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National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire

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TABLE OF CONTENTS

PART 1 SUMMARY

1.1 Introduction
1.2 Property Description, Location and Ownership
1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography
1.4 History
1.5 Geology and Mineralisation
1.6 Status of Exploration, Development and Operations
1.7 Data Verification, Sample Preparation, Analysis and Security
1.8 Mineral Processing and Metallurgical Test Work
1.9 Mineral Resource Estimate
1.10 Mineral Reserve Estimate
1.11 Mining Methods
1.12 Recovery Methods
1.13 Project Infrastructure
1.14 Market Studies and Contracts
1.15 Environmental Studies, Permitting and Social or Community Impact
1.16 Capital and Operating Costs
1.17 Economic Analysis
1.18 Schedule, Project Implementation and Other Relevant Information
1.19 Interpretations and Conclusions
1.20 Recommendations

PART 2 INTRODUCTION

2.1 Issuer
2.2 Scope and Terms of Reference
2.3 Sources of Information
2.4 Contributing Consultants
2.5 Risks and Forward Looking Statements
2.6 Qualified Persons
2.7 Site Visits and Purpose

PART 3 RELIANCE ON OTHER EXPERTS

PART 4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

4.2 Tenure

4.3 Datum and Projection

4.4 Royalties

4.5 Environmental Liabilities

4.6 Permitting

4.7 Other Significant Factors and Risks

PART 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.2 Physiography

5.3 Climate

5.4 Site Characteristics

5.4 Local Resources and Infrastructure

PART 6 HISTORY

PART 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

7.2 Project Geology

PART 8 DEPOSIT TYPES

8.1 Introduction

PART 9 EXPLORATION

9.1 Coordinates, Survey Controls and Topographic Surveys

9.2 Geological Reconnaissance, Mapping and Rock Chip Sampling

9.3 Airborne Geophysical Survey

9.4 Soil Sampling

9.5 Auger Drilling

9.6 Trenching

9.7 Regolith Mapping and Interpretation

9.8 Gradient Array Induced Polarisation Survey

9.9 Aircore Drilling

PART 10 DRILLING

10.1 Drilling Summary

10.2 Collar Surveys

10.3 Downhole Surveys

10.4 Drill Coverage and Orientation

10.5 Sample Recovery and Grade

10.6 Wet RC Samples

10.7 Logging

PART 11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Reverse Circulation Sampling Methods

11.2 Diamond Core Sampling Methods

11.3 Chain of Custody and Transport

11.4 Sample Preparation

11.5 Sample Analysis at Laboratory

11.6 Bulk Density Determinations

11.7 Quality Assurance and Quality Control Sampling

PART 12 DATA VERIFICATION

12.1 Site Visit

12.2 Data Verification and Validation

12.3 Topography

12.4 Summary Opinion of Qualified Person

PART 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgy and Mineral Processing Test Work

13.2 Feasibility Test Work

13.3 Comminution Test Work

13.4 Sample Provenance

13.5 Head Assay Analysis

13.6 Mineralogy

13.7 Grind and Cyanide Optimization Test Work
13.8 Oxygen Versus Air Direct Cyanidation Leach Tests
13.9 Cyanide Optimization Test Work
13.10 Pre-Oxidation Shear Reactor Test Work (Mach©)
13.11 Optimised CIL and Pre-Oxidation CIL Test Work
13.12 Thickener Test Work
13.13 Cyanide Detox Test Work
13.14 Gold Recovery - Regression Analysis

PART 14 MINERAL RESOURCE ESTIMATES

14.1 Introduction
14.2 Software
14.3 Data Used for Estimation
14.4 Drill Database
14.5 Actions on Undefined/Null and Below Detection Limit Samples
14.6 Wireframe Models and Data Coding
14.6.1 Mineralisation
14.6.2 Weathering, Oxidation and Regolith
14.7 Sample Compositing
14.8 Boundary Analysis on Weathering Surfaces
14.9 Basic Statistics
14.10 Gold Grade Caps and Distance Limiting
14.11 Diffusivity Tests
14.12 Variography
14.13 Block Model Definition
14.14 Ordinary Kriging Interpolation
14.15 Localised Uniform Conditioning
14.16 Density Assignments
14.17 Mining Depletion
14.18 Mineral Resource Estimate Validation
14.19 Mineral Resource Estimate Classification
14.20 Mineral Resource Statement
14.21 Factors that May Affect the Mineral Resource

14.22 Comparison to Previous Mineral Resource
14.23 Audits and Reviews

PART 15 MINERAL RESERVES ESTIMATES
15.1 Introduction
15.2 Reliance On Other Experts
15.3 Mineral Reserve Estimate

PART 16 MINING METHODS
16.1 Introduction
16.2 Mining Method Selection
16.3 Geotechnical Considerations
16.4 Mining Contractor Request for Budget Pricing
16.5 Mining Dilution and Recovery Factors
16.6 Open Pit Optimisation
16.7 Mine Design
16.8 Life of Mine Production Schedule
16.9 Mining Cost Rates and Assumptions
16.10 LOM Mining Contractor Cost Estimate

PART 17 RECOVERY METHODS
17.1 Introduction
17.2 Summary Flowsheet
17.3 Process Description
17.4 Process Design
17.5 Plant Design
17.6 Plant General
17.7 Equipment Selection

PART 18 PROJECT INFRASTRUCTURE
18.1 Introduction
18.2 Roads and Site Access
18.3 Logistics

18.4 Geotechnical Considerations
18.5 Tailings Storage Facility
18.6 Surface Water Management
18.7 Power Supply and Distribution
18.8 Bulk Earthworks
18.9 Mine Infrastructure
18.10 Administration and Processing Infrastructure
18.11 Accommodation Village
18.12 Sewage Treatment
18.13 Fuel and Lubricants
18.14 Communications
18.15 Security and Fencing

PART 19 MARKET STUDIES AND CONTRACTS

19.1 Introduction
19.2 Contracts
19.3 Market Outlook
19.4 Gold Demand
19.5 Available Smelting and Refining Options and Costs
19.6 Payment Terms
19.7 Shipping

PART 20 ENVIRONMENTAL AND SOCIAL ASSESSMENT

20.1 Introduction
20.2 Legislative Requirements, Guidelines and Corporate Commitments
20.3 Baseline Setting
20.4 Management Systems/Approach
20.5 Physical Environmental Impacts, Management and Mitigation
20.6 Socio-economic Impacts, Management and Mitigation
20.7 Biological and Ecological Impacts, Management and Mitigation
20.8 Archaeology and Cultural Heritage
20.9 Cumulative Impacts
20.10 Stakeholder Engagement

20.11 Occupational Health and Safety
20.12 Emergency Preparedness and Response
20.13 ESMMP Framework
20.14 Rehabilitation and Mine Closure
20.15 Resettlement and Livelihood Restoration
20.16 Conclusions

PART 21 CAPITAL AND OPERATING COSTS

21.1 Introduction
21.2 Capital Cost Estimate Summary
21.3 Basis of Capital Cost Estimate Development
21.4 Operating Cost Estimate
21.5 Mining Operating Costs
21.6 Processing Plant Operating Costs
21.7 Site General and Administration
21.8 Exclusions

PART 22 ECONOMIC ANALYSIS

22.1 Introduction
22.2 Summary
22.3 Principal Assumptions and Inputs

PART 23 ADJACENT PROPERTIES

PART 24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation
24.2 Development Methodology
24.3 Project Management
24.4 Implementation
24.5 Engineering
24.6 Procurement and Logistics
24.7 Construction
24.8 Completions and Commissioning Process
24.9 Operations Readiness

PART 25 INTERPRETATION AND CONCLUSIONS

25.1 Metallurgy
25.2 Mineral Resources
25.3 Mining Methods
25.4 Recovery Methods
25.5 Project Infrastructure Risks and Opportunities
25.6 Environmental, Permitting, Social and Community Impact
25.7 Project Implementation Schedule

PART 26 RECOMMENDATIONS

26.1 Mineral Resource Recommendations
26.2 Mining Methods Recommendations
26.3 Metallurgical Test Work Recommendations
26.4 Environmental and Social Recommendations
26.5 Project Infrastructure Recommendations
26.6 Project Implementation Recommendations

PART 27 REFERENCES

FIGURES

Part 1

Figure 1.1	*Doropo Gold Project Location (Source: Centamin, 2024)*
Figure 1.2	*Major lithological domains*
Figure 1.3	*Geological Modelling of Weathering Zones (Source: SRK, 2024)*
Figure 1.4	*LOM Total Mining by Mining Area - Annual (Source: Orelogy, 2024)*
Figure 1.5	*Mining Sequence by Pit Stage (Source: Orelogy, 2024)*
Figure 1.6	*Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade (Source: Orelogy, 2024)*
Figure 1.7	*Doropo Simplified Process Flow Diagram (Source: GRES, 2024)*
Figure 1.8	*Site Layout (Source: Knight Piésold, 2024)*
Figure 1.9	*Mine Infrastructure Layout (Source: Knight Piésold, 2024)*
Figure 1.10	*Main Project Development Area (MPDA) (Source: Centamin, 2024)*
Figure 1.11	*All proposed Project Development Areas, including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs (Source: Centamin, 2024)*
Figure 1.12	*Project Timeline (Source: Centamin, 2024)*

Part 4

Figure 4.1	*Location of the Doropo Project - Map of Côte d'Ivoire*

Part 5

Figure 5.1	*Elevations Over the Project Area - SRTM Data*
Figure 5.2	*Main Vegetation Zones in West Africa*

Part 7

Figure 7.1	*Map of West African Craton*
Figure 7.2	*Geology of the Leo-Man Shield - from the BRGM Interpretations*
Figure 7.3	*Geology Map of Doropo*
Figure 7.4	*Deposit Scale Geology and Prospect Locations*
Figure 7.5	*Distal Epidote-Chlorite-Weak Hematite Alteration on Granodiotite (from Footwall Zone at Souwa - DPDD1382 @ 103.4 m Depth)*
Figure 7.6	*Distal Alteration: Haematite-Chlorite Pervasive Alteration (from Direct Footwall of Mineralisation at Souwa* - DPRC0504 @ 157 m Depth)
Figure 7.7	*Proximal Intense Sericite and Fine Grained Disseminated Pyrite Alteration (from the Han Deposit - DPRD0470 @ 94 m Depth)*
Figure 7.8	*Strong Silica-Sericite Alteration - High Grade Mineralisation (from the Han Deposit - DPRD0470 @ 98.7 m Depth)*
Figure 7.9	*Two Examples of Doropo Saprolite Showing Quartz Vein Fragments in the Right Hand Photo*
Figure 7.10	*Saprock (Original Fabric and Mineralogy is Partially Preserved)*
Figure 7.11	*Saprock*
Figure 7.12	*Transition Zone Between Lower Saprolite and Fresh Rock*
Figure 7.13	*Fresh Granodiorite*
Figure 7.14	*Souwa Prospect Geological History*

Part 8

Figure 8.1	*Schematic Geological Model Interpretation of the Resource Area (Shear Zones in Black, Quartz Veins in Yellow and Doleritic Dykes in Green)*

Part 9

Figure 9.1	*Location and Results of Rock Chip Sampling Programmes*
Figure 9.2	*Exploration Data: Magnetic Imagery (top) and Regolith Mapping*
Figure 9.3	*Location and Results of Soil Sampling Programmes*
Figure 9.4	*Location and Results of AUGER Sampling Programmes*
Figure 9.5	*Exploration Data: Gridded Gold Values in Surface Geochemistry (Merged Soils and Auger Data)*
Figure 9.6	*Sterilisation AC drilling completed at the Main Cluster in 2024 (Source: Centamin, 2024)*
Figure 9.7	*Combined Aircore, RC and DD drill hole locations (Source: Centamin, 2024)*

Part 10

Figure 10.1	*The Geodrill UDR650 Multipurpose Drill Rig with Compressor Visible in the Background*
Figure 10.2	*An Example of a Concrete Plinth Set Around PVC Casing at Souwa*
Figure 10.3	*The Reflex EZ Shot Tool About to be Inserted into a Drill Hole to Take a Downhole Survey Reading*
Figure 10.4	*Plan View of the Project Area Showing Drilling at the Various Prospects*
Figure 10.5	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at ATI*
Figure 10.6	*Cross-Section View at ATI Looking South-East*
Figure 10.7	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at CHG and NOK*
Figure 10.8	*Cross-Section View at CHG Main Looking North-East*
Figure 10.9	*Cross-Section View at CHG South Looking North-Northwest*
Figure 10.10	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at ENI*
Figure 10.11	*Cross-Section View at ENI Looking North*
Figure 10.12	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at HAN*
Figure 10.13	*Cross-Section View at HAN Looking North-East*
Figure 10.14	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at KEK*
Figure 10.15	*Cross-Section View at KEK Looking North-East*
Figure 10.16	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at KILO*
Figure 10.17	*Cross-Section View at KILO Looking South-East*
Figure 10.18	*Plan View of the Drill Holes and Mineralisation Domain Wireframes at SWA*
Figure 10.19	*Cross-Section View at SWA Looking North-East*
Figure 10.20	*Scatter Plot of RC Sample Weight Versus Downhole Depth*
Figure 10.21	*Scatter Plot of RC Sample Weight Versus Gold Grade*
Figure 10.22	*Histogram of Drill Core Recovery Percentage for Doropo DD Holes*
Figure 10.23	*Scatter Plot of DD Core Recovery Percentage Versus Downhole Depth*
Figure 10.24	*Scatter Plot of Gold Grade Versus DD Core Recovery Percentage*

Part 11

Figure 11.1	*Accuracy and Precision Concepts*
Figure 11.2	*OREAS 211 - MSA_YAM Results - Au ppm*
Figure 11.3	*OREAS 211 - VER_AB Results - Au ppm*
Figure 11.4	*OREAS 219 - VER_AB Results - Au ppm*

Figure 11.5	*OREAS 219 - ALS_IRL Results - Au ppm*
Figure 11.6	*OREAS 235 - VER_AB Results - Au ppm*
Figure 11.7	*OREAS 237 - MSA_YAM Results - Au ppm*
Figure 11.8	*OREAS 237 - VER_AB Results - Au ppm*
Figure 11.9	*OREAS 237 - VALS_IRL - Au ppm*
Figure 11.10	*OREAS 239 - MSA_YAM Results - Au ppm*
Figure 11.11	*OREAS 239 - VER_AB Results - Au ppm*
Figure 11.12	*OREAS 239 - ALS_IRL Results - Au ppm*
Figure 11.13	*OREAS 250b - MSA_YAM Results - Au ppm*
Figure 11.14	*OREAS 250b - VER_AB Results - Au ppm*
Figure 11.15	*OREAS 250b - ALS_IRL Results - Au ppm*
Figure 11.16	*OREAS 251 - VER_AB Results - Au ppm*
Figure 11.17	*OREAS 253 - MSA_YAM Results - Au ppm*
Figure 11.18	*OREAS 253 - VER_AB Results - Au ppm*
Figure 11.19	*OREAS 253 - ALS_IRL Results - Au ppm*
Figure 11.20	*OREAS 253b - MSA_YAM Results - Au ppm*
Figure 11.21	*OREAS 253b - VER_AB Results - Au ppm*
Figure 11.22	*OREAS 256b - MSA_YAM Results - Au ppm*
Figure 11.23	*OREAS 256b - VER_AB Results - Au ppm*
Figure 11.24	*OREAS 256b - ALS_IRL Results - Au ppm*
Figure 11.25	*OREAS 22h Blank - MSA_YAM Results - Au ppm*
Figure 11.26	*OREAS 22h Blank - VER_AB Results - Au ppm*
Figure 11.27	*OREAS 22h Blank - ALS_IRL Results - Au ppm*
Figure 11.28	*CBLKY Coarse Blank - MSA_YAM Results - Au ppm*
Figure 11.29	*CBLKY Coarse Blank - VER_AB Results - Au ppm*
Figure 11.30	*CBLKY Coarse Blank - ALS_IRL Results - Au ppm*
Figure 11.31	*Prospect ATI, RC Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.32	*Prospect ATI, RC Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.33	*Prospect ATI, RC Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.34	*Prospect ATI, DD Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.35	*Prospect ATI, DD Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.36	*Prospect ATI, DD Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.37	*Prospect CHG, RC Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.38	*Prospect CHG, RC Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.39	*Prospect CHG, RC Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.40	*Prospect CHG, DD Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.41	*Prospect CHG, DD Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.42	*Prospect CHG, DD Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.43	*Prospect CHG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.44	*Prospect CHG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.45	*Prospect CHG, DD Pulp Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.46	*Prospect ENI, RC Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.47	*Prospect ENI, RC Field Duplicate Q-Q Plot - To Date - Au ppm*

Figure 11.48 Prospect ENI, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.49 Prospect HAN, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.50 Prospect HAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.51 Prospect HAN, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.52 Prospect HAN, DD Pulp Duplicate Scatter Plot - To Date - Au ppm
Figure 11.53 Prospect HAN, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.54 Prospect HAN, DD Pulp Duplicate RMDP Plot - To Date - Au ppm
Figure 11.55 Prospect HND, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.56 Prospect HND, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.57 Prospect HND, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.58 Prospect KEK, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.59 Prospect KEK, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.60 Prospect KEK, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.61 Prospect KEK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm
Figure 11.62 Prospect KEK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.63 Prospect KEK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm
Figure 11.64 Prospect KLG, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.65 Prospect KLG, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.66 Prospect KLG, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.67 Prospect KLG, DD Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.68 Prospect KLG, DD Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.69 Prospect KLG, DD Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.70 Prospect KLG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm
Figure 11.71 Prospect KLG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.72 Prospect KLG, DD Pulp Duplicate RMPD Plot - To Date - Au ppm
Figure 11.73 Prospect NAR, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.74 Prospect NAR, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.75 Prospect NAR, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.76 Prospect NOK, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.77 Prospect NOK, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.78 Prospect NOK, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.79 Prospect NOK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm
Figure 11.80 Prospect NOK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.81 Prospect NOK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm
Figure 11.82 Prospect SAN, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.83 Prospect SAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.84 Prospect SAN, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.85 Prospect SOL, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.86 Prospect SOL, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.87 Prospect SOL, RC Field Duplicate RMDP Plot - To Date - Au ppm
Figure 11.88 Prospect SWA, RC Field Duplicate Scatter Plot - To Date - Au ppm
Figure 11.89 Prospect SWA, RC Field Duplicate Q-Q Plot - To Date - Au ppm
Figure 11.90 Prospect SWA, RC Field Duplicate RMDP Plot - To Date - Au ppm

Figure 11.91 *Prospect SWA, DD Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.92 *Prospect SWA, DD Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.93 *Prospect SWA, DD Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.94 *Prospect SWA, Pulp Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.95 *Prospect SWA, Pulp Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.96 *Prospect SWA, Pulp Duplicate RMPD Plot - To Date - Au ppm*
Figure 11.97 *Prospect THN, RC Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.98 *Prospect THN, RC Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.99 *Prospect THN, RC Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.100 *Prospect VAKO, RC Field Duplicate Scatter Plot - To Date - Au ppm*
Figure 11.101 *Prospect VAKO, RC Field Duplicate Q-Q Plot - To Date - Au ppm*
Figure 11.102 *Prospect VAKO, RC Field Duplicate RMDP Plot - To Date - Au ppm*
Figure 11.103 *QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 5 m*
Figure 11.104 *QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 7.5 m*
Figure 11.105 *QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 10 m*
Figure 11.106 *Log-Probability Plots Between Au from DD Holes and RC-Migrated Holes with Migration Distances of 5 m (a), 7.5 m (b), and 10 m (c)*
Figure 11.107 *RC-DD Comparison - NOK*
Figure 11.108 *RC-DD Comparison - CHG*
Figure 11.109 *Suspected RC Contamination - CHG*
Figure 11.110 *Suspected RC Contamination - HAN*

Part 12

Figure 12.1 *Log Scatterplot of the Independent vs Original Sample Results from Hole DPRC3081*

Part 13

Figure 13.1 *Ore Comminution Characteristics*
Figure 13.2 *Provenance of Lithology Composites from the Souwa Open Pit*
Figure 13.3 *Provenance of Lithology Composites from the Nokpa Open Pit*
Figure 13.4 *Provenance of Lithology Composites from the Kilosegui Open Pit (NW Section)*
Figure 13.5 *Provenance of Lithology Composites from the Kilosegui Open Pit (Central Section)*
Figure 13.6 *Provenance of Lithology Composites from the Kilosegui Open Pit (SE Section)*
Figure 13.7 *Provenance of Lithology Composites from the Kekeda Open Pit*
Figure 13.8 *Provenance of Lithology Composites from the Enioda Open Pit*
Figure 13.9 *Provenance of Lithology Composites from the Han Open Pit*
Figure 13.10 *Provenance of Lithology Composites from the Chegue Main*
Figure 13.11 *Provenance of Lithology Composites from the Chegue Main (North)*
Figure 13.12 *Provenance of Lithology Composites from the Chegue South*
Figure 13.13 *Master Composite - Model Mineralogy*
Figure 13.14 *Souwa Grind Optimization Leach Kinetic Curves*
Figure 13.15 *Nokpa Grind Optimization Leach Kinetic Curves*
Figure 13.16 *Kilosegui Grind Optimization Leach Kinetic Curves*
Figure 13.17 *Kekeda Grind Optimization Leach Kinetic Curves*
Figure 13.18 *Enioda Grind Optimization Kinetic Leach Curves*

Figure 13.19 Han Grind Optimization Leach Kinetic Curves
Figure 13.20 Chegue Main Grind Optimization Leach Kinetic Curves
Figure 13.21 Chegue South Grind Optimization Leach Kinetic Curves
Figure 13.22 Souwa Cyanide Optimization Leach Kinetic Curves
Figure 13.23 Nokpa Cyanide Optimization Leach Kinetic Curves
Figure 13.24 Kilosegui Cyanide Optimization Leach Kinetic Curves
Figure 13.25 Kekeda Cyanide Optimization Kinetic Leach Curves
Figure 13.26 Enioda Cyanide Optimization Kinetic Leach Curves
Figure 13.27 Han Cyanide Optimization Kinetic Leach Curves
Figure 13.28 Chegue Main Cyanide Optimization Kinetic Leach Curves
Figure 13.29 Chegue South Cyanide Optimization Kinetic Leach Curves
Figure 13.30 Souwa Regression Curve
Figure 13.31 Nokpa Regression Curve
Figure 13.32 Kilosegui Regression Curve
Figure 13.33 Kekeda Regression Curve
Figure 13.34 Enioda Regression Curve
Figure 13.35 Chegue Main Regression Curve
Figure 13.36 Chegue Main Regression Curve
Figure 13.37 Chegue South Regression Curve

Part 14
Figure 14.1 Log Probability Plots per Prospect for Gold Grade Assays, Showing Grade Inflexions at 0.2 g/t to 0.3 g/t, the Approximate Threshold Used to Define the Mineralised Zone Boundaries
Figure 14.2 ATI Interpreted Mineralised Lodes with Drill Holes
Figure 14.3 CHG and NOK Interpreted Mineralised lodes with Drill Holes
Figure 14.4 ENI Interpreted Mineralised Lodes with Drill Holes
Figure 14.5 HAN Interpreted Mineralised Lodes with Drill Holes
Figure 14.6 HND and HNDS Interpreted Mineralised Lodes with Drill Holes
Figure 14.7 KEK Interpreted Mineralised Lodes with Drill Holes
Figure 14.8 KILO Interpreted Mineralised Lodes with Drill Holes
Figure 14.9 NAR Interpreted Mineralised Lodes with Drill Holes
Figure 14.10 SAN Interpreted Mineralised Lodes with Drill Holes
Figure 14.11 SOL Interpreted Mineralised Lodes with Drill Holes
Figure 14.12 SWA Interpreted Mineralised Lodes with Drill Holes
Figure 14.13 THN Interpreted Mineralised Lodes with Drill Holes
Figure 14.14 VAKO Interpreted Mineralised Lodes with Drill Holes
Figure 14.15 Histogram of Raw Assay Length (m)
Figure 14.16 Scatter Plot of Raw Assay Length Versus Gold Grade
Figure 14.17 Boundary Analysis Between Oxide (200 Series) Vs Fresh (100 series) at Enioda Prospect
Figure 14.18 CHG Domain 5001 Log-probability Plot with Population Break at 0.2 g/t Au
Figure 14.19 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ATI Domains
Figure 14.20 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - CHG Domains
Figure 14.21 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ENI Domains

Figure 14.22 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HAN Domains
Figure 14.23 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HND Domains
Figure 14.24 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HNDS Domains
Figure 14.25 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KEK Domains
Figure 14.26 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KLG Major Domains
Figure 14.27 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NAR Minor Domains
Figure 14.28 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NOK Domains
Figure 14.29 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SAN Domains
Figure 14.30 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SOL Domains
Figure 14.31 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SWA Domains
Figure 14.32 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - THN Domains
Figure 14.33 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - VAKO Domains
Figure 14.34 Diffusion Test for Gold Grade Based on 1 m Composites (2022 MRE)
Figure 14.35 Example Gaussian Variogram (left) and Back-Transformed Variogram (right) for Domain 2001 at VAKO
Figure 14.36 KNA Plot Using the Kriging Slope of Regression and Kriging Efficiency as Criteria for Selection of Minimum and Optimum Informing Composites - CHG domain 50041 (Ore domain 5004)
Figure 14.37 Example Swath Plots for Gold Grade in Domain 12001 at ATI
Figure 14.38 Cross-section View Looking SE at ATI. Block Grades and Drill Assay Grades
Figure 14.39 Plan View of the Mineral Resource Classification at ATI
Figure 14.40 Isometric View Looking SW of the Mineral Resource Classification at ATI
Figure 14.41 Plan View of the Mineral Resource Classification at CHG-NOK
Figure 14.42 Plan View of the Mineral Resource Classification at ENI
Figure 14.43 Isometric View Looking E of the Mineral Resource Classification at ENI
Figure 14.44 Plan View of the Mineral Resource Classification at HAN
Figure 14.45 Isometric View Looking SE of the Mineral Resource Classification at HAN
Figure 14.46 Plan View of the Mineral Resource Classification at HND
Figure 14.47 Plan View of the Mineral Resource Classification at KEK
Figure 14.48 Plan View of the Mineral Resource Classification at KILO
Figure 14.49 Zoom View of the Mineral Resource Classification at KILO
Figure 14.50 Plan View of the Mineral Resource Classification at NAR
Figure 14.51 Plan View of the Mineral Resource Classification at SAN
Figure 14.52 Plan View of the Mineral Resource Classification at SOL
Figure 14.53 Plan View of the Mineral Resource Classification at SWA
Figure 14.54 Isometric View Looking ESE of the Mineral Resource Classification at SWA
Figure 14.55 Plan View of the Mineral Resource Classification at THN
Figure 14.56 Plan View of the Mineral Resource Classification at VAKO
Figure 14.57 Grade and Tonnage Curves for In-pit Classified Material - ATI
Figure 14.58 Grade and Tonnage Curves for In-pit Classified Material - CHG Main
Figure 14.59 Grade and Tonnage Curves for In-pit Classified Material - CHG South
Figure 14.60 Grade and Tonnage Curves for In-pit Classified Material - ENI
Figure 14.61 Grade and Tonnage Curves for In-pit Classified Material - HAN
Figure 14.62 Grade and Tonnage Curves for In-pit Classified Material - HND

Figure 14.63 Grade and Tonnage Curves for In-pit Classified Material - HNDS
Figure 14.64 Grade and Tonnage Curves for In-pit Classified Material - KEK
Figure 14.65 Grade and Tonnage Curves for In-pit Classified Material - KILO
Figure 14.66 Grade and Tonnage Curves for In-pit Classified Material - NAR
Figure 14.67 Grade and Tonnage Curves for In-pit Classified Material - NOK
Figure 14.68 Grade and Tonnage Curves for In-pit Classified Material - SAN
Figure 14.69 Grade and Tonnage Curves for In-pit Classified Material - SOL
Figure 14.70 Grade and Tonnage Curves for in-pit Classified Material - SWA
Figure 14.71 Grade and Tonnage Curves for In-pit Classified Material - THN
Figure 14.72 Grade and Tonnage Curves for In-pit Classified Material - VAKO

Part 16
Figure 16.1 Location of Mining Areas
Figure 16.2 Geological Modelling of Weathering Zones
Figure 16.3 Pit Slope Design Parameters
Figure 16.4 Comparison of November 2023 RBP Submission ($/t Mined Basis)
Figure 16.5 Best & Worst Case DCF Mining Sequence (Cross-Section View)
Figure 16.6 Souwa Pit Optimisation - Tonne/Value Chart
Figure 16.7 Souwa Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.8 Kilosegui Pit Optimisation - Tonne/Value Chart
Figure 16.9 Kilosegui Pit Optimisation -Strip Ratio and Cost Per Oz
Figure 16.10 Han Pit Optimisation - Tonne/Value Chart
Figure 16.11 Han Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.12 Kekeda Pit Optimisation - Tonne/Value Chart
Figure 16.13 Kekeda Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.14 Nokpa Pit Optimisation - Tonne/Value Chart
Figure 16.15 Nokpa Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.16 Chegue Main Pit Optimisation - Tonne/Value Chart
Figure 16.17 Chegue Main Pit Optimisation - Strip Ratio and Cost per oz
Figure 16.18 Chegue South Pit Optimisation - Tonne/Value Chart
Figure 16.19 Chegue South Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.20 Enioda Pit Optimisation - Tonne/Value Chart
Figure 16.21 Enioda Pit Optimisation - Strip Ratio and Cost Per Oz
Figure 16.22 Two-way Ramp Layout
Figure 16.23 One-way Ramp Layout
Figure 16.24 General Site Arrangement
Figure 16.25 Pit Design - Souwa Borrow Pit
Figure 16.26 Pit Design - Souwa Stage 1
Figure 16.27 Pit Design - Souwa Stage 1A
Figure 16.28 Pit Design - Souwa Stage 2
Figure 16.29 Pit Design - Souwa Stage 3
Figure 16.30 Pit Design - Souwa Stage 4
Figure 16.31 Pit Design - Nokpa Stage 1

Figure 16.32 *Pit Design - Nokpa Stage 2*
Figure 16.33 *Pit Design - Kilosegui East*
Figure 16.34 *Pit Design - Kilosegui West*
Figure 16.35 *Pit Design - Enioda*
Figure 16.36 *Pit Design - Han*
Figure 16.37 *Pit Design - Kekeda*
Figure 16.38 *Pit Design - Chegue Main and South*
Figure 16.39 *General Site Arrangement - Hub Mining Area*
Figure 16.40 *General Site Arrangement - Kekeda, Han, Enioda Satellite Mining Area*
Figure 16.41 *General Site Arrangement - Kilosegui Mining Area*
Figure 16.42 *ROM Pad Layout Design*
Figure 16.43 *Process Plant Ramp-Up (% of Nominal Throughput)*
Figure 16.44 *Pit Stage Layout for LOM Production Schedule (Hub and Satellite Pits)*
Figure 16.45 *Pit Stage Layout for LOM Production Schedule (Kilosegui)*
Figure 16.46 *LOM Total Mining by Mining Area (Annual)*
Figure 16.47 *Total Mining by Mining Area (Pre-Production to End Year 2)*
Figure 16.48 *Mining Sequence by Pit Stage*
Figure 16.49 *Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade*
Figure 16.50 *Utilisation of Plant Capacity (hours)*
Figure 16.51 *Feed Ounces, Recovered Ounce and Recovery*
Figure 16.52 *Stockpile Balance and Grade by Weathering*
Figure 16.53 *Annual Mining Costs*
Figure 16.54 *Mining Contractor Cost Estimate Breakdown*

Part 17
Figure 17.1 *Process Flowsheet*
Figure 17.2 *Laboratory Scale Gravity Recovery by Master Composite*
Figure 17.3 *Gold Extraction Curves for Oxide/Transition and Fresh Ores*

Part 18
Figure 18.1 *Site Layout (Source: Knight Piésold, 2024)*
Figure 18.2 *Mine Infrastructure Layout (Source: Knight Piésold, 2024)*
Figure 18.3 *Plant and infrastructure Layout (Source: GRES, 2024)*
Figure 18.4 *Transport Route*
Figure 18.5 *TSF General Arrangement - Stage 1 (Source: Knight Piésold, 2024)*
Figure 18.6 *TSF General Arrangement - Final Stage (Source: Knight Piésold, 2024)*
Figure 18.7 *Final Stage TSF Embankment - Typical Section (Source: Knight Piésold, 2024)*
Figure 18.8 *WHD Layout (Source: Knight Piésold, 2024)*
Figure 18.9 *WSD Layout (Source: Knight Piésold, 2024)*

Part 20
Figure 20.1 *Main Project Development Area (MPDA)*
Figure 20.2 *All proposed Project Development Areas, Including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs*

Figure 20.3 *Topography over the Doropo Gold Project Area*
Figure 20.4 *Doropo Gold Project Location and Administrative Boundaries*
Figure 20.5 *Hand Dug Well Lagbo*
Figure 20.6 *Hand Pump Dupindouo*
Figure 20.7 *Lagbo Health Centre (H&B, May 2023)*
Figure 20.8 *Holidouo Community School (H&B, May 2023)*
Figure 20.9 *Shrub Savannah (Wet Season)*
Figure 20.10 *West African Bowal (Dry Season)*
Figure 20.11 *Gallery Forest (Wet Season)*
Figure 20.12 *Swampy Thicket (Wet Season)*
Figure 20.13 *Patas Monkey (Erythrocebus patas)*
Figure 20.14 *Bushbuck (Tragelaphus scriptus)*
Figure 20.15 *Dwarf Crocodile (Osteolaemus tetraspis)*
Figure 20.16 *C.rouxae*

Part 21
Figure 21.1 *Processing Operating Cost Estimate Summary by Cost Centre (Fresh)*
Figure 21.2 *Processing Operating Cost Estimate Summary by Cost Centre (Oxide/Transition)*
Figure 21.3 *Processing Operating Cost Estimate Summary by Area (Fresh)*
Figure 21.4 *Processing Operating Cost Estimate Summary by Area (Oxide/Transition)*

Part 24
Figure 24.1 *Summary of Project Implementation Schedule*

TABLES

Part 1

Table 1.1	*Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023*
Table 1.2	*Gold Grade Versus Recovery (Regression Analysis)*
Table 1.3	*Doropo Updated Mineral Resource Estimate (0.3 g/t Au COG) (CIM Definition Standards), 31 October 2023*
Table 1.4	*Breakeven Cut-Off Grade - Modifying Factors*
Table 1.5	*Breakeven Cut-Off Grades*
Table 1.6	*Doropo Gold Project FS Mineral Reserve Estimate*
Table 1.7	*Description of Geological Weathering Domains*
Table 1.8	*Saprolite (CW & HW) Design Recommendations*
Table 1.9	*Saprock and Fresh Rock (MW, SW & FR) Design Recommendations*
Table 1.10	*LOM Production Schedule Physicals by Period (Pre-Production to Year 3)*
Table 1.11	*LOM Production Schedule Physicals by Period (Year 4 to Year 9)*
Table 1.12	*Historical Annual Average LBMA Gold Prices (S&P Capital IQ)*
Table 1.13	*Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023)*
Table 1.14	*Indicative Employment Numbers and Classification – Construction and Operations*
Table 1.15	*Land Cover Within the Doropo Project Development Areas (PDAs)*
Table 1.16	*Project Capital Cost Estimate Summary*
Table 1.17	*Sustaining Capital Estimate Summary*
Table 1.18	*Project Operating Cost Estimate Summary*
Table 1.19	*Project Economic Summary*
Table 1.20	*Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz*
Table 1.21	*Operating Expenditure % Change vs Gold Price US$/oz, NPV8% in US$M*
Table 1.22	*Up-front Capital Cost % Change vs Gold Price US$/oz, NPV8% in US$M*
Table 1.23	*Project Implementation Milestones*

Part 2

Table 2.1	*Contributing Consultants to the FS Report*
Table 2.2	*Qualified Persons*
Table 2.3	*Site Visits by QPs*

Part 4

Table 4.1	*Summary of the Exploration Permits - as of June 2023*

Part 10

Table 10.1	*Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023*
Table 10.2	*Breakdown of Available Drill Data for the Mineral Resource Estimate Update of the Updated Prospects Only, as at 27 August 2023*

Part 11
Table 11.1 *Analytical Methods Used in the Doropo Project for Gold Assays*
Table 11.2 *Average Density Values and Number of Samples Taken for Each Oxidation Profile*
Table 11.3 *Average Density for Each Rock Type*
Table 11.4 *Nominal QAQC Sample Insertion Rates for Doropo*
Table 11.5 *Actual QAQC Sample Insertion Rates for the Doropo Gold Project - Project to Date*
Table 11.6 *Doropo Gold Project CRM Breakdown (September 2021 - July 2023)*
Table 11.7 *Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Performance Summary by Type and Laboratory - Au (ppm)*
Table 11.8 *Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Blanks Breakdown*
Table 11.9 *Doropo Gold Project - To Date Period - Duplicates Breakdown by Prospect*
Table 11.10 *Reverse Circulation Field Original vs. Duplicate Statistics - To Date*
Table 11.11 *Diamond Drilling Field Original vs. Duplicate Statistics - To Date*
Table 11.12 *Diamond Drilling Pulp Original vs. Duplicate Statistics - To Date*
Table 11.13 *Field Duplicates Performance by Deposit - Project to Date*
Table 11.14 *Pulp Duplicate Performance by Deposit - Project to Date*
Table 11.15 *Paired Bivariate Statistics Between Au Values from DD Holes and Au Values from RC Holes Migrated to the DD Holes (the Migrated Distance is Shown After the Variable Name)*

Part 12
Table 12.1 *Drill Collar Position, Azimuth and Inclination Checks*
Table 12.2 *Listing of Original versus Independent Sample Results from Hole DPRC3081*

Part 13
Table 13.1 *Gold Grade Versus Recovery (Regression Analysis)*
Table 13.2 *Comminution Test Work Samples*
Table 13.3 *SMC Tests*
Table 13.4 *SMC Parameters*
Table 13.5 *Comminution Test Work Results Summary*
Table 13.6 *Feasibility Study Sample Summary*
Table 13.7 *Feasibility Study Composite Sample Weights*
Table 13.8 *Souwa Oxide Master Composite Details (MC1)*
Table 13.9 *Souwa Transitional Master Composite Details (MC2)*
Table 13.10 *Souwa Oxide Master Composite Details (MC3)*
Table 13.11 *Souwa Oxide Low Grade Variability Composite Details (VC1)*
Table 13.12 *Souwa Oxide High Grade Variability Composite Details (VC2)*
Table 13.13 *Souwa Transitional Low Grade Variability Composite Details (VC3)*
Table 13.14 *Souwa Transitional High Grade Variability Composite Details (VC4)*
Table 13.15 *Souwa Fresh Low Grade Variability Composite Details (VC5)*
Table 13.16 *Souwa Fresh High Grade Variability Composite Details (VC6)*
Table 13.17 *Nokpa Oxide Master Composite Details (MC4)*
Table 13.18 *Nokpa Transitional Master Composite Details (MC5)*
Table 13.19 *Nokpa Fresh Master Composite Details (MC6)*

Table 13.20 Nokpa Oxide Low Grade Variability Composite Details (VC7)
Table 13.21 Nokpa Transitional Low Grade Variability Composite Details (VC8)
Table 13.22 Nokpa Transitional High Grade Variability Composite Details (VC9)
Table 13.23 Nokpa Fresh Low Grade Variability Composite Details (VC10)
Table 13.24 Nokpa Fresh High Grade Variability Composite Details (VC11)
Table 13.25 Kilosegui Oxide Master Composite Details (MC7)
Table 13.26 Kilosegui Transitional Master Composite Details (MC8)
Table 13.27 Kilosegui Fresh Master Composite Details (MC9)
Table 13.28 Kilosegui Transitional Low Grade Variability Composite Details (VC12)
Table 13.29 Kilosegui Transitional High Grade Variability Composite Details (VC13)
Table 13.30 Kilosegui Fresh Low Grade Variability Composite Details (VC14)
Table 13.31 Kilosegui Fresh High Grade Variability Composite Details (VC15)
Table 13.32 Kekeda Oxide Master Composite Details (MC10)
Table 13.33 Kekeda Transitional Master Composite Details (MC11)
Table 13.34 Kekeda Fresh Master Composite Details (MC12)
Table 13.35 Kekeda Oxide Low Grade Variability Composite Details (VC16)
Table 13.36 Kekeda Oxide High Grade Variability Composite Details (VC17)
Table 13.37 Kekeda Transitional Low Grade Variability Composite Details (VC18)
Table 13.38 Kekeda Transitional High Grade Variability Composite Details (VC19)
Table 13.39 Kekeda Fresh Low Grade Variability Composite Details (VC20)
Table 13.40 Kekeda Fresh High Grade Variability Composite Details (VC21)
Table 13.41 Enioda Oxide Master Composite Details (MC13)
Table 13.42 Enioda Transitional Master Composite Details (MC14)
Table 13.43 Enioda Fresh Master Composite Details (MC15)
Table 13.44 Enioda Oxide Low Grade Variability Composite Details (VC22)
Table 13.45 Enioda Oxide High Grade Variability Composite Details (VC23)
Table 13.46 Enioda Transitional Low Grade Variability Composite Details (VC24)
Table 13.47 Enioda Transitional High Grade Variability Composite Details (VC25)
Table 13.48 Enioda Fresh Low Grade Variability Composite Details (VC26)
Table 13.49 Enioda Fresh High Grade Variability Composite (VC27)
Table 13.50 Han Oxide Master Composite Details (MC16)
Table 13.51 Han Transitional Master Composite Details (MC17)
Table 13.52 Han Fresh Master Composite Details (MC18)
Table 13.53 Han Transitional Low Grade Variability Composite Details (VC28)
Table 13.54 Han Transitional High Grade Variability Composite Details (VC29)
Table 13.55 Han Fresh Low Grade Variability Composite Details (VC30)
Table 13.56 Han Fresh High Grade Variability Composite Details (VC31)
Table 13.57 Chegue Main Oxide Master Composite Details (MC19)
Table 13.58 Chegue Main Traditional Master Composite Details (MC20)
Table 13.59 Chegue Main Fresh Master Composite Details (MC21)
Table 13.60 Chegue Main Oxide Low Grade Variability Composite Details (VC32)
Table 13.61 Chegue Main Oxide High Grade Variability Composite Details (VC33)
Table 13.62 Chegue Main Transitional Low Grade Composite Details (VC34)

Table 13.63 Chegue Main Transitional High Grade Variability Composite Details (VC35)
Table 13.64 Chegue Main Fresh Low Grade Variability Composite Details (VC36)
Table 13.65 Chegue Main Fresh High Grade Variability Composite Details (VC37)
Table 13.66 Chegue South Oxide Master Composite Details (MC22)
Table 13.67 Chegue South Transitional Master Composite Details (MC23)
Table 13.68 Chegue South Fresh Master Composite Details (MC24)
Table 13.69 Chegue South Transitional Low Grade Variability Composite Details (VC38)
Table 13.70 Chegue South Transitional High Grade Variability Composite Details (VC39)
Table 13.71 Chegue South Fresh Low Grade Variability Composite Details (VC40)
Table 13.72 Chegue South Fresh High Grade Variability Composite Details (VC41)
Table 13.73 Head Assay Analysis MC1 - MC6
Table 13.74 Head Assay Analysis MC7 - MC12
Table 13.75 Head Assay Analysis MC13 - MC18
Table 13.76 Head Assay Analysis MC19 - MC24
Table 13.77 Souwa Variability Head Assay Analysis VC1 - VC6
Table 13.78 Nokpa Variability Head Assay Analysis VC7 - VC11
Table 13.79 Kilosegui Variability Head Assay Analysis VC12 - VC15
Table 13.80 Kekeda Variability Head Assay Analysis VC17 - VC21
Table 13.81 Enioda Variability Head Assay Analysis VC22 - VC27
Table 13.82 Han Variability Head Assay Analysis VC28 - VC31
Table 13.83 Chegue Main Variability Head Assay Analysis VC32 - VC37
Table 13.84 Chegue South Variability Head Assay Analysis VC38 - VC41
Table 13.85 Master Composites Souwa & Nokpa - Model Mineralogy
Table 13.86 Master Composites Kilosegui & Kekeda - Model Mineralogy
Table 13.87 Master Composites Enioda & Han - Model Mineralogy
Table 13.88 Master Composites Chegue Main & Chegue South - Model Mineralogy
Table 13.89 Master Composites Souwa & Nokpa - Sulphur Deportment
Table 13.90 Master Composites Kiloegui & Kekeda - Sulphur Deportment
Table 13.91 Master Composites Enioda & Han - Sulphur Deportment
Table 13.92 Master Composites Chegue Main & Chegue South - Sulphur Deportment
Table 13.93 Souwa Grind Optimization Leach Results
Table 13.94 Nokpa Grind Optimization Leach Results
Table 13.95 Kilosegui Grind Optimization Leach Results
Table 13.96 Kekeda Grind Optimization Leach Results
Table 13.97 Enioda Grind Optimization Leach Results
Table 13.98 Han Grind Optimization Leach Results
Table 13.99 Chegue Main Grind Optimization Leach Results
Table 13.100 Chegue South Grind Optimization Leach Results
Table 13.101 Souwa Master Composites - Oxygen Versus Air Leach Results
Table 13.102 Nokpa Master Composites - Oxygen Versus Air Leach Results
Table 13.103 Kilosegui Master Composites - Oxygen Versus Air Leach Results
Table 13.104 Kekeda Master Composites - Oxygen Versus Air Leach Results
Table 13.105 Enioda Master Composites - Oxygen Versus Air Leach Results

Table 13.106 Han Master Composites - Oxygen Versus Air Leach Results
Table 13.107 Chegue Main Master Composites - Oxygen Versus Air Leach Results
Table 13.108 Chegue South Master Composites - Oxygen Versus Air Leach Results
Table 13.109 Souwa Cyanide Optimization Leach Results
Table 13.110 Nokpa Cyanide Optimization Leach Results
Table 13.111 Kilosegui Cyanide Optimization Leach Results
Table 13.112 Kekeda Cyanide Optimization Leach Results
Table 13.113 Enioda Cyanide Optimization Leach Results
Table 13.114 Han Cyanide Optimization Leach Results
Table 13.115 Chegue Main Cyanide Optimization Leach Results
Table 13.116 Chegue South Cyanide Optimization Leach Results
Table 13.117 Souwa Fresh Master Composite - Mach Reactor Leach Results
Table 13.118 Nokpa Fresh Master Composite - Mach Reactor Leach Results
Table 13.119 Kilosegui Fresh Master Composite - Mach Reactor Leach Results
Table 13.120 Kekeda Fresh Master Composite - Mach Reactor Leach Results
Table 13.121 Souwa Composite Pre-Oxidation Leach Results
Table 13.122 Nokpa Composites Pre-Oxidation Leach Results
Table 13.123 Kilosegui Composites Pre-Oxidation Leach Results
Table 13.124 Kekeda Composites Pre-Oxidation Leach Results
Table 13.125 Enioda Composites Pre-Oxidation Leach Results
Table 13.126 Han Composites Pre-Oxidation Leach Results
Table 13.127 Chegue Main Composites Pre-Oxidation Leach Results
Table 13.128 Chegue South Pre-Oxidation Leach Results
Table 13.129 Thickener Test Work Results
Table 13.130 Cyanide Detox Test Work Conditions
Table 13.131 Souwa Oxide Continuous Cyanide Detoxification Results
Table 13.132 Souwa Oxide Bulk Cyanide Detoxification Results
Table 13.133 Souwa Trans Continuous Cyanide Detoxification Results
Table 13.134 Souwa Trans Bulk Cyanide Detoxification Results
Table 13.135 Souwa Fresh Continuous Cyanide Detoxification Results
Table 13.136 Souwa Fresh Bulk Cyanide Detoxification Results
Table 13.137 Nokpa Trans Continuous Cyanide Detoxification Results
Table 13.138 Nokpa Trans Bulk Cyanide Detoxification Results
Table 13.139 Nokpa Fresh Continuous Cyanide Detoxification Results
Table 13.140 Nokpa Fresh Bulk Cyanide Detoxification Results
Table 13.141 Leach Recovery Regression Format
Table 13.142 Leach Recovery Regression Co-efficients

Part 14
Table 14.1 Estimation Databases and Prospects
Table 14.2 Weathering Zones Modelled and the Logging Codes ('WI' Field) Used
Table 14.3 Regolith Zones Modelled and the Logging Codes ('Lith1' Field) Used
Table 14.4 Oxidation Zones Modelled and the Logging Codes ('WI' Field) Used

Table 14.5	*Basic Statistics of the Composites (naïve) all Domains in the Various Updated Prospect Areas*
Table 14.6	*Listing of Gold Grade Caps and Distance Limiting Parameters per Mineralisation/Estimation Domain*
Table 14.7	*Variogram Model Parameters per Mineralisation/Estimation Domain - Sills Normalised to 100%*
Table 14.8	*Block Model Definition per Prospect Area*
Table 14.9	*OK Panel Block Sizes Used for Gold Interpolation per Prospect*
Table 14.10	*OK Panel Block Search Parameters for Gold Grades*
Table 14.11	*Basic Statistics for Dry Density (t/m3) per Prospect*
Table 14.12	*Dry Density (t/m3) Assignments in the Block Models for the Updated Mineral Resource*
Table 14.13	*Comparison Between 2022 MRE (drilling at 25 m x 25 m) and 2023 MRE (Drilling at 10 m x 10 m) Within the Grade Control Drilled Areas*
Table 14.14	*Doropo Updated Measured Mineral Resource Estimate (CIM Definition Standards), 31 October 2023*
Table 14.15	*Doropo Updated Indicated Mineral Resource Estimate (CIM Definition Standards), 31 October 2023*
Table 14.16	*Doropo Updated Inferred Mineral Resource Estimate (CIM Definition Standards), 31 October 2023*
Table 14.17	*Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.18	*Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.19	*Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.20	*Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.21	*Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.22	*Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.23	*Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.24	*Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.25	*Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.26	*Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.27	*Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.28	*Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.29	*Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*
Table 14.30	*Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au*
Table 14.31	*Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au*

Table 14.32 Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.33 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.34 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.35 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.36 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.37 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.38 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.39 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.40 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.41 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.42 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.43 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.44 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.45 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade cut-off of 0.3 g/t Au
Table 14.46 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.47 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au
Table 14.48 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au
Table 14.49 Reported Measured Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE
Table 14.50 Reported Indicated Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE
Table 14.51 Reported Inferred Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE

Part 15

Table 15.1 Breakeven Cut-Off Grade - Modifying Factors
Table 15.2 Breakeven Cut-Off Grades
Table 15.3 Doropo Gold Project FS Mineral Reserve Estimate

Part 16
Table 16.1 *Description of Geological Weathering Domains*
Table 16.2 *Saprolite (CW & HW) Design Recommendations*
Table 16.3 *Saprock and Fresh Rock (MW, SW & FR) Design Recommendations*
Table 16.4 *Request for Budget Pricing Items*
Table 16.5 *Average of Competitive Contractor RBP Submission ($/t Mined Basis)*
Table 16.6 *Resource Block Model Parameters*
Table 16.7 *Dilution Parameters by Orebody*
Table 16.8 *Approximate Ore Loss and Dilution*
Table 16.9 *Saprolite (Oxide) Weathering Zones Geotechnical Parameters*
Table 16.10 *Saprock (Transition) and Fresh Weathering Zones Geotechnical Parameters*
Table 16.11 *Slope Angles by Mining Area used in the Whittle Optimisation Process*
Table 16.12 *Average Waste Mining Cost by Deposit*
Table 16.13 *Process Recoveries*
Table 16.14 *Ore Cost*
Table 16.15 *Satellite Ore Haulage Costs*
Table 16.16 *Product Revenue Parameters*
Table 16.17 *Optimisation Results by Mining Area*
Table 16.18 *Pit Design Inventory and % Difference from Optimisation Shells*
Table 16.19 *Waste Rock Dump Design Criteria*
Table 16.20 *Waste Rock Dump Designed Capacity Versus Required Capacity*
Table 16.21 *LOM Production Schedule Periods*
Table 16.22 *Schedule Production Rate Cat 6015*
Table 16.23 *Schedule Production Rate Cat 6020*
Table 16.24 *Digging Fleet and Mining Capacity by Period*
Table 16.25 *LOM Production Schedule Physicals by Period (Pre-Production to Year 3)*
Table 16.26 *LOM Production Schedule Physicals by Period (Year 4 to Year 9)*
Table 16.27 *Selected RFQ Submission - Mobilisation and Demobilisation*
Table 16.28 *Selected RFQ Submission - Site Establishment and Infrastructure*
Table 16.29 *Selected RFQ Submission - Drill and Blast Rates ($/BCM)*
Table 16.30 *Selected RFQ Submission - Waste Load and Haul Rates ($/BCM)*
Table 16.31 *Selected RFQ Submission - Ore Load and Haul Rates ($/BCM)*
Table 16.32 *Selected RBP Submission - Fixed Fees and Unit Rates*
Table 16.33 *Selected RBQ Submission - Ore Reclaim Rates*
Table 16.34 *Selected RFQ Submission - Satellite Pit Ore Haulage Rates*
Table 16.35 *Total Mining Contract Costs to Year 3 by Quarterr*
Table 16.36 *Total Mining Contract Costs to Year 4 - Year 9 by Quarter/Year*

Part 17
Table 17.1 *Plant Power Requirements*
Table 17.2 *Ore Sources*
Table 17.3 *Metallurgical Tested Comminution Properties*
Table 17.4 *Circuit Design Criteria*

Table 17.5 Processing Plant Annual Consumption Rates for Key Reagents and Consumables
Table 17.6 Comminution Design Parameters
Table 17.7 SAG Mill Operating Conditions
Table 17.8 Ball Mill Operating Conditions
Table 17.9 Leach Circuit Design Parameters
Table 17.10 Adsorption (CIL) Design Parameters
Table 17.11 Elution Circuit Design Parameters

Part 18
Table 18.1 Site Access Roads Design Parameters
Table 18.2 Site Haul Roads Design Parameters
Table 18.3 Site Airstrip Design Parameters
Table 18.4 Geotechnical Investigation Summary
Table 18.5 TSF Design Parameters
Table 18.6 Staged Embankment Construction
Table 18.7 Case 1 - Indicative Population at Risk (PAR)
Table 18.8 Case 2 - Indicative Population at Risk (PAR)
Table 18.9 Case 3 - Indicative Population at Risk (PAR)
Table 18.10 ANCOLD Consequence Category - Dam Failure
Table 18.11 ANCOLD Consequence Category - Environmental Spill
Table 18.12 ANCOLD Design Criteria
Table 18.13 Water Harvesting Dam Design Parameters
Table 18.14 Water Storage Dam Design Parameters
Table 18.15 Sediment Management System Design Parameters
Table 18.16 SCS Design Information
Table 18.17 Surface Water Management System Design Parameters

Part 19
Table 19.1 Historical Annual Average LBMA Gold Prices (S&P Capital IQ)
Table 19.2 Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023)

Part 20
Table 20.1 Key Studies Conducted to Support the Project Feasibility Studies and Statutory ESIA
Table 20.2 Key Air Quality, Noise and Water Standards, Guidelines and Legal Requirements
Table 20.3 Project Design Criteria
Table 20.4 Land Cover within the Doropo PDAs

Part 21
Table 21.1 Foreign Exchange Rates
Table 21.2 Project Capital Cost Estimate Summary
Table 21.3 Processing and infrastructure Capital Cost Estimate Summary
Table 21.4 Owner's Cost Estimate Summary
Table 21.5 Mechanical Equipment Pricing
Table 21.6 Sustaining Capital Estimate Summary

Table 21.7 *Project Operating Cost Estimate Summaries*
Table 21.8 *Mining Operating Cost Estimate Summaries*
Table 21.9 *Processing Operating Cost Estimate Summaries by Cost Centre*
Table 21.10 *Processing Operating Cost Estimate Summaries by Area*
Table 21.11 *Power Summary - Fresh Feed*
Table 21.12 *Power Summary - Oxide-Transition Feed*
Table 21.13 *Fresh Ore Reagent Cost and Consumptions*
Table 21.14 *Oxide-Transitional Ore Reagent Cost and Consumptions*
Table 21.15 *Processing Plant Labour Cost Summary*

Part 22
Table 22.1 *Economic Summary*
Table 22.2 *Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz*
Table 22.3 *Operating Expenditure % Change vs Gold Price US$/oz, NPV8% in US$M*
Table 22.4 *Up-front Capital Cost % Change vs Gold Price US$/oz, NPV8% in US$M*

Part 24
Table 24.1 *Key Milestone Dates for Project Implementation*

TABLE OF CONTENTS

1. SUMMARY 1
1.1 Introduction 1
1.2 Property Description, Location and Ownership 1
1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography 2
1.4 History 3
1.5 Geology and Mineralisation 4
1.6 Status of Exploration, Development and Operations 5
1.7 Data Verification, Sample Preparation, Analysis and Security 6
1.8 Mineral Processing and Metallurgical Test Work 7
1.8.1 Mineralogy 9
1.8.2 Comminution 9
1.8.3 Cyanidation 10
1.8.4 Thickening and Cyanide Detoxification 11
1.8.5 Metallurgical Recovery 11
1.9 Mineral Resource Estimate 13
1.9.1 Mineral Resource Estimate Validation 14
1.9.2 Factors that May Affect the Mineral Resource 14
1.10 Mineral Reserve Estimate 14
1.11 Mining Methods 16
1.11.1 Mining Strategy 16
1.11.2 Geotechnical Considerations 17
1.11.3 Mining Equipment 22
1.11.4 Mining Schedule 23
1.12 Recovery Methods 29
1.13 Project Infrastructure 31
1.13.1 TSF Design Summary 34
1.13.2 Water Management 34
1.13.3 Water Harvest Dam 35
1.13.4 Water Storage Dam 35
1.13.5 Access Roads 35
1.13.6 Airstrip 35
1.13.7 Power Supply 35
1.14 Market Studies and Contracts 36
1.15 Environmental Studies, Permitting and Social or Community Impact 37
1.15.1 Project Setting 38
1.15.2 Socio-Economic Benefit 40

1.15.3 Employment and Local Economy ... 41
1.15.4 Training and Skills Development ... 42
1.15.5 Land Acquisition, Livelihood Restoration and Resettlement ... 43
1.15.6 Land and Livelihood ... 44
1.15.7 Soil Geochemistry ... 45
1.16 Capital and Operating Costs ... 47
1.16.1 Capital Costs ... 47
1.16.2 Sustaining Capital ... 48
1.16.3 Operating Costs ... 49
1.16.4 Closure Costs ... 50
1.17 Economic Analysis ... 50
1.18 Schedule, Project Implementation and Other Relevant Information ... 52
1.19 Interpretations and Conclusions ... 54
1.19.1 Economic and Social Benefits ... 54
1.19.2 Metallurgy ... 55
1.19.3 Mineral Resources ... 55
1.19.4 Mineral Reserves ... 55
1.20 Recommendations ... 56
1.20.1 Metallurgical Test Work Recommendations ... 56
1.20.2 Mineral Resource Recommendations ... 57
1.20.3 Mineral Reserve Recommendations ... 58

LIST OF FIGURES

Figure 1.1 Doropo Gold Project Location (Source: Centamin, 2024) ... 2
Figure 1.2 Major lithological domains ... 18
Figure 1.3 Geological Modelling of Weathering Zones (Source: SRK, 2024) ... 19
Figure 1.4 LOM Total Mining by Mining Area - Annual (Source: Orelogy, 2024) ... 25
Figure 1.5 Mining Sequence by Pit Stage (Source: Orelogy, 2024) ... 25
Figure 1.6 Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade (Source: Orelogy, 2024) ... 26
Figure 1.7 Doropo Simplified Process Flow Diagram (Source: GRES, 2024) ... 30
Figure 1.8 Site Layout (Source: Knight Piésold, 2024) ... 32
Figure 1.9 Mine Infrastructure Layout (Source: Knight Piésold, 2024) ... 33
Figure 1.10 Main Project Development Area (MPDA) (Source: Centamin, 2024) ... 39
Figure 1.11 All proposed Project Development Areas, including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs (Source: Centamin, 2024) ... 40
Figure 1.12 Project Timeline (Source: Centamin, 2024) ... 53

LIST OF TABLES

Table 1.1 Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023 6
Table 1.2 Gold Grade Versus Recovery (Regression Analysis) 12
Table 1.3 Doropo Updated Mineral Resource Estimate (0.3 g/t Au COG) (CIM Definition Standards), 31 October 2023....14
Table 1.4 Breakeven Cut-Off Grade - Modifying Factors....15
Table 1.5 Breakeven Cut-Off Grades 16
Table 1.6 Doropo Gold Project FS Mineral Reserve Estimate....16
Table 1.7 Description of Geological Weathering Domains....19
Table 1.8 Saprolite (CW & HW) Design Recommendations....20
Table 1.9 Saprock and Fresh Rock (MW, SW & FR) Design Recommendations....22
Table 1.10 LOM Production Schedule Physicals by Period (Pre-Production to Year 3)....27
Table 1.11 LOM Production Schedule Physicals by Period (Year 4 to Year 9) 28
Table 1.12 Historical Annual Average LBMA Gold Prices (S&P Capital IQ)....36
Table 1.13 Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023) 36
Table 1.14 Indicative Employment Numbers and Classification – Construction and Operations....43
Table 1.15 Land Cover Within the Doropo Project Development Areas (PDAs)....45
Table 1.16 Project Capital Cost Estimate Summary....48
Table 1.17 Sustaining Capital Estimate Summary....49
Table 1.18 Project Operating Cost Estimate Summary 49
Table 1.19 Project Economic Summary....51
Table 1.20 Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz....51
Table 1.21 Operating Expenditure % Change vs Gold Price US$/oz, NPV8% in US$M....51
Table 1.22 Up-front Capital Cost % Change vs Gold Price US$/oz, NPV8% in US$M 52
Table 1.23 Project Implementation Milestones....53

1. SUMMARY

1.1 Introduction

This Technical Report has been prepared by Centamin plc (Centamin) and GR Engineering Services Limited (GRES), in conjunction with various consultants listed below, in the third quarter of 2024. This Technical Report documents the results of the investigations completed for the Feasibility Study (FS, DFS or the Study) undertaken for the proposed development of Centamin's Doropo Gold Project (Doropo or "the Project") in northeastern Côte d'Ivoire.

The Project was previously the subject of an NI 43-101 Technical Report, Prefeasibility Study (PFS), dated August 10, 2023 (by Centamin and Lycopodium Minerals Pty Ltd).

This Technical Report has been prepared in compliance with the disclosure requirements of the Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

This Technical Report incorporates the work of Qualified Persons (QPs) from Centamin, GRES, Cube Consulting Pty Ltd (Cube), Orelogy Consulting Pty Ltd (Orelogy), Knight Piésold Pty Ltd (Knight Piésold), ECG Engineering Pty Ltd (ECG), SRK Consulting (UK) Ltd (SRK) and Independent Metallurgical Operations Ltd (IMO).

1.2 Property Description, Location and Ownership

The Project is located in north-eastern Côte d'Ivoire, in the Bounkani region, 480 km north of the capital Abidjan and 50 km north of the city of Bouna.

The Project comprises sixteen prospects/deposits namely: Souwa (SWA), Nokpa (NOK), Chegue Main and Chegue South (CHG), Kekeda (KEK), Han (HAN), Enioda (ENI), Kilosegui (KILO), Attire (ATI), Hinda (HND), Hinda South (HNDS), Nare (NAR), Sanboyoro (SAN), Solo (SOL), Tchouahinin (THN) and Vako (VAK). Most of the deposits (11) are within a 7 km radius, with Vako and Kilosegui at a radius of approximately 15 km and 30 km respectively.

The Project is contained within seven current exploration permits that were granted to Ampella Mining Côte d'Ivoire (AMCI) and Ampella Mining Exploration Côte d'Ivoire (AMEXCI), which are both 100% owned Ivoirian subsidiaries of Centamin. The block of permits covers a total area of 1,847 km^2.

The Doropo prospects listed above are located on five out of the seven exploration permits. The sixteen deposits that comprise the Mineral Resource occur within a radius of approximately 25 km centred on UTM 482,450 mE and 1,074,951 mN (WGS84, zone 30N).

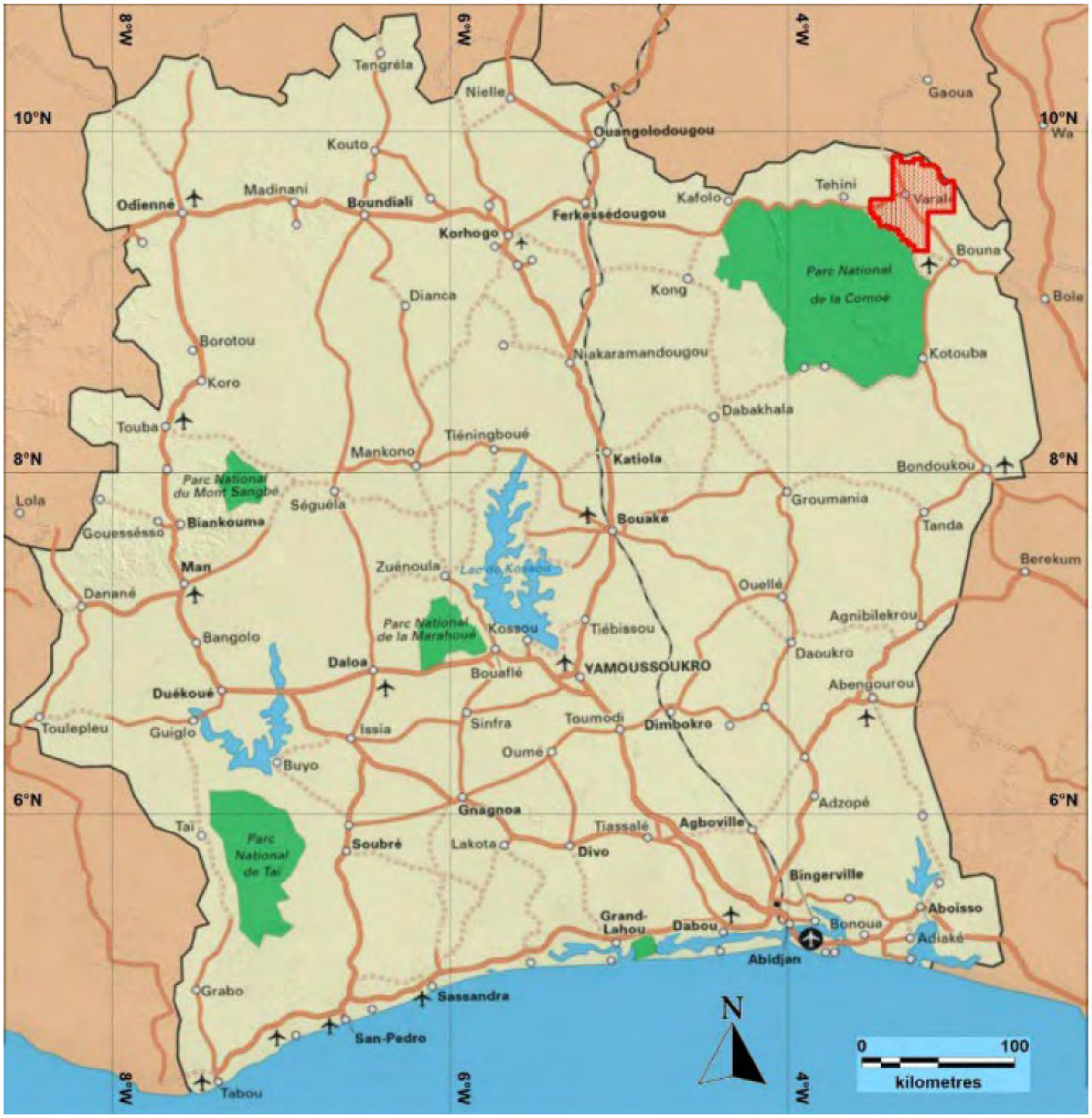


Figure 1.1 Doropo Gold Project Location (Source: Centamin, 2024)

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project area is accessible by a national sealed road called the A1, which crosses through the centre of the Project. The A1 is a major road that links Abidjan and Ouagadougou, the capitals of Côte d'Ivoire and Burkina Faso, respectively. Doropo prefecture is 76 km (about 1.5 hours' drive) from Bouna, the Capital of the Bounkani Region. It is also 240 km (about 3.5 hours' drive) from Bondoukou, the capital of the district, and 645 km (about 10 to 11 hours' drive) from Abidjan, the economic capital. A dense network of small dirt/sandy roads allows easy access to all parts of the Project, even during the wet season. The sandy nature of the soil allows a rapid drainage of the water on the access roads generally.

The Doropo area has relatively subdued relief due to the nature of the underlying rocks - the granites. The surface soils are mostly sandy, and outcrops are rare. The ridges form small plateaus and are covered by laterites and occasionally duricrust of limited thickness.

The climate is of Sudanese type, with two distinct seasons, a rainy season and a dry season. The rainy season extends from May/June to September/October when rainfall totals between 1,100 mm and 1,200 mm. The dry season extends from September/October to May/June. The Harmattan, a hot dry wind coming from the Sahara regions, generally blows in December and January, sometimes extending to March, and brings dust clouds, which reduce visibility. The average annual temperature is 28°C, ranging between 21°C and 33°C. The hottest times of the year occur at the change of seasons.

Local infrastructure remains limited, therefore the Project development will need to include access roads, power, water, accommodation, and communications.

The company La Société des Energies de Côte d'Ivoire (CI-ENERGIES) own the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government. The power grid is available from the Bouna substation, south of Doropo, which is fed a 90 kV transmission line from the 225 kV Bondoukou substation.

The mobile phone network is well deployed, from at least two main national providers. Internet access has overall proven reliable, via the general 3G and 4G mobile connections, or dedicated microwave connections for the sites.

Underground water is not abundant in the Project area and hence hydrological studies completed have indicated that water harvesting via a water harvest dam (WHD) and distribution pumps is required to supply the required water demand of the Project. A suitably sized water storage dam (WSD) has been designed to cater for long term-supply of water to the Project.

Due to the rural aspect of the area, the specialised professional skills and trade skills are very limited in the near vicinity, but adequate workforces are available from elsewhere in the country.

1.4 History

The first exploration permits of the area were granted to Ampella Mining Côte d'Ivoire, an Ivoirian subsidiary, in June 2013. Prior to that time, no systematic mineral exploration had ever been conducted in the area.

Some evidence of historical gold mining during the Colonial times (under the French management) is seen at Varale, where a small open pit type operation occurred, most likely shallow surface workings on outcropping quartz veins. However, this operation seems to not have been documented.

Ampella Mining Ltd made application for the Kalamon, Varale and Doropo Ouest permits in 2010, and the permits were granted in June 2013. This was followed by a reconnaissance visit to the Doropo area at the end of 2013 and the identification of several exploration targets based on rock chip sampling of exposures in artisanal pits. The Doropo Ouest permit was later relinquished in August 2019. The Kalamon and Varale permits cover the area of the Main Cluster of gold deposits. A further seven (7) exploration permits were granted to Ampella between 2015 and 2018 (Danoa, Gogo, Bouna, Bouna Nord and the 3 Tehini permits). Bouna and Bouna Nord were also relinquished in August 2019 and May 2017, respectively.

Centamin acquired Ampella Mining Côte d'Ivoire via the takeover of Ampella Mining Ltd. in March 2014. Exploration activities then started on the Doropo Project from mid-2014.

1.5 Geology and Mineralisation

The Doropo permits lie entirely within the Tonalite-Trondhjemite-Granodiorite (TTG) domain, bounded on its eastern side by the Boromo-Batie greenstone belt in Burkina Faso and by the Tehini greenstone belt in the west.

At the Project scale, the geology consists of granite-gneiss terrain, the granite being mostly of granodioritic composition. The granites are intruded by an abundant series of pegmatitic veins and quartz veins, ranging from the decimetre scale to several hundreds of metres scale. Some of this veining hosts gold mineralisation, often as primary native gold, across the entire area. This generates regular dispersed gold anomalism in the surface geochemistry; it is also the main source of the gold extracted by the artisanal miners but is mostly uneconomic at the industrial scale.

Large, late doleritic dykes criss-cross the whole domain at the regional scale.

The host rocks of the Doropo gold deposits comprise a homogeneous medium to coarse-grained granite-granodiorite complex, which has been locally intruded by late-stage gabbro-dolerite dykes and some pegmatite veins. In addition, there are occasional biotite rich or aplitic dykes, the latter mostly associated with the Enioda deposit, which also shows some amphibolite layers. Outcrops of recognisable rock is rare, but is generally confined to erosional valley sides. Interfluvial ridge lines are often covered by hard laterite cuirasse, while drainage lines are filled with lateritic soils and transported sediments (alluvium, colluvium).

Economically interesting mineralisation is associated with discrete structures of intense silica-sericite alteration, focused within and along the margins of narrow (5-10 m wide to locally 20-25 m wide) dextral shear zones. Outside of the mineralised zones, the granodiorite is fairly undeformed.

The alteration assemblages seen at Doropo from distant to proximal are epidote ± chlorite, chlorite ± magnetite, haematite-silica, silica-sericite-leucoxene then silica flooding ± pyrite ± gold.

In distant regions from mineralised zones, the mineral assemblage is dominated by epidote-chlorite and haematite. Haematite alteration is widespread, ranging from weak to medium intensity. Close to doleritic dykes, haematite alteration intensifies, making it challenging to determine mineralisation direction, notably at the Nokpa prospect.

Proximal mineral assemblages include strong silica-sericite alteration that often overprints earlier haematite and silica alteration. The sulphides, mostly pyrite, are abundant throughout the core of the shear zone; they host part of the gold mineralisation. The other portion of the gold mineralisation occurs as native gold in quartz veins and selvages. Chlorite-magnetite alteration precedes the main pyrite-associated silica-sericite alteration phase.

1.6 Status of Exploration, Development and Operations

Only minor exploration work was conducted before Centamin took over the Doropo Project in 2014. This work was limited to field reconnaissance and rock chip sampling and was carried out by Ampella Mining Ltd.

Centamin started exploration work in 2014, progressing from regional field mapping to the surface geochemistry sampling, via soils and auger, to the geophysical surveys, ground surveys and airborne surveys, to trenching, Air Core (AC) drilling and then Reverse Circulation (RC) and Diamond Drilling (DD).

A regional aeromagnetic and radiometric survey, with additional detailed infill surveying over the Doropo Project, was flown by UTS Geophysics/Geotech Airborne Limited (UTS) between 24 March and 27 May in 2015 (Wood, 2015). The resulting imagery supported the initial regional interpretations and then the first regional exploration programmes.

Auger drilling was largely used on the Project to complete the soil grid surveys where the thickest lateritic plateaus cover the insitu material and where the transported horizons (alluvium and sand) average over 3 m thickness. A total of 27,999 auger holes have been drilled up until January 2022 with an average depth of 6.22 m and a maximum grade of 28 g/t Au.

In total, 92,307 soil samples were collected between 2014 and December 2022, including 20,815 samples on infill grids on the Project. Most of the deposits and current prospects are well highlighted by the soils results.

Campaigns of AC drilling were regularly conducted to quickly test coherent geochemical gold anomalism, conceptual targets, or extensions to known mineralised structures. From June 2015 to December 2019, 142,947 m were drilled at an average length of 29.27 m and predominant dip of -55 degrees. From December 2023 to January 2024, 188 holes for 4,732 m at an average length of 25 m were drilled to sterilise planned infrastructure (e.g. waste dumps, processing plant, laydown areas, camp and water treatment facilities) as part of the FS process. Programmes were drilled on a 500 x 50 to 25 m spacing or one section across the middle of the planned infrastructure to ensure no economic zones of mineralisation were present as a first phase.

The Reverse Circulation (RC) and Diamond Drilling (DD) programmes have been undertaken using a phased approach on the Doropo Project since the end of November 2015, following the first significant hits from the aircore (AC) drill programmes.

A summary of the hole types and meterage is provided in Table 1.1.

Hole Type	No. Holes	Total Length (m)	Ave. Depth (m)
DD	450	43,404	96
RC	5,245	487,440	93
RCD	99	16,961	171
TOTAL	5,794	547,805	95

Table 1.1 Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023

1.7 Data Verification, Sample Preparation, Analysis and Security

Bureau Veritas Minerals Laboratory Abidjan, Côte d'Ivoire (VER_AB), was the primary analytical laboratory used for gold fire assay on the Doropo Project. The Bureau Veritas Minerals head office is in Paris, France, and Bureau Veritas Minerals is independent of Centamin.

The OMAC Laboratories Ltd ALS Loughrea laboratory in Galway, Ireland (ALS_IRL) was used as a secondary laboratory. ALS_IRL is independent of Centamin.

MSA Laboratory Yamoussoukro (MSA_YAM) was utilised in 2023 to conduct Chrysos photon analysis on 500 g, 2 mm crushed samples. MSA_YAM is accredited by the International Accreditation Service (IAS) which accredits laboratories to ISO/IEC Standard 17025 (Testing and Calibration) and Intertek for both ISO/IEC Standard 17025 (Testing and Calibration Laboratories) and ISO 9001 (Quality Management Systems) accreditation.

Centamin established an on-site sample preparation laboratory at Danoa in late 2021, the construction of which was observed by the Qualified Person in August 2021. Prior to this preparation facility becoming operational, all samples were prepared at either VER_AB or ALS_IRL.

Drilling has been carried in several campaigns, spread across the time period from 2015 to July 2023. For the purposes of this report, the QAQC analysis of the field duplicates is on the whole dataset and split on deposit, while for the certified reference materials (standards/CRMs) the analysis was conducted on samples submitted from September 2021 to July 2023.

Cube has independently assessed the QAQC data for the Project and the following summary and conclusions have been noted from the periods under review.

The QAQC for the Doropo Gold Project involves insertion of CRMs and Blank material following a set of company Standard Operating Procedures (SOPs).

The QAQC data show that the sampling and analysis process has generally been robust, support the use of the RC and DD samples for Mineral Resource estimation. Although it appears that in some instances DD samples produce a lower grade, the impact is not considered material, as the DD holes only make up around 10% of the total Mineral Resource estimation dataset.

Based on an assessment of the available data, the Qualified Person considers the dataset collected by Centamin to be acceptable for Mineral Resource estimation, with QA protocols and QC results posing minimal risk to the overall confidence level of the Mineral Resource estimation for the period assessed. Analysis of QAQC results at prior Mineral Resource updates similarly showed that the data were acceptable for adoption in Mineral Resource estimation.

1.8 Mineral Processing and Metallurgical Test Work

The FS test work scope followed on from the PFS test work reported in the 2023 PFS Technical Report. Independent Metallurgical Operations (IMO) was retained by Centamin in June 2023 to initiate and oversee metallurgical test work on samples from the Doropo gold deposits.

The test work objectives were to:

- Establish and demonstrate the optimum process operating parameters for each lithology type in each orebody;
- Conduct leach tests on master composites from each lithology in each orebody to determine the optimum conditions and expected gold recoveries;
- Conduct leach tests on variability composites at optimised conditions to determine gold recoveries at varying head grades for the three lithology types in each orebody; and
- Determine key process design criteria (PDC) to allow selection and sizing of mechanical equipment.

A key focus of the FS phase test work has been the assessment of the pre-oxidation and cyanidation leach circuit configuration to optimise gold recovery and reduce cyanide consumption over the base case reported at the close of the PFS. The FS test work scope included MACH© shear reactor testing to increase leach kinetics, as well as a range of PDC supporting test work performed by ALS in Perth.

The FS test work programme has provided focus on optimisation of the leaching response across a range of representative master and variability composites formed predominately from PQ and HQ drill-core, with some RC drill samples obtained for variability testing.

The following metallurgical flowsheet development testing has been performed:

- Historical comminution test work (as reported for the 2023 PFS Technical Report);
- Comprehensive Master and Variability lithology composite head assay analysis;
- Mineralogical analysis by QEMSCAN and XRD;
- Grind optimisation cyanidation testing on all master composites;
- Cyanide optimisation cyanidation testing on all master composites;
- Bulk Carbon-in-Leach (CIL) test work on Souwa, Nokpa and Kilosegui composites under optimised test conditions;
- Direct cyanidation leach tests on Souwa and Nokpa master composites under optimised test conditions comparing oxygen versus air sparging;
- Pre-oxidation shear reactor MACH© testing targeting improved gold recovery and faster leach kinetics;
- Optimised CIL and pre-oxidation test work with lead nitrate on all master composites;
- Bulk CIL test work including cyanide detoxification testing and dynamic thickener testing of detox tailings.

The results reported in this document are from test work completed as at Friday 14 June 2024. Further test work was well advanced at this time, however, the results were not expected to necessitate changes to the process flowsheet. The outstanding test work will be reported and used in the next phase of the development of the Project.

The outstanding test work not deemed necessary for finalising the process flowsheet but are still required for the project development are as follows:

- Further dynamic thickening and cyanide detoxification test work;
- Pre-Leach and CIL test work;
- Oxygen uptake testing;
- Slurry rheology characterisation on process slurry at different densities;
- CIL cyanidation testing of all Variability composites under optimised conditions;
- Sequential CIP testing - Triple Carbon Contact/Carbon Kinetics test work on all Master composites.

Key findings from the test work are summarised in the following sub-sections.

1.8.1 Mineralogy

No native gold particles were detected, however gravity separation test work has confirmed the presence of gravity recoverable gold in all master composites.

Pyrite was detected reporting between 0.9% and 1.63% by mass in the fresh samples and measured less than 0.5% in the transition samples, and less than 0.1% in oxide samples.

The Nokpa & Han oxide composites and Chegue Main and Chegue South fresh composites reported elevated organic carbon levels at 0.42%, 0.78%, 0.32% and 0.24% respectively. All other composites reported low levels (<0.03% Corg). The high levels of organic carbon reported, upon testing, however, did not exhibit preg-robbing behaviour in the leach kinetic response.

Deleterious elements mercury, arsenic and antimony are low in the composites and should not present an environmental or occupational health risk in the elution or electrowinning circuit.

Relatively high carbonate content was detected in the eight fresh samples. Calcite was observed in all the fresh composites and, of the transition samples, Enioda was the only composite containing a significant proportion of carbonate, largely as 'ankerite-dolomite'.

1.8.2 Comminution

The fresh ore has medium competency, moderate to high grinding energy requirements and is moderately abrasive. Transition ore has moderate competency with average grinding energy requirements and is moderately abrasive. The oxide ore has low to moderate competency with average to high grinding energy requirements and is low to moderately abrasive.

Crushing work indices (BCWi) results ranged from 1.6 to 15.8 kWh/t, averaging 8.3 kWh/t over the complete set of samples. The BCWi are low to medium, indicating a low crushing energy requirement. The maximum BCWi values, ranging from 4.1 to 15.8 kWh/t were for the Kekeda ores, with an 85th percentile value of 14.9 kWh/t.

Rod mill work indices (BRWi) results ranged from 7.2 to 21.4 kWh/t. A BRWi of 18.5 kWh/t for fresh and 14.9 kWh/t for oxide were selected for design.

The Bond Ball Mill Indices (BBWi) results ranged from 11 to 22.2 kWh/t. The BBWi are medium to high indicating a high grinding energy requirement. Selected 85th percentile data for design were 20.4 kWh/t for oxide and 18.5 kWh/t for fresh.

The abrasion indices ranged from very low at 0.018 to medium at 0.38. The oxide samples demonstrated the highest abrasion indices, averaging 0.248. Selected 85th percentile data for design were 0.248 for oxide and 0.20 for fresh ores.

1.8.3 *Cyanidation*

The grind versus recovery leach tests have confirmed the optimum target grind sizes to be consistent with those selected in the PFS test work programme. P_{80} 106 µm for oxide and transitional ores and P_{80} 75 µm for fresh ores.

Gold recoveries for the FS test work were largely consistent compared to the PFS test work. The three lithology types reported the following average gold recoveries across the eight (8) open pits: oxide - 96.2%, transitional - 93.0% and fresh - 87.9%.

The gravity recoverable gold component averaged at 31% across all open pits and was consistent with levels reported in the PFS test programme.

The results of the FS CIL leach tests were consistent with the PFS tests and confirmed the fresh ores require an extended leach residence time of 12 hours over and above the 24 hour leach residence time required by the oxide and transitional ores to maximise gold recovery.

Cyanide consumption rates reported in the FS CIL tests averaged at 0.57 kg/t which were significantly higher than reported in the PFS direct cyanide (no carbon) leach tests at 0.12 kg/t. Higher cyanide consumption rates are generally reported when carbon is used in the leach test.

Lime consumption in the FS CIL tests were reported in similar ranges to the PFS direct cyanide leaches. In both test programmes the lime requirements were highest in the oxide ores (~1.5 kg/t) followed by transitional (~0.60 kg/t) and lowest in the fresh ores (~0.30 kg/t).

Cyanide addition rates were optimised at 0.05% sodium cyanide.

Oxygen demand has been observed in the low range.

Oxygen sparging reported an increase in gold recoveries between 3 to 10% when processing fresh ores compared to air only sparging. Significant improvements were also reported using oxygen in the leaching of transitional ores at between 2 to 4% and to a lesser extent the oxide ores.

The addition of lead nitrate at 500 g/t proved to have little effect on gold recovery.

A six (6) hour pre-oxidation (no cyanide) step identified in the PFS report as a potential benefit to overall recovery proved to have little benefit with the FS conventional CIL test work. This was further demonstrated during the MACH© shear reactor tests which reported marginal and, in some cases inconsistent results.

Further tests were underway at the time of writing this report to assess the effect of a pre-leach step in advance of conventional CIL circuit on gold recovery.

Gold adsorption kinetics are noted to be in the medium range based on an average Fleming (K) value of 160 $hour^{-1}$ relative to industry typical values in the range 150 $hour^{-1}$ to 250 $hour^{-1}$, and intercept 'n' at an average of 0.70.

Elevated levels of tellurium (Te) were observed in some of the composites and were not confined to any particular lithology type. The leach test results have confirmed the presence of tellurides has no measurable impact on gold recovery.

Deleterious elements mercury, arsenic and antimony were low in the composites and should not present an environmental or occupational health risk in the elution or electrowinning circuits.

1.8.4 Thickening and Cyanide Detoxification

Dynamic thickening test work was conducted on the lithology master composite samples from Souwa and Nokpa deposits. Solids loadings were reported in the 0.75 $t/m^2/h$ to 1.0 $t/m^2/h$ range. Underflow slurry densities at or above 55% solids were achieved, with the exception of the Nokpa Transitional sample reporting at 48% solids which also required 300 g/t of coagulant to meet the <100 mg/L total suspended solids overflow clarity guidelines.

The Souwa oxide master composite dynamic thickener test achieved an underflow slurry density of 55% solids, however required high flocculant dose rates at 120 g/t and reported poor overflow clarity in excess of 1,000 mg/L.

Cyanide detoxification tests were conducted on the bulk Souwa thickener test samples and reported residence times in the range 80-95 mins from the three lithology types. Tests conducted on the Nokpa transitional and fresh composites however, reported contrasting residence time requirements at 50 minutes and 150 minutes respectively.

1.8.5 Metallurgical Recovery

Samples reported gold head grades via fire assay in a range from 0.74 g/t to 3.70 g/t Au and averaged 1.63 g/t Au. The duplicate gold assays for several composites varied significantly which is consistent with the PFS master composites supporting the presence of coarse gold.

Gold recoveries were largely consistent with those reported in the PFS at an average of 96.2% for oxide, 93% for transition and 87.9% for fresh ores.

Gold dissolution kinetics were consistent with the rates reported in the PFS, with the oxide and transitional ores leaching effectively complete in approximately 24 hours, however, gold extraction for some fresh ores were not complete until 40 hours of leach time.

There were no elements, including sulphur, that displayed any relationship with gold recovery. A regression analysis was conducted on the leach results from all FS master composite leach tests and PFS leach test results to assess the relationship between leach feed grade on the final leach residue grade and extraction.

The regression format and coefficients derived from a regression analysis exhibited a high level of correlation across all lithology types. All leach results were analysed to produce three regression recovery curves, one for each of the three lithology types. These curves were then applied to the varying head grades of the mine schedule to derive overall recovery. The following Table 1.2 presents the predicted recoveries derived from the regression analysis versus reserve head grades.

	Unit	Souwa			Nokpa			Kilosegui			Kekeda		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC1	MC2	MC3	MC4	MC5	MC6	MC7	MC8	MC9	MC10	MC11	MC12
Head Grade	g/t	1.27	1.54	1.70	n/a	1.62	1.78	1.42*	1.08	1.08	1.00	1.11	1.20
Au Recovery	%	95.0	94.4	87.6	n/a	95.1	89.0	97.7	92.0	85.5	96.0	93.7	88.3

	Unit	Enioda			Han			Chegue Main			Chegue South		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC13	MC14	MC15	MC16	MC17	MC18	MC19	MC20	MC21	MC22	MC23	MC24
Head Grade	g/t	1.40	1.37	1.82	n/a	2.15	1.89	1.11	1.01	1.58	n/a	1.04	1.34
Au Recovery	%	96.8	93.9	92.7	n/a	92.2	88.8	95.7	92.7	89.7	n/a	89.9	81.9

Notes:

* *Denotes assay gold grade and not reserve gold grade*

n/a Denotes no oxide samples tested

Table 1.2 Gold Grade Versus Recovery (Regression Analysis)

Cyanide consuming elements, copper, zinc, lead and nickel and antimony were reported at similar ranges to the PFS composites and are not expected to impact on the circuit.

The silver head grades reported were consistent with the PFS composite head assays and reported low grades at or under the 2.0 g/t Ag detection limit. The Han (MC16) and Enioda (MC13) oxide composites were exceptions reporting elevated silver head grades at 12 g/t Ag and 6 g/t Ag respectively.

All Master composites reported varying grades of tellurium at an average 2.54 g/t. The presence of tellurides at these grades warranted monitoring during subsequent test work. However, there was no measurable correlation observed between Tellurium grades and gold cyanide solubility and therefore is not expected to be a significant metallurgical issue.

The oxide and transition composites reported very low or below detection sulphide sulphur assays. Fresh composites reported sulphide sulphur levels in a range of 0.24% - 0.62%.

1.9 Mineral Resource Estimate

This Doropo updated Mineral Resource Estimate (MRE) has an effective date of 31 October 2023, with drill hole data cut-off dates from 22 May 2023 to 27 August 2023. The MRE has been reported in accordance with the CIM Definition Standards (CIM Council, 2014).

The Doropo Gold Project comprises sixteen prospects, namely Attire, Enioda, Chegue Main, Chegue South, Han, Hinda, Hinda South, Kekeda, Kilosegui, Nare, Nokpa, Sanboyoro, Solo, Souwa, Tchouahinin, and Vako. All of these prospects were updated in this study following the collection of the additional FS phase drill data. The primary objective of the infill drilling was to raise the Mineral Resource confidence classification rating of the prospects for input into the FS.

Of the sixteen prospects listed, eight prospects - Attire (ATI), Enioda (ENI), Chegue Main (CHG), Chegue South (CHS), Han (HAN), Kekeda (KEK), Kilosegui (KLG), Nokpa (NOK), and Souwa (SWA) were previously updated in 2022 for the PFS. The remainder were either last updated in 2020 or estimated for the first time in this update.

Previously, a cut-off grade of 0.5 g/t Au was used to report Mineral Resources. However, the PFS demonstrated that a proportion of the oxide Mineral Resources were economically viable at grades lower than 0.5 g/t au. As a result, the Mineral Resource reporting cut-off grade for this update has been lowered to 0.3 g/t.

Cube made use of Geoaccess Professional, Supervisor, Leapfrog Geo, Surpac and Isatis v2018.5 software to undertake the MRE update.

The updated Doropo MRE, as constrained to the optimised pit shells, is summarised in Table 1.3 and Table 1.1. Grade-tonnage curves per updated prospect are presented in Part 14. The tonnage and grade curves show a relatively slow change in both these parameters for all prospects except CHS and KEK, which display steeper transitions through the range of cut-offs considered.

Mineral Resources (0.3 g/t Au COG)			
Category	Mt	Au g/t	Au Moz
Measured	1.51	1.60	0.077
Indicated	75.34	1.25	3.027
Measured + Indicated	76.85	1.85	3.10
Inferred	7.37	1.23	0.292

Notes:

- *Some numerical differences may occur due to rounding;*
- *RPEEE is defined by optimised pit shells based on a gold price of US$2,000/oz;*
- *Reported at a gold grade cut-off of 0.3 g/t Au;*
- *Includes drill holes up to and including 27 August 2023;*
- *Measured and Indicated categories includes Mineral Reserves, no Mineral Reserves included for Inferred category.*

Table 1.3 Doropo Updated Mineral Resource Estimate (0.3 g/t Au COG) (CIM Definition Standards), 31 October 2023

No mining has taken place at the Project to date and hence no depletion is recorded in the block model. Artisanal workings do occur in the area, but these have had, at most, a negligible impact on the modelled Mineral Resource.

1.9.1 Mineral Resource Estimate Validation

Mineral Resource estimate validation has been undertaken by the following means:

- Global and local statistical comparisons of the mean grade of the estimated blocks to the declustered mean grade of the composites. The results of Inverse distance squared and moving window average (MWA) check estimates were also considered. The global mean of estimated gold grades were found to match the informing composite data satisfactorily. Any larger variances could be readily explained by extrapolation of sample grades into relatively poorly informed volumes, such as around the periphery of the mineralised domains. Local estimated informed cells mean was found to match the MWA estimates satisfactorily as well;
- Using swath plots to compare estimated block grades to the informing composite grades;
- By visual validation, both in cross-section and 3D isometric views, of the estimated block grades overlaid on drill assay data.

The block estimates were considered to be an accurate reflection of the input sample data.

1.9.2 Factors that May Affect the Mineral Resource

The drilling, sampling and analytical methods utilised by Centamin are considered appropriate for Mineral Resource modelling and the input data have been found to be of sufficient quality.

Most of the Doropo prospects remain open, predominantly at depth. Additional drilling would be required to fully delineate the mineralisation extents.

The estimation method employed is considered to be appropriate for open pit mining studies.

There are no current known environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors that could materially impact the MRE.

1.10 Mineral Reserve Estimate

The Mineral Reserves were estimated for the Doropo Gold Project as part of this FS by Orelogy. The total Proven and Probable Mineral Reserve is estimated at 38.2 Mt at 1.53 g/t Au with a contained gold content of 1,876 koz.

The Mineral Reserve for the Project is reported according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The Mineral Resource was converted by applying Modifying Factors (refer Table 1.4).

The modifying factors used to develop the cut-off grade were those available at the time of the life of mine (LOM) production scheduling and are detailed in Table 1.4.

<table>
<tr><th colspan="2">Item</th><th>Unit</th><th>Value</th></tr>
<tr><td rowspan="4">Ore Related Mining Costs</td><td>ROM Rehandle</td><td>$US/dmt</td><td>$0.62</td></tr>
<tr><td>Grade Control</td><td>$US/dmt</td><td>$0.50</td></tr>
<tr><td rowspan="2">Owner's Team</td><td>$US/dmt weathered</td><td>$0.63</td></tr>
<tr><td>$US/dmt fresh</td><td>$0.85</td></tr>
<tr><td colspan="2" rowspan="3">Processing Cost</td><td>$US/dmt oxide</td><td>$11.43</td></tr>
<tr><td>$US/dmt trans</td><td>$11.26</td></tr>
<tr><td>$US/dmt fresh</td><td>$14.35</td></tr>
<tr><td colspan="2" rowspan="3">G & A Cost</td><td>$US/dmt oxide</td><td>$3.81</td></tr>
<tr><td>$US/dmt trans</td><td>$3.81</td></tr>
<tr><td>$US/dmt fresh</td><td>$5.14</td></tr>
<tr><td rowspan="4">Ore Haulage</td><td>Enioda</td><td>$US/dmt</td><td>$2.95</td></tr>
<tr><td>Han</td><td>$US/dmt</td><td>$2.50</td></tr>
<tr><td>Kekeda</td><td>$US/dmt</td><td>$2.13</td></tr>
<tr><td>Kilosegui</td><td>$US/dmt</td><td>$5.50</td></tr>
<tr><td rowspan="2">Process Recovery</td><td>Weathered</td><td>%</td><td rowspan="2">Refer to Table 16.13 (Part 16)</td></tr>
<tr><td>Fresh</td><td>%</td></tr>
<tr><td rowspan="8">Gold Price</td><td>Base Gold Price</td><td>$US/oz</td><td>$1,450.00</td></tr>
<tr><td>Govt. Royalty</td><td>%</td><td>4.00%</td></tr>
<tr><td>Tenement Royalty</td><td>%</td><td>0.50%</td></tr>
<tr><td>Social Fund Royalty</td><td>%</td><td>0.50%</td></tr>
<tr><td>Gold Loss</td><td>%</td><td>0.05%</td></tr>
<tr><td>Charge</td><td>$US/oz</td><td>$4.00</td></tr>
<tr><td rowspan="2">Net Gold Price</td><td>$US/oz</td><td>$1,372.78</td></tr>
<tr><td>$US/gram</td><td>$44.14</td></tr>
</table>

Table 1.4 Breakeven Cut-Off Grade - Modifying Factors

As costs and process recovery vary by weathering and/or location, the calculated cut-off grade also varies as detailed in Table 1.5.

Mining Area	Unit	Oxide	Trans	Fresh
Souwa	g/t	0.43	0.45	0.62
Kilosegui	g/t	0.56	0.58	0.77
Nokpa	g/t	1.00[1]	0.42	0.62
Chegue Main	g/t	0.43	0.45	0.62
Cheque South	g/t	N/A	0.45	0.62
Kekeda	g/t	0.49	0.55	0.68
Han	g/t	N/A	0.50	0.69
Enioda	g/t	0.52	0.57	0.70

[1] Nokpa oxide only has recovery for >1.0 g/t material (refer to Table 16.13, in Part 16)

Table 1.5 Breakeven Cut-Off Grades

The Proven Mineral Reserve estimate is based on the Mineral Resource classified as Measured. The Probable Mineral Reserve estimate is based on the Mineral Resource classified as Indicated. A summary of the Mineral Reserves on a 100% Project basis at a gold price of US$1,450/oz is presented in Table 1.6.

Classification	Mt	Grade	Contained Ounces
Proven	1.3	1.73	70,100
Probable	37.0	1.52	1,805,559
Total	38.2	1.53	1,875,659

Notes:

- *The Mineral Reserve conforms with and uses the CIM (2014) definitions.*
- *The Mineral Reserve was evaluated using a gold price of US$1,450 per ounce.*
- *The Mineral Reserve was evaluated using variable cut-off grades as described in Table 1.5.*
- *Block grade and tonnage dilution was incorporated into the model.*
- *All figures are rounded to reflect appropriate levels of confidence.*
- *Apparent differences may occur due to rounding.*

Table 1.6 Doropo Gold Project FS Mineral Reserve Estimate

All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur as a result of rounding. Based on the information presented in this FS, the Mineral Reserve estimation process has converted approximately 85% of the Measured Mineral Resources to Proven Mineral Reserve and 50% of the Indicated Mineral Resources to Probable Mineral Reserves.

1.11 Mining Methods

1.11.1 Mining Strategy

The Doropo Project will use a conventional open pit operation utilising a conventional truck and excavator approach undertaking a typical drill, blast, load and haul production cycle. Mining is based on all production related activities being undertaken by a suitably qualified and experienced mining contractor. These activities include, but are not limited to:

- Construction of all mining infrastructure required for undertaking mining operations such as workshops, warehousing, fuel and lubricants area, washdown facilities, administration building, crib rooms and ablutions;
- Construction of all mine haul roads and ore haulage roads from the satellite pits to the processing plant;
- Site preparation for pits, waste rock dumps (WRD's), and roads (i.e., clearing and grubbing of vegetation, removal and stockpiling of topsoil material);
- Primary production drilling and blasting;
- Primary production loading and hauling;
- Support of primary production activities with a suitable sized fleet of support equipment (i.e., bulldozers, graders, and water trucks);
- Pit-dewatering via in pit sump pumps;
- Establishment and maintenance of suitable surface water management infrastructure;
- Site rehabilitation and closure works such as re-profiling WRDs to final landform, rehandling and spreading stockpiled topsoil material.

Centamin will provide oversight and management of the mining contractors and undertake all technical requirements for the mining operation such as grade control, mine planning, and mine surveying.

1.11.2 Geotechnical Considerations

The FS geotechnical evaluation for the open pits was undertaken by SRK.

As part of the feasibility assessment, a total of 8 holes of approximately 820 m total length were completed. These drillholes were designed for two purposes:

1. To infill geotechnical drilling and inform the geotechnical model; and
2. To collect samples for laboratory testing.

These are in addition to 33 drillholes completed for the 2023 PFS and 23 drillholes completed for the Scoping Study in 2018.

Modelling

Open pit stability will be controlled by shear strength of the rock mass within the weathered horizons and structurally controlled within the fresh rock. The geotechnical domain model considers the weathering model from a rock mass failure perspective. The objective of the geotechnical domain model is to provide a representation of the variability of the rock properties across the deposit area. The rock mass is variable by nature, and as such, a simplification of the parameters with similar units is required for analysis and design purposes.

Lithological Domains

Based on the regional geology context, the deposits have been sub-domained into three major areas: Doropo, Kilosegui, and Enioda (Figure 5.1). For the Doropo area, laboratory testing was completed at Chegue and Souwa. The lithologies considered for the geotechnical domain model include granodiorite, the dolerite dykes in Chegue South and Nokpa, and the mafic units at Enioda. Each domain was individually assessed for rock mass quality and rock strength as part of the geotechnical characterization work.

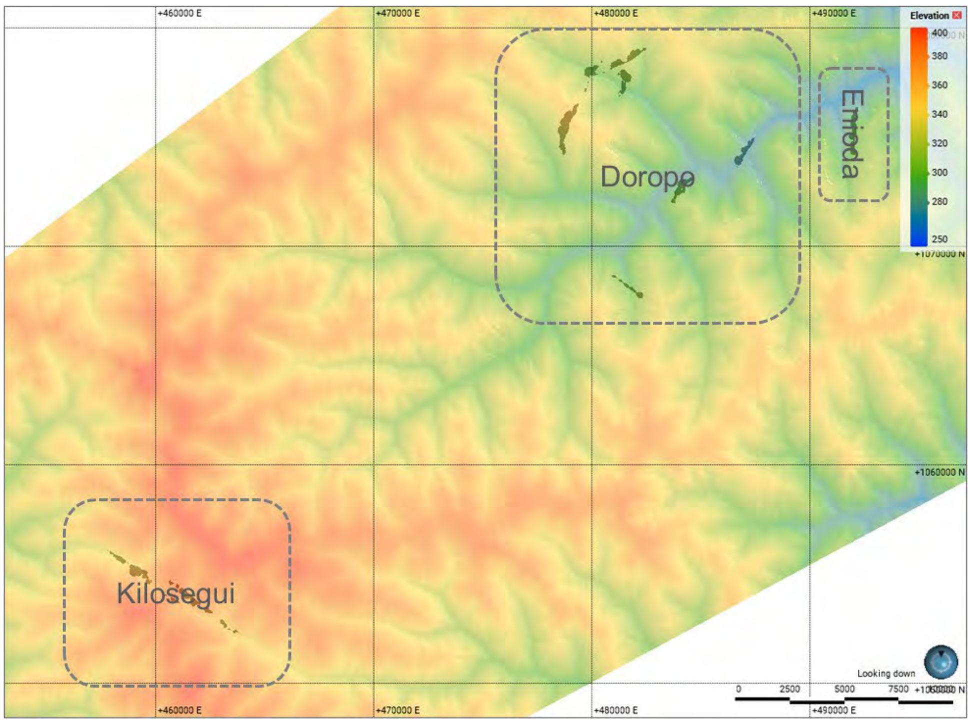


Figure 1.2 Major lithological domains

Weathering Domains

The geotechnical model primarily comprises the weathering domains modelled for each pit. There are six modelled weathering horizons (refer to Figure 1.3). These have been modelled into three distinct weathering domains for design purposes as described in Table 1.7.

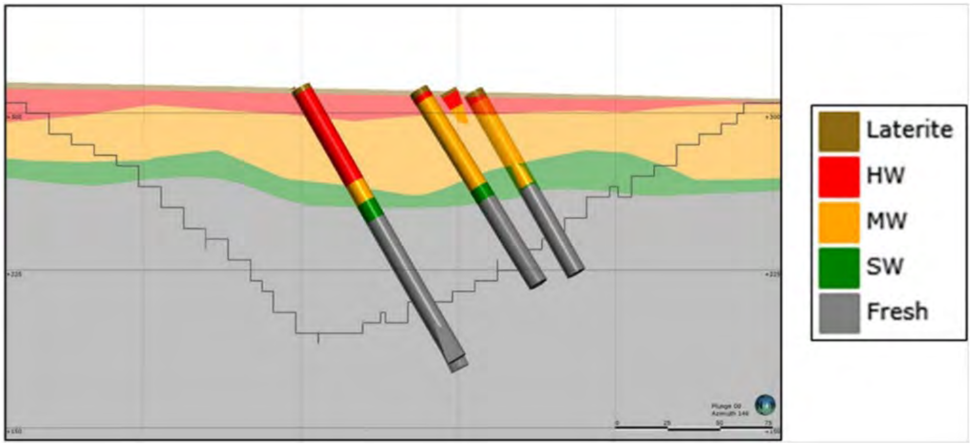


Figure 1.3 Geological Modelling of Weathering Zones (Source: SRK, 2024)

Weathering Horizons	Weathering Domain	Description	Rock Strength	Fracture Frequency
Completely Weathered (CW)	Saprolite/ Laterite	No constituent minerals are recognisable, except in part quartz grain (in the case of Doropo).	R0	40
Highly Weathered (HW)		Unconsolidated, some constituent minerals are visible including quartz which is abundant and some rare feldspars	R0 - R1	40
Moderately Weathered (MW)	Saprock	Consolidated although crumbly, only few minerals are recognisable, quartz and feldspars are little or no altered, but micas are almost completely altered (all rusty).	R0 - R2	20-40
Slightly Weathered (SW)		Consolidated, most of the minerals are recognisable, quartz and feldspars are not altered, but micas are partially altered (rusty).	R3 - R5	0 - 5
Fresh (FR)	Fresh	None of the constituents are altered.	R3 - R5	0 - 5

Table 1.7 Description of Geological Weathering Domains

Deposit-scale structures typically emulate regional-scale structures. Structural data, such as the orientation of faults, fractures and joints in drill holes, geotechnical logging data (RQD/recoveries) and gold grade interpolants, allow for the modelling of mineralised structures and structural discontinuities in the deposit, that can be linked to shear zones, faults, and cross-cutting dykes.

The orientations of the deposit-scale mineralised structures, interpreted as brittle-ductile shear zones and veins in the Doropo deposits, show variable trends from NE-SW, NW-SE, E-W to N-S. The dominant trends in Chegue, Enioda, Han, Kekeda, Nokpa and Souwa is N-S and NE-SW, whereas Kilosegui and Attire show NW-SE and WNW-ESE trends.

Slope Stability Assessment

The bench and inter-ramp design for the pits were based on the following considerations:

- Saprolite stability will be controlled by the slope angles and influence of groundwater; and
- Slopes within the saprock and fresh rock domains will be structurally controlled, and the achievable slope geometries will be driven by the bench scale kinematics.

Two-dimensional limit equilibrium (LE) slope stability analysis was used to evaluate the expected rock and saprolite slope stability conditions. The LE analysis was conducted using Slide2D™. The stability analysis considered the potential for overall non-circular failures through the various units. Stability modelling was focused on evaluating the designs in the saprolite and saprock domains, as these will be the most sensitive to groundwater.

Saprolite Design

Bench face design philosophy in saprolites represents a compromise between minimising erosion from surface water run-off (steeper bench faces more favourable) and maintaining stability (where shallower bench faces are more stable). It is assumed that benches in saprolite will not require blasting (i.e. “free dig”) and can be cut back to the desired bench face angle (BFA).

The design of benches and inter-ramps in saprolite will depend on the overall thickness of the saprolite horizons. This will be managed by increasing bench widths to reduce the Inter-Ramp Angle (IRA) as the thickness of the weathered domain increases. Bench face angles for the saprolite will be set at 60° to promote drainage of the faces. Berms should be graded at 2% to 3% into the pit to encourage sheet flow and surface water run-off, thus reducing erosion. The design recommendations for the saprolite domain are summarised in Table 1.8. Kilosegui and Enioda have their own slope geometries due to variations in thickness and orientation.

Domain	Thickness	Face Height	Berm Width	BFA	IRA	Stack Height	Factor of Safety	
	(m)	(m)	(m)	(°)	(°)	(m)	Hu = 0.8	Hu = 0.6
Doropo	< 20 m	5	4	60	36	20	1.6	1.6
	< 40 m	5	5	60	32.4	20	1.26	1.3
Enioda	All	5	5.5	60	30.8	20	1.3	1.36
Kilosegui	All	5	4.5	60	34.1	20	1.4	1.37

Table 1.8 Saprolite (CW & HW) Design Recommendations

Mining in the saprolite will be unpredictable at times, with local changes in material properties, groundwater conditions, or relict structures that may cause instabilities. Inter-ramp stack heights should be limited to 20 m, with a 15 m geotechnical berm placed between stacks. The base of the saprolite should also have a 15 m geotechnical berm before entering the saprock and fresh rock domains.

Saprock and Fresh Rock Design

The structural data was grouped into structural domains based on similar identified joint sets, with the Doropo area being split into two domains defined by a large, NW striking dolerite dyke:

- North (comprising Chegue Main, Chegue South and Han);
- South (comprising Souwa, Nokpa and Kekeda).

Enioda and Kilosegui remain independent domains, reflecting the different geological context of the deposit.

Bench design for the saprock and fresh rock domains is driven by kinematic instabilities and increasing bench widths to maintain suitable catchment. The achievable inter-ramp angles are a function of the bench geometries. The contact between the saprock and fresh rock is expected to be variable in terms of depths around the pits, and some flexibility will be required during implementation. The saprock designs will be limited to a bench height of 10 m to mitigate potential kinematic failures exacerbated by weathered fractures. The fresh rock will optimise to a 20 m bench height based on improved geotechnical conditions. The design recommendations for the saprock and fresh rock domain are summarised in Table 1.9.

No geotechnical berms are required for the current design configurations in the saprock and fresh rock domains, as the pit depths within these units are less than a practical geotechnical berm spacing of 120 m. However, a geotechnical berm of 15m is required at the top of the saprock, at the base of saprolite.

Domain	Weathering Domain	Azimuth From (°)	Azimuth To (°)	Bench Height (m)	Bench Width (m)	BFA (°)	IRA (°)	Stack Height (m)
Enioda	Saprock (MW/SW)	140	220	10	10	75	38.3	120
		220	140	10	6.5	75	47.4	120
	Fresh	100	140	20	10	75	52.5	120
		140	220	20	12	75	49	120
		220	100	20	8.5	75	55.3	120
Chegue Main/Chegue South/Han	Saprock (MW/SW)	40	320	10	6.5	75	47.4	120
		320	40	10	10	75	38.3	120
	Fresh	60	320	20	8.5	75	55.3	120
		320	60	20	13	75	47.4	120

Domain	Weathering Domain	Azimuth From (°)	Azimuth To (°)	Bench Height (m)	Bench Width (m)	BFA (°)	IRA (°)	Stack Height (m)
Souwa/Nokpa/Kekeda	Saprock (MW/SW)	20	300	10	6.5	75	47.4	120
		300	20	10	8	75	43.1	120
	Fresh	20	300	20	8.5	75	55.3	120
		300	20	20	11	75	50.7	120
Kilosegui	Saprock (MW/SW)	100	340	10	6.5	75	47.4	120
		340	100	10	8	75	43.1	120
	Fresh	0	60	20	12	75	49	120
		60	120	20	10	75	52.5	120
		120	360	20	8.5	75	55.3	120

Table 1.9 Saprock and Fresh Rock (MW, SW & FR) Design Recommendations

Operational Considerations

The slope design guidelines are presented to global best-practice design standards.

Operational recommendations in regard to final wall slopes include:

- Saprolite cut slopes and partially blasted saprock slopes should be excavated utilising batter boards and guidance from the survey team to ensure design compliance;
- Convex slopes of one or more stack heights should be minimised in final designs to minimise the effects of unconfinement;
- Pits designed with single-ramp access should trigger a review of geotechnical hazards, slope and bench performance, and evaluation of possible risks to access.

Opportunities remain during mining to optimise the bench heights and berm widths. Specifically, the requirements for additional catchment in certain aspects of the design domains may be reduced should design compliance be maintained during implementation.

1.11.3 Mining Equipment

Mining of both ore and waste will be undertaken using an excavator (backhoe configuration) to load rigid body dump trucks. Drill and blast will be carried out on 5 m bench heights. It was assumed that 20% of the highly weathered oxide material will be free-dig (i.e. not require blasting) with a small amount of dozer ripping. The ore will be excavated in 2.5 m flitches to minimise dilution and ore loss. Waste will be mined in either 2.5 m flitches or at the 5 m blast height depending on the size of the excavator. The configuration for the mining fleet of the contractor submission on which the FS is based consists of:

- Up to 2 x 150-tonne class excavators;

- Up to 2 x 300-tonne class excavators;
- Up to 24 x 90-tonne payload rigid bodied dump trucks.

Waste will be placed in designated WRD's adjacent to the open pits. Ore mined at the hub deposits will be transported directly to the central processing plant with the mining dump trucks. In the case of the satellite deposits, ore will be dumped to a Run-of-Mine (ROM) stockpile area immediately adjacent to the pit ramp exit for each satellite pit. From here it will be rehandled by a front end-loader to road type tipper trucks for transport to the process plant.

Ore arriving at the process plant from the hub pits will be either:

- Direct tipped into the primary crusher bin (80% of plant feed); or
- Placed on stockpile fingers on top of the primary crusher ROM pad (15% of plant feed); or
- Dumped to a low-grade stockpile (5% of plant feed) on natural ground level adjacent to the primary crusher ROM pad (designated stockpile material).

All ore arriving at the process plant from the satellite pits will be placed in a "satellite ore stockpile" on natural ground level adjacent to the low-grade stockpile.

It is assumed that all ore from the satellite ore and low-grade stockpiles will be reclaimed by front-end loader (or excavator) and hauled by mine trucks to the ROM Pad, where it will be direct tipped into the primary crusher bin.

Mining is scheduled such that the hub pits and up to two satellite pits can be in production concurrently.

Rehabilitation works will be carried out on an ongoing basis as areas of WRD's are finalised or infrastructure such as roads, stockpiles etc. are no longer required.

No consideration has been made for underground extensions of the operation in this FS.

1.11.4 Mining Schedule

The Life of Mine (LOM) production schedule was developed based on the following periods:

- 1 x quarter (pre-production);
- 24 x monthly periods (i.e. 2 years);
- 12 x quarterly periods (i.e. 3 years);
- Annual periods thereafter.

The schedule assumes mining contractor mobilisation during the two quarters before plant production commences, with initial stripping, clearing and pre-strip mining occurring in the quarter preceding plant production.

A processing plant feed rate of 5.4 Mt/a for weathered (oxide and transitional) material and 4.0 Mt/a for fresh material was utilised throughout the schedule. The schedule assumed 6,000 operating hrs per year for the crusher and converted the weathered and fresh throughput rates into an hourly rate of 900 t/h and 667 t/h respectively.

A process plant ramp up was also utilised, reaching 100% of nominal throughput three months after first ore. There were no requirements for feed blending to the process plant.

The LOM Production Schedule was undertaken as a Total Material Movement (TMM) schedule, which assumes the loading unit utilisation is the constraint and that sufficient trucking is available to “service” the loader. Effective equipment management will be necessary to ensure that the truck fleet size is maintained at a reasonable level.

The digging fleet for the LOM production schedule was based on the contractor submissions to the request for budget pricing and assumed a combination of 150-tonne (Cat 6015 equivalent) and 200-tonne (Cat 6020 equivalent) excavators matched to a 92-tonne payload truck (Cat 777 equivalent).

The LOM Production Schedule results are present in Figure 1.4 to Figure 1.6. The physicals are then presented by quarters and years in Table 1.10 and Table 1.11.

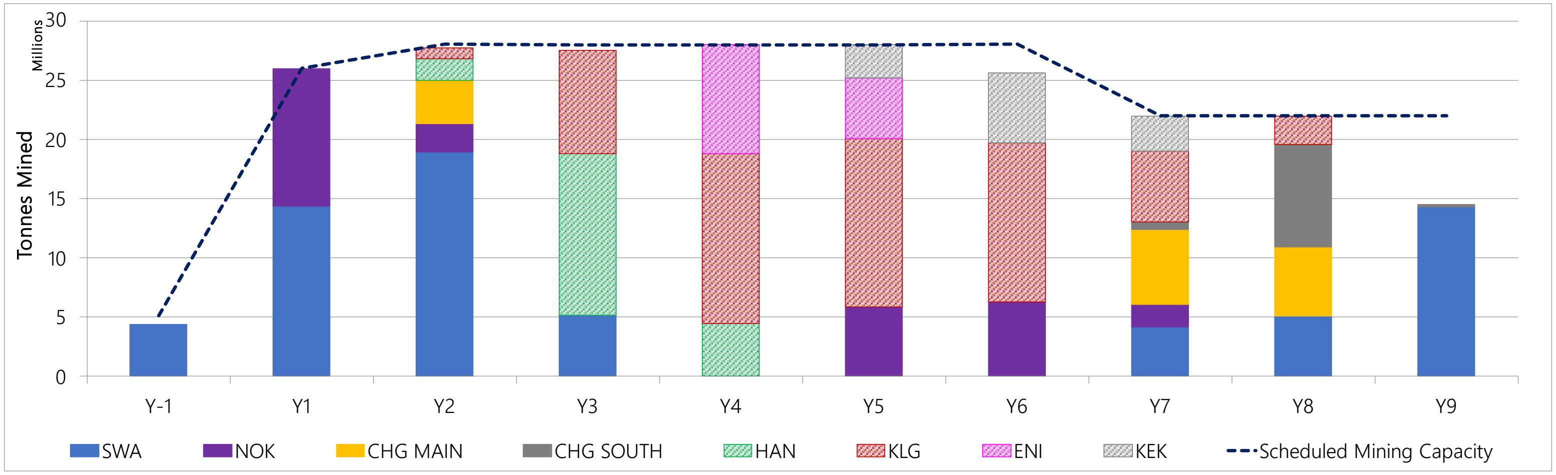


Figure 1.4 LOM Total Mining by Mining Area - Annual (Source: Orelogy, 2024)

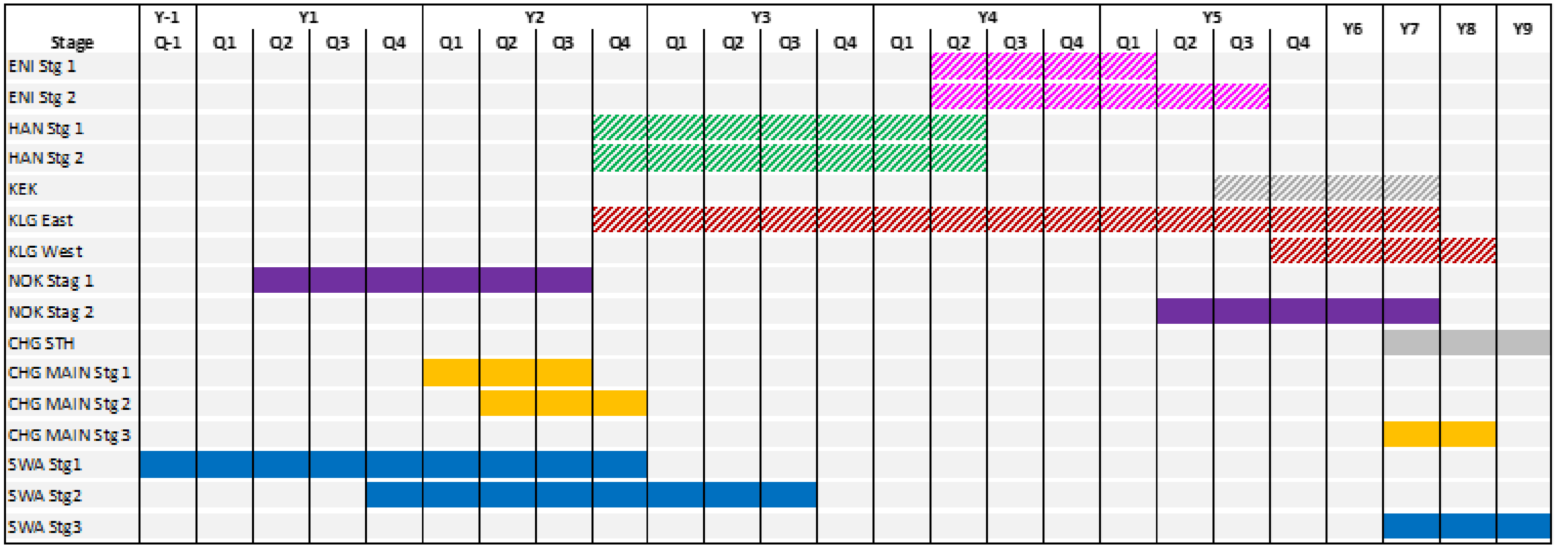


Figure 1.5 Mining Sequence by Pit Stage (Source: Orelogy, 2024)

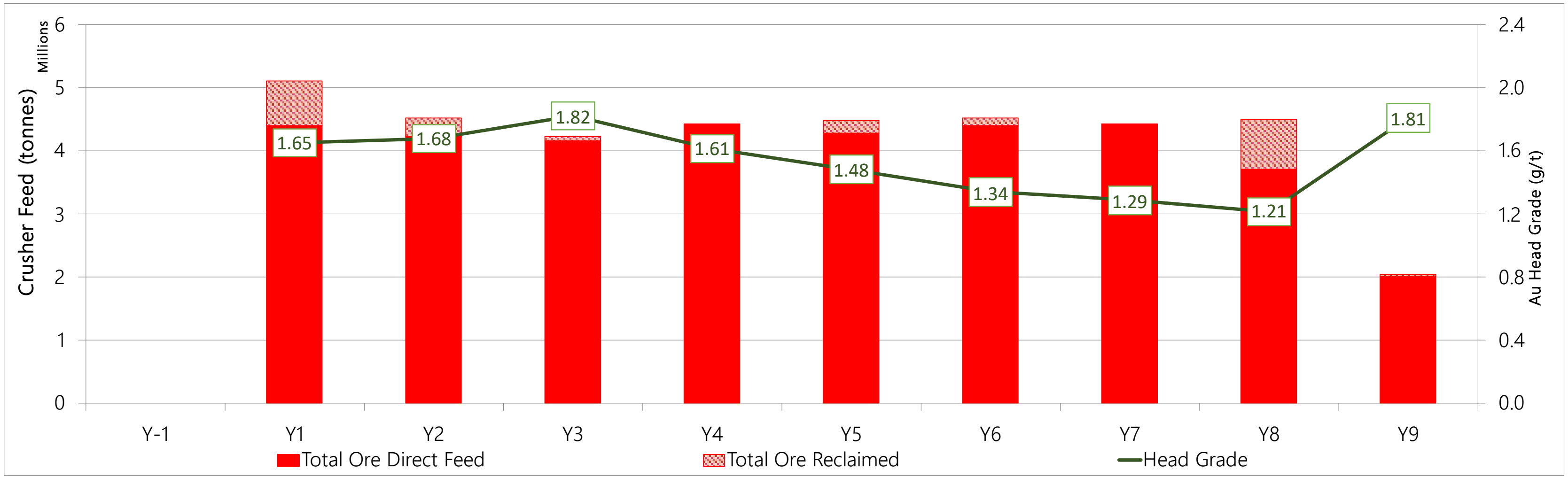


Figure 1.6 Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade (Source: Orelogy, 2024)

Items	Units	Total	Y-1	Y1				Y2				Y3			
			Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Total Material Mined	Mt	225.8	4.4	5.0	6.9	7.0	7.1	7.1	7.0	6.7	7.1	7.1	6.9	6.5	7.1
Total Material Moved	Mt	228.0	4.4	5.1	7.0	7.5	7.1	7.1	7.0	6.9	7.1	7.1	6.9	6.5	7.1
Ore Mined	Mt	38.2	0.80	1.18	1.28	0.85	1.39	1.19	1.18	1.09	1.15	1.06	1.08	1.05	1.04
	g/t	1.53	1.25	1.72	1.94	1.53	1.51	1.63	1.37	1.58	1.97	1.73	1.72	2.13	1.67
Waste Mined	Mt	187.6	3.6	3.9	5.6	6.1	5.7	5.9	5.8	5.6	5.9	6.0	5.8	5.5	6.0
Strip Ratio	W:O	4.9	4.5	3.3	4.4	7.2	4.1	4.9	4.9	5.1	5.2	5.6	5.4	5.2	5.8
Direct Feed Ore	Mt	36.0		1.2	1.3	0.8	1.2	1.2	1.2	0.8	1.0	1.0	1.0	1.0	1.0
	g/t	1.55		1.7	2.0	1.6	1.5	1.6	1.4	1.8	2.0	1.7	1.8	2.1	1.7
Stockpiled Ore	Mt	2.2	0.8	0.0	0.0	0.1	0.2	0.0	0.0	0.2	0.1	0.0	0.0	0.0	
	g/t	1.18	1.25	0.00	0.82	0.79	1.57	0.67	0.68	0.91	1.33	1.00	1.00	1.02	
Reclaimed Ore	Mt	2.2		0.0	0.1	0.6	0.1		0.1	0.2					0.1
	g/t	1.18		1.90	1.51	1.26	1.09		1.78	1.33					1.31
Ore Processed	Mt	38.2	0.0	1.2	1.3	1.3	1.2	1.2	1.2	1.1	1.0	1.0	1.0	1.0	1.1
	g/t	1.53	0.00	1.72	1.93	1.46	1.48	1.64	1.40	1.68	2.05	1.74	1.75	2.14	1.65
	koz	1,876	0.0	66.3	83.4	62.4	59.1	62.4	55.0	58.7	67.9	58.7	58.6	71.2	58.9
	koz rec.	1,667	0.0	62.7	78.5	58.7	54.5	56.6	50.1	52.2	59.6	51.3	51.4	62.7	52.2
	% rec.	88.9%	0.0%	94.5%	94.1%	94.1%	92.3%	90.8%	91.0%	89.0%	87.8%	87.5%	87.6%	88.1%	88.8%
TMM (Mt)	ENI	14.4													
	HAN	19.9									1.8	3.2	3.4	3.5	3.6
	KEK	11.7													
	KLG	60.1									1.0	1.2	1.6	2.5	3.5
	NOK	28.0			3.6	4.5	3.5	1.6	0.6	0.1					
	CHG STH	9.5													
	CHG MAIN	15.8						1.5	0.9	1.2	0.1				
	SWA	66.4	4.4	5.0	3.3	2.5	3.5	3.9	5.5	5.4	4.2	2.7	1.9	0.6	

Table 1.10 LOM Production Schedule Physicals by Period (Pre-Production to Year 3)

Items	Units	Y4				Y5				Y6	Y7	Y8	Y9
		Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4				
Total Material Mined	Mt	7.1	6.9	7.0	7.1	7.1	6.9	7.0	7.1	25.6	22.0	22.0	14.5
Total Material Moved	Mt	7.1	6.9	7.0	7.1	7.1	6.9	7.1	7.1	25.8	22.0	22.8	14.5
Ore Mined	Mt	1.13	1.07	1.13	1.11	1.19	1.13	1.28	1.24	4.44	4.42	3.71	2.01
	g/t	1.57	1.88	1.47	1.52	1.46	1.40	1.39	1.55	1.34	1.28	1.26	1.82
Waste Mined	Mt	5.9	5.8	5.8	5.9	5.9	5.8	5.7	5.8	21.2	17.6	18.3	12.5
Strip Ratio	W:O	5.2	5.4	5.2	5.4	4.9	5.1	4.4	4.7	4.8	4.0	4.9	6.2
Direct Feed Ore	Mt	1.1	1.1	1.1	1.1	1.1	1.1	1.0	1.0	4.4	4.4	3.7	2.0
	g/t	1.6	1.9	1.5	1.5	1.5	1.4	1.5	1.6	1.3	1.3	1.3	1.8
Stockpiled Ore	Mt	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.2	0.0	0.0	0.0	
	g/t	0.89	1.14	1.07	0.88	1.43	0.98	1.11	1.20	0.86	0.49	0.62	
Reclaimed Ore	Mt			0.0				0.1	0.1	0.1		0.8	0.0
	g/t			1.43				1.38	1.05	1.06		0.98	0.66
Ore Processed	Mt	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	4.5	4.4	4.5	2.0
	g/t	1.58	1.88	1.48	1.52	1.46	1.42	1.46	1.59	1.34	1.29	1.21	1.81
	koz	56.2	64.7	54.3	54.2	53.8	49.3	53.8	56.1	194.4	182.6	175.4	118.3
	koz rec.	49.8	57.4	48.3	47.7	47.7	43.4	47.9	48.5	170.5	160.8	152.2	102.6
	% rec.	88.6%	88.8%	89.0%	88.1%	88.6%	88.0%	89.1%	86.5%	87.7%	88.0%	86.8%	86.7%
TMM (Mt)	ENI		2.0	3.6	3.6	3.4	1.4	0.4					
	HAN	3.3	1.2										
	KEK							1.2	1.7	5.9	3.0		
	KLG	3.8	3.7	3.4	3.4	3.7	3.7	3.4	3.4	13.5	6.0	2.4	
	NOK						1.9	2.0	2.0	6.2	1.9		
	CHG STH										0.6	8.7	0.2
	CHG MAIN										6.3	5.8	
	SWA										4.2	5.1	14.3

Table 1.11 LOM Production Schedule Physicals by Period (Year 4 to Year 9)

1.12 Recovery Methods

The key project and ore specific design criteria for the plant design are as follows:

- 4,000,000 t/y of primary ore (fresh) and 5,400,000 t/y of oxide/transition ore (higher due to volumetric process capacity i.e. not mill constrained);
- Primary crushing plant mechanical utilisation of 80% (7,008 h/y);
- Mechanical utilisation for the remainder of the plant of 91.3% (8,000 h/y) supported by crushed ore storage and standby equipment in critical areas;
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

The treatment plant design incorporates the following unit process operations:

- Primary crushing with a jaw crusher to produce a coarse crushed product;
- A live stockpile with a 12 hour capacity;
- A comminution circuit comprising a SAG mill in closed circuit with a pebble crusher and a ball mill in closed circuit with hydrocyclones to produce an 80% passing (P_{80}) 75 µm grind size on fresh ores and 106 µm on oxide and transition ores;
- The cyclone overflow slurry gravitates, via trash screening, to a pre-leach thickener;
- Gravity concentration and recovery of coarse gold from the milling circuit, with treatment of gravity concentrates by intensive sodium cyanide leaching;
- Trash screen to remove any trash before the pre-leach thickener;
- Pre-leach thickening of the trash screen underflow to increase the slurry density feeding the carbon in leach (CIL) circuit to minimise slurry tank volume requirements and reduce overall reagent consumption;
- Gold leaching in two dedicated leach tanks followed by six CIL adsorption tanks to provide 36 hours of leaching residence time when processing fresh ores at design plant throughput. Sodium cyanide solution will be added to the first leach tank slurry to start the gold leaching process in the presence of the elevated dissolved oxygen levels;
- An acid wash column to remove inorganic contaminants from the carbon with hydrochloric acid;
- Gold recovery via batch elution processes using the pressure Zadra methodology;
- Dedicated electrowinning circuits for the gravity and CIL process streams followed by gold smelting to produce doré;
- Carbon reactivation kiln to remove organic contaminants from the carbon with heat;
- Tailings treatment incorporating cyanide destruction to less than 30 ppm weak acid dissociable (WAD) cyanide using the INCO SO_2/air process (sodium metabisulphite/oxygen) in line with International Cyanide Management Code standards;
- Tailings pumping to the tailings storage facility (TSF);
- Reagent mixing, storage and distribution facilities.

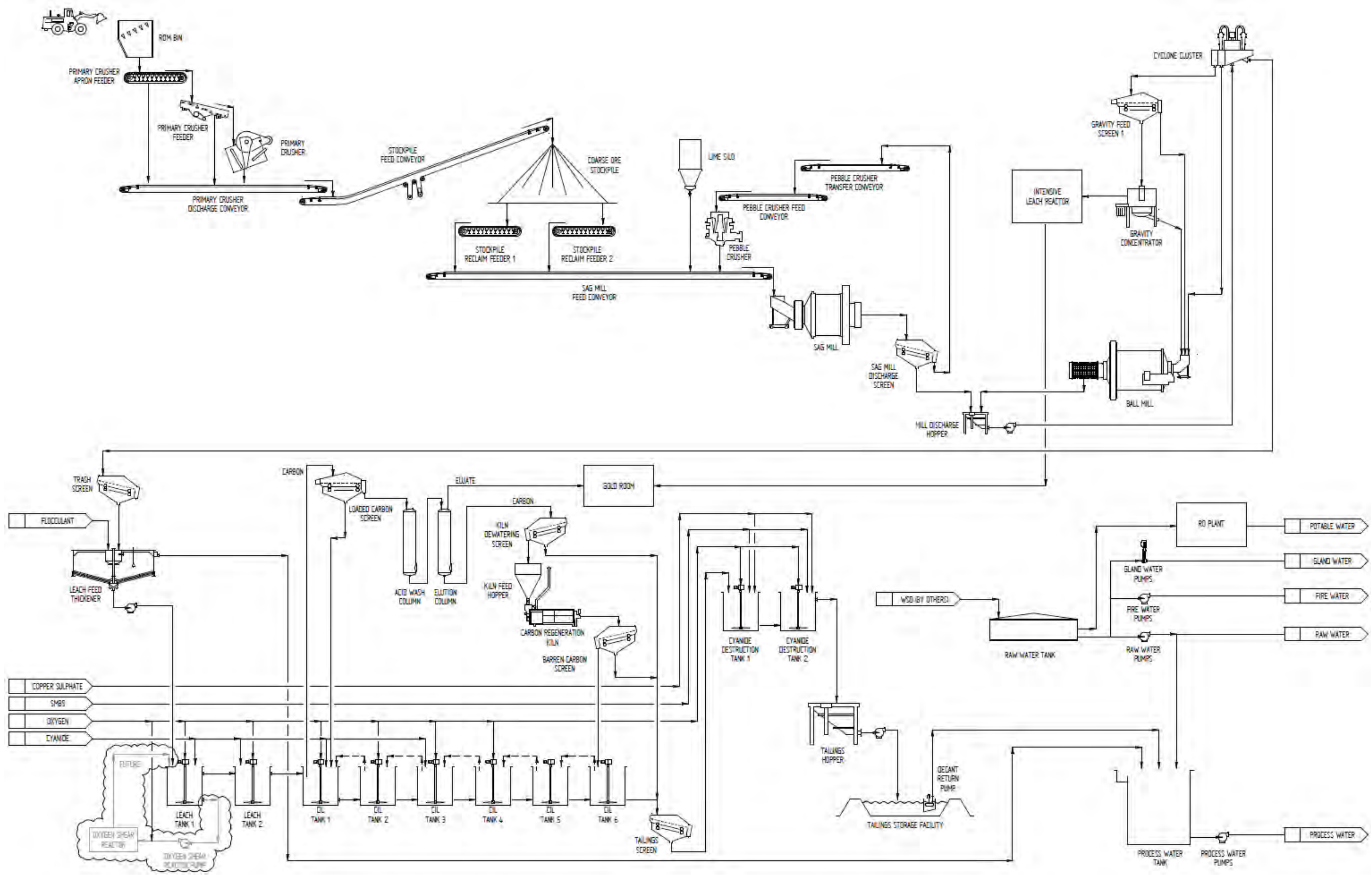


Figure 1.7 Doropo Simplified Process Flow Diagram (Source: GRES, 2024)

1.13 Project Infrastructure

Project facilities and infrastructure that support the operations will include:

- Access:
  - A site access road from the existing national road network to the mine site;
  - Haul roads from mine pits to the plant and waste storage areas;
  - Internal access roads in an around the plant and infrastructure areas; and
  - An airstrip;
- Water supply:
  - A WHD for harvesting surface water;
  - A WSD;
  - Surface water management and sediment control structures;
  - A potable water supply and waste water treatment;
- Power supply and distribution:
  - Connection to the Côte d'Ivoire grid, via a 90 kV transmission line from Bouna;
  - A mine site substation, including 90/11 kV transformer;
  - 11 kV power distribution from the mine site substation to the plant and facilities; and
  - Emergency power via on-site diesel generation;
- On-site ancillary facilities:
  - Administration buildings and offices, including messing and ablutions facilities;
  - A medical clinic and emergency response facilities;
  - Security facilities and change house for plant operations and maintenance personnel;
  - Plant maintenance facilities, including a workshop, warehouse and stores;
  - Fuel storage and distribution facilities;
  - A fire protection system; and
  - Communications infrastructure;
- Other supporting infrastructure:
  - TSF;
  - Mining services area, comprising infrastructure to support the mining operation;
  - WRDs; and
  - Explosives storage facilities;
- Accommodation facilities:
  - A 300-bed main camp; and
  - An 80-bed security camp.

The site layout and layout drawings for the plant and infrastructure are provided in Figure 18.1, Figure 18.2 and Figure 18.3.

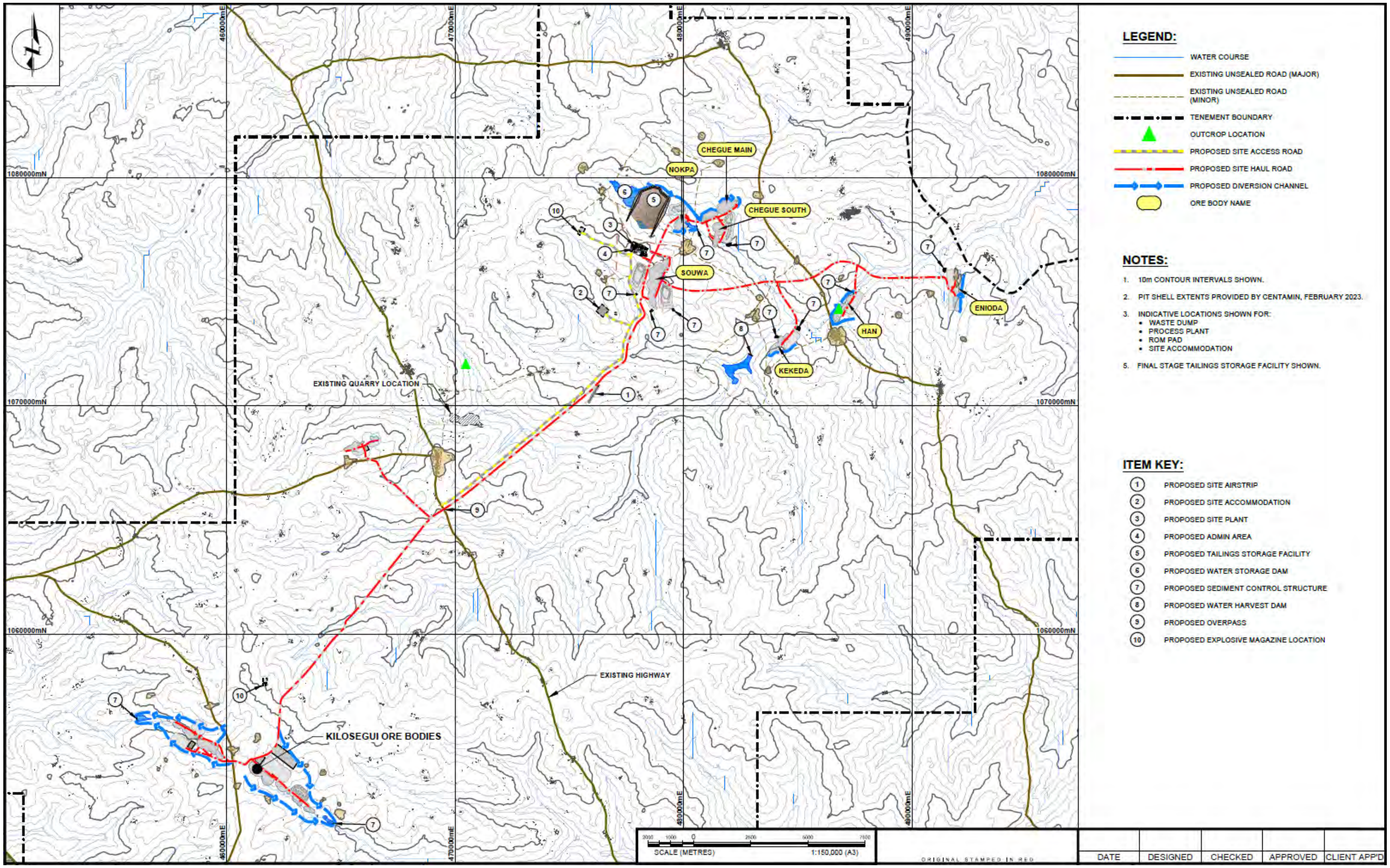


Figure 1.8 Site Layout (Source: Knight Piésold, 2024)

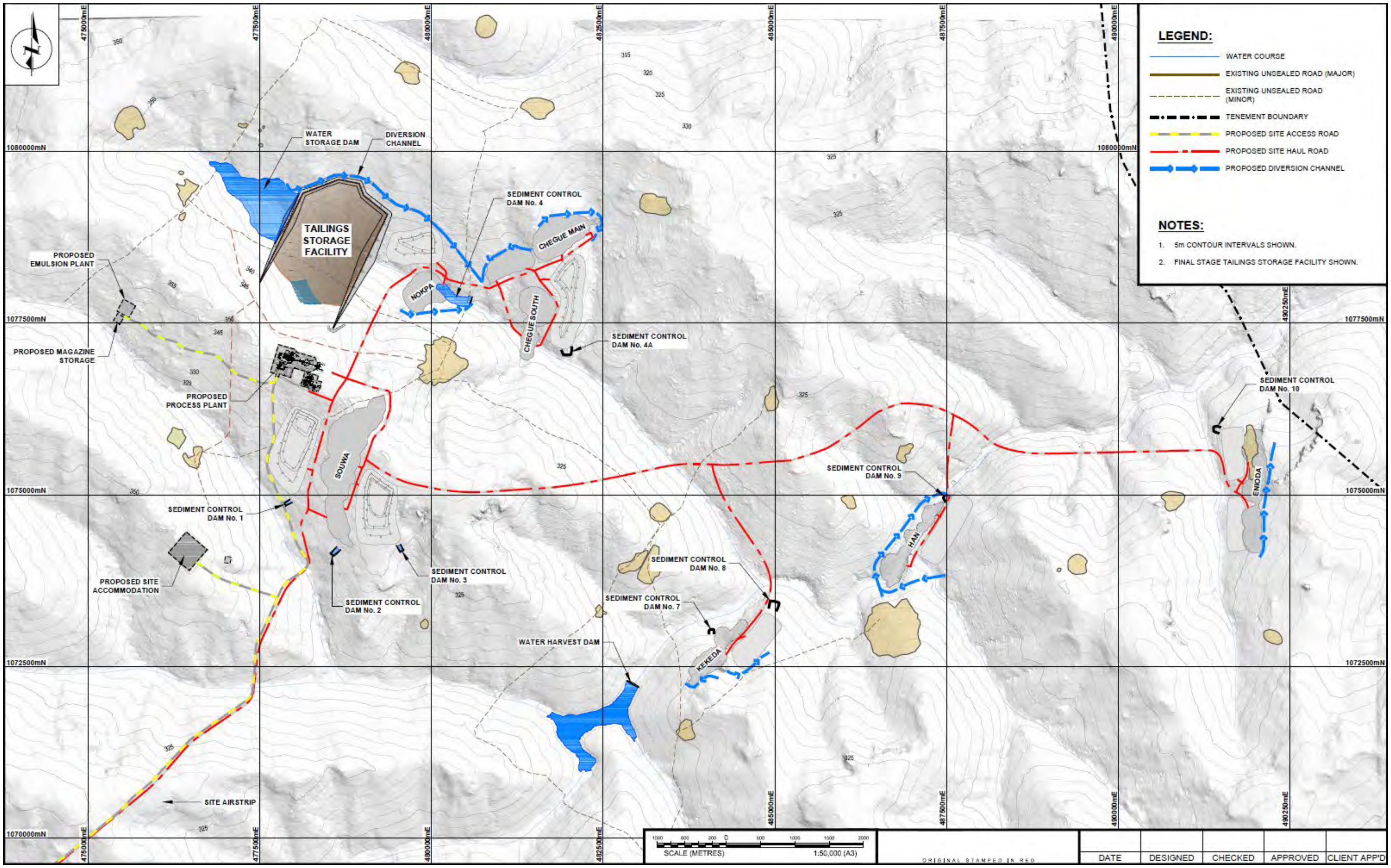


Figure 1.9 Mine Infrastructure Layout (Source: Knight Piésold, 2024)

1.13.1 TSF Design Summary

The TSF has been designed to Australian National Committee on Large Dams (ANCOLD) guidelines and is consistent with Centamin's Group commitment to be in conformance with the Global Industry Standard on Tailings Management (GISTM).

The TSF will comprise a side hill storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 146 ha for Stage 1 increasing to 287 ha for the final stage facility.

The TSF is designed to accommodate a total of 41.1 Mt of tailings (a slightly higher volume than the minimum LOM requirements of 38.2 Mt). The Stage 1 TSF is designed for 18 months of storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements however, this may be adjusted to biennial raises to suit mine scheduling during the operation. Downstream raise construction methods will be utilised for all TSF embankment raises.

1.13.2 Water Management

The water balance modelling included the TSF, WHD, WSD and process plant, with a view to determining site water storage requirements. Design wet conditions were modelled to ensure that the TSF is designed with sufficient storage capacities to comply with design criteria. Key findings from the water balance modelling are as follows:

- The TSF is designed to hold the tailings generated, plus the design rainfall conditions, and thus has sufficient stormwater storage capacity for all design storm events and rainfall sequences;
- The supernatant pond volume peaks in October each year (at the end of the wet season);
- A process water shortfall is expected to occur under average and design dry climatic conditions;
- All make-up water requirements can be provided by the WSD reservoir, supplemented by the WHD for design dry conditions;
- It is necessary that the WSD is completed early to allow a full wet season of filling prior to commissioning;
- For a basin permeability of 1 x 10^{-7} m/s, a WSD storage capacity of 2,000,000 m^3 is required to provide sufficient make-up water, supplemented by an abstraction rate of 125 L/s (unfactored) from the WHD;
- A WHD capacity of 500,000 m^3 is required to reduce the risk of shortfalls under design dry conditions.

1.13.3 Water Harvest Dam

The WHD will be the primary water collection structure and able to store up to 500,000 m^3 of water at the maximum operating level. The design intent of the WHD is that the reservoir will be frequently pumped to the WSD during each wet season, with a view to filling the WSD reservoir to its maximum storage level prior to each dry season. The WHD will have a catchment area of 43,000 ha.

1.13.4 Water Storage Dam

The WSD will be the primary storage pond for clean process water on site and be able to store up to 2,000,000 m^3 of water at its maximum operating level.

The WSD will have a catchment area of 1,110 ha. The WSD is intended to be recharged by water abstracted from the WHD and rainfall runoff from its upstream catchment. Pit dewatering will be pumped to the WSD. It is assumed that dust suppression and wash down water will be sourced from the WSD. The water collected in the WSD will be pumped back to the plant to supply plant raw water requirements and process make-up water requirements. Water will be recovered from the WSD by a floating pump.

1.13.5 Access Roads

The following were proposed as part of the road design:

- Haul roads have been designed to connect the Chegue/Nokpa pit and Souwa pits in the north and the Kilosegui Ore bodies in the south with the proposed process plant (total distance of 30 km). An overpass is proposed where the haul road intersects the existing highway (sealed A1 national highway);
- Site access roads have been designed to connect the A1 national highway with the proposed plant site, accommodation village, magazine, and site airstrip, with a total distance of 16 km. The site access road will generally run parallel to the haul access road.

1.13.6 Airstrip

The airstrip was designed in accordance with the Australian Government Civil Aviation Safety Authority (CASA), RACI 6001 Aerodrome Design and Operation (ANAC) and International Civil Aviation Organisation (ICAO) guidelines. The design aircraft for the project is a Cessna Caravan. The runway surface will be 850 m and 20 m wide, with an 80 m wide surrounding runway strip. Cut and fill operations will be required to achieve a design compliance with CASA and ICAO guidelines.

1.13.7 Power Supply

The power supply for the project will be via Côte d'Ivoire national High Voltage (HV) electricity grid.

The proposed power supply solution for the project is via the existing Bouna Substation which is approximately 55 m southeast of the project area. The Bouna substation is fed via a 90 kV transmission line from the 225/90 kV Bondoukou Substation further south.

The design of the proposed power supply solution has been based on an estimated installed load for the site of 27 MW and a maximum demand (operating) load of 21 MW.

1.14 Market Studies and Contracts

Gold is a readily traded market that operates internationally. Major trading centres are located across all time zones with most global gold trading volumes passing through the London Over-The-Counter (OTC) market, US futures market and the Shanghai Gold Exchange.

In 2023, the average London Bullion Market Association ("LBMA") gold price increased by 13% over the year closing at US$2,065/oz, having started the year at US$1,823/oz. Similarly, with annualised average prices, 2023 prices averaged US$1,943/oz versus an average price of US$1,802/oz in 2022. Prices rose steadily throughout the year with a low of US$1,810/oz and a high of US$2,079/oz. During the first half of 2024, gold prices have risen by 13% starting the year at US$2,065/oz and closing the half at US$2,326/oz.

Period	Annual Average US$/oz
2017	1,258
2018	1,269
2019	1,393
2020	1,771
2021	1,799
2022	1,802
2023	1,943
2024 H1	2,205

Table 1.12 Historical Annual Average LBMA Gold Prices (S&P Capital IQ)

Gold prices are expected to be supported by expectations of interest rate cuts by the major global economies and ongoing global geopolitical uncertainty and conflict, these are expected to sustain demand for safe haven assets such as gold, from investors and central banks.

Period	Consensus Forecast US$/oz
2024	2,235
2025	2,297
2026	2,226
2027	2,148
2028	2,154

Table 1.13 Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023)

The gold price used for the reserve optimisation parameters was US$1,450/oz which is well below current market prices and price forecasts, it is also well aligned with industry norms for reserve pricing.

The gold price used for the economic analysis was US$1,900/oz.

For the doré produced from the proposed Doropo treatment plant, in the absence of letters of interest or letters of intent from potential smelters or buyers of gold, general smelter terms for similar projects have been applied.

1.15 Environmental Studies, Permitting and Social or Community Impact

The Doropo Gold Project currently covers eight discrete mineable mineral resources, located over four of the seven Ampella exploration tenement areas of 1,847 km^2, named Souwa, Nokpa, Chegue Main, Chegue South, Kekeda, Han, Enioda and Kilosegui.

H&B Consulting SARL and Earth Systems Consulting Pty Ltd were engaged by Ampella to prepare an Environmental and Social Impact Assessment ("ESIA") in compliance with key Ivoirian regulatory requirements, and to align with international good practice.

The draft ESIA report was submitted to the National Environment Agency (ANDE) in the first quarter 2024. The Environmental Permit for Doropo was granted on the 13 June 2024, following a public enquiry and validation of the Environmental and Social Impact Assessment ("ESIA") by the interministerial committee on the 10 May 2024.

The Doropo development will consist of eight open mine pits, one processing plant, one tailings storage facility, waste rock dumps adjacent to each pit, accommodation and office facilities. The development of the Project will be phased, commencing with a 'Main' Project Development Area (MPDA) with extension to the Kekeda, Han, Enioda, and Kilosegui satellite pits in subsequent years. All deposits are located within a 7 km radius of the MPDA, except for the Kilosegui pit which is approximately 30 km to the south-west, and will be connected by ore haul roads.

Various project design and layout alternatives for the Doropo Project have been examined as part of the ESIA process. Significant changes have been made to the Project layout in the design process which has allowed potential negative impacts on environmental and social values to be minimised and positive impacts to be enhanced.

Project Development Areas (PDAs) have been defined to encompass key elements of project infrastructure, including appropriate safety and security buffer zones of approximately 500 m from pit shell extents. PDAs have been carefully designed to minimise potential environmental and social impacts. Proposed PDAs comprise of:

- One, larger, 'Main Project Development Area' (MPDA), containing three resource deposits (Souwa, Nokpa, Chegue) and the primary mine infrastructure (i.e. TSF, WSD, process plant and mine services area);
- Four separate PDAs for each of the satellite deposits (Kekeda, Han, Enioda and Kilosegui), their respective WRDs and buffer zones. Each satellite PDA will be connected to the Main PDA via ore haul roads.

Some mining areas will be mined for less than a 2-year period prior to rehabilitation and closure. Ampella will require full control over land access within the PDAs, to maintain project security and community safety. Ampella will acquire rights of use to the land within PDA boundaries from host communities.

A number of mine components and supporting facilities will be established outside PDA boundaries. These include the WHD, a sediment pond downstream of the MPDA, the accommodation camp, the explosives magazine and emulsion plant, and the airstrip.

An environmental and social baseline has been established for the Doropo Project with extensive field studies undertaken by H&B Consulting and Earth Systems Consulting since February 2022 to support pre-feasibility and feasibility design studies as well as the statutory ESIA. These studies have included those related to socio-economic conditions, land and water use, surface and groundwater resources, terrestrial and aquatic ecology and biodiversity, air quality, noise and vibration, traffic and transportation, as well as archaeology and cultural heritage.

1.15.1 Project Setting

Four of the prospects are clustered in the sub-prefecture of Kalamon, which includes the largest prospect of Souwa. This is the proposed location of the processing plant, TSF, airstrip and office facilities. Four of the satellite prospects are located within 7 km of this main cluster, with Kilosegui located approximately 30 km to the southwest, all of which would require haul roads to truck ore to the processing plant.

The area around the mineral deposits is gently undulating with old erosional and weathered surfaces. Small land holding agriculture dominates the area with artisanal mining often found in the mineralised areas.

There is a distinct dry and wet season with surface water flow at the Doropo area consisting of a network of non-perennial streams and smaller tributaries.

A majority of the PDA is comprised of modified habitats, predominantly agricultural areas such as cashew plantations as well as artisanal mining, settlements, roads and tracks. Four principal natural vegetation types occur within the PDA, these are wooded savannah, shrub savannah, gallery forest and swampy thicket. All habitats are highly degraded due to agricultural, grazing and artisanal mining activities.

The Project is located in a poor and underserviced part of Côte d'Ivoire. Education, health, transport, water supply, energy and communication services are basic. There are very few job opportunities apart from within agriculture and artisanal mining.

The Bounkani Region is administered by a regional prefect who is supported in this role by Department-level prefects and sub-prefects and various governmental technical agencies. The region is characterised by a customary system of land governance held by the Koulango authorities, according to which land resources are the exclusive property of the King of Bounkani. The King entrusts administrative responsibilities for land management to local-level Canton Chiefs. Within the PDA, legal land title has yet to be implemented and the customary system of land tenure is the only established land management mechanism.

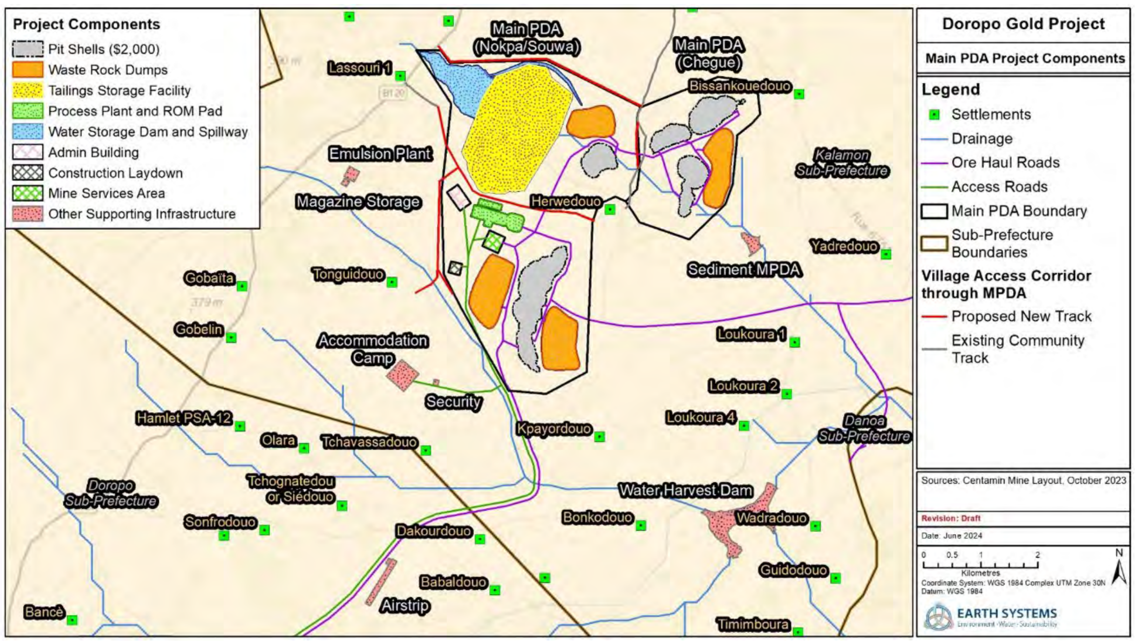


Figure 1.10 Main Project Development Area (MPDA) (Source: Centamin, 2024)

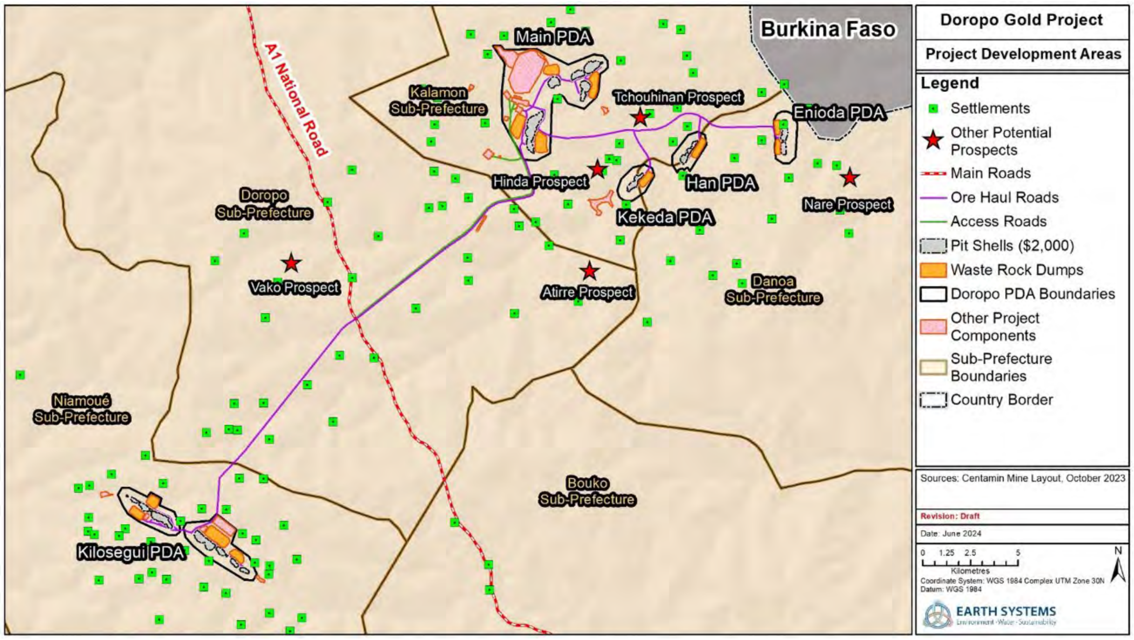


Figure 1.11 All proposed Project Development Areas, including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs (Source: Centamin, 2024)

1.15.2 Socio-Economic Benefit

The key benefits of the proposed Project for Côte d'Ivoire will be financial, economic and social benefits at the national, regional and local levels. On a local level, the Project can provide a significant injection of income and employment opportunities to the region, providing opportunities for training and skill development. It has the capacity to catalyse economic and social development in the area, improve access to infrastructure, goods and services, and leave a lasting positive legacy. Royalties and taxes from the Project would also provide needed income for the national government of Côte d'Ivoire.

The generation of royalties, taxes and dividends from gold mining to the Côte d'Ivoire National Government contributes significantly to the national goals of the country. The three main taxes and fees imposed on companies operating in the mining sector are royalties, salary withholding tax and industrial and commercial profits tax. The Côte d'Ivoire government also has the right of a 10% free carried ownership interest in the development.

Based on the current gold price, which is in excess of US$2,000 per oz, government royalties would be 6% of sales revenue or approximately US$20 million per year.

Overall the Doropo Project would be expected to contribute over US$2 billion (based on the current gold price) to the Ivorian economy in the form of taxes, royalties, dividends, salaries, payments to local suppliers and infrastructure development. During operation Ampella can be expected to be one of the largest corporate taxpayers in Côte d'Ivoire.

Mines operating in local communities in Cote d'Ivoire are required to pay 0.5% of their turnover to a Local mining development fund, which aims to finance community development projects in mining regions. This fund is expected to generate approximately US$17 million over a 10-year life of the Project with approximately US$2 million generated each year for the first five years of operation.

Through effective management, the injection of income and economic opportunity into the local community as a result of the Project will be a major benefit. Through employment, training, procurement and livelihood improvement programmes, the Project can be a catalyst for sustained economic growth in the Doropo region. The Company's existing social investment programme will also continue to complement Project-related activities and contribute to achieving local development objectives.

1.15.3 Employment and Local Economy

The Doropo Gold Project is expected to be a driver for economic growth in the region through employment expenditure, infrastructure development, and social investment. A significant proportion of the capital expenditure and operating costs for the Project will be spent in the prefecture of Doropo, resulting in flow-on benefits to the local community.

Through its mineral exploration and feasibility study activities, the Doropo Project has already generated job opportunities and economic activity in the region. This will be magnified significantly if the Project moves forward to development. During the construction phase, a period of approximately two years, the Project will create more than 1,000 new employment opportunities through direct and contractor hiring. Employment for the Project will result in an injection of income in nearby villages of the surrounding area and is expected to provide strong socio-economic opportunities for local communities surrounding the Project.

Ampella will bring professional, safety and quality management systems to its operations which will positively impact and influence employees and local communities. The introduction of new employment, training and business opportunities (both direct and indirect) is likely to increase average income in the Project affected villages and result in an improvement in living standards. Ampella is committed to training and skills development of local workers. Through these activities, the Project will raise the skills base in the local area allowing the local workforce to enter the mining industry and advance throughout Ampella's organisation.

Ampella will uphold international requirements concerning modern slavery risks across the supply chain, and the approach to diversity, equal opportunity and pay, discrimination, employee relations and labour conditions.

Employment opportunities will continue during the operations phase with an overall workforce expected to stabilise at approximately 1,000 direct employees and contractors. Operations will continue for at least 10 years, although most large gold mining Projects typically extend their life through further mineral exploration.

Based on the economic opportunities created during mining operation and the more stable employment of the operations phase it could be expected that a significant town would develop proximal to the mine. This would be expected to develop near to the MPDA. There are many examples of successful and thriving towns throughout the world that commenced their life as a mining town.

Local procurement of goods and services for the Doropo operation will be promoted during the operations phase and continue to enhance business opportunities for local villages and towns. Continued business opportunities (both direct and indirect) will increase average income in the Project affected localities, the region, and the country and result in an improvement in living standards leading to positive impact on the country's socio-economic development.

1.15.4 Training and Skills Development

Ampella is committed to supporting the development of mining labour skills in Côte d'Ivoire including supporting the Government's efforts to promote the national mining industry. Mining requires diverse skill sets including engineering, geology, human resource management, accounting, construction, and administration, which are all transferrable across companies and industry sectors.

Significant benefits from increased training and skills development will continue throughout the construction and operations phases of the Project. Operations employment will be longer term than typical construction jobs and will provide opportunity for entry into well paid mining employment.

Project Phases	Ampella Mining		Contractors		Total
	Permanents	Temporary	Permanents	Temporary	
Pre-Construction Phase: -Year 2	39	19	0	22	80
Labourer (Ouvrier)	10	9	0	10	29
Middle skill (Agent de Maîtrise)	20	8	0	10	38
Senior (Cadre)	9	2	0	2	13
Construction Phase: -Year 1	85	145	30	1,350	1,610
Labourer (Ouvrier)	35	100	17	500	652
Middle skill (Agent de Maîtrise)	30	40	8	800	878
Senior (Cadre)	20	5	5	50	80

Project Phases	Ampella Mining		Contractors		Total
	Permanents	Temporary	Permanents	Temporary	
Operation Phase: Year 0	246	90	110	780	1226
Process	130	50	0	0	180
Labourer (Ouvrier)	70	30	0	0	100
Middle skill (Agent de Maîtrise)	40	15	0	0	55
Senior (Cadre)	20	5	0	0	25
Mining	46	0	30	520	596
Labourer (Ouvrier)	10	0	5	250	265
Middle skill (Agent de Maîtrise)	25	0	15	200	240
Senior (Cadre)	11	0	10	70	91
Administration	70	40	0	20	130
Labourer (Ouvrier)	15	15	0	10	40
Middle skill (Agent de Maîtrise)	45	20	0	8	73
Senior (Cadre)	10	5	0	2	17
Sub-contractors (Lab, Explosive, Catering, Fuel...)	0	0	80	240	320
Labourer (Ouvrier)	0	0	50	155	205
Middle skill (Agent de Maîtrise)	0	0	20	70	90
Senior (Cadre)	0	0	10	15	25

Table 1.14 Indicative Employment Numbers and Classification – Construction and Operations

1.15.5 Land Acquisition, Livelihood Restoration and Resettlement

Within the PDA, legal land title has yet to be implemented and the customary system of land tenure is the only established land management mechanism. The PDA has a complex land ownership history that will be addressed sensitively through stakeholder engagement during development.

Currently most land occupants and users are from the Lobi ethnic group who mostly settled in the area from Burkina Faso over the last 100 years. Historically the Koulango had a kingdom centred on Bouna and were the traditional custodians of the land. Low numbers of Peuhl or Fulani livestock herders also traditionally used the land and occupy a few small hamlets in the areas identified for project development.

The key social issues and risks that will need to be managed very carefully by the Project are:

- Resettlement and livelihood restoration for impacted communities with high economic and social vulnerability;
- Implementing strong livelihood restoration in addition to appropriate cash compensation;
- Managing the potential for land conflicts especially those associated with the overlapping rights of Koulango and Lobi communities;

- Ensuring a fair system of employment especially across different ethnic groups;
- Community health and safety particularly associated with transportation routes; and
- Land and socio-economic fragmentation resulting from the extended Project footprint.

1.15.6 Land and Livelihood

The loss of access to land will have direct and indirect impacts to the communities living in these areas. A total of 23 villages own land within PDA or supporting infrastructure boundaries, including 18 villages that own land within PDA boundaries and an additional five villages that own land within supporting infrastructure footprints. Village land impacts are summarised as follows:

- Seven villages with major impacts (>25% land impacted): Herwedouo, Bissankouedouo, Holidouo, Wadaradouo, Dupindouo, Gbelta, Lehitedouo;
- Nine villages with medium impacts (5-25% land impacted): Tonguidouo, Lassouri, Lagbo, Fafoudouo, Behindjinadouo, Mabridouo, Fangadouo, Gangata, Norfadouo/Yolkoubiel;
- Seven villages with minor impacts (<5% impacted): Gola, Simatedouo, Bonkodouo, Douhoundo, Loukoura, Sonfrodouo, Talo.

To follow the mine schedule and reduce the impacts to village land, the MPDA have been subdivided into two separate areas. The MPDA section containing the Souwa and Nokpa Pits will be mostly inaccessible for the duration of the Project due to the construction and operation of infrastructure required for the full LOM. The MPDA Chegue section contains the Chegue Main and Chegue South pits which are not expected to be mined until Year 7 of the Project, and so land within this section will remain likely to be accessible to landowners and land users until required, and under the provision that no further settlement or illegal artisanal mining activities are established.

The Kilosegui PDA has been split into east and west sections, with development of the eastern area expected to occur during Years 3 to 8 and development of the western area expected to occur during Years 5 to 8 of the Project.

PDAs cover a combined total area of 3,783.4 a as detailed in Table 1.15 below.

Land Cover Category	MPDA	Enioda PDA	Han PDA	Kekeda PDA	Kilosegui PDS	Total
Grazing Land/Forest Resource Use Areas	728.4	48.9	16.8	35.1	95.6	924.8
Cashew Plantation (Mature)	391.7	71.8	20.4	44.9	297.5	826.3
Forest and Natural Habitat Fragments	363.2	38.8	62.3	96.1	219.4	779.8
Other Crops (e.g. Maize, Yam, Millet)	250.1	54.4	24.1	32.7	292.6	653.9
Cashew Plantation (Immature) and Other Crops	72.2	30.4	2.9	0.8	112.8	219.1

Land Cover Category	MPDA	Enioda PDA	Han PDA	Kekeda PDA	Kilosegui PDS	Total
Recently Fallow	46.9	6.0	74.4	11.8	37.7	176.8
Artisanal Mining (Active and Inactive)	53.2	10.4	26.2	17.4	3.0	110.2
Intensive Agriculture (e.g. Rice)	27.8	3.3	0.0	0.0	7.6	38.7
Roads	9.7	4.1	3.6	4.5	4.9	26.8
Settlement Area	1.7	15.2	0.7	0.7	0.6	18.9
West African Bowal	6.4	0.0	0.0	0.0	0.0	6.4
Sacred Forest	2.2	0.0	0.0	0.0	0.0	2.2
TOTAL	1953.6	283.3	231.4	243.5	1071.7	3783.4

Table 1.15 Land Cover Within the Doropo Project Development Areas (PDAs)

1.15.7 Soil Geochemistry

The Doropo Project encompasses thirteen areas of mineralisation extending over an area of 1,847 km^2. Eight of these areas are currently considered “mineable” and are planned to be exploited through open pit mining. These include Chegue Main and Chegue South, Enioda, Han, Kekeda, Kilosegui, Nokpa, and Souwa. Ten WRDs are anticipated for storage of waste rock extracted from the open pits during mining. A total of 187.5 million tonnes of waste rock will be produced and stored on the ten anticipated WRDs over the 10-year life of mine.

The geology of the project area comprises homogenous granodiorite with some large post-mineralisation dolerite dykes. The weathering profile includes a surface layer of transported sediments and granitic soil, underlain by a saprolite horizon. A thin transition zone of saprock overlies a sharp contact with the fresh granodiorite below.

Testing

Geochemical characterisation of waste rock and ore material for the Doropo project has been carried out, with the aim of providing guidance to waste management strategies, and to provide inputs to predictive water quality models for pit sumps and waste rock dump seepage. An industry-standard suite of static and kinetic tests was performed on a population of samples representing each of the fresh rock, saprock and saprolite material types associated with the Doropo deposits. The sample population includes 196 individual samples collected and analysed over two campaigns, the results of which have been compiled into a unified geochemistry database for Doropo.

Analysis

The results of static testing show that the ore (>0.2 g/t Au) and waste rock material has a generally low total sulphur (S) content, with all samples containing less than 1% total S. The highest total S was recorded for an ore-grade sample of granodiorite, at 0.88% S. The average total S content for waste material is 0.04%.

Nearly all of the S present is shown to be in the form of sulphide. As a result of the low S contents, the sample population has a low acid generation potential (AP), and generally higher values of neutralisation potential (NP). The average neuralisation potential ratio (NPR) for the population is 15, implying that samples have on average 15 times greater NP than AP. All waste samples are classified on the basis of static test indices as non-potential acid generating (PAG), while 10% of ore samples are classified as PAG.

Bulk geochemical analysis indicates low levels of enrichment of the Doropo ore and waste samples with respect to the average crustal abundance of elements of environmental significance. Enrichment factors across the sample suite are routinely less than 5. Exceptions are the Mo content in the granodiorite lithology and in the vein samples, with averages of ~6.3x and 10x crustal abundance, respectively. The vein material also shows an enrichment factor of 6.3x crustal abundance for Pb. The results of net acid generating (NAG) extract analysis and shake flask extraction (SFE) leachates indicate low leachability of most solutes from the material, including Mo and Pb. The average concentrations of Fe and Al in NAG extract and SFE leachates exceed the proposed drinking water standards, but are compliant with the discharge and/or surface water quality thresholds.

Results

No detectable cation exchange capacity (CEC) was recorded in nine out of the ten samples of saprolite submitted for analysis, while 1 sample had a low CEC value of 13 cmol/kg.

A total of 10 humidity cell tests, representing each of the fresh, saprock and saprolite materials, including one ore sample, were also carried out. The results of humidity cell tests indicate generally low mobility of sulphate and metals from the Doropo material. The fresh samples produced more alkaline pH leachates (8.5 to 9.5) than the other lithotypes, reflecting the higher calcite abundance and resultant neutralisation potential of the fresh material. The saprolite composites produced the lowest pH leachates of between 5.5 and 7.0, consistent with the lower NP values in the static test results. Sulphate concentrations are highest in the fresh rock leachates, reaching a maximum of ~40 mg/L and declining to values of <10 mg/L later in the test period. The concentrations of Al and Fe in one or more humidity cell test (HCT) leachates exceed the proposed drinking water standards for several weeks of testing, while the concentrations are compliant with discharge standards. Most samples, however, yield leachates with Fe and Al concentrations within the acceptable range for drinking water. Concentrations of Mo and Pb are well below drinking water standards in all sample leachates.

The results of humidity cell testing of the main material types at Doropo were used to generate inputs to facility-scale geochemical models of contact water quality for the open pits and WRDs. The model input solutions are derived from the 16 weeks of HCT data available at the time of preparation of this report. Modelling was undertaken to predict the water quality associated with pit sumps and WRD seepage at four of the deposit areas (Kilosegui, Han, Souwa and Nokpa). Model simulations were run on a monthly basis, based on the average monthly rainfall for the site.

The results of pit sump geochemical modelling indicate that sump water will have a pH ranging from 8.1 to 8.7, with a maximum sulphate concentration of ~25 mg/L. The concentrations of sulphate and most metals will be highest at the very start of the wet season, as accumulated weathering products from the dry season are flushed from the pit walls by the first significant rainfall of the wet season. The concentrations will be lowest in the peak of the wet season as a result of dilution by greater rainfall volumes. The concentrations of all parameters are predicted to be within the acceptable range of the proposed drinking water standards for Doropo.

Conclusion

Model simulations for WRD seepage were conducted for 2 scenarios. Scenario 1 assumes 50% infiltration of incident rainfall, while Scenario 2 is based on 25% infiltration. The results for Scenario 2 indicate higher concentrations of sulphate and metals, due to smaller volumes of seepage moving through the WRD rock mass and resulting in more concentrated solutions. Under Scenario 2, the maximum sulphate concentration is predicted for the Han WRD, as this has the greatest proportion of fresh material stored. The maximum sulphate concentration predicted for Han under Scenario 2 is just less than the 250 mg/L drinking water threshold, while the maximum Scenario 1 concentration is ~150 mg/L. For all modelled WRDs, the concentrations of all solutes are predicted to be less than the drinking water thresholds for the site, under both Scenario 1 and Scenario 2. Concentrations of Fe and Al, while shown to be elevated in some HCT leachates, are predicted to be low in WRD seepage (and pit sump water), as a result of oversaturation of Fe and Al hydroxides at the predicted pH levels, and the resultant precipitation of these minerals.

1.16 Capital and Operating Costs

The capital and operating cost estimates reflect the Project scope as described in this Technical Report. All estimated costs are expressed in United States dollars (US$), unless noted otherwise, with a base date of the second quarter 2024 (2Q24).

1.16.1 Capital Costs

The capital cost estimate for the Doropo Gold Project has been compiled by GRES based on inputs received from:

- Orelogy - provided costs associated with mine establishment and associated facilities;
- GRES - provided the costs for the processing plant and associated infrastructure;
- Knight Piésold - provided quantities for the tailings storage facility, water harvest dam, water storage dam, site access roads, haul roads, airstrip and sediment control structures. Pricing, incorporating unit rates and costs were developed and applied by GRES;
- ECG - provided the estimate for the power supply connection infrastructure; and
- Centamin - provided Owner's costs and compiled the LOM sustaining capital cost estimate, supported by Knight Piésold.

The Project Capital cost estimate (mining, processing and infrastructure) developed for the Feasibility Study (FS) is based upon an Engineering, Procurement, and Construction Management (EPCM) approach for the process plant and infrastructure and contract mining for mine development.

The estimate includes all costs associated with project management, process engineering, design engineering, drafting, procurement, construction management, and commissioning services required to construct and commission the processing facility' and its associated support infrastructure. Additionally, the estimate also includes costs related to the establishment of the mining services facilities, spare parts, and the provision of first fills, and consumables required for the commencement of operations.

The estimate has been based upon preliminary engineering designs, material quantity estimates derived from these designs, multiple budget price quotations for major process equipment, and budget rates for the supply of bulk commodities. Unit rates for site installation works were based on market inquiries specific to the Project from contractors experienced in the construction of minerals processing projects in Côte d'Ivoire and throughout West Africa.

The overall project capital cost estimate has been summarised in Table 1.16. The estimate accuracy is considered to be -10%/+15%.

Area	Total US$M
Mining, including pre-strip	22.4
Processing, including infrastructure	271.3
Owner's Costs	50.6
Project Contingency	29.0
Estimate Total (-10%/+15%)	373.2

Table 1.16 Project Capital Cost Estimate Summary

1.16.2 Sustaining Capital

The development of the sustaining capital included inputs from:

- Knight Piesold - for the estimate phases of the:
  - TSF;
  - WSD;
  - Sediment control structures;
  - Haul road development from satellite pits.
- Piteau Associates - for the pit dewatering requirements;
- Centamin - for the ESG requirements including resettlement and loss compensation.
- Centamin – closure and rehabilitation costs.

Area	Total US$M LOM
Infrastructure Phased Development	49.5
Pit Dewatering	4.5
ESG	6.0
Closure and Rehabilitation	36.0
Estimate Total (-10%/+15%)	96.0

Table 1.17 Sustaining Capital Estimate Summary

1.16.3 Operating Costs

Operating cost estimates for the Doropo Gold Project has been compiled by GRES based on inputs developed by:

- Orelogy - for mining contractor and mine management costs;
- GRES - for the processing costs;
- ECG - for the cost of power;
- Centamin - for the Site General and Administration (G&A) costs, as well as labour organisation charts, project manning, labour rates and operational manning build-up.

The operating cost for the Project, exclusive of sustaining and deferred capital costs is presented in Table 1.18.

Project Area	LOM	
	Cost US$M	Unit Cost US$/t ore treated
Mining	869	22.7
Processing		
Power	161	4.21
Maintenance Spares & Consumables	25	0.65
Operating Consumables	179	4.68
Labour	91	2.38
Laboratory	3	0.09
Other	3	0.09
General and Administration	156	4.1
Total	1,486	38.9

Note: 1) General and Administration (G&A) costs presented above include Site G&A costs of US$147M over the LOM and mine lease / permitting costs of US$9M over the LOM.

Table 1.18 Project Operating Cost Estimate Summary

1.16.4 Closure Costs

In the Centamin economic model, rehabilitation is ongoing during the operation with an additional final allowance for demolition and closure activities to be carried out after mining and processing activities have ceased. The total allowance for rehabilitation and closure in the model is USD36M (as per Table 1.17).

1.17 Economic Analysis

An economic analysis has been undertaken by Centamin and incorporates the FS outputs, including milled tonnages and grades for the ore and the associated recoveries, gold price (revenue), operating costs, bullion transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The evaluation method assumes that the Project has been assessed on a 100% ownership basis, with no debt financing.

The model assumes a long-term gold price of US$1,900/oz, which is below consensus forecasts, applied on a flat line basis from commencement of production. The Project is estimated to produce 167 koz per annum over its 10-year mine life, with an average cash cost of US$892/oz of gold produced and an average all-in sustaining cost (ASIC) of US$1,047/oz of gold sold. The initial capital is expected to be US$373M.

The Project economics indicate a Pre-tax NPV8% of US$568M and an IRR of 40% and a Post-tax NPV8% of US$426M and an IRR of 34%. Table 1.19 provides a summary of the Projects physical, financial and economics metrics.

Economic Summary @ US$1,900/oz Au	Units	Value
Mine life	Years	10
LOM ore processed	kt	38,226
LOM strip ratio	w:o	4.9:1
LOM feed grade processed	Au g/t	1.53
LOM gold recovery	%	89%
LOM gold production	koz	1,667
Upfront capital cost	US$M	373
Life of Mine Average:		
Gold, average annual production	oz	167
Cash costs per ounce	US$/oz	892
AISC per ounce	US$/oz	1,047
Project years 1 to 5		
Gold, average annual production	oz	207
Cash costs per ounce	US$/oz	817
AISC per ounce	US$/oz	971
Pre-Tax Economics		
Net present value - 8%	US$M	568
Internal Rate of Return	%	40%

Economic Summary @ US$1,900/oz Au	Units	Value
Post-Tax Economics		
Net present value - 8%	US$M	426
Internal rate of return	%	34%
Payback period	Years	2.1

Table 1.19 Project Economic Summary

An analysis of the post-tax NPV sensitivity to key input assumptions was conducted.

The after tax NPV sensitivity (in US$M) comparing varying discount rate percents and gold price in US$/oz is presented in Table 1.20. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

Discount Rate	1,500	1,600	1,700	1,800	1,900	2,000
5%	199	275	364	454	543	613
6%	175	247	332	416	501	568
7%	153	221	301	382	462	526
8%	132	197	273	350	426	487
9%	113	174	247	320	393	450
10%	95	154	223	293	362	417

Table 1.20 Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz

The after tax NPV sensitivity (in US$M) comparing operating expenditure fluctuations with varying gold prices in US$/oz is presented in Table 1.21. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

% Change	1,500	1,600	1,700	1,800	1,900	2,000
-20%	316	380	457	533	610	670
-10%	224	288	365	442	518	578
0%	132	197	273	350	426	487
10%	40	105	182	258	335	395
20%	-51	13	90	167	243	303

Table 1.21 Operating Expenditure % Change vs Gold Price US$/oz, NPV8% in US$M

The after tax NPV sensitivity (in US$M) comparing up-front capital expenditure fluctuations with varying gold prices in US$/oz is presented in Table 1.22. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

% Change	1,500	1,600	1,700	1,800	1,900	2,000
-20%	201	265	342	419	495	555
-10%	166	231	308	384	461	521
0%	132	197	273	350	426	487
10%	98	162	239	316	392	452
20%	64	128	205	281	358	418

Table 1.22 Up-front Capital Cost % Change vs Gold Price US$/oz, NPV8% in US$M

1.18 Schedule, Project Implementation and Other Relevant Information

The Project implementation approach proposed for the Doropo Gold Project is for Centamin (the 'Owner') to engage a principal EPCM Contractor to provide design, procurement, and construction management services for the delivery of the processing plant and selected infrastructure facilities, which will be handed over to the Owner's operating team on completion.

The development of the mine and construction of the TSF, WHD, WSD, airstrip, incoming 90 kV transmission line and 90 kV/11 kV switchyard, security camp and main camp will be undertaken by specialist consultants/contractors directly engaged by the Owner, then handed over to the Owner's operating team.

The capital cost estimate and preliminary implementation schedule have been developed based on an EPCM implementation strategy, which is consistent with this approach.

The anticipated project timeline is as follows:

- Submit mining license application during H2-2024;
- First gold 27 months from final investment decision (FID) (T=0).

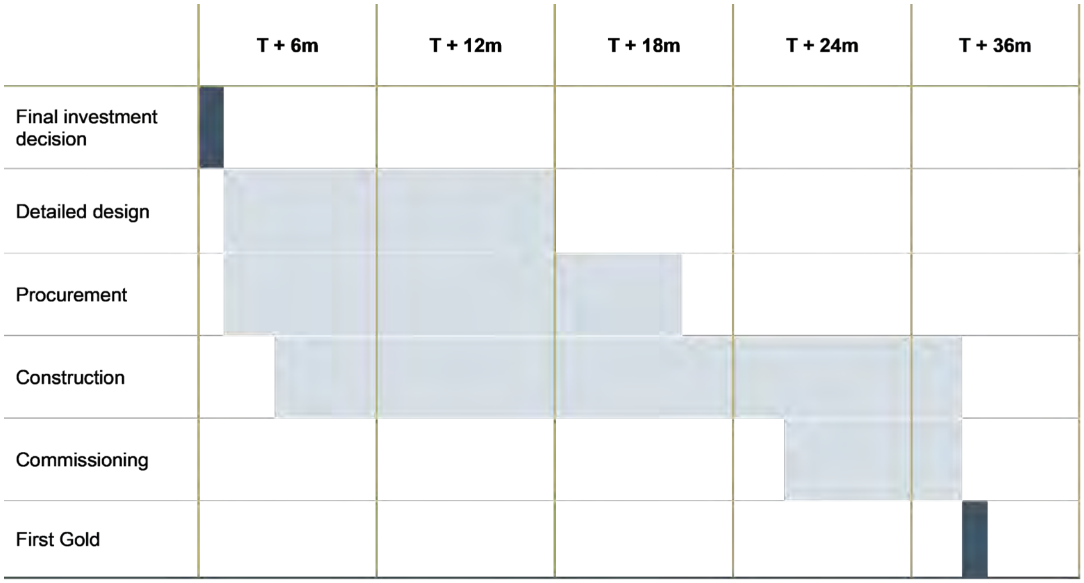


Figure 1.12 Project Timeline (Source: Centamin, 2024)

The estimated durations of key project activities are provided in Table 1.23.

Milestones	Months from EPCM Award
Early Works commencement	-3.8
Board Approval & FID	0
EPCM Award	0.2
Award of Long Lead Equipment	1.6
Site Access & Initial Construction Accommodation	2.9
Commence Process Plant Bulk Earthworks	5.0
Commence Process Plant Concrete Works	8.5
Permanent Camp Completion	10.9
Detailed Design and Engineering Completed	11.4
Processing Plant Practical Completion	27.1
First Ore into Plant	27.1
First Gold Pour	27.4

Table 1.23 Project Implementation Milestones

The timing of early works construction activities will be critical to ensure they coincide with the dry seasons, maximising performance and productivity, particularly for earthworks and initial/major civil works. These activities are to be completed by the Owner's team ahead of FID to facilitate overall project schedule compression as indicated.

Owner activities, including the construction of access roads, accommodation, water supply for construction, and bulk earthworks, have been carefully planned to ensure subsequent activities can proceed under the most favourable environmental conditions. This is especially important for constructing the WHD before the first wet season to enable early water harvesting.

1.19 Interpretations and Conclusions

1.19.1 Economic and Social Benefits

The key benefits of the proposed Project for Côte d'Ivoire will be financial, economic and social benefits at the national, regional and local levels. On a local level, the Project can provide a significant injection of income and employment opportunities to the region, providing opportunities for training and skill development. It has the capacity to catalyse economic and social development in the area, improve access to infrastructure, goods and services, and leave a lasting positive legacy. Royalties and taxes from the Project would also provide needed income for the National Government of Cote d'Ivoire.

The generation of royalties, taxes and dividends from gold mining to the Côte d'Ivoire National Government contributes significantly to the national goals of the country. The three main taxes and fees imposed on companies operating in the mining sector are royalties, salary withholding tax and industrial and commercial profits tax. Additionally, the Côte d'Ivoire Government holds the right to a 10% free carried ownership interest in the development.

Based on the current gold price, which is in excess of US$2,000 per oz, government royalties would be 6% of sales revenue or approximately US$20 million per year.

Overall, the Doropo Project would be expected to contribute over US$2 billion (based on the current gold price) to the Ivoirian economy in the form of taxes, royalties, dividends, salaries, payments to local suppliers and infrastructure development. During operation, Ampella can be expected to be one of the largest corporate taxpayers in Côte d'Ivoire.

Mines operating in local communities in Cote d'Ivoire are required to pay 0.5% of their turnover to a Local Mining Development Fund, which aims to finance community development projects in mining regions. This Fund is expected to generate approximately US$17 million over a 10-year life of the Project with approximately US$2 million generated each year for the first five years of operation.

Through effective management, the injection of income and economic opportunity into the local community as a result of the Project will be a major benefit. Through employment, training, procurement and livelihood improvement programmes, the Project can be a catalyst for sustained economic growth in the Doropo Region. The Company's existing social investment programme will also continue to complement Project-related activities and contribute to achieving local development objectives.

1.19.2 Metallurgy

The metallurgical work carried out to date indicates that gold can be satisfactorily recovered from Doropo ores using conventional Carbon in Leach (CIL) cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

1.19.3 Mineral Resources

Cube has produced a Mineral Resource estimation update for the Doropo Gold Project in line with the scope of its engagement by Centamin. The Mineral Resource has been reported in accordance with CIM Definition Standards – For Mineral Resources and Mineral Reserves (CIM Council, 2014), with an effective date of 31 October 2023.

The Mineral Resource is reported within pit shells using a metal price assumption of US$2,000/oz Au, and is reported above 0.3 g/t Au in-pit. The qualified person believes this is a reasonable approach, considering the potential mine life and considerations for reporting Mineral Resources in accordance with the CIM Definition Standards (CIM Council, 2014).

The Mineral Resource is considered to have Reasonable Prospects for Eventual Economic Extraction (RPEEE) on the following basis:

- The Project is located in a mining jurisdiction with multiple operating gold mines, with no known impediments to land access or tenure status;
- The volume, orientation and grade of the Mineral Resource is amenable to mining extraction via traditional open pit methods;
- Current metallurgical recovery based on available definitive feasibility level metallurgical test work was used in a pit optimisation to generate the resource pit shells.

The Mineral Resource Qualified Person is of the opinion that the data used in the preparation of the Mineral Resource estimates were collected in a manner consistent with industry good practice and are therefore fit-for-purpose. The Mineral Resource estimate was undertaken using a range of appropriate statistical, geostatistical and 3D visual analysis tools and methods, using reliable and proven software. The Doropo mineralisation is still open, especially at depth; the potential therefore exists to further extend it.

1.19.4 Mineral Reserves

Given the comprehensive mining study undertaken for the Doropo definitive feasibility Study and considering the conditions provided in this report, Orelogy's Qualified Person considers the Doropo Gold Project to be technically and economically viable.

- Mining selectivity and ore definition will be important to the success of the mining operation to ensure ore loss and dilution is minimised. Early grade control drilling ahead of operations will assist to mitigate the upfront risk of grade certainty;
- The geotechnical wall slope criteria has been developed to a high level of detail based on the modelled weathering surfaces. These surfaces are now based on more detailed information and better predicts variability in depth and thickness. It is possible that during operations these surfaces will change and therefore the resulting wall slopes may vary, particularly through the weathered zones;
- Due to the multiple pit mining operation, careful selection of a reliable mining contractor will need to be made to ensure strong management of the multiple working faces and fleets. The Owners mining team will need to adequately staffed to manage the multiple working areas and contractors;
- The multiple pit scheduling will require some ore being mined in advance and stockpiled to ensure the ore is available for processing. If this ore is stockpiled at the pits, it may present a potential ore security risk. Therefore, the ore should be transported and stockpiled at the central stockpiling area adjacent to the processing plant. Although this mitigates the security risk of the ore, it will be at the cost of incurring the ore transport costs in advance of the ore being required for feed;
- It was assumed that 20% of the highly weathered oxide material will be free-dig (i.e. not require blasting), which is seen to be practical;
- The interaction of both mining extraction activities and the ore haulage activities from the satellite pits will need to be carefully and systematically managed to minimise any negative impacts on regional communities;
- The large number of satellite pits that require ore transported to the processing facility means there is a potential risk to ore continuity.

1.20 Recommendations

1.20.1 Metallurgical Test Work Recommendations

IMO notes that the detailed metallurgical test work programme was continuing for the Doropo Gold Project under the direction of Centamin to support the design as part of the continuous improvement programme. Ongoing test work is listed below:

- Bulk Carbon-in-Leach (CIL) cyanidation testing of Souwa, Nokpa, Kilosegui, Enioda and Chegue Main Master composites under optimised conditions;
- Dynamic thickener and detoxification test work;
- Pre-Leach & Carbon-in-Leach test work;
- Carbon-in-Leach (CIL) cyanidation testing of all Variability composites under optimised conditions;

- Equilibrium carbon loading tests on all Master composites;
- Rheology test work has recently been reported and currently under review. Initial results indicate that 55% and 60% solids are a viable option;
- Oxygen demand tests; and
- Diagnostic leach tests on CIL leach tails.

GRES recommends the completed FS metallurgical test work be reviewed and used to supplement existing data in the next design phase.

Additionally, GRES recommends the testing discussed below be undertaken, to supplement the plant design data set.

- Collection and analysis of site water;
- Leach cyanidation on the FS master composites, at optimised CIL conditions using site water;
- Settling tests on leached CIL residues focused on flocculant screening, flocculant dosing and if necessary, coagulant dose optimisation;
- Further mineralogy testing to distinguish between pyrite and pyrrhotite;
- Diagnostic leaching on CIL leach residues, and
- Detoxification testwork to further define reagent consumption.

Grade Recovery Curves

The current grade-recovery curves were prepared from limited variability results from the FS leach test campaign. The regression analysis confidence can be improved by addition of the full FS master and variability composite leach results.

Reduce Cyanide Consumption and Improve Leach Kinetics

The results from the shear reactor tests conducted in the FS programme were inconclusive. Further tests at optimum cyanide concentrations may identify reductions in cyanide consumption. The tests conducted at optimised conditions may also report higher overall gold recovery. Lower cyanide requirements will also translate to lower cyanide detoxification reagent costs.

1.20.2 Mineral Resource Recommendations

Cube recommends the following with respect to the Doropo Gold Project:

- Centamin should give consideration to constructing a geo-metallurgical model to support future studies;

- Consideration should be given to pre-production work, such as the conversion of the first two years of Mineral Resources in the production profile into Measured Resources to de-risk the production profile. This would include drilling the starter pits for Souwa, Nokpa and Kilosegui. Consideration could then be given to how to best set up a fit-for-purpose mine geology system.

1.20.3 Mineral Reserve Recommendations

Orelogy recommends the following with respect to the Doropo Gold Project.

- Mining selectivity and ore definition will need be mitigated through the use of industry standard ore control techniques such as:
  - In-advance and detailed RC grade control drilling and modelling;
  - Digital ore mark-out combined with ore spotting where required;
  - The use of smaller excavators and blast balls are recommended for highly selective zones;
  - A robust mining contractor tender process is required in the next phase to ensure the eventual contractor selection has the capacity, capability and experience to undertake the work to the required complexity and quality.

TABLE OF CONTENTS

2.	INTRODUCTION	1
2.1	Issuer	1
2.2	Scope and Terms of Reference	1
2.3	Sources of Information	2
2.4	Contributing Consultants	2
2.5	Risks and Forward-looking Statements	3
2.6	Qualified Persons	3
2.7	Site Visits and Purpose	4

LIST OF TABLES

Table 2.1	Contributing Consultants to the FS Report	3
Table 2.2	Qualified Persons	4
Table 2.3	Site Visits by QPs	4

2. INTRODUCTION

2.1 Issuer

Centamin plc ('Centamin' or 'the Company') commissioned GR Engineering Services Limited (GR Engineering), in conjunction with Cube Consulting Pty Ltd (Cube), Orelogy Consulting Pty Ltd (Orelogy), Knight Piésold Pty Ltd (Knight Piésold), SRK Consulting (UK) Ltd (SRK) and Independent Metallurgical Operations Pty Ltd (IMO), to compile a Technical Report on the Doropo Gold Project ('the Project' or 'the Property') located in the Bounkani Region, north-eastern Côte d'Ivoire. This Report is compiled recognising the disclosure and reporting requirements set forth in NI 43-101.

Centamin is a public company whose shares trade on both the London Stock Exchange (LSX) under the ticker CEY, and on the Toronto Stock Exchange (TSX) under the ticker CEE. The Company is headquartered in St Helier, Jersey.

2.2 Scope and Terms of Reference

This Feasibility Study (FS) Technical Report includes the additional drill data from the Property and the metallurgical test work required to support the FS. Pit and infrastructure geotechnical drilling, logging and sampling was also completed.

The Mineral Resource Estimate (MRE) work was undertaken by Cube in stages between February 2023 and August 2023, as drill assay data became available from successive prospects in the Project area. The prospects for which Mineral Resources were updated are Attire (ATI), Enioda (ENI), Chegue Main, Chegue South (CHG), Han (HAN), Hinda (HND), Hinda South (HNS), Kekeda (KEK), Kilosegui (KLG), Nare (NAR), Nokpa (NOK), Sanboyoro (SAN), Solo (SOL), Souwa (SWA), Tchouahinin (THN), and Vako (VAK).

Cube undertook the following general tasks as part of the MRE update:

- Modelling of weathering/oxidation and mineralisation wireframe models;
- An independent review of the Centamin Quality Assurance-Quality Control (QAQC) processes and outcomes;
- Validation of the drill database;
- Statistical and geostatistical analysis of the supplied assay and density data to establish estimation domains and interpolation parameters. The Au grade variable was estimated using geostatistical interpolation. Bulk dry density was assigned following a statistical analysis by relevant domains;
- Classification of the MRE.

Centamin, GR Engineering Services, Cube, SRK, IMO, Knight Piésold and Orelogy are co-authors of this Technical Report. The individuals responsible for the various sections of the report are provided in Table 2.1, and by virtue of their education, experience and professional association are considered Qualified Persons (QPs) as defined by NI 43-101 for this Report.

This Technical Report details the work undertaken to complete the FS and documents the results of the work.

This Technical Report is not independent of the Issuer and has been prepared in accordance with the disclosure requirements of the Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

2.3 Sources of Information

GR Engineering Services, Cube, SRK, IMO, Knight Piésold and Orelogy have undertaken the work based mostly on data provided by Centamin, but also on public domain information and third-party Technical Reports and relevant published and unpublished third- party information (see references in Section 27).

GR Engineering Services, Cube, SRK, IMO, Knight Piésold and Orelogy have made all reasonable endeavours to confirm the authenticity and completeness of the technical data on which this Report is based; however, GR Engineering Services, Cube, SRK, IMO, Knight Piésold and Orelogy cannot guarantee the authenticity or completeness of such third-party information.

2.4 Contributing Consultants

Consultants who contributed to this Report are as noted below:

FS Component	Specialist Study Field	Consulting Company
Mineral Resource	Mineral Resource estimation	Cube Consulting Pty Ltd
Mine Design	Pit optimization	Orelogy Consulting Pty Ltd
	Mine design	
	Mine scheduling	
	Mineral Reserve estimation	
Metallurgy	Metallurgical test work program	Independent Metallurgical Operations Pty Ltd
Process Design	Recovery methods	GR Engineering Services Limited
	Process plant design	
	Processing capital and operating costs	
Infrastructure Design	Geotechnical assessment for infrastructure	Knight Piésold Pty Ltd
	Facility design and costing	
Environmental and Social Studies	Environmental and Social Studies/Closure Costs	Earth Systems Consulting Pty Ltd

FS Component	Specialist Study Field	Consulting Company
Other Information	Hydrology, hydrogeology, geochemical studies	Piteau Associates Pty Ltd
	High voltage power supply to the Project site	ECG Consulting Pty Ltd
	Pit geotechnical design and study	SRK Consulting (United Kingdom) Pty Ltd

Table 2.1 Contributing Consultants to the FS Report

2.5 Risks and Forward-looking Statements

The business of mining and mineral exploration, development and production by its nature has significant operational risks. The business depends upon, amongst other things, successful prospecting programs, and competent management. Profitability and asset values can be affected by unforeseen changes in operating circumstances and by technical issues.

Factors such as political and industrial disruption, currency fluctuation and interest rates could have an impact on the proposed Project's future operations, and potential revenue streams can also be affected by these factors. The majority of these factors are, and will be, beyond the control of Centamin or any other operating entity.

This Technical Report contains forward-looking statements. These forward-looking statements are based on the opinions and estimates of Centamin, GR Engineering Services, Cube, SRK, IMO, Knight Piésold and Orelogy at the date the statements were made. The statements are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those anticipated in the forward-looking statements. Factors that could cause such differences include changes in world gold markets, equity markets, costs and supply of materials relevant to the Project, and changes to regulations affecting them.

Although the authors believe the expectations reflected in the forward-looking statements to be reasonable, they do not guarantee future results, levels of activity, performance or achievements.

2.6 Qualified Persons

The QPs responsible for the preparation of this Technical Report and the sections under their responsibility are provided in Table 2.1.

Qualified Person	Position	Employer	Professional Designation	Report Sections
Michael Millad	Principal Geologist	Cube Consulting	MAIG	1.7, 1.9, 1.19.3, 1.20.2, 10, 11, 12, 14, 25.2, 26.1
Flavie Isatelle	Principal Geologist	Cube Consulting	MAIG	1.7, 1.9, 1.19.3, 1.20.2, 10, 11, 12, 14, 25.2, 26.1
Ross Cheyne	Principal Consultant	Orelogy	FAusIMM	1.10, 1.11, 1.19.4, 1.20.3, 15, 16, 18.9, 21.2.2, 21.3.2, 21.5, 25.3, 26.2

Qualified Person	Position	Employer	Professional Designation	Report Sections
David Morgan	Director	Knight Piesold	MIEAust	1.13, 1.16.2, 18.2, 18.4, 18.5, 18.6, 25.5, 26.5
Grant Harding	Principal Process Consultant	Independent Metallurgical Operations	FAusIMM	1.8, 1.19.2, 1.20.1, 13, 25.1, 26.3
Deepak Malhotra	Process Consultant	GR Engineering Services	CIMM	1.12, 1.16.1, 1.16.3, 1.20.1, 17, 18.1, 18.7.2, 18.8, 18.10, 18.11, 18.12, 18.15, 21.1, 21.2.1, 21.2.3, 21.3, 21.4, 21.6, 21.8, 24.4, 24.5, 24.6, 24.8, 25.4, 25.7, 26.3
Craig Barker	Group Mineral Resource Manager	Centamin	FAIG	1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.14, 1.15, 1.16, 1.17, 1.18, 1.19.1, 2, 3, 4, 5, 6, 7, 8, 9, 18.3, 18.7.1,18.13, 18.14, 19, 20, 21.2.4, 21.3.22, 21.7, 22, 23, 24.1, 24.2, 24.3, 24.7, 24.9, 25.5.12, 25.6, 25.7, 26.4, 26.5, 26.6, 27
Samson Tims	Senior Consultant (Rock Mechanics)	SRK	P.Geo. - EGBC and APEGA	1.11.2, 16.3

Table 2.2 Qualified Persons

2.7 Site Visits and Purpose

Site visits to the Property were undertaken by the QPs indicated in Table 2.2

Qualified Person	Employer	Dates
Michael Millad	Cube	29-30 August 2021
David Morgan	Knight Piésold	29-31 January 2022
Craig Barker	Centamin	29-30 August 2021
Samson Tims	SRK	11-16 October 2023

Table 2.3 Site Visits by QPs

Mr Michael Millad of Cube, and Mr Craig Barker of Centamin visited the Property on the 29 and 30 August 2021, while the primary assay laboratory, Bureau Veritas (Abidjan) was visited on 4 September 2021.

The following activities and inspections were undertaken:

- View selected drill cores and discuss geological framework and mineralisation controls;

- Discuss and note data capture, storage and management processes;
- QAQC and sampling discussion - procedures and processes;
- Tour of facilities, including site of the under-construction Danoa sample preparation building;
- View outcrops and orpaillage workings in the prospecting area;
- Observe current drilling activities, including drilling methods, sample collection and sample security procedures;
- Independently check several drill collar positions and azimuth/inclinations;
- Observation and direction of the collection of independent drill samples;
- Inspection of the sample storage, sample preparation and sample analysis sections of the Bureau Veritas (Abidjan) laboratory.

The facilities and equipment were considered fit for purpose, and the procedures were well-designed and being implemented consistently. The sample preparation and analytical laboratories were well equipped and operated to a high standard, with the only concerning issue noted being the over-crowded sample storage area at Bureau Veritas in Abidjan, which was explained to be due to late collection of sample residue by various mining companies. A few minor issues were noted within the sample preparation area, but the overall impression was of a well-run and sufficiently clean environment.

In the Qualified Person's opinion, the methods, procedures and processes for sample collection and analysis are fit-for-purpose and the geology of the deposits are well understood, ensuring the suitability of the Project data for Mineral Resource estimation.

Mr David Morgan conducted a site visit from the 29 to 31 January 2022 to inspect the Property and, in particular, the sites for the key infrastructure. In addition, typical core was inspected and discussions were held with the site exploration team regarding the ore characteristics, hanging and foot wall attributes as well as sterilisation drilling progress.

Mr Sam Tims conducted a 2-day site visit between 11 and 16 October 2023 to inspect the Property and, in particular, the core was inspected and discussions were held with the site exploration team regarding the geological logging procedures and the material characteristics. All pits were visited except for Enioda and Han due to security constraints.

Ms Flavie Isatelle, Mr Grant Harding, Mr Deepak Malhotra and Mr Ross Cheyne did not visit the Doropo Gold Project property that is the subject of the Technical Report, as it was not required for the purpose of this mandate.

TABLE OF CONTENTS

3. RELIANCE ON OTHER EXPERTS 1

3. RELIANCE ON OTHER EXPERTS

This Technical Report was prepared by GR Engineering Services, Independent Metallurgical Operations, Cube Consulting, Knight Piesold, Orelogy Consulting, SRK Consulting and Centamin under the supervision of the authors of the Technical Report who are qualified persons (QPs) pursuant to NI 43-101 for Centamin. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QPs at the time of this Report;
- Assumptions and conditions as set forth in this Report;
- Data, reports, and opinions supplied by Centamin and other third-party sources referred to in the References section of this Report.

In preparing this Report, the QPs have fully relied upon certain work, opinions and statements of experts concerning legal, political, environmental, or tax matters relating to the Project. The authors consider the reliance on other experts as being reasonable based on their knowledge, experience, and qualifications.

The QPs believe the information provided by the third-parties to be reliable, but cannot guarantee the accuracy of conclusions, opinions or estimates that rely on such third-party sources for information that is outside their area of technical expertise. This Report is intended to be used by Centamin as a Technical Report for Canadian securities regulatory authorities pursuant to applicable Canadian provincial securities laws.

The QPs have fully relied upon, and disclaim responsibility for, information derived from Centamin and experts retained by Centamin for information in Section 20 related to:

- Environmental, Social and Permitting Studies., reference documentation as follows:
  - Environmental and Social Impact Assessment for the Doropo Gold Project Definitive Feasibility Study, DOROPO2339 Doropo ESIA Report, prepared by Earth Systems;
  - Environment and Social Management and Monitoring Plan for the Doropo Gold Project, DOROPO2339 ESMMP - May 2024, prepared by Earth Systems.

For the purpose of this Report Cube Consulting, Knight Piesold, GR Engineering Services, Independent Metallurgical Operations, SRK Consulting and Orelogy have relied on ownership information in and other local knowledge provided by Centamin as described in Section 4 of this Technical Report. Cube Consulting, Knight Piesold, GR Engineering Services, Independent Metallurgical Operations, SRK Consulting and Orelogy have no independent information regarding property title or mineral rights for the Doropo Gold Project and expresses no opinion as to the ownership status of the property.

Except for the purposes legislated under Canadian or other securities laws, any use of this Report by any third-party is at that party's sole risk.

The major components of this FS comprise of:

- Resource modelling based on available data;
- Mine design and production scheduling;
- Mining cost estimation; metallurgical test work;
- Process design and process plant cost estimation;
- Environmental assessment;
- Financial analysis and other supporting studies on geology;
- Hydrogeology;
- Hydrology;
- Geochemistry;
- Pit geotechnical studies for pit slope design and geotechnical engineering.

TABLE OF CONTENTS

4. PROPERTY DESCRIPTION AND LOCATION 1
4.1 Location 1
4.2 Tenure 2
4.3 Datum and Projection 2
4.4 Royalties 3
4.5 Environmental Liabilities 3
4.6 Permitting 3
4.7 Other Significant Factors and Risks 3

LIST OF FIGURES

Figure 4.1 Location of the Doropo Project - Map of Côte d'Ivoire 1

LIST OF TABLES

Table 4.1 Summary of the Exploration Permits - as of June 2023 2

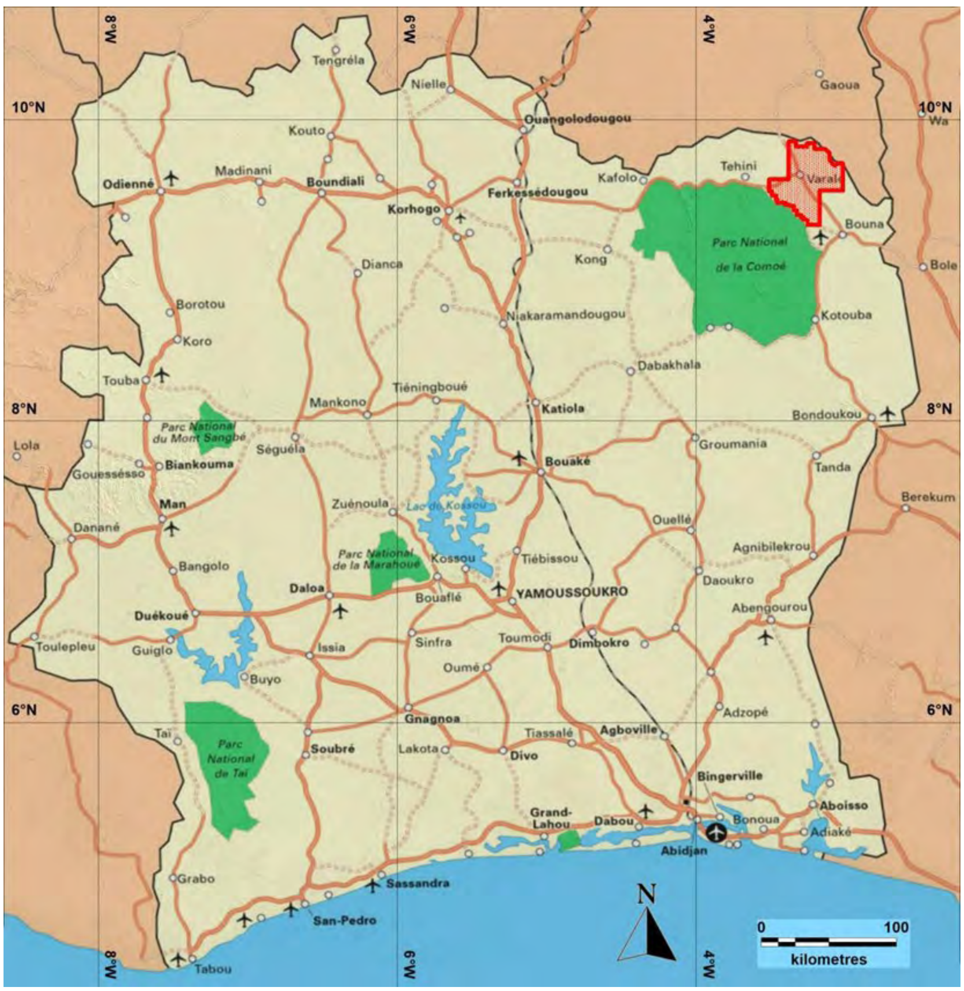
4. PROPERTY DESCRIPTION AND LOCATION

Much of the content in this section is sourced from Osborn (2019) and Centamin (2021), and has been updated only where necessary.

4.1 Location

The Doropo Project is located in north-eastern Côte d'Ivoire, in the Bounkani region, 480 km north of the capital Abidjan and 50 km north of the city of Bouna. The block of exploration permits lies between the border with Burkina Faso and the Comoe National Park (Figure 4.1).

The Mineral Resource area, which fits in a 30 km radius, is centred on about UTM 482,450 mE and 1,074,951 mN, otherwise Latitude 9°43'28'' N and Longitude 3°9'36'' W.



Source: Centamin, 2021

Figure 4.1 Location of the Doropo Project - Map of Côte d'Ivoire

4.2 Tenure

The block of permits includes seven granted exploration permits, all covering granitic rocks. Ampella Mining Côte d'Ivoire (AMCI) and Ampella Mining Exploration Côte d'Ivoire (AMEXCI), both 100% owned Ivorian subsidiaries of Centamin, own these permits, as detailed in Table 4.1. The block of permits covers a total area of 1,847 km^2.

The Mineral Resources reported in this Report are located in five out of the seven exploration permits. All of the exploration permits are subject to the 2014 Ivorian Mining Code.

The 2023 MRE update considered sixteen prospects/deposits namely: Souwa (SWA), Nokpa (NOK), Chegue Main and Chegue South (CHG), Kekada (KEK), Han (HAN), Enioda (ENI), Kilosegui (KILO), Attire (ATI), Hinda (HND), Hinda South (HNDS), Nare (NAR), Sanboyoro (SAN), Solo (SOL), Tchouahinin (THN) and Vako (VAK).

Permit Name	Permit ID	Surface (km^2)	Status	Company	Date Granted	Expiry Date
Varale	PR 335	284.9	Granted	Ampella Mining Côte d'Ivoire S.A.	13.06.2013	12.06.2024
Kalamon*	PR 334	398.9	Granted	Ampella Mining Côte d'Ivoire S.A.	13.06.2013	12.06.2024
Danoa*	PR 559	240.3	Granted	Ampella Mining Côte d'Ivoire S.A.	10.06.2015	09.06.2025
Tehini 1*	PR 535	253.0	Pending	Ampella Mining Exploration C.I. S.A.	08.03.2017	07.03.2024
Tehini 2*	PR 536	228.0	Pending	Ampella Mining Exploration C.I. S.A.	01.03.2017	29.02.2024
Tehini 3	PR 778	241.0	Granted	Ampella Mining Exploration C.I. S.A.	16.04.2018	15.05.2025
Gogo**	PR 633	201.93	Granted	Ampella Mining Exploration C.I. S.A.	19.10.2016	18.10.2023

**These permits host the estimated Mineral Resources and Mineral Reserves reported in this document.*
***Gogo was relinquished in July 2023*
NOTE: The relevant permits were under renewal for the Exploitation Permit application process at the date of this Technical Report

Table 4.1 Summary of the Exploration Permits - as of May 2024

4.3 Datum and Projection

The coordinate system utilised for the Project is the Universal Transverse Mercator (UTM) projection, WGS84, zone 30 north. It is centred on about UTM 475,000 mE and 1,068,000 mN, otherwise Latitude 9°39'42'' N and Longitude 3°13'40'' W.

4.4 Royalties

Royalties payable to the Côte d'Ivoire government are calculated on a sliding scale from 3% to 6% based on the gold price at the time of calculation.

The corporate income tax is payable at a rate of 25% to the Côte d'Ivoire Government.

Centamin has not entered into any contracts for mine development, mining, processing, transportation, handling, sales and hedging, or forward sales contracts or arrangements. The Qualified Person recommends continued discussions with 3rd parties and the establishment of tender documents in anticipation of obtaining the mining permit for the Project.

4.5 Environmental Liabilities

Compensation for crop destruction is paid to local communities. These compensations are paid according to the guidelines set by the Ministry of Agriculture directly to the landowners.

4.6 Permitting

Each presidential decree sets minimum expenditure requirements and type of work by year in order to maintain the rights on the exploration permits. The total expenditures, the work achieved and the results are summarised in bi-annual and annual reporting to the Directorate General of Mines. Regular field visits are conducted by representatives of the Directorate General of Mines in order to reconcile the reports.

The exploration activities, including the drilling, need no other specific permitting in the field other than the aforementioned compensation for crop destruction to the local communities.

4.7 Other Significant Factors and Risks

Environmental, permitting, legal, title, taxation, socio-economic, marketing, security and political or other relevant issues could potentially materially affect access, title or the right or ability to perform work on the Property. However, as of the Effective Date of this Report, the Qualified Person(s) is unaware of any such potential issues affecting the Property.

TABLE OF CONTENTS

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 1
5.1 Accessibility 1
5.2 Physiography 1
5.3 Climate 3
5.4 Site Characteristics 4
5.5 Local Resources and Infrastructure 5

LIST OF FIGURES

Figure 5.1 Elevations Over the Project Area - SRTM Data 2
Figure 5.2 Main Vegetation Zones in West Africa 3

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Much of the content in this section is sourced from Osborn (2019) and Centamin (2021), and has been updated only where necessary.

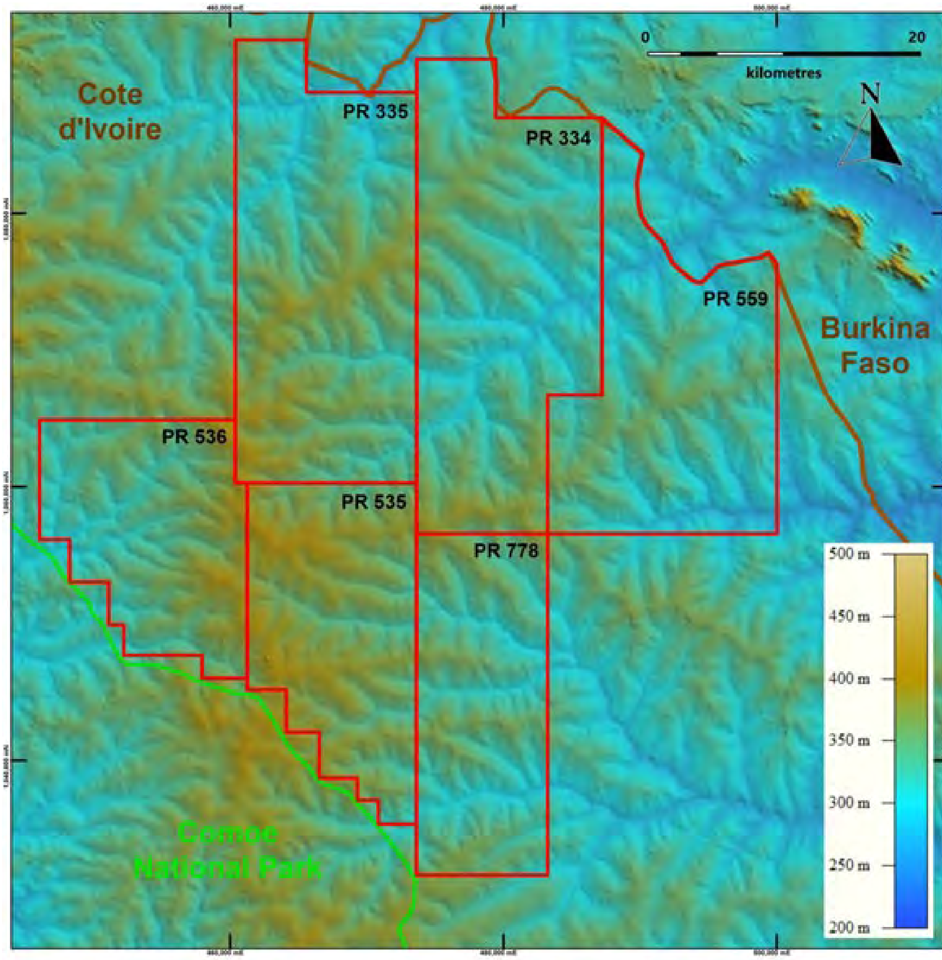
5.1 Accessibility

The Project area is accessible by a national sealed road called the A1 which crosses through the centre of the Project. The A1 is a major road that links Abidjan and Ouagadougou, the capitals of and Côte d'Ivoire and Burkina Faso, respectively. Doropo prefecture is 76 km (about 1.5 hours' drive) from Bouna, the Capital of the Bounkani Region. It is also 240 km (about 3.5 hours' drive) from Bondoukou, the capital of the district and 645 km (about 10 to 11 hours' drive) from Abidjan, the economic capital. A dense network of small dirt/sandy roads allows easy access to all parts of the Project, even during the wet season. The sandy nature of the soil allows a rapid drainage of the water on the access roads generally.

5.2 Physiography

The Doropo area has relatively subdued relief, due to the nature of the underlying rocks - the granites. The surface soils are mostly sandy and outcrops are rare. The ridges form small plateaus and are covered by laterites and occasionally duricrust of limited thickness. Large peneplains bound the area on the north and south, while hill chains bound the eastern and the western sides where greenstone belts crop out (the Tehini-Hounde belt on the West and the Bonomo/Batie belt on the East).

Elevations range from about 250 m to 407 m at the highest point, which is more or less in the middle of the Doropo Project, and forms a drainage divide between the Volta Noire basin on the East and the Comoe basin on the West as shown in Figure 5.1. Streams and rivers on the Project are seasonal.



Source: Centamin, 2021

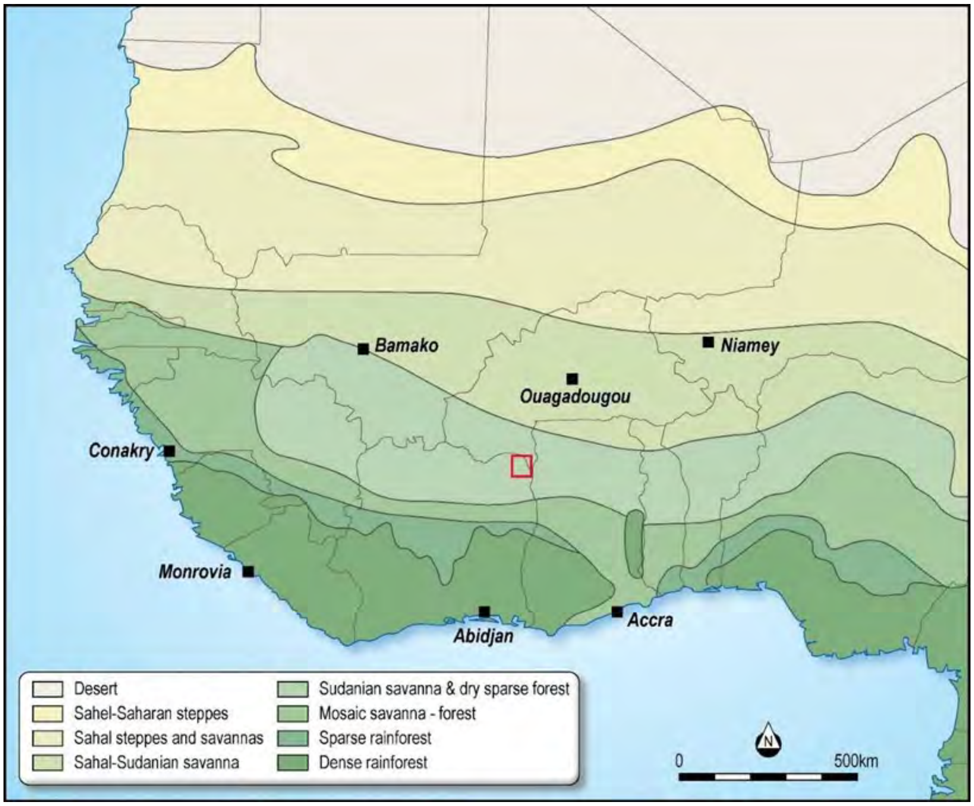
Figure 5.1 Elevations Over the Project Area - SRTM Data

The vegetation is characterised by the sparse forests and savannah where natural environment exists, as shown in Figure 5.2. However, a large extent of ground is covered by seasonal crops, mostly yams, peanuts, rice, millet and sorghum and plantations of cashew trees - Côte D'Ivoire is one of the main producers of cashew nuts in the world.

The National Comoe Park limits the Project all along its south-west side. The park covers 11,500 km^2, which is the largest protected area in West Africa. It is a biosphere reserve and a UNESCO world heritage site since 1983.

There is a dense network of rural villages in the area of the Project, mostly populated by the ethnic group of Lobi. Bigger villages, such as (but not only) Danoa, Kalamon, Kodo, Varale, Niamoin, are mostly populated by the Koulango ethnic group. The third ethnic group present in the area is the Fula, who are often nomads, living from cattle farming.

The main economic activity is represented by rural agriculture and farming. However, for several years, mostly since the civil war times, some villagers also live from artisanal gold mining (mostly superficial rocks digging and laterite panning). To some extent, the illegal mining increased more recently with the arrival of nomadic Burkinabes (Mossi and Dioula ethnic groups mainly).



Source: Centamin, 2021

Figure 5.2 Main Vegetation Zones in West Africa

5.3 Climate

The climate is of Sudanese type, with two distinct seasons, a rainy season and a dry season. The rainy season extends from May/June to September/October when rainfall totals between 1,100 mm and 1,200 mm. The dry season extends from September/October to May/June. The Harmattan, a hot dry wind coming from the Sahara regions, generally blows in December and January, sometimes extending to March, and brings dust clouds, which reduce visibility.

The average annual temperature is 28°C, ranging between 21°C and 33°C. The hottest times of the year occur at the change of seasons.

5.4 Site Characteristics

The site characteristics for the Doropo Gold Project (Doropo) are summarised below:

- A site-specific climate assessment was completed for Doropo;
- Daily precipitation records (1944 to 2000) from the Batie meteorological station (34 km north east of the project site) were used for the short-duration climatic analysis and summed to produce monthly and annual totals for long-duration climatic analysis;
- September is the wettest month of the year and appreciable rainfall starts (on average) in April and finishes in October;
- The wet season typically lasts for 4 months out of each year, starting in July;
- The average annual rainfall for the project area is 1,133 mm;
- The maximum recorded annual rainfall for Batie was 1,838 mm (in 1968);
- The minimum recorded annual rainfall for Batie was 718 mm (in 1948);
- The 10-year ARI wet season duration is the period in which 70% of the annual rainfall occurs on average. For Doropo, this period was determined to be 120 days. The magnitude of the design wet season is 1,065 mm;
- Daily pan evaporation values were found for Batie and Gaoua weather stations, approximately 34 km to the north east and 67.3 km to the north of the site. The average annual lake evaporation for the project area used for water balance modelling is 1,469 mm;
- Design annual rainfall sequences, determined by analysing the Batie rainfall data, are provided in Table 5.1. The 100-year ARI, wet annual rainfall total is 1,931 mm. The 100-year ARI, dry annual rainfall total is 704 mm.

Month	Average Rainfall (mm)	100 ARI Wet Annual Rainfall (mm)	100 ARI Dry Annual Rainfall (mm)	Average Pan Evaporation (mm)	Average Lake Evaporation (mm)
January	0	0	0	199	143
February	7	56	0	210	148
March	0	93	48	234	164
April	85	149	133	204	146
May	121	189	69	189	136
June	64	258	100	152	111
July	229	366	79	128	94
August	267	303	111	111	82
September	223	394	77	121	89
October	84	83	88	146	107

Month	Average Rainfall (mm)	100 ARI Wet Annual Rainfall (mm)	100 ARI Dry Annual Rainfall (mm)	Average Pan Evaporation (mm)	Average Lake Evaporation (mm)
November	2	26	0	162	118
December	50	14	0	181	131
Total	1,133	1,931	704	2,037	1,469

Table 5.1 Design rainfall and evaporation annual sequences

- Short-term storm event magnitudes for the project area were calculated using published methods. The design 100-year ARI, 72-hour duration storm event magnitude is 220 mm;
- The design Probable Maximum Precipitation (PMP), 24-hour duration storm event is 602 mm;
- A probabilistic seismic hazard analysis was carried out to determine appropriate seismic design parameters for Doropo. The analysis was carried out using seismic hazard model developed using the computer program Open Quake (Ref. 15). It is recommended that the 1,000-year ARI earthquake be adopted as the Operating Basis Earthquake (OBE) for Doropo. The estimates peak ground acceleration (PGA) for the 1,000-year earthquake was calculated as 0.006 g for site class – Rock Site and 0.007 g for site class – Class C;
- Due to the ANCOLD Consequence Category (Ref. 14) of the Doropo TSF, it is recommended that the 5,000-year ARI earthquake be adopted as the Safety Evaluation Earthquake (SEE) for Doropo. The estimated peak ground acceleration (PGA) for the 5,000-year earthquake was calculated as 0.014 g for site class – Rock Site and 0.017 g for site class – Class C.

5.5 Local Resources and Infrastructure

Local infrastructure remains limited so the Project development will have to include a self-sufficient aspect or backup options.

The company La Société des Energies de Côte d'Ivoire (CI-ENERGIES) own the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government. The 90 kV high voltage grid supply at Bouna should be of good quality and reliable. The recently constructed 90 kV transmission line from Bondoukou to Bogofa and Bouna is lightly loaded. The supply at Bondoukou is from the 225 kV network and is part of a ring-main system which is very reliable and based on the HV power consultant risk assessment, it is not expected that a full back-up power station is required as part of the grid connection works.

The mobile phone network is well deployed, from at least two main national providers. Internet access has overall proven reliable, via the general 3G and 4G mobile connections, or dedicated microwave connections for the sites.

Underground water is not abundant in the Project area and hence hydrological studies completed have indicated that water harvesting via a water harvest dam and distribution pumps is required is supply the required water demand of the Project. A suitably sized water storage dam has been designed to cater for long term supply of water to the Project.

Due to the rural aspect of the area, the specialised professional skills and trade skills are very limited in the near vicinity, but adequate workforces are available from elsewhere in the country.

TABLE OF CONTENTS

6. HISTORY 1

6. HISTORY

Much of the content in this section is sourced from Osborn (2019) and Centamin (2021), and has been updated only where necessary.

The first exploration permits of the area were granted to Ampella Mining Côte d'Ivoire, Ivoirian subsidiary, in June 2013. Prior to that time, no systematic mineral exploration had ever been conducted in the area.

The region (the north-eastern part of the country) was first mapped by French Geologists from 1950 to 1958, in order to produce the first Geological map at the scale 1:500,000, prepared by the Bureau de Recherche Geologique et Minière (BRGM), printed in 1963.

Some evidence of historical gold mining during the Colonial times (under the French management) are seen at Varale, where a small open pit type operation occurred, most likely shallow surface workings on outcropping quartz veins. However, this operation seems to not have been documented.

The granitic domain that characterises the Doropo Project had always been considered as not prospective for gold deposits.

Ampella Mining Ltd made application for the Kalamon, Varale and Doropo Ouest permits in 2010 based on the area's close proximity with Ampella's Batie West Project in neighbouring Burkina Faso, and reports of artisanal gold mining (ASM) in the Doropo area. The permits were granted in June 2013, and this was followed by a reconnaissance visit to the Doropo area at the end of 2013 and the identification of several exploration targets based on rock chip sampling of exposures in artisanal pits. The Doropo Ouest permit was later relinquished in August 2019 due to a lack of positive soils and rock chip sample results. The Kalamon and Varale permits cover the area of the Main Cluster of gold deposits. A further 7 exploration permits were granted to Ampella between 2015 and 2018 (Danoa, Gogo, Bouna, Bouna Nord and the 3 Tehini permits). Bouna and Bouna Nord were also relinquished in August 2019 and May 2017, respectively due to the same reasons as Doropo Ouest.

Centamin acquired Ampella Mining Côte d'Ivoire via the takeover of Ampella Mining Ltd. in March 2014. Exploration activities then started on the Doropo Project from mid-2014.

TABLE OF CONTENTS

7. GEOLOGICAL SETTING AND MINERALISATION 1
7.1 Regional Scale Geology 1
7.2 Project Scale Geology 4
7.2.1 Lithology 4
7.2.2 Structure and Mineralisation 5
7.2.3 Alteration and Mineralisation 6
7.2.4 Weathering Profiles 9
7.2.5 Genesis 12

LIST OF FIGURES

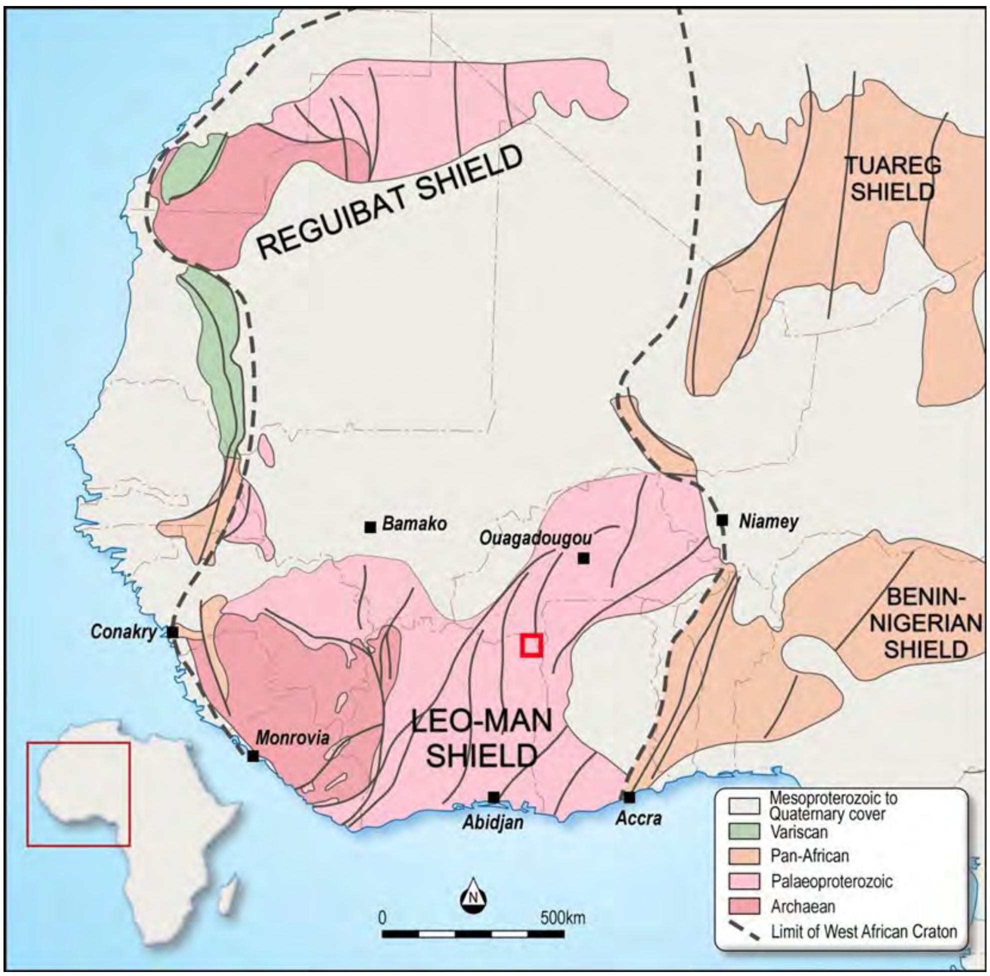
Figure 7.1 Map of West African Craton 1
Figure 7.2 Geology of the Leo-Man Shield - from the BRGM Interpretations 2
Figure 7.3 Geology Map of Doropo 3
Figure 7.4 Deposit Scale Geology and Prospect Locations 5
Figure 7.5 Distal Epidote-Chlorite-Weak Hematite Alteration on Granodiorite (from Footwall Zone at Souwa - DPDD1382 @ 103.4 m Depth) 7
Figure 7.6 Distal Alteration: Haematite-Chlorite Pervasive Alteration (from Direct Footwall of Mineralisation at Souwa - DPRC0504 @ 157 m Depth) 7
Figure 7.7 Proximal Intense Sericite and Fine Grained Disseminated Pyrite Alteration (from the Han Deposit - DPRD0470 @ 94 m Depth) 8
Figure 7.8 Strong Silica-Sericite Alteration - High Grade Mineralisation (from the Han Deposit - DPRD0470 @ 98.7 m Depth) 8
Figure 7.9 Two Examples of Doropo Saprolite Showing Quartz Vein Fragments in the Right Hand Photo 10
Figure 7.10 Saprock (Original Fabric and Mineralogy is Partially Preserved) 10
Figure 7.11 Saprock 11
Figure 7.12 Transition Zone Between Lower Saprolite and Fresh Rock 11
Figure 7.13 Fresh Granodiorite 12
Figure 7.14 Souwa Prospect Geological History 14

7. GEOLOGICAL SETTING AND MINERALISATION

Much of the content in this section is sourced from Osborn (2019) and Centamin (2021), and has been updated only where necessary.

7.1 Regional Scale Geology

The West African craton covers a surface area of 4.5 million km^2, extending from the northern portion of Mauritania in the north, to the southernmost West African countries of Liberia, Côte d'Ivoire and Ghana in the south. It crops out in two major areas, the Reguibat shield in the north and the Leo-Man shield in the south, as shown in Figure 7.1. The Leo-Man shield includes the major gold producing provinces in Ghana, Burkina Faso, Southern Mali, Guinea and Côte d'Ivoire.



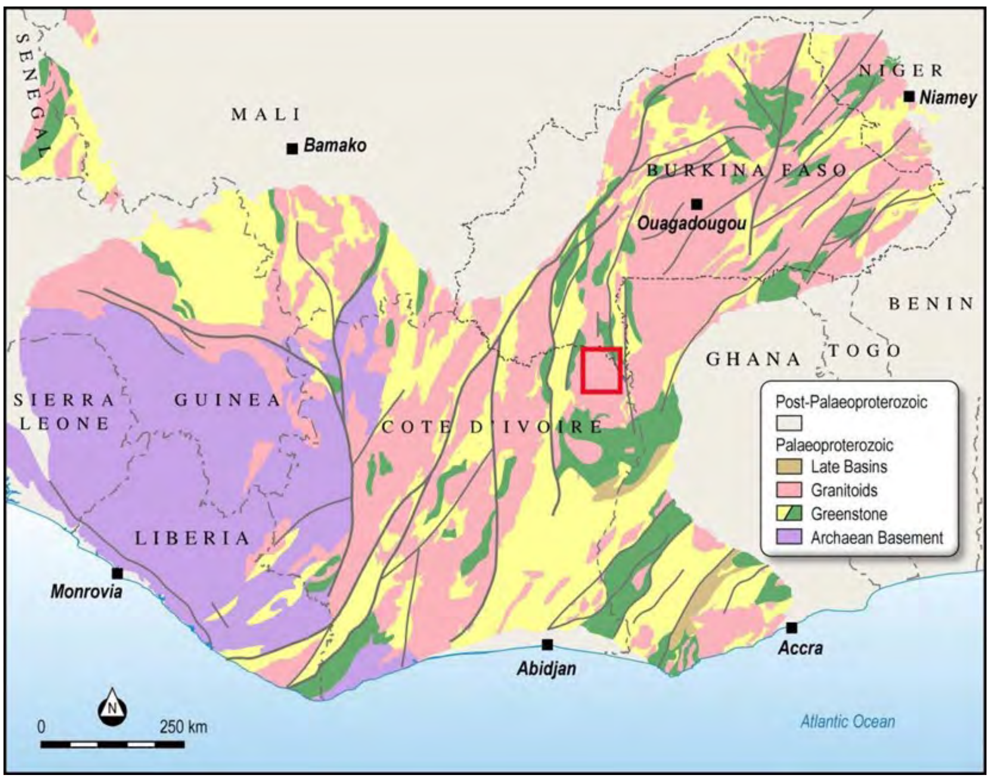
Source: Centamin, 2021

Figure 7.1 Map of West African Craton

In the Leo-Man shield, shown in Figure 7.1, Paleoproterozoic rocks, known as the 'Birimian domain' are tectonically juxtaposed to the Archaean basement, separated by the Sassandra fault. The gold deposits largely lie within the Birimian domain, which covers about 85% of the Côte d'Ivoire ground.

The structure within the Birimian domain was formed during the Eburnean megacycle between 2.5 to 1.6 billion years ago and the main tectono-metamorphism events occurred between 2.2 to 2.0 billion years ago. This Paleoproterozoic domain includes greenstone belts (volcano-sediments) bounded by large areas of tonalitic granite-gneiss, trondhjémite and granodiorite (TTG orthogneiss suite, Tonalite- Trondhjemite-Granodiorite). Later stages of alkaline and calc-alkaline granitic plutons intrude this rock package.

The post Eburnean deformation events ended with large regional brittle deformation, often of a NW-SE orientation marked by the doleritic dykes.



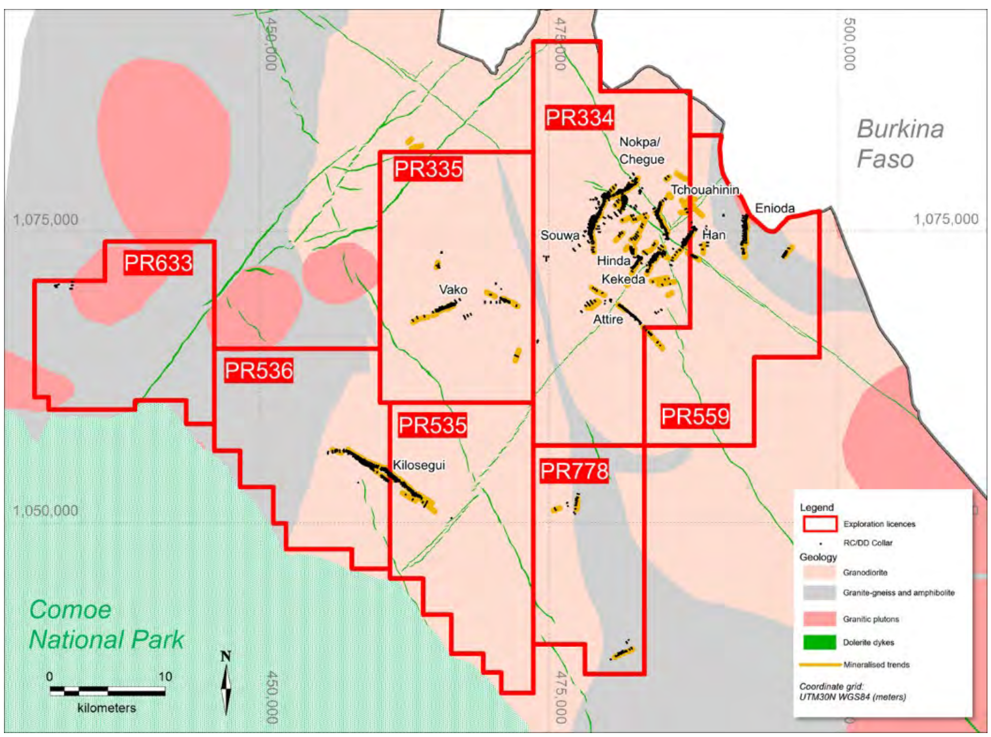
Source: Centamin, 2021

Figure 7.2 Geology of the Leo-Man Shield - from the BRGM Interpretations

The Doropo permits, shown in red in Figure 7.2 and Figure 7.3, lie entirely within the Tonalite-Trondhjemite-Granodiorite (TTG) domain, bounded on its eastern side by the Boromo-Batie greenstone belt, in Burkina Faso, and by the Tehini greenstone belt in the west. At the Project scale, the geology consists of granite-gneiss terrain, the granite being mostly of granodioritic composition. Outcrop is sparse, and generally confined to some slope sides with the flat ridge tops and low-lying areas being covered by lateritic soils and transported sediments (alluvium and colluvium). The transitions with the greenstone belts, on both sides of the granitic domain, span progressive changes in the lithologies, encompassing layers of volcanic rocks (greenstones), as pyroxenites, amphibolites or more generally migmatites (mostly on the western side).

The granites are intruded by an abundant series of pegmatitic veins and quartz veins, ranging from the decimetre scale to several hundreds of metres scale. Some of this veining hosts gold mineralisation, often as primary native gold, across the entire area. This generates regular dispersed gold anomalism in the surface geochemistry; it is also the main source of the gold extracted by the artisanal miners but is mostly uneconomic at the industrial scale.

Large, late doleritic dykes criss-cross the whole domain at the regional scale.



Source: Centamin, 2021

Figure 7.3 Geology Map of Doropo

7.2 Project Scale Geology

7.2.1 Lithology

The MRE update presented in this document covers sixteen deposits named Souwa (SWA), Nokpa (NOK), Chegue Main and Chegue South (CHG), Kekada Kekeda (KEK), Han (HAN), Enioda (ENI), Kilosegui (KILO), Attire (ATI), Hinda (HND), Hinda South (HNDS), Nare (NAR), Sanboyoro (SAN), Solo (SOL), Tchouahinin (THN) and Vako (VAK). Most of the deposits (7) are within a 7 km radius with Enioda ~10 km east of and Kilosegui ~30 km southwest of the main deposit camp.

The main rock types across the Doropo area were initially distinguished based on aeromagnetic, radiometric and soil geochemical attributes into broad domains. However, a Centamin supported PhD research program by Wilfried Digbeu at the Felix Houphouët-Boigny University of Abidjan, will provide further information on the regional scale litho-structural context of the Doropo Project area.

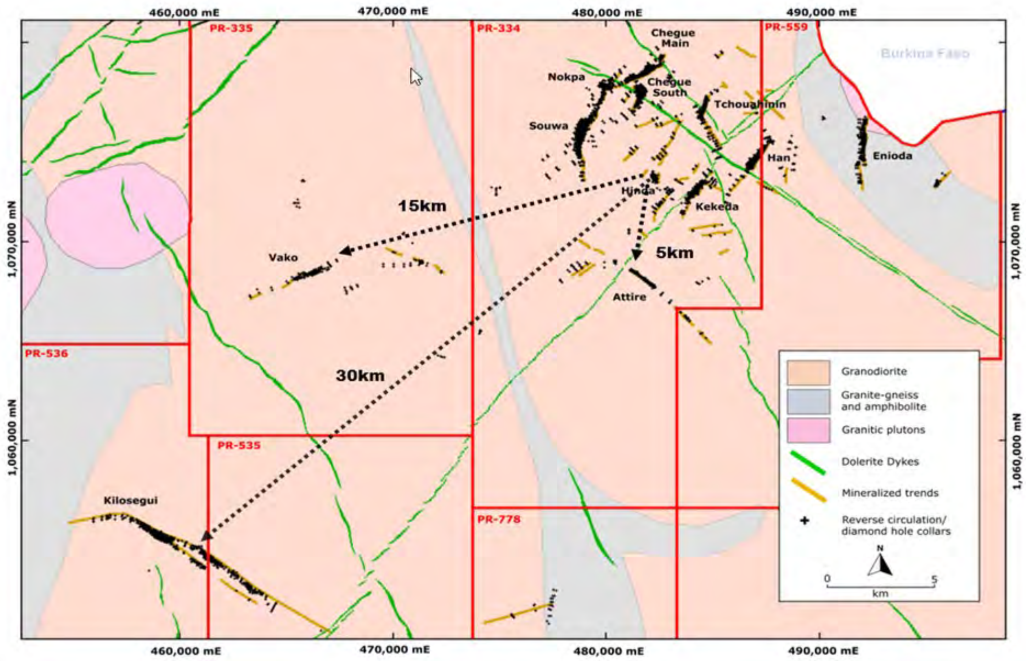
One of the aims of the PhD research is to accurately map, interpret and date the various granitic facies, which comprise biotite granites, granodiorites and tonalites, plus some amphibole enclaves.

The geochemical composition of the granitoid rocks characterise the suite as calc-alkaline and the REE signatures are like Archaean age TTG granitoids.

Age dating of the rocks (U-Pb dating of zircons) suggests a long history of evolving continental crust during the Palaeoproterozoic from 2,450 Ma to 2,100 Ma.

The timing of gold mineralisation has not been determined although if typical of Birimian orogenic gold deposits the gold mineralising events are likely to be late in the magmatic-tectonic evolution of the Doropo terrane, around 2,100 Ma. (Figure 7.4).

The host rocks of the Doropo gold deposits comprise a homogeneous medium to coarse grained granite-granodiorite complex, which has been locally intruded by late-stage gabbro-dolerite dykes (clearly seen in airborne magnetic data) and some pegmatite veins. In addition, there are occasional biotite rich or aplitic dykes, the latter mostly associated with the Enioda deposit, which also shows some amphibolite layers. Outcrop of recognisable rock is rare, but is generally confined to erosional valley sides. Interfluvial ridge lines are often covered by hard laterite cuirasse, while drainage lines are filled with lateritic soils and transported sediments (alluvium, colluvium).



Source: Centamin, 2021

Figure 7.4 Deposit Scale Geology and Prospect Locations

7.2.2 Structure and Mineralisation

Economically interesting mineralisation is associated with discrete structures of intense silica-sericite alteration, focused within and along the margins of narrow (5-10 m wide to locally 20-25 m wide) dextral shear zones. Outside of the mineralised zones, the granodiorite is fairly undeformed.

The planar zones of mineralisation define a great circle on the stereonet with a plunge of 30->295 (excluding Kilosegui). This direction appears to be coincident with the linear shoot directions within the planar zones of mineralisation and can be used to further explore the deposits (e.g. Nokpa).

Even though Kilosegui appears to be in a completely different strike orientation to the planar zones of gold mineralisation at Doropo, the trend could well be related and may have been formed under the same stress conditions. If Kilosegui is completely unrelated to Doropo, the poles of the planar grade continuity would not lie close to the great circle defined by the poles of the Doropo zones. With the inclusion of Kilosegui, the average pole to the great circle that fits through the local planar orientations is 25->266. Further drilling will support or negate this hypothesis.

When comparing the grade distributions with 1 VD magnetic data it is evident that the gold mineralisation is laterally terminated by NE and NW striking late-stage fractures, some of which are intruded by younger dykes. Most of the prospects are not along these trends but are more NNE-SSW striking. Some observed patterns are as follows:

- The eastern extent of CHG North prospect is terminated by a NW-striking transverse fracture;
- CHG South prospect is terminated to the south by NW-striking fracture;
- HAN is terminated to the SW by a NW-striking fracture;
- ATI is terminated to the NW by a NE-striking transverse fracture.

Also, some prospects are parallel to the transverse fractures:

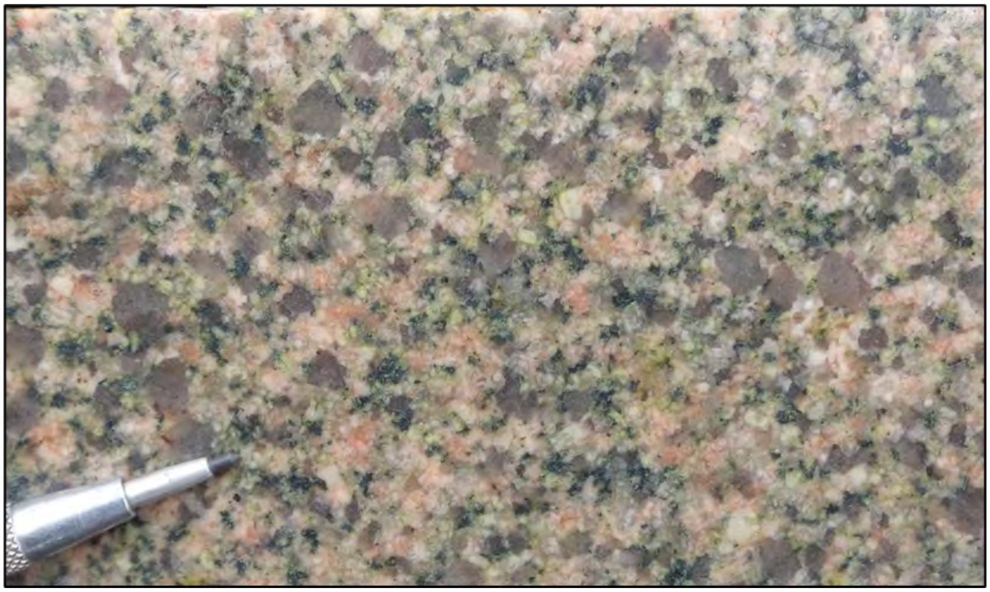
- VAKO appears to run along a NE-striking fracture;
- KLG and ATI are both running parallel to a NW-striking transverse fracture.

These patterns of gold mineralisation are common in Archean belts where the gold is locally continuous along the tectonic grain but are terminated by fractures and faults that clearly post-date and crosscut the tectonic grain. This means that the gold mineralisation itself is very late in the orogenic process. If gold is found to be laterally non-continuous at Doropo, it is highly likely that they are terminating against these late fractures. By recognising this relationship, it would save in drilling costs by drilling wider spaced drill fences on the 'barren' side of these fractures.

7.2.3 Alteration and Mineralisation

The alteration assemblages seen at Doropo from distant to proximal are epidote ± chlorite, chlorite ± magnetite, haematite-silica, silica-sericite-leucoxene then silica flooding ± pyrite ± gold.

In distant regions from mineralised zones, the mineral assemblage is dominated by epidote-chlorite and haematite. Examples of these can be seen in Figure 7.5 and Figure 7.6. Haematite alteration is widespread, ranging from weak to medium intensity. Close to doleritic dykes, haematite alteration intensifies, making it challenging to determine mineralization direction, notably at the Nokpa prospect.



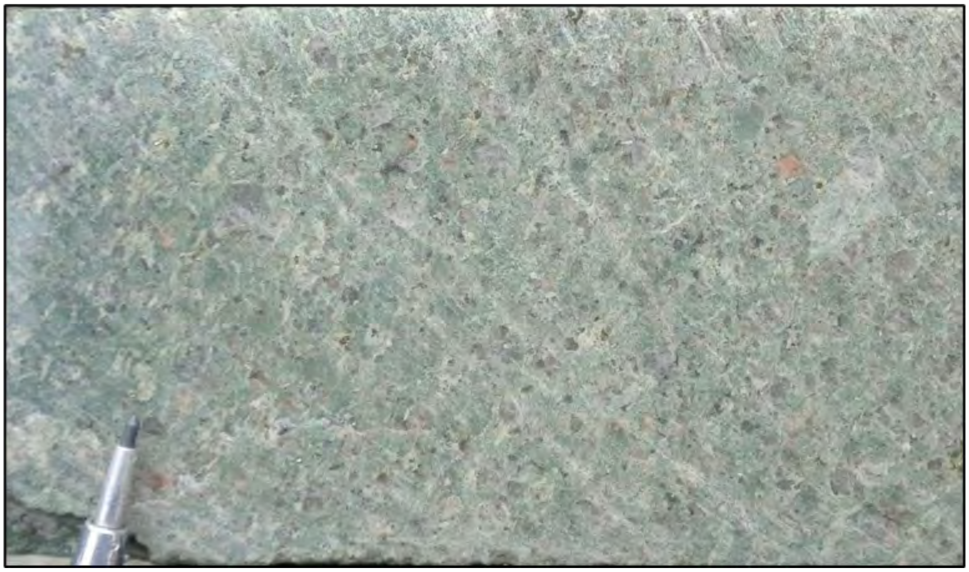
Source: Centamin, 2021

Figure 7.5 Distal Epidote-Chlorite-Weak Hematite Alteration on Granodiorite (from Footwall Zone at Souwa - DPDD1382 @ 103.4 m Depth)

Source: Centamin, 2021

Figure 7.6 Distal Alteration: Haematite-Chlorite Pervasive Alteration (from Direct Footwall of Mineralisation at Souwa - DPRC0504 @ 157 m Depth)

Proximal mineral assemblages, as shown in Figure 7.7 and Figure 7.8, include strong silica-sericite alteration that often overprints earlier haematite and silica alteration. The sulphides, mostly pyrite, are abundant throughout the core of the shear zone; they host part of the gold mineralisation. The other portion of the gold mineralisation occurs as native gold in quartz veins and selvages. Chlorite-magnetite alteration precedes the main pyrite-associated silica-sericite alteration phase.



Source: Centamin, 2021

Figure 7.7 Proximal Intense Sericite and Fine Grained Disseminated Pyrite Alteration (from the Han Deposit - DPRD0470 @ 94 m Depth)

Figure 7.8 shows a photograph of a sheared granodiorite with strong silica-sericite overprinting earlier weak haematite alteration. This interval contains disseminated coarse pyrite and returned a gold assay of 3.9 g/t.

Source: Centamin, 2021

Figure 7.8 Strong Silica-Sericite Alteration - High Grade Mineralisation (from the Han Deposit - DPRD0470 @ 98.7 m Depth)

7.2.4 Weathering Profiles

The weathering profiles encountered across the gold deposits in the Doropo Project area includes a surficial layer or soil profile, which has some degree of transported material but is dominated by sandy granite-derived soils. The surficial layer rests on a mottled or unmottled saprolite layer and a saprock layer which then transitions to fresh rock. The weathered, in situ bedrock is divided into two reasonably distinct material types for the purposes of drill hole logging and modelling of the weathered zone, which overlies fresh granodiorite. The transition from saprock to fresh rock is generally a sharp contact zone.

The characteristics of the constituent parts of the weathered bedrock are as follows:

- Surficial material (soil profile): In situ and transported sandy soils, colluvium and alluvium in drainage lines and limited areas of hard laterite cuirasse on stable interfluvial areas. The soil profiles vary in thickness from 0 m to 5 m with an average thickness (outside the drainage lines) of 2 m. The BOT code (base of transported/soil profile) is used for geological modelling;
- Saprolite: Highly weathered insitu rock (mainly granodiorite), which is reduced to an orange-brown clayey sand. The original granitic texture is barely recognisable but some of the constituent minerals are present, mainly a skeletal framework of quartz grains with some remnant feldspars and clay minerals. The saprolite layer varies in thickness from 2 m to 40 m with an average thickness of 18 m. The BOS code (base of saprolite) is used for the geological modelling;
- Saprock (transition material): Weathered granodiorite which retains it's original rock texture. Most of the constituent minerals are recognizable, including quartz, milky, partially weathered feldspars and any mica is only partially altered. The saprock layer varies in thickness from 10 m to 30 m with an average thickness of 16 m;
- Fresh rock: None of the mineral components are altered. The TPFR code (top of fresh rock) is used for geological modelling.

The following photographs illustrate typical weathered rock material seen at the Doropo Project gold deposits.

Source: Centamin, 2021

Figure 7.9 Two Examples of Doropo Saprolite Showing Quartz Vein Fragments in the Right Hand Photo

The saprolite, example shown in Figure 7.10, has very little to no fabric preserved. The granitic origin can be interpreted by the remaining coarse quartz grains that are supported by clay matrix. Thickness of saprolite varies from 2 to 40 m, being very irregular across the drill sections and the various gold deposits. Overall, the thickest saprolite section is found at the Souwa and Enioda deposits and the thinnest saprolite section at the Han deposit.

Source: Centamin, 2021

Figure 7.10 Saprock (Original Fabric and Mineralogy is Partially Preserved)

The saprock example shown in Figure 7.11, while retaining much of the original granitic rock texture and being reasonably consolidated can still be broken by hand. Thickness of the saprock layer is less variable than saprolite, varying from about 10-30 m.

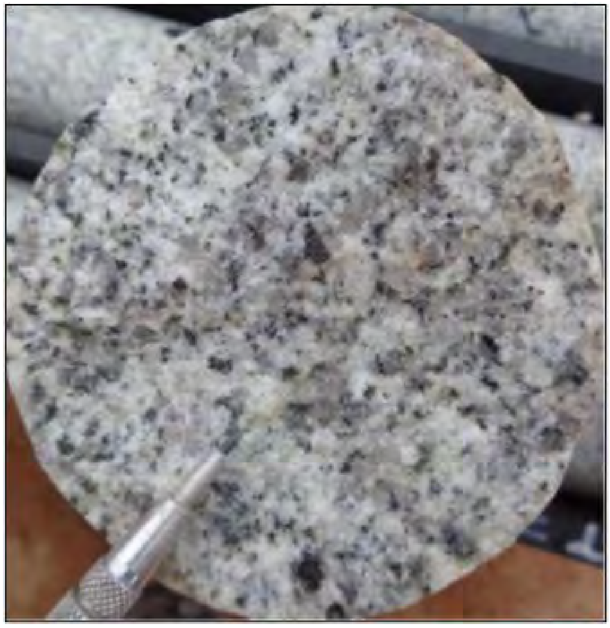
Source: Centamin, 2021

Figure 7.11 Saprock

The transition from saprock to fresh rock, shown in Figure 7.12, is generally sharp, typically no more than 1-3 m thick. A maximum thickness of about 10 m can be found in a few places where weathering is controlled by the fractures in the rock. Figure 7.13 shows a photograph of fresh granodiorite drill core for comparison.

Source: Centamin, 2021

Figure 7.12 Transition Zone Between Lower Saprolite and Fresh Rock



Source: Centamin, 2021

Figure 7.13 Fresh Granodiorite

7.2.5 Genesis

From August 2017 Centamin commissioned Orefind, an Australian based structural geology and geological modelling consulting company, to investigate the geological history of the Doropo Project area. The following text is an extract of their detailed Report (Davis, 2017):

All granite hosted mineralised structures examined have developed via the same fluid-structural history, comprising three broad cycles. Overprinting relationships indicate a massive deformation-controlled fluid flow system acting in tandem with a deformation regime that progressed from ductile through to brittle/ brittle-ductile in the waning stages of fluid ingress. Broadly similar sequences of quartz-carbonate veining were introduced at all prospects and indicate the following history:

- Ductile shear zone initiation;
- Ingress of the first silica-dominated fluid phase during ductile deformation;
- Hiatus in fluid flow;
- Ingress of the second major silica dominated fluid phase with deposition of base metals and gold in the waning stages of this fluid deformation cycle. Deformation caused pervasive brecciation of the first stage of quartz-dominant veins, with cementation and silicification of the breccias being facilitated by massive second-stage silica inflow;
- Hiatus in fluid flow;
- Deformation progresses to a dominantly brittle system. Deformation of the earlier silica stages, with breccia being cemented/silicified by the final major silica-dominant fluid phase.

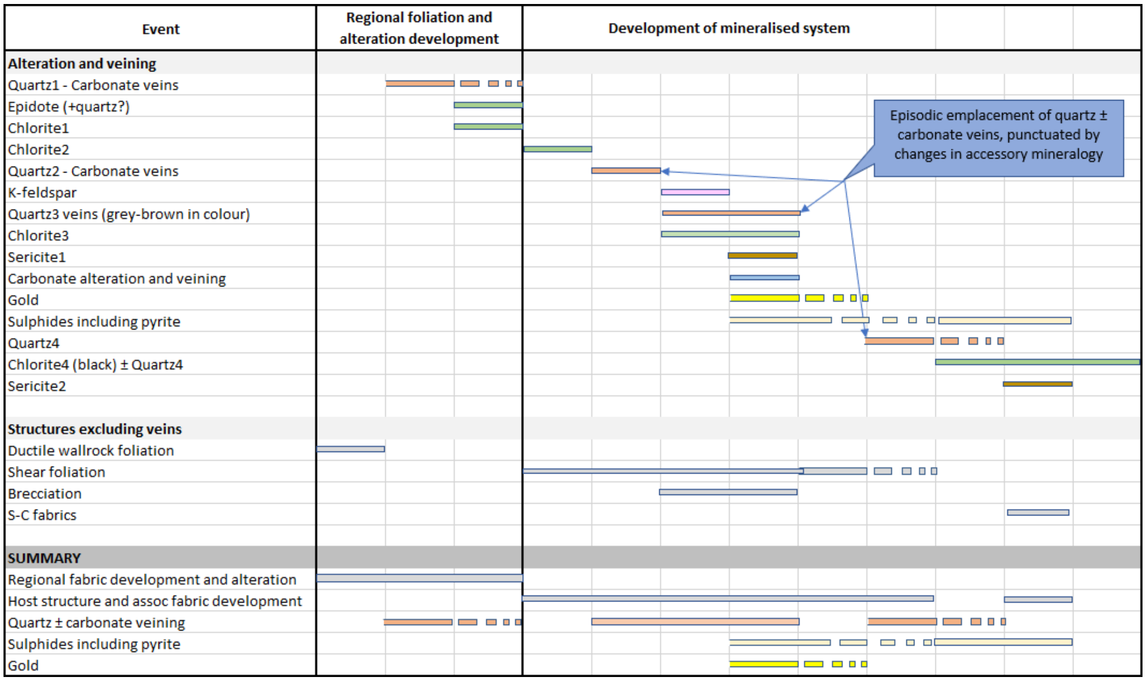
The progressive/repeated reactivation of the host structures has channelled numerous fluid cycles. Each of the three major silica-dominated fluid flow episodes described above will have comprised many individual fluid pulses, resulting in progressive increases in vein volume. Silicification of the host structures will have modified the rheology of the host rock, resulting in strain accumulation and ongoing localisation of deformation.

The first major stage of quartz rich fluid is inferred as being controlled by permeability associated with the accommodation of strain on structures that initiated as ductile shears in the granite. These structures likely also accommodated the greatest volume of silica bearing fluids, resulting in incremental formation of the largest white quartz veins seen in the deposits.

The second largest stage of quartz rich fluid was coeval with cycles of brittle deformation that overprinted the large, first-generation quartz veins. Angular breccia fragments were produced and then 'cemented' by a matrix of translucent to grey to black quartz infill and veining. The distinctive dark-coloured veins are commonly the host to sulphides and are inferred as coeval with, and host to, gold mineralisation. Accumulation of shearing strain at the margins of the first-generation veins commonly produced shear-induced lamination. Brecciation was sponsored in zones where the strain rate was great to accommodate ductile deformation. Overall, the special distribution of highest grades coincides with the deformed margins of the early formed veins.

The third major stage of quartz rich fluid was the volumetrically smallest, and manifests as cross-cutting white to translucent veins that inferred as forming under dominantly brittle conditions.

The geological history for Souwa, Kekeda and Han were established through the documentation of overprinting and geometric relationships in diamond core with Souwa displayed below in Figure 7.14. In 2023, geological histories were also developed for Kilosegui and Enioda. *Note: that evolution of the gold mineralising event overprints regional epidote-chlorite alteration*. Sulphide deposition is coeval with gold, with the exception if pyrite manifests as several generations. Both sulphides and gold occupy structural sites, including fractures and stylolites, that are spatially associated with deformed portions of the first-generation veins. These spatial relationships, and the common alignment of sulphide grain accumulations, indicate deposition of gold under an imposed stress. The S-C fabrics represent the final stage of foliation development noted in Souwa diamond core and have accommodated a sinistral sense of movement.



Source: Davis, 2017

Figure 7.14 Souwa Prospect Geological History

TABLE OF CONTENTS

8. DEPOSIT TYPES 1
8.1 Introduction 1

LIST OF FIGURES

Figure 8.1 Schematic Geological Model Interpretation of the Resource Area (Shear Zones in Black, Quartz Veins in Yellow and Doleritic Dykes in Green) 1

8. DEPOSIT TYPES

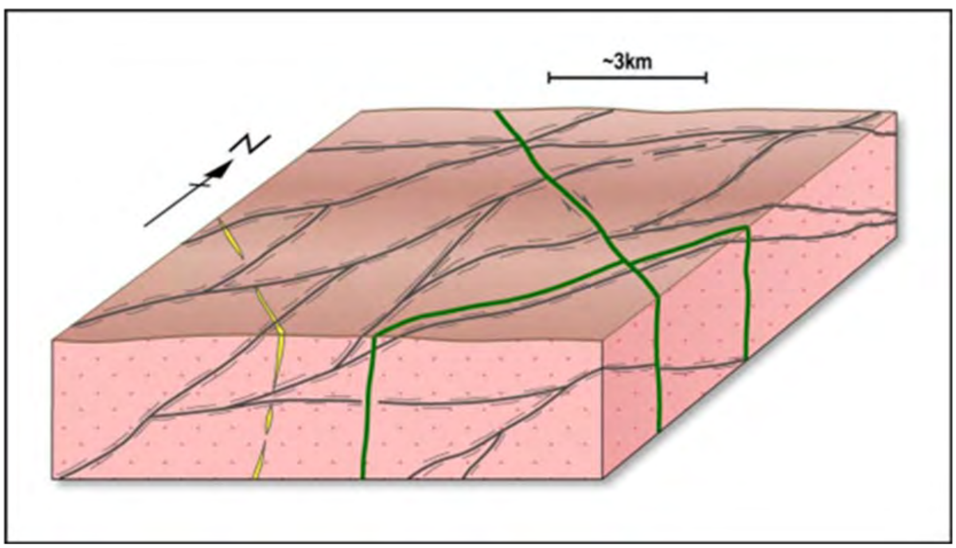
8.1 Introduction

The Doropo Project currently includes thirteen distinct mineralised bodies that host the actual resource plus numerous prospects and geochemical surface anomalies yet to be tested, fitting an area of about 170 km^2, or within a circular area of 28 km radius.

The mineral occurrences tested to date include two model types, a ‘classic’ orogenic shear-hosted gold deposit model and a quartz vein hosted gold deposit model (Figure 8.1). Both these models are coherent in nature with the majority of the other West African deposits, except on the issue of the host lithology (the granitic domain).

The granitic complex displays lozenge-shaped arrays of anastomosing shear zones. The shears have a broad south-southwest to north-northeast orientation, dipping shallowly towards the northwest. This interpreted model has been developed by several authors but was formalised in 3D by the Orefind geologists. The mineralisation occurs all along the shears, which were all channel ways for the fluid flows, however, at a generally low-grade gold deposition. Significant higher-grade mineralisation occurs on specific localised trend orientations, or when shears intersect, and are often spatially associated with doleritic dyke swarms (Davis, 2017).

The quartz veins mainly occur along the NW-SE orientation, and are sub-vertical or steeply dipping towards the SW. These veins show significant gold grades and often visible gold but have a limited width.



Source: Centamin, 2021

Figure 8.1 Schematic Geological Model Interpretation of the Resource Area (Shear Zones in Black, Quartz Veins in Yellow and Doleritic Dykes in Green)

TABLE OF CONTENTS

9. EXPLORATION ... 1
9.1 Coordinates, Survey Controls and Topographic Surveys ... 1
9.2 Geological Reconnaissance, Mapping and Rock Chip Sampling ... 2
9.3 Airborne Geophysical Survey ... 2
9.4 Soil Sampling ... 4
9.5 Auger Drilling ... 5
9.6 Trenching ... 7
9.7 Regolith Mapping and Interpretation ... 7
9.8 Gradient Array Induced Polarisation Survey ... 7
9.9 Aircore Drilling ... 7

LIST OF FIGURES

Figure 9.1 Location and Results of Rock Chip Sampling Programmes ... 2
Figure 9.2 Exploration Data: Magnetic Imagery (top) and Regolith Mapping ... 3
Figure 9.3 Location and Results of Soil Sampling Programmes ... 5
Figure 9.4 Location and Results of AUGER Sampling Programmes ... 6
Figure 9.5 Exploration Data: Gridded Gold Values in Surface Geochemistry (Merged Soils and Auger Data) ... 6
Figure 9.6 Sterilisation AC drilling completed at the Main Cluster in 2024 (Source: Centamin, 2024) ... 8
Figure 9.7 Combined Aircore, RC and DD drill hole locations (Source: Centamin, 2024) ... 9

9. EXPLORATION

Much of the content in this section is sourced from Osborn (2019) and Centamin (2021), and has been updated only where necessary.

Only minor exploration work was conducted before Centamin took over the Doropo Project in 2014. This work was limited to field reconnaissance and rock chip sampling and was carried out by Ampella Mining Ltd. Centamin started exploration work in 2014, progressing from regional field mapping, to the surface geochemistry sampling, via soils and auger, to the geophysical surveys, ground surveys and airborne surveys, to trenching, aircore drilling and then Reverse Circulation (RC) and Diamond Drilling (DD).

The above strategy currently remains unchanged and continues to be applied to the pipeline of targets within the permitted area.

All exploration work has started from a fixed base setup in the Doropo town. Following up the momentum in drilling activities on the prospects host of the actual resource, a second camp was set up in the Danoa village, located closer to the drill sites, about 45 km by tracks from Doropo. The regional exploration work (mapping, soil sampling, auger sampling, aircore drilling) is based out of fly camps or other temporary camps depending on the location of the programmes.

9.1 Coordinates, Survey Controls and Topographic Surveys

The default coordinates system used on the Project is based on the UTM coordinates, Zone 30 North in the World Geodetic System (WGS) 84. The Shuttle Radar Topography Mission (SRTM) digital data is used as the topographic reference for all the exploration work carried out to date.

GEDES International SARL Surveyors (Geo-Engineering Design and Surveying) is an accredited surveying company that is contracted to carry all ground surveys on the Project, including recording the location of all drill hole collars.

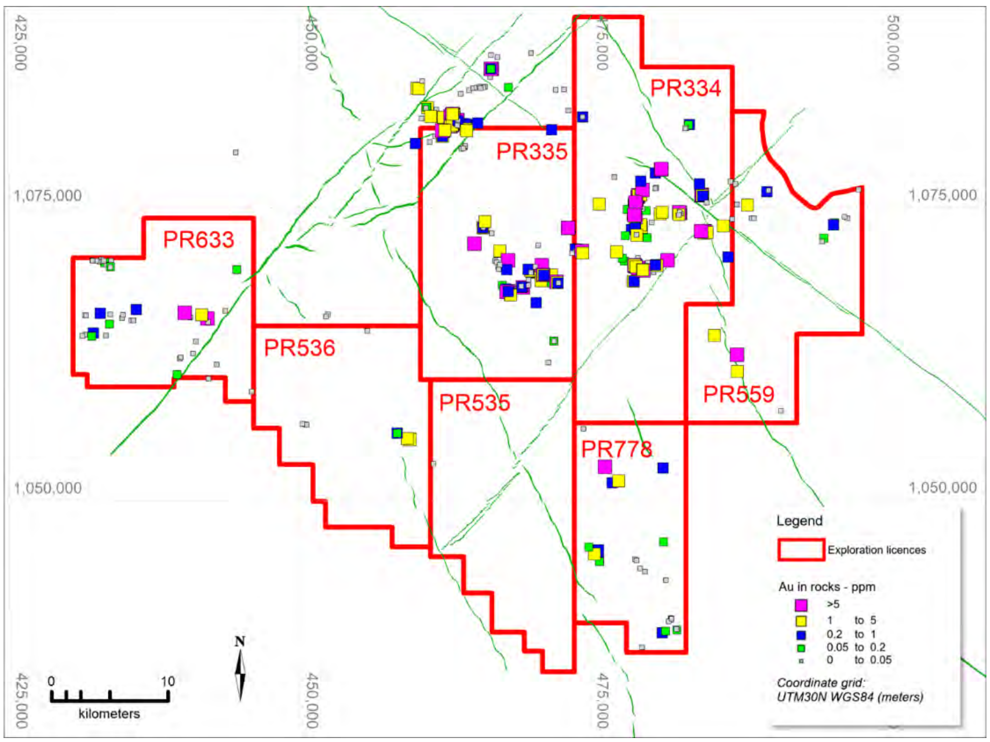
Ground fixed control points are regularly established, to follow the exploration progress. At the end of 2018, height control points were setup by GEDES. They are cemented in the ground, generally located on duricrust plateaus in the vicinity of the major prospects.

All drill hole collars (including RC and diamond collars) are surveyed using differential GPS unless accuracy is deemed to be low due to issues such as poor satellite coverage or abundant vegetation cover. In these cases, a total station is used to record the location of the collars. All other programmes including soil samples, rock samples, auger collars, trenches, aircore collars are located using hand-held GPS units. The collar elevations are linked to the NGCI system (Nivellement General de Côte d'Ivoire), that is the standard system for recording elevation in Côte d'Ivoire.

9.2 Geological Reconnaissance, Mapping and Rock Chip Sampling

Outcrops on the Project area are uncommon due to the granitic nature of the underlying rock. In the resource area, the main access to outcrops is generated by the artisanal mining.

The initial reconnaissance focused on mapping all the artisanal mining spots, the outlines of the excavations and any data on the quartz veins that were mined. All quartz vein, and the veins selvages were sampled by rock chipping. A total of 519 rock chip samples have been taken up until January 2022 with results of the rock chip sampling programmes shown in Figure 9.1.



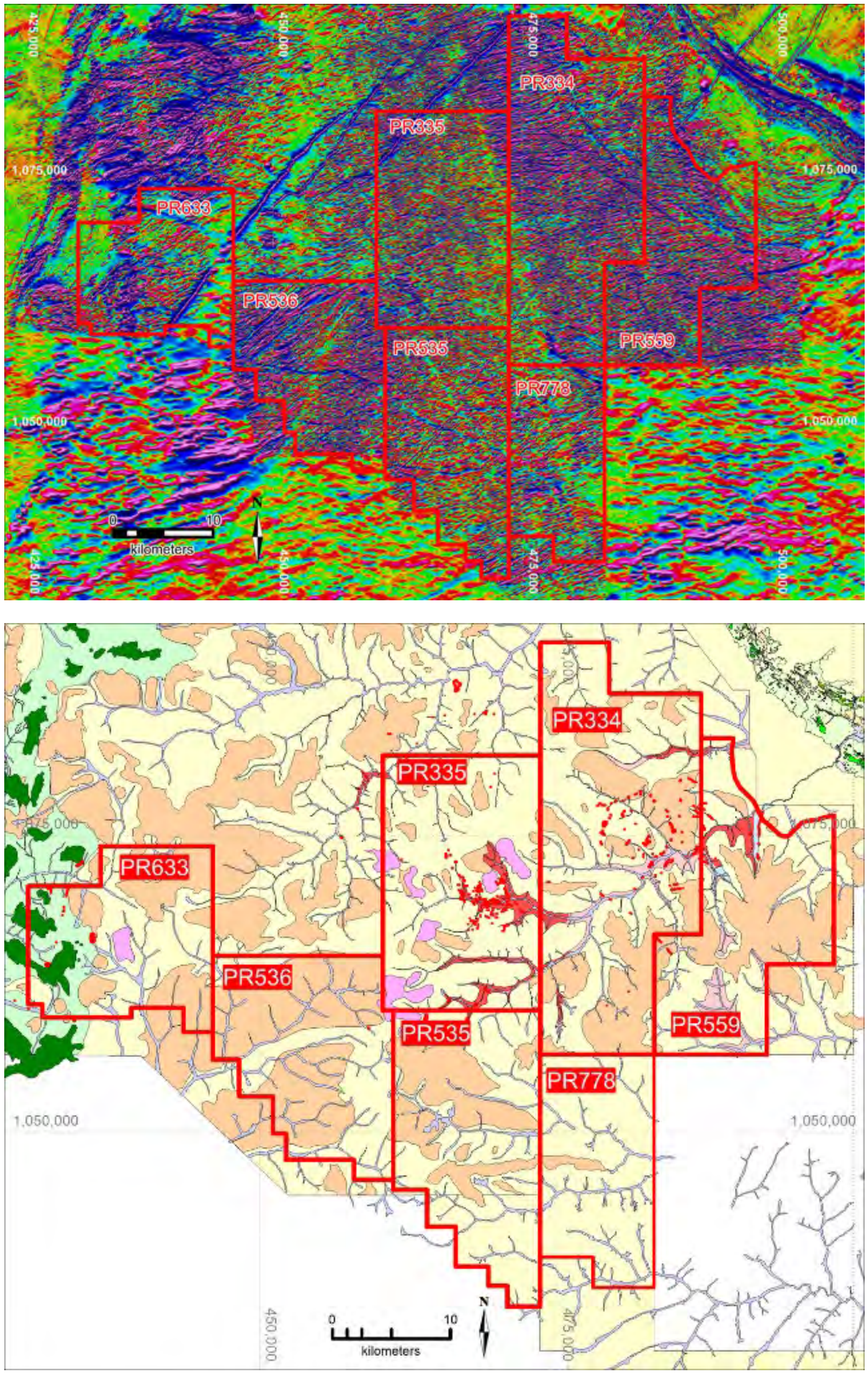
Source: Centamin, 2023

Figure 9.1 Location and Results of Rock Chip Sampling Programmes

9.3 Airborne Geophysical Survey

A regional aeromagnetic and radiometric survey, with additional detailed infill surveying over the Doropo Project, was flown by UTS Geophysics/Geotech Airborne Limited (UTS) between March 24 and May 27 in 2015 (Wood, 2015). The survey was flown using NNW-SSE oriented survey lines spaced either 200 m or 100 m apart, and covered a total of 21,827 line km.

The resulting imagery supported the initial regional interpretations and then the first regional exploration programmes. The results of the magnetic imagery can be seen in Figure 9.2.



Source: Centamin, 2023

Figure 9.2 Exploration Data: Magnetic Imagery (top) and Regolith Mapping

9.4 Soil Sampling

Soil sampling remains an efficient reconnaissance tool on the granitic domain and has proven to return representative results.

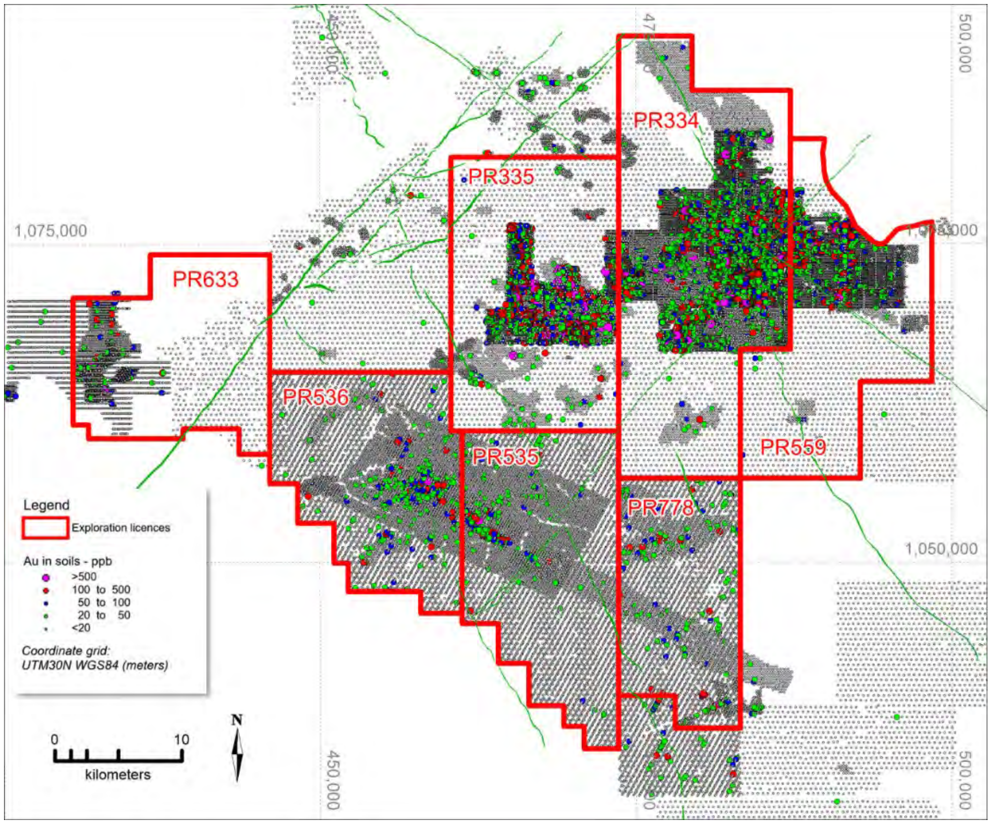
Several orientation surveys were originally conducted within the permit areas, across zones of artisanal mining activity as well as areas with no specific activity. The results showed that the upper most surface material, of sandy composition and often transported, is not representative at the prospect scale and returned irregular widespread dispersions of the gold anomalism. However, the horizon of mottle zone, or sometimes stripped top of saprolite, returned more coherent and reliable gold anomalism in the vicinity of the mineralised structures.

All subsequent soil sampling surveys sampled these representative levels, which are accessible from about 0.5 m depth to about maximum 3 m depth by quick pitting. Beyond this depth, the auger drilling methods are more efficient.

The regional soil grid is set on a staggered 400 m x 400 m grid. Infill sampling is then carried out where necessary on 200 m or 100 m spaced grids.

All soil samples (and geochemistry samples in general) are analysed using a standard 50 g gold fire assay with an atomic absorption finish at Bureau Veritas Laboratories in Abidjan. Multi-elements are also analysed by four-acid digest with ICP-AES and ICP-MS finish at the ACME Laboratories in Vancouver.

In total, 92,307 soil samples were collected between 2014 and December 2022, including 20,815 samples on infill grids on the Project. Most of the deposits and current prospects are well highlighted by the soils results. A map showing the location and results of the soil sampling programmes can be seen in Figure 9.2 and the combined results of the soil sampling and auger drilling surveys can be seen as a colour contour map in Figure 9.3.



Source: Centamin, 2023

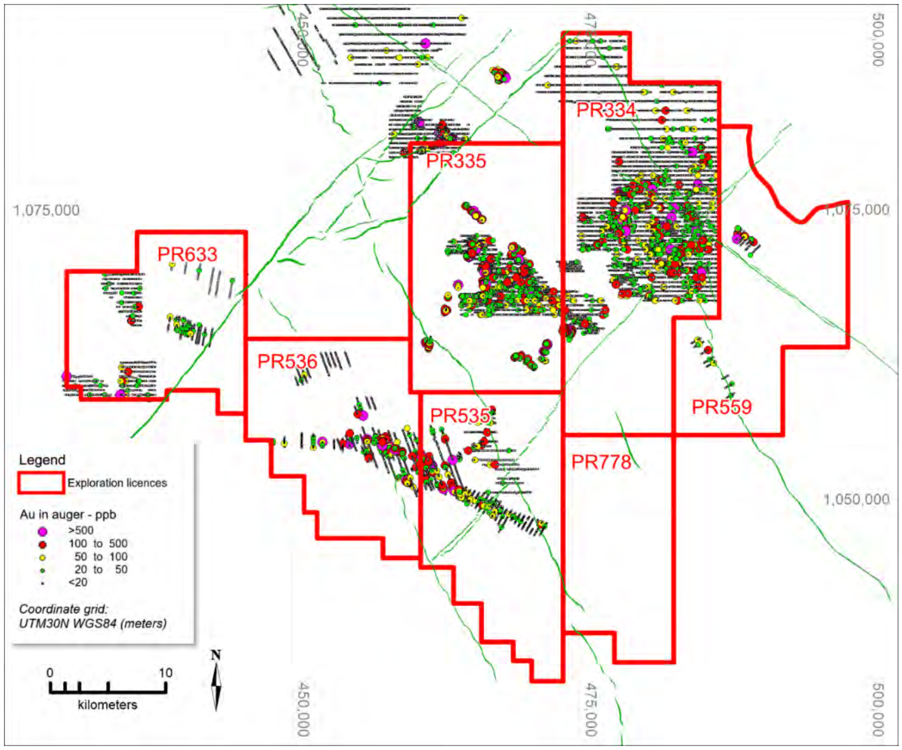
Figure 9.3 Location and Results of Soil Sampling Programmes

9.5 Auger Drilling

Auger drilling was largely used on the Project to complete the soil grid surveys where the thickest lateritic plateaus cover the in-situ material and where the transported horizons (alluvium and sand) average over 3 m thickness.

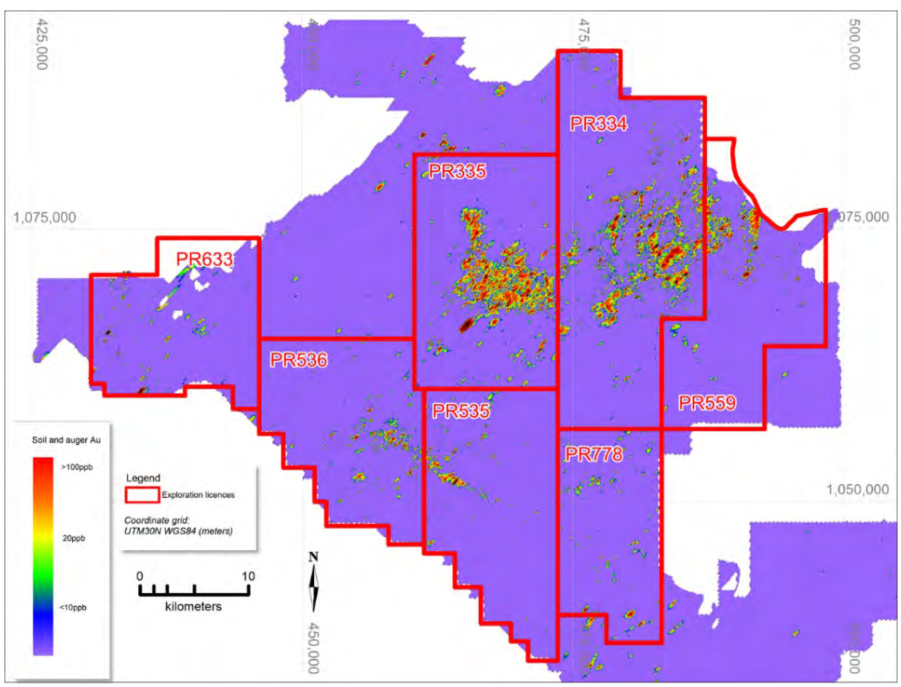
The powered augers, mounted on Land Cruisers, from Sahara Mining Services have been used on the Project to date. Generally, one sample from the top of the saprolitic horizon is collected per auger hole and is analysed for gold only (same analysis methods as the soils). In some cases, a second sample is also collected at the base of the lateritic horizon, aiming to test for mineralised lateritic layers. The samples collected from the auger drilling carried out in 2014 were analysed for gold by aqua regia digest with atomic absorption finish at SGS Laboratory in Ouagadougou.

A total of 27,999 auger holes have been drilled up until January 2022 with an average depth of 6.22 m and a maximum grade of 28 g/t Au. A map showing the location and results of the auger drilling programmes can be seen in Figure 9.4 and the combined results of the soil sampling and auger drilling surveys can be seen as a colour contour map Figure 9.5.



Source: Centamin, 2023

Figure 9.4 Location and Results of AUGER Sampling Programmes



Source: Centamin, 2023

Figure 9.5 Exploration Data: Gridded Gold Values in Surface Geochemistry (Merged Soils and Auger Data)

9.6 Trenching

Trenching was employed in certain remote areas to validate in-situ mineralized structures that had been identified through geochemical sampling. In total, 32 trenches were dug over two separate periods - 2015/2016, and 2022. Among these, 21 trenches were excavated in four phases around the Vako deposit and to the south of Doropo town during 2015 and 2016. Additionally, in 2022, ten trenches were completed to assess the northwest extension of Kilosegui.

Despite the extensive effort, the results from most of the trenches did not yield positive outcomes. Only two trenches excavated south of Doropo town exhibited promising mineralization with grades of 15 m @ 1.4 g/t and 9 m @ 0.7 g/t. Unfortunately, the area covered by the permit containing these promising trenches had already been relinquished to the government.

9.7 Regolith Mapping and Interpretation

The regolith map was generated to cover the entire Project area, using a combination of satellite imagery, radiometrics, soils database and field checking.

Weathering processes result in the depletion of potassium and relative enrichment of thorium and uranium, therefore radiometric maps including Th, K and U are extremely useful in identifying lateritic plateaus, as well as areas with little to no weathering profile. Large areas of lateritic plateaus, rivers and alluvial deposits can be easily identified using satellite imagery.

The soil sample descriptions, that include comments on the surface landscape, also provided good insight into the profile of the regolith.

9.8 Gradient Array Induced Polarisation Survey

Gradient Array Induced Polarisation (GAIP) surveys are regularly used to interpret the continuity of structures when already highlighted by other methods. Multiple blocks were surveyed in 2015 and 2016 in the resource area (Toni, 2017). The Nokpa deposit was targeted directly from the interpretation of the GAIP imagery. SAGAX is used to run the ground survey while Resource Potentials (RESPOT) worked on the QAQC and data processing.

9.9 Aircore Drilling

Campaigns of Aircore (AC) drilling were regularly conducted to quickly test coherent geochemical gold anomalism, conceptual targets or extensions to known mineralised structures. From June 2015 to December 2019, 142,947 m were drilled at an average length of 29.27 m and predominant dip of -55 degrees. From December 2023 to January 2024, 188 holes for 4,732 m at an average length of 25 m were drilled to sterilise planned infrastructure (e.g. waste dumps, processing plant, laydown areas, camp and water treatment facilities) as part of the Definitive Feasibility Study (DFS) process. Programmes were drilled on a 500 x 50 to 25 m spacing or one section across the middle of the planned infrastructure to ensure no economic zones of mineralisation were present as a first phase (Figure 9.6).

A map showing the combined aircore, RC and diamond drill hole locations can be seen in Figure 9.7. Aircore drilling is used as an exploration tool but is not included in the database used for resource estimates due to issues relating to sample representivity. All the drilling completed to date was conducted by Geodrill Ltd. The aircore holes are usually planned on lines across the targets to test; collars are planned heel to toe based on ground refusal along the lines.

The aircore programmes identified several mineralised structures which have now been followed up by RC and diamond drilling, including Souwa, Kekeda and, Enioda. The samples are composited on 2 m lengths and analysed for gold by fire assay at the Bureau Veritas Laboratories in Abidjan.

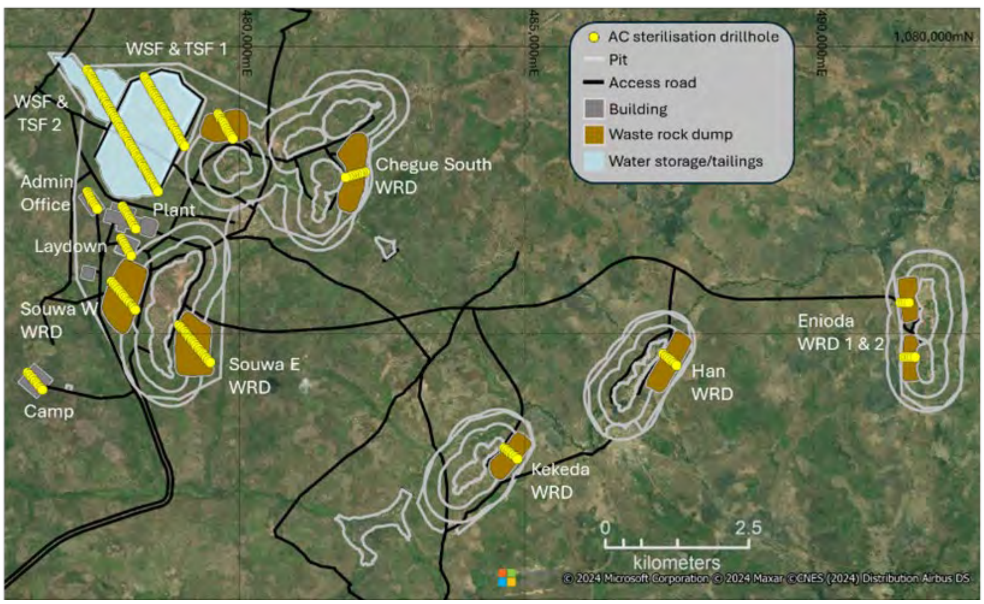


Figure 9.6 Sterilisation AC drilling completed at the Main Cluster in 2024 (Source: Centamin, 2024)

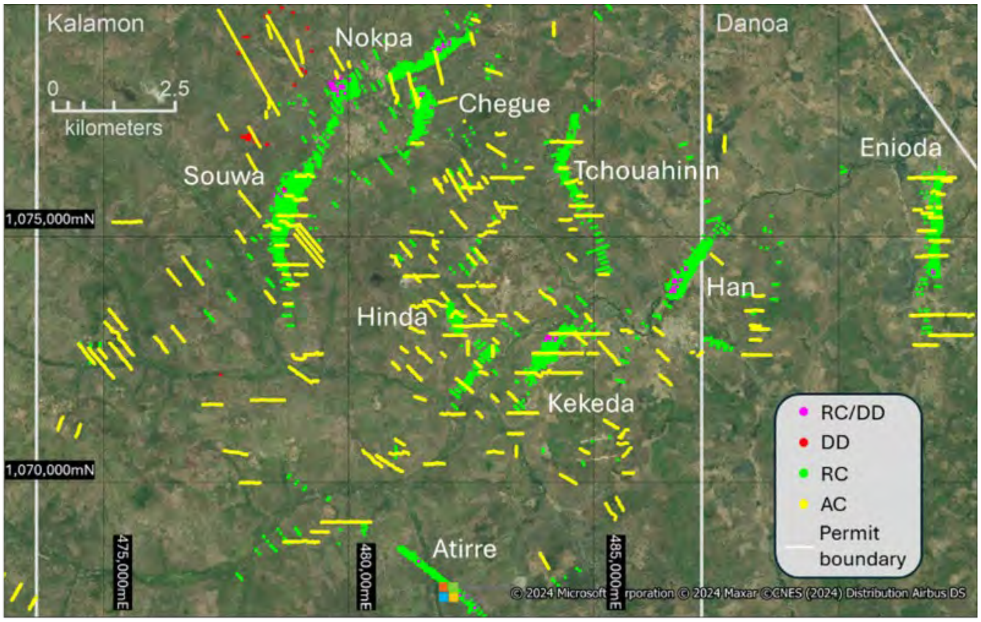


Figure 9.7 Combined Aircore, RC and DD drill hole locations (Source: Centamin, 2024)

TABLE OF CONTENTS

10. DRILLING 1
10.1 Drilling Summary 1
10.1.1 Reverse Circulation Drilling 5
10.1.2 Diamond Drilling 5
10.2 Collar Surveys 5
10.3 Downhole Surveys 5
10.4 Drill Coverage and Orientation 6
10.5 Sample Recovery and Grade 16
10.5.1 RC Holes 16
10.5.2 DD Holes 18
10.6 Wet RC Samples 20
10.7 Logging 20

LIST OF FIGURES

Figure 10.1 The Geodrill UDR650 Multipurpose Drill Rig with Compressor Visible in the Background 2
Figure 10.2 An Example of a Concrete Plinth Set Around PVC Casing at Souwa 3
Figure 10.3 The Reflex EZ Shot Tool About to be Inserted into a Drill Hole to Take a Downhole Survey Reading 6
Figure 10.4 Plan View of the Project Area Showing Drilling at the Various Prospects 7
Figure 10.5 Plan View of the Drill Holes and Mineralisation Domain Wireframes at ATI 7
Figure 10.6 Cross-Section View at ATI Looking South-East 8
Figure 10.7 Plan View of the Drill Holes and Mineralisation Domain Wireframes at CHG and NOK 8
Figure 10.8 Cross-Section View at CHG Main Looking North-East 9
Figure 10.9 Cross-Section View at CHG South Looking North-Northwest 9
Figure 10.10 Plan View of the Drill Holes and Mineralisation Domain Wireframes at ENI 10
Figure 10.11 Cross-Section View at ENI Looking North 11
Figure 10.12 Plan View of the Drill Holes and Mineralisation Domain Wireframes at HAN 11
Figure 10.13 Cross-Section View at HAN Looking North-East 12
Figure 10.14 Plan View of the Drill Holes and Mineralisation Domain Wireframes at KEK 12
Figure 10.15 Cross-Section View at KEK Looking North-East 13
Figure 10.16 Plan View of the Drill Holes and Mineralisation Domain Wireframes at KILO 13
Figure 10.17 Cross-Section View at KILO Looking South-East 14
Figure 10.18 Plan View of the Drill Holes and Mineralisation Domain Wireframes at SWA 15
Figure 10.19 Cross-Section View at SWA Looking North-East 16

Figure 10.20 Scatter Plot of RC Sample Weight Versus Downhole Depth 17
Figure 10.21 Scatter Plot of RC Sample Weight Versus Gold Grade 17
Figure 10.22 Histogram of Drill Core Recovery Percentage for Doropo DD Holes 18
Figure 10.23 Scatter Plot of DD Core Recovery Percentage Versus Downhole Depth 19
Figure 10.24 Scatter Plot of Gold Grade Versus DD Core Recovery Percentage 19

LIST OF TABLES

Table 10.1 Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023 3
Table 10.2 Breakdown of Available Drill Data for the Mineral Resource Estimate Update of the Updated Prospects Only, as at 27 August 2023 4

10. DRILLING

10.1 Drilling Summary

The Reverse Circulation (RC) and Diamond Drilling (DD) drill programmes have been undertaken using a phased approach on the Doropo Project since the end of November 2015, following the first significant hits from the aircore (AC) drill programmes. Drill procedures are well documented and have been specifically adapted to the Project from the experience gained by the team on previous projects.

All the drilling to date has been largely undertaken by two drilling companies, both are reputable contractors who respect good industry practices (Geodrill Ltd and Energold Drilling Ltd). The drill rigs are well maintained and the maintenance crew is quickly responsive. All the staff, from the drillers to the offsiders, are well trained and operate smoothly. The drill rigs used on the Project are UDR200 (for diamond drilling only), UDR650 (small multipurpose, truck mounted rig, Figure 10.1) and UDR900 (large multipurpose, track mounted rig).

The drill programmes are planned using on-site cross-sectional interpretations, which are based on previous exploration programmes, surface geochemistry, aircore drilling or other previous drilling completed, geophysical imageries and on conceptual interpretations.

The latest database provided to Cube contained a total of 5,794 holes relating to the prospects updated and comprises DD, RC and RC pre-collar holes with DD tails (RCD holes). No AC holes have been used for Mineral Resource modelling due to quality concerns associated with this method, which is better suited to exploration target and sterilisation drilling. A summary of the hole types and metreage is provided in Table 10.1, with a more detailed breakdown per updated prospect given in Table 10.2.

The drill sites are prepared by hand clearing or dozer depending on the areas. By default, infill lines are cleared by dozer. The drill pad sizes are set according to the needs of the drilling contractor.

After the completion of a drill hole, the drill site is cleaned and any contaminated soil is removed. A concrete plinth of approximatively 40 cm x 40 cm x 20 cm is set around the PVC casing for future reference (Figure 10.2).

Figure 10.1 The Geodrill UDR650 Multipurpose Drill Rig with Compressor Visible in the Background

Figure 10.2 An Example of a Concrete Plinth Set Around PVC Casing at Souwa

Hole Type	No. Holes	Total Length (m)	Ave. Depth (m)
DD	450	43,404	96
RC	5,245	487,440	93
RCD	99	16,961	171
TOTAL	5,794	547,805	95

Table 10.1 Summary of DD, RC and RCD Holes Provided to Cube for Mineral Resource Estimation at Doropo for the Updated Prospects Only, as at 27 August 2023

Prospect	Hole Type	No. Holes	Total Length (m)
ATI (Attire)	DD	21	1955.65
	RC	238	24184
	RCD	4	603.65
CHG (Chegue)	DD	83	8054.99
	RC	682	62767.8
	RCD	12	1,667.8
ENI (Enioda)	DD	42	3282.43
	RC	342	28973
	RCD	2	210.5

Prospect	Hole Type	No. Holes	Total Length (m)
HAN (Han)	DD	39	3044.34
	RC	371	28378
	RCD	18	2347.92
KEK (Kekeda)	DD	40	2814.6
	RC	377	27900
	RCD	11	1725.6
KLG (Kilosegui)	DD	96	10762.15
	RC	1138	101372
	RCD	6	858.5
NOK (Nokpa)	DD	33	4246.21
	RC	300	31165
	RCD	29	6347.23
SWA (Souwa)	DD	76	7594.53
	RC	739	78405.2
	RCD	17	3200.22
HND & HNDS (Hinda)	DD	0	0
	RC	181	17342
	RCD	0	0
NAR (Nare)	DD	0	0
	RC	43	3720
	RCD	0	0
SAN (Sanboyoro)	DD	6	444.8
	RC	94	9375
	RCD	0	0
THN (Tchouahinin)	DD	0	0
	RC	175	17507
	RCD	0	0
VAKO (Vako)	DD	14	1204.65
	RC	244	21769
	RCD	0	0
OTHER	DD	0	0
	RC	321	34582
	RCD	0	0

Table 10.2 Breakdown of Available Drill Data for the Mineral Resource Estimate Update of the Updated Prospects Only, as at 27 August 2023

10.1.1 *Reverse Circulation Drilling*

RC drilling comprises the primary drilling method on the Project from the end of November 2015. One to three multipurpose rigs (to keep the opportunity to switch to diamond drilling) are rotating, depending on the programme. Some 5,245 RC holes totalling 487,440 m have been drilled up to 18 July 2023 at all Doropo prospects, with the additional infill drilling used to inform the current Mineral Resource update being a subset of this total. The numbers quoted in Table 10.1 refer only to the prospects updated in this Study.

The drilling is dominantly dry and the moisture content (dry, moist or wet) of the bulk sample has been recorded since the end of 2016. For resource definition drilling, the drilling stops when the water table is reached and the booster air pressure cannot keep the samples dry. The hole may then be continued by DD if the targeted mineralisation has not been intersected yet.

The RC drilling uses hammer bits of nominally 5¼, 5½ and 5¾ inch diameter; the bit size was poorly recorded by drill holes until November 2016. From this time onward, the bit sizes used by depth and by hole has been recorded.

10.1.2 *Diamond Drilling*

A total of 450 DD holes for 43,404 m and some 99 RCD hole for 16,961 m had been drilled at the Project as at 27 August 2023 for all prospects, including those not updated in this estimate. The DD is used as drilling tails after RC pre-collar in the case of the deepest drilling (over about 180 m depth) as well as holes drilled to obtain structural data. Some DD was also completed to collect composite samples for metallurgical test-work samples. Either quarter or half core is retained in metallurgical holes and analysed for gold assays that are used in resource estimation.

10.2 Collar Surveys

The initial location of drill collars is undertaken by the geologist using a hand-held GPS, to rapidly enter the data into the database. Regular surveying campaigns are subsequently undertaken by an independent surveyor company (GEDES International) to accurately pick up collar coordinates with either a Total Station or differential GPS.

10.3 Downhole Surveys

The initial downhole survey reading is taken at 12 m depth, and then at every subsequent 30 m depth interval. A single shot Reflex EZ SHOT system has been used to undertake the downhole surveys (Figure 10.3). The drill inclination and azimuth are set using a compass and clinometer by the geologist, and this is recorded as the survey reading at the collar. Every survey is validated at the rig site by the geologist before being entered into the database.

10.4 Drill Coverage and Orientation

The drill coverage for the Project as a whole is shown in Figure 10.4. The latest infill drilling has reduced the nominal drill collar spacing to between 25 m and 40 m (including 10 m by 10 m grade control patterns across eight Projects), especially in areas identified as being of most economic interest at the preceding 60 m to 80 m grid. The drill lines and azimuths are orientated so as to be near-perpendicular to the strike of each prospect. Drill inclination is most commonly at -60°, which for the vast majority of the lodes in the updated prospects results in an intersection angle to the lode planes that is orthogonal or sufficiently close to orthogonal to avoid significant sample bias.

Zoomed in plots of the drill plan and a representative cross-section, overlaid on the latest interpreted mineralised lodes, are shown in Figure 10.5 through Figure 10.19.

It is the Qualified Person’s opinion that the drilling configuration is appropriate for the geometry and orientation of the planar lodes at Doropo and is therefore suitable for Mineral Resource estimation.

Figure 10.3 The Reflex EZ Shot Tool About to be Inserted into a Drill Hole to Take a Downhole Survey Reading

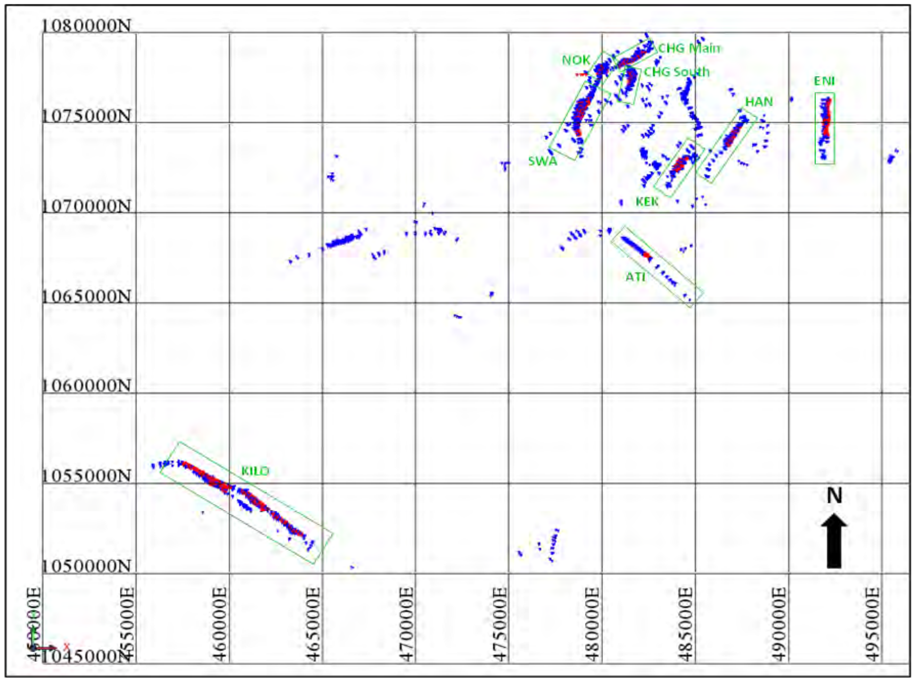
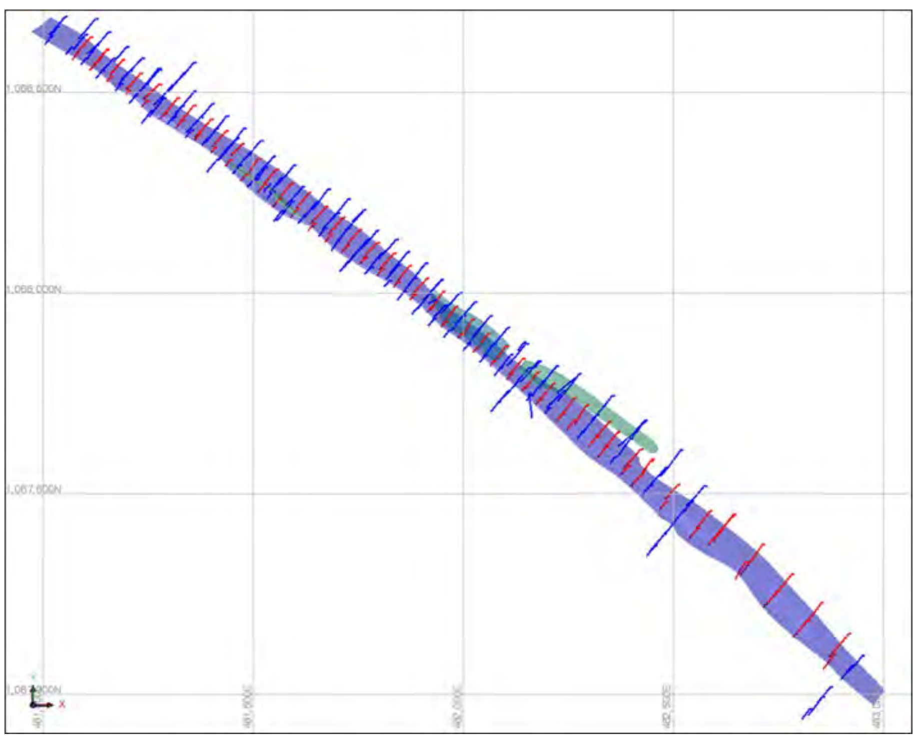
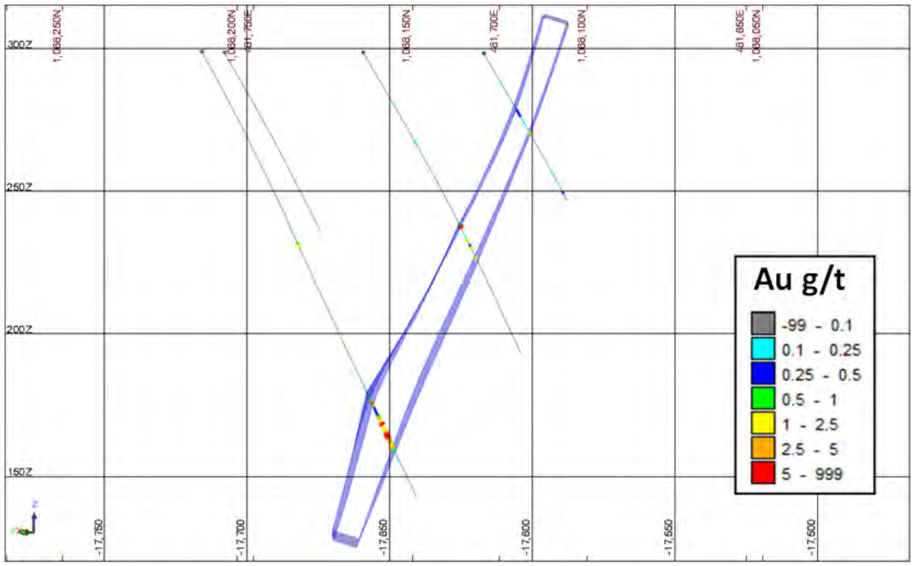


Figure 10.4 Plan View of the Project Area Showing Drilling at the Various Prospects



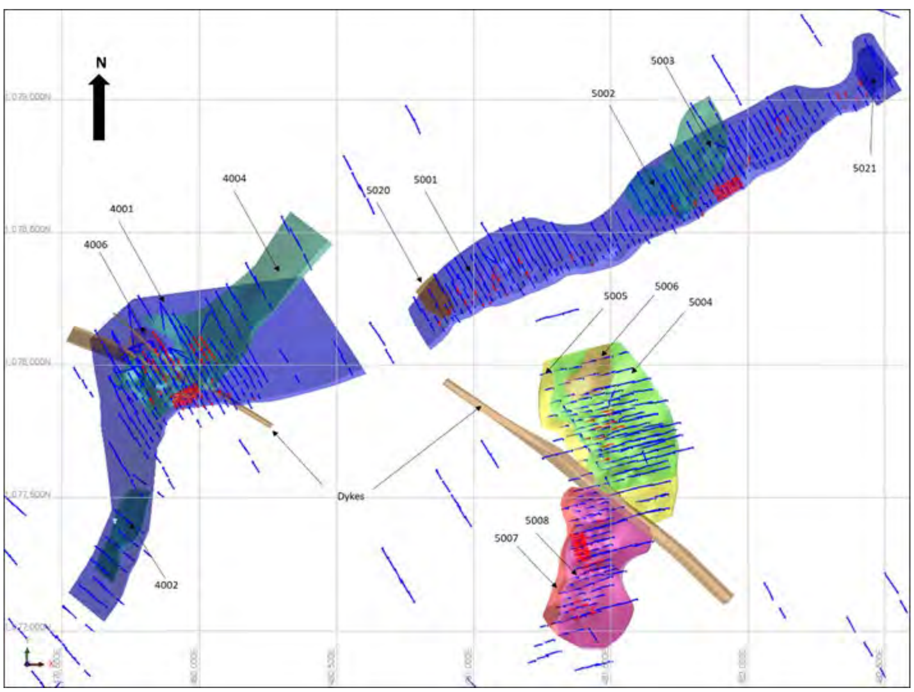
Note: Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.

Figure 10.5 Plan View of the Drill Holes and Mineralisation Domain Wireframes at ATI



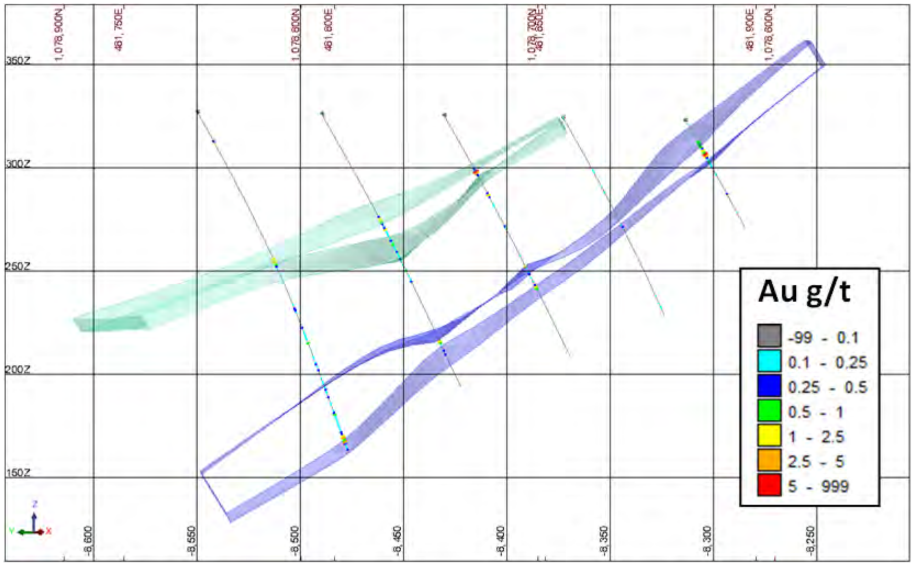
Note: *Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.*

Figure 10.6 Cross-Section View at ATI Looking South-East



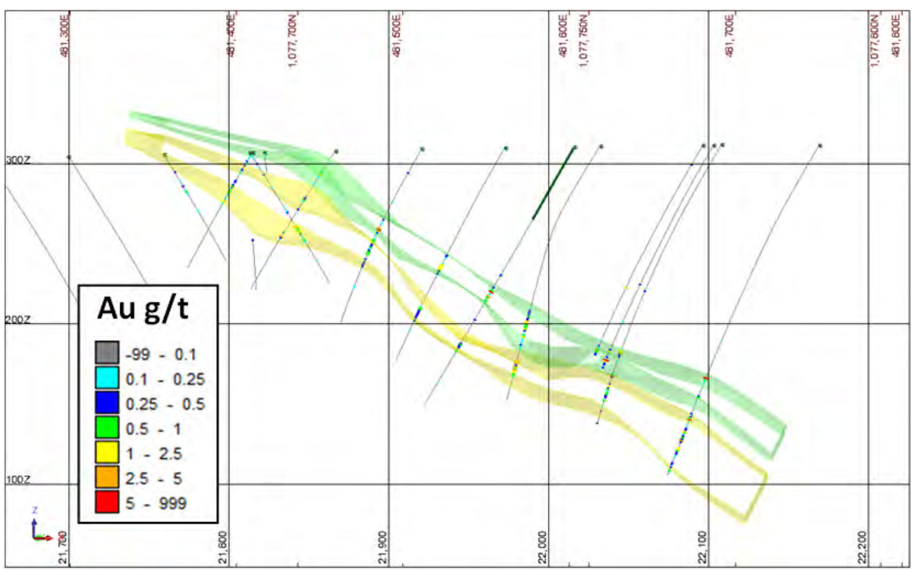
Note: *Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.*

Figure 10.7 Plan View of the Drill Holes and Mineralisation Domain Wireframes at CHG and NOK



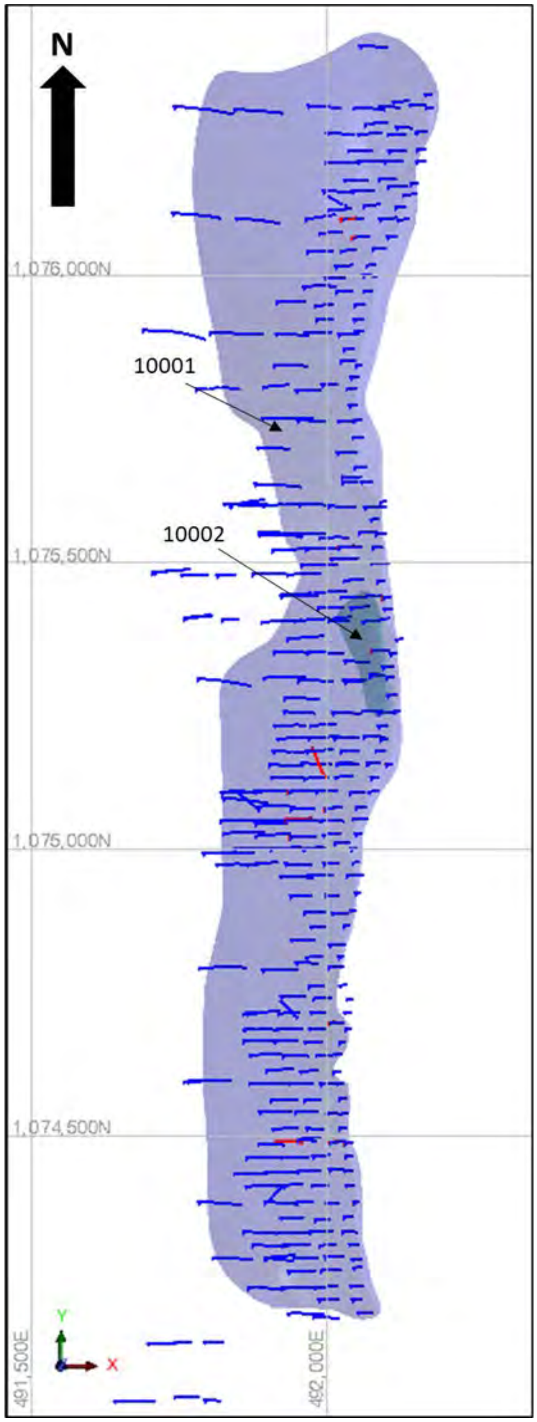
Note: *Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.*

Figure 10.8 Cross-Section View at CHG Main Looking North-East



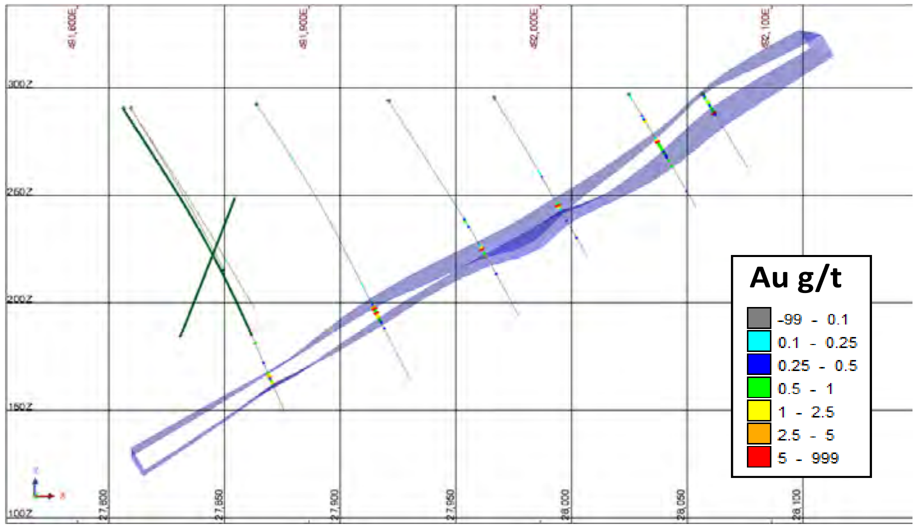
Note: *Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.*

Figure 10.9 Cross-Section View at CHG South Looking North-Northwest



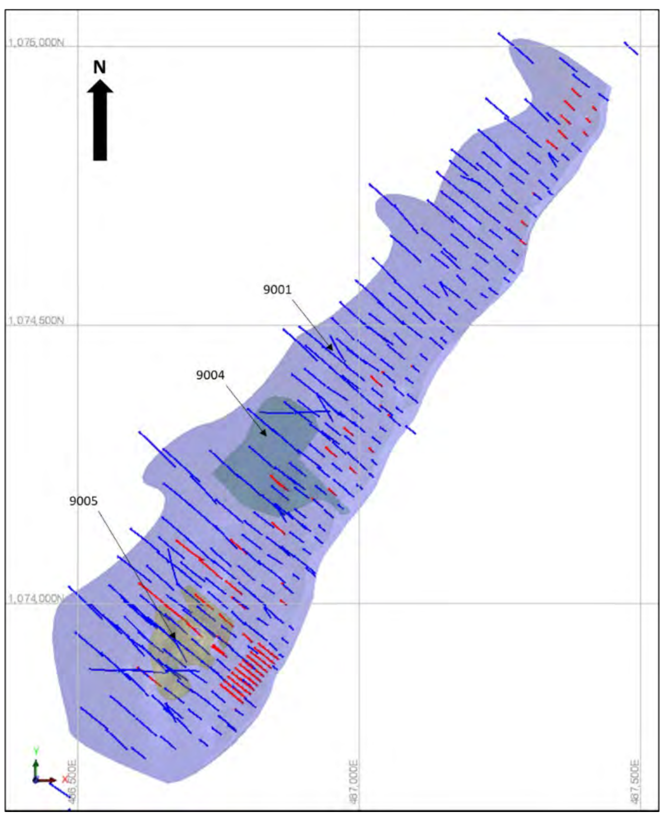
Note: Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.

Figure 10.10 Plan View of the Drill Holes and Mineralisation Domain Wireframes at ENI



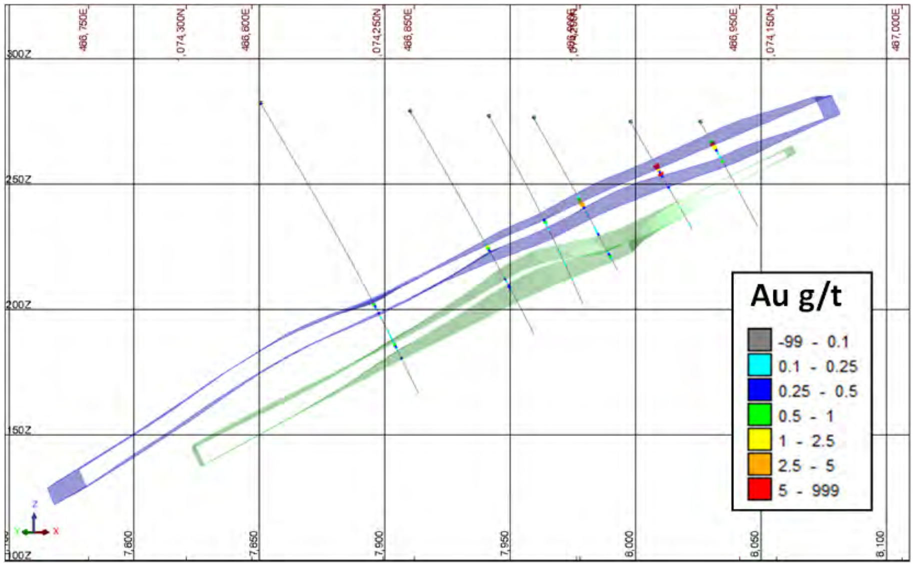
Note: *Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.*

Figure 10.11 Cross-Section View at ENI Looking North



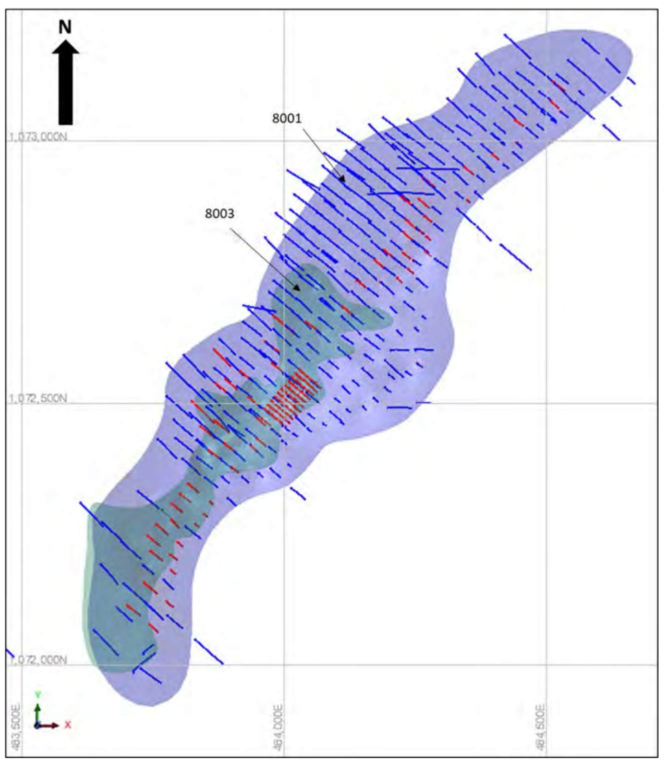
Note: *Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.*

Figure 10.12 Plan View of the Drill Holes and Mineralisation Domain Wireframes at HAN



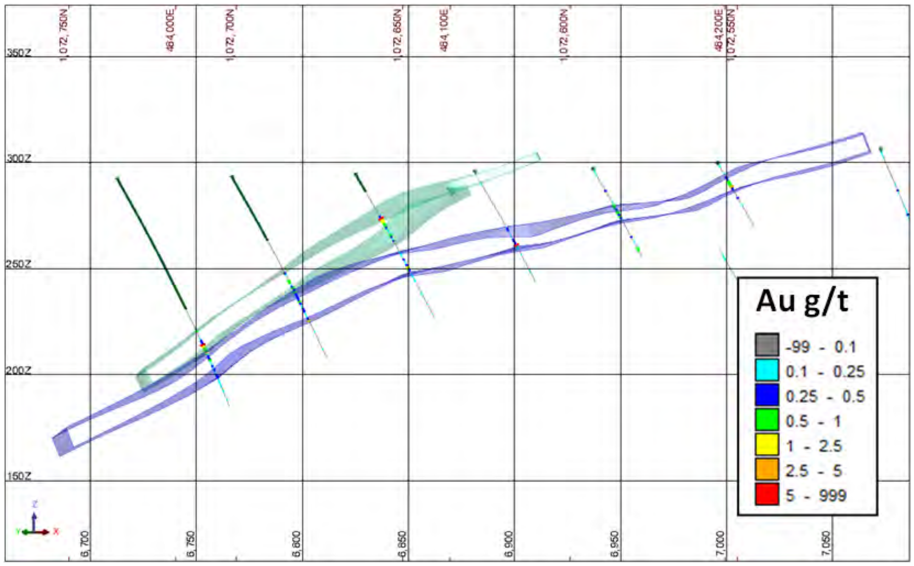
Note: *Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.*

Figure 10.13 Cross-Section View at HAN Looking North-East



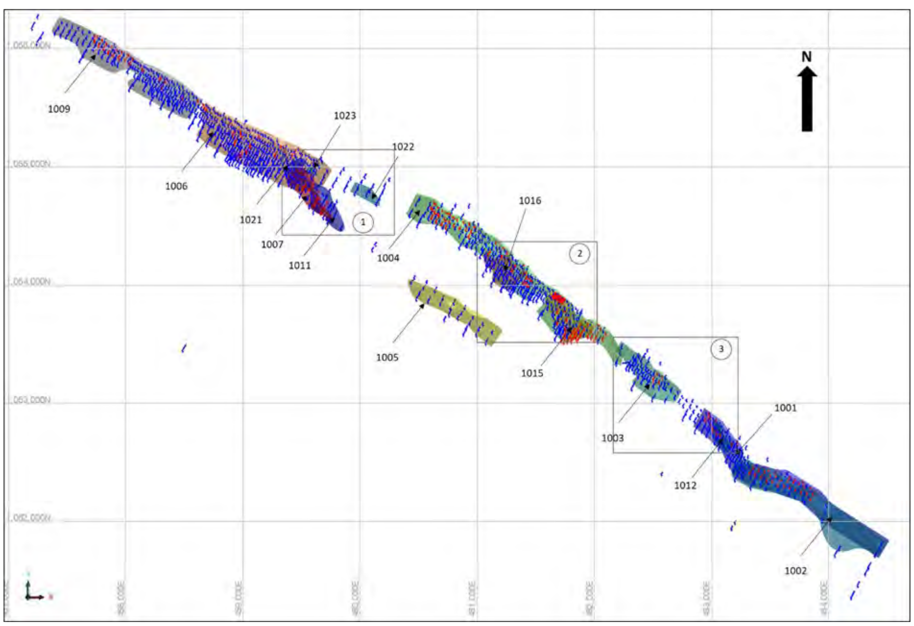
Note: *Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.*

Figure 10.14 Plan View of the Drill Holes and Mineralisation Domain Wireframes at KEK



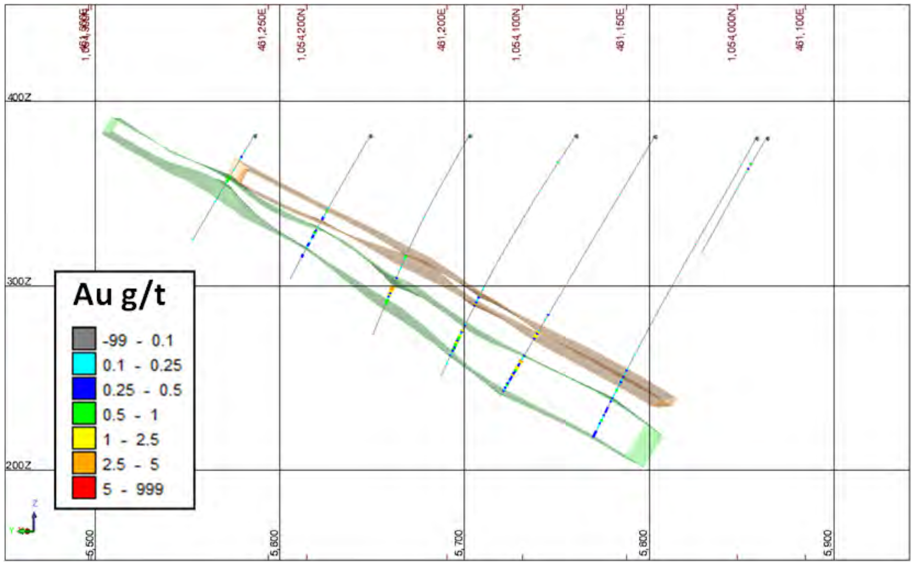
Note: Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.

Figure 10.15 Cross-Section View at KEK Looking North-East



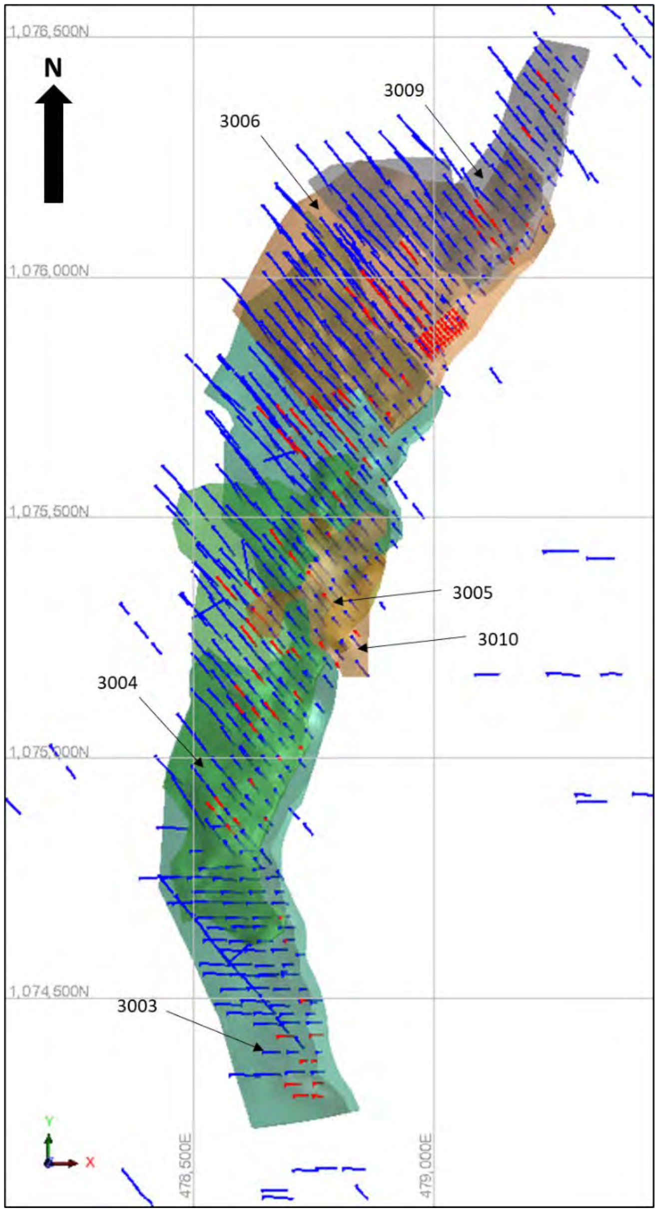
Note: Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.

Figure 10.16 Plan View of the Drill Holes and Mineralisation Domain Wireframes at KILO



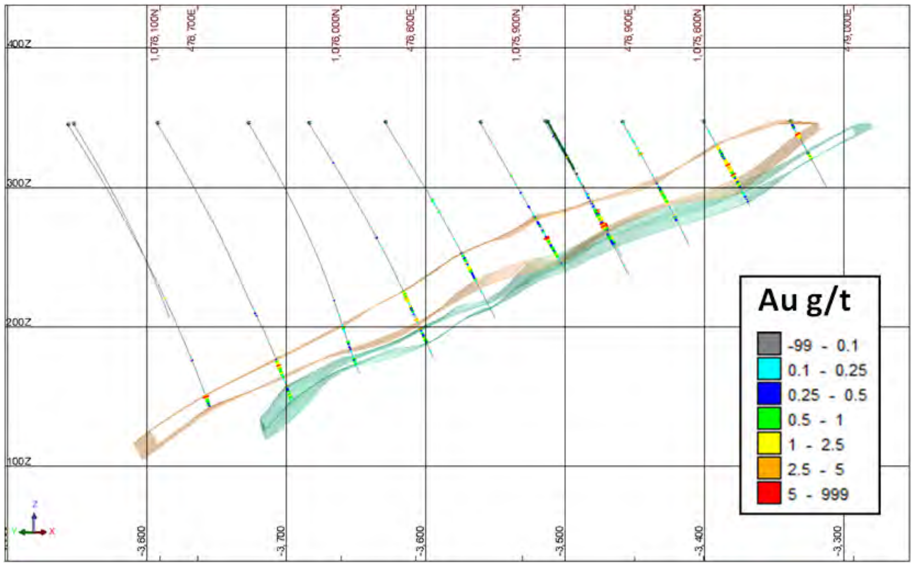
Note: Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.

Figure 10.17 Cross-Section View at KILO Looking South-East



Note: *Blue traces represent pre-existing holes and red traces are the additional holes drilled subsequent to the PFS resource estimate, which were utilised in this DFS update.*

Figure 10.18 Plan View of the Drill Holes and Mineralisation Domain Wireframes at SWA



Note: Drill holes coloured by gold assay grade with mineralisation domain wireframe outlines shown. Slicing window is 15 m either side of the section line.

Figure 10.19 Cross-Section View at SWA Looking North-East

10.5 Sample Recovery and Grade

A study examining both RC and DD sample recovery with respect to gold grade was undertaken as part of a 2019 MRE by Osborn (2019). That study concluded that there was no significant bias brought about either by selective loss of gangue or ore material in the drill hole dataset as it stood at that time. The only issue picked up was that there was a slight reduction in gold grade in DD samples with lower core recovery. Cube has undertaken some additional checks in this study, given that additional drilling has been undertaken since 2019.

10.5.1 RC Holes

A total of 477,612 RC sample weight records were provided to Cube, and of these, some 447,401 had accompanying gold assay records. A plot of RC sample weights against downhole depths shows that the sample weight initially increases with depth before levelling out to between 30 kg and 40 kg (Figure 10.20). The lower weights in the upper part of the holes are probably mostly due to lower densities in the highly weathered zone but could also include a component of sample loss, which would be more prevalent in the less competent soils and saprolites. The relationship between the RC sample weights and gold grade shows evidence of a slight trend of lower grades with lower weight at grades below 1 g/t Au, but this correlation is very weak (Figure 10.21) within the grade range that would typify the modelled mineralisation zones (i.e. at grades approximately above 0.2 g/t Au). It is therefore concluded that there is no problematic gold grade bias on the basis of RC sample weights.

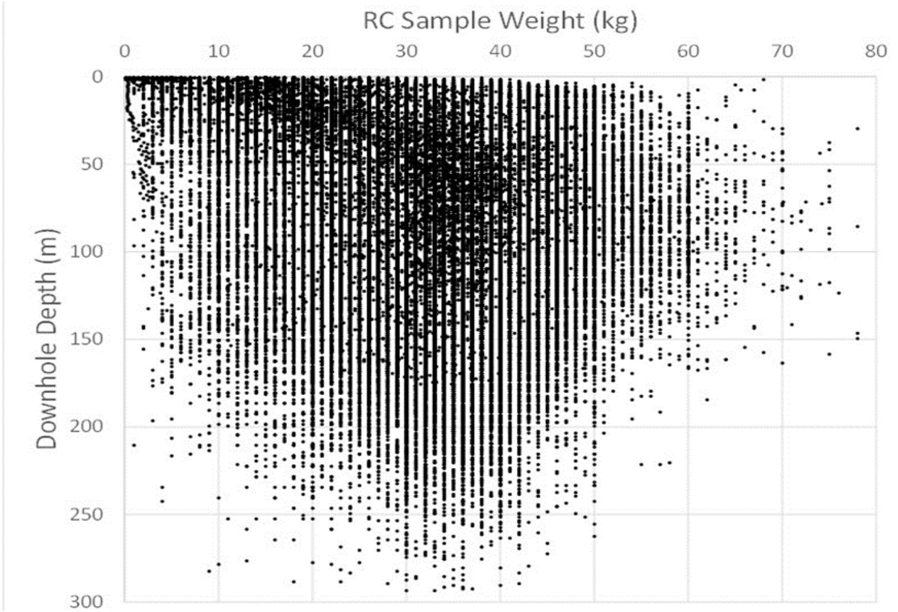


Figure 10.20 Scatter Plot of RC Sample Weight Versus Downhole Depth

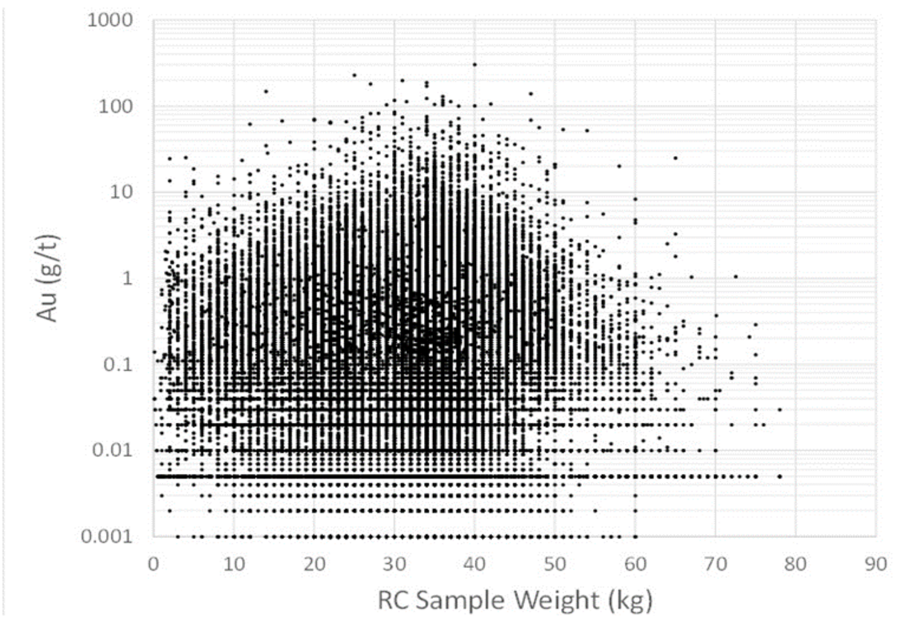


Figure 10.21 Scatter Plot of RC Sample Weight Versus Gold Grade

10.5.2 DD Holes

A total of 20,552 DD core recovery records are available for analysis at Doropo, with some 13,886 having corresponding gold assays. The average core recovery is ~96% (Figure 10.22), and a plot of downhole depth against core recovery shows that a significant proportion of the lower recoveries are at shallower depths (Figure 10.23). The proportion of samples with core recovery greater than 90% is 89.5% overall, but this rises to 97.5% at downhole depths greater than 50 m. This is to be expected since the drilling mud tends to wash away fines in the weathered zone. When gold grade is plotted against core recovery, no significant correlation trend can be observed (Figure 10.24). The evidence suggests there is no systematic grade bias resulting from under-recovery of drill core, and the proportion of intervals reflecting lesser core recovery is small.

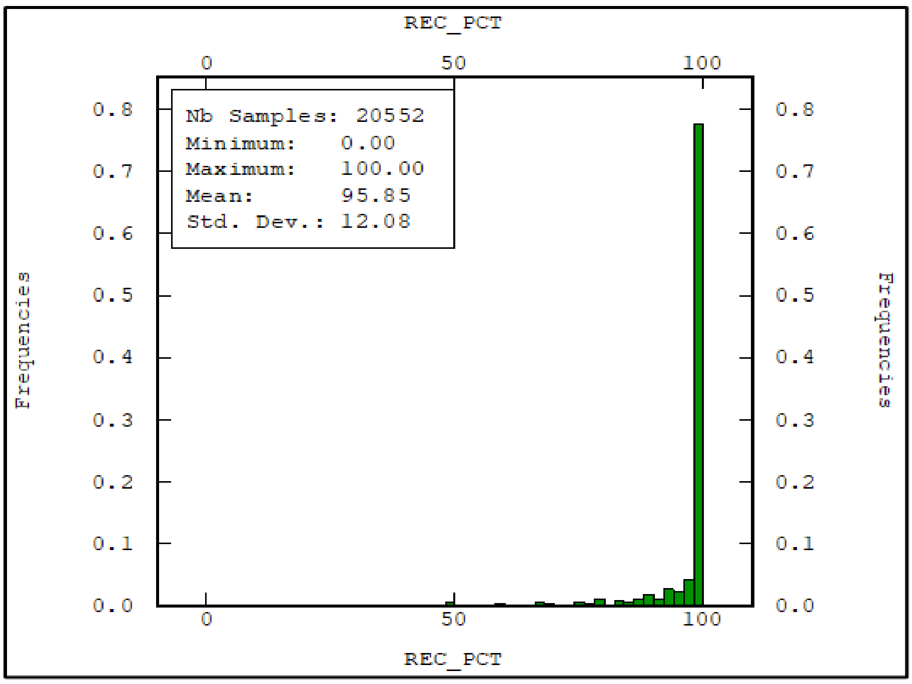


Figure 10.22 Histogram of Drill Core Recovery Percentage for Doropo DD Holes

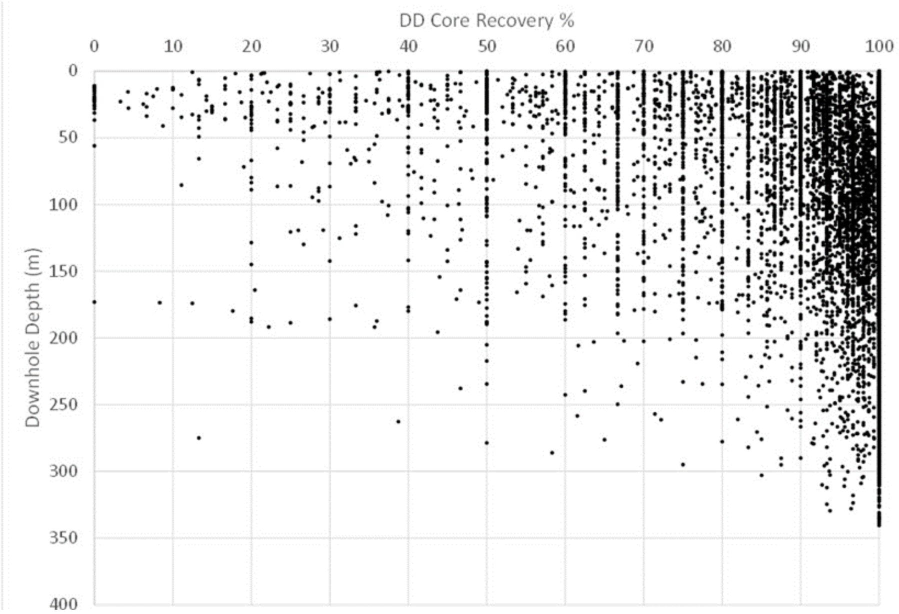


Figure 10.23 Scatter Plot of DD Core Recovery Percentage Versus Downhole Depth

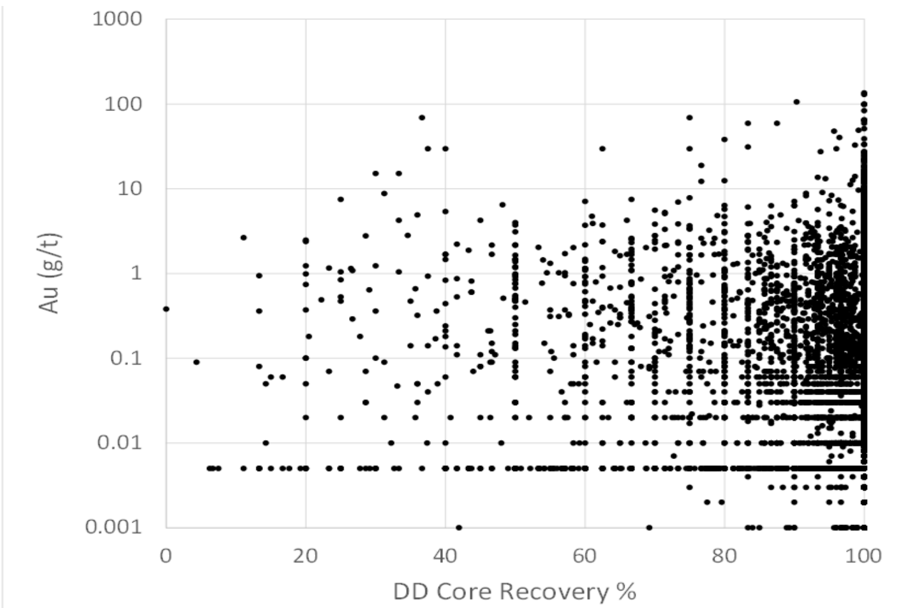


Figure 10.24 Scatter Plot of Gold Grade Versus DD Core Recovery Percentage

10.6 Wet RC Samples

During the site visit undertaken by the Qualified Person, RC drilling operations were observed for several hours, with the late-August date being at the end of the wet season. The booster system was observed to be highly effective at keeping the RC samples dry and cleaning of the cyclone after each 1 m sample and at the rod changes was seen to be satisfactory. These observations are supported by the drill logs show that dry samples comprise ~96% of all the RC assay samples while moist samples comprise ~3% and wet samples ~1%. The vast majority of the RC samples have been logged for moisture content.

A selection of RC hole traces containing wet intervals were inspected, with gold grade displayed, and no significant evidence of downhole contamination due to the wet samples was detected. This, along with the fact that wet samples comprise a very small proportion of logged samples, lead the Qualified Person to conclude that wet RC samples pose an insignificantly small risk to the veracity of the RC assay data.

10.7 Logging

All drill holes were logged geologically and geotechnically, with information recorded including:

- Weathering;
- Lithology;
- Structure;
- Texture;
- Alteration;
- Mineralisation;
- Rock Quality Designation.

All holes were geologically logged in full. Diamond drill core is photographed wet and dry before cutting.

TABLE OF CONTENTS

11. SAMPLE PREPARATION, ANALYSES AND SECURITY 1
11.1 Reverse Circulation Sampling Methods 1
11.2 Diamond Core Sampling Methods 2
11.3 Chain of Custody and Transport 2
11.4 Sample Preparation 3
11.4.1 RC Samples 3
11.4.2 DD Samples 3
11.5 Sample Analysis at Laboratory 4
11.6 Bulk Density Determinations 5
11.7 Quality Assurance and Quality Control Sampling 6
11.7.1 Certified Reference Materials (CRMs or Standards) 9
11.7.2 Blanks 24
11.7.3 Duplicates 28
11.7.4 Duplicate Performance Conclusions 79
11.7.5 Summary Opinion of Qualified Person 80

LIST OF FIGURES

Figure 11.1 Accuracy and Precision Concepts 7
Figure 11.2 OREAS 211 - MSA_YAM Results - Au ppm 12
Figure 11.3 OREAS 211 - VER_AB Results - Au ppm 12
Figure 11.4 OREAS 219 - VER_AB Results - Au ppm 13
Figure 11.5 OREAS 219 - ALS_IRL Results - Au ppm 13
Figure 11.6 OREAS 235 - VER_AB Results - Au ppm 14
Figure 11.7 OREAS 237 - MSA_YAM Results - Au ppm 14
Figure 11.8 OREAS 237 - VER_AB Results - Au ppm 15
Figure 11.9 OREAS 237 - VALS_IRL - Au ppm 15
Figure 11.10 OREAS 239 - MSA_YAM Results - Au ppm 16
Figure 11.11 OREAS 239 - VER_AB Results - Au ppm 16
Figure 11.12 OREAS 239 - ALS_IRL Results - Au ppm 17
Figure 11.13 OREAS 250b - MSA_YAM Results - Au ppm 17
Figure 11.14 OREAS 250b - VER_AB Results - Au ppm 18
Figure 11.15 OREAS 250b - ALS_IRL Results - Au ppm 18
Figure 11.16 OREAS 251 - VER_AB Results - Au ppm 19
Figure 11.17 OREAS 253 - MSA_YAM Results - Au ppm 19

Figure 11.18 OREAS 253 - VER_AB Results - Au ppm....20
Figure 11.19 OREAS 253 - ALS_IRL Results - Au ppm....20
Figure 11.20 OREAS 253b - MSA_YAM Results - Au ppm....21
Figure 11.21 OREAS 253b - VER_AB Results - Au ppm....21
Figure 11.22 OREAS 256b - MSA_YAM Results - Au ppm....22
Figure 11.23 OREAS 256b - VER_AB Results - Au ppm....22
Figure 11.24 OREAS 256b - ALS_IRL Results - Au ppm....23
Figure 11.25 OREAS 22h Blank - MSA_YAM Results - Au ppm....25
Figure 11.26 OREAS 22h Blank - VER_AB Results - Au ppm....25
Figure 11.27 OREAS 22h Blank - ALS_IRL Results - Au ppm....26
Figure 11.28 CBLKY Coarse Blank - MSA_YAM Results - Au ppm....26
Figure 11.29 CBLKY Coarse Blank - VER_AB Results - Au ppm....27
Figure 11.30 CBLKY Coarse Blank - ALS_IRL Results - Au ppm....27
Figure 11.31 Prospect ATI, RC Field Duplicate Scatter Plot - To Date - Au ppm....35
Figure 11.32 Prospect ATI, RC Field Duplicate Q-Q Plot - To Date - Au ppm....35
Figure 11.33 Prospect ATI, RC Field Duplicate RMDP Plot - To Date - Au ppm....36
Figure 11.34 Prospect ATI, DD Field Duplicate Scatter Plot - To Date - Au ppm....36
Figure 11.35 Prospect ATI, DD Field Duplicate Q-Q Plot - To Date - Au ppm....37
Figure 11.36 Prospect ATI, DD Field Duplicate RMDP Plot - To Date - Au ppm....37
Figure 11.37 Prospect CHG, RC Field Duplicate Scatter Plot - To Date - Au ppm....38
Figure 11.38 Prospect CHG, RC Field Duplicate Q-Q Plot - To Date - Au ppm....38
Figure 11.39 Prospect CHG, RC Field Duplicate RMDP Plot - To Date - Au ppm....39
Figure 11.40 Prospect CHG, DD Field Duplicate Scatter Plot - To Date - Au ppm....39
Figure 11.41 Prospect CHG, DD Field Duplicate Q-Q Plot - To Date - Au ppm....40
Figure 11.42 Prospect CHG, DD Field Duplicate RMDP Plot - To Date - Au ppm....40
Figure 11.43 Prospect CHG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm....41
Figure 11.44 Prospect CHG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm....41
Figure 11.45 Prospect CHG, DD Pulp Duplicate RMDP Plot - To Date - Au ppm....42
Figure 11.46 Prospect ENI, RC Field Duplicate Scatter Plot - To Date - Au ppm....42
Figure 11.47 Prospect ENI, RC Field Duplicate Q-Q Plot - To Date - Au ppm....43
Figure 11.48 Prospect ENI, RC Field Duplicate RMDP Plot - To Date - Au ppm....43
Figure 11.49 Prospect HAN, RC Field Duplicate Scatter Plot - To Date - Au ppm....44
Figure 11.50 Prospect HAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm....44
Figure 11.51 Prospect HAN, RC Field Duplicate RMDP Plot - To Date - Au ppm....45
Figure 11.52 Prospect HAN, DD Pulp Duplicate Scatter Plot - To Date - Au ppm....45
Figure 11.53 Prospect HAN, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm....46
Figure 11.54 Prospect HAN, DD Pulp Duplicate RMDP Plot - To Date - Au ppm....46
Figure 11.55 Prospect HND, RC Field Duplicate Scatter Plot - To Date - Au ppm....47

Figure 11.56 Prospect HND, RC Field Duplicate Q-Q Plot - To Date - Au ppm....47
Figure 11.57 Prospect HND, RC Field Duplicate RMDP Plot - To Date - Au ppm....48
Figure 11.58 Prospect KEK, RC Field Duplicate Scatter Plot - To Date - Au ppm....48
Figure 11.59 Prospect KEK, RC Field Duplicate Q-Q Plot - To Date - Au ppm....49
Figure 11.60 Prospect KEK, RC Field Duplicate RMDP Plot - To Date - Au ppm....49
Figure 11.61 Prospect KEK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm....50
Figure 11.62 Prospect KEK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm....50
Figure 11.63 Prospect KEK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm....51
Figure 11.64 Prospect KLG, RC Field Duplicate Scatter Plot - To Date - Au ppm....51
Figure 11.65 Prospect KLG, RC Field Duplicate Q-Q Plot - To Date - Au ppm....52
Figure 11.66 Prospect KLG, RC Field Duplicate RMDP Plot - To Date - Au ppm....52
Figure 11.67 Prospect KLG, DD Field Duplicate Scatter Plot - To Date - Au ppm....53
Figure 11.68 Prospect KLG, DD Field Duplicate Q-Q Plot - To Date - Au ppm....53
Figure 11.69 Prospect KLG, DD Field Duplicate RMDP Plot - To Date - Au ppm....54
Figure 11.70 Prospect KLG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm....54
Figure 11.71 Prospect KLG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm....55
Figure 11.72 Prospect KLG, DD Pulp Duplicate RMPD Plot - To Date - Au ppm....55
Figure 11.73 Prospect NAR, RC Field Duplicate Scatter Plot - To Date - Au ppm....56
Figure 11.74 Prospect NAR, RC Field Duplicate Q-Q Plot - To Date - Au ppm....56
Figure 11.75 Prospect NAR, RC Field Duplicate RMDP Plot - To Date - Au ppm....57
Figure 11.76 Prospect NOK, RC Field Duplicate Scatter Plot - To Date - Au ppm....57
Figure 11.77 Prospect NOK, RC Field Duplicate Q-Q Plot - To Date - Au ppm....58
Figure 11.78 Prospect NOK, RC Field Duplicate RMDP Plot - To Date - Au ppm....58
Figure 11.79 Prospect NOK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm....59
Figure 11.80 Prospect NOK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm....59
Figure 11.81 Prospect NOK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm....60
Figure 11.82 Prospect SAN, RC Field Duplicate Scatter Plot - To Date - Au ppm....60
Figure 11.83 Prospect SAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm....61
Figure 11.84 Prospect SAN, RC Field Duplicate RMDP Plot - To Date - Au ppm....61
Figure 11.85 Prospect SOL, RC Field Duplicate Scatter Plot - To Date - Au ppm....62
Figure 11.86 Prospect SOL, RC Field Duplicate Q-Q Plot - To Date - Au ppm....62
Figure 11.87 Prospect SOL, RC Field Duplicate RMDP Plot - To Date - Au ppm....63
Figure 11.88 Prospect SWA, RC Field Duplicate Scatter Plot - To Date - Au ppm....63
Figure 11.89 Prospect SWA, RC Field Duplicate Q-Q Plot - To Date - Au ppm....64
Figure 11.90 Prospect SWA, RC Field Duplicate RMDP Plot - To Date - Au ppm....64
Figure 11.91 Prospect SWA, DD Field Duplicate Scatter Plot - To Date - Au ppm....65
Figure 11.92 Prospect SWA, DD Field Duplicate Q-Q Plot - To Date - Au ppm....65
Figure 11.93 Prospect SWA, DD Field Duplicate RMDP Plot - To Date - Au ppm....66

Figure 11.94 Prospect SWA, Pulp Duplicate Scatter Plot - To Date - Au ppm....66
Figure 11.95 Prospect SWA, Pulp Duplicate Q-Q Plot - To Date - Au ppm....67
Figure 11.96 Prospect SWA, Pulp Duplicate RMPD Plot - To Date - Au ppm....67
Figure 11.97 Prospect THN, RC Field Duplicate Scatter Plot - To Date - Au ppm....68
Figure 11.98 Prospect THN, RC Field Duplicate Q-Q Plot - To Date - Au ppm....68
Figure 11.99 Prospect THN, RC Field Duplicate RMDP Plot - To Date - Au ppm....69
Figure 11.100 Prospect VAKO, RC Field Duplicate Scatter Plot - To Date - Au ppm....69
Figure 11.101 Prospect VAKO, RC Field Duplicate Q-Q Plot - To Date - Au ppm....70
Figure 11.102 Prospect VAKO, RC Field Duplicate RMDP Plot - To Date - Au ppm....70
Figure 11.103 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 5 m....73
Figure 11.104 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 7.5 m....73
Figure 11.105 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 10 m....74
Figure 11.106 Log-Probability Plots Between Au from DD Holes and RC-Migrated Holes with Migration Distances of 5 m (a), 7.5 m (b), and 10 m (c)....75
Figure 11.107 RC-DD Comparison - NOK....76
Figure 11.108 RC-DD Comparison - CHG....77
Figure 11.109 Suspected RC Contamination - CHG....78
Figure 11.110 Suspected RC Contamination - HAN....79

LIST OF TABLES

Table 11.1 Analytical Methods Used in the Doropo Project for Gold Assays....4
Table 11.2 Average Density Values and Number of Samples Taken for Each Oxidation Profile....5
Table 11.3 Average Density for Each Rock Type....6
Table 11.4 Nominal QAQC Sample Insertion Rates for Doropo....8
Table 11.5 Actual QAQC Sample Insertion Rates for the Doropo Gold Project - Project to Date....9
Table 11.6 Doropo Gold Project CRM Breakdown (September 2021 - July 2023)....10
Table 11.7 Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Performance Summary by Type and Laboratory - Au (ppm)....11
Table 11.8 Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Blanks Breakdown....24
Table 11.9 Doropo Gold Project - To Date Period - Duplicates Breakdown by Prospect....29
Table 11.10 Reverse Circulation Field Original vs. Duplicate Statistics - To Date....31
Table 11.11 Diamond Drilling Field Original vs. Duplicate Statistics - To Date....33
Table 11.12 Diamond Drilling Pulp Original vs. Duplicate Statistics - To Date....34
Table 11.13 Field Duplicates Performance by Deposit - Project to Date....71
Table 11.14 Pulp Duplicate Performance by Deposit - Project to Date....72

Table 11.15 Paired Bivariate Statistics Between Au Values from DD Holes and Au Values from RC Holes Migrated to the DD Holes (the Migrated Distance is Shown After the Variable Name) 72

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

Bureau Veritas Minerals Laboratory Abidjan, Côte d'Ivoire (VER_AB), was the primary analytical laboratory used for gold fire assay on the Doropo Project. VER_AB is currently acquiring its accreditation (completion expected Q3 2024) while aligned with ISO 9001:2015, ISO 14001:2015, ISO 45001:2018 held by head office. VER_AB also uses the same standard operating procedures as Bureau Veritas Minerals Laboratory Vancouver who are accredited under ISO 17025. VER_AB actively participates every year in the CANMET and Geostats round-robin proficiency tests. The Bureau Veritas Minerals head office is in Paris, France, and Bureau Veritas Minerals is independent of Centamin.

The OMAC Laboratories Ltd ALS Loughrea laboratory in Galway, Ireland (ALS_IRL) was used as a secondary laboratory. ALS_IRL is accredited by the Irish National Accreditation Board (INAB) to undertake testing as detailed in the scope bearing the registration number 173T, in conformity with ISO/IEC 17025:2017. ALS_IRL is independent of Centamin.

MSA Laborarory Yamoussoukro (MSA_YAM) was utilised in 2023 to conduct Chrysos photon analysis on 500 g, 2 mm crushed samples. MSA_YAM is accredited by the International Accreditation Service (IAS) which accredits laboratories to ISO/IEC Standard 17025 (Testing and Calibration) and Intertek for both ISO/IEC Standard 17025 (Testing and Calibration Laboratories) and ISO 9001 (Quality Management Systems) accreditation.

Centamin established an on-site sample preparation laboratory at Danoa in late 2021, the construction of which was observed by the Qualified Person in August 2021. Prior to this preparation facility becoming operational, all samples were prepared at either VER_AB or ALS_IRL.

Drilling has been carried in several campaigns, spread across the time period from 2015 to July 2023. For the purposes of this report, the QAQC analysis of the field duplicates is on the whole dataset and split on deposit, while for the certified reference materials (standards/CRMs) the analysis was conducted on samples submitted from September 2021 to July 2023.

11.1 Reverse Circulation Sampling Methods

RC drill holes are sampled in 1 m intervals. Each 1 m bulk sample is collected from the cyclone splitter, placed in a large plastic bag, and labelled with Hole ID and a sampling interval number. The bulk sample weight is recorded. The 1 m bulk sample is passed through a 3-tier riffle splitter and the primary split sample is collected in a plastic sample bag, labelled with a Sample ID and sent to the laboratory for assay analysis. The secondary split of 1/8th of the remaining 7/8th is taken for storage in the camp. The sample condition (Dry, Wet or Moist) is also recorded.

During the PFS and DFS campaigns, the sampling was undertaken in the predefined mineralisation zone of ±15 m off the hanging wall and footwall of the interpreted mineralisation zone. The predefined zone is given to the geologist in the field as a preliminary drill fence cross-section interpretation.

11.2 Diamond Core Sampling Methods

DD is typically drilled using an HQ bit from the surface down to the top of fresh rock material, then NTW or NQ to the end of the hole. DD sampling intervals were defined after geological logging was completed. Sampling is constrained within the lithological domain at every 1 m interval. Quartz veins with at least 0.5 m width are defined and sampled separately. Sampling intervals are marked up on the core and assigned with SAMPLE ID. The core is cut in half; one half is placed in a labelled plastic bag, assigned as the primary sample and sent to the laboratory for assay analysis. The remaining half is returned to the original core tray and retained for storage.

11.3 Chain of Custody and Transport

All RC samples and core trays are transported by Centamin personnel between the drill sites and the sample processing facility. The processing area consists of an open logging area for core trays and a covered sample handling area for the staging of the RC and DD samples for transport. The sample processing area is adjacent to the main office and in the main compound. The compound is completely fenced and under 24 hour guard.

The core is laid out, logged and sampled by Centamin personnel. After RC and core samples are prepared, they are placed in sealed polyweave in groups of 10 - 15 samples per sack.

Samples are transported to Abidjan by a VER_AB truck directly to the lab facility. A sample submission form accompanies each shipment of samples. An email copy of the submission form and sample list is also sent to the laboratory. BV Abidjan signs and sends back a scanned copy of the submission form to acknowledge formal receipt of the samples when they take custody of the samples.

All pulp rejects are returned by VER_AB transport to the site office and stored in locked shipping containers.

For shipments of samples to ALS_IRL , a systematic and secure packaging process is followed. The samples are first packed in lots of 20 sample packets, which are then placed into plastic bags and securely sealed. These sample bags are further organized into sturdy blue plastic 44-gallon barrels with screw-top lids, each capable of holding up to 200 pulp samples. To ensure proper identification and tracking, each barrel is labelled with the despatch number, sample interval, destination address, and barrel number. The barrels are then transported from the site to the Centamin office in Abidjan. They are kept securely in a locked room until all the export documents are complete.

To facilitate the shipping process, a comprehensive sample shipment checklist is prepared. This includes a sample submission form, sample list, and an invoice, which may be required for customs purposes. The shipment also involves several essential documents, such as:

- Commercial Invoice: This document clarifies that the samples hold no commercial value;
- Declaration of Non-Radioactivity: To confirm that the contents i.e., drill pulps, are not radioactive;
- Sample Shipment Details: This document contains all the necessary information about the samples being shipped;
- Sample Export Authorisation: Obtained from the General Directorate of Mine and Geology in Abidjan, this authorization allows for the legal export of the samples.

Once the export authorization is received, the barrels, along with all relevant documents, are dispatched to DHL for shipment to ALS_IRL, via aircraft. DHL provides a waybill via email to enable easy tracking of the shipment. At the same time, all the documents and waybill are sent to ALS_IRL by email as well.

Typically, an email from ALS_IRL is sent to Centamin indicating that all samples have been received. No formal receipt is received. On completion of the analytical process by ALS the samples remain after the free storage period, and then the laboratory responsibly disposes of the pulp rejects.

11.4 Sample Preparation

11.4.1 RC Samples

For the PFS and DFS campaigns (September 2021 to July 2023), chip samples were transported to the VER_AB up until December 2021. An onsite sample preparation laboratory was constructed at Danoa. Pulp samples were subsequently submitted from the Danoa facility to VER_AB and ALS_IRL for analysis. Danoa Laboratory sample preparation procedures are as follows:

- Oven dry RC chip samples;
- Crushed sample to 75% passing 2 mm;
- Riffle split to 800 g - 1,000 g, retain coarse reject;
- Pulverise 800 g - 1,000 g to 85% passing 75 µm;
- Collect 150 g for analysis;
- Retain pulp reject.

11.4.2 DD Samples

For the Pre-PFS campaign, core sample was transported to the VER_AB. VER_AB sample preparation procedures were as follows:

- Oven dry core samples;

- Crushed core to 70% passing 2 mm;
- Riffle split 1,000 g, retain coarse reject;
- Pulverise 1,000 g to 85% passing 75 µm;
- Collect 250 g for analysis;
- Retain pulp reject.

For the PFS and DFS campaigns, core samples were transported to the VER_AB until December 2021, when the onsite facility was commissioned. Pulp samples were subsequently generated at the Danoa onsite preparation facility before being submitted to the BV facility in Abidjan and ALS_IRL for analysis. Danoa Laboratory core sample preparation procedures are as follows:

- Oven dry core samples;
- Crushed core to 75% passing 2 mm;
- Riffle split to 800 g - 1,000 g, retain coarse reject;
- Pulverise 800g - 1,000 g to 85% passing 75 µm;
- Collect 150 g for analysis;
- Retain pulp reject.

11.5 Sample Analysis at Laboratory

A standard fire assay for gold was undertaken at both VER_AB and ALS_IRL, but with different finishes. A 50 g sub-sample is taken from the pulverised material, mixed with flux and then fired. The resultant lead button is then transformed to a prill using cupellation. The prill is dissolved in Aqua Regia solution and the resultant liquor read by either AAS (VER_AB) or ICP-AES (ALS_IRL). Fire assay is considered to be a total gold content assaying technique. For over-range results, typically 10 g/t Au or above, a gravimetric finish was undertaken and used in preference to the AAS or ICP-AES reading. The photon assay method, undertaken on 500 g of coarsely crushed sample material, was utilised at MSA_YAM. The analytical methods used are summarised Table 11.1.

Laboratory	Period	Generic Method	Lab Code	Detection Limit (ppm)
MSA_YAM	DFS (2023)	Au_CPA_Au1_ppm	CPA_Au1	0.015
VER_AB	DFS (2023)	Au_FA450_ppm	FA450	0.010
VER_AB	DFS (2023)	Au_FA550_ppm	FA550	0.05
VER_AB	PFS (2021 - 2022)	Au_FA450_ppm	FA450	0.010
VER_AB	PFS (2021 - 2022)	Au_Au_ICP22_ppm	ICP22	0.001
ALS_IRL	PFS (2021 - 2022)	Au_Au_ICP22_ppm	ICP22	0.001
ALS_IRL	PFS (2021 - 2022)	Au_Au_GRA22_ppm	GRA22	0.050
VER_AB	Pre PFS (2015- 2020)	Au_FA450_ppm	FA450	0.010
VER_AB	Pre PFS (2015- 2020)	Au_FA550_ppm	FA550	0.05

Table 11.1 Analytical Methods Used in the Doropo Project for Gold Assays

A more detailed description of the analytical methods is as follows:

- Au_FA450_ppm - Bureau Veritas Fire Assay using a 50 g charge, AAS finish DL (0.01 ppm);
- Au_FA550_ppm - Over range method for Au_FA450_ppm; Bureau Veritas Fire Assay using a 50 g charge, Gravimetric Finish using a Micro - Balance DL (0.05 ppm);
- Au_Au_ICP22_ppm - ALS Fire Assay and ICP-AES finish;
- Au_Au_GRA22_ppm - ALS Fire Assay and gravimetric finish for over range results;
- Au_CPA_Au1_ppm - MSALABS Chrysos Photon Assay Services using 500 g crushed samples to 2 mm.

11.6 Bulk Density Determinations

Bulk density measurements were determined on half and whole drill core using the Archimedes method (water immersion measurements) using a scale with a precision of ±0.02 g. Oxide and transition materials were coated in wax prior to the immersion process to avoid intake of water and sample breakdown.

Density measurements were taken in every lithology and/or at least two samples per core tray (i.e. every 2.5 metres down the hole) and results have been recorded on the density log sheet. Sample lengths range between 10-15 cm where possible, but <10 cm samples were used in areas of broken zones.

A total of 20,011 bulk density measurements were recorded from HQ, NTW and NQ drill core; 2,709 from oxide material, 3,486 from transitional material and 13,816 from fresh material. Of the 20,011 measurements, 12,735 were taken from granodiorites as the dominant domain with an average of 2.69 g/cm^3, 2,143 from saprolite, 3,492 from saprock, 467 from quartz veins, 354 from gneiss, 127 from amphibolite, 141 from pegmatites, 27 from aplite, 101 from diorite dyke and 424 from other rock units (refer to Table 11.2 and Table 11.3 below for weathering and rock type densities respectively).

Oxidation	Bulk Density g/cm^3			
	#Samples	Mean	Minimum	Maximum
Oxide	2,709	1.89	1.30	2.01
Transition	3,486	2.31	1.50	2.43
Fresh Rock	13,816	2.69	2.30	2.80

Table 11.2 Average Density Values and Number of Samples Taken for Each Oxidation Profile

Litho Type	#Samples	Average Bulk Density g/cm³
Granodiorite	12,735	2.69
Saprolite	2,143	1.84
Saprock	3,492	2.31
Quartz vein	467	2.65
Gneiss	354	2.73
Amphibolite	127	2.9
Pegmatite	141	2.64
Aplite	27	2.65
Diorite Dyke	101	2.86

Table 11.3 Average Density for Each Rock Type

11.7 Quality Assurance and Quality Control Sampling

Cube has reviewed and independently assessed the provided QAQC sample data that were collected from 2015 to July 2023 from the Doropo Gold Project.

Gold assay values are reported in units of ppm and values less than the lower detection limit have been replaced with a value of half the detection limit for the QAQC review.

The quality of the assay data was assessed by analysing the Certified Reference Materials (CRMs or Standards) and duplicate samples in terms of accuracy and precision. The precision analysis determines how closely the results can be repeated, while the accuracy analysis determines how similar the results are to the reported CRM value.

All projects and every assay batch should strive to achieve both high precision and accuracy. It is possible to have good accuracy without good precision or conversely good precision without good accuracy as shown in Figure 11.1. Precision analysis is measured by the use of duplicate and replicate assays, whereas accuracy analysis is measured through the use of CRMs. Calculations defining acceptable levels of accuracy and precision are based on Abzalov (2011).

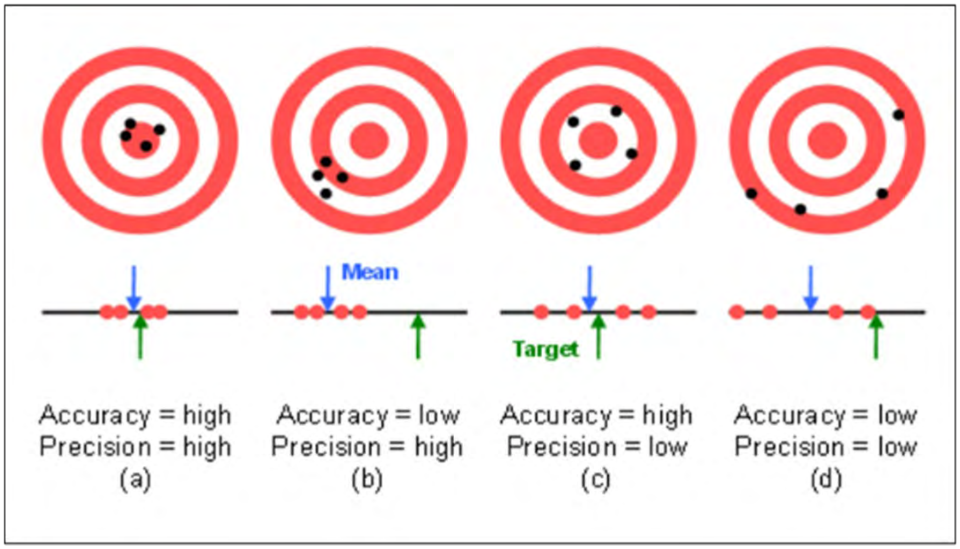


Figure 11.1 Accuracy and Precision Concepts

The QAQC for the Doropo Gold Project involves insertion of CRMs and Blank material following a set of company Standard Operating Procedures (SOPs). The geologist is responsible for the insertion of the CRM and blanks and adhering to the SOPs.

For CRMs, the pulp is transferred from the original sachet into a new plastic sachet and stapled. The standard ID is written on the new ticket and stapled on the new plastic sachet. The standards are then sent to the geologist on the rig. The geologist writes the standard sample ID on the sample sachet and photographs it. The ticket is then stapled in the sample booklet. The standard is then placed in the sample bag.

For Blank material, the samples are prepared at the camp, weighed, placed in plastic bags and stapled and then sent to the rig where the geologist assigns a sample ID.

For RC Field Duplicates, the samples are collected after the primary RC split of $1/8^{th}$ is taken. The bulk sample is once again run through the 3-tier riffle splitter to obtain a second sample that is a split of $1/8^{th}$ of the remaining $7/8^{th}$ material.

In February 2022, the above QAQC processes were changed due to the establishment of the onsite Danoa preparation laboratory. The standards are provided to the geologist at the rig. The geologist photographs the standard with an assigned sample ID with the original packaging in the field, then inserts it into the sample bag and submits it to the preparation lab. The onsite preparation lab is responsible for transferring the standards into the new packet as soon as the batch's sample preparation process has been completed.

For Blanks, an empty sample bag with an assigned sample ID ticket staple on the bag is submitted to the preparation lab, and after pulverising, the Blank sample is inserted.

For DD samples, the original half core sample is submitted to the laboratory including an empty sample bag with a duplicate sample ID ticket stapled on the bag; after crushing and pulverising, a Pulp Duplicate is sampled according to the reserved sample ID. Field Duplicates are collected from the remaining half core, which is sawn to produce a quarter core duplicate sample.

A list of samples containing the QAQC is submitted to the preparation lab to allow them to identify the different quality control samples. CRMs, Blanks, and Duplicates were initially inserted every 20 m. The nominal insertion rates in place for Doropo Gold Project are listed in Table 11.4.

In November 2021, the method was changed to insert QAQC samples consecutively within the mineralisation zone only, following discussions between the site personnel, Mr Michael Millad and Mr. Craig Barker of Centamin. In addition, a recommendation to use a limited range of CRMs was adopted, in order to allow for improved analysis of each CRM's results. A Pulp Blank has been introduced into the sequence. The samples are inserted in the following order:

- Field Duplicate;
- Coarse Blank;
- Standard;
- Pulp Blank.

Sample Type	CRMs	Duplicates	Coarse Blank
DD	5% (1 every 20 samples with Sample ID ending 5, 25, 45, 65, 85)	5% (1 every 20 samples with sample ID ending 15, 35, 55, 75, 95)	5% (1 every 20 samples in ODD multiples of 10)
RC	5% (1 every 20 samples with Sample ID ending 5, 25, 45, 65, 85)	5% (1 every 20 samples with sample ID ending 15, 35, 55, 75, 95)	5% (1 every 20 samples in ODD multiples of 10)

Table 11.4 Nominal QAQC Sample Insertion Rates for Doropo

A comparison of all QAQC samples submitted reported by prospect is presented in Table 11.5 and shows the breakdown of achieved QAQC insertion rates. These figures show that the achieved insertion rates for blanks mostly exceed the nominal rates, and that duplicates and CRMs sometimes fall slightly short of the desired rate - nevertheless, the most economically material deposits are considered to generally be adequately covered. Special attention must be drawn to Sanboroyo and Solo for which the insertion rate is below the recommended standard. However, these deposits are also poorly drilled and it is reasonable to believe that the rates will increase as the drilling intensifies.

Prospect	Routine Samples	Duplicates		CRMs		Blanks	
		Number	%	Number	%	Number	%
ATI	22,760	923	4%	798	4%	1,251	5%
CHG	61,929	2,820	5%	2,650	4%	4,154	7%
ENI	26,802	1,340	5%	1,117	4%	1,638	6%
HAN	28,964	1,366	5%	1,222	4%	1,864	6%
HND	17,330	706	4%	552	3%	770	4%
KEK	26,844	1,272	5%	1,214	5%	1,927	7%
KLG	90,656	4,213	5%	3,926	4%	6,618	7%
NAR	2,641	120	5%	101	4%	165	6%
NOK	38,491	1,564	4%	1,614	4%	2,418	6%
SAN	8,229	300	4%	278	3%	478	6%
SOL	3,174	76	2%	61	2%	89	3%
SWA	74,652	3,298	4%	3,110	4%	4,153	6%
THN	17,501	656	4%	520	3%	736	4%
VAKO	22,460	969	4%	834	4%	1,340	6%
Grand Total	442,433	19,623	4%	17,997	4%	27,601	6%

Table 11.5 Actual QAQC Sample Insertion Rates for the Doropo Gold Project - Project to Date

11.7.1 Certified Reference Materials (CRMs or Standards)

A number of CRMs have been used at the Doropo Project over time. A breakdown of the CRM ID's used from September 2021 to July 2023 is presented in Table 11.6.

For the purposes of this report, an analysis of CRMs has only been undertaken for samples submitted during the PFS-DFS period (September 2021 to July 2023) and their performance has been analysed for each laboratory separately. The CRMs submitted during this time period comprise a large proportion of the total number submitted in the Project to date.

The performance of CRMs pre the PFS-DFS has been completed previously (Cube, 2022).

LAB_ID	CRM_ID	Standard Type	Expected Au ppm	First Use	Last Use
VER_AB	OREAS219	CRM	0.76	13-Sep-21	13-Sep-21
VER_AB	OREAS235	CRM	1.59	13-Sep-21	13-Sep-21
VER_AB	OREAS237	CRM	2.21	13-Sep-21	13-Sep-21
VER_AB	OREAS239	CRM	3.55	13-Sep-21	16-Mar-23
VER_AB	OREAS250b	CRM	0.332	13-Sep-21	19-Dec-22
VER_AB	OREAS256b	CRM	7.84	13-Sep-21	13-Sep-21
ALS_IRL	OREAS219	CRM	0.76	1-Mar-22	1-Mar-22
ALS_IRL	OREAS237	CRM	2.21	1-Mar-22	1-Mar-22
ALS_IRL	OREAS239	CRM	3.55	1-Mar-22	1-Mar-22
ALS_IRL	OREAS250b	CRM	0.332	1-Mar-22	1-Mar-22

LAB_ID	CRM_ID	Standard Type	Expected Au ppm	First Use	Last Use
ALS_IRL	OREAS253	CRM	1.22	1-Mar-22	1-Mar-22
ALS_IRL	OREAS256b	CRM	7.84	2-Mar-22	2-Mar-22
ALS_IRL	OREAS22h	CRM Blank	0.005	1-Apr-22	1-Apr-22
VER_AB	OREAS22h	CRM Blank	0.0025	3-May-22	7-Jul-22
VER_AB	OREAS211	CRM	0.768	7-Dec-22	7-Jan-23
MSA_YAM	OREAS221	CRM	1.06	4-Jan-23	4-Jan-23
MSA_YAM	OREAS22h	CRM Blank	0.0075	4-Jan-23	4-Jan-23
MSA_YAM	OREAS238	CRM	3.03	4-Jan-23	4-Jan-23
MSA_YAM	OREAS251	CRM	0.504	4-Jan-23	4-Jan-23
VER_AB	OREAS253b	CRM	1.24	10-Jan-23	10-Jan-23
MSA_YAM	OREAS250b	CRM	0.332	3-Mar-23	3-Mar-23
MSA_YAM	OREAS253	CRM	1.22	3-Mar-23	3-Mar-23
MSA_YAM	OREAS253b	CRM	1.24	3-Mar-23	3-Mar-23
MSA_YAM	OREAS256b	CRM	7.84	3-Mar-23	3-Mar-23
MSA_YAM	OREAS211	CRM	0.768	15-Mar-23	15-Mar-23
MSA_YAM	OREAS237	CRM	2.21	16-Mar-23	16-Mar-23
MSA_YAM	OREAS239	CRM	3.55	16-Mar-23	16-Mar-23

Table 11.6 Doropo Gold Project CRM Breakdown (September 2021 - July 2023)

PFS-DFS Phase Results

A summary of CRM performance for the gold CRMs for all laboratories submitted during the PFS-DFS campaign is presented in Table 11.7. The results show that all standards pass the accuracy test, but there was a high precision failure rate. Samples suspected of being misallocated or incorrectly identified have been removed from the overall statistics. The overall performance of the laboratories is generally considered to be satisfactory.

Control charts for all CRMs analysed by each laboratory can be found from Figure 11.2 through to Figure 11.24. The control charts for each CRM are grouped together and separated for each laboratory.

Std ID	Laboratory	Expected Value	SD	No. Assays	Actual Mean	Actual SD	Bias %	Accuracy Test	Precision Test	No. Outside 2SD	% Outside 2SD	No. Outside 3SD	% Outside 3SD
OREAS 211	MSA_YAM	0.768	0.027	35	0.749	0.037	-2.52%	PASS	FAIL	8	22.86	1	2.86
	VER_AB	0.768	0.027	625	0.764	0.029	0.55	PASS	FAIL	25	4.00	0	-
OREAS 219	ALS_IRL	0.760	0.024	238	0.755	0.026	-0.67	PASS	FAIL	4	1.68	3	1.26
	VER_AB	0.760	0.024	691	0.768	0.028	1.05	PASS	FAIL	83	12.01	0	-
OREAS 235	VER_AB	1.59	0.038	117	1.581	0.053	-0.59	PASS	FAIL	24	20.51	0	-
OREAS 237	ALS_IRL	2.21	0.054	262	2.182	0.066	-1.26	PASS	PASS	13	4.96	4	1.53
	MSA_YAM	2.21	0.054	49	2.199	0.075	-0.5	PASS	FAIL	6	12.24	1	2.04
	VER_AB	2.21	0.054	1,463	2.210	0.134	0.03	PASS	FAIL	196	13.40	0	-
OREAS 239	ALS_IRL	3.55	0.086	190	3.51	0.136	-1.14	PASS	PASS	6	3.16	0	-
	MSA_YAM	3.55	0.086	43	3.573	0.095	0.66	PASS	PASS	3	6.98	1	2.33
	VER_AB	3.55	0.086	1,193	3.572	0.153	0.63	PASS	FAIL	84	7.04	1	0.08
OREAS 250b	ALS_IRL	0.332	0.011	121	0.317	0.007	-4.47	PASS	PASS	17	14.05	0	-
	MSA_YAM	0.332	0.011	54	0.325	0.018	-2.23	PASS	FAIL	21	38.89	0	-
	VER_AB	0.332	0.011	705	0.323	0.292	-2.84	PASS	FAIL	104	14.75	1	0.14
OREAS 251	VER_AB	0.504	0.015	91	0.512	0.018	1.65	PASS	FAIL	11	12.09	0	-
OREAS 253	MSA_YAM	1.22	0.044	7	1.223	0.050	0.27	PASS	PASS	1	14.29	0	-
	VER_AB	1.22	0.044	480	1.232	0.069	1.01	PASS	PASS	16	3.33	0	-
	ALS_IRL	1.22	0.044	135	1.205	0.031	-1.24	PASS	PASS	2	1.48	2	1.48
OREAS 253b	MSA_YAM	1.24	0.044	41	1.236	0.054	-0.34	PASS	FAIL	3	7.32	0	0
	VER_AB	1.24	0.044	219	1.240	0.08	0.34	PASS	PASS	4	1.83	0	-
OREAS 265b	MSA_YAM	7.84	0.207	44	7.968	0.152	1.63	PASS	PASS	1	2.27	0	-
	VER_AB	7.84	0.207	587	7.821	0.267	-0.24	PASS	FAIL	20	3.41	0	-
	ALS_IRL	7.84	0.207	112	7.705	0.212	-1.73	PASS	PASS	4	3.57	3	2.68

Total	7,502
MSA_YAM	273
VER_AB	6,171
ALS_IRL	1,058

Table 11.7 Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Performance Summary by Type and Laboratory - Au (ppm)

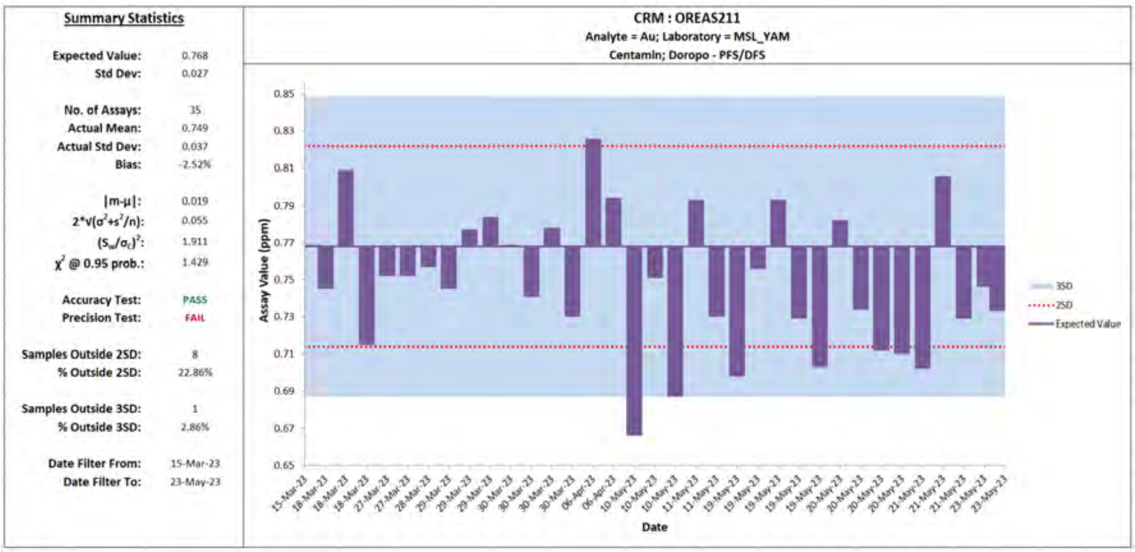


Figure 11.2 OREAS 211 - MSA_YAM Results - Au ppm

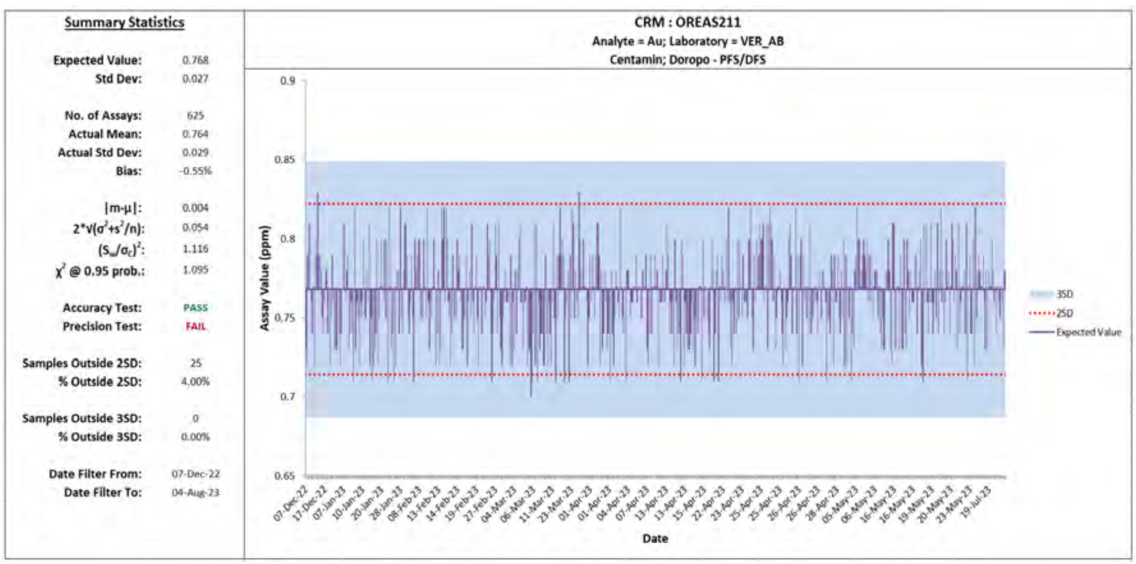


Figure 11.3 OREAS 211 - VER_AB Results - Au ppm

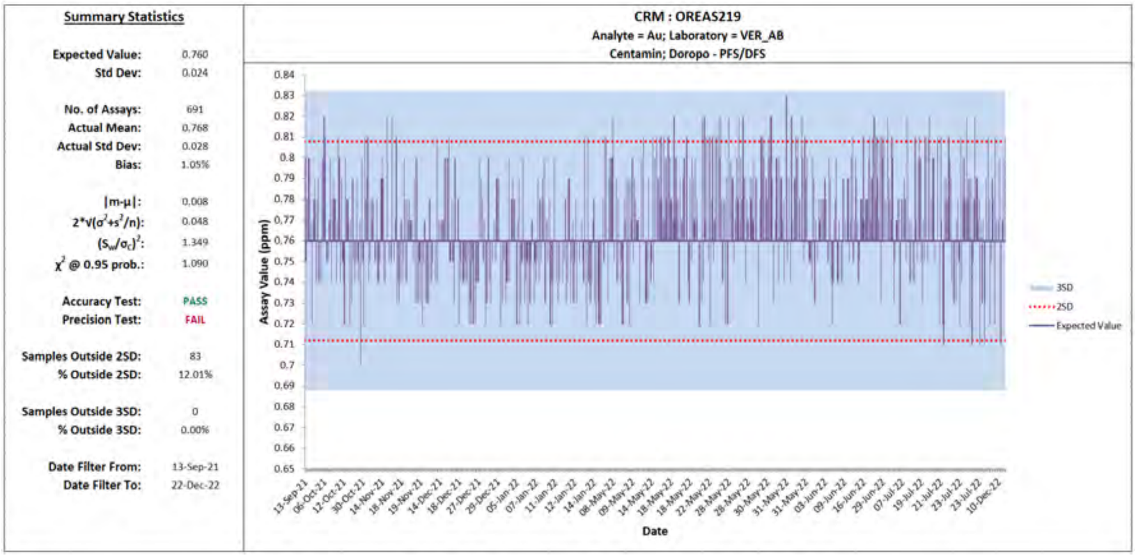


Figure 11.4 OREAS 219 - VER_AB Results - Au ppm

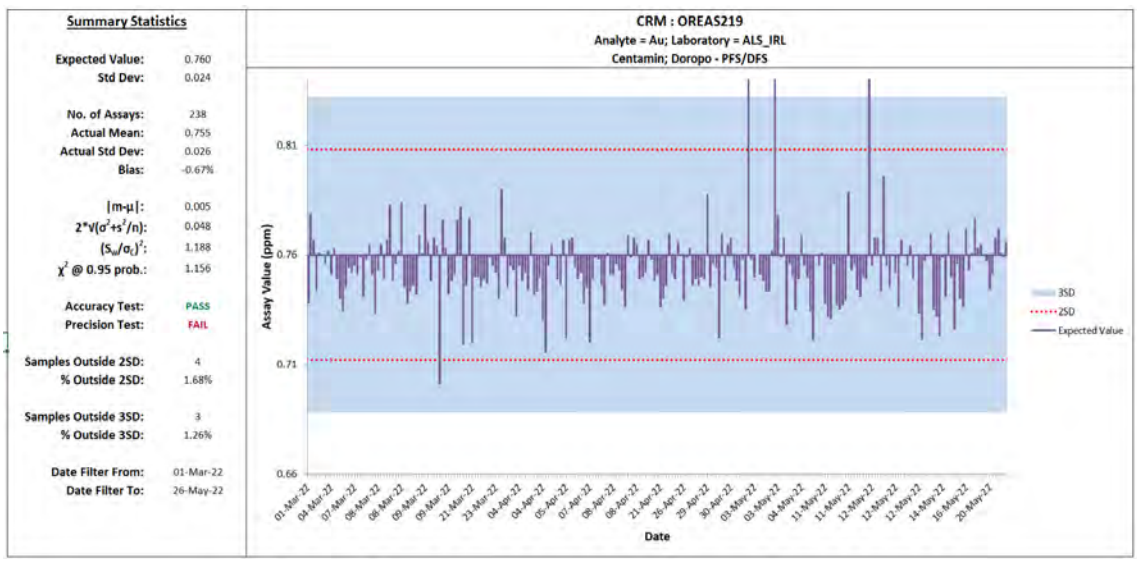


Figure 11.5 OREAS 219 - ALS_IRL Results - Au ppm

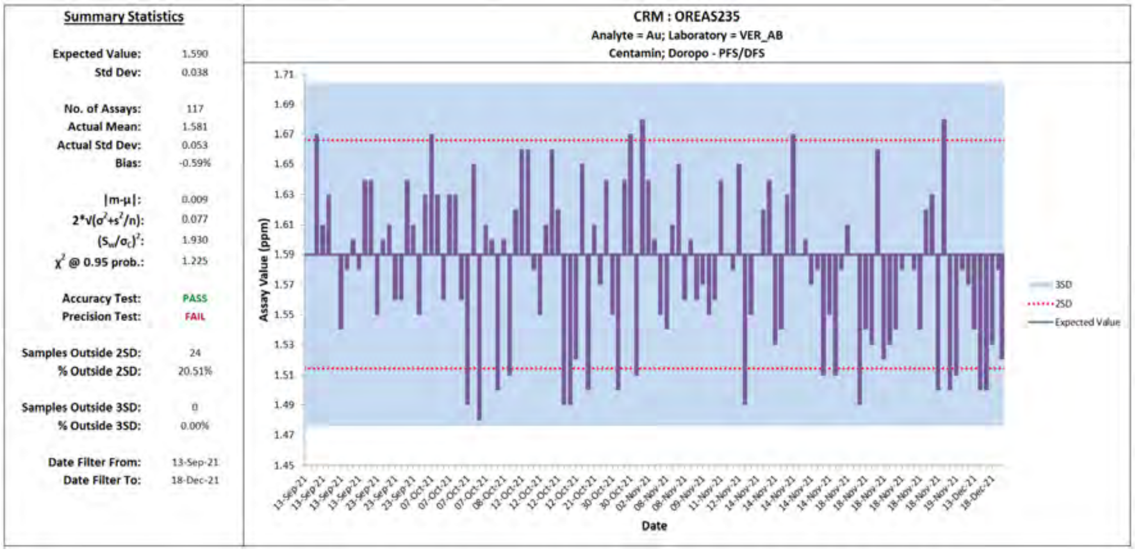


Figure 11.6 OREAS 235 - VER_AB Results - Au ppm

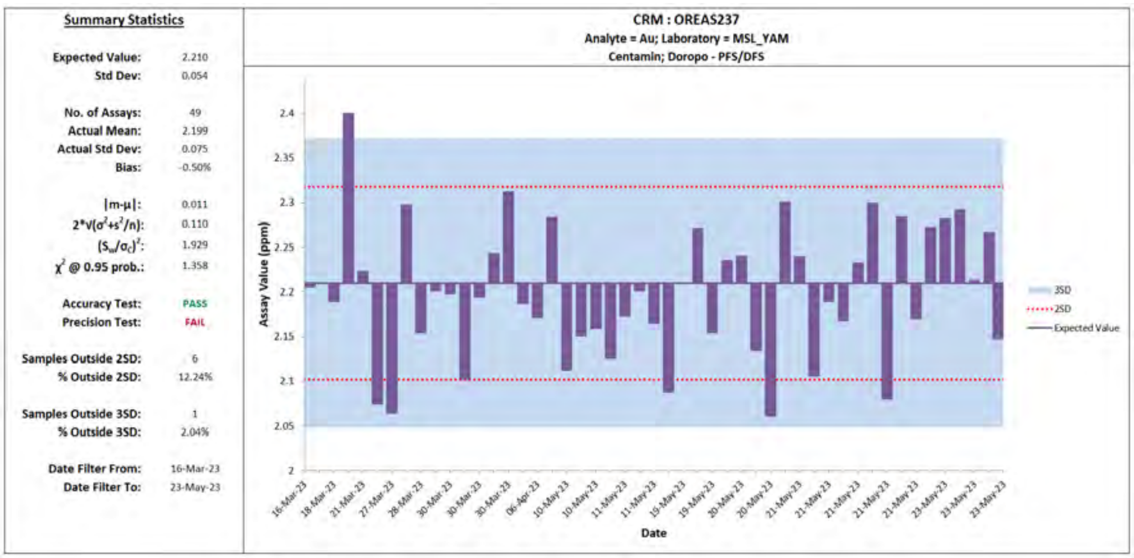


Figure 11.7 OREAS 237 - MSA_YAM Results - Au ppm

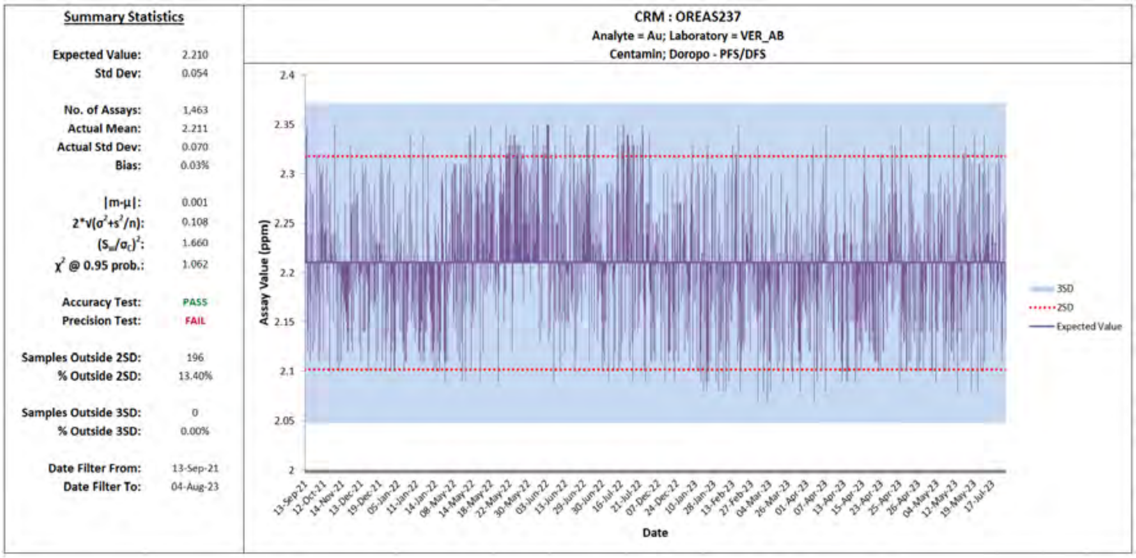


Figure 11.8 OREAS 237 - VER_AB Results - Au ppm

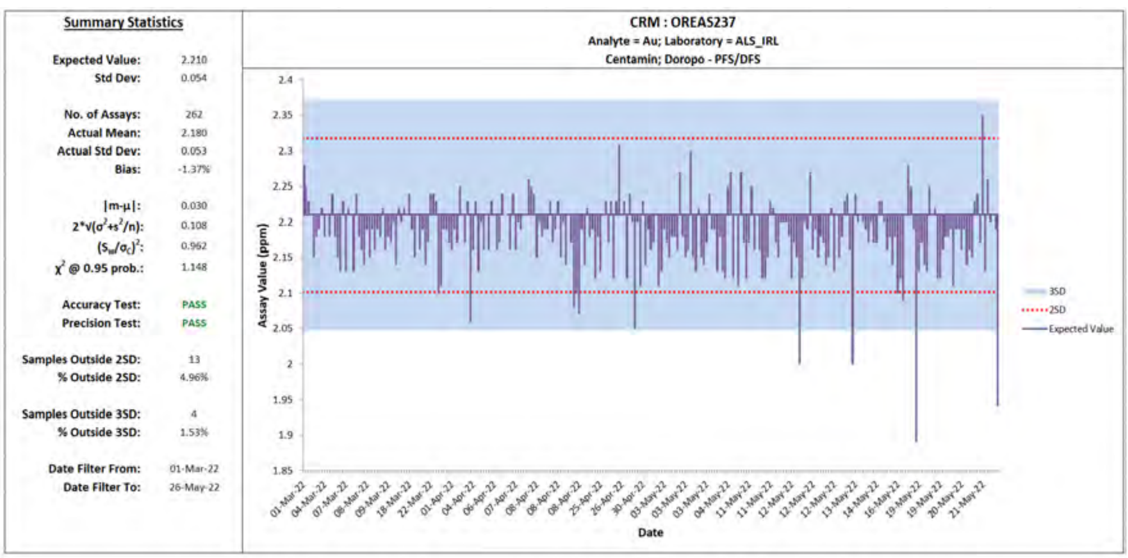


Figure 11.9 OREAS 237 - VALS_IRL - Au ppm

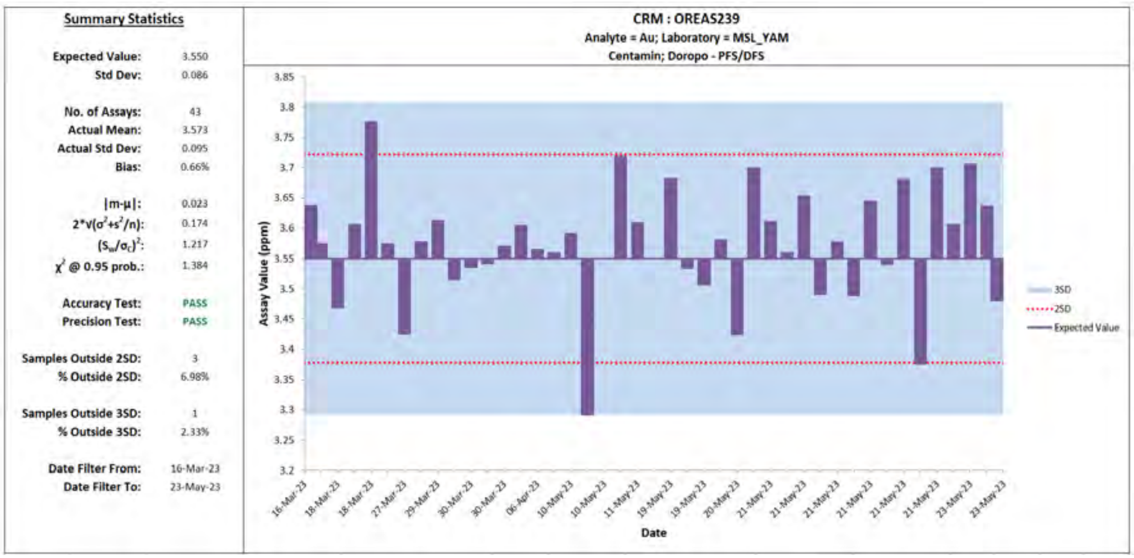


Figure 11.10 OREAS 239 - MSA_YAM Results - Au ppm

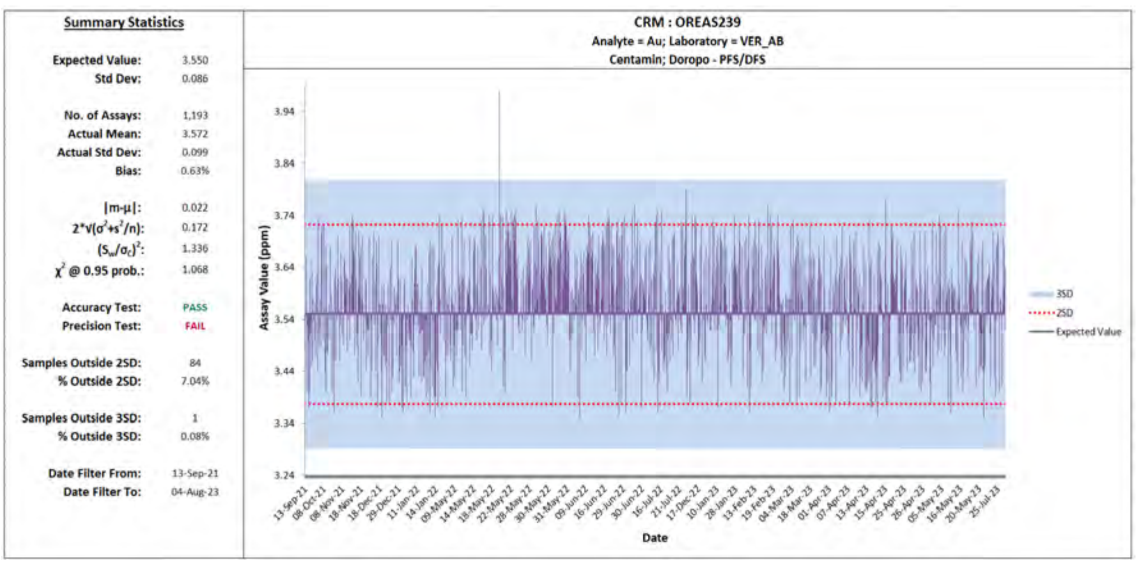


Figure 11.11 OREAS 239 - VER_AB Results - Au ppm

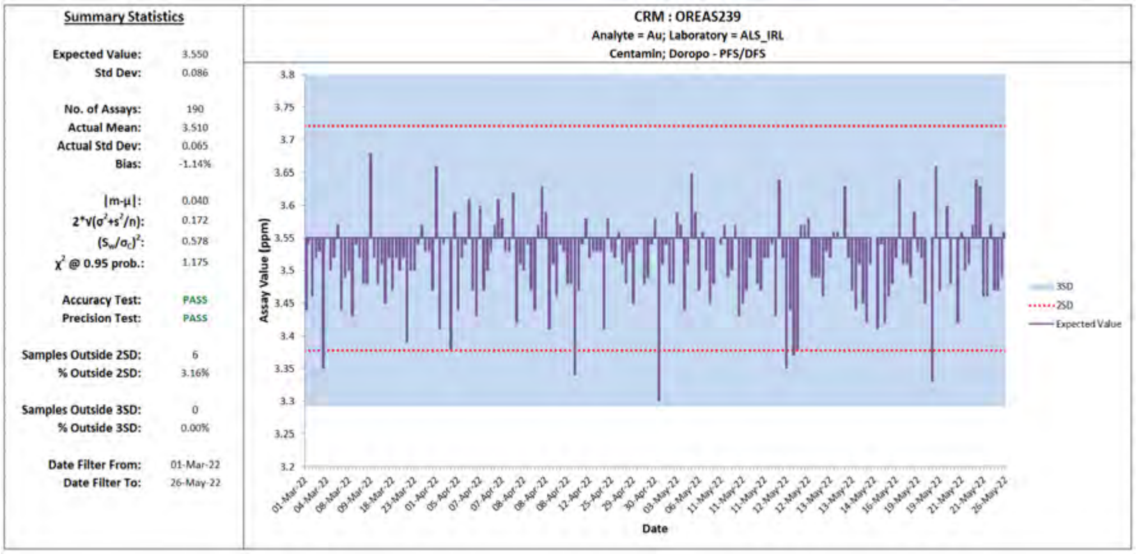


Figure 11.12 OREAS 239 - ALS_IRL Results - Au ppm

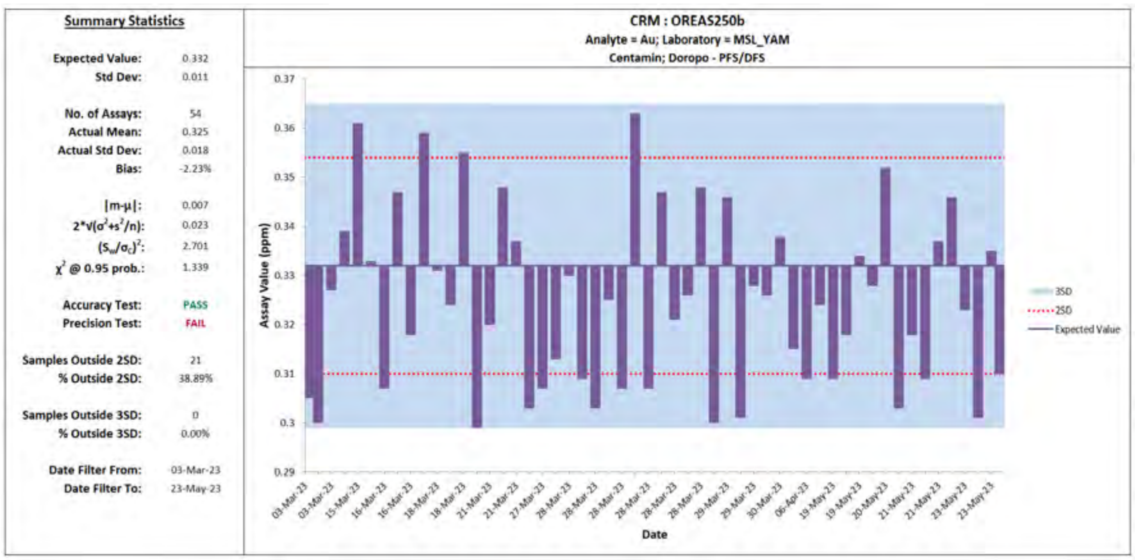


Figure 11.13 OREAS 250b - MSA_YAM Results - Au ppm

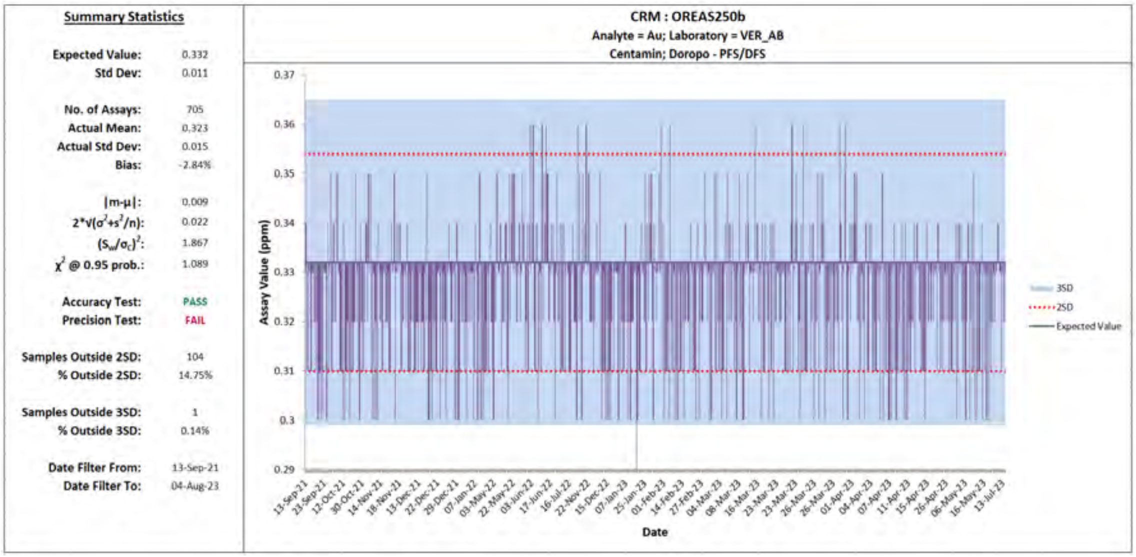


Figure 11.14 OREAS 250b - VER_AB Results - Au ppm

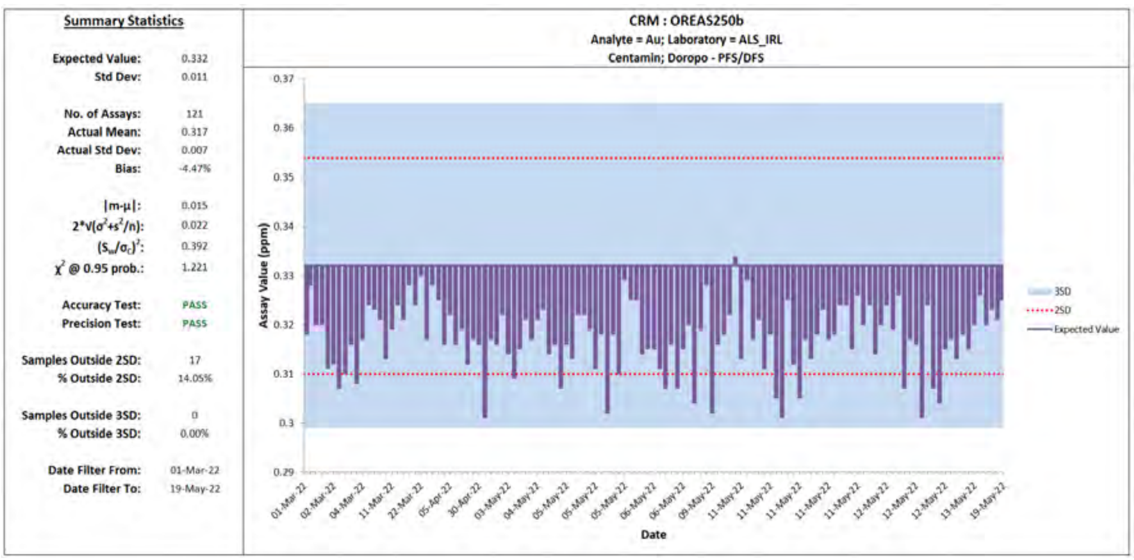


Figure 11.15 OREAS 250b - ALS_IRL Results - Au ppm

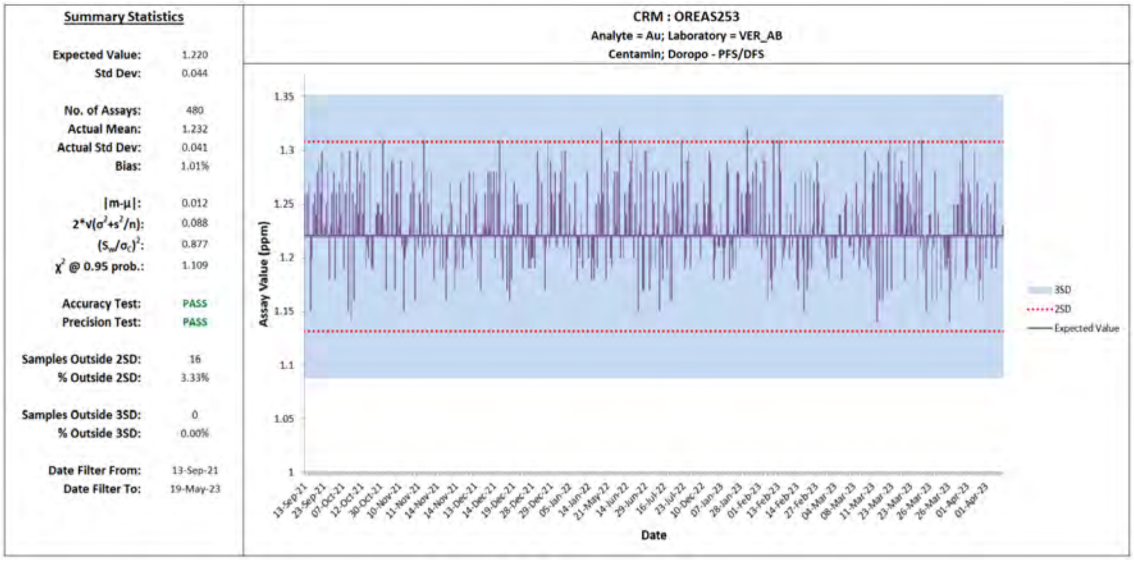


Figure 11.16 OREAS 251 - VER_AB Results - Au ppm

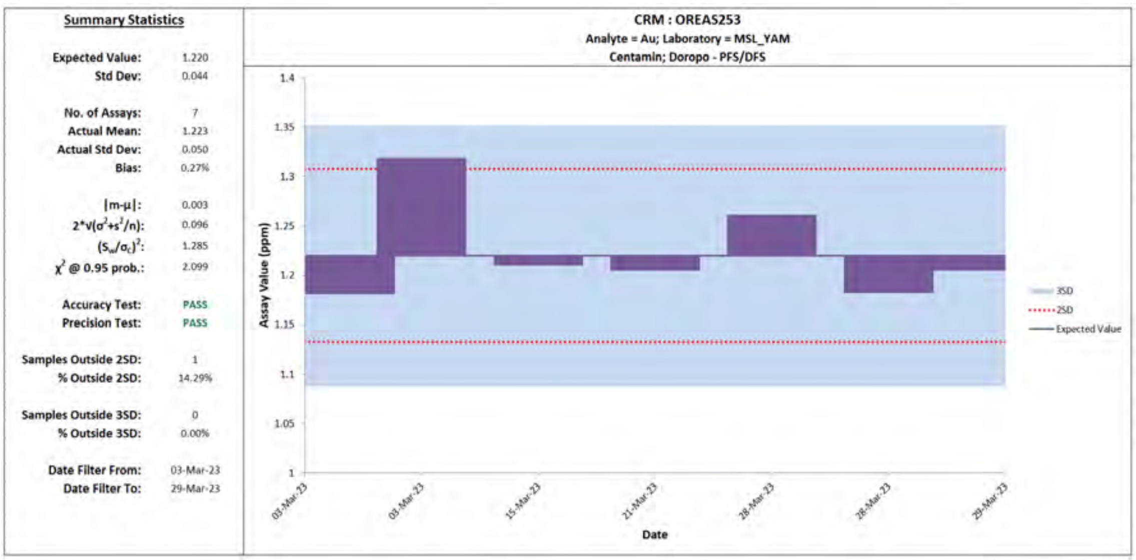


Figure 11.17 OREAS 253 - MSA_YAM Results - Au ppm

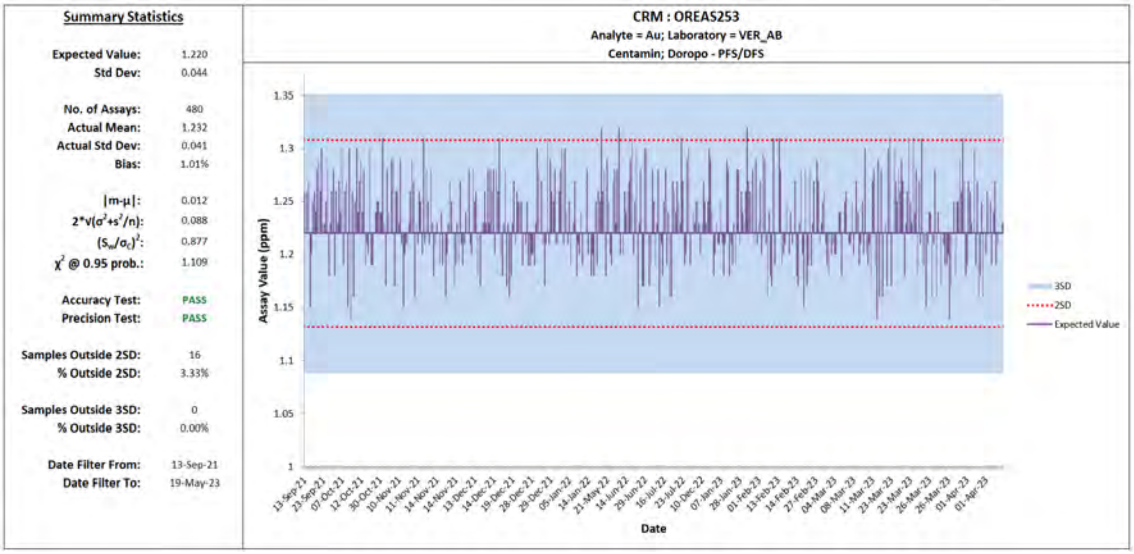


Figure 11.18 OREAS 253 - VER_AB Results - Au ppm

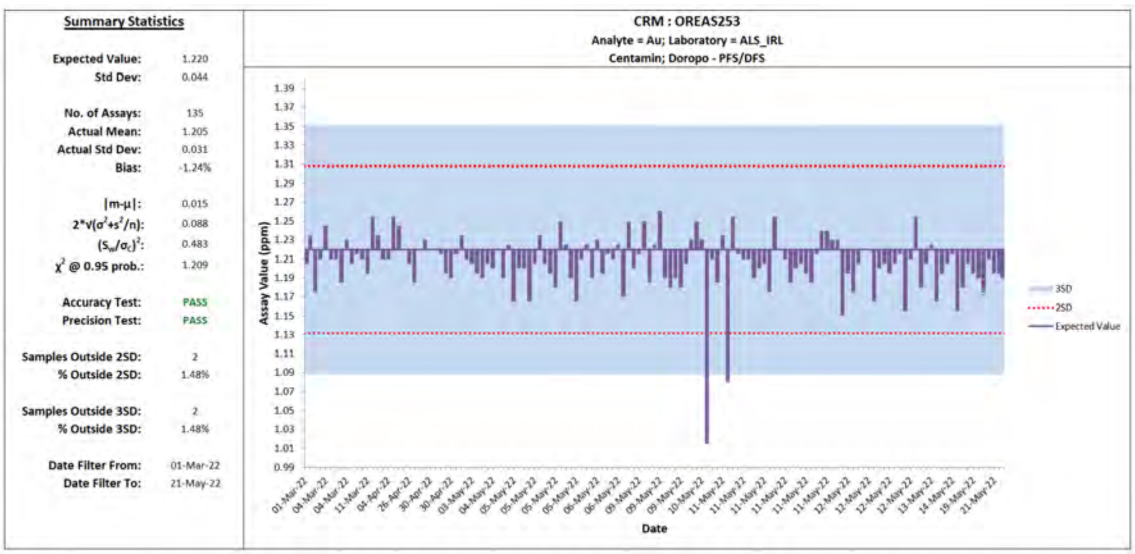


Figure 11.19 OREAS 253 - ALS_IRL Results - Au ppm

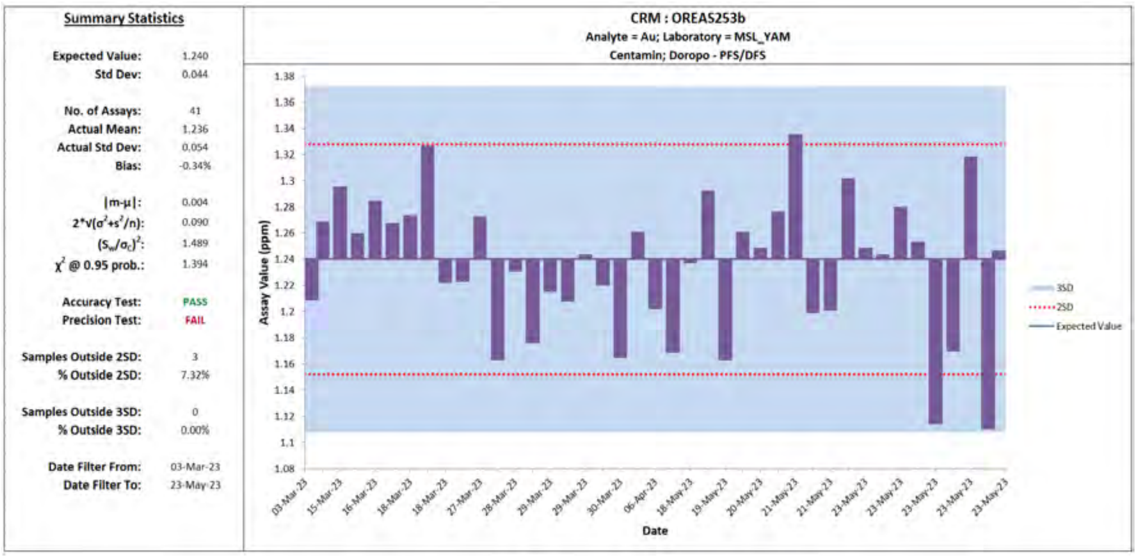


Figure 11.20 OREAS 253b - MSA_YAM Results - Au ppm

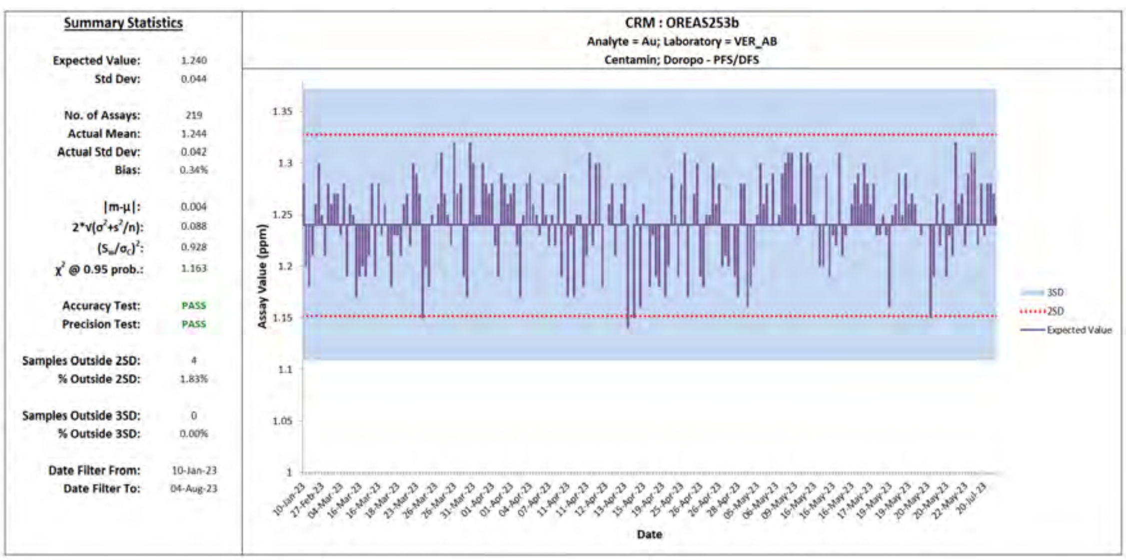


Figure 11.21 OREAS 253b - VER_AB Results - Au ppm

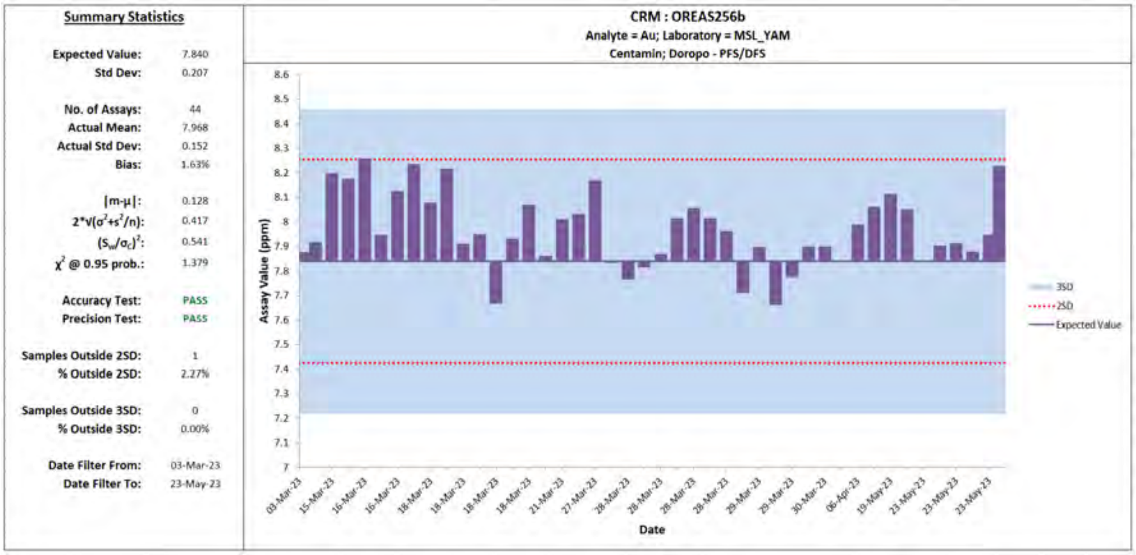


Figure 11.22 OREAS 256b - MSA_YAM Results - Au ppm

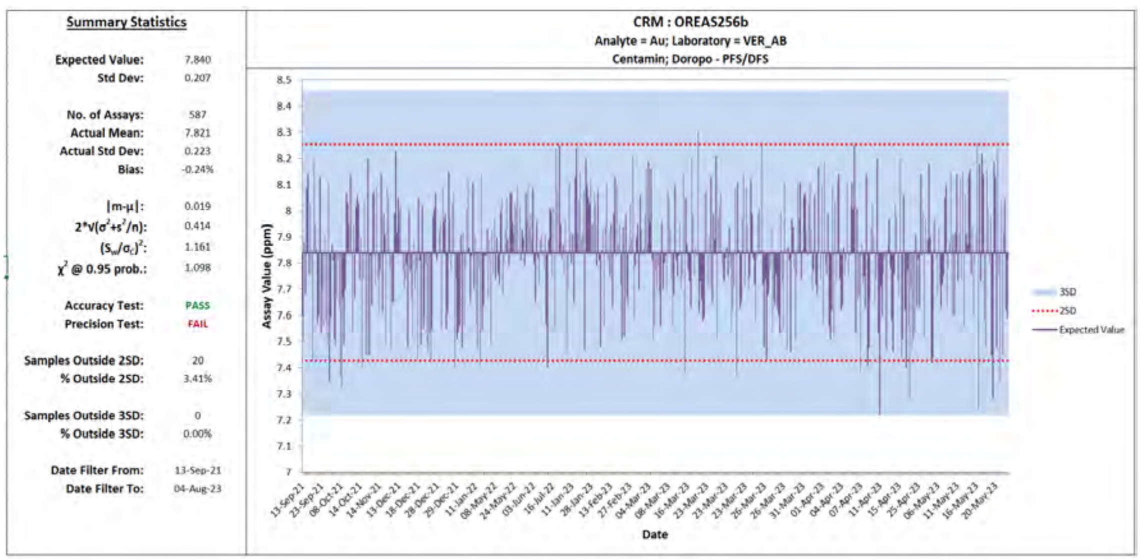


Figure 11.23 OREAS 256b - VER_AB Results - Au ppm

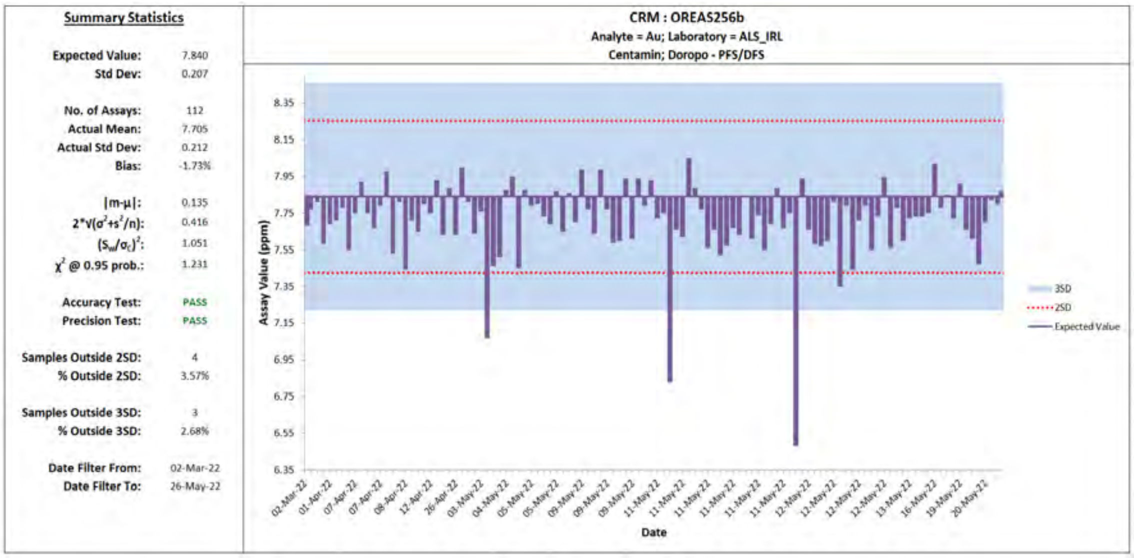


Figure 11.24 OREAS 256b - ALS_IRL Results - Au ppm

CRM Performance Comments

The following points summarise the relevant observations for the overall CRM performance for the PFS-DFS period:

- It should be noted that an analysis of the performance of CRMs analysed by MSA_YAM should be treated with caution as the analysis was completed by Photon, however the CRMs certified values are determined by Fire Assay;
- Overall accuracy performance for all laboratories is good, with all CRMs passing the accuracy test;
- In terms of precision, MSA_YAM had three out of seven CRMs pass the precision test, ALS_IRL, five out of six pass, and for VER_AB, two out of ten passed the precision test;
- In terms of failures outside 2SD, MSA_YAM had the two highest failure rates (39% and 23%) with an average failure rate of 15%. ALS_IRL has the lowest average 2SD failure rate of 4%. The high 2SD failure rate for MSA_YAM, especially at the lower grade ranges is expected as in the Qualified Person's experience the Au_CPA_Au1_ppm analysis method is less accurate at these lower grades;
- ALS_IRL CRM control charts show several CRMs returning a negative bias, especially OREAS250b with all results, except one (120 in total) below the certified mean. The negative bias is only slightly reflected in the other laboratories. All the other CRMs results show a reasonable spread around the certified mean;
- As a general comment, the CRM results are reporting with the given accuracy error limits. While there are some precision fails, the sample data are considered to be suitable from an accuracy perspective for use in Mineral Resource estimation.

11.7.2 Blanks

Blank samples (i.e. material with a very low grade) are usually inserted after high-grade mineralisation samples. The primary purpose of using blanks is to monitor the laboratory for possible contamination of samples mainly caused by poor housekeeping and insufficient cleaning of equipment. If equipment has not been appropriately cleaned, the blank samples will be contaminated, which is reflected on the chart as increased values for the element of interest.

During the DFS period, blank samples at the Doropo Gold Project consist of certified blank (CRM) Oreas 22h and a non-certified coarse blank. The coarse blank (CBLKY) is sourced from Yamoussoukro; forty samples were assayed at BV, and all samples returned values less than the Au ppm detection limit.

Details of the blanks submitted are presented in Table 11.8 and Table 11.5 by drilling campaign period.

More than 1,500 pulp blanks were submitted, however since these are not suitable for detecting contamination in the laboratory, they were not used in the analysis.

Blank ID	Laboratory	No. Samples	Mean Au (ppm)	Max Au (ppm)*
OREAS22h	MSA_YAM	271	0.021	0.024
	ALS_IRL	553	0.001	0.121
	VER_AB	4,608	0.05	0.006
CRM Blank Total		5,433		
CBLKY	MSA_YAM	279	0.049	0.059
	ALS_IRL	1,067	0.032	0.307
	VER_AB	6,198	0.016	0.28
Non-Certified Coarse Blank Total		7,544		
Grand Total		12,977		

**Obvious outliers have been removed from the final analysis*

Table 11.8 Doropo Gold Project Certified Reference Materials - PFS-DFS Period - Blanks Breakdown

PFS-DFS Phase Results

Overall performance for MSA_YAM, ALS_IRL and VER_AB for CRM OREAS 22h is acceptable. MSA_YAM and VER_AB both pass the precision test, however ALS_IRL fails in both cases, with several instances of suspected contamination. Control charts are shown in Figure 11.25 through to Figure 11.27.

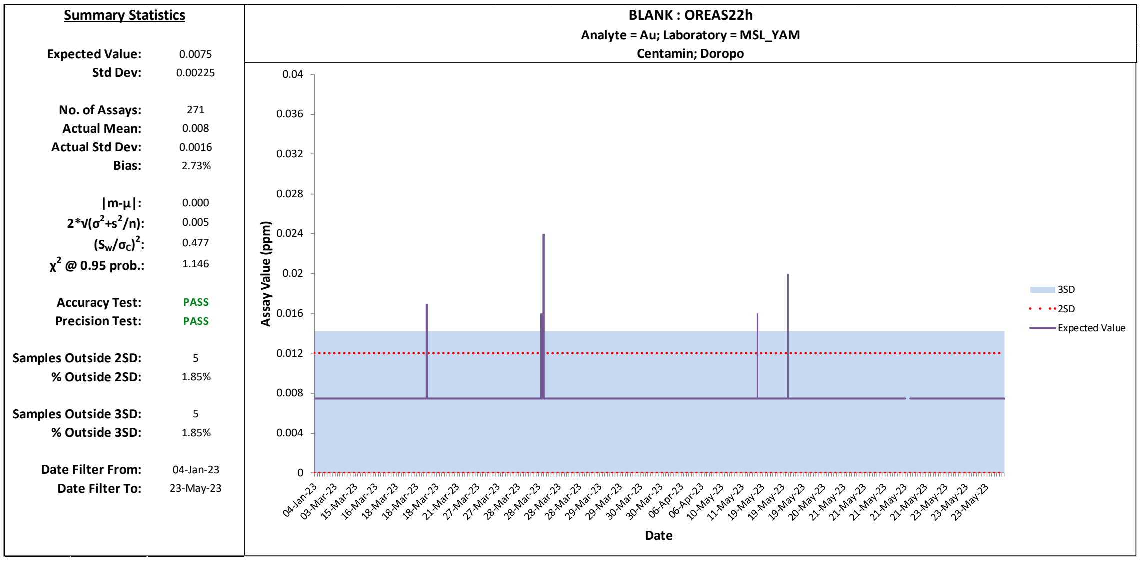


Figure 11.25 OREAS 22h Blank - MSA_YAM Results - Au ppm

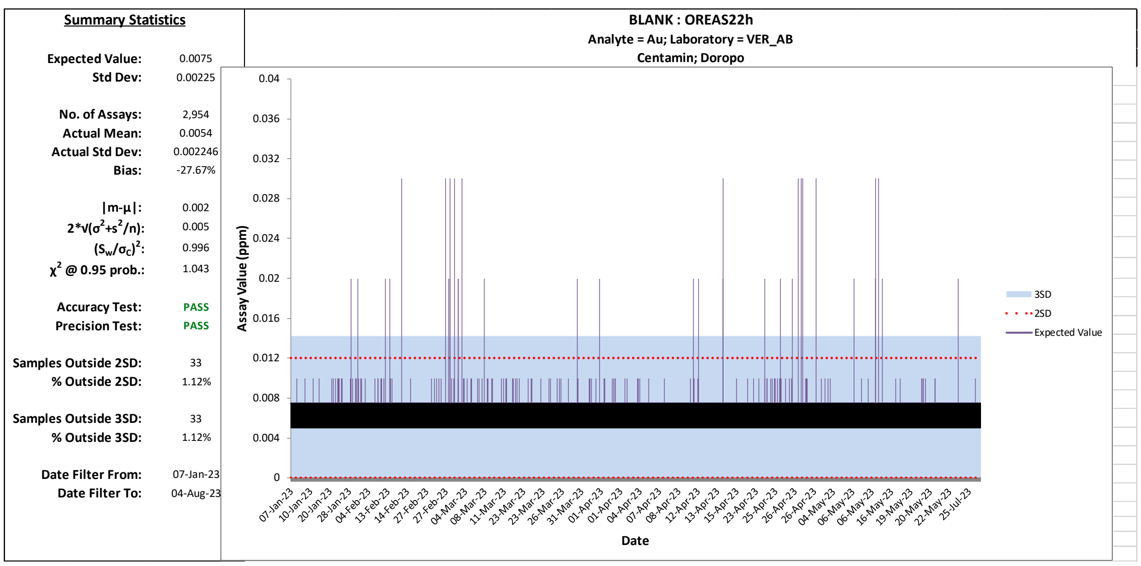


Figure 11.26 OREAS 22h Blank - VER_AB Results - Au ppm

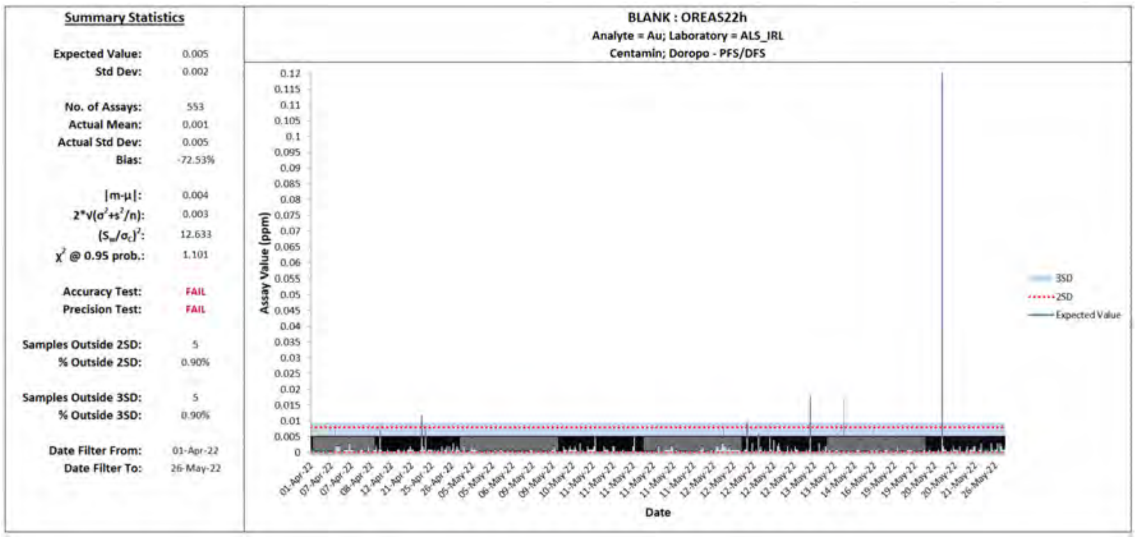


Figure 11.27 OREAS 22h Blank - ALS_IRL Results - Au ppm

Blank performance for the coarse blank (CBLKY) is shown in Figure 11.28 through to Figure 11.30.

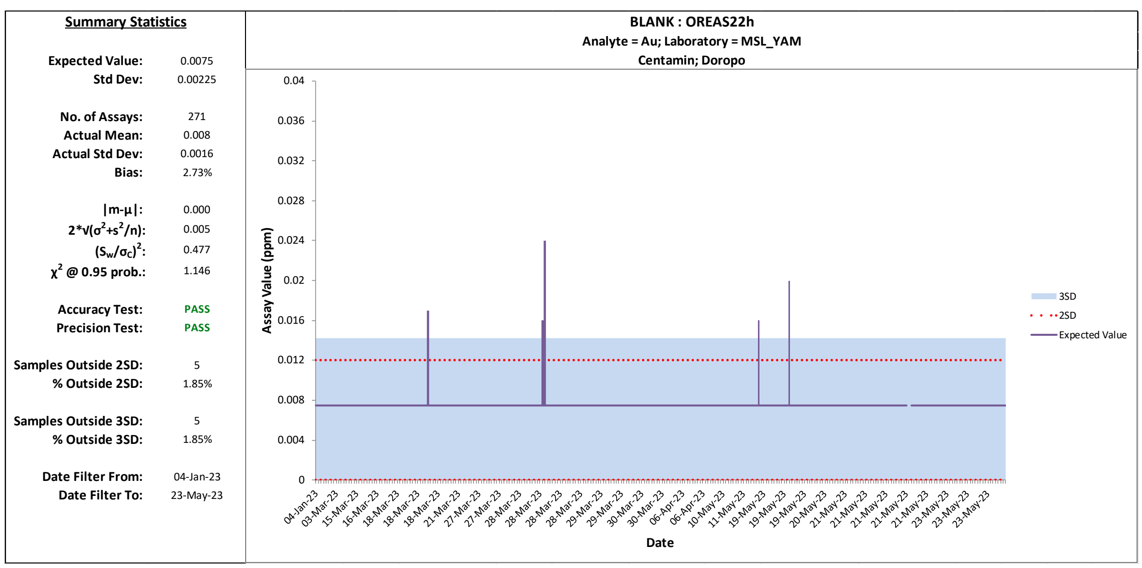


Figure 11.28 CBLKY Coarse Blank - MSA_YAM Results - Au ppm

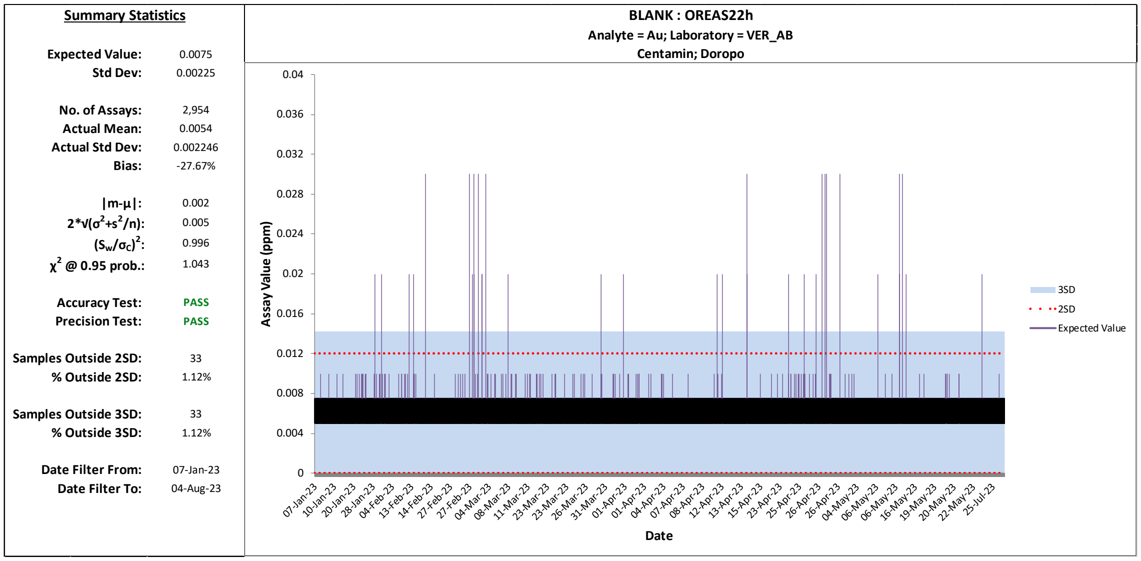


Figure 11.29 CBLKY Coarse Blank - VER_AB Results - Au ppm

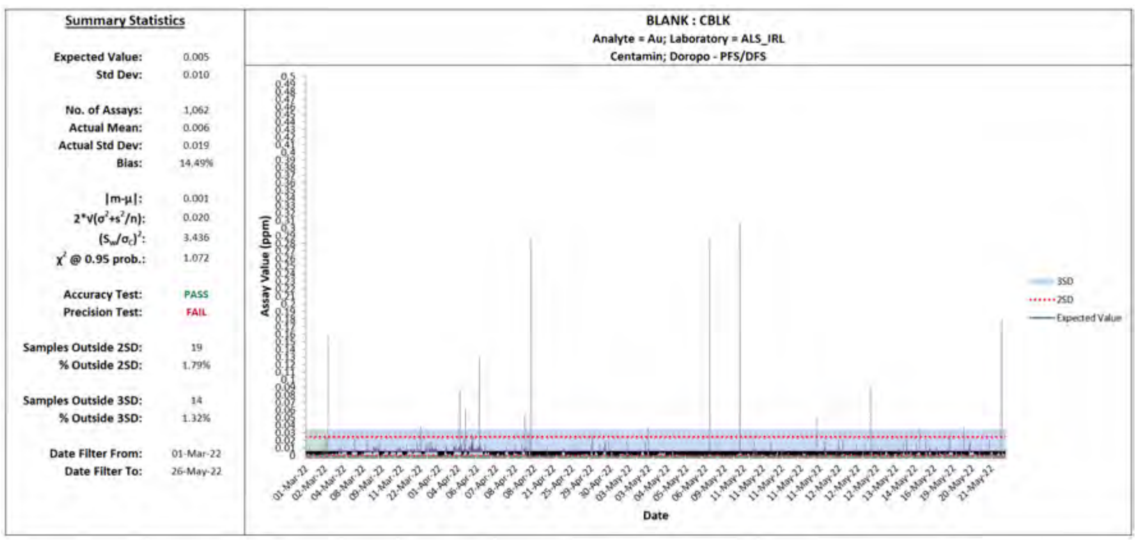


Figure 11.30 CBLKY Coarse Blank - ALS_IRL Results - Au ppm

Blank Performance Comments

The following points summarise the relevant observations for the overall coarse blank performance for the Doropo Gold Project during the PFS-DFS period:

- The CRM blank performance is acceptable, with results typically reporting below the detection limits;
- The coarse blank performance is average to acceptable. The source of the coarse blanks and the potential for them to contain background values of Au may be affecting the performance;

- There are potential cases of contamination, however levels are very low;
- It was noted there was several potential cases of misallocation. These should be further investigated, and their results corrected in the database.

11.7.3 Duplicates

Duplicates (or sample pairs) represent the most common type of matching pairs of data and are an important measure of precision error. Field duplicate samples for the Doropo Gold Project are collected at the first stage of sub-sampling for RC chips and DD core.

Data analysed in this report includes all Project data to date. This analysis is then further split by deposit.

The RC field duplicates were collected after the primary RC sample of 1/8th is taken from the first split using a 3-tier riffle splitter. The remaining 7/8th of bulk sample is run through the 3-tier riffle splitter once again to obtain a second sample for the field duplicate.

For diamond core, two types of sample duplicates are collected; field (DUPL) and pulp (PDUP) duplicates. Field duplicates (DUPL) are collected during the pre-PFS to DFS period, and pulp duplicates (PDUP) were introduced during the later stage of the PFS period (April 2022). Field duplicates were sampled as quarter core up until April 2022 and then half core was taken thereafter and then sent to the laboratory for analysis. In the Qualified Person's opinion, the quarter core duplicates are not true duplicates hence analysis on their performance should be treated with caution.

In April 2022, pulp duplicates were introduced as the primary duplicate sampling method. The pulp duplicate is collected after crushing and pulverising during the laboratory sampling process.

For core duplicates the primary half-core sample is submitted to the laboratory accompanied by an empty sample bag with a duplicate sample ID ticket stapled on the bag, which indicates that a duplicate (half of the pulp) sample must be collected.

Duplicate data is assessed via various methods including:

- Relative Mean Paired Difference (RMPD) - estimated as the differences between matching pairs of data which are normalised to the means of the corresponding pairs of the data;
- Scatter plot - The original and duplicate values are plotted as XY scatter and compared to a 1:1 unbiased relationship;
- Quantile-Quantile (Q-Q) plot - Original and duplicate populations are assessed overall and ranked by value, with resulting corresponding quantile values plotted on an XY scatter against an unbiased 1:1 trend line. Deviation from the 1:1 line indicates a bias between results.

A breakdown of the overall sample duplicate data for Project to date and split on deposit is tabulated in Table 11.9.

The performance of the RC and DD duplicates, divided by deposit are shown in Table 11.10 and Table 11.11 respectively. The performance of the pulp duplicates is tabulated in Table 11.12.

Prospect	Period	No. Samples			Mean Assay Au (ppm) - Original			Mean Assay Au (ppm) - Duplicate		
		RC (Field)	DD (Field)	DD (Pulp Repeat)	RC (Field)	DD (Field)	DD (Pulp Repeat)	RC (Field)	DD (Field)	DD (Pulp Repeat)
ATI	To date	885	38	3	0.48	9.33	0.09	0.49	9.43	0.08
CHG	To date	2763	57	187	5.78	2.71	1.48	5.40	2.48	1.71
ENI	To date	1321	19	66	0.48	1.18	2.21	0.51	1.34	2.26
HAN	To date	1328	38	73	1.02	0.77	6.54	1.08	0.66	6.50
HND	To date	706	-	-	0.58	-	-	0.50	-	-
KEK	To date	1254	18	71	0.82	0.51	1.22	0.77	0.52	1.20
KLG	To date	4130	83	256	0.46	1.68	1.99	0.46	1.55	2.04
NAR	To date	120	-	-	0.46	-	-	0.46	-	-
NOK	To date	1425	139	63	1.08	0.30	4.69	1.12	0.33	5.00
SAN	To date	289	11	-	0.77	1.11	-	0.86	1.43	-
SOL	To date	76	-	-	0.87	-	-	0.90	-	-
SWA	To date	3182	116	114	0.77	0.39	2.47	0.73	0.79	2.49
THN	To date	656	-	-	0.28	-	-	0.27	-	-
VAKO	To date	929	40	-	0.31	0.84	-	0.31	0.94	-

Table 11.9 Doropo Gold Project - To Date Period - Duplicates Breakdown by Prospect

Statistic	Prospect													
	ATI		CHG		ENI		HAN		HND		KEK		KLG	
	Original	Duplicat	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
No. Pairs	885		2,763		1,321		1,328		706		1,254		4,130	
RMPD														
% Assays within 10%	66.7%		57.0%		50.3%		58.9%		59.9%		52.9%		59.7%	
% Assays within 20%	73.6%		70.1%		59.7%		70.9%		68.8%		67.8%		72.3%	
% Assays within 50%	83.5%		84.2%		76.4%		84.9%		80.7%		83.3%		85.5%	
Average RMPD%	-2.0%		-1.4%		-1.2%		-1.9%		-0.4%		-3.3%		-1.3%	
Average CV%	30.3%		30.4%		38.8%		29.6%		30.2%		30.6%		27.9%	
Scatter Plot														
Ori. Mean	0.482	0.490	5.781	5.405	0.476	0.506	1.018	1.082	0.578	0.502	0.819	0.768	0.464	0.459
Average Diff. %	1.6%		-6.5%		6.4%		6.3%		-13.2%		-6.2%		-1.1%	
Coeff. Correlation	0.96		1.00		0.97		0.98		0.95		1.00		0.97	
Q-Q														
10^{th} %	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
25^{th} %	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.010	0.020	0.005	0.005
50^{th} %	0.005	0.005	0.032	0.030	0.020	0.020	0.030	0.023	0.010	0.010	0.100	0.090	0.050	0.050
75^{th} %	0.070	0.070	0.310	0.320	0.212	0.210	0.391	0.390	0.130	0.130	0.370	0.360	0.470	0.468
90^{th} %	0.662	0.620	1.050	1.068	0.841	0.790	1.175	1.293	0.745	0.650	0.780	0.780	1.341	1.331

Statistic	Prospect													
	NAR		NOK		SAN		SOL		SWA		THN		VAKO	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
No. Pairs	120		1,425		289		76		3,128		656		929	
RMPD														
% Assays within 10%	60.8%		59.9%		54.0%		51.3%		57.8%		66.2%		49.5%	
% Assays within 20%	72.5%		69.8%		66.4%		57.9%		69.1%		74.1%		62.9%	
% Assays within 50%	85.8%		82.9%		80.3%		75.0%		83.1%		84.1%		48.6%	
Average RMPD%	-1.5%		-1.3%		0.3%		12.3%		-2.7%		-1.7%		-2.3%	
Average CV%	25.1%		31.8%		32.2%		35.0%		31.2%		30.8%		33.6%	
Scatter Plot														
Ori. Mean	0.458	0.459	1.078	1.120	0.774	0.864	0.870	0.900	0.774	0.733	0.279	0.270	0.309	0.312
Average Diff. %	0.1%		3.9%		11.7%		3.4%		-5.3%		-3.3%		1.1%	
Coeff. Correlation	0.99		0.99		0.97		0.99		0.99		1.00		0.90	
Q-Q														
10^{th} %	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
25^{th} %	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
50^{th} %	0.025	0.020	0.020	0.020	0.030	0.030	0.010	0.015	0.030	0.030	0.010	0.010	0.050	0.050
75^{th} %	0.448	0.403	0.275	0.280	0.250	0.240	0.120	0.155	0.250	0.250	0.160	0.150	0.330	0.330
90^{th} %	1.112	1.040	1.202	1.198	0.650	0.754	0.715	0.770	0.999	0.970	0.590	0.620	0.862	0.840

Table 11.10 Reverse Circulation Field Original vs. Duplicate Statistics - To Date

Statistic	Prospect													
	ATI		CHG		ENI		HAN		HND		KEK		KLG	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
No. Pairs	38		57		19		38		-		18		83	
RMPD														
% Assays within 10%	31.6%		31.6%		15.8%		47.4%		-		38.9%		28.9%	
% Assays within 20%	57.9%		50.9%		26.3%		60.5%		-		55.6%		39.8%	
% Assays within 50%	68.4%		77.2%		63.2%		78.9%		-		94.4%		80.7%	
Average RMPD%	7.6%		-6.3%		-13.0%		-18.5%		-		0.2%		-16.3%	
Average CV%	44.8%		35.3%		46.1%		45.2%		-		19.9%		40.7%	
Scatter Plot														
Ori. Mean	9.334	9.433	2.711	2.476	1.177	1.336	0.773	0.659	-	-	0.514	0.515	1.678	1.552
Average Diff. %	1.1%		-8.6%		13.5%		-14.9%		-		0.1%		-7.6%	
Coeff. Correlation	1.00		0.97		0.92		0.94		-		0.98		0.90	
Q-Q														
10^{th} %	0.0170	0.0085	0.016	0.020	0.2242	0.1484	0.005	0.005	-	-	0.005	0.005	0.030	0.040
25^{th} %	0.053	0.095	0.090	0.106	0.301	0.223	0.012	0.020	-	-	0.025	0.052	0.720	0.546
50^{th} %	0.455	0.445	0.470	0.394	0.600	0.320	0.375	0.164	-	-	0.310	0.255	1.250	0.940
75^{th} %	2.868	3.080	1.710	1.380	1.295	1.835	1.048	0.748	-	-	0.663	0.725	2.105	1.810
90^{th} %	19.155	21.966	7.921	10.388	2.164	3.104	1.649	1.624	-	-	1.435	1.235	3.500	3.252

Statistic	Prospect													
	NAR		NOK		SAN		SOL		SWA		THN		VAKO	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
No. Pairs	-		139		11		-		116		-		40	
RMPD														
% Assays within 10%	-		72.7%		9.1%		-		44.8%		-		32.5%	
% Assays within 20%	-		79.1%		63.6%		-		54.3%		-		50.0%	
% Assays within 50%	-		82.7%		100.0%		-		69.8%		-		80.0%	
Average RMPD%	-		3.4%		7.6%		-		-1.8%		-		5.5%	
Average CV%	-		32.8%		16.0%		-		39.8%		-		22.9%	
Scatter Plot														
Ori. Mean	-	-	0.298	0.327	1.114	1.428	-	-	0.389	0.794	-	-	0.844	0.940
Average Diff. %	-		9.4%		28.2%		-		104.3%		-		11.4%	
Coeff. Correlation	-		0.90		1.00		-		0.78		-		0.86	
Q-Q														
10^{th} %	-	-	0.005	0.005	0.110	0.130	-	-	0.005	0.005	-	-	0.194	0.186
25^{th} %	-	-	0.005	0.005	0.175	0.180	-	-	0.005	0.005	-	-	0.325	0.390
50^{th} %	-	-	0.005	0.005	0.390	0.400	-	-	0.030	0.020	-	-	0.785	0.670
75^{th} %	-	-	0.030	0.030	1.485	1.965	-	-	0.390	0.470	-	-	1.143	1.358
90^{th} %	-	-	0.564	0.606	2.520	2.800	-	-	1.035	4.015	-	-	1.731	2.053

Table 11.11 Diamond Drilling Field Original vs. Duplicate Statistics - To Date

Statistic	Prospect															
	ATI[1]		CHG		ENI		HAN		KEK		KLG		NOK		SWA	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
No. Pairs	-		66		30		28		38		84		45		95	
RMPD																
% Assays within 10%	-		43.9%		53.3%		17.9%		39.5%		47.6%		44.4%		41.1%	
% Assays within 20%	-		65.2%		76.7%		50.5%		81.6%		70.2%		71.1%		64.2%	
% Assays within 50%	-		92.4%		90.0%		100.0%		97.4%		88.1%		88.9%		87.4%	
Average RMPD%	-		-2.1%		-2.7%		-5.5%		3.7%		-3.4%		4.4%		-7.1%	
Average CV%	-		24.4%		18.2%		17.8%		13.9%		35.6%		21.4%		23.4%	
Scatter Plot																
Ori. Mean	-	-	2.099	2.180	1.127	1.101	11.082	10.868	1.526	1.477	2.070	1.863	4.304	4.624	1.748	7.747
Average Diff. %	-		3.9%		-2.3%		-1.9%		-3.2%		-10.0%		7.4%		-0.1%	
Coeff. Correlation	-		0.99		0.96		0.93		0.99		0.89		0.99		0.94	
Q-Q																
10th %	-	-	0.136	0.131	0.1027	0.1626	0.2661	0.3277	0.2818	0.2992	0.2554	0.2320	0.1188	0.1300	0.260	0.284
25th %	-	-	0.310	0.350	0.342	0.348	0.415	0.437	0.380	0.593	0.600	0.758	0.387	0.424	0.520	0.546
50th %	-	-	0.676	0.654	0.820	0.705	1.027	0.999	0.932	0.844	1.116	1.073	0.681	0.720	1.000	1.132
75th %	-	-	1.534	1.454	1.491	1.514	6.642	7.331	1.650	1.747	2.213	2.094	2.296	2.180	1.910	1.965
90th %	-	-	3.306	3.070	2.275	2.379	16.652	15.627	3.845	3.329	4.303	4.062	4.297	6.648	3.389	3.291

[1] *Only 3 Pulp rejects duplicates were found for ATI which is insufficient to compile the statistics*

Table 11.12 Diamond Drilling Pulp Original vs. Duplicate Statistics - To Date

Results - Prospect ATTIRE

For RC field duplicates, 885 sample pairs have been collected Project to date and all were analysed at VER_AB. There is a mean difference of 1.6% between original and duplicate samples (Figure 11.31). There is no bias within any grade ranges (Figure 11.32) and the average CV of 30.3% (Figure 11.33) which is within the limits of acceptability (i.e. < 40% for nuggety gold deposits).

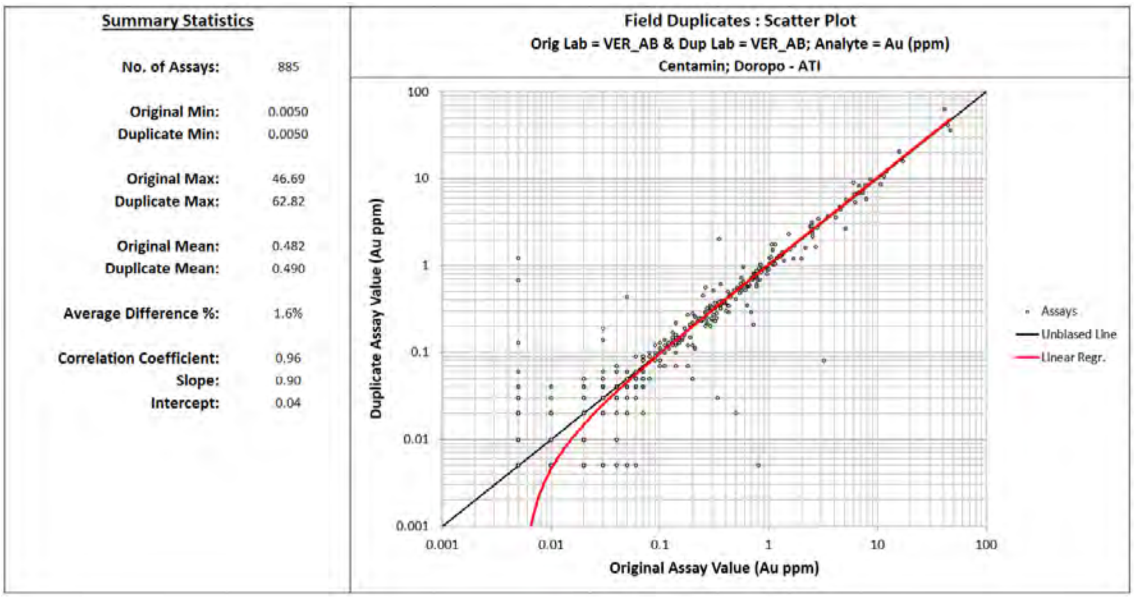


Figure 11.31 Prospect ATI, RC Field Duplicate Scatter Plot - To Date - Au ppm

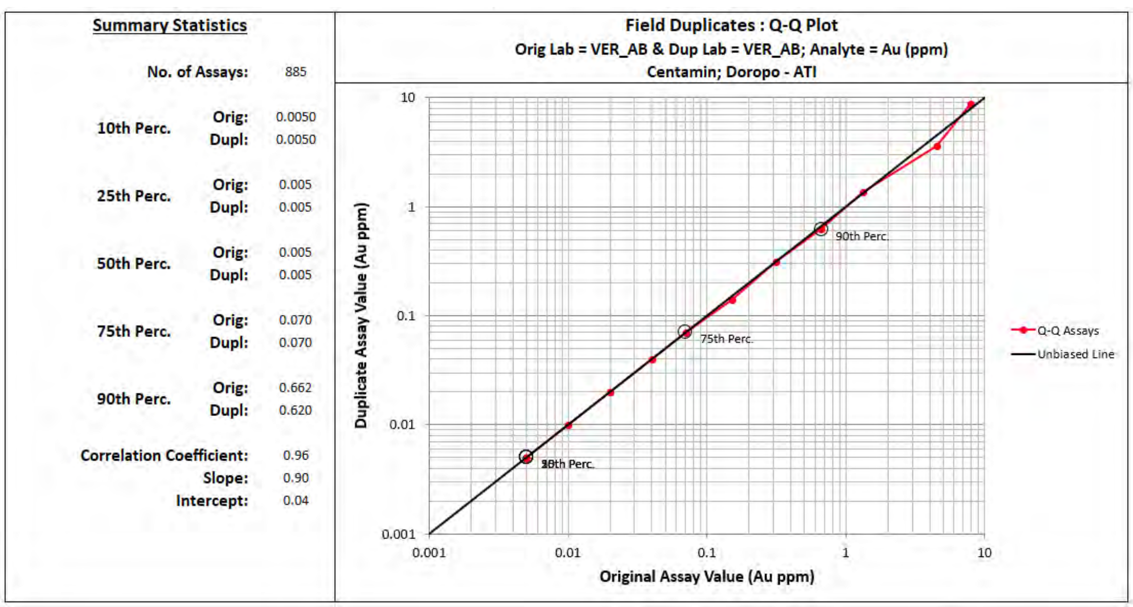


Figure 11.32 Prospect ATI, RC Field Duplicate Q-Q Plot - To Date - Au ppm

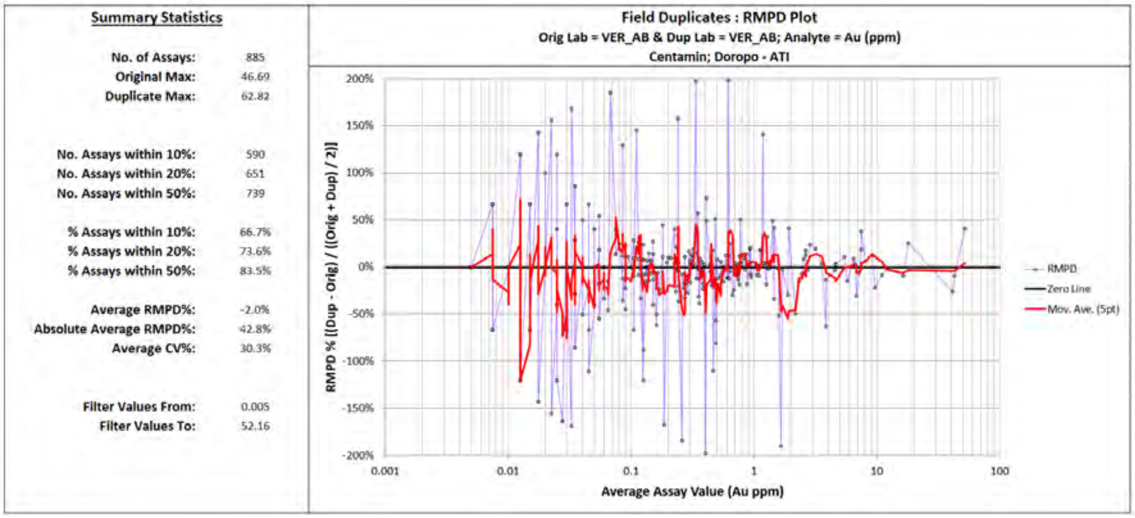


Figure 11.33 Prospect ATI, RC Field Duplicate RMDP Plot - To Date - Au ppm

DD field duplicates performance is acceptable with a mean difference of 1.1% (Figure 11.34). The Q-Q plot (Figure 11.35) does show some bias in the 0.1 - 1.5 g/t Au range, but at higher grade there is a good correlation. The precision is poor with an average CV of 44.8% (Figure 11.36). It is considered acceptable to have a CV lower than 40% for field duplicates within nuggety gold deposits.

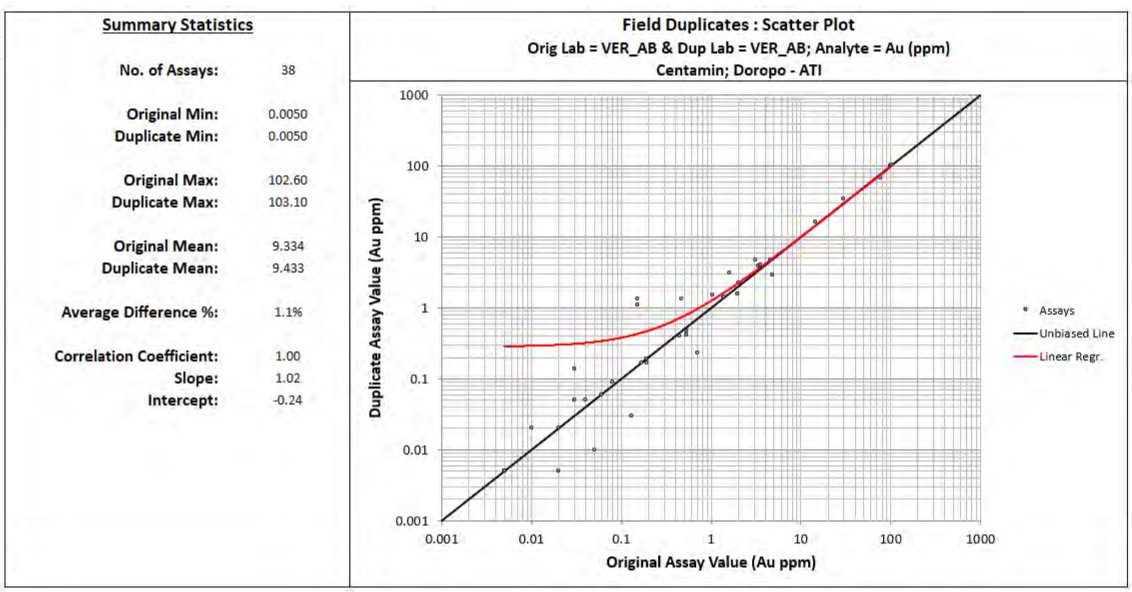


Figure 11.34 Prospect ATI, DD Field Duplicate Scatter Plot - To Date - Au ppm

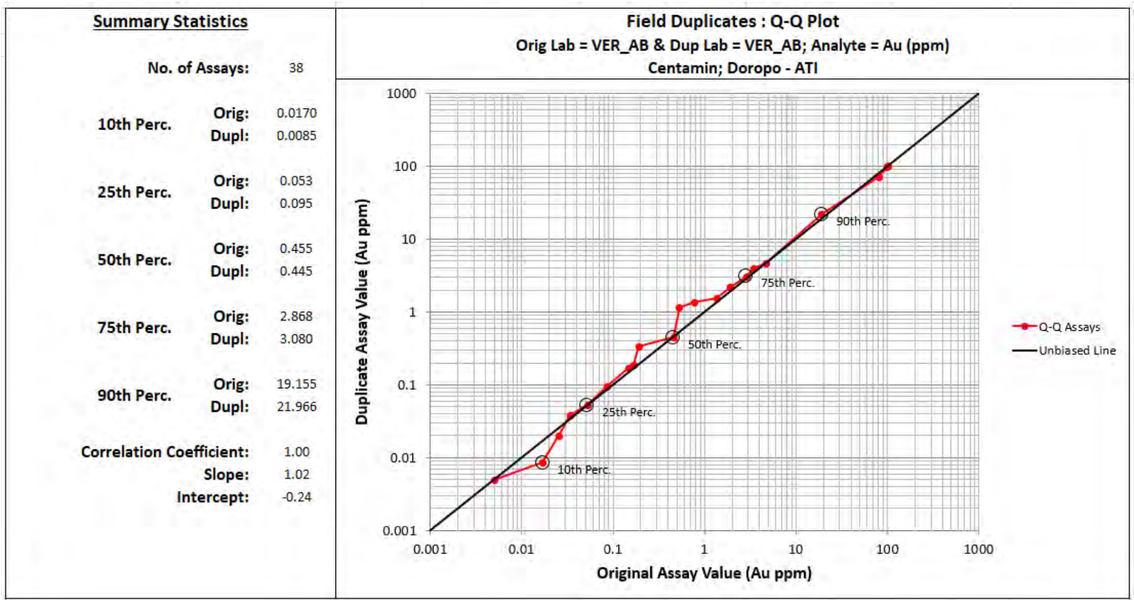


Figure 11.35 Prospect ATI, DD Field Duplicate Q-Q Plot - To Date - Au ppm

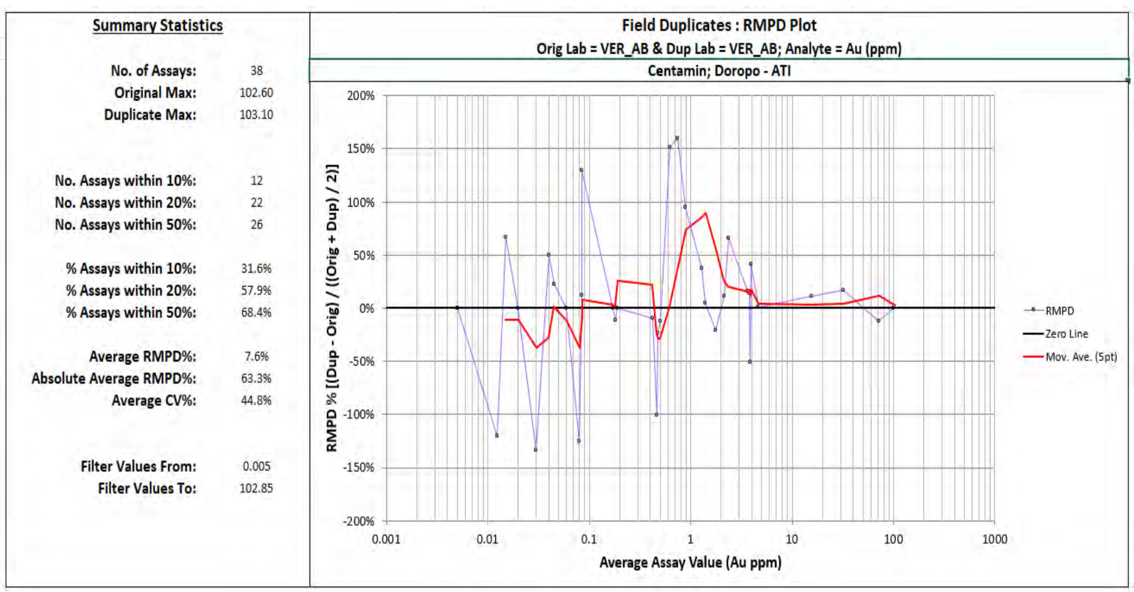


Figure 11.36 Prospect ATI, DD Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect CHEGUE

There were 2,763 RC field duplicate pairs collected which were sent to all three laboratories, however VER_AB saw the majority of samples. There were no perceptible biases (Figure 11.38), however the average CV of 30.4% is within what is considered acceptable.

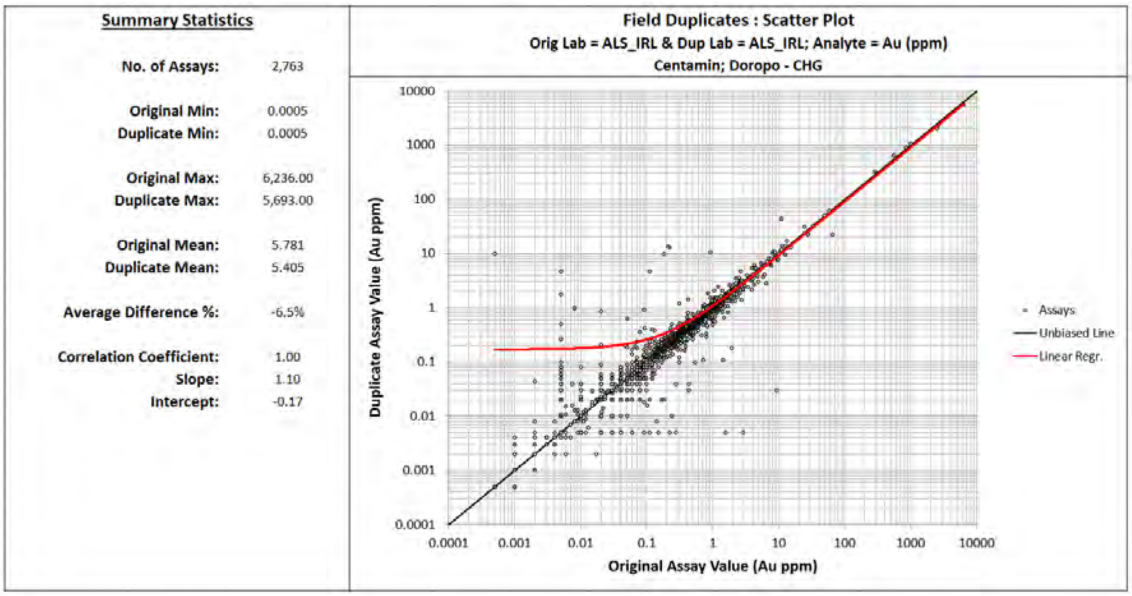


Figure 11.37 Prospect CHG, RC Field Duplicate Scatter Plot - To Date - Au ppm

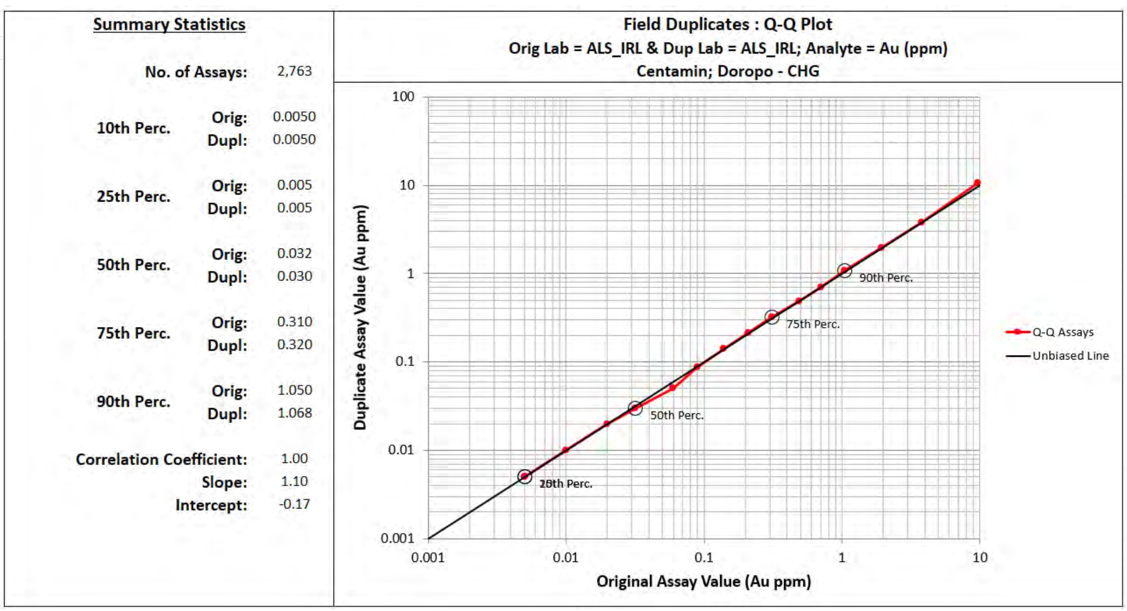


Figure 11.38 Prospect CHG, RC Field Duplicate Q-Q Plot - To Date - Au ppm

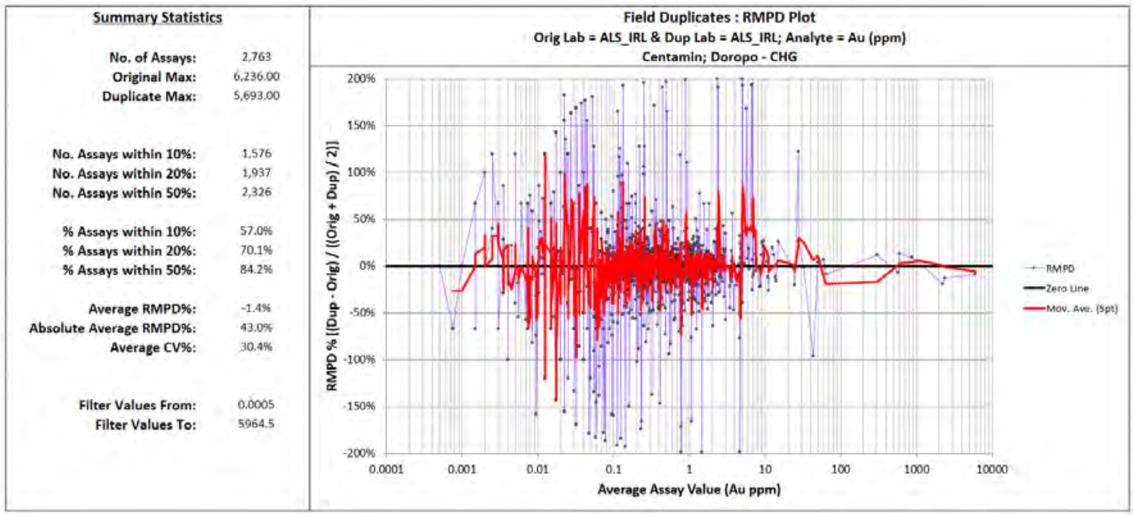


Figure 11.39 Prospect CHG, RC Field Duplicate RMDP Plot - To Date - Au ppm

There were only 57 DD field duplicates, which is at the lower limit of data points to make a definitive conclusion. The scatter plot (Figure 11.40) shows a bias of 8.6% relative to the original mean which mainly occurs at grades > 0.2 g/t Au (Figure 11.41). An average CV of around 35% is acceptable for the DD samples.

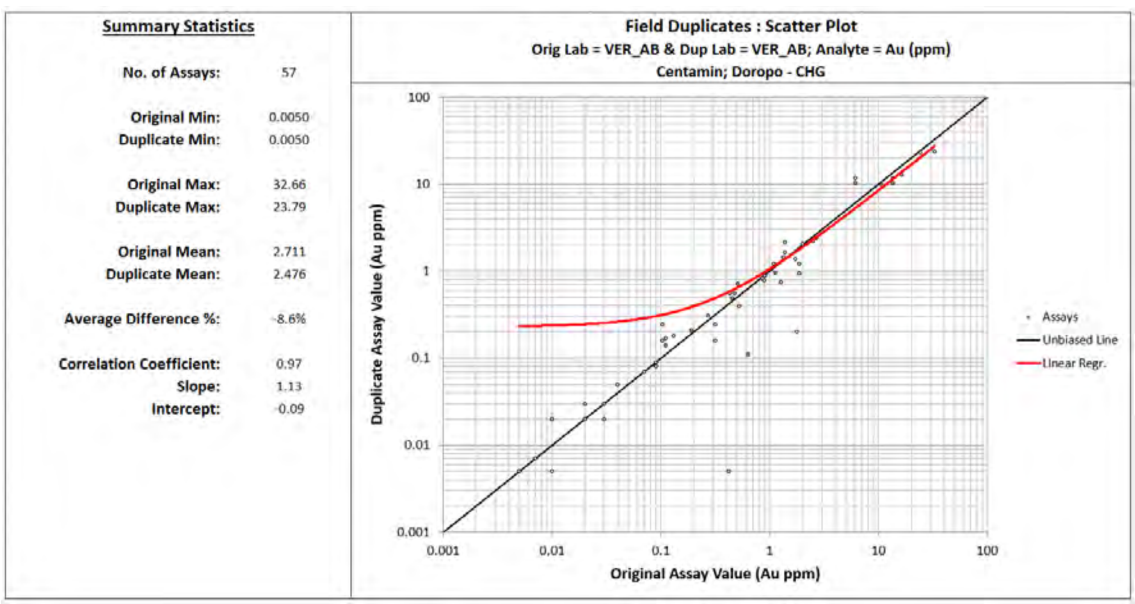


Figure 11.40 Prospect CHG, DD Field Duplicate Scatter Plot - To Date - Au ppm

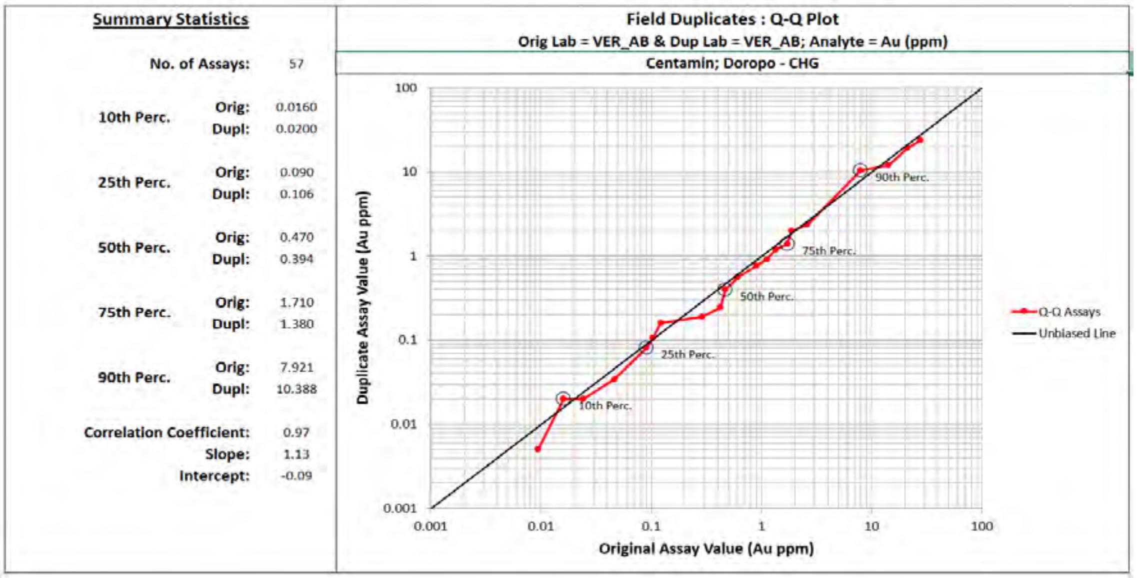


Figure 11.41 Prospect CHG, DD Field Duplicate Q-Q Plot - To Date - Au ppm

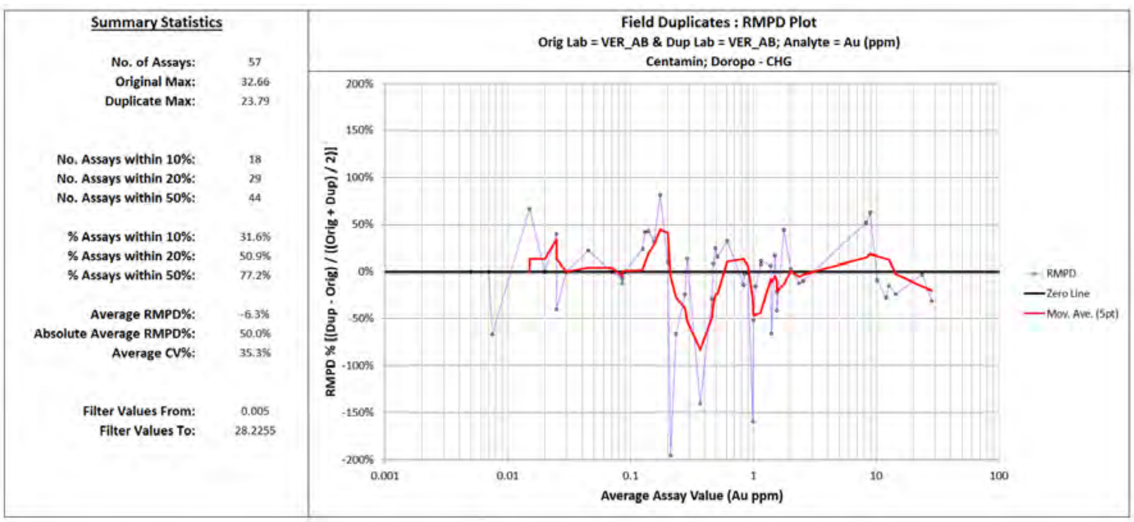


Figure 11.42 Prospect CHG, DD Field Duplicate RMDP Plot - To Date - Au ppm

The scatter plot for pulp duplicates shows a difference in means of 15.9% (Figure 11.43) which is considered high. Overall, there is a consistent high bias towards the duplicate across all grade ranges (Figure 11.44) and the average CV of 31% (Figure 11.45) is well outside what is considered acceptable (i.e. 10 - 20%).

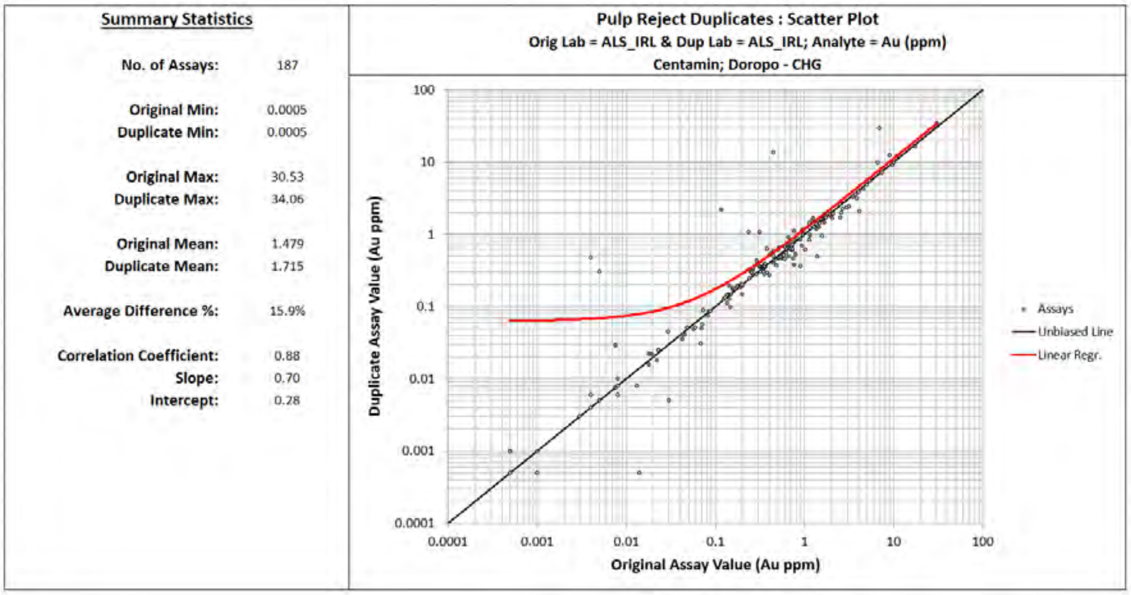


Figure 11.43 Prospect CHG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm

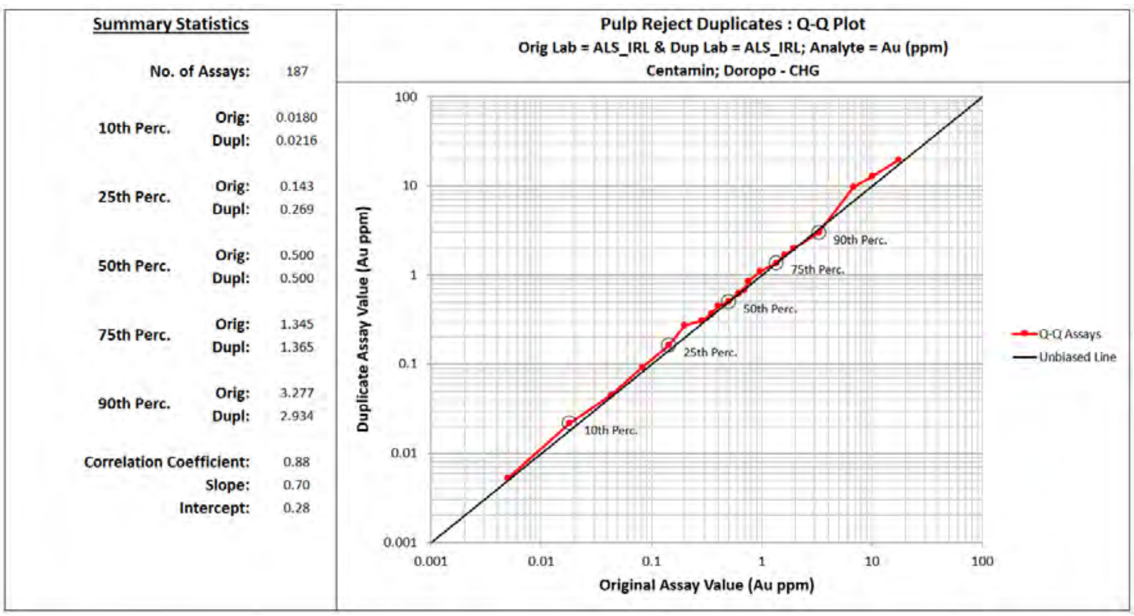


Figure 11.44 Prospect CHG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm

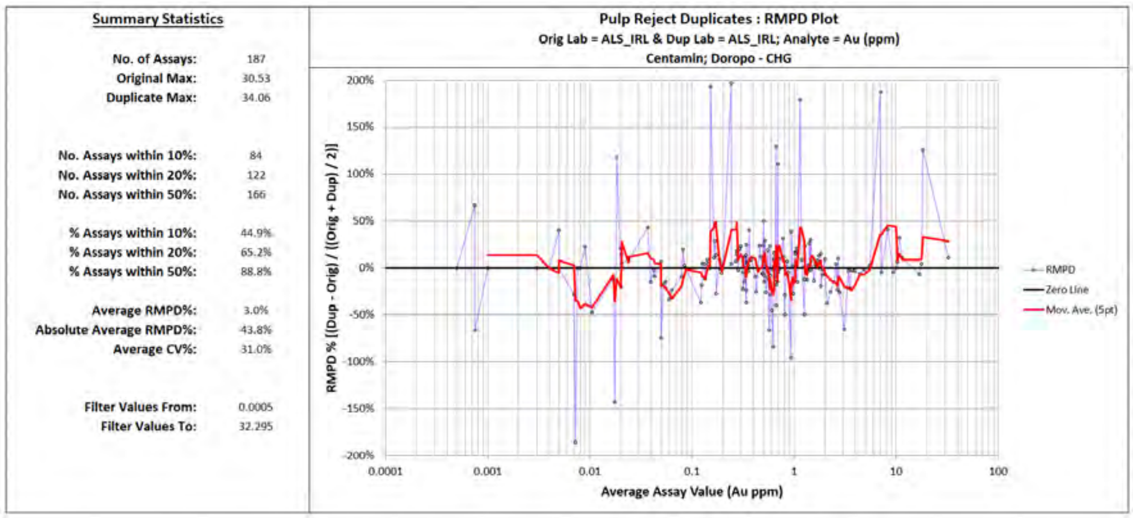


Figure 11.45 Prospect CHG, DD Pulp Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect ENIODA

There was a significant number of RC duplicate pairs from the Enioda Project. There was a slight high bias of 6.4% in the duplicates (Figure 11.46), with no perceptible significant bias within any grade ranges (Figure 11.47) and an average CV of 38.8% which is at the upper end of what is considered acceptable.

There were less than 20 DD duplicates and only 66 pulp duplicates so no analysis was undertaken.

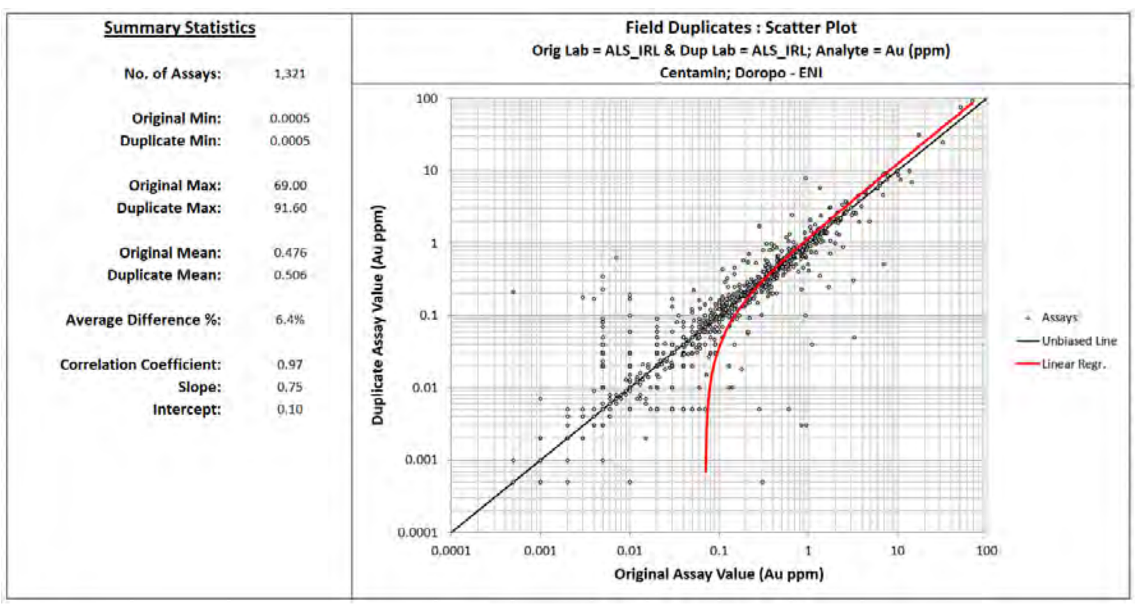


Figure 11.46 Prospect ENI, RC Field Duplicate Scatter Plot - To Date - Au ppm

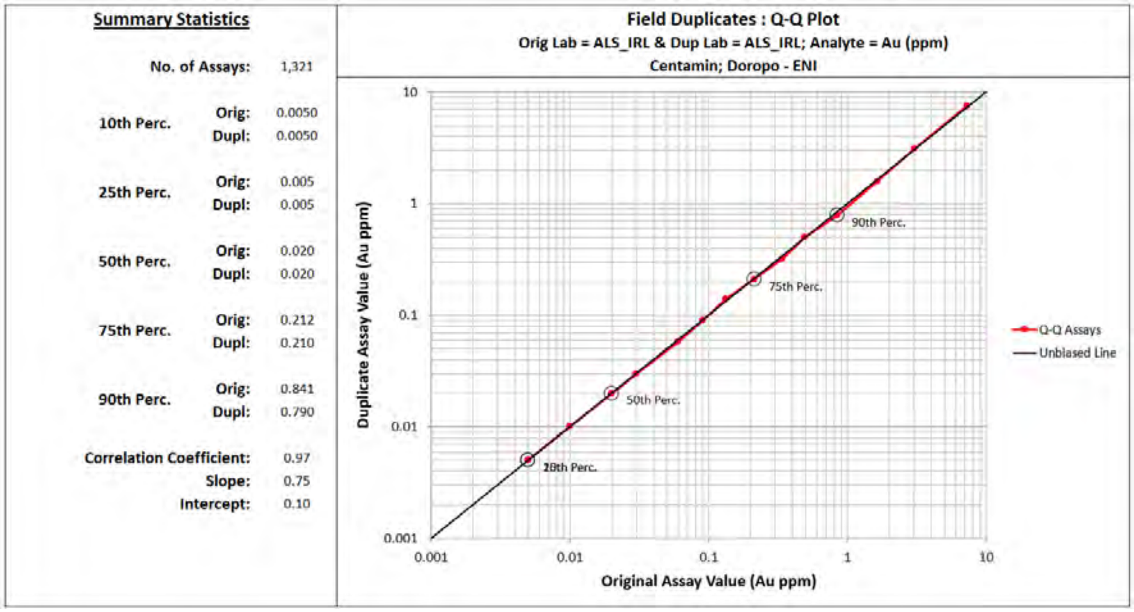


Figure 11.47 Prospect ENI, RC Field Duplicate Q-Q Plot - To Date - Au ppm

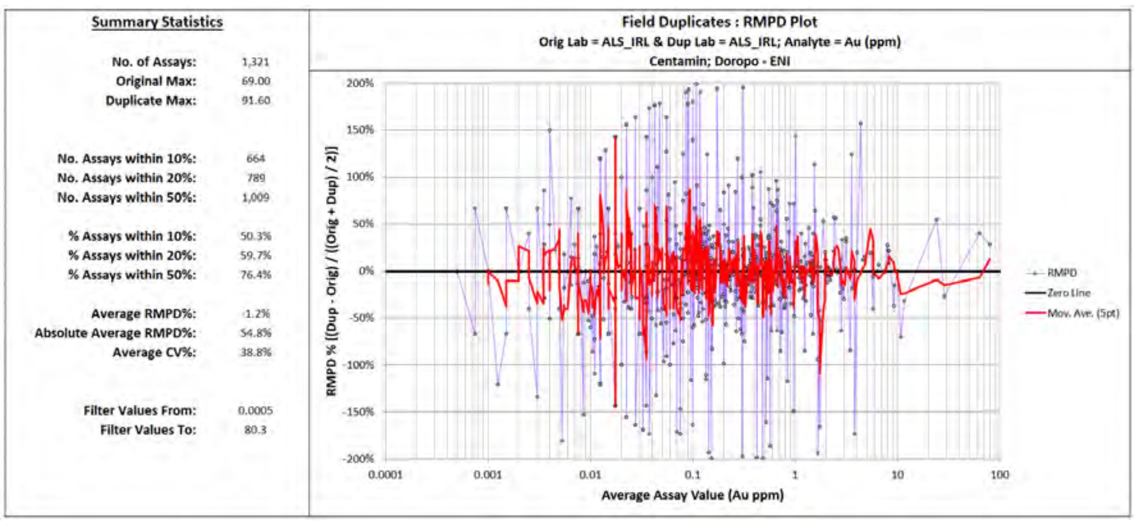


Figure 11.48 Prospect ENI, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect HAN

There were over 1,300 RC duplicates submitted to ALS_IRL and VER_AB laboratories, with the majority being submitted to VER_AB. There was a slight high bias (6.3%) of the duplicate mean (Figure 11.49), no systematic biases (Figure 11.50), with a reasonable level of precision, as 70% of samples were within 20% and average CV of around 30% (Figure 11.51).

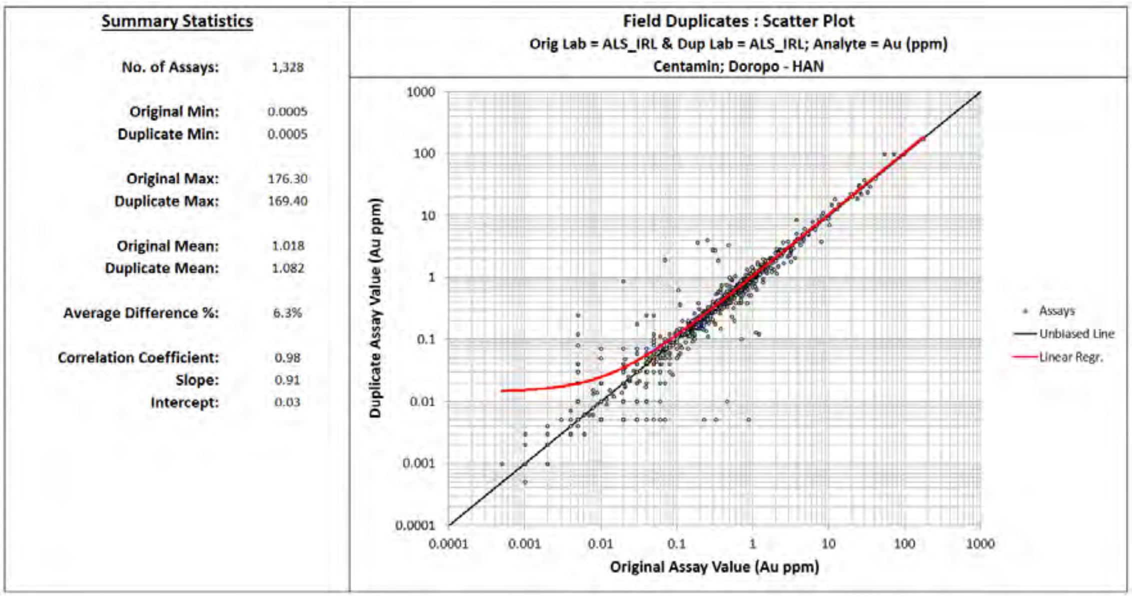


Figure 11.49 Prospect HAN, RC Field Duplicate Scatter Plot - To Date - Au ppm

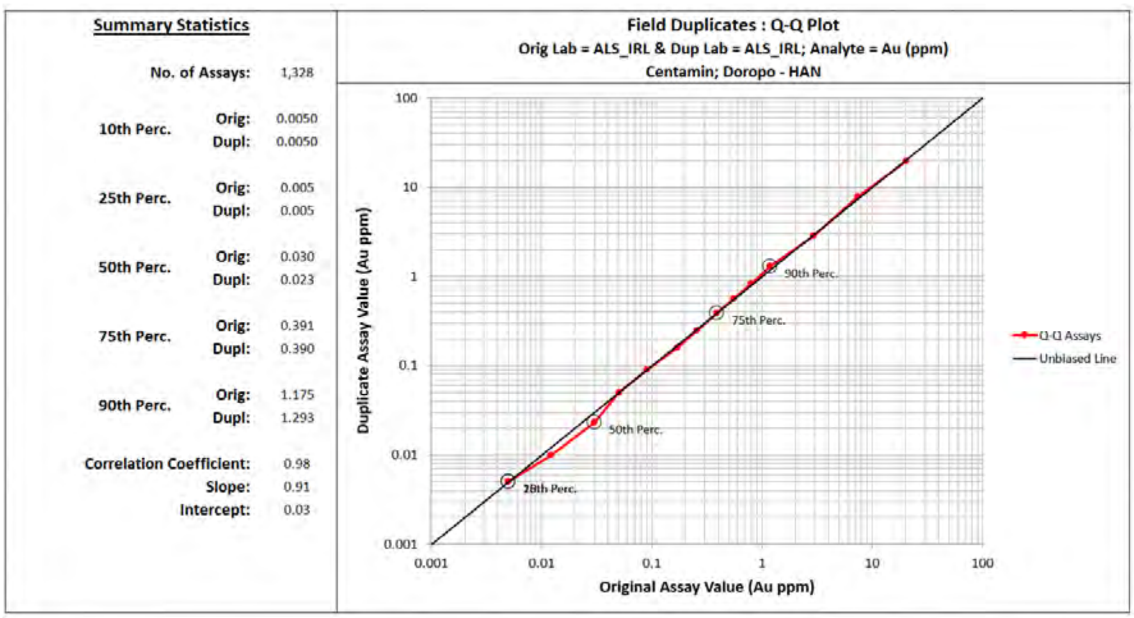


Figure 11.50 Prospect HAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm

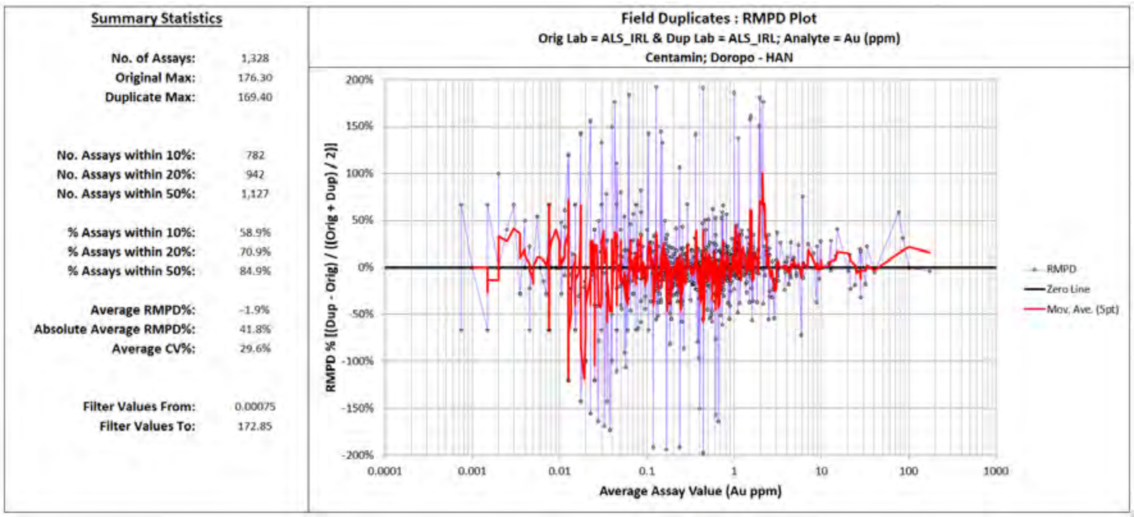


Figure 11.51 Prospect HAN, RC Field Duplicate RMDP Plot - To Date - Au ppm

There were only 38 DD duplicates, which is considered too small to make a meaningful analysis.

The pulp duplicates showed a very minor difference in means of 0.7% (Figure 11.52), no considerable biases within specific grade ranges (Figure 11.53) and an average CV of 25.8%, which is at the upper end of what is deemed acceptable.

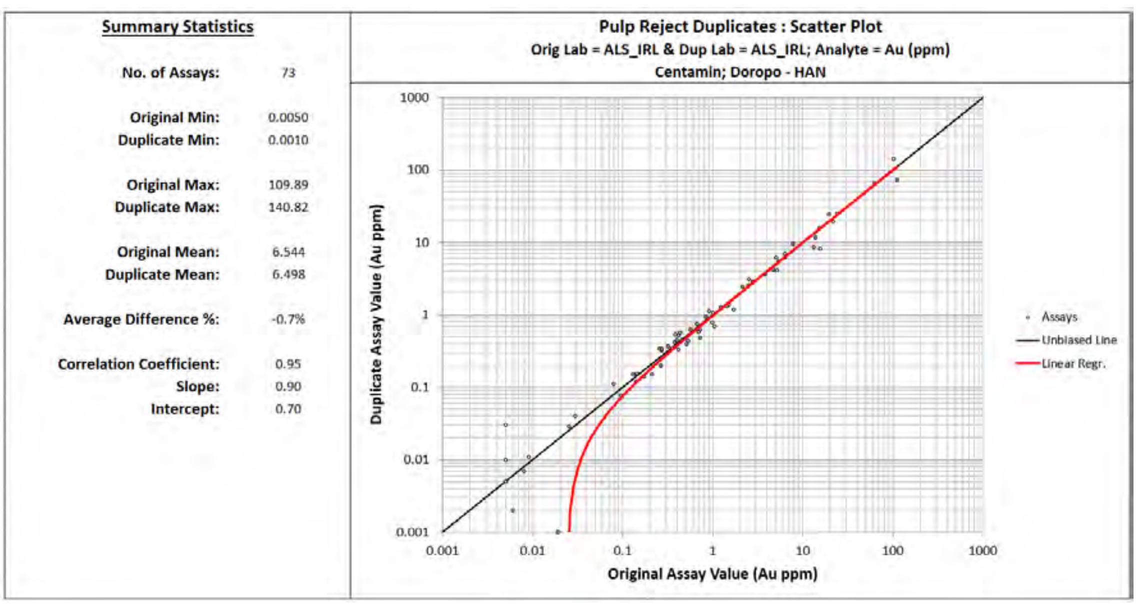


Figure 11.52 Prospect HAN, DD Pulp Duplicate Scatter Plot - To Date - Au ppm

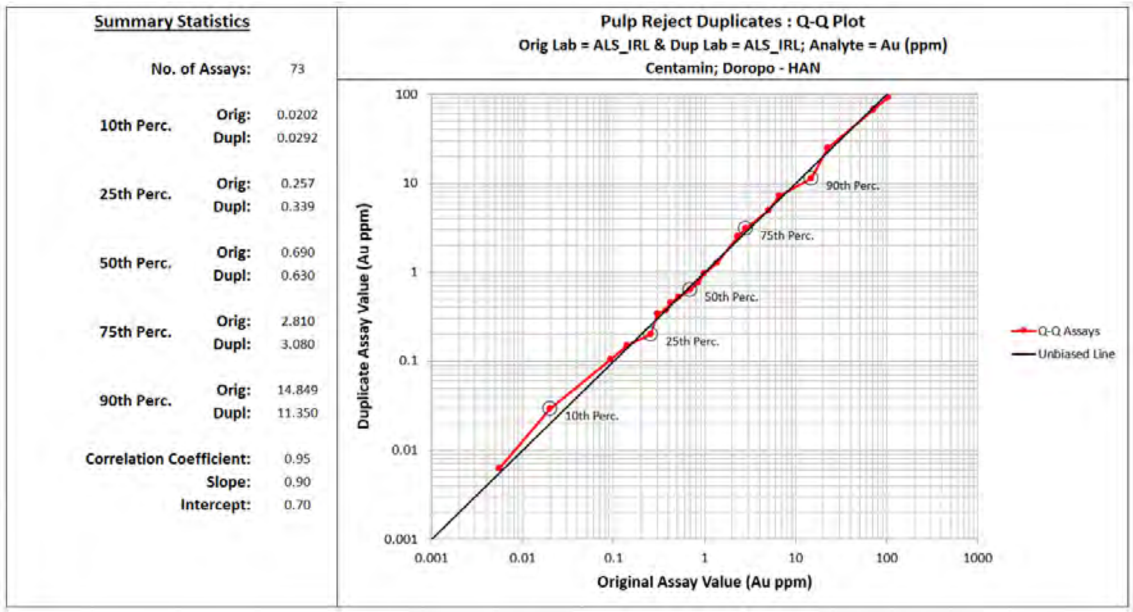


Figure 11.53 Prospect HAN, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm

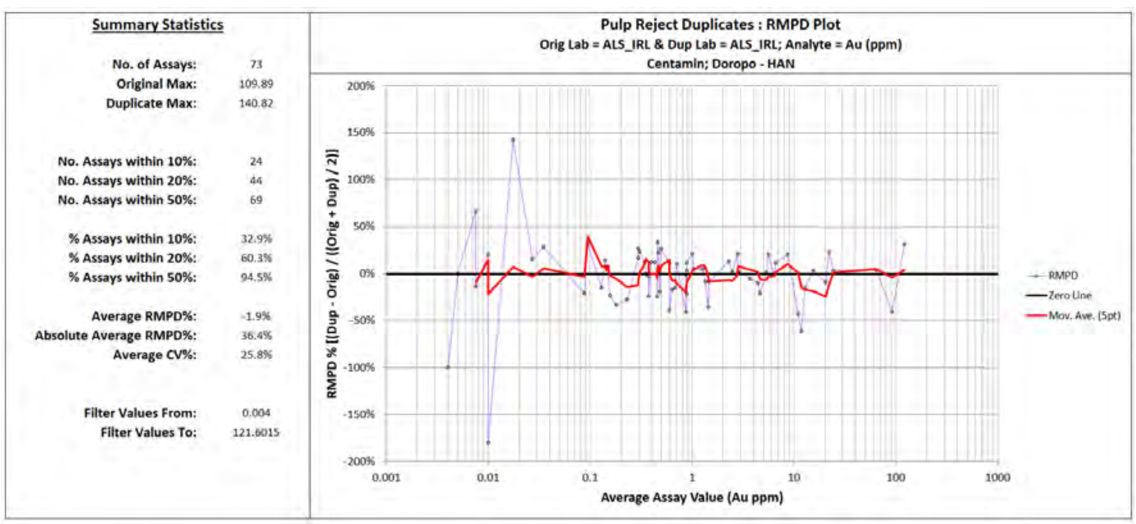


Figure 11.54 Prospect HAN, DD Pulp Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect HINDA

RC duplicates from Hinda show a reasonably large difference in means of 13.2% (Figure 11.55), with the duplicate being biased low. The high bias of the primary sample is reasonably consistent throughout all grade ranges (Figure 11.56); an average CV of 30.2% is considered to be within acceptable limits.

There are no DD or pulp duplicates at this prospect.

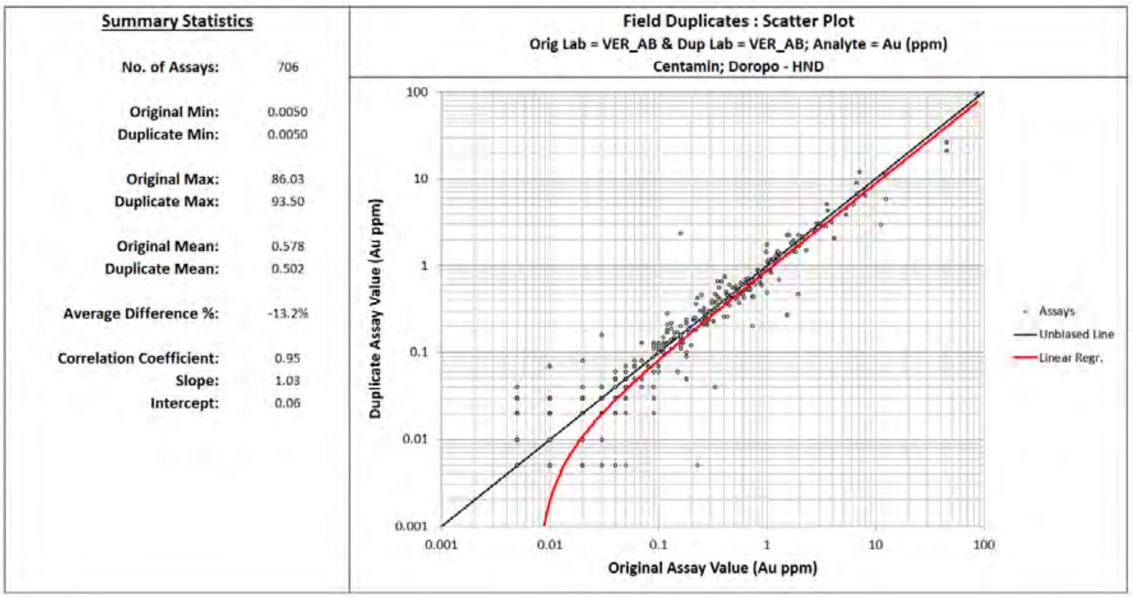


Figure 11.55 Prospect HND, RC Field Duplicate Scatter Plot - To Date - Au ppm

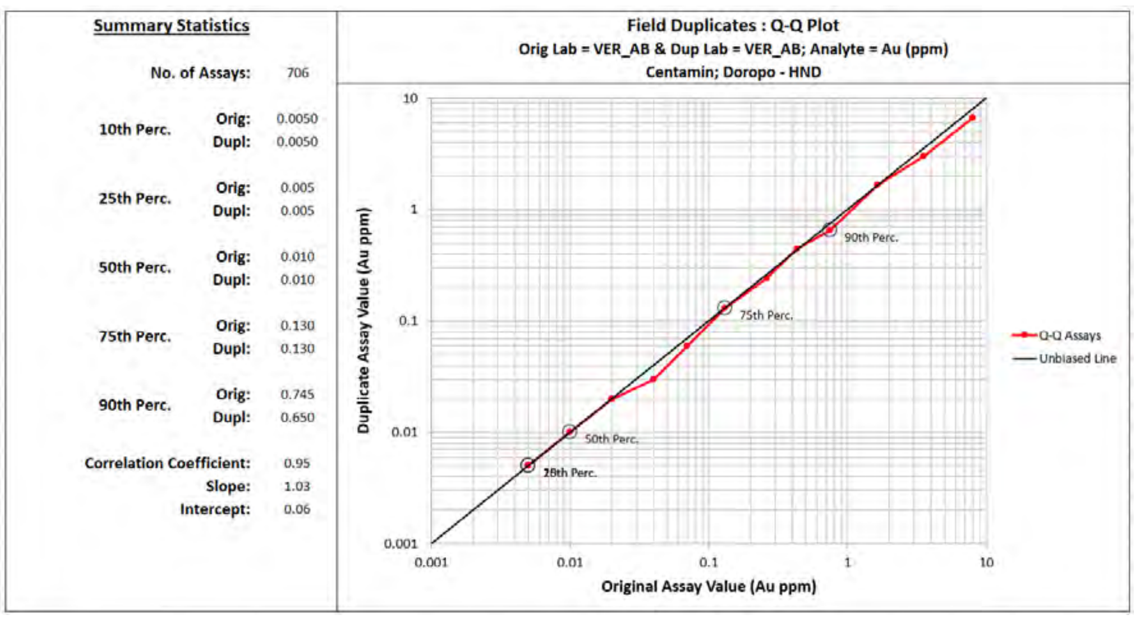


Figure 11.56 Prospect HND, RC Field Duplicate Q-Q Plot - To Date - Au ppm

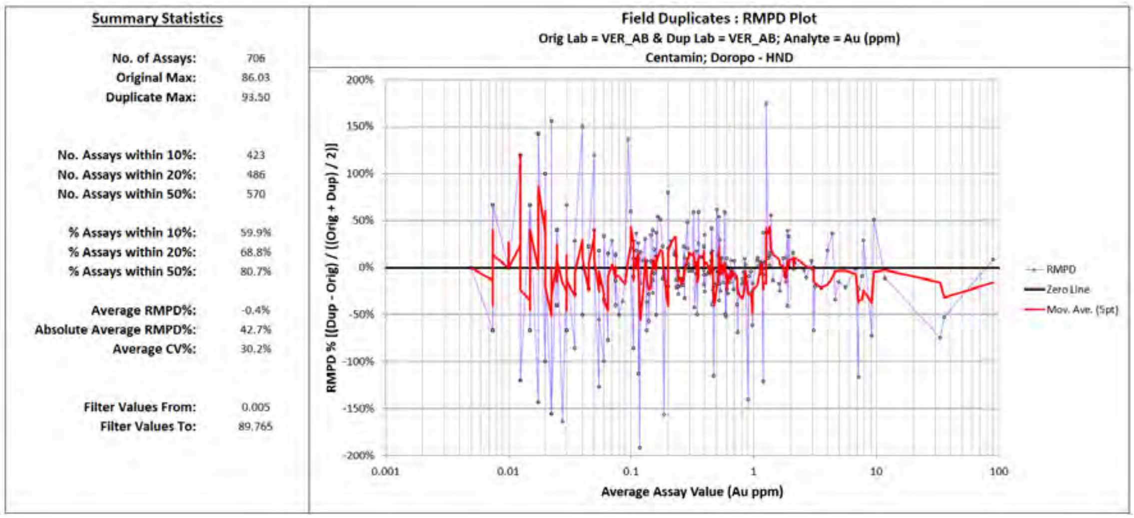


Figure 11.57 Prospect HND, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect KEKEDA

There was a minor 6% difference in means (Figure 11.58), biased high towards the original sample, but no significant grade range biases (Figure 11.59). The CV average of 30.6% (Figure 11.60) is within the acceptable limits.

There were only 18 DD duplicate samples, so no analysis was completed.

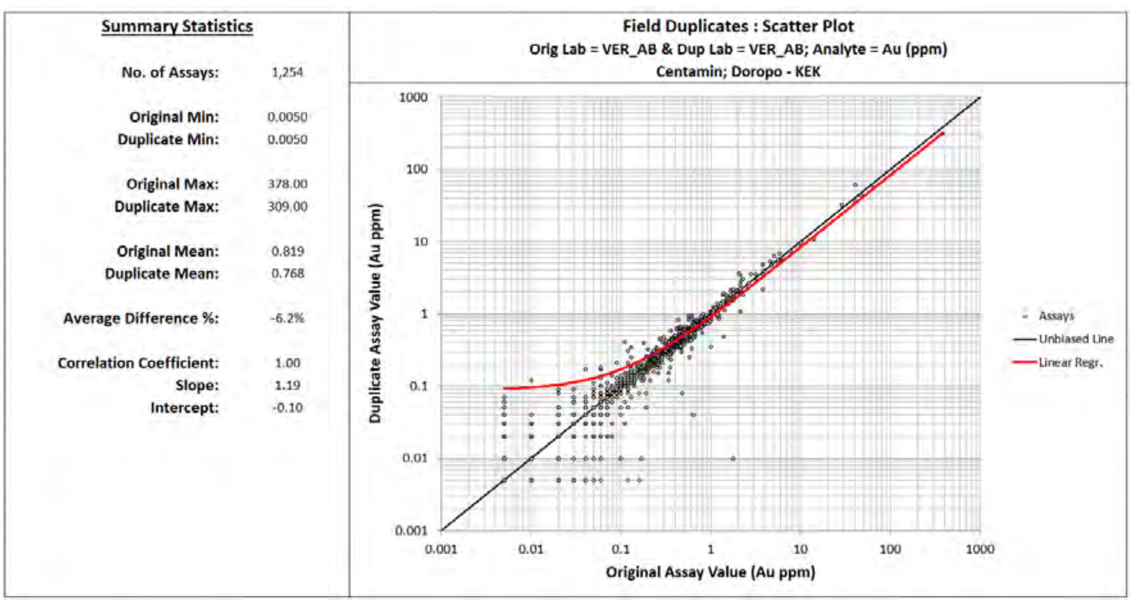


Figure 11.58 Prospect KEK, RC Field Duplicate Scatter Plot - To Date - Au ppm

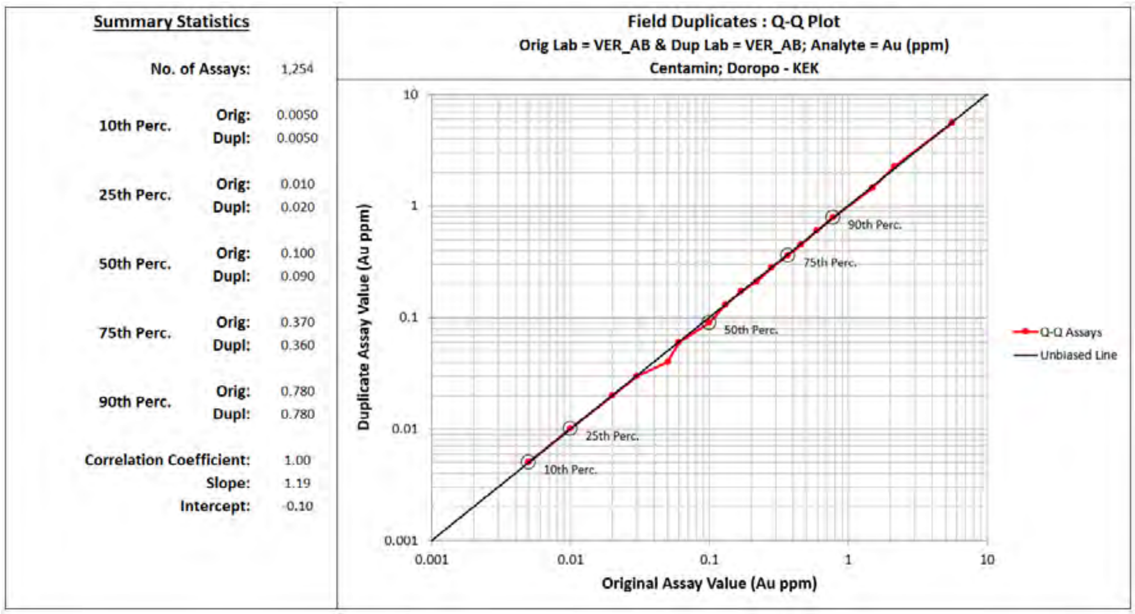


Figure 11.59 Prospect KEK, RC Field Duplicate Q-Q Plot - To Date - Au ppm

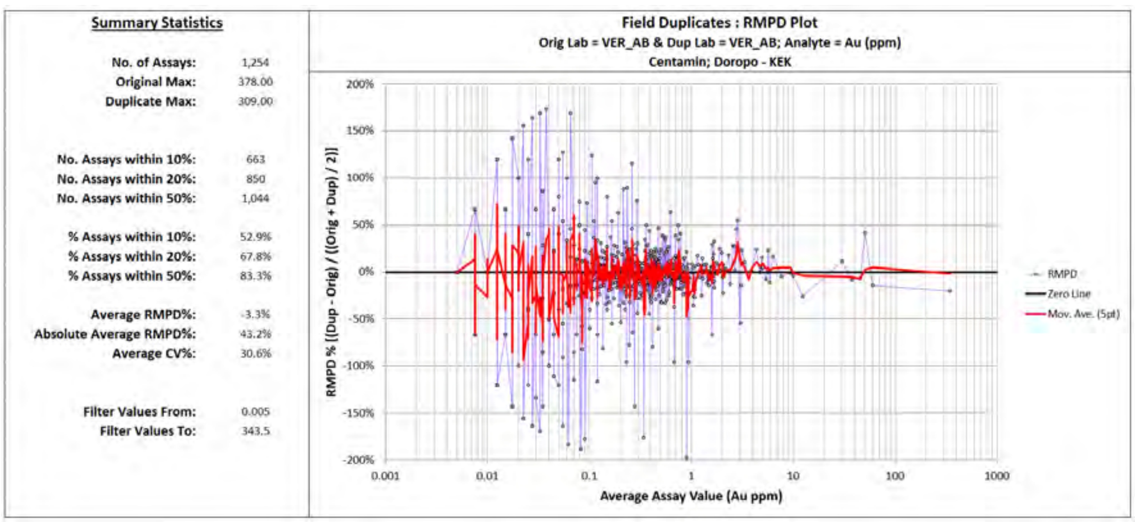


Figure 11.60 Prospect KEK, RC Field Duplicate RMDP Plot - To Date - Au ppm

There were 71 pulp duplicates analysed at all three laboratories utilised Project to date. Although the difference in average means of 1.4% appears to be minor, there is a consistent high bias towards the duplicate up to around 1.5 g/t Au, which encompasses around 75% of the sample population. The average CV of over 22% is considered to be slightly outside of what is an acceptable performance.

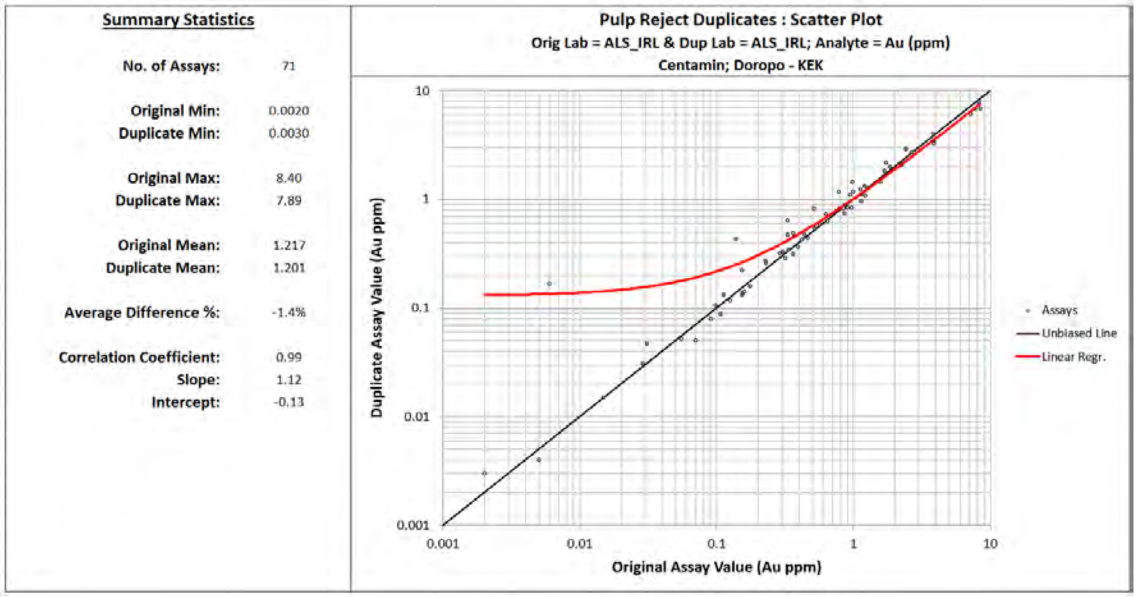


Figure 11.61 Prospect KEK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm

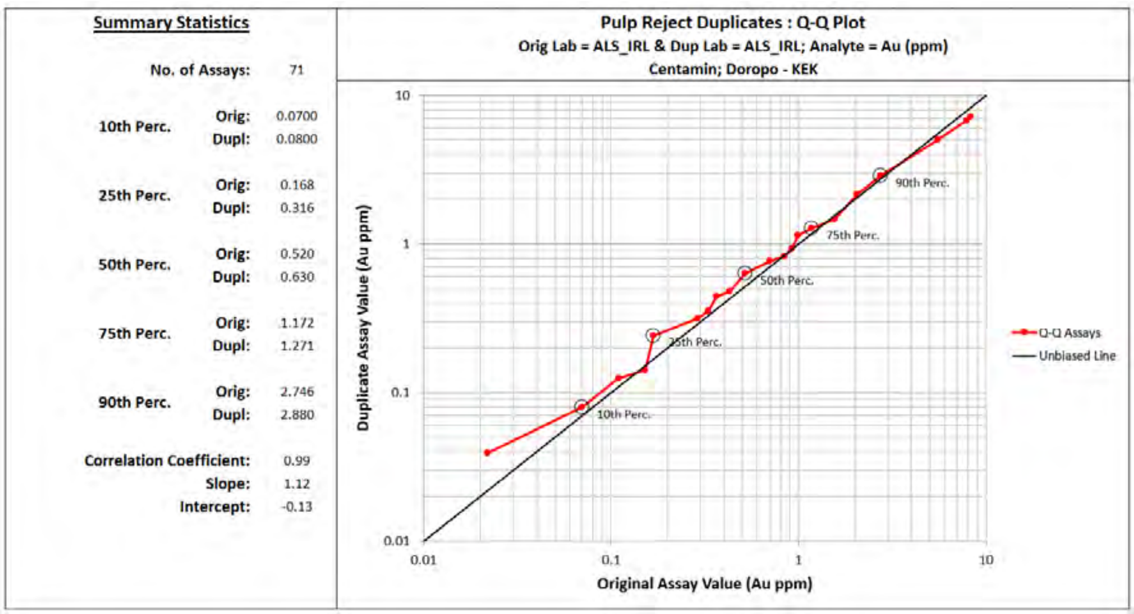


Figure 11.62 Prospect KEK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm

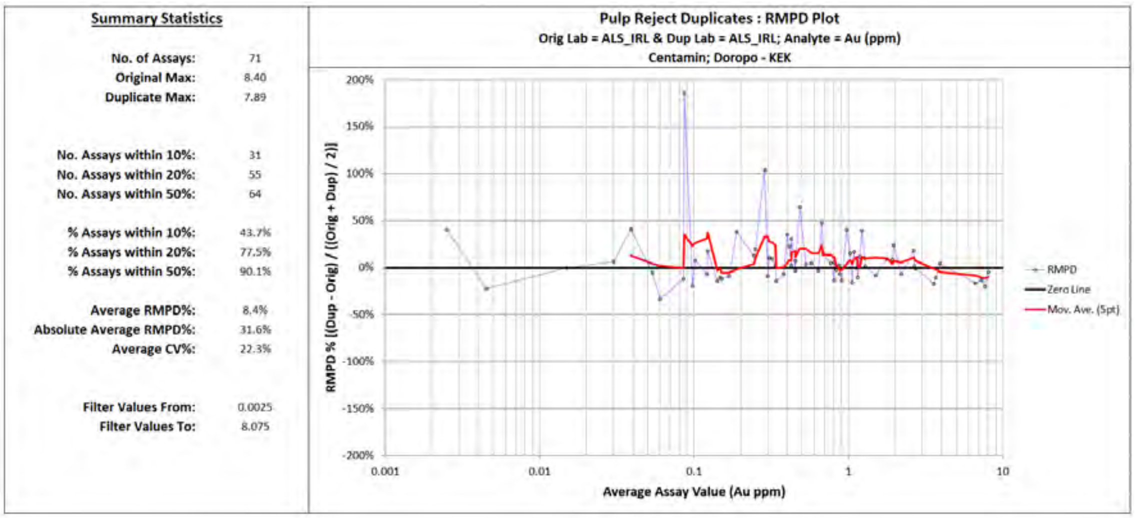


Figure 11.63 Prospect KEK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect KILOSEGUI

For RC field duplicates, over 4,000 sample pairs were collected Project to date. The average mean difference is minor at 1.1% (Figure 11.64) and with no perceptible grade range biases (Figure 11.65). The average CV of 27.9% (Figure 11.66) is within acceptable limits.

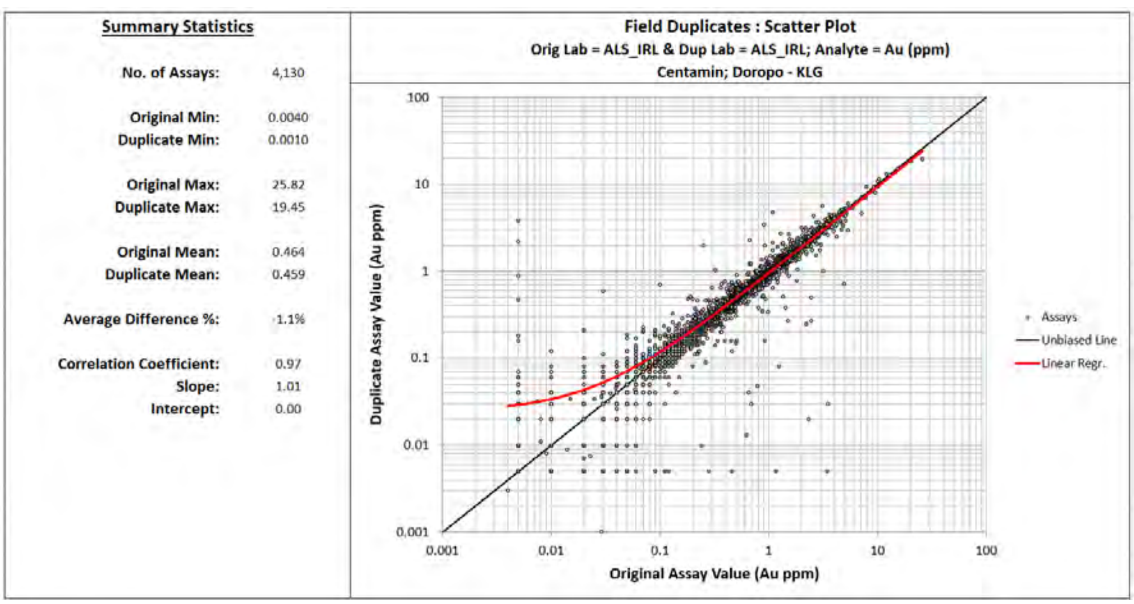


Figure 11.64 Prospect KLG, RC Field Duplicate Scatter Plot - To Date - Au ppm

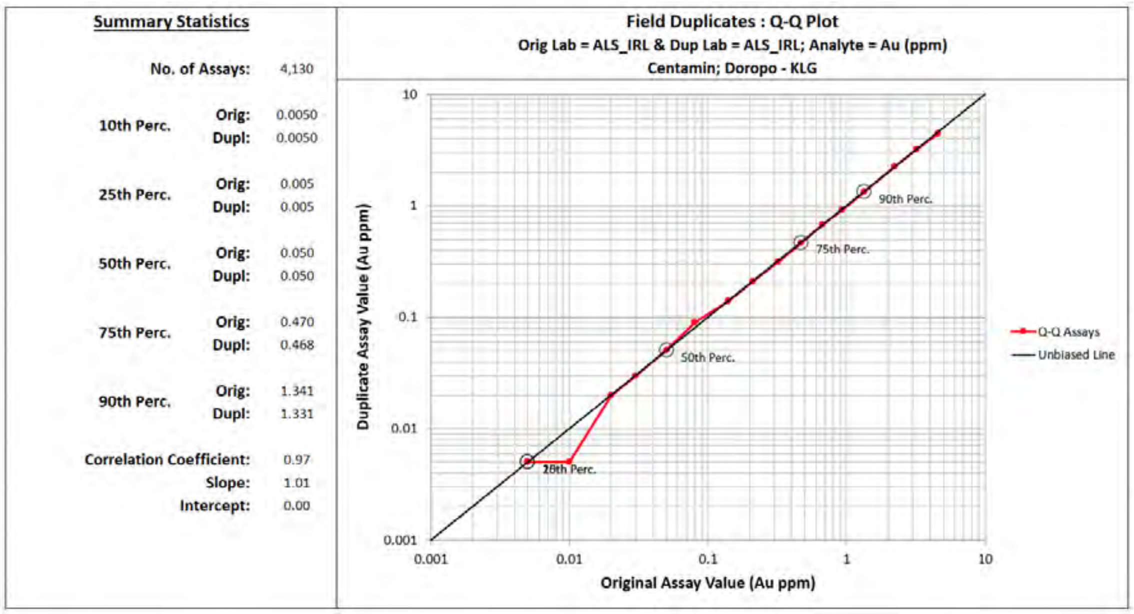


Figure 11.65 Prospect KLG, RC Field Duplicate Q-Q Plot - To Date - Au ppm

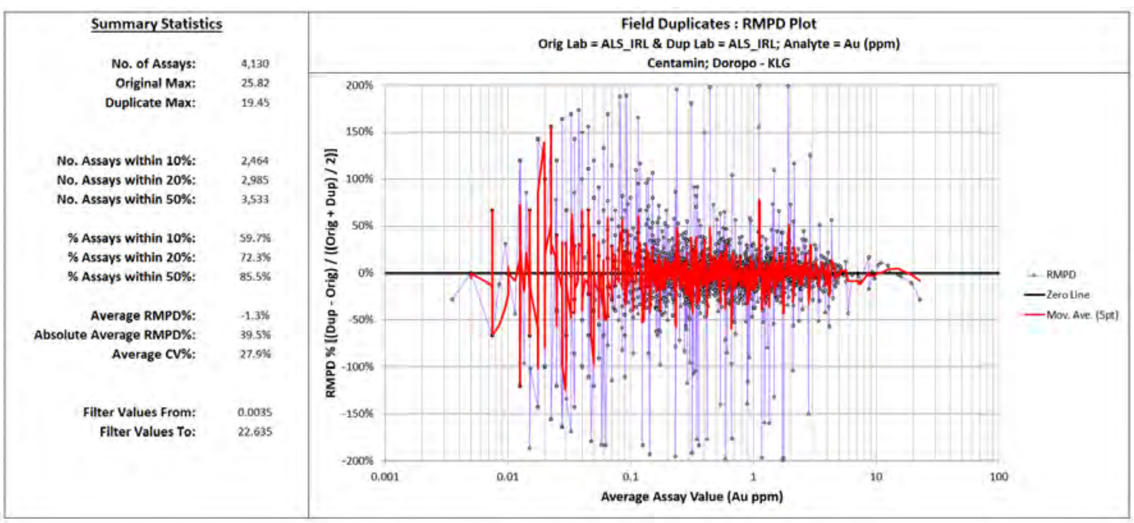


Figure 11.66 Prospect KLG, RC Field Duplicate RMDP Plot - To Date - Au ppm

There were 83 DD field duplicates submitted, all of which were analysed at VER_AB. The average difference in means of 7.6% is reasonable, but there is a perceptible bias towards the original between 0.2 - 3.0 g/t Au (Figure 11.67). The average CV = 40.7% is considered to be just outside what is considered acceptable.

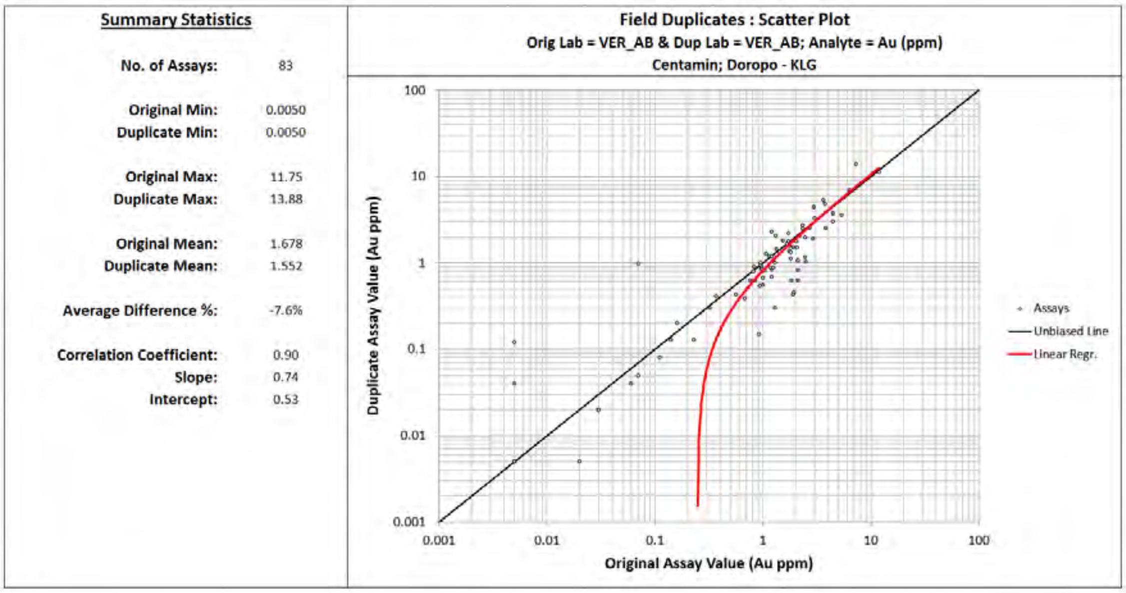


Figure 11.67 Prospect KLG, DD Field Duplicate Scatter Plot - To Date - Au ppm

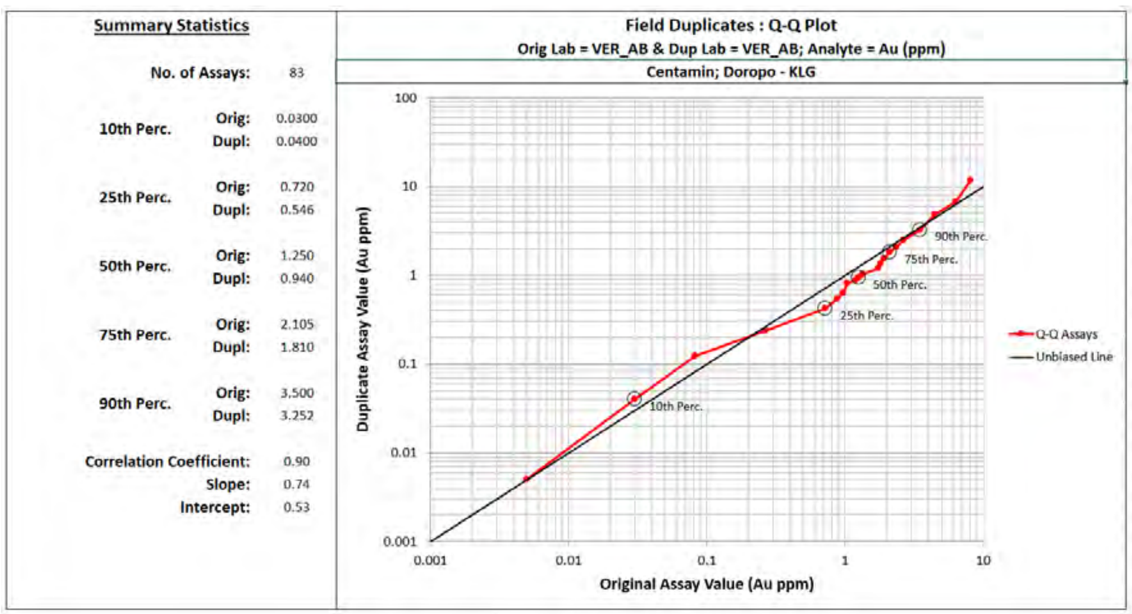


Figure 11.68 Prospect KLG, DD Field Duplicate Q-Q Plot - To Date - Au ppm

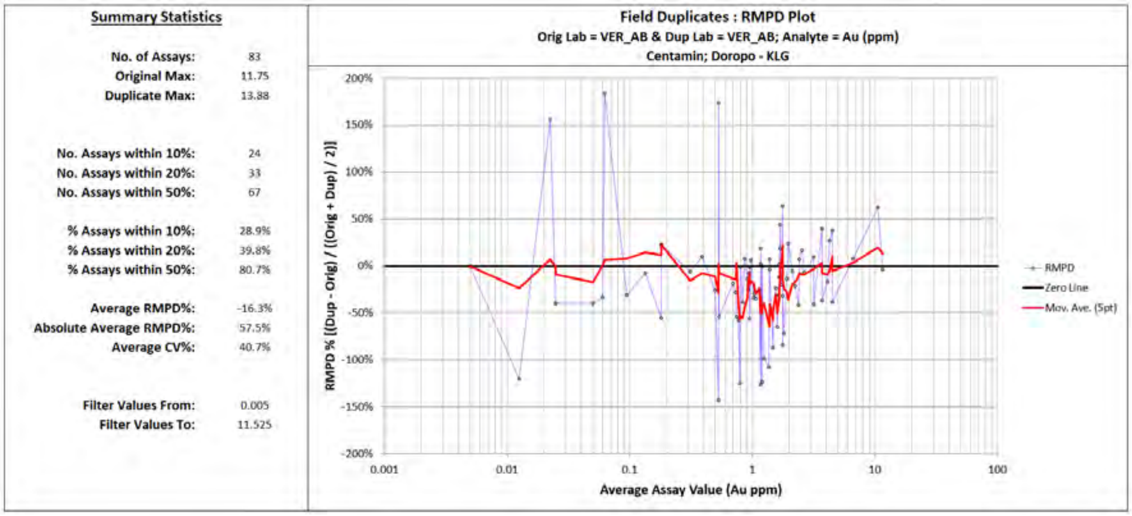


Figure 11.69 Prospect KLG, DD Field Duplicate RMDP Plot - To Date - Au ppm

The pulp duplicates, the majority of which were analysed at VER_AB, had a relative low (2.4%) difference in means (Figure 11.70) and no significant grade based biases (Figure 11.71), however the average CV of 27.4% is high (Figure 11.72) for pulps.

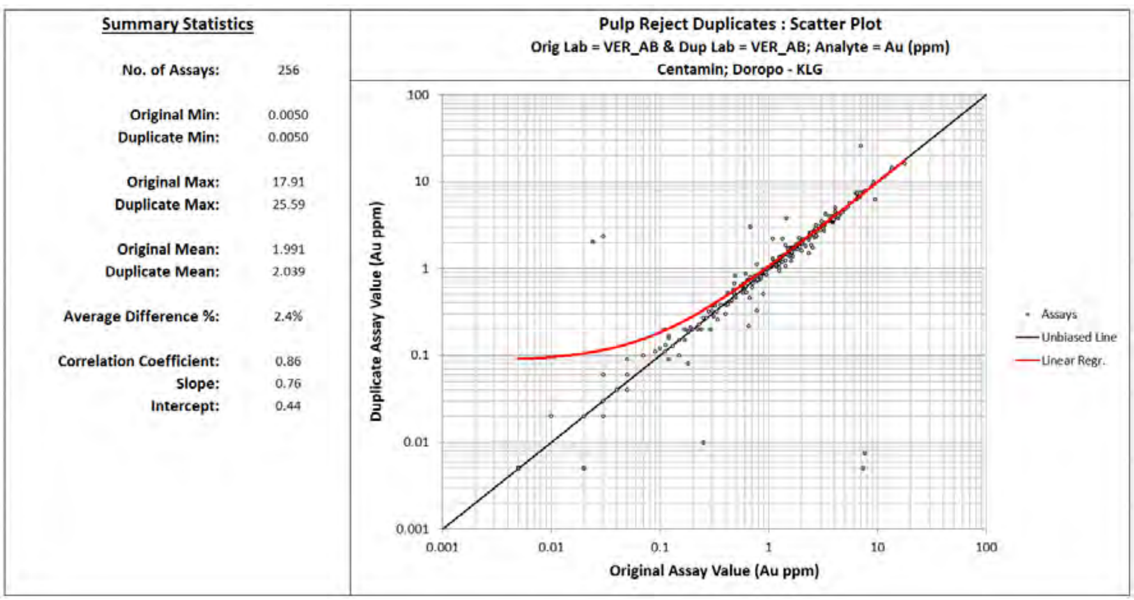


Figure 11.70 Prospect KLG, DD Pulp Duplicate Scatter Plot - To Date - Au ppm

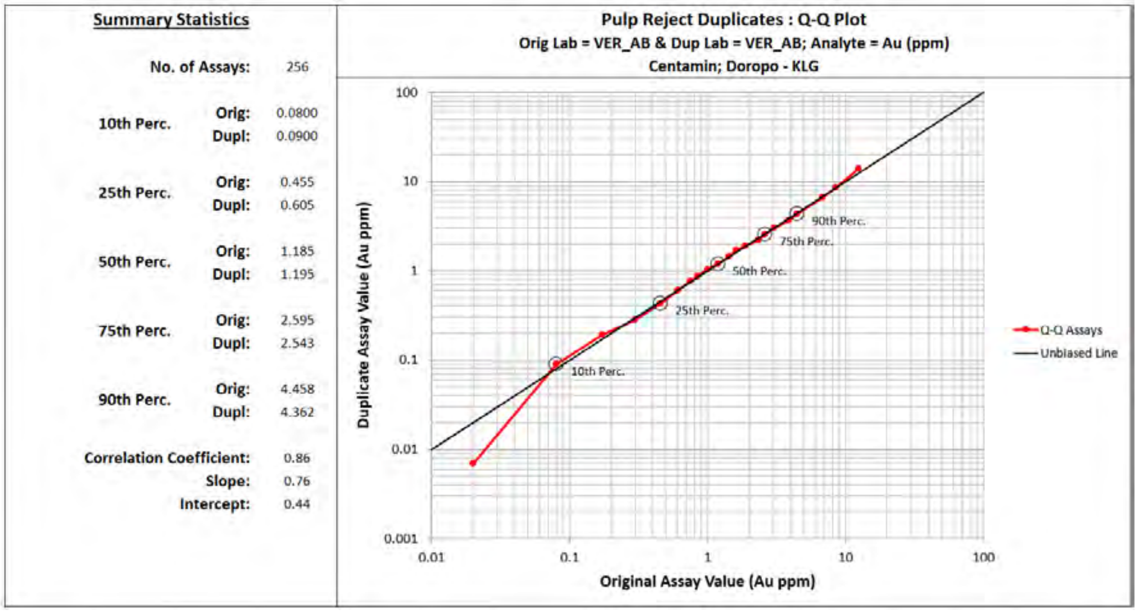


Figure 11.71 Prospect KLG, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm

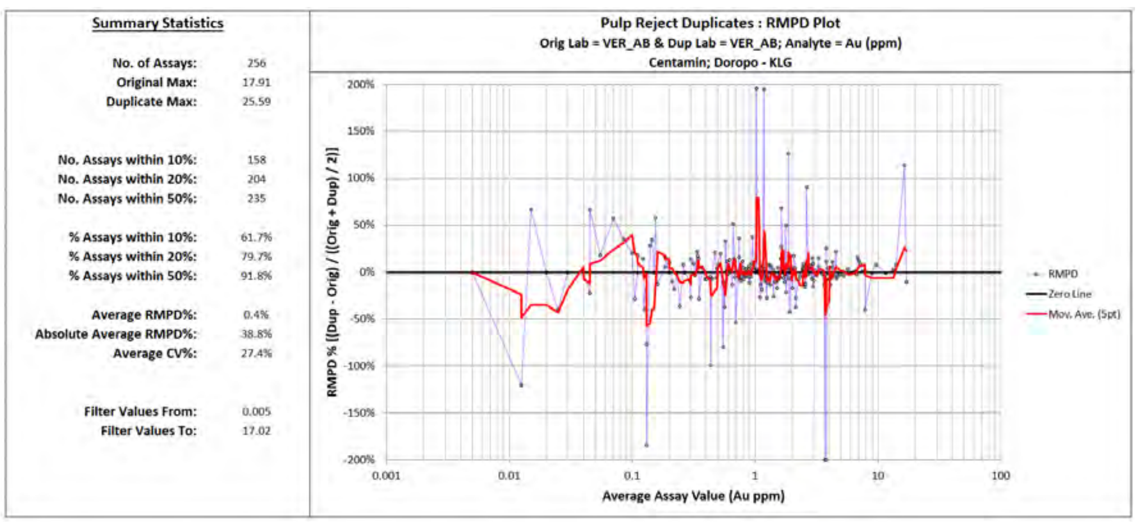


Figure 11.72 Prospect KLG, DD Pulp Duplicate RMPD Plot - To Date - Au ppm

Results - Prospect NARE

For RC field duplicates, 120 sample pairs were collected. All sample pairs were analysed by VER_AB. The average difference in means of 0.1% (Figure 11.73) is a good result and with no significant grade biases > 0.4 g/t Au (Figure 11.74). The average CV of 25.1% is considered an acceptable result (Figure 11.75).

There have been no DD or pulp duplicates carried out Project to date.

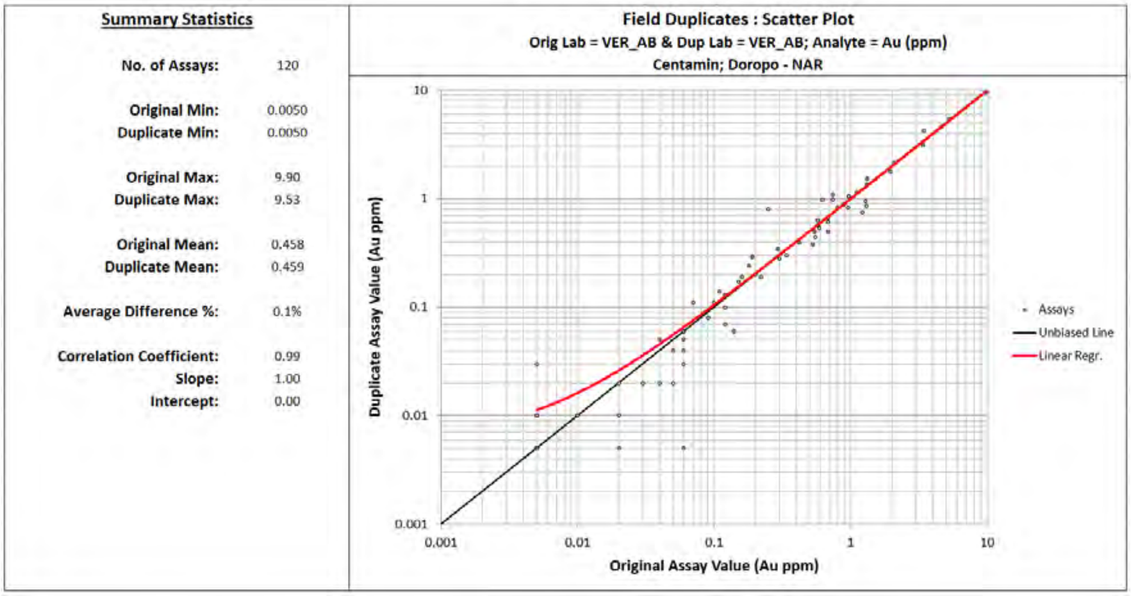


Figure 11.73 Prospect NAR, RC Field Duplicate Scatter Plot - To Date - Au ppm

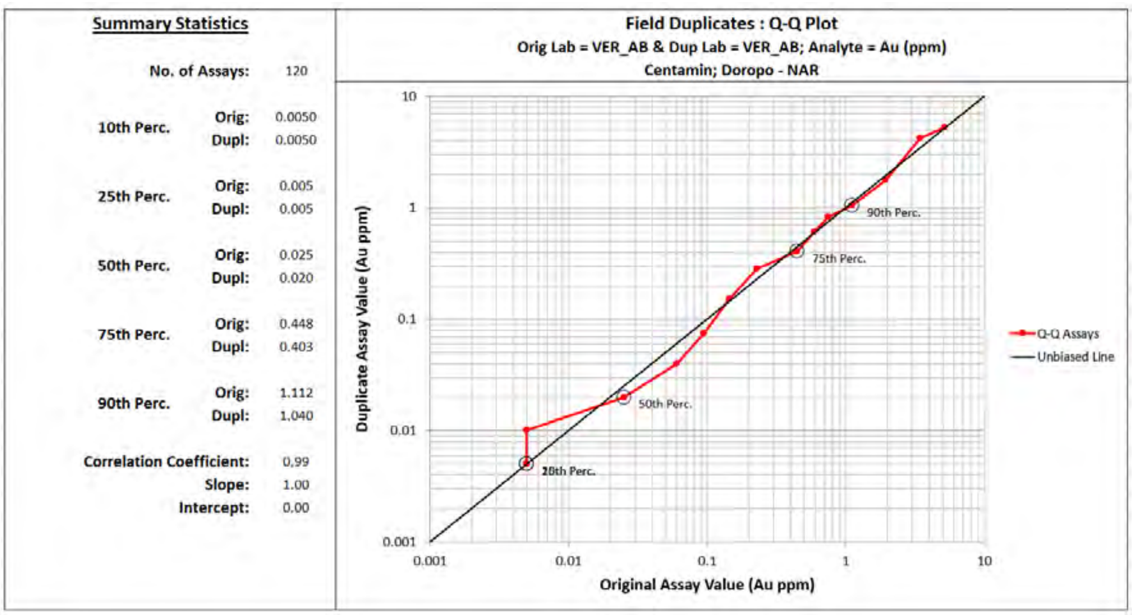


Figure 11.74 Prospect NAR, RC Field Duplicate Q-Q Plot - To Date - Au ppm

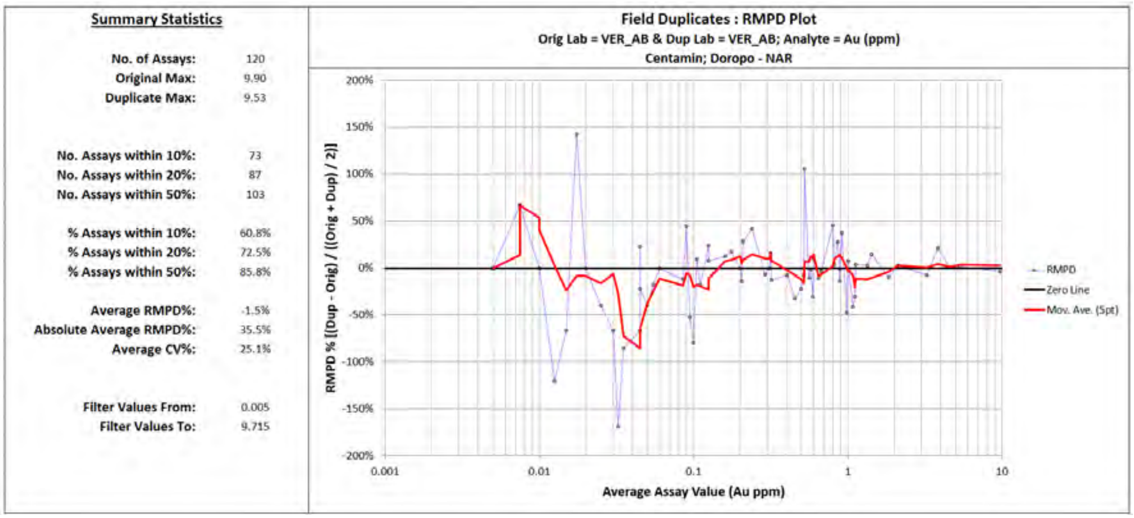


Figure 11.75 Prospect NAR, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect NOKPA

There were over 1,400 RC duplicate pairs analysed, with the vast majority analysed at VER_AB. The difference in average means of 3.9% is acceptable (Figure 11.76) and with no significant grade biases (Figure 11.77) - the average CV = 31.8% is deemed to be within acceptable limits (Figure 11.78).

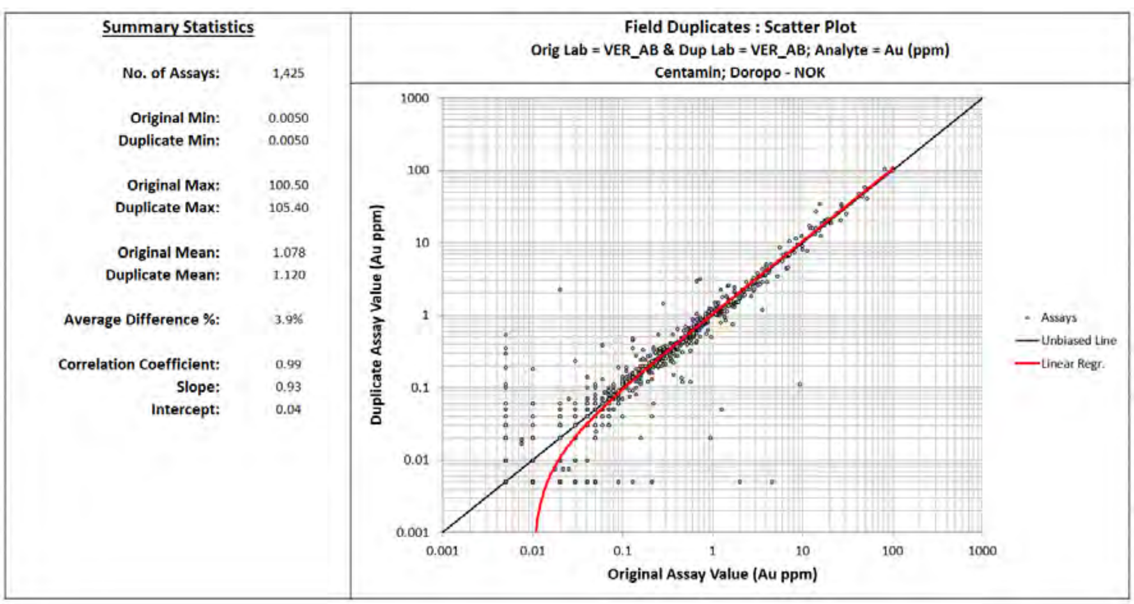


Figure 11.76 Prospect NOK, RC Field Duplicate Scatter Plot - To Date - Au ppm

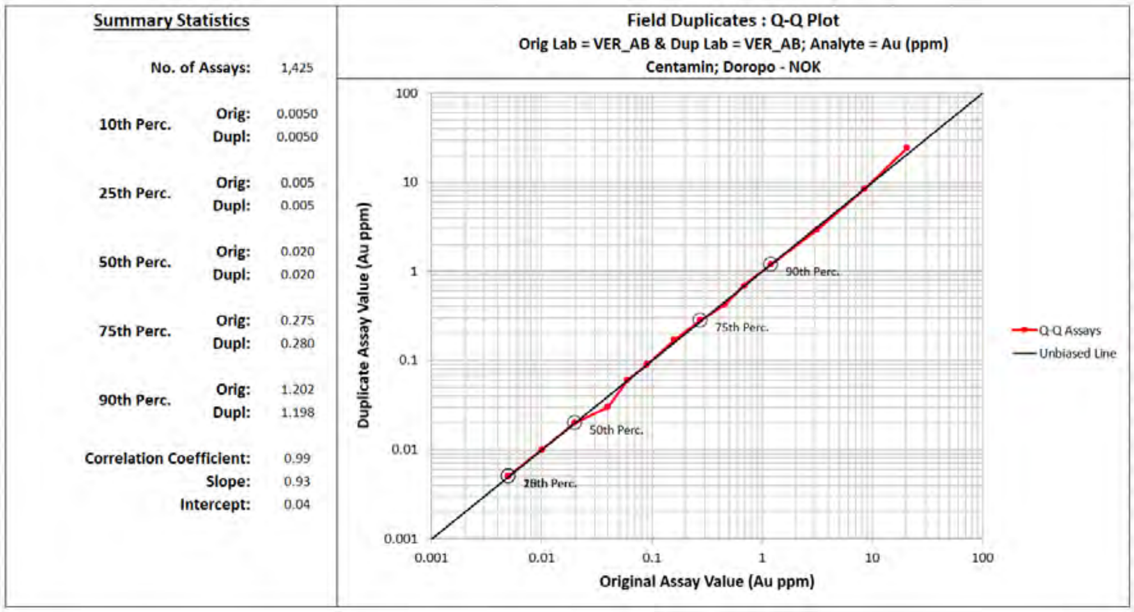


Figure 11.77 Prospect NOK, RC Field Duplicate Q-Q Plot - To Date - Au ppm

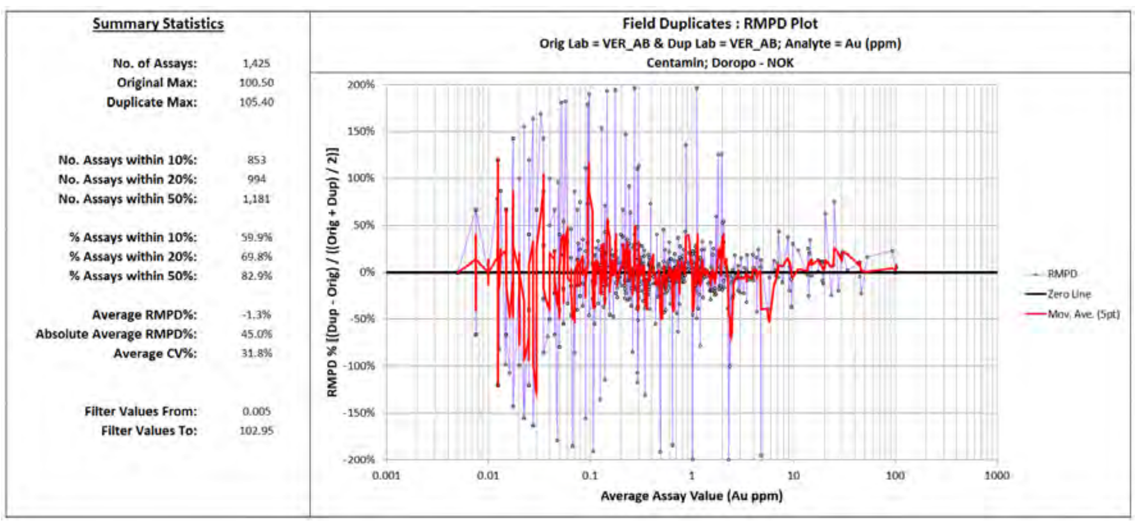


Figure 11.78 Prospect NOK, RC Field Duplicate RMDP Plot - To Date - Au ppm

There were 139 DD duplicate pairs analysed and all at VER_AB. As over 80% of the samples analysed were <0.1 g/t Au, it is difficult to make any definitive conclusions on performance and hence no performance charts have been displayed.

Pulp duplicates were submitted to all three laboratories. Overall, the performance was good, with a relatively small difference in means (6.7%), a slight grade bias towards the duplicate and an acceptable average CV of 19.3%. These statistics are displayed in Figure 11.79 through to Figure 11.81.

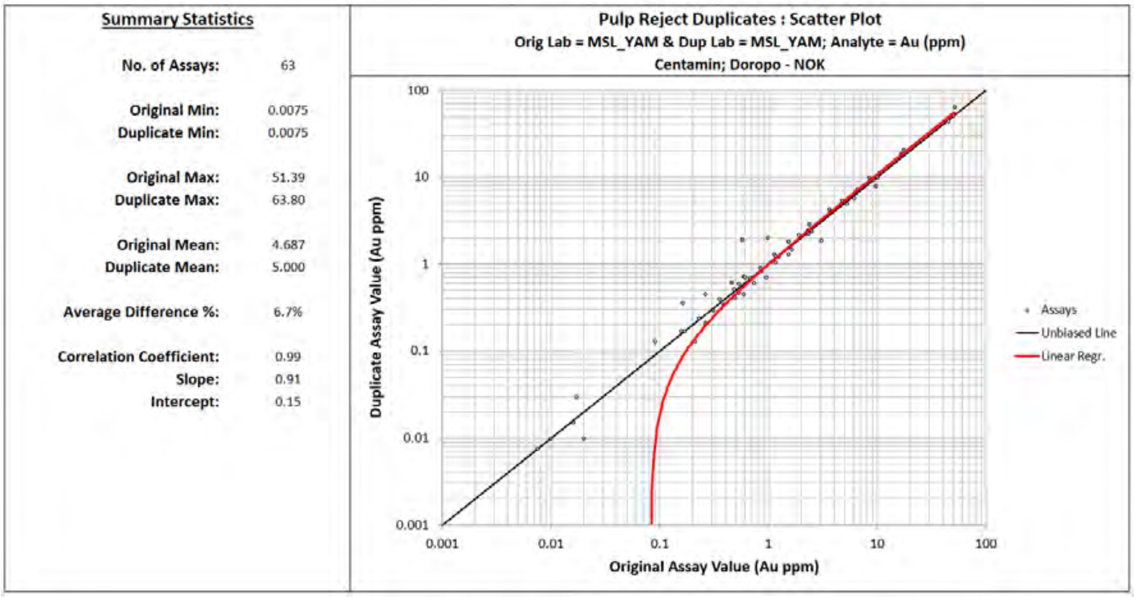


Figure 11.79 Prospect NOK, DD Pulp Duplicate Scatter Plot - To Date - Au ppm

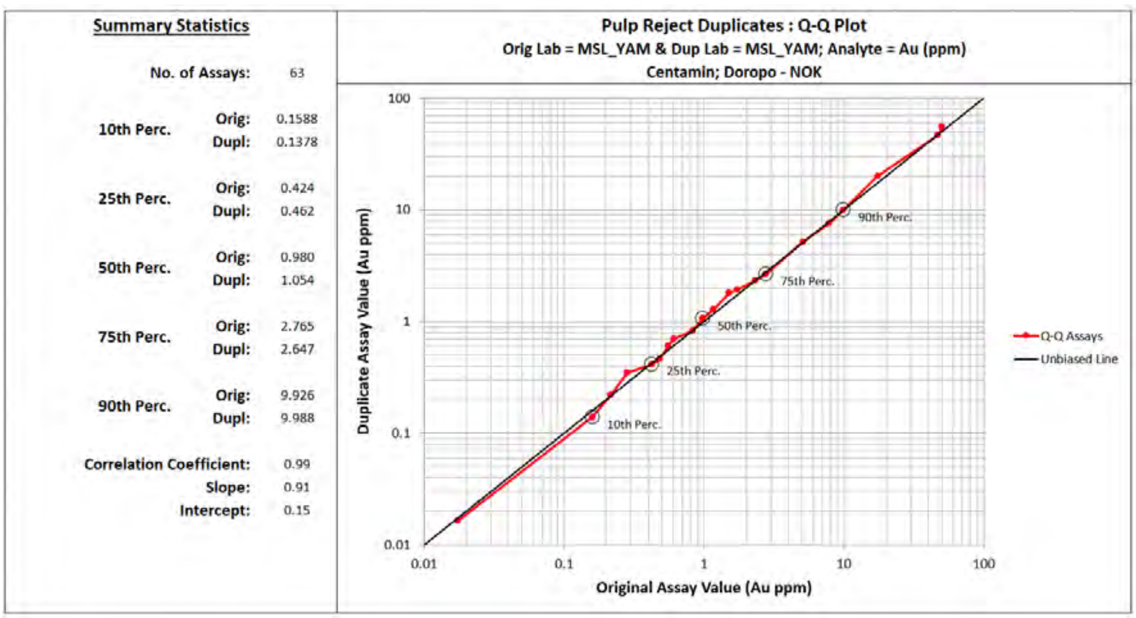


Figure 11.80 Prospect NOK, DD Pulp Duplicate Q-Q Plot - To Date - Au ppm

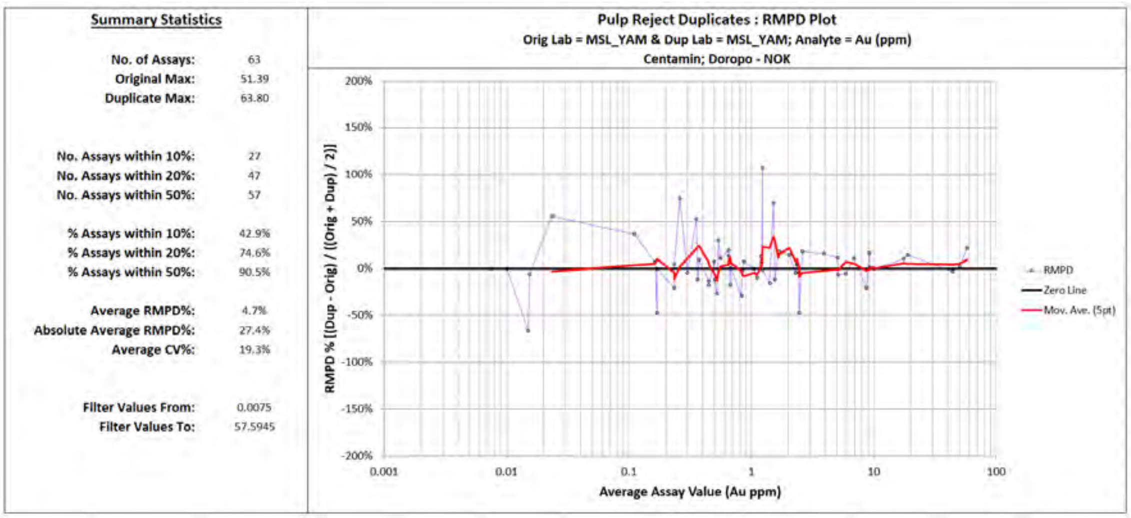


Figure 11.81 Prospect NOK, DD Pulp Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect SANBOROYO

There were 289 RC duplicate sample pairs analysed, all from VER_AB. The average difference in means of 11.7% is considered high (Figure 11.82), but there are no significant grade biases for assays >0.1 g/t Au, except for a small sample population (~5%) at >2.0 g/t Au (Figure 11.83). The average CV of 32.2% is considered to be within acceptable limits (Figure 11.84).

There was only 11 DD duplicates so no analysis of performance was undertaken. There are no pulp duplicates from this prospect.

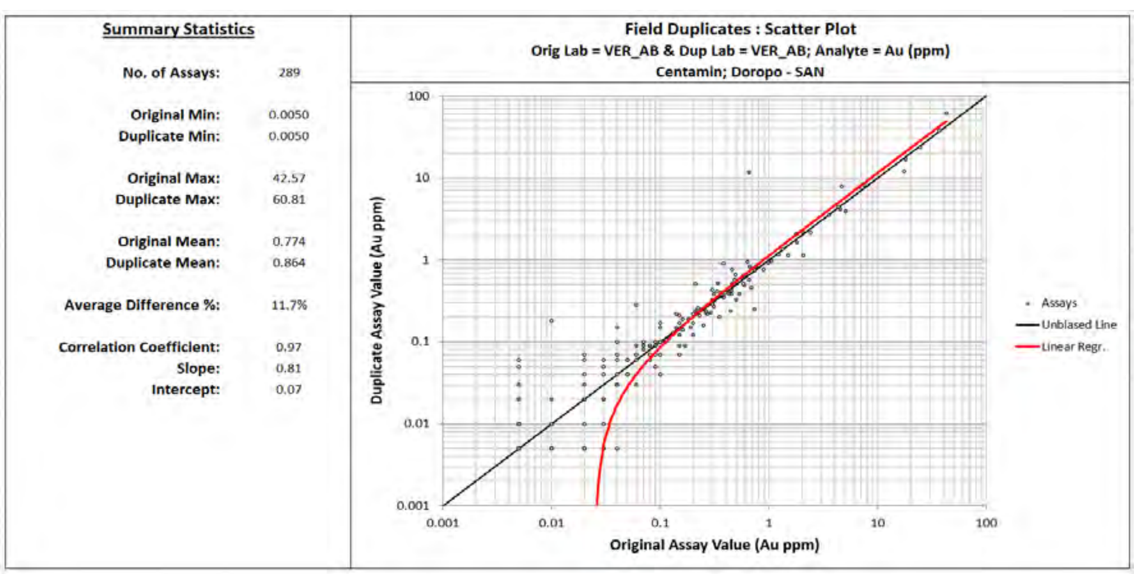


Figure 11.82 Prospect SAN, RC Field Duplicate Scatter Plot - To Date - Au ppm

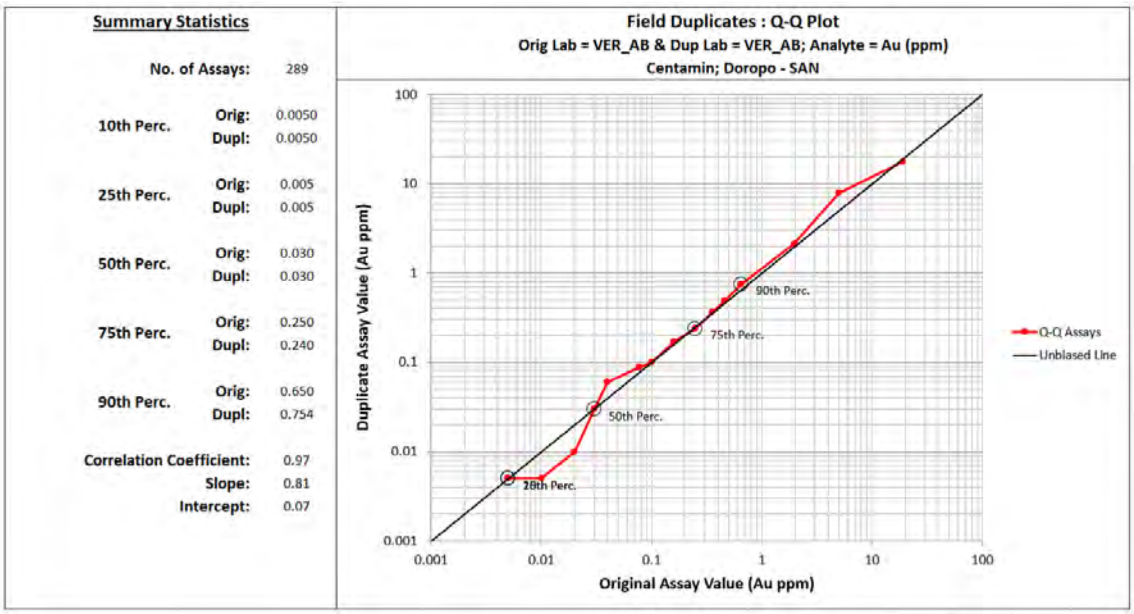


Figure 11.83 Prospect SAN, RC Field Duplicate Q-Q Plot - To Date - Au ppm

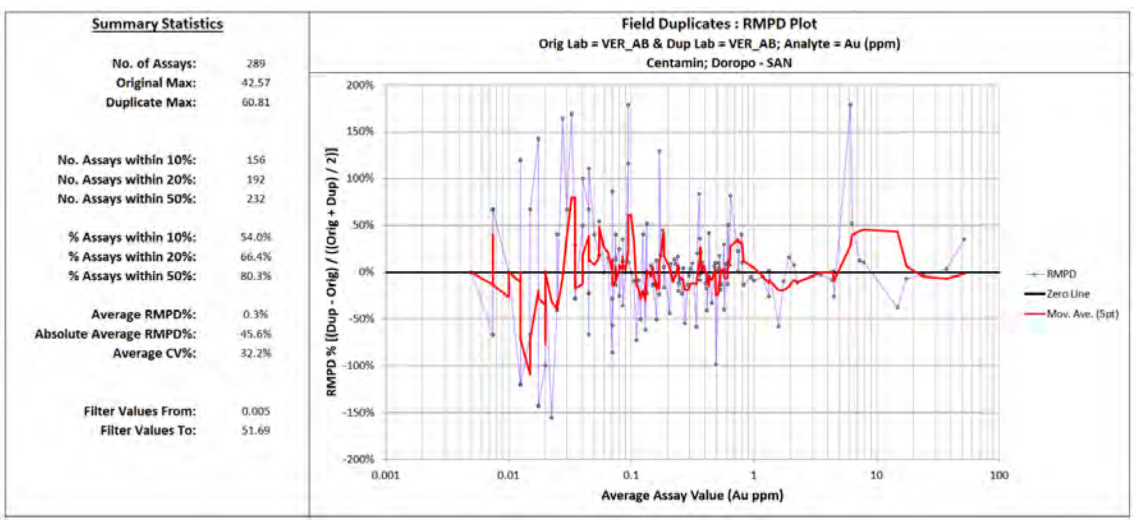


Figure 11.84 Prospect SAN, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect SOLO

There were 76 RC sample pairs submitted for Solo prospect to date. All were analysed at VER_AB. The average differences in means of 3.4% (with the duplicate being higher) is relatively small (Figure 11.85), but there is a perceptible high grade bias towards the duplicate between 0.1 - 0.7 g/t Au, which makes up about 15% of the sample population (Figure 11.86). The average CV of 35% is considered to be at the upper limits of acceptability (Figure 11.87).

There were no DD or pulp duplicates from this Project.

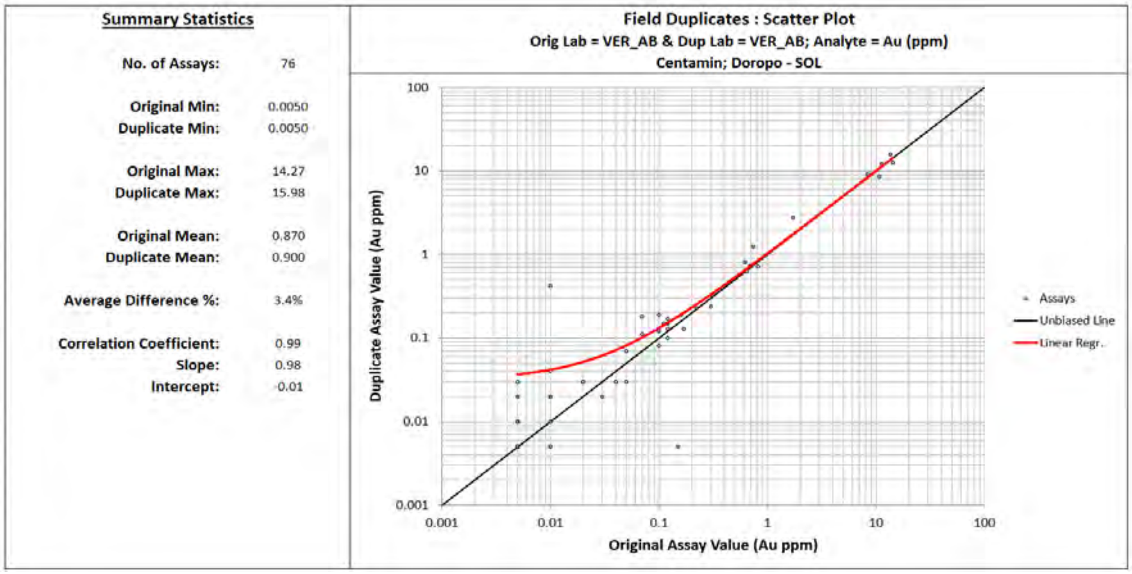


Figure 11.85 Prospect SOL, RC Field Duplicate Scatter Plot - To Date - Au ppm

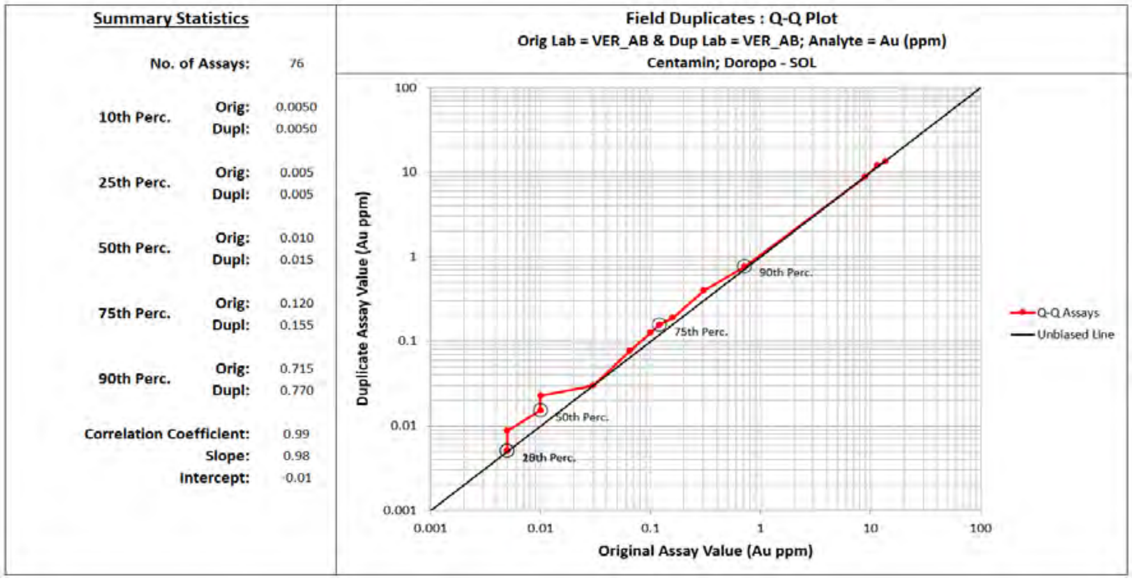


Figure 11.86 Prospect SOL, RC Field Duplicate Q-Q Plot - To Date - Au ppm

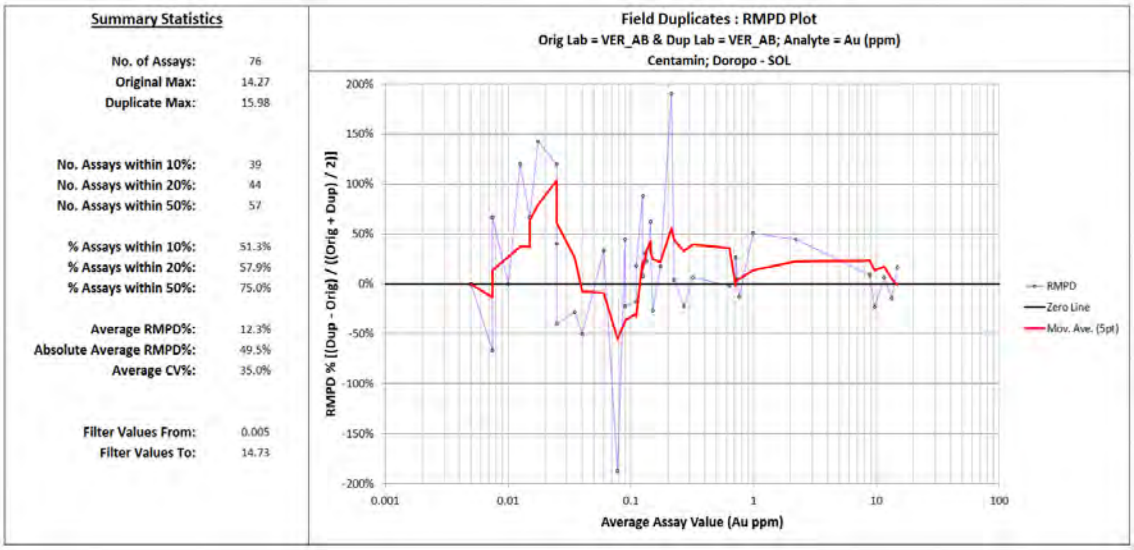


Figure 11.87 Prospect SOL, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect SOUWA

There were over 3,000 RC field duplicate pairs submitted, with the overwhelming majority analysed by VER_AB. There is a difference in average means of 5.3%, favouring the original sample (Figure 11.88), however the Q-Q plot shows no grade biases (Figure 11.89). The CV average of 31.2% is within acceptable limits (Figure 11.90).

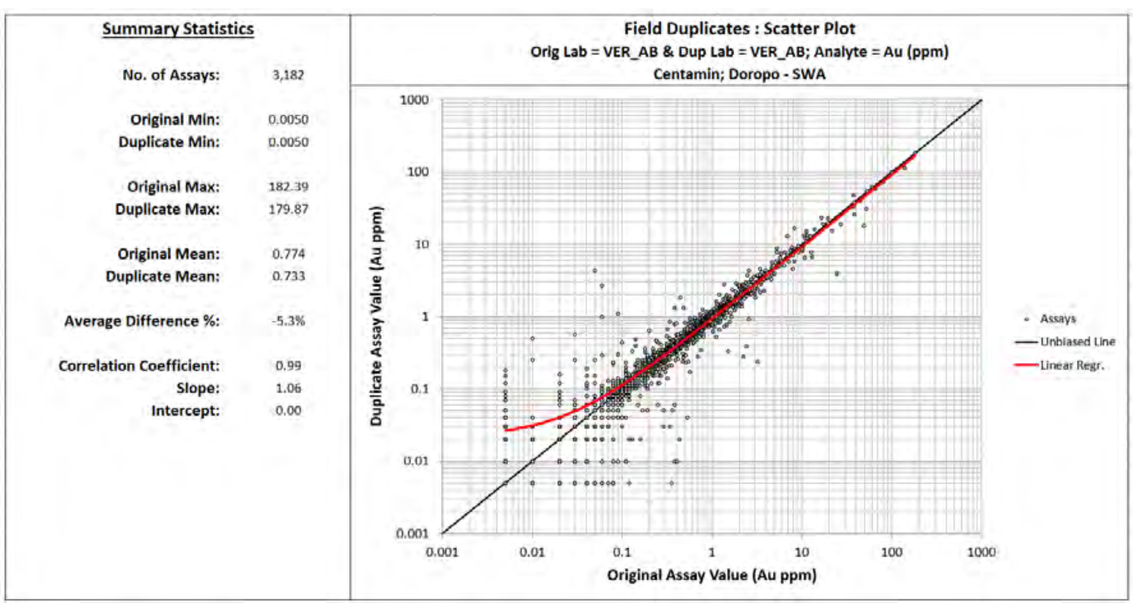


Figure 11.88 Prospect SWA, RC Field Duplicate Scatter Plot - To Date - Au ppm

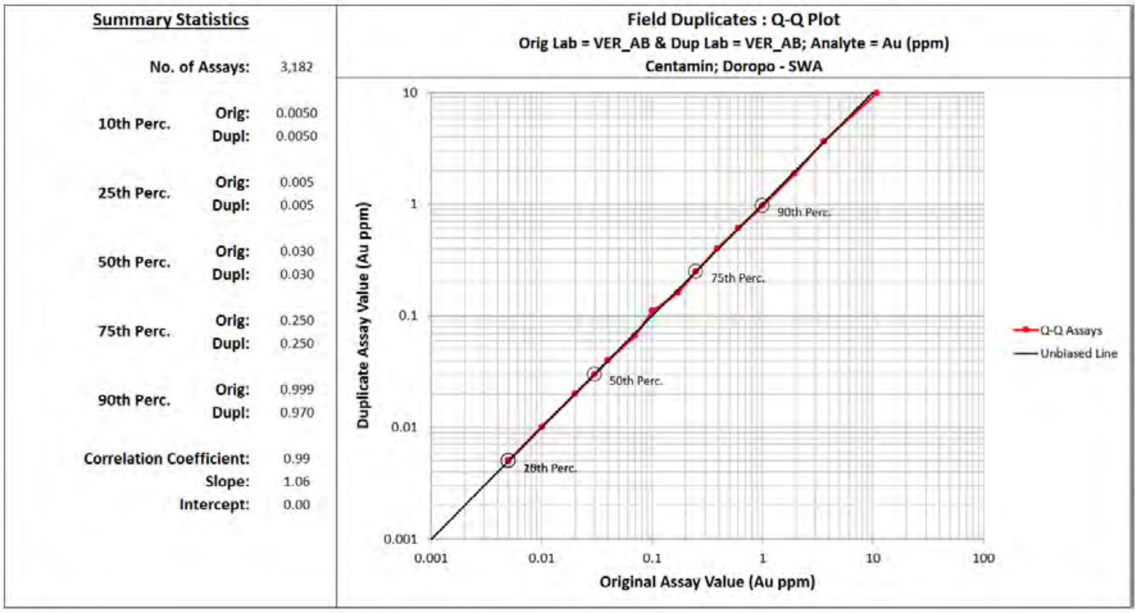


Figure 11.89 *Prospect SWA, RC Field Duplicate Q-Q Plot - To Date - Au ppm*

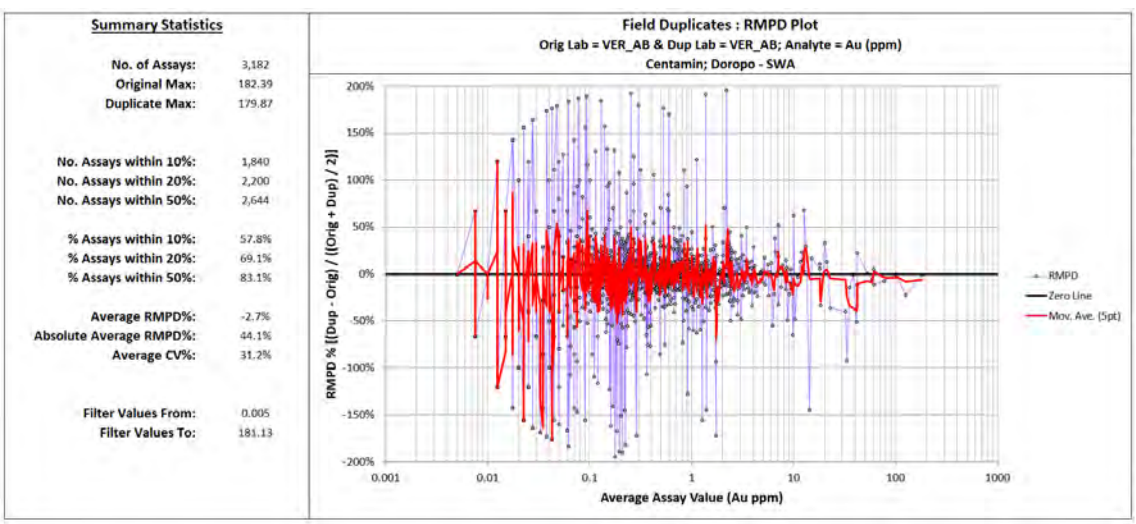


Figure 11.90 *Prospect SWA, RC Field Duplicate RMDP Plot - To Date - Au ppm*

The DD field duplicates show very poor performance in all analysis metrics (Figure 11.91 through to Figure 11.93). All samples were analysed at VER_AB. From this poor performance it is assumed that the mineralisation style at this deposit is more nuggety than the majority of the other deposits.

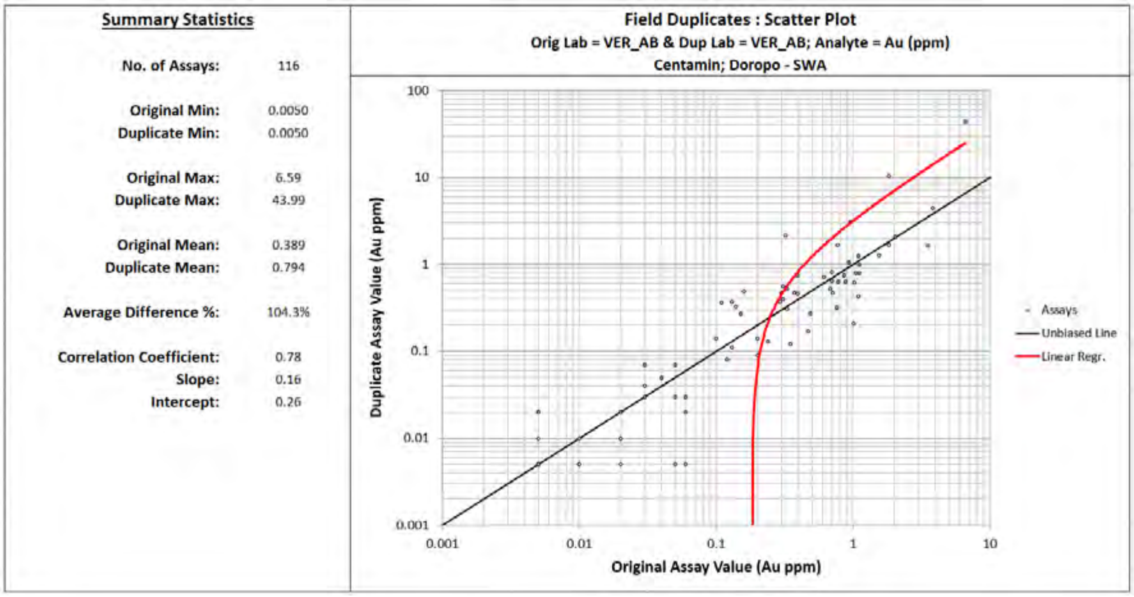


Figure 11.91 Prospect SWA, DD Field Duplicate Scatter Plot - To Date - Au ppm

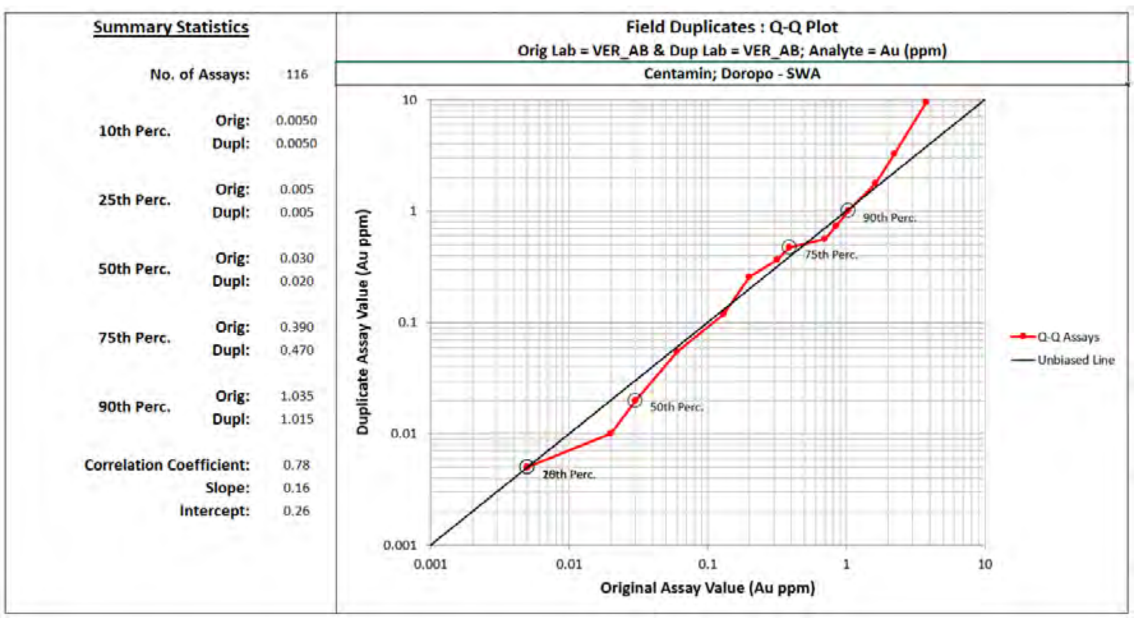


Figure 11.92 Prospect SWA, DD Field Duplicate Q-Q Plot - To Date - Au ppm

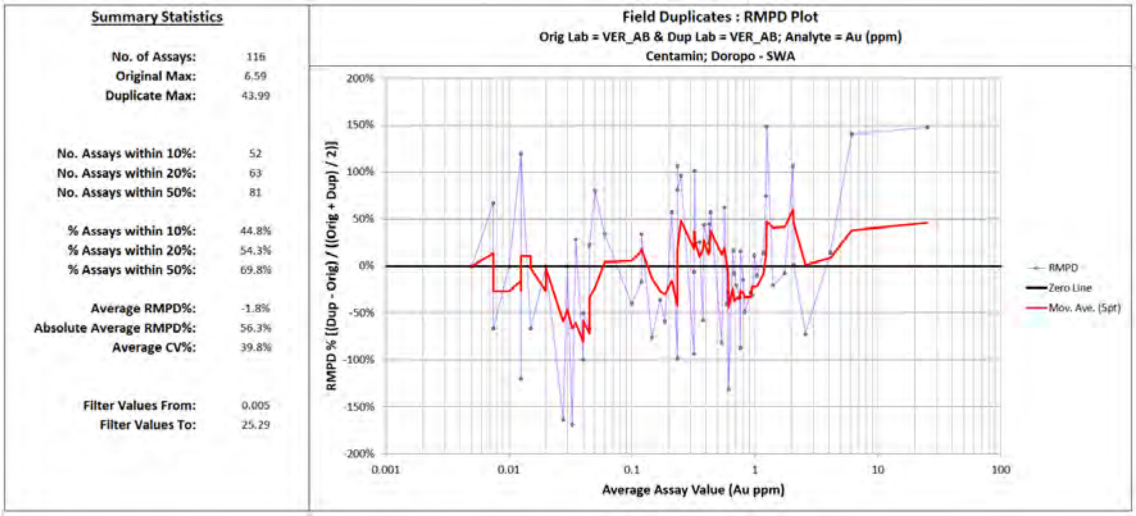


Figure 11.93 Prospect SWA, DD Field Duplicate RMDP Plot - To Date - Au ppm

All three laboratories were used for pulp duplicate analysis. The scatter plot (Figure 11.94) shows a low difference in means (0.9%), no significant grade ranges biases (Figure 11.95), but the average CV of 22.4% is above what is considered acceptable (Figure 11.96).

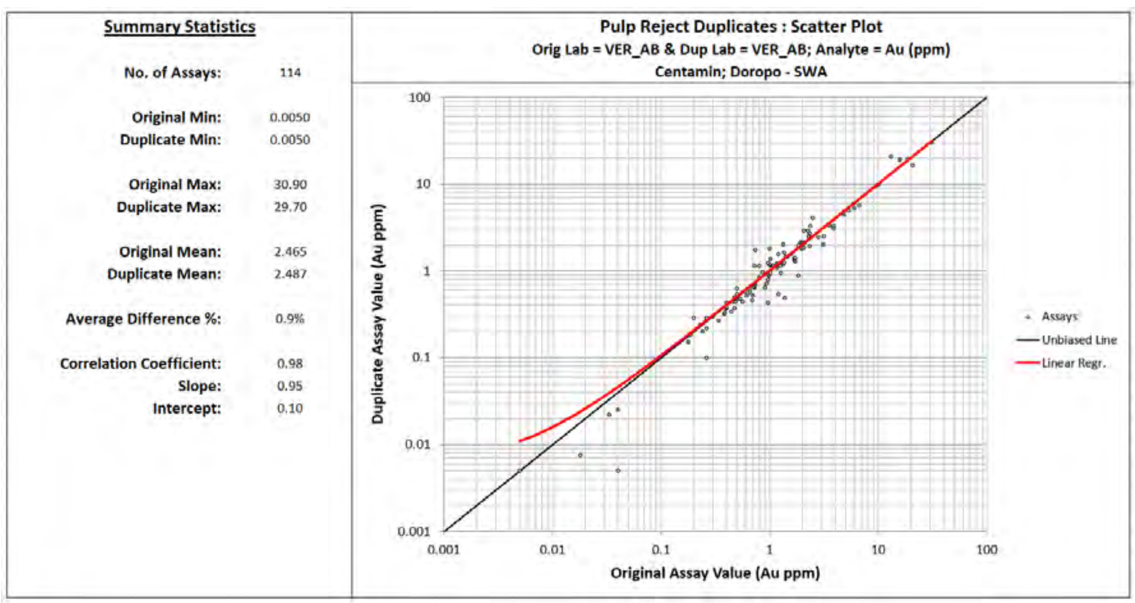


Figure 11.94 Prospect SWA, Pulp Duplicate Scatter Plot - To Date - Au ppm

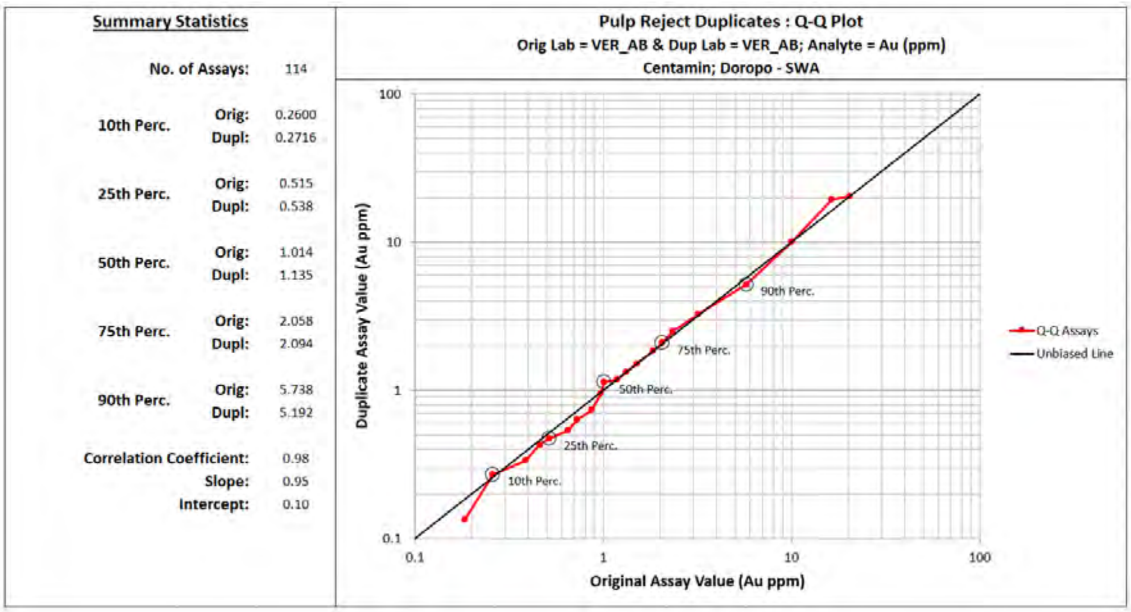


Figure 11.95 Prospect SWA, Pulp Duplicate Q-Q Plot - To Date - Au ppm

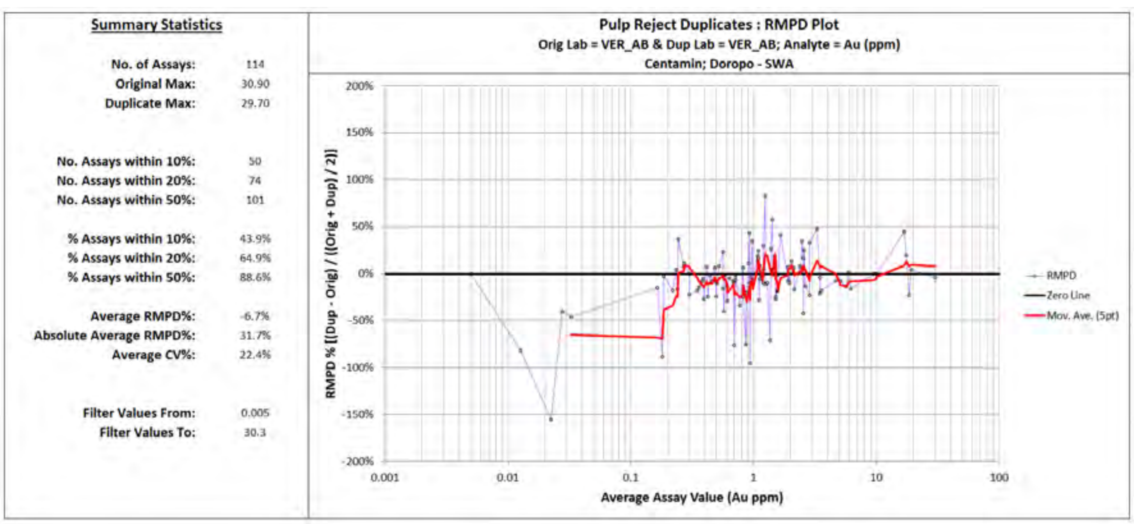


Figure 11.96 Prospect SWA, Pulp Duplicate RMPD Plot - To Date - Au ppm

Results - Prospect TCHOUAHIN

There were 656 RC field duplicates pairs analysed at VER_AB for the Project to date. The scatter (Figure 11.97) and Q-Q plots (Figure 11.98) show an acceptable performance, and the RMPD plot (Figure 11.99) has calculated an average CV of 30.8% which is acceptable performance.

There are no DD or pulp duplicates for this deposit.

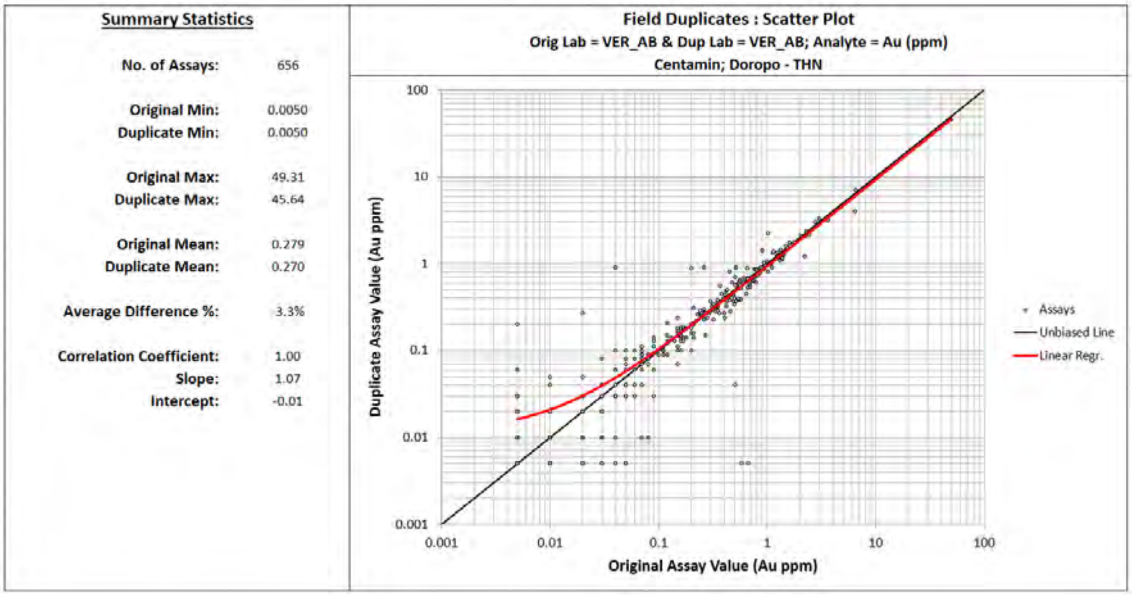


Figure 11.97 Prospect THN, RC Field Duplicate Scatter Plot - To Date - Au ppm

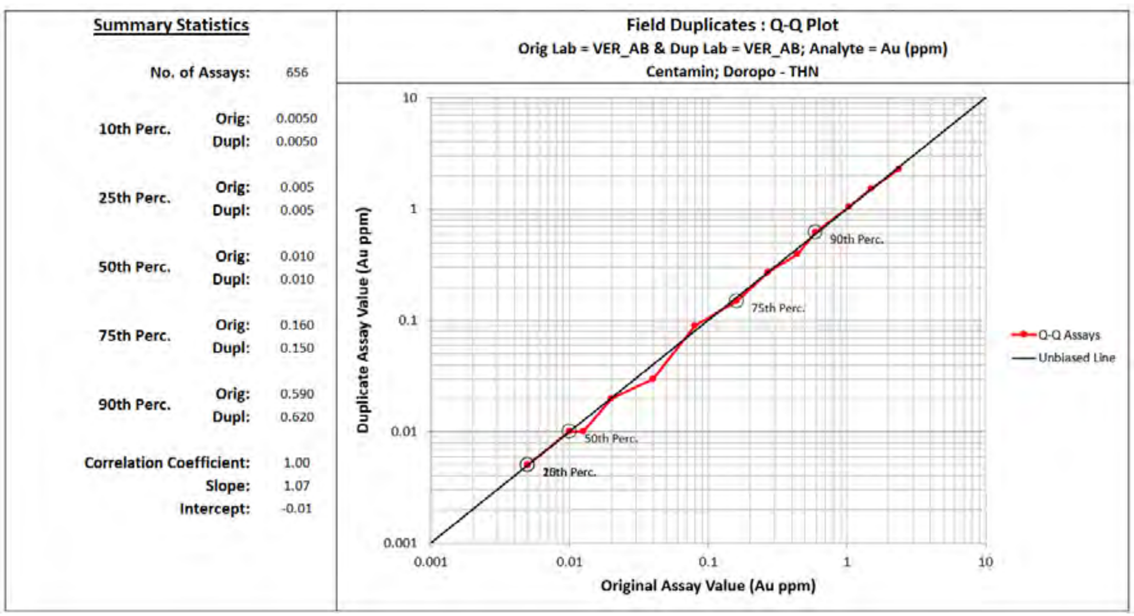


Figure 11.98 Prospect THN, RC Field Duplicate Q-Q Plot - To Date - Au ppm

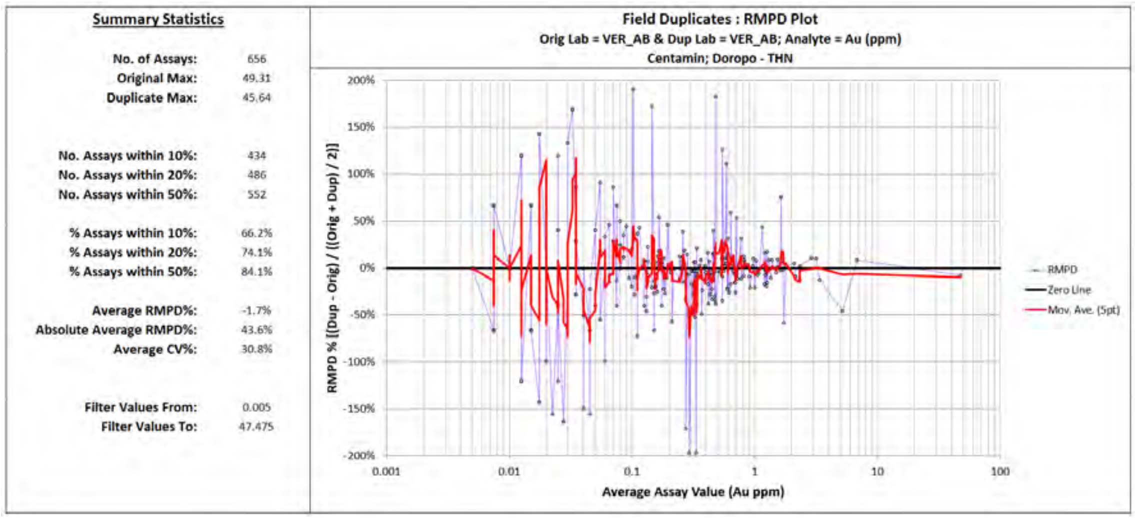


Figure 11.99 Prospect THN, RC Field Duplicate RMDP Plot - To Date - Au ppm

Results - Prospect VAKO

There was over 900 RC field duplicate sample pairs analysed, all at VER_AB. The scatter plot (Figure 11.100) shows a close comparison of average means (1.1%), no significant grade ranges biases (Figure 11.101), but an average CV% of 33.6% is considered acceptable (Figure 11.102).

There was only 40 DD field duplicates, so no analysis was completed. There were no pulp duplicates from this Project.

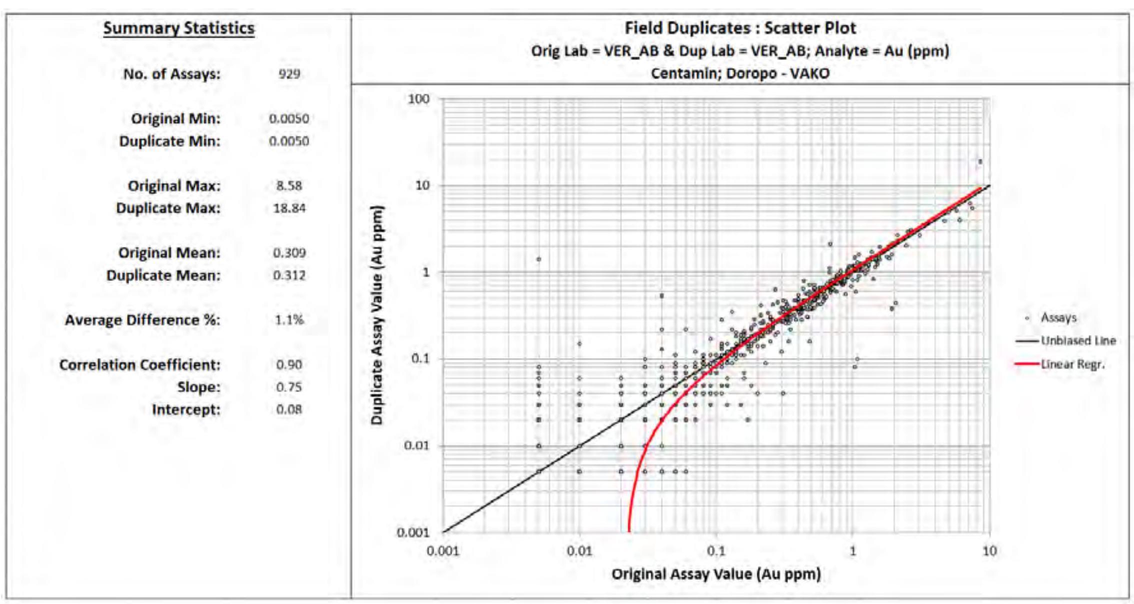


Figure 11.100 Prospect VAKO, RC Field Duplicate Scatter Plot - To Date - Au ppm

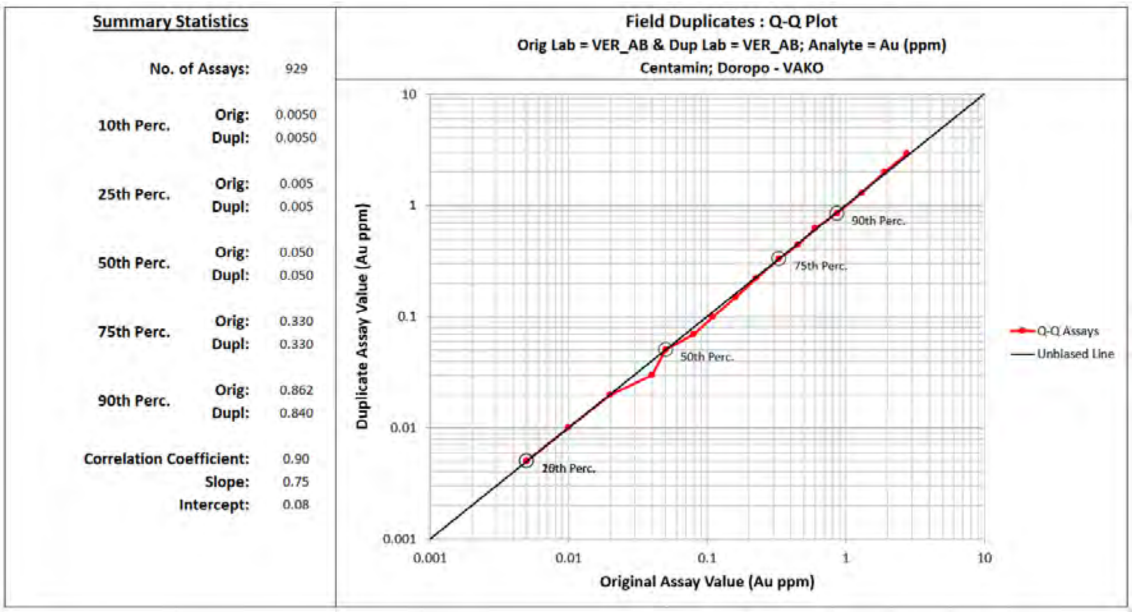


Figure 11.101 Prospect VAKO, RC Field Duplicate Q-Q Plot - To Date - Au ppm

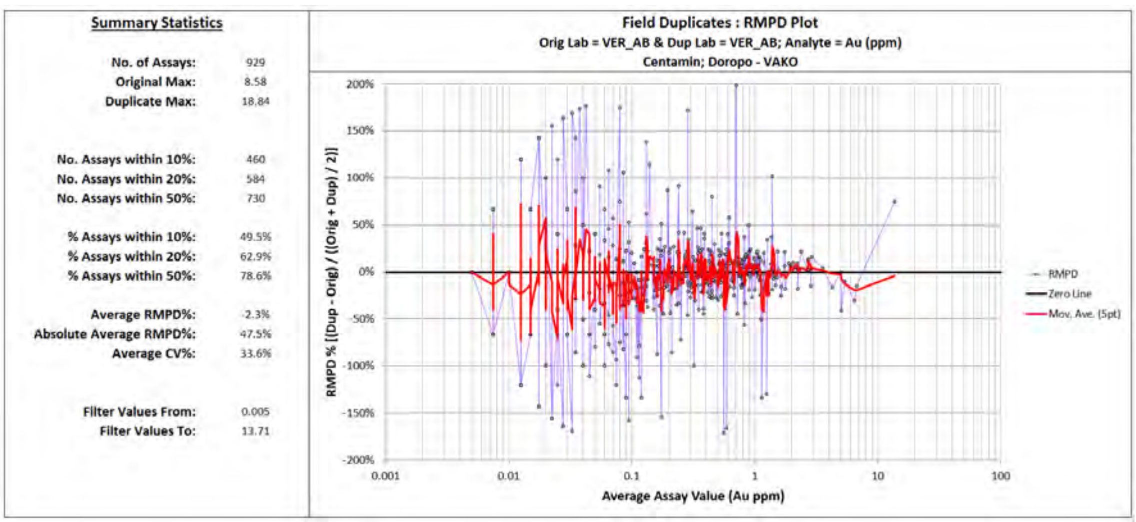


Figure 11.102 Prospect VAKO, RC Field Duplicate RMDP Plot - To Date - Au ppm

Summary RC, DD and Pulp Duplicate Performance

A summary of overall performance of RC and DD in terms of average CV% and split on deposits is tabulated in Table 11.13.

The mean average CV% for DD field duplicated was higher than RC at 39%. This is to be expected as the smaller hole diameter increases the variability of the nugget effect. The potential effect is exaggerated in this case as there were ¼ core duplicates taken in the early drill out stages.

The intended aim of RC field duplicates is to determine the performance of the sampling equipment but looking at this analysis in isolation can give a wrong impression on the performance of the sampling equipment due to the nugget effect. However, the information gained can still be used by the Qualified Person to gauge the potential nugget effect from each of the deposits.

Deposit	RC		DD	
	No. pairs	Average CV %	No pairs*	Average CV %
ATI	885	30.3%	38	44.8%
CHG	2,763	30.4%	57	35.3%
ENI	1,321	38.8%	19	46.1%
HAN	1,328	29.6%	73	25.8%
HND	706	30.2%	-	-
KEK	*1,254*	*30.6%*	-	-
KLG	4,130	27.9%	83	40.7%
NAR	120	25.1%	-	-
NOK	1,425	31.8%	-	-
SAN	*289*	*32.2%*	-	-
SOL	76	35.0%	-	-
SWA	3,128	31.2%	116	39.8%
THN	656	30.8%	-	-
VAKO	929	33.6%	-	-
All	19,010	31.2%	386	39.0%

**In some cases there were duplicates but not enough for a relevant analysis or there was a large population at low levels of Au*

Table 11.13 Field Duplicates Performance by Deposit - Project to Date

Table 11.14 shows the number of pulp duplicate pairs from each deposit. Some of the deposits showing no pulp duplicates did have some data, but the number of pairs was considered to be statistically too low for a meaningful analysis.

Deposit	Pulp Duplicates	
	No. Pairs*	Average CV %
ATI	-	-
CHG	187	31%
ENI	-	-
HAN	73	25.8%
HND	-	-
KEK	*71*	*22.3%*
KLG	256	27.4%
NAR	-	-
NOK	-	-
SAN	-	-

Deposit	Pulp Duplicates	
	No. Pairs*	Average CV %
SOL	-	-
SWA	114	22.4%
THN	-	-
VAKO	-	-
All	701	26%

** In some cases there were duplicates but not enough for a relevant analysis.*

Table 11.14 Pulp Duplicate Performance by Deposit - Project to Date

RC and DD Twinning Analysis

Paired statistics between RC and DD samples were computed for the fourteen deposits and show that for a migration distance of 5 m to 10 m; the DD distribution is 17% under than the RC distribution (Table 11.15).

VARIABLE	Count	Minimum	Maximum	Mean	Std. Dev.	CV	DD/RC Rel. Diff
AU_DD	1938	0	134.37	1.86	7.05	3.79	-16%
AU_RC_5m	1938	0	229.77	2.21	8.25	3.73	
AU_DD	2341	0	134.37	1.83	6.73	3.68	-18%
AU_RC_7.5m	2341	0	229.77	2.23	7.96	3.57	
AU_DD	2420	0	134.37	1.82	6.74	3.70	-17%
AU_RC_10m	2420	0	229.77	2.2	7.85	3.57	

Table 11.15 Paired Bivariate Statistics Between Au Values from DD Holes and Au Values from RC Holes Migrated to the DD Holes (the Migrated Distance is Shown After the Variable Name)

Q-Q plots of the DD values and the paired RC values at distances of 5 m, 7.5 m and 10 m show a relatively low and consistent bias within all grade ranges from the DD assays (Figure 11.103 to Figure 11.105).

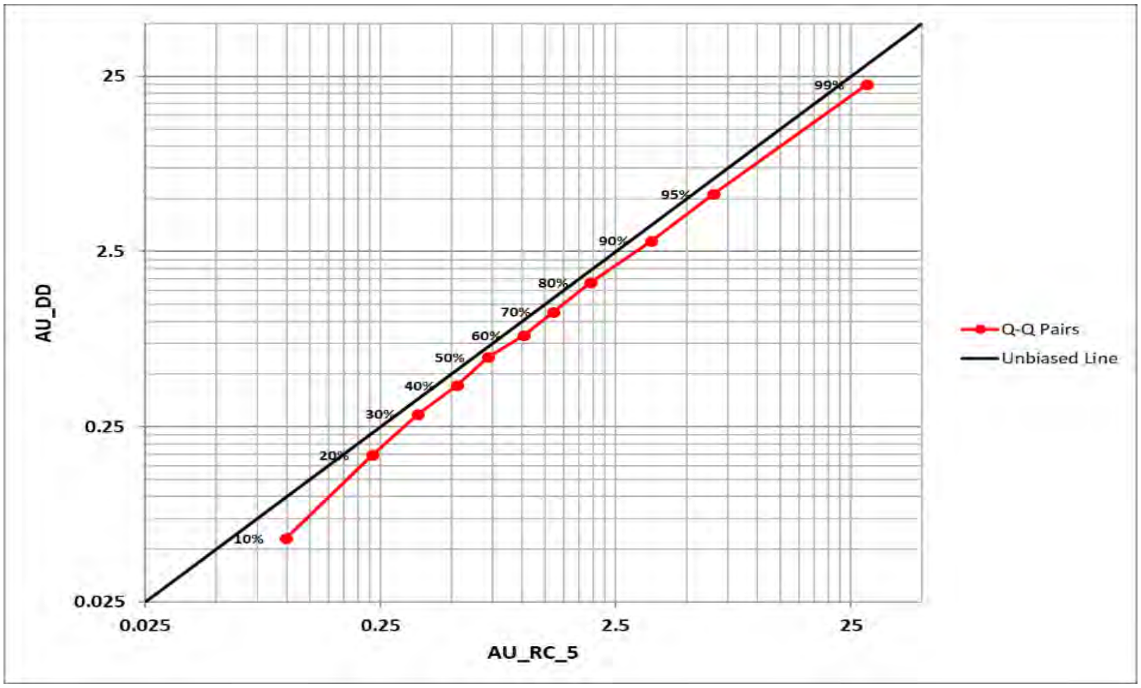


Figure 11.103 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 5 m

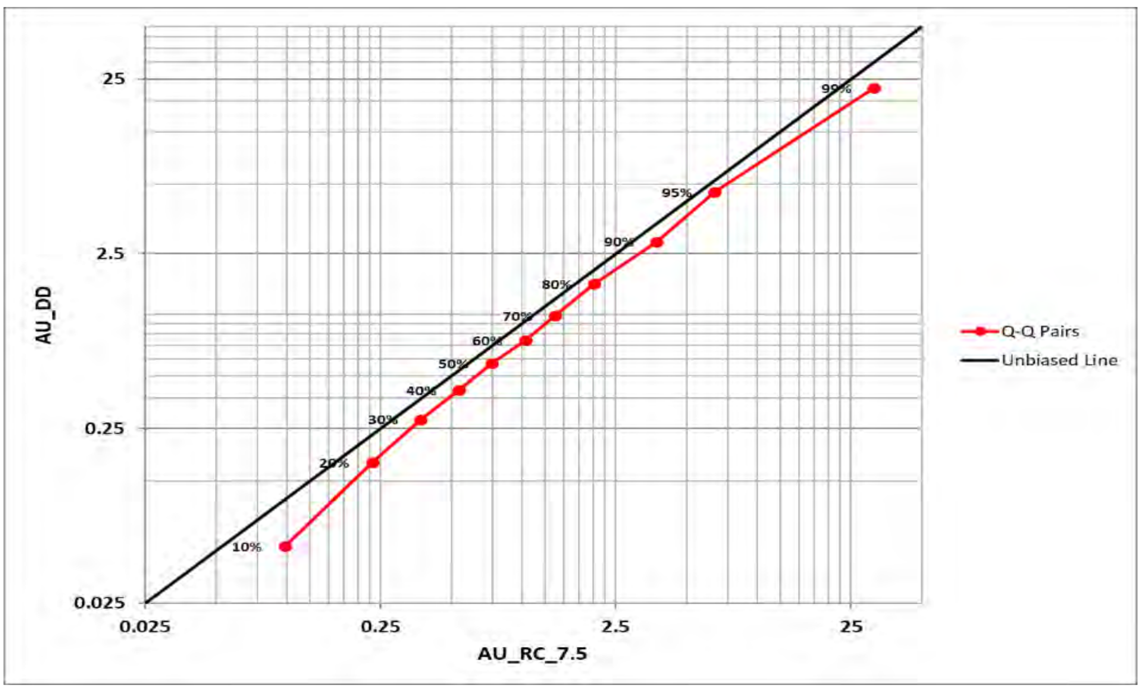


Figure 11.104 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 7.5 m

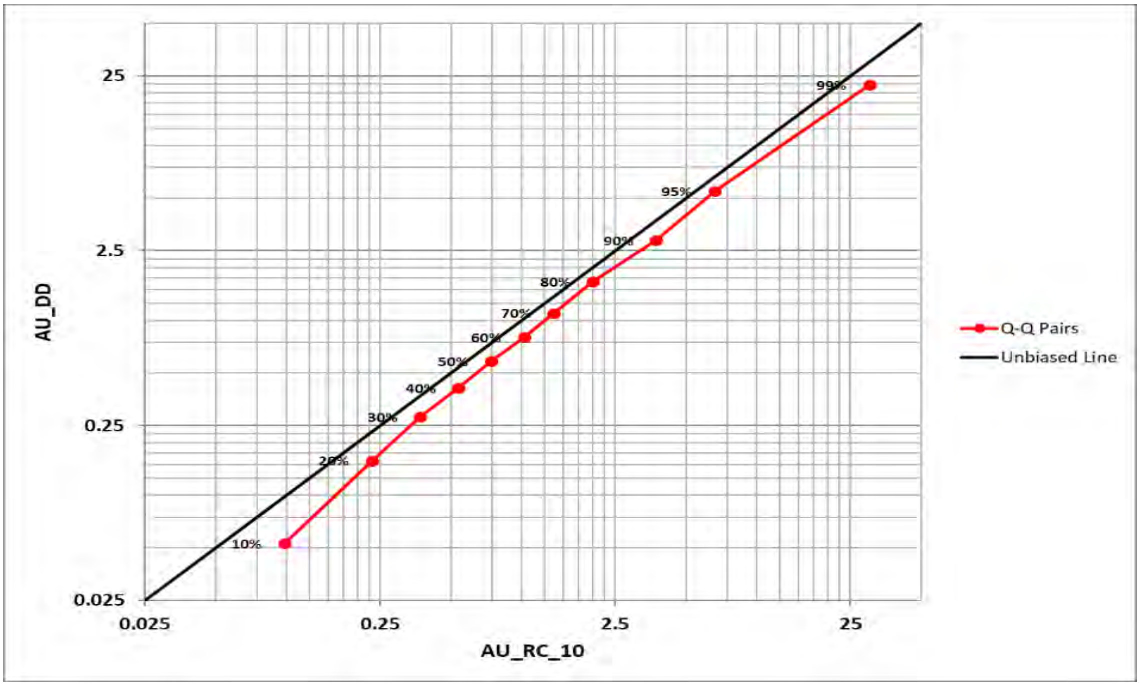


Figure 11.105 QQ-plots Between DD Assay Values and the RC Values Migrated from Maximum 10 m

Log-probability plots between the Au values from DD holes and RC-migrated holes show a similar trend with the DD distribution being systematically under the RC distribution (Figure 11.106), but due to the log scale, it is clear that the upper, high-grade tail of the distribution is having the most impact.

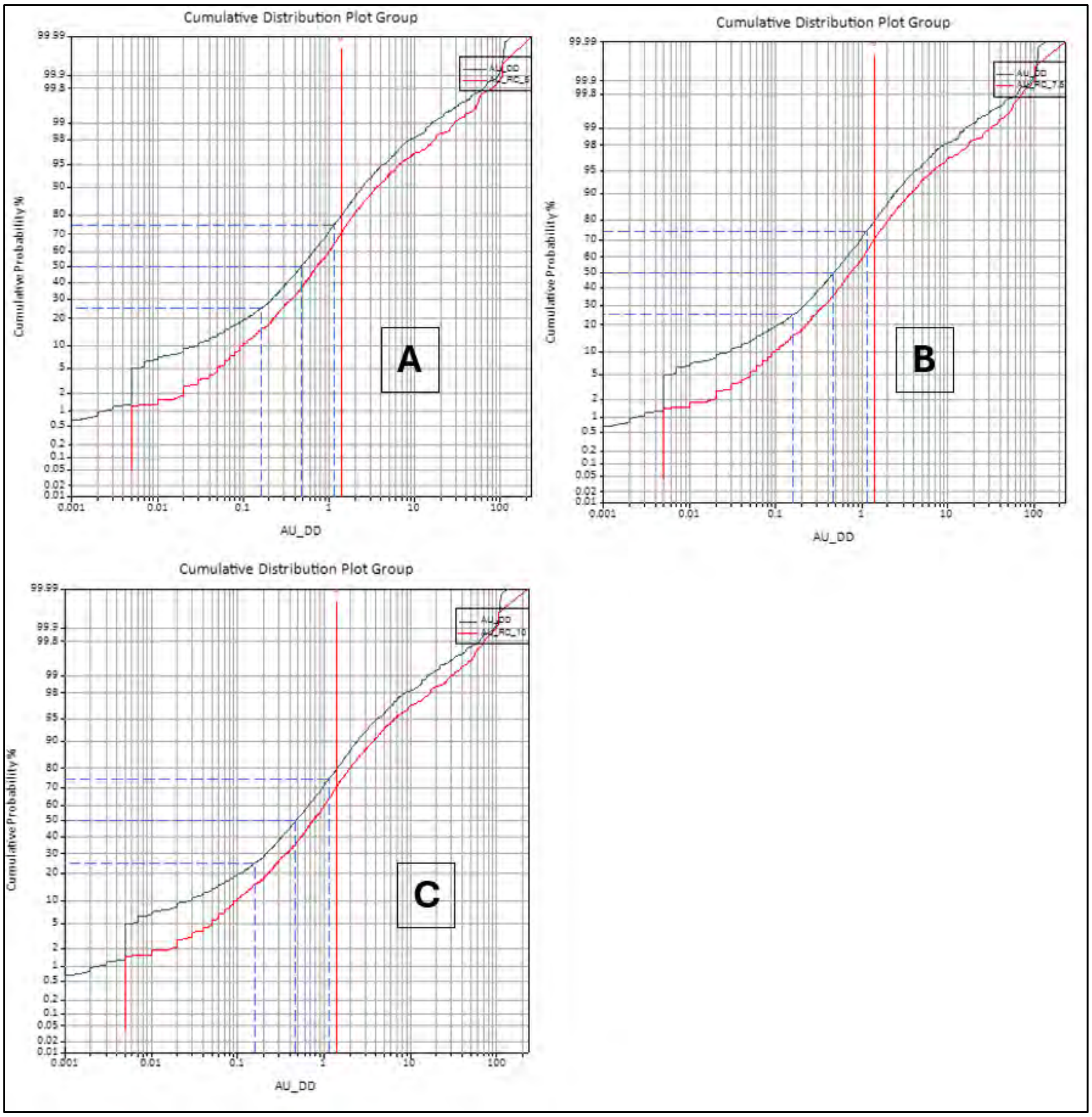


Figure 11.106 Log-Probability Plots Between Au from DD Holes and RC-Migrated Holes with Migration Distances of 5 m (a), 7.5 m (b), and 10 m (c)

To confirm the statistical analyses shown above, a visual analysis was completed on instances where there were RC-DD twins. As can be seen in Figure 11.107 and Figure 11.108 there are instances where the DD returns lower grades, especially on the hanging wall intercepts.

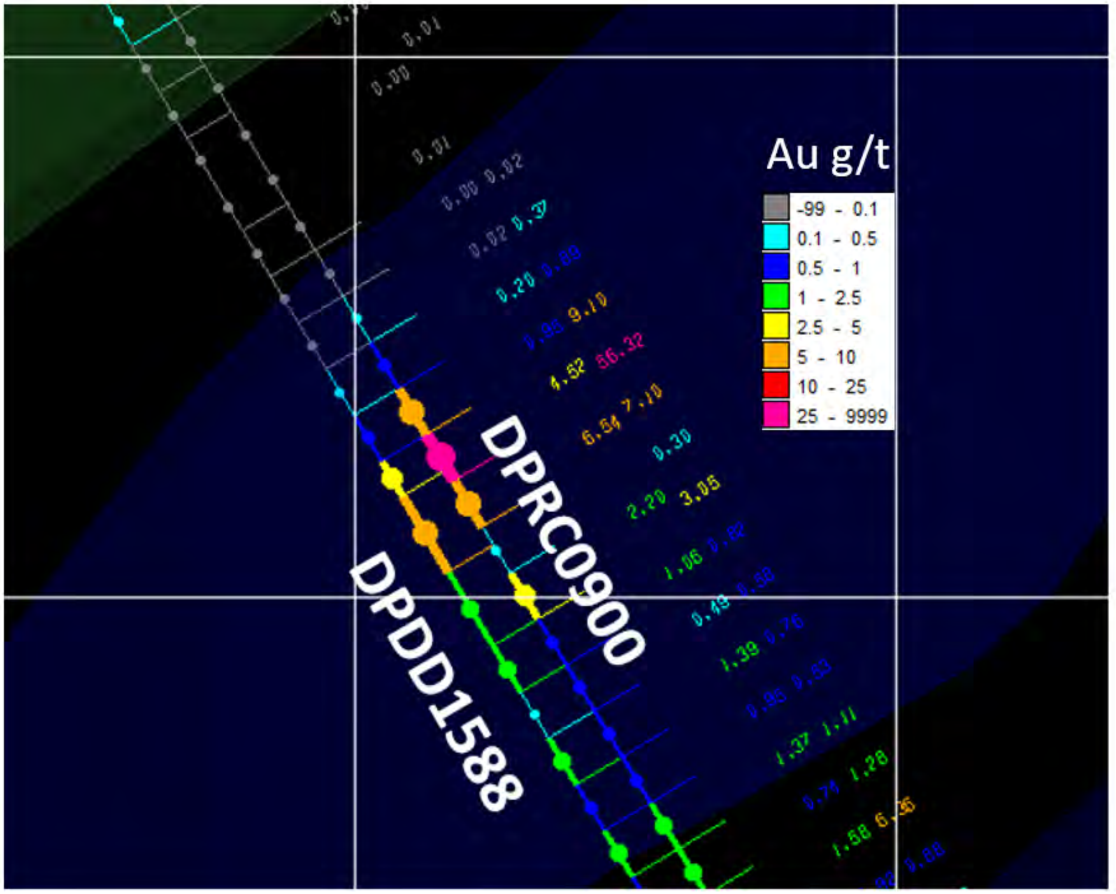


Figure 11.107 RC-DD Comparison - NOK

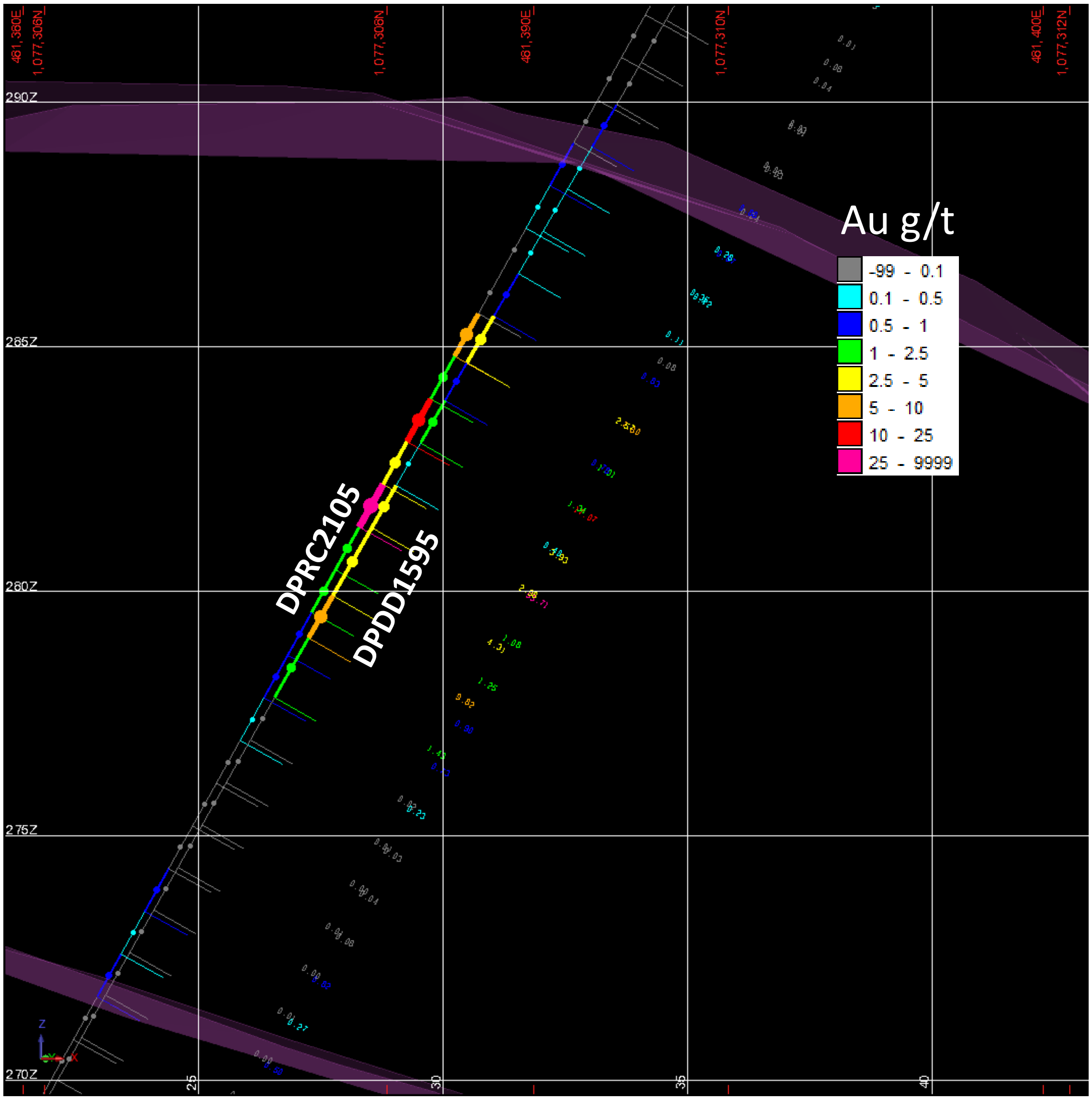


Figure 11.108 RC-DD Comparison - CHG

All these observations tend towards the same conclusion that the DD holes under-sample very-high grades - possibly due to a coarse gold component. This could be due to the primary sample size not being large enough to be representative of the upper tail of the gold grade distribution, whereas RC primary samples are much larger and therefore able to better represent the higher-grade mineralisation.

A visual review also highlighted potential smearing issues in the RC holes, although these appear to be relatively rare. Most of the twins show comparative length intervals of anomalous mineralisation. Examples of possible smearing are shown in Figure 11.109 and Figure 11.110. It is recommended that in future estimations, an emphasis is placed on identifying suspected instances of contamination and dealing with them accordingly.

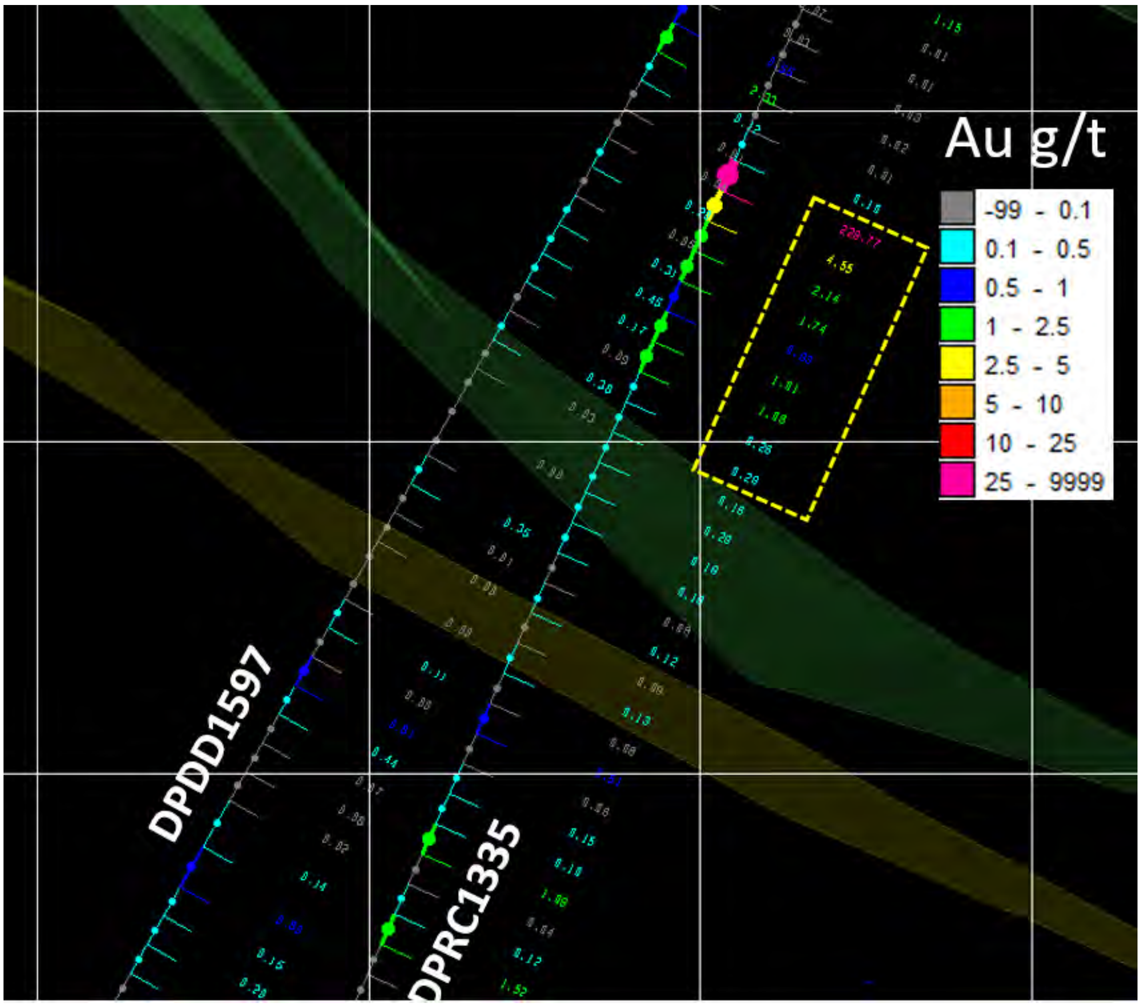


Figure 11.109 Suspected RC Contamination - CHG

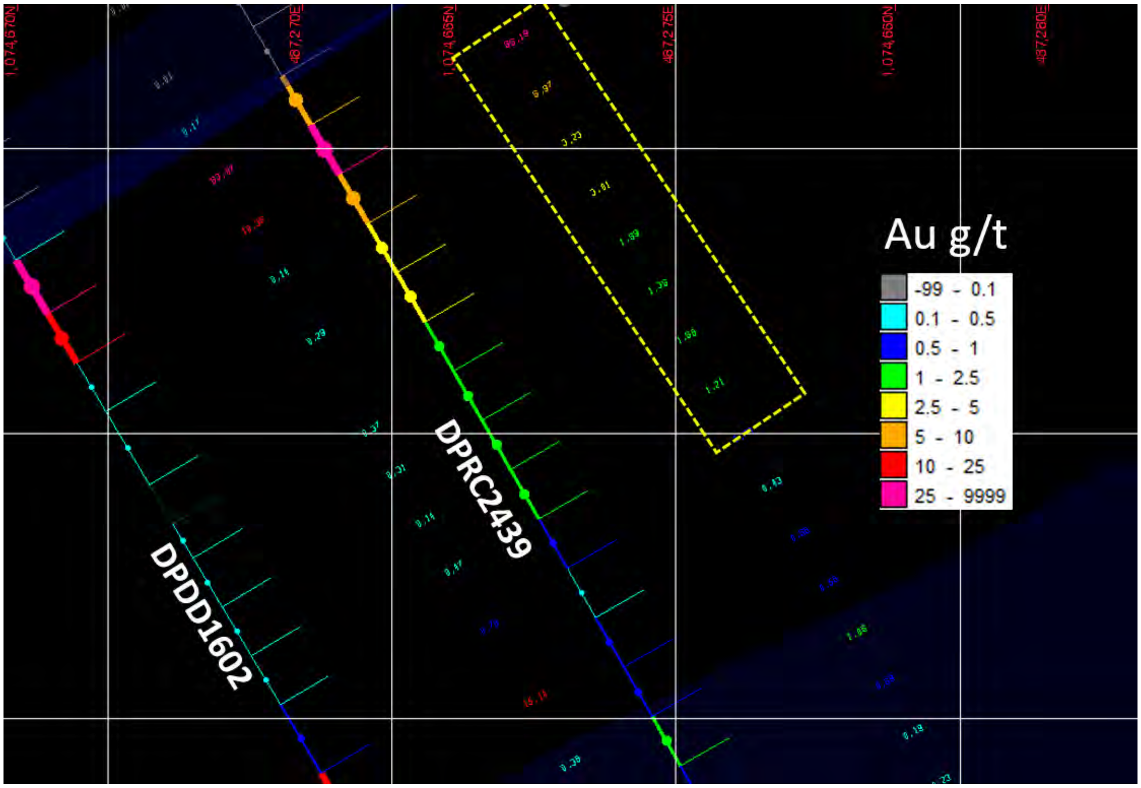


Figure 11.110 Suspected RC Contamination - HAN

11.7.4 Duplicate Performance Conclusions

The following points summarise the relevant observations for the overall duplicate performance for Doropo Gold Project.

- For RC field duplicates, the average CV% of all deposits combined of around 31% is considered to be within acceptable limits (i.e. less than 40%). When looking at the deposits individually, only ENI has an average CV% which is considered to be at the upper limits (38.8%) of what is acceptable, however the high average CV% is also reflected in the DD duplicates, indicating the mineralisation style at this deposit, is more 'nuggety' than the other deposits;
- RC field duplicate performance for all prospects to date shows acceptable correlation and precision; the average mean difference is -0.08% - duplicate is lower than original mean;
- In terms of the spread of assays, the percentage within 20% is deemed acceptable with results around 70%. ENI (see above) and SOL are the only deposits with a higher spread of assays;
- Diamond field duplicate performance is highly variable, but due to the nature of the duplicate sampling of DD core and also the instances of ¼ core duplicates, a definitive analysis is not possible. However, it does point to the nuggetty nature of most of the deposits;

- A limited dataset of pulp duplicates usually shows a difference in means of <5%, with the exception of KLG and NOK with 10% and 7.4% respectively. The average CV for all deposits was 26%, which is outside what is considered to be the acceptable limits for pulps, which is 10 - 20%. When comparing to the performance of the RC and DD field duplicates, it appears that there is a component of the nugget effect to the poor performance of the pulp duplicates, however there is a suspicion that poor homogenisation during the sample preparation and pulp selection could be a contributing factor;
- Paired statistics between DD and RC holes show relative difference of -17% - DD is lower than RC - at 5 m 7.5 m and 10 m migration distances. This could indicate that DD are an inadequate sample for this type of deposit in terms of primary sample size: the very high-grade portion of the distribution is probably poorly represented by the DD due to the presence of coarse gold that requires larger primary samples such as RC.

11.7.5 Summary Opinion of Qualified Person

Cube has independently assessed the QAQC data for the Project and the following summary and conclusions have been noted from the periods under review.

- CRM and Blanks Performance:
  - Overall blank performance is considered acceptable, with little evidence of persistent contamination observable in the results;
  - Coarse blank (CBLKY) performance is good with 97% of samples being within the pass limit. CRM OREAS 22h performance is good with 99% of samples being within the pass limit. Some potentially misallocated samples were observed. These should be further investigated, and their results corrected in the database;
- Duplicate Performance:
  - RC field duplicate performance for all prospects to date shows acceptable correlation and precision for these types of deposits;
  - Pulp duplicate performance for all Projects to date show acceptable correlation with the difference means usually <5%, however the precision is poor with an average CV of 26% which is above the acceptable <20% for orogenic gold deposits. This is likely the combination of the presence of coarse gold, poor homogenisation during the sample preparation and pulp selection;
  - Paired statistics between DD and RC holes show relative difference of -17% - DD is lower than RC - at 5 m, 7.5 m and 10 m migration distances. This could indicate that DD produces an inadequate sample for this type of deposit: the upper tail of the grade distribution appears to be poorly represented by DD, probably due to the presence of coarse gold that requires larger primary samples such as RC.

The QAQC data show that the sampling and analysis process has generally been robust, support the use of the RC and DD samples for Mineral Resource estimation. Although it appears the in some instances DD samples produce a lower grade, the impact is not considered material, as the DD holes only make up around 10% of the total Mineral Resource estimation dataset.

Based on an assessment of the available data, the Qualified Person considers the dataset collected by Centamin to be acceptable for Mineral Resource estimation, with QA protocols and QC results posing minimal risk to the overall confidence level of the Mineral Resource estimation for the period assessed. Analysis of QAQC results at prior Mineral Resource updates similarly showed that the data were acceptable for adoption in Mineral Resource estimation.

TABLE OF CONTENTS

12. DATA VERIFICATION 1
12.1 Site Visit 1
12.1.1 Drill Collar Checks 1
12.1.2 Independent Samples 2
12.2 Data Verification and Validation 4
12.3 Topography 5
12.4 Summary Opinion of Qualified Person 5

LIST OF FIGURES

Figure 12.1 Log Scatterplot of the Independent vs Original Sample Results from Hole DPRC3081 4

LIST OF TABLES

Table 12.1 Drill Collar Position, Azimuth and Inclination Checks 2
Table 12.2 Listing of Original versus Independent Sample Results from Hole DPRC3081 3

12. DATA VERIFICATION

12.1 Site Visit

A site visit to the Project was undertaken by Mr Michael Millad of Cube, accompanied by Centamin's Group Mineral Resource Manager, Mr Craig Barker. The site visit took place on the 29 and 30 August 2021, while the primary assay laboratory, BV Abidjan, was visited on 4 September 2021.

The following activities and inspections were undertaken:

- View selected drill cores and discuss geological framework and mineralisation controls. The core was compared to logs of lithology and to assay results contained in the drill database;
- Discuss and note data capture, storage and management processes;
- QAQC and sampling discussion - procedures and processes;
- Tour of facilities, including site of the under-construction Danoa sample preparation building;
- View outcrops and orpaillage workings in the prospecting area;
- Observe current RC drilling activities, including drilling methods, sample collection and sample security procedures;
- Independently check several drill collar positions and azimuth/inclinations;
- Observation and direction of the collection of independent drill samples;
- Inspection of the sample storage, sample preparation and sample analysis sections of the Bureau Veritas (Abidjan) laboratory.

The facilities and equipment were considered fit for purpose, and the procedures were well-designed and being implemented consistently. The sample preparation and analytical laboratories were well equipped and operated to a high standard, with the only concerning issue noted being the over-crowded sample storage area, which was explained to be due to late collection of sample residue by various mining companies. A few minor issues were noted within the sample preparation area, but the overall impression was of a well-run and sufficiently clean environment.

12.1.1 Drill Collar Checks

A selection of drill collar positions, azimuths and inclinations at Doropo were checked using an Apple iPhone 12, with relevant applications downloaded from the Apple App Store. The results were compared to the surveyed values in the drill database (Table 12.1).

Hole ID	Area	Cube Site Check				Centamin/Ampella Database				Distance Difference (m)
		UTM X (m)	UTM Y (m)	Azimuth	Dip	UTM X (m)	UTM Y (m)	Azimuth	Dip	
DPRC0437	HAN	486,818.0	1,074,199.7	125	61	486,825.8	1,074,195.3	129.8	59.8	9.0
DPRC0039	SWA	478,982.8	1,075,966.2	143	63	478,981.8	1,075,965.5	140.9	60.0	1.2
DPRC0843	SWA	478,688.2	1,074,547.7	91	59	478,680.9	1,074,549.2	87.9	60.7	7.5
DPRC0837	SWA	478,604.5	1,074,702.5	97	53	478,608.3	1,074,699.2	89.2	59.8	5.1
DPRC0836	SWA	478,661.4	1,074,701.7	92	57	478,661.4	1,074,699.1	91.5	59.9	2.6
DPDD1443	NOK	479,896.4	1,077,874.9	150	65	479,894.4	1,077,873.7	148.7	59.4	2.3
DPRC2261	NOK	479,904.6	1,077,883.8	143	57	479,907.3	1,077,881.4	152.1	58.0	3.6
DPRC2120	CHG	481,746.6	1,078,683.9	146	58	481,748.3	1,078,677.4	151.9	60.5	6.7
DPRC1451	CHG	482,082.9	1,078,784.9	150	58	482,085.0	1,078,787.0	150.6	58.1	3.0

Table 12.1 Drill Collar Position, Azimuth and Inclination Checks

The check collar positions were found to be within 10 m in all instances and within 5 m for five out of the nine collars checked. The check azimuths and inclinations were also within a reasonable margin of the database values, given that the cemented PVC piping used to do the check readings may have shifted slightly since they were set. The database values are considered to be more accurate, given they were measured using precision survey methods and because the hole azimuths and inclinations were set at the time of drilling.

12.1.2 Independent Samples

The Qualified Person, Mr Michael Millad, directly observed the collection of a set of independent samples in hole DPRC3081, which was targeting the Souwa mineralised zone close to where it outcrops, hence the mineralisation intercept was relatively shallow. The independent samples were collected on 30 August 2021. Some 22 duplicate samples were collected and sealed in polyweave bags for delivery to BV Abidjan for preparation and then onto ALS Loughrea in Ireland for analysis. The multi-purpose Geodrill UDR900 rig (can drill RC and DD [PQ, HQ & NQ] holes) was used to collect the RC samples, which were collected from the cyclone before being passed through a three-tier riffle splitter to produce a 2 to 3 kg sample.

The results are summarised in Table 12.2 below and a scatter plot is shown in Figure 12.1. The agreement between the original and independent samples is acceptable.

Samp Id Independent	Samp ID Original	Depth From (m)	Depth To (m)	Au ppm Independent	Au ppm Original
CIS384973	CIS671836	4	5	0.18	0.22
CIS384974	CIS671837	5	6	1.76	1.26
CIS384953	CIS671838	6	7	0.5	0.56
CIS384954	CIS671839	7	8	0.361	0.37
CIS384955	CIS671840	8	9	0.254	0.25
CIS384956	CIS671841	9	10	0.104	0.09
CIS384957	CIS671842	10	11	0.01	0.005
CIS384958	CIS671843	11	12	0.009	0.005
CIS384959	CIS671844	12	13	0.008	0.005
CIS384960	CIS671846	13	14	0.01	0.005
CIS384961	CIS671847	14	15	0.005	0.005
CIS384962	CIS671848	15	16	0.005	0.005
CIS384963	CIS671849	16	17	0.007	0.005
CIS384964	CIS671851	17	18	0.008	0.005
CIS384965	CIS671852	18	19	0.038	0.02
CIS384966	CIS671853	19	20	0.006	0.005
CIS384967	CIS671854	20	21	0.057	0.03
CIS384968	CIS671856	21	22	0.014	0.005
CIS384969	CIS671857	22	23	0.005	0.04
CIS384970	CIS671858	23	24	0.023	0.03
CIS384971	CIS671859	24	25	0.005	0.005
CIS384972	CIS671860	25	26	0.001	0.005

Note: *Mineralised zone highlighted.*

Table 12.2 Listing of Original versus Independent Sample Results from Hole DPRC3081

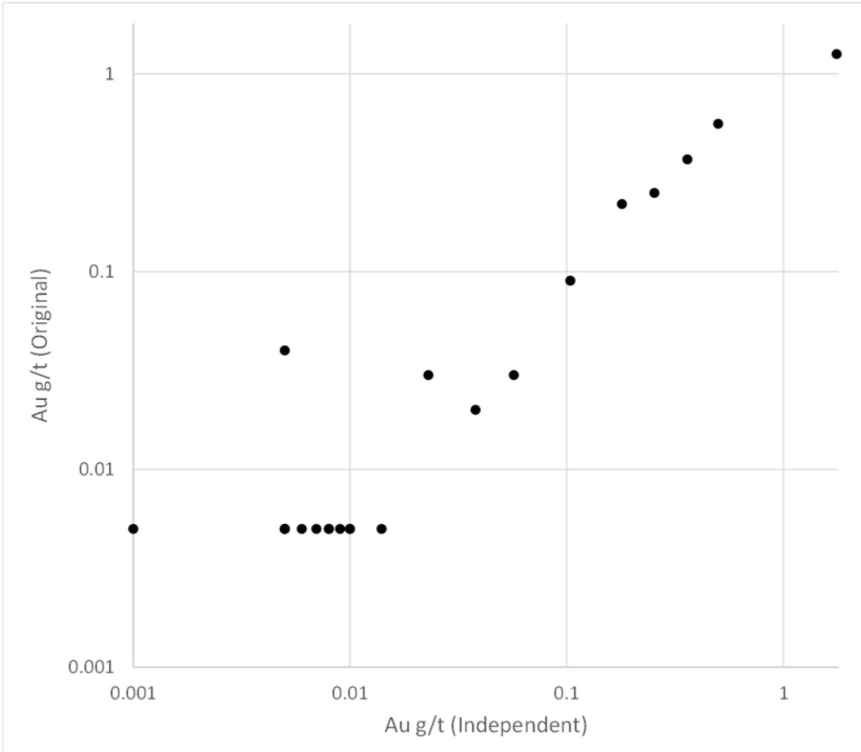


Figure 12.1 Log Scatterplot of the Independent vs Original Sample Results from Hole DPRC3081

12.2 Data Verification and Validation

The exploration database has been maintained on site in acQuire since the beginning of the project. Field data is collected on paper and transcribed to excel spreadsheets by field geologists and a dedicated data entry person. Spreadsheets are then imported to acQuire by a dedicated database manager.

Data is internally validated by acQuire as it is entered and ensures:

- Collar, survey, assay and geology end of hole depths are compatible;
- No overlapping intervals are allowed;
- No repeat sample identification numbers can occur within the database;
- Laboratory assays are loaded to the correct sample identification number;
- All analytical results are stored in the database as reported by the laboratory. Assay values below detection are converted to half detection limit for reporting and modelling purposes;
- All logged codes adhere to the accepted libraries.

The Qualified Person undertook spot checks of assay values in the database against laboratory certificates while on the site visit and no errors were found.

Database validation checks completed by Cube include the following work:

- Checking for absent collars;
- Flagging of collars with no survey and/or assay data;
- Sample data exceeding the recorded depth of hole;
- Checking for out-of-range assay or density values;
- Checking for sample interval overlaps;
- Reporting missing assay intervals;
- Visual validation of downhole survey data by inspection of drill traces;
- Visual validation of geological models against the drill logs.

No major issues were detected, and the files were considered suitable for use in the Mineral Resource estimation.

12.3 Topography

The topographic models supplied to Cube by Centamin for use in the Mineral Resource estimation are based on the surveyed drill collars over each deposit. Cube has visually checked the topography DTM against the drill collars and observed a high degree of spatial correspondence between the two datasets. The relief is subdued in the Project area, and so the use of the drill collars to generate the topographic surface models is not considered to pose a significant risk. It is recommended though that a higher resolution survey is undertaken to inform more advanced mining studies.

12.4 Summary Opinion of Qualified Person

Based on an assessment of the data verification procedures undertaken by Centamin, and the Qualified Person's own verification and validation checks, it is the Qualified Person's opinion that the input data are of the required quality and integrity for Mineral Resource estimation.

TABLE OF CONTENTS

13. MINERAL PROCESSING AND METALLURGICAL TESTING 1
13.1 Metallurgy and Mineral Processing Test Work 1
13.1.1 Mineralogy 1
13.1.2 Comminution 2
13.1.3 Cyanidation 2
13.1.4 Thickening and Cyanide Detoxification 4
13.1.5 Metallurgical Recovery 4
13.1.6 Further Metallurgical Testing 5
13.2 Feasibility Test Work 6
13.3 Comminution Test Work 7
13.3.1 Comminution Overview 8
13.3.2 PFS Study Comminution Sample Selection 8
13.3.3 SMC Tests 11
13.3.4 Comminution Tests 13
13.3.5 Results Interpretation 16
13.4 Sample Provenance 16
13.4.1 Feasibility Study Samples 16
13.4.2 Sample Spatial Distribution Indication 34
13.5 Head Assay Analysis 40
13.5.1 Master Composite Analysis 40
13.5.2 Variability Composite Analysis 44
13.6 Mineralogy 53
13.6.1 Master Composite Mineralogy 53
13.7 Grind and Cyanide Optimization Test Work 60
13.7.1 Grind Optimization Test Work 60
13.8 Oxygen Versus Air Direct Cyanidation Leach Tests 74
13.8.1 Oxygen Versus Air Leach Results 75
13.9 Cyanide Optimization Test Work 83
13.9.1 Cyanide Optimization Test Work 83
13.10 Pre-Oxidation Shear Reactor Test Work (Mach©) 95
13.11 Optimised CIL and Pre-Oxidation CIL Test Work 100
13.11.1 Souwa Master Composites - MC-1, MC-2 and MC-3 101
13.11.2 Nokpa Master Composites - MC-5 and MC-6 102
13.11.3 Kilosegui Master Composites - MC-8 and MC-9 102
13.11.4 Kekeda Master Composites - MC-11 and MC-12 103

13.11.5 Enioda Master Composites - MC-13, MC-14 and MC-15 ... 104
13.11.6 Han Master Composites - MC-17 & MC-18 ... 105
13.11.7 Cheque Main Master Composites - MC-20 and MC-21 ... 106
13.11.8 Chegue South Master Composites - MC23 and MC24 ... 107
13.12 Thickener Test Work ... 108
13.13 Cyanide Detox Test Work ... 109
13.14 Gold Recovery - Regression Analysis ... 112
13.14.1 Souwa Recovery Regression ... 113
13.14.2 Nokpa Recovery Regression ... 115
13.14.3 Kilosegui Recovery Regression ... 116
13.14.4 Kekeda Recovery Regression ... 117
13.14.5 Enioda Recovery Regression ... 118
13.14.6 Han Recovery Regression ... 120
13.14.7 Chegue Main Recovery Regression ... 121
13.14.8 Chegue South Recovery Regression ... 122

LIST OF FIGURES

Figure 13.1 Ore Comminution Characteristics ... 15
Figure 13.2 Provenance of Lithology Composites from the Souwa Open Pit ... 34
Figure 13.3 Provenance of Lithology Composites from the Nokpa Open Pit ... 35
Figure 13.4 Provenance of Lithology Composites from the Kilosegui Open Pit (NW Section) ... 35
Figure 13.5 Provenance of Lithology Composites from the Kilosegui Open Pit (Central Section) ... 36
Figure 13.6 Provenance of Lithology Composites from the Kilosegui Open Pit (SE Section) ... 36
Figure 13.7 Provenance of Lithology Composites from the Kekeda Open Pit ... 37
Figure 13.8 Provenance of Lithology Composites from the Enioda Open Pit ... 37
Figure 13.9 Provenance of Lithology Composites from the Han Open Pit ... 38
Figure 13.10 Provenance of Lithology Composites from the Chegue Main ... 38
Figure 13.11 Provenance of Lithology Composites from the Chegue Main (North) ... 39
Figure 13.12 Provenance of Lithology Composites from the Chegue South ... 39
Figure 13.13 Master Composite - Model Mineralogy ... 55
Figure 13.14 Souwa Grind Optimization Leach Kinetic Curves ... 62
Figure 13.15 Nokpa Grind Optimization Leach Kinetic Curves ... 64
Figure 13.16 Kilosegui Grind Optimization Leach Kinetic Curves ... 66
Figure 13.17 Kekeda Grind Optimization Leach Kinetic Curves ... 67
Figure 13.18 Enioda Grind Optimization Kinetic Leach Curves ... 69
Figure 13.19 Han Grind Optimization Leach Kinetic Curves ... 70

Figure 13.20 Chegue Main Grind Optimization Leach Kinetic Curves 72
Figure 13.21 Chegue South Grind Optimization Leach Kinetic Curves 74
Figure 13.22 Souwa Cyanide Optimization Leach Kinetic Curves 84
Figure 13.23 Nokpa Cyanide Optimization Leach Kinetic Curves 86
Figure 13.24 Kilosegui Cyanide Optimization Leach Kinetic Curves 87
Figure 13.25 Kekeda Cyanide Optimization Kinetic Leach Curves 89
Figure 13.26 Enioda Cyanide Optimization Kinetic Leach Curves 90
Figure 13.27 Han Cyanide Optimization Kinetic Leach Curves 92
Figure 13.28 Chegue Main Cyanide Optimization Kinetic Leach Curves 93
Figure 13.29 Chegue South Cyanide Optimization Kinetic Leach Curves 95
Figure 13.30 Souwa Regression Curve 114
Figure 13.31 Nokpa Regression Curve 115
Figure 13.32 Kilosegui Regression Curve 117
Figure 13.33 Kekeda Regression Curve 118
Figure 13.34 Enioda Regression Curve 119
Figure 13.35 Chegue Main Regression Curve 120
Figure 13.36 Chegue Main Regression Curve 121
Figure 13.37 Chegue South Regression Curve 122

LIST OF TABLES

Table 13.1 Gold Grade Versus Recovery (Regression Analysis) 5
Table 13.2 Comminution Test Work Samples 10
Table 13.3 SMC Tests 11
Table 13.4 SMC Parameters 12
Table 13.5 Comminution Test Work Results Summary 15
Table 13.6 Feasibility Study Sample Summary 17
Table 13.7 Feasibility Study Composite Sample Weights 17
Table 13.8 Souwa Oxide Master Composite Details (MC1) 17
Table 13.9 Souwa Transitional Master Composite Details (MC2) 18
Table 13.10 Souwa Oxide Master Composite Details (MC3) 18
Table 13.11 Souwa Oxide Low Grade Variability Composite Details (VC1) 18
Table 13.12 Souwa Oxide High Grade Variability Composite Details (VC2) 19
Table 13.13 Souwa Transitional Low Grade Variability Composite Details (VC3) 19
Table 13.14 Souwa Transitional High Grade Variability Composite Details (VC4) 19
Table 13.15 Souwa Fresh Low Grade Variability Composite Details (VC5) 19
Table 13.16 Souwa Fresh High Grade Variability Composite Details (VC6) 19

Table 13.17 Nokpa Oxide Master Composite Details (MC4) 20
Table 13.18 Nokpa Transitional Master Composite Details (MC5) 20
Table 13.19 Nokpa Fresh Master Composite Details (MC6) 20
Table 13.20 Nokpa Oxide Low Grade Variability Composite Details (VC7) 21
Table 13.21 Nokpa Transitional Low Grade Variability Composite Details (VC8) 21
Table 13.22 Nokpa Transitional High Grade Variability Composite Details (VC9) 21
Table 13.23 Nokpa Fresh Low Grade Variability Composite Details (VC10) 21
Table 13.24 Nokpa Fresh High Grade Variability Composite Details (VC11) 21
Table 13.25 Kilosegui Oxide Master Composite Details (MC7) 22
Table 13.26 Kilosegui Transitional Master Composite Details (MC8) 22
Table 13.27 Kilosegui Fresh Master Composite Details (MC9) 23
Table 13.28 Kilosegui Transitional Low Grade Variability Composite Details (VC12) 23
Table 13.29 Kilosegui Transitional High Grade Variability Composite Details (VC13) 23
Table 13.30 Kilosegui Fresh Low Grade Variability Composite Details (VC14) 23
Table 13.31 Kilosegui Fresh High Grade Variability Composite Details (VC15) 24
Table 13.32 Kekeda Oxide Master Composite Details (MC10) 24
Table 13.33 Kekeda Transitional Master Composite Details (MC11) 24
Table 13.34 Kekeda Fresh Master Composite Details (MC12) 25
Table 13.35 Kekeda Oxide Low Grade Variability Composite Details (VC16) 25
Table 13.36 Kekeda Oxide High Grade Variability Composite Details (VC17) 25
Table 13.37 Kekeda Transitional Low Grade Variability Composite Details (VC18) 25
Table 13.38 Kekeda Transitional High Grade Variability Composite Details (VC19) 26
Table 13.39 Kekeda Fresh Low Grade Variability Composite Details (VC20) 26
Table 13.40 Kekeda Fresh High Grade Variability Composite Details (VC21) 26
Table 13.41 Enioda Oxide Master Composite Details (MC13) 27
Table 13.42 Enioda Transitional Master Composite Details (MC14) 27
Table 13.43 Enioda Fresh Master Composite Details (MC15) 27
Table 13.44 Enioda Oxide Low Grade Variability Composite Details (VC22) 27
Table 13.45 Enioda Oxide High Grade Variability Composite Details (VC23) 28
Table 13.46 Enioda Transitional Low Grade Variability Composite Details (VC24) 28
Table 13.47 Enioda Transitional High Grade Variability Composite Details (VC25) 28
Table 13.48 Enioda Fresh Low Grade Variability Composite Details (VC26) 28
Table 13.49 Enioda Fresh High Grade Variability Composite (VC27) 28
Table 13.50 Han Oxide Master Composite Details (MC16) 29
Table 13.51 Han Transitional Master Composite Details (MC17) 29
Table 13.52 Han Fresh Master Composite Details (MC18) 29
Table 13.53 Han Transitional Low Grade Variability Composite Details (VC28) 29
Table 13.54 Han Transitional High Grade Variability Composite Details (VC29) 30

Table 13.55 Han Fresh Low Grade Variability Composite Details (VC30) ... 30
Table 13.56 Han Fresh High Grade Variability Composite Details (VC31) ... 30
Table 13.57 Chegue Main Oxide Master Composite Details (MC19) ... 30
Table 13.58 Chegue Main Traditional Master Composite Details (MC20) ... 31
Table 13.59 Chegue Main Fresh Master Composite Details (MC21) ... 31
Table 13.60 Chegue Main Oxide Low Grade Variability Composite Details (VC32) ... 31
Table 13.61 Chegue Main Oxide High Grade Variability Composite Details (VC33) ... 31
Table 13.62 Chegue Main Transitional Low Grade Composite Details (VC34) ... 32
Table 13.63 Chegue Main Transitional High Grade Variability Composite Details (VC35) ... 32
Table 13.64 Chegue Main Fresh Low Grade Variability Composite Details (VC36) ... 32
Table 13.65 Chegue Main Fresh High Grade Variability Composite Details (VC37) ... 32
Table 13.66 Chegue South Oxide Master Composite Details (MC22) ... 33
Table 13.67 Chegue South Transitional Master Composite Details (MC23) ... 33
Table 13.68 Chegue South Fresh Master Composite Details (MC24) ... 33
Table 13.69 Chegue South Transitional Low Grade Variability Composite Details (VC38) ... 33
Table 13.70 Chegue South Transitional High Grade Variability Composite Details (VC39) ... 33
Table 13.71 Chegue South Fresh Low Grade Variability Composite Details (VC40) ... 34
Table 13.72 Chegue South Fresh High Grade Variability Composite Details (VC41) ... 34
Table 13.73 Head Assay Analysis MC1 - MC6 ... 41
Table 13.74 Head Assay Analysis MC7 - MC12 ... 42
Table 13.75 Head Assay Analysis MC13 - MC18 ... 43
Table 13.76 Head Assay Analysis MC19 - MC24 ... 44
Table 13.77 Souwa Variability Head Assay Analysis VC1 - VC6 ... 46
Table 13.78 Nokpa Variability Head Assay Analysis VC7 - VC11 ... 47
Table 13.79 Kilosegui Variability Head Assay Analysis VC12 - VC15 ... 48
Table 13.80 Kekeda Variability Head Assay Analysis VC17 - VC21 ... 49
Table 13.81 Enioda Variability Head Assay Analysis VC22 - VC27 ... 50
Table 13.82 Han Variability Head Assay Analysis VC28 - VC31 ... 51
Table 13.83 Chegue Main Variability Head Assay Analysis VC32 - VC37 ... 52
Table 13.84 Chegue South Variability Head Assay Analysis VC38 - VC41 ... 53
Table 13.85 Master Composites Souwa & Nokpa - Model Mineralogy ... 55
Table 13.86 Master Composites Kilosegui & Kekeda - Model Mineralogy ... 56
Table 13.87 Master Composites Enioda & Han - Model Mineralogy ... 57
Table 13.88 Master Composites Chegue Main & Chegue South - Model Mineralogy ... 58
Table 13.89 Master Composites Souwa & Nokpa - Sulphur Deportment ... 59
Table 13.90 Master Composites Kiloegui & Kekeda - Sulphur Deportment ... 59
Table 13.91 Master Composites Enioda & Han - Sulphur Deportment ... 60
Table 13.92 Master Composites Chegue Main & Chegue South - Sulphur Deportment ... 60

Table 13.93 Souwa Grind Optimization Leach Results....62
Table 13.94 Nokpa Grind Optimization Leach Results....64
Table 13.95 Kilosegui Grind Optimization Leach Results....65
Table 13.96 Kekeda Grind Optimization Leach Results....67
Table 13.97 Enioda Grind Optimization Leach Results....68
Table 13.98 Han Grind Optimization Leach Results....70
Table 13.99 Chegue Main Grind Optimization Leach Results....71
Table 13.100 Chegue South Grind Optimization Leach Results....73
Table 13.101 Souwa Master Composites - Oxygen Versus Air Leach Results....75
Table 13.102 Nokpa Master Composites - Oxygen Versus Air Leach Results....76
Table 13.103 Kilosegui Master Composites - Oxygen Versus Air Leach Results....77
Table 13.104 Kekeda Master Composites - Oxygen Versus Air Leach Results....78
Table 13.105 Enioda Master Composites - Oxygen Versus Air Leach Results....79
Table 13.106 Han Master Composites - Oxygen Versus Air Leach Results....80
Table 13.107 Chegue Main Master Composites - Oxygen Versus Air Leach Results....81
Table 13.108 Chegue South Master Composites - Oxygen Versus Air Leach Results....82
Table 13.109 Souwa Cyanide Optimization Leach Results....84
Table 13.110 Nokpa Cyanide Optimization Leach Results....85
Table 13.111 Kilosegui Cyanide Optimization Leach Results....87
Table 13.112 Kekeda Cyanide Optimization Leach Results....88
Table 13.113 Enioda Cyanide Optimization Leach Results....90
Table 13.114 Han Cyanide Optimization Leach Results....91
Table 13.115 Chegue Main Cyanide Optimization Leach Results....93
Table 13.116 Chegue South Cyanide Optimization Leach Results....94
Table 13.117 Souwa Fresh Master Composite - Mach Reactor Leach Results....96
Table 13.118 Nokpa Fresh Master Composite - Mach Reactor Leach Results....98
Table 13.119 Kilosegui Fresh Master Composite - Mach Reactor Leach Results....99
Table 13.120 Kekeda Fresh Master Composite - Mach Reactor Leach Results....100
Table 13.121 Souwa Composite Pre-Oxidation Leach Results....101
Table 13.122 Nokpa Composites Pre-Oxidation Leach Results....102
Table 13.123 Kilosegui Composites Pre-Oxidation Leach Results....103
Table 13.124 Kekeda Composites Pre-Oxidation Leach Results....104
Table 13.125 Enioda Composites Pre-Oxidation Leach Results....105
Table 13.126 Han Composites Pre-Oxidation Leach Results....106
Table 13.127 Chegue Main Composites Pre-Oxidation Leach Results....107
Table 13.128 Chegue South Pre-Oxidation Leach Results....108
Table 13.129 Thickener Test Work Results....109
Table 13.130 Cyanide Detox Test Work Conditions....109

Table 13.131 Souwa Oxide Continuous Cyanide Detoxification Results 110
Table 13.132 Souwa Oxide Bulk Cyanide Detoxification Results 110
Table 13.133 Souwa Trans Continuous Cyanide Detoxification Results 110
Table 13.134 Souwa Trans Bulk Cyanide Detoxification Results 110
Table 13.135 Souwa Fresh Continuous Cyanide Detoxification Results 111
Table 13.136 Souwa Fresh Bulk Cyanide Detoxification Results 111
Table 13.137 Nokpa Trans Continuous Cyanide Detoxification Results 111
Table 13.138 Nokpa Trans Bulk Cyanide Detoxification Results 111
Table 13.139 Nokpa Fresh Continuous Cyanide Detoxification Results 112
Table 13.140 Nokpa Fresh Bulk Cyanide Detoxification Results 112
Table 13.141 Leach Recovery Regression Format 113
Table 13.142 Leach Recovery Regression Co-efficients 113

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical test work continued to focus on the flowsheet presented in Centamin's Doropo Gold Project Prefeasibility Study, National Instrument 43-101 Technical Report completed in August 2023 by Lycopodium (Report# 2234-GREP-001).

The Feasibility Study (FS) is based on developing eight open pit mines which will be processed in a conventional Carbon-in-Leach (CIL) circuit. The FS test work was carried out by ALS Metallurgy in Perth, Western Australia between August 2023 and June 2024. The test work programme was managed by IMO.

The test work objectives were to:

- Establish and demonstrate the optimum process operating parameters for each lithology type in each orebody;
- Conduct leach tests on Master composites from each lithology in each orebody to determine the optimum conditions and expected gold recoveries;
- Conduct leach tests on Variability composites at optimised conditions to determine gold recoveries at varying head grades for the three lithology types in each orebody; and
- Determine key process design criteria (PDC) for the Engineer to allow selection and sizing of mechanical equipment.

Key findings from the test work are summarised in the following sub-sections.

13.1 Metallurgy and Mineral Processing Test Work

13.1.1 Mineralogy

No native gold particles were detected, however gravity separation test work has confirmed the presence of gravity recoverable gold in all Master composites.

Pyrite was detected reporting between 0.9% and 1.63% by mass in the Fresh samples and measured less than 0.5% in the 'Transition' samples, and less than 0.1% in 'Oxide' samples.

The Nokpa & Han Oxide composites and Chegue Main and Chegue South Fresh composites reported elevated organic carbon levels at 0.42%, 0.78%, 0.32% and 0.24% respectively. All other composites reported low levels (<0.03% C_{org}). The high levels of organic carbon reported, upon testing, however, did not exhibit preg-robbing behaviour in the leach kinetic response.

Deleterious elements mercury, arsenic and antimony are low in the composites and should not present an environmental or occupational health risk in the elution or electrowinning circuit.

Relatively high carbonate content was detected in the eight 'Fresh' samples. Calcite was observed in all the 'Fresh' composites and of the Transition samples Enioda was the only composite containing a significant proportion of carbonate, largely as 'ankerite-dolomite'.

13.1.2 *Comminution*

The fresh ore has medium competency, moderate to high grinding energy requirements and is moderately abrasive. Transition ore has moderate competency with average grinding energy requirements and is moderately abrasive. The oxide ore has low to moderate competency with average to high grinding energy requirements and is low to moderately abrasive.

Crushing Work Indices (BCWi) results ranged from 1.6 to 15.8 kWh/t, averaging 8.3 kWh/t over the complete set of samples. The BCWi are low to medium, indicating a low crushing energy requirement. The maximum BCWi values, ranging from 4.1 to 15.8 kWh/t were for the Kekeda ores, with an 85th percentile value of 14.9 kWh/t.

Rod Mill Work Indices (BRWi) results ranged from 7.2 to 21.4 kWh/t. A BRWi of 18.5 kWh/t for fresh and 14.9 kWh/t for oxide were selected for design.

The Bond Ball Mill Indices (BBWi) results ranged from 11 to 22.2 kWh/t. The BBWi are medium to high indicating a high grinding energy requirement. Selected 85th percentile data for design were 20.4 kWh/t for oxide and 18.5 kWh/t for fresh.

The abrasion indices ranged from very low at 0.018 to medium at 0.38. The oxide samples demonstrated the highest abrasion indices, averaging 0.248. Selected 85th percentile data for design were 0.248 for oxide and 0.20 for fresh ores.

13.1.3 *Cyanidation*

The grind versus recovery leach tests have confirmed the optimum target grind sizes to be consistent with those selected in the PFS test work programme. P_{80} 106 µm for Oxide and Transitional ores and P_{80} 75 µm for Fresh ores.

Gold recoveries for the FS test work were largely consistent compared to the PFS test work. The three lithology types reported the following average gold recoveries across the eight (8) open pits: Oxide - 96.2%, Transitional - 93.0% and Fresh - 87.9%.

The gravity recoverable gold component averaged at 31% across all open pits and was consistent with levels reported in the PFS test programme.

The results of the FS Carbon in Leach (CIL) leach tests were consistent with the PFS tests and confirmed the Fresh ores require an extended leach residence time of 12 hours over and above the 24 hour leach residence time required by the Oxide and Transitional ores to maximize gold recovery.

Cyanide consumption rates reported in the FS CIL tests averaged at 0.57 kg/t which were significantly higher than reported in the PFS direct cyanide (no carbon) leach tests at 0.12 kg/t. Higher cyanide consumption rates are generally reported when carbon is used in the leach test.

Lime consumption in the FS CIL tests were reported in similar ranges to the PFS direct cyanide leaches. In both test programmes the lime requirements were highest in the oxide ores (~1.5 kg/t) followed by Transitional (~0.60 kg/t) and lowest in the Fresh ores (~0.30 kg/t).

Cyanide addition rates were optimized at 0.05% NaCN.

Oxygen demand has been observed in the low range.

Oxygen sparging reported an increase in gold recoveries between 3 to 10% when processing Fresh ores compared to air only sparging. Significant improvements were also reported using oxygen in the leaching of Transitional ores at between 2 to 4% and to a lesser extent the Oxide ores.

The addition of Lead Nitrate at 500 g/t proved to have little effect on gold recovery.

A six (6) hour pre-oxidation (no cyanide) step identified in the PFS report as a potential benefit to overall recovery proved to have little benefit with the FS conventional CIL test work. This was further demonstrated during the MACH shear reactor tests which reported marginal and, in some cases inconsistent results.

Further tests were underway at the time of writing this report to assess the effect of a pre-leach step in advance of conventional CIL circuit on gold recovery.

Gold adsorption kinetics are noted to be in the medium range based on an average Fleming (K) value of 160 hour^{-1} relative to industry typical values in the range 150 hour^{-1} to 250 hour^{-1}, and intercept 'n' at an average of 0.70.

Elevated levels of Tellurium were observed in some of the composites and were not confined to any particular lithology type. The leach test results have confirmed the presence of Tellurides has no measurable impact on gold recovery.

Deleterious elements mercury, arsenic and antimony were low in the composites and should not present an environmental or occupational health risk in the elution or electrowinning circuits.

13.1.4 Thickening and Cyanide Detoxification

Dynamic thickening test work was conducted on the lithology Master composite samples from Souwa and Nokpa deposits. Solids loadings were reported in the 0.75 to 1.0 t/m^2/h range indicating a thickener diameter of between 30-35 m will be required. Underflow slurry densities at or above 55% solids were achieved, with the exception of the Nokpa Transitional sample reporting at 48% solids which also required 300 g/t of coagulant to meet the <100 mg/L total suspended solids (TSS overflow) clarity guidelines.

The Souwa Oxide Master composite dynamic thickener test achieved an underflow slurry density of 55% solids, however required high flocculant dose rates at 120 g/t and reported poor overflow clarity in excess of 1,000 mg/L.

Cyanide detoxification tests were conducted on the bulk Souwa thickener test samples and reported residence times in the range 80-95 mins from the three lithology types. Tests conducted on the Nokpa Transitional and Fresh composites however, reported contrasting residence time requirements at 50 mins and 150 mins respectively.

13.1.5 Metallurgical Recovery

Samples reported gold head grades via fire assay in a range from 0.74 g/t to 3.70 g/t Au and averaged 1.63 g/t Au. The duplicate gold assays for several composites varied significantly which is consistent with the PFS Master composites supporting the presence of coarse gold.

Gold recoveries were largely consistent compared to the PFS test work at an average of 96.2% for oxide, 93% for transition and 87.9% for fresh ores.

Gold dissolution kinetics were consistent with the rates reported in the PFS, with the oxide and transitional ores leaching effectively complete in approximately 24 hours, however, gold extraction for some fresh ores were not complete until 40 hours leach time.

There were no elements, including Sulphur that displayed any relationship with gold recovery. A regression analysis was conducted on the leach results from all FS Master composite leach tests and PFS leach test results to assess the relationship between leach feed grade on the final leach residue grade and extraction.

The regression format and coefficients derived from a regression analysis exhibited a high level of correlation across all lithology types. All leach results were analysed to produce three regression recovery curves, one for each of the three lithology types. These curves were then applied to the varying head grades of the mine schedule to derive overall recovery. The following Table 13.1 presents the predicted recoveries derived from the regression analysis versus reserve head grades.

	Unit	Souwa			Nokpa			Kilosegui			Kekeda		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC1	MC2	MC3	MC4	MC5	MC6	MC7	MC8	MC9	MC10	MC11	MC12
Head Grade	g/t	1.27	1.54	1.70	n/a	1.62	1.78	1.42*	1.08	1.08	1.00	1.11	1.20
Au Recovery	%	95.0	94.4	87.6	n/a	95.1	89.0	97.7	92.0	85.5	96.0	93.7	88.3

	Unit	Enioda			Han			Chegue Main			Chegue South		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC13	MC14	MC15	MC16	MC17	MC18	MC19	MC20	MC21	MC22	MC23	MC24
Head Grade	g/t	1.40	1.37	1.82	n/a	2.15	1.89	1.11	1.01	1.58	n/a	1.04	1.34
Au Recovery	%	96.8	93.9	92.7	n/a	92.2	88.8	95.7	92.7	89.7	n/a	89.9	81.9

** Denotes assay gold grade and not reserve gold grade*
n/a Denotes no oxide samples tested

Table 13.1 Gold Grade Versus Recovery (Regression Analysis)

Cyanide consuming elements, Cu, Zn, Pb and Ni and Sb were reported at similar ranges to the PFS composites and are not expected to impact on the circuit.

The silver head grades reported were consistent with the PFS composite head assays and reported low grades at or under the 2.0 g/t Ag detection limit. The Han (MC16) and Enioda (MC13) Oxide composites were exceptions reporting elevated silver head grades at 12 g/t Ag and 6 g/t Ag respectively.

All Master composites reported varying grades of Tellurium (Te) at an average 2.54 g/t. The presence of Tellurides at these grades warranted monitoring during subsequent test work. However, there was no measurable correlation observed between Te grades and Au cyanide solubility and therefore is not expected to be a significant metallurgical issue.

The Oxide and Transition composites reported very low or below detection sulphide sulphur assays. Fresh composites reported sulphide sulphur levels in a range of 0.24% - 0.62%.

13.1.6 Further Metallurgical Testing

Detailed metallurgical test work is continuing for the Doropo Gold Project under the direction of Centamin to support the design as part of the continuous improvement programme. Ongoing test work is listed below:

- Bulk Carbon-in-Leach (CIL) cyanidation testing of Souwa, Nokpa, Kilosegui, Enioda and Chegue Main Master composites under optimised conditions;
- Dynamic thickener and detoxification test work;
- Pre-Leach & Carbon-in-Leach test work;
- Carbon-in-Leach (CIL) cyanidation testing of all Variability composites under optimised conditions;

- Equilibrium carbon loading tests on all Master composites;
- Rheology test work has recently been reported and currently under review. Initial results indicate that 55% and 60% solids are a viable option;
- Oxygen demand tests; and
- Diagnostic leach tests on CIL leach tails.

The metallurgical work carried out to date indicates that gold can be satisfactorily recovered from Doropo ores using conventional Carbon in Leach (CIL) cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

13.2 Feasibility Test Work

Feasibility test work scope followed on from the PFS test work reported in Lycopodium (2023). Independent Metallurgical Operations (IMO) was retained by Centamin in June 2023 to initiate and oversee metallurgical test work on samples from the Doropo gold deposits.

A key focus of this current phase has been the assessment of the pre-oxidation and cyanidation leach circuit configuration to optimise gold recovery and reduce cyanide consumption over the Base Case reported at the close of the PFS. Feasibility test scope included MACH© shear reactor testing to increase leach kinetics, as well as a range of PDC supporting test work performed by ALS in Perth.

The current test work programme has provided focus on optimisation of the leaching response across a range of representative master and variability composites formed predominately from PQ and HQ drill-core, with some RC drill samples obtained for variability testing.

The following metallurgical flowsheet development testing is reported in this chapter:

- Historical comminution test work;
- Comprehensive Master and Variability lithology composite head assay analysis;
- Mineralogical analysis by QEMSCAN and XRD;
- Grind optimization cyanidation testing on all Master composites;
- Cyanide optimization cyanidation testing on all Master composites;
- Direct cyanidation leach tests on Souwa and Nokpa Master composites under optimised test conditions comparing Oxygen versus Air sparging;
- Pre-oxidation shear reactor MACH© testing targeting improved gold recovery and faster leach kinetics;
- Optimised CIL and Pre-oxidation test work with Lead Nitrate on all Master composites;
- Cyanide detoxification testing and dynamic thickener testing of detox tailings.

The results reported in this document are from test work completed as at Friday 14 June 2024. Further test work was well advanced at this time, however, the results were not expected to necessitate changes to the process flowsheet. The outstanding test work will be reported and used in the next phase of the development of the Project.

The outstanding test work not deemed necessary for finalising the process flowsheet but are still required for the project development are as follows:

- Further dynamic thickening and cyanide detoxification test work, including cyanide speciation;
- Pre-Leach and Carbon-in-Leach test work;
- Oxygen uptake testing;
- Slurry rheology characterization on process slurry at different densities;
- CIL cyanidation testing of all Variability composites under optimised conditions;
- Sequential CIP testing - Triple Carbon Contact/Carbon Kinetics test work on all Master composites;
- Diagnostic leaching of Carbon-in-Leach (CIL) cyanidation tails of all Master composites under optimised conditions.

13.3 Comminution Test Work

IMO notes that all relevant metallurgical test work completed and reported during the Lycopodium PFS has been utilised within the FS as the test work results are considered relevant and support the Doropo flowsheet and process design criteria.

IMO believes the PFS test work programme included the necessary comminution tests and sample provenance requirements to fulfill the prerequisites of a FS. The results of the comminution test work undertaken during the PFS on samples across all predominant lithologies and weathered rock types were interpreted by Orway Mineral Consultants Pty Ltd (OMC).

Based on the test work results and on the mine schedule information, OMC developed a milling circuit design to provide sufficient flexibility to accommodate all ore types likely to be processed. The circuit configuration and mill selection were based on a circuit treating 4.0 Mt/a of fresh ore to achieve a grind P_{80} of 75 µm.

13.3.1 Comminution Overview

Comminution test work was conducted on 43 samples of oxide, transition and fresh material from the nine deposits to determine the variability of comminution parameters throughout the deposit/orebody and allow parameters for design of the comminution circuit to be derived. The results of comminution tests conducted on samples from the Attire deposit have been presented, however no further metallurgical tests were conducted and as such do not form part of the FS.

The 43 samples consisted of 4 oxide, 30 fresh and 9 transition selected from throughout the Doropo mineralised zones.

Comminution tests conducted included the following:

- Unconfined Compressive Strength (UCS);
- Bond Crushing Work Index (BCWi);
- Bond Rod Mill Work Index (BRWi);
- Bond Ball Mill Work Index (BBWi);
- SAG Mill Comminution value (SMC);
- Abrasion index (Ai);
- In-situ Specific Gravity (SG).

All tests were conducted at ALS Perth and the results from the SMC tests were interpreted and ranked by JKTech.

13.3.2 PFS Study Comminution Sample Selection

A summary of the samples selected for the PFS comminution test work programme is provided in Table 13.2.

Deposit	Sample Name	Drill Hole	Depth		Weathering	Lithology	Alteration
			From (m)	To (m)			
Souwa	Souwa Oxide	DPDD1517	0	10	Oxide		
	Souwa Transition	DPDD1462	22.5	32.5	Transition		
	Souwa Fresh 1	DPDD1520	60	70	Fresh	Granodiorite	Sericite
	Souwa Fresh 2	DPDD1523	133	143	Fresh	Granodiorite	Veining
	Souwa Fresh 3	DPDD1527	96	103	Fresh	Granodiorite	Hematite
	Souwa Fresh 4	DPDD1382	79	90	Fresh	Granodiorite	Chlorite & Hematite & Silica
	Souwa Fresh 5	DPDD1473	80	87	Fresh	Granodiorite + Vein	Silica & Sericite
Nokpa	Nokpa Transition	DPDD1533	12.5	21.5	Transition	Saprolite	Veining
	Nokpa Fresh 1	DPDD1528	81	88	Fresh	Granodiorite	
	Nokpa Fresh 2	DPDD1531	108	118	Fresh	Granodiorite	Veining
	Nokpa Fresh 3	DPDD1425	120	137.7	Fresh	Granodiorite + Vein	Hematite & Silica
Chegue Main	Chegue Main Oxide	DPDD1541	5	11	Oxide	Saprolite	Clay Undiff
	Chegue Main Transition	DPDD1541	21	26	Transition	Saprock	Limonite
	Chegue Main Fresh 1	DPDD1536	75	80	Fresh	Granodiorite	Sericite
	Chegue Main Fresh 2	DPDD1537	52	57	Fresh	Stockwork veins	Silica
	Chegue Main Fresh 3	DPDD1538	78	83	Fresh	Granodiorite	Hematite
	Chegue Main Fresh 4	DPDD1543	65	70	Fresh	Vein (>10cm)	Silica
	Chegue Main Fresh 5	DPDD1494	50	58	Fresh	Granodiorite + Vein	Silica & Sericite
Chegue South	Chegue South Transition	DPDD1551	23	28	Transition	Saprock + Vein	Limonite + Clay
	Chegue South Fresh 1	DPDD1544	55	61	Fresh	Granodiorite	Hematite
	Chegue South Fresh 2	DPDD1549	82	90	Fresh	Granodiorite + Vein	Chlorite + Sericite

Deposit	Sample Name	Drill Hole	Depth		Weathering	Lithology	Alteration
			From (m)	To (m)			
Kekeda	Kekeda Transition 1	DPDD1554	38	47	Transition	Saprock & Stockwork Veins	Epidote & Limonite
	Kekeda Transition 2	DPDD1559	39	46	Transition	Granodiorite	Silica & Sericite
	Kekeda Fresh 1	DPDD1554	57	64	Fresh	Granodiorite with Veins	Sericite
	Kekeda Fresh 2	DPDD1556	49	53	Fresh	Vein	Silica
	Kekeda Fresh 3	DPDD1558	68	73	Fresh	Granodiorite	Sericite & Hematite
	Kekeda Fresh 4	DPDD1479	44	50.3	Fresh	Granite + Vein	Sericite & Hematite & Silica & Epidote
Han	Han Transition	DPDD1562	12	17	Trans/Fresh	Granodiorite	Sericite
	Han Fresh 1	DPDD1564	81	86	Fresh	Granodiorite	Hematite
	Han Fresh 2	DPDD1561	61	67	Fresh	Granodiorite	Chlorite & Hematite bands
	Han Fresh 3	DPDD1565	48	54	Fresh	Granodiorite	Chlorite
Enioda	Enioda Oxide 1	DPDD1571	20	27	Oxide	Saprock & Vein	Hematite & Limonite
	Enioda Oxide 2	DPDD1574	10	20	Oxide	Saprolite	Hematite & Limonite
	Enioda Fresh 1	DPDD1567	79	86	Fresh	Granodiorite	Chlorite & Hematite
	Enioda Fresh 2	DPDD1573	78	85	Fresh	Granodiorite	Sericite & Hematite & Silica
Attire	Attire Transition	DPDD1580	30	39	Transition	Saprock	Hematite
	Attire Fresh 1	DPDD1578	55	60	Fresh	Granodiorite	Chlorite & Hematite
	Attire Fresh 2	DPDD1575	73	81	Fresh	Granodioritic, Gneiss & Vein	Sericite & Hematite & Silica
Kilosegui	Kilosegui Transition	DPDD3012	23	28	Transition	Saprock	Sericite & Hematite & Silica
	Kilosegui Fresh 1	DPDD3008	37	42	Fresh	Granodiorite with small Veins	Sericite & Hematite & Silica
	Kilosegui Fresh 2	DPDD3010	28	33	Fresh	Granodiorite	Hematite
	Kilosegui Fresh 3	DPDD3011	63	68	Fresh	Granodiorite with large Qtz Vein	Sericite & Hematite & Silica
	Kilosegui Fresh 4	DPDD3013	91	96	Fresh	Granodiorite	Hematite

Table 13.2 Comminution Test Work Samples

13.3.3 SMC Tests

A series of SMC tests were performed on 36 of the comminution samples. A summary of the SMC test results and the derived parameters is presented in Table 13.3 and Table 13.4.

Test Sample	DWi (kWh/m³)	DWi (%)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	S.G.
Attire Fresh 1	6.8	52	19.7	14.6	7.6	2.69
Attire Fresh 2	7.4	60	21	15.8	8.2	2.71
Attire Trans	2.3	6	8.6	5.1	2.7	2.57
Chegue Main Fresh 1	6.8	53	20	14.8	7.7	2.68
Chegue Main Fresh 2	6.4	46	18.8	13.8	7.1	2.67
Chegue Main Fresh 3	6	41	18.2	13.2	6.8	2.65
Chegue Main Fresh 4	5.8	38	17.4	12.5	6.4	2.68
Chegue Main Trans	0.5	<1	2.7	1.2	0.6	2.48
Chegue South Fresh 1	7.2	58	20.8	15.7	8.1	2.68
Chegue South Fresh 2	7.4	60	21	15.8	8.2	2.72
Chegue South Trans	3.3	12	11.5	7.4	3.8	2.56
Enioda Fresh 1	8.6	74	23.6	18.4	9.5	2.7
Enioda Fresh 2	6.9	53	19.7	14.6	7.6	2.73
Enioda Oxide 1	2.5	7	10.4	6.3	3.3	2.29
Han Fresh 1	5.8	38	17.5	12.5	6.5	2.67
Han Fresh 2	7.3	59	20.9	15.7	8.1	2.7
Han Fresh 3	5.7	38	17.2	12.4	6.4	2.69
Han Trans/Fresh	3.2	12	11	7	3.6	2.65
Kekeda Fresh 1	6.1	43	18.1	13.2	6.8	2.7
Kekeda Fresh 2	4.5	23	14.2	9.7	5	2.68
Kekeda Fresh 3	5.7	38	17.1	12.2	6.3	2.72
Kekeda Trans 1	1.5	3	6.3	3.4	1.8	2.58
Kekeda Trans 2	2.8	9	10.2	6.3	3.3	2.58
Kilosequi Fresh 1	4.8	27	15.7	10.8	5.6	2.57
Kilosequi Fresh 2	4.3	21	13.9	9.4	4.9	2.63
Kilosequi Fresh 3	5.1	30	15.8	11	5.7	2.68
Kilosequi Fresh 4	5.5	35	17.3	12.3	6.4	2.58
Kilosequi Trans 1	2.6	8	9.4	5.7	3	2.6
Nokpa Fresh 1	3.1	11	10.7	6.8	3.5	2.66
Nokpa Fresh 2	4.7	25	14.8	10.2	5.3	2.67
Nokpa Transition	3.1	11	11	7	3.6	2.58
Souwa Fresh 1	5.8	39	17.6	12.6	6.5	2.67
Souwa Fresh 2	6.8	51	19.7	14.6	7.6	2.69
Souwa Fresh 3	7.5	61	21.3	16.1	8.3	2.68
Souwa Oxide	0.8	1	4	1.9	1	2.32
Souwa Transition Saprolite	2.6	8	9.6	5.8	3	2.56

Table 13.3 SMC Tests

Test Sample	A	b	A * b	ta	SCSE (kWh/t)
Attire Fresh 1	73.6	0.54	39.7	0.38	9.88
Attire Fresh 2	75	0.49	36.8	0.35	10.29
Attire Trans	68.8	1.64	112.8	1.14	6.57
Chegue Main Fresh 1	66	0.59	38.9	0.38	9.96
Chegue Main Fresh 2	66.4	0.63	41.8	0.41	9.62
Chegue Main Fresh 3	63.4	0.69	43.7	0.43	9.4
Chegue Main Fresh 4	65.4	0.71	46.4	0.45	9.2
Chegue Main Trans	77.1	6.23	480.3	5.02	4.88*
Chegue South Fresh 1	77.8	0.48	37.3	0.36	10.15
Chegue South Fresh 2	69.7	0.53	36.9	0.35	10.29
Chegue South Trans	61.8	1.28	79.1	0.80	7.38
Enioda Fresh 1	80.2	0.39	31.3	0.30	11.1
Enioda Fresh 2	69.7	0.57	39.7	0.38	9.96
Enioda Oxide 1	67.6	1.36	91.9	1.04	7.24
Han Fresh 1	75	0.62	46.5	0.45	9.18
Han Fresh 2	78.2	0.47	36.8	0.35	10.27
Han Fresh 3	73.4	0.64	47	0.45	9.17
Han Trans/Fresh	69.6	1.19	82.8	0.81	7.28
Kekeda Fresh 1	76	0.58	44.1	0.42	9.44
Kekeda Fresh 2	72.6	0.82	59.5	0.58	8.28
Kekeda Fresh 3	69.9	0.68	47.5	0.45	9.17
Kekeda Trans 1	68.1	2.47	168.2	1.69	5.87
Kekeda Trans 2	69.9	1.31	91.6	0.92	7.02
Kilosequi Fresh 1	73.2	0.73	53.4	0.54	8.57
Kilosequi Fresh 2	77.5	0.79	61.2	0.60	8.15
Kilosequi Fresh 3	70.5	0.75	52.9	0.51	8.7
Kilosequi Fresh 4	70.9	0.66	46.8	0.47	9.06
Kilosequi Trans 1	73.4	1.37	100.6	1	6.81
Nokpa Fresh 1	60	1.42	85.2	0.83	7.22
Nokpa Fresh 2	69.3	0.82	56.8	0.55	8.43
Nokpa Transition	69.1	1.21	83.6	0.84	7.24
Souwa Fresh 1	76.4	0.60	45.8	0.44	9.24
Souwa Fresh 2	79.4	0.50	39.7	0.38	9.89
Souwa Fresh 3	84.6	0.43	36.4	0.35	10.28
Souwa Oxide	74.8	4.07	304.4	3.4	5.44
Souwa Transition Saprolite	65.9	1.51	99.5	1.01	6.84

Table 13.4 SMC Parameters

The drop weight index (DWi) is a measure of the strength of the rock when broken under impact conditions. The fresh ore has medium competency. Transition ore has moderate competency. The oxide ore has low to moderate competency.

The A x b value indicates the material's resistance to impact breakage. As there is an inverse relationship between the DWi and the product of the 'A' and 'b' rock breakage parameters, a lower A x b value represents a greater ore hardness. In general, ore types which return A x b values of less than ~40 are considered hard. JK Tech Axb results ranged from 31.3 to 480.3. The average Axb parameters in increasing order of competency were: 198 for oxide ore, 144 for transitional ore and 49 for fresh ore. An A x b of 37.2 for fresh and 84.2 for oxide were selected for design.

13.3.4 Comminution Tests

The comminution test work results are summarised in Table 13.5 and illustrated graphically in Figure 13.1 with the following salient outcomes:

- The fresh ore has medium competency, moderate to high grinding energy requirements and is moderately abrasive. Transition ore has moderate competency with average grinding energy requirements and is moderately abrasive. The oxide ore has low to moderate competency with average to high grinding energy requirements and is low to moderately abrasive;
- Crushing work indices results ranged from 1.6 to 15.8 kWh/t, averaging 8.3 kWh/t over the complete set of samples. The crushing work indices are low to medium, indicating a low crushing energy requirement. The maximum BCWi values, ranging from 4.1 to 15.8 kWh/t were for the Kekeda ores, with an 85th percentile value of 14.9 kWh/t;
- Rod mill work indices results ranged from 7.2 to 21.4 kWh/t. A BRWi of 18.5 kWh/t for fresh and 14.9 for oxide were selected for design;
- BBWi results ranged from 11 to 22.2 kWh/t. The Bond ball mill indices are medium to high indicating a high grinding energy requirement. Selected 85th percentile data for design were 20.4 kWh/t for oxide and 18.5 kWh/t for fresh;
- The abrasion indices measured ranged from very low at 0.018 to medium at 0.38. The oxide samples demonstrated the highest abrasion indices, averaging 0.248. Selected 85th percentile data for design were 0.248 for oxide and 0.20 for fresh ores.

Oxidation	Ore Type	Sample Name	Ai (g)	BRWi (kWh/t)	BBWi (kWh/t)	Avg BCWi (kWh/t)	Avg UCS (Mpa)
Transition	Attire	Attire Transition	0.122		16.6	5.0	-
Fresh	Attire	Attire Fresh1	0.116		17.2	11.3	73
Fresh	Attire	Attire Fresh2	0.072		18.7	10.8	87
Oxide	Chegue Main	Chegue Main Oxide	-	-	19.8	-	-
Transition	Chegue Main	Chegue Main Transition	0.045	7.2	12.6	1.6	-
Fresh	Chegue Main	Chegue Main Fresh1	0.120	18.2	17.7	8.8	60
Fresh	Chegue Main	Chegue Main Fresh2	0.347	18.7	16.8	11.0	44
Fresh	Chegue Main	Chegue Main Fresh3	0.125	17.2	16.9	2.1	-
Fresh	Chegue Main	Chegue Main Fresh4	0.380	17.4	17.7	12.3	40
Fresh	Chegue Main	Chegue Main Fresh5	0.191	17.3	17.8	6.5	40
Transition	Chegue South	Chegue South Transition	0.214	11.9	16.1	1.8	14
Fresh	Chegue South	Chegue South Fresh1	0.291	17.9	19.4	13.3	38
Fresh	Chegue South	Chegue South Fresh2	0.161	19.4	18.5	10.7	85
Oxide	Enioda	Enioda Oxide1	0.360	-	18.3	3.9	-
Oxide	Enioda	Enioda Oxide2	-	-	22.2	-	-
Fresh	Enioda	Enioda Fresh1	0.244	21.4	16.9	13.5	134
Fresh	Enioda	Enioda Fresh2	0.155	17.8	16.6	7.9	56
Transition	Han	Han Transition	0.168	11.8	17.0	7.8	88
Fresh	Han	Han Fresh1	0.251	16.3	19.4	14.6	98
Fresh	Han	Han Fresh2	0.223	17.4	17.8	10.6	99
Fresh	Han	Han Fresh3	0.262	15.3	17.2	9.1	94
Transition	Kekeda	Kekeda Transition1	0.135	10.6	14.0	4.1	34
Transition	Kekeda	Kekeda Transition2	0.119	12.5	15.2	12.1	37
Fresh	Kekeda	Kekeda Fresh1	0.217	18.1	16.4	13.8	106
Fresh	Kekeda	Kekeda Fresh2	0.162	14.3	17.1	7.1	83
Fresh	Kekeda	Kekeda Fresh3	0.172	17.4	17.3	15.8	93
Fresh	Kekeda	Kekeda Fresh4	0.163	13.3	16.1	6.2	93
Transition	Kilosegui	Kilosegui Transition	0.062	12.3	11.0	-	-
Fresh	Kilosegui	Kilosegui Fresh1	0.315	16.5	16.5	9.9	60
Fresh	Kilosegui	Kilosegui Fresh2	0.309	12.7	16.1	10.4	128
Fresh	Kilosegui	Kilosegui Fresh3	0.183	16.4	16.7	10.3	96
Fresh	Kilosegui	Kilosegui Fresh4	0.193	17.3	16.7	10.4	82
Transition	Nokpa	Nokpa Transition	0.326	15.9	19.7	-	-
Fresh	Nokpa	Nokpa Fresh1	0.018	15.7	18.3	3.1	-
Fresh	Nokpa	Nokpa Fresh2	0.122	16.0	17.9	5.6	22
Fresh	Nokpa	Nokpa Fresh3	0.113	18.4	19.6	4.6	22

Oxidation	Ore Type	Sample Name	Ai (g)	BRWi (kWh/t)	BBWi (kWh/t)	Avg BCWi (kWh/t)	Avg UCS (Mpa)
Oxide	Souwa	Souwa Oxide	0.136	11.9	17.5	-	-
Transition	Souwa	Souwa Transition	0.148	12.7	13.6	2.9	-
Fresh	Souwa	Souwa Fresh1	0.223	16.6	15.9	4.8	22
Fresh	Souwa	Souwa Fresh2	0.267	17.9	17.8	9.3	30
Fresh	Souwa	Souwa Fresh3	0.255	18.8	17.7	10.0	54
Fresh	Souwa	Souwa Fresh4	0.169	16.5	16.9	6.9	54
Fresh	Souwa	Souwa Fresh5	0.145	16.3	17.0	5.3	54
85th Percentile	All Samples		0.190	15.8	17.1	8.3	66
85th Percentile	Fresh		0.199	17.0	17.4	9.2	70
85th Percentile	Oxide		0.248	11.9	19.4	3.9	-
85th Percentile	Transition		0.149	11.9	15.1	5.0	43

Table 13.5 Comminution Test Work Results Summary

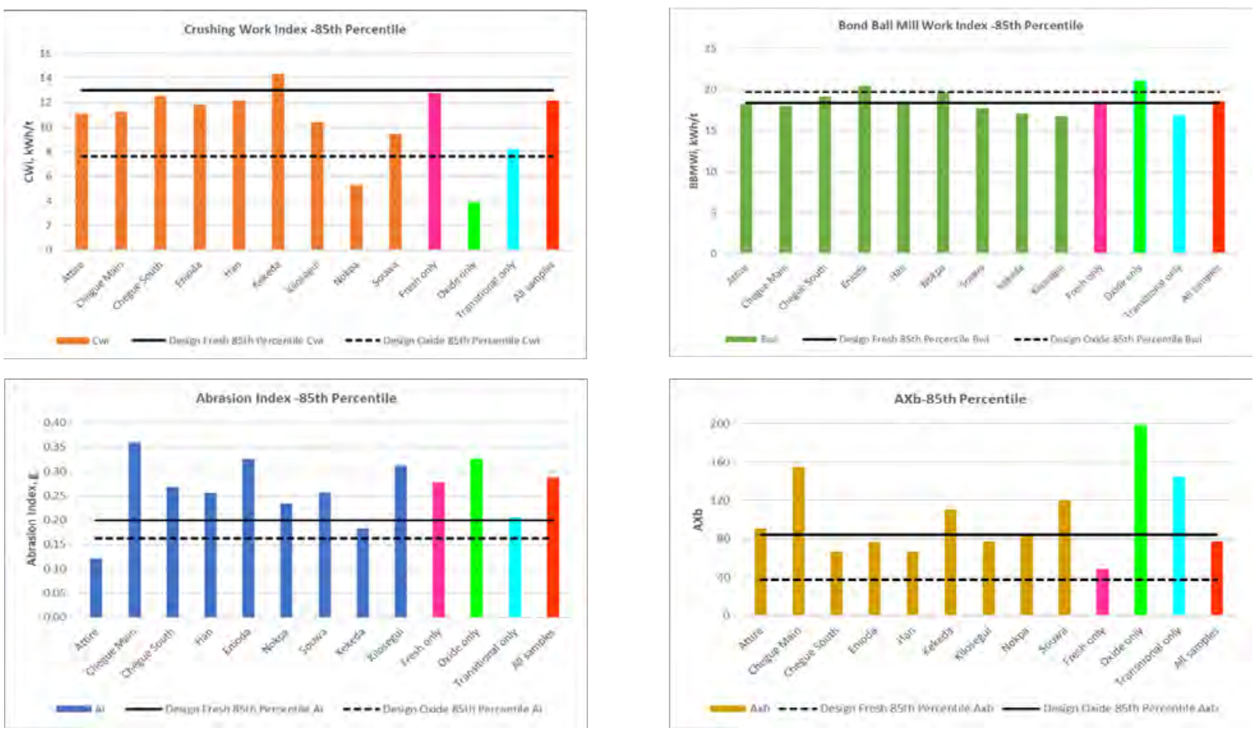


Figure 13.1 Ore Comminution Characteristics

13.3.5 Results Interpretation

The results of the PFS comminution test work programme conducted on 36 ore samples across all predominant lithologies and weathered rock types were forwarded to Orway Mineral Consultants Pty Ltd (OMC) for interpretation. Based on the test work results and on the mine schedule information, OMC developed a milling circuit design to provide sufficient flexibility to accommodate all ore types likely to be processed. The circuit configuration and mill selection were based on a circuit treating 4,000,000 tpa of fresh ore to achieve a grind P_{80} of 75 µm.

13.4 Sample Provenance

13.4.1 Feasibility Study Samples

IMO note that all relevant metallurgical test work completed and reported during the client Prefeasibility Study has been utilised within the Feasibility Study (FS) as the test work results are considered relevant and support the Doropo flowsheet and process design criteria. The process flowsheet has not changed from the PFS to the issuance of this FS.

To ensure that the processing characteristics of all major ore lithologies over the Life of Mine (LoM) are understood, Centamin, with assistance from IMO, selected appropriate diamond drill core samples as part of the FS test work programme. Diamond drill (DD) core was used to produce the majority of the metallurgical composites of the different lithologies, with only a small proportion of reverse circulation (RC) drill chips utilised.

A total of eight (8) open pits were tested as part of the feasibility study including the following, Souwa, Nokpa, Kilosegui, Kekeda, Enioda, Han, Chegue Main and Chegue South. A summary of the composite samples prepared for the feasibility test work is provided in Table 13.6.

The composite samples were prepared from drill holes selected across each of the proposed eight open pit shells to be spatially representative, and at depth. The intervals of core were selected to provide for the preparation of Master, Low-Grade and High-Grade gold composites of the three lithology types found in the Doropo ores. The number of drill holes used, and weights of the composites prepared for each ore deposit is provided in Table 13.7.

Pit Name	Oxide	Trans	Fresh
Souwa	MC1, VC1, VC2	MC2, VC3, VC4	MC3, VC5, VC6
Nokpa	MC4, VC7	MC5, VC8, VC9	MC6, VC10, VC11
Kilosegui	MC7	MC8, VC12, VC13	MC9, VC14, VC15
Kekeda	MC10, VC16, VC17	MC11, VC18, VC19	MC12, VC20, VC21
Enioda	MC13, VC22, VC23	MC14, VC24, VC25	MC15, VC26, VC27
Han	MC16	MC17, VC28, VC29	MC18, VC30, VC31

Pit Name	Oxide	Trans	Fresh
Chegue Main	MC19, VC32, VC33	MC20, VC34, VC35	MC21, VC36, VC37
Chegue South	MC22	MC23, VC38, VC39	MC24, VC40, VC41

Note: MC- master composite & VC - variability composites (low & high grade)

Table 13.6 Feasibility Study Sample Summary

Pit	Oxide				Transitional			Fresh		
	Drill Holes	Master	LG Oxide	HG Oxide	Master	LG Trans	HG Trans	Master	LG Fresh	HG Fresh
	#	kg	kg	kg	kg	kg	kg	kg	kg	kg
Souwa	23	142.0	10.5	12.2	326.3	109.0	22.9	239.3	104.4	98.3
Nokpa	15	20.0	8.4		173.3	16.7	30.5	174.7	65.8	52.9
Kilosegui	18	65.0			367.6	87.6	33.5	474.9	109.9	16.0
Kekeda	26	43.1		6.3	434.4	176.5	79.0	295.2	75.5	73.0
Enioda	16	143.1	38.7	28.4	146.9	23.9	6.6	146.6	92.2	52.9
Han	16	7.2			123.4	5.9	28.2	203.7	82.9	39.3
Chegue Main	20	36.3		11.6	207.1	45.8	16.1	238.7	106.1	69.7
Chegue South	13	7.5			64.3	45.1	23.4	99.7	26.3	72.6

Table 13.7 Feasibility Study Composite Sample Weights

Souwa

Three (3) master composites (MC1 - MC3) were prepared for the Souwa pit for fresh, transitional and oxide material. Six (6) variability composites were prepared (VC1 - VC6) with all composite information presented in Table 13.8 to Table 13.16.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4701	3	7
	DPDD4702R	3	7
	DPDD4704	10	21
	DPDD4705	21-27, 31-35	
	DPDD4715	21-25, 30-36, 38-39	
	Head Grade Assay	(Avg Au g/t)	1.47
	Resource	Au (g/t)	1.27

Table 13.8 Souwa Oxide Master Composite Details (MC1)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4700	10-15, 23-26	
	DPDD4701	9	12
	DPDD4703	20	30
	DPDD4706R	35-36, 37-43, 45-51	
	DPDD4707	35	46
	DPDD4708	45	51
	DPDD4709	38-41, 45-50, 51-57, 59-61	
	DPDD4710R	38-39, 42-46	
	DPDD4712	78-80, 81-85, 86-87	
	DPDD4715	45	49
	Head Grade Assay	(Avg Au g/t)	1.45
	Resource	Au (g/t)	1.54

Table 13.9 Souwa Transitional Master Composite Details (MC2)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4706R	51	55
	DPDD4707	53-55, 59-60	
	DPDD4711	82	95
	DPDD4713	109	115
	DPDD4714	78-88, 91-92	
	DPDD4716	70-80, 83-86	
	DPDD4717	89	112
	Head Grade Assay	(Avg Au g/t)	1.77
	Resource	Au (g/t)	1.70

Table 13.10 Souwa Oxide Master Composite Details (MC3)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4715	25	27
CIS339768	DPDD1460	11	20
CIS339783	DPDD1524	38	43
CIS 790019	DPDD1491	7	11
	Head Grade Assay	(Avg Au g/t)	0.72

Table 13.11 Souwa Oxide Low Grade Variability Composite Details (VC1)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4715	36	38
CIS339782	DPDD1524	30	38
	Head Grade Assay	(Avg Au g/t)	4.31

Table 13.12 Souwa Oxide High Grade Variability Composite Details (VC2)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4709	57	59
CIS339758	DPDD1462	12.5	22.5
	Head Grade Assay	(Avg Au g/t)	0.71

Table 13.13 Souwa Transitional Low Grade Variability Composite Details (VC3)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4710R	39	42
CIS 790007	DPDD1584	40	45
	Head Grade Assay	(Avg Au g/t)	3.04

Table 13.14 Souwa Transitional High Grade Variability Composite Details (VC4)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4714	88	90
CIS339790	DPDD1527	86	90
CIS 790006	DPDD1473	80	94
	Head Grade Assay	(Avg Au g/t)	1.04

Table 13.15 Souwa Fresh Low Grade Variability Composite Details (VC5)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4716	80	83
CIS 790002	DPDD1382	79	100
CIS339778	DPDD1523	133	143
	Head Grade Assay	(Avg Au g/t)	4.63

Table 13.16 Souwa Fresh High Grade Variability Composite Details (VC6)

Nokpa

Three (3) master composites (MC4 - MC6) were prepared for the Nokpa pit for fresh, transitional and oxide material. Five (5) variability composites were prepared (VC7 - VC11) with all composite information presented in Table 13.17 to Table 13.24.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPRC4633	0	2
CIS 790021	DPDD1589	0	3
	Head Grade Assay	(Avg Au g/t)	1.39
	Resource	Au (g/t)	n/a

Table 13.17 Nokpa Oxide Master Composite Details (MC4)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4718	28	31
	DPDD4719	8	15
	DPDD4720	35-36, 39-40	
	DPRC4631	0	4
	DPRC4632	4	7
	DPRC4633	2	4
	Head Grade Assay	(Avg Au g/t)	1.64
	Resource	Au (g/t)	1.62

Table 13.18 Nokpa Transitional Master Composite Details (MC5)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4718	53	58
	DPDD4721	90-95, 99-103, 104-112, 115-129	
	DPDD4722	48-62, 63-64, 79-87	
	DPDD4723	128	133
	Head Grade Assay	(Avg Au g/t)	1.68
	Resource	Au (g/t)	1.78

Table 13.19 Nokpa Fresh Master Composite Details (MC6)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPRC4634	0	1
CIS 790019	DPDD1491	7	11
	Head Grade Assay	(Avg Au g/t)	0.72

Table 13.20 Nokpa Oxide Low Grade Variability Composite Details (VC7)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4720	33	35
CIS339802	DPDD1533	12.5	21.50
	Head Grade Assay	(Avg Au g/t)	0.84

Table 13.21 Nokpa Transitional Low Grade Variability Composite Details (VC8)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4720	37	39
CIS 790039	DPDD1589	25	36.00
CIS 790040	DPDD1588	26	35.00
CIS 790041	DPDD1491	28	33.00
	Head Grade Assay	(Avg Au g/t)	3.56

Table 13.22 Nokpa Transitional High Grade Variability Composite Details (VC9)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4721	112	115
CIS339799	DPDD1531	118	128
	Head Grade Assay	(Avg Au g/t)	0.76

Table 13.23 Nokpa Fresh Low Grade Variability Composite Details (VC10)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4721	95	98
CIS339797	DPDD1531	98	108
	Head Grade Assay	(Avg Au g/t)	4.36

Table 13.24 Nokpa Fresh High Grade Variability Composite Details (VC11)

Kilosegui

Three (3) master composites (MC7 - MC9) were prepared for the Kilosegui pit for fresh, transitional and oxide material. Four (4) variability composites were prepared (VC12 - VC15) with all composite information presented in Table 13.25 to Table 13.31.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPRC4642	2	6
	DPRC4644	15	17
	Head Grade Assay	(Avg Au g/t)	1.42
	Resource	Au (g/t)	n/a

Table 13.25 Kilosegui Oxide Master Composite Details (MC7)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6300	10	20
	DPDD6301	4	14
	DPDD6302	8	10
	DPDD6303	42	49
	DPDD6304	29	45
	DPDD6305	11	34
	DPDD6306	1	2
	DPDD6307	6-8, 10-11	
	DPDD6309	24	27
	DPRC4640	2	6
	DPRC4642	6-9, 12-14	
	Head Grade Assay	(Avg Au g/t)	1.49
	Resource	Au (g/t)	1.08

Table 13.26 Kilosegui Transitional Master Composite Details (MC8)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6300	20	22
	DPDD6302	10	21
	DPDD6303	49	57
	DPDD6306	2-10, 28-36	
	DPDD6307	11	32
	DPDD6308	43	56
	DPDD6309	47	57
	DPDD6310R	44-46, 57-73	
	Head Grade Assay	(Avg Au g/t)	1.65
	Resource	Au (g/t)	1.32

Table 13.27 Kilosegui Fresh Master Composite Details (MC9)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6305	8.0	12.0
	DPDD6307	8.0	10.0
CIS734014	DPDD3005	6.0	11.0
CIS734048	DPDD3015	6.3	13.0
	Head Grade Assay	(Avg Au g/t)	0.54

Table 13.28 Kilosegui Transitional Low Grade Variability Composite Details (VC12)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6308	37	40
	DPRC4642	9	12
	Head Grade Assay	(Avg Au g/t)	2.92

Table 13.29 Kilosegui Transitional High Grade Variability Composite Details (VC13)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6310R	46	48
CIS734035	DPDD3011	58	68
CIS734046	DPDD3014	48	58
CIS734049	DPDD3015	13	23
	Head Grade Assay	(Avg Au g/t)	0.86

Table 13.30 Kilosegui Fresh Low Grade Variability Composite Details (VC14)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD6308	40	43
	Head Grade Assay	(Avg Au g/t)	3.35

Table 13.31 Kilosegui Fresh High Grade Variability Composite Details (VC15)

Kekeda

Three (3) master composites (MC10 - M12) were prepared for the Kekeda pit for fresh, transitional and oxide material. Six (6) variability composite were prepared (VC16 - VC21) with all composite information presented in Table 13.32 to Table 13.40.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4742	0	4
	DPDD4743	1-3, 11-12	
	DPDD4746	0	4
CIS 790059	DPRC3281	2	6
	Head Grade Assay	(Avg Au g/t)	0.86
	Resource	Au (g/t)	0

Table 13.32 Kekeda Oxide Master Composite Details (MC10)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4742	23	30
	DPDD4744	2-7, 14-19	
	DPDD4745	9-11, 13-16	
	DPDD4746	9	12
	DPDD4747	18-19, 22-25	
	DPDD4748	6-10, 14-24	
	DPDD4749	7-15, 16-20	
	DPDD4750	24	30
	DPRC4638	3	9
	DPRC4639	4	7
	Head Grade Assay	(Avg Au g/t)	1.29
	Resource	Au (g/t)	1.11

Table 13.33 Kekeda Transitional Master Composite Details (MC11)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4745	16	23
	DPDD4749	20	21
	DPDD4751	57	63
	DPDD4752	46-48, 49-53	
	DPDD4753	22-24, 26-34	
	DPDD4754	48-52, 56-60	
	DPDD4755	25	42
	Head Grade Assay	(Avg Au g/t)	1.34
	Resource	Au (g/t)	1.20

Table 13.34 Kekeda Fresh Master Composite Details (MC12)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS 790056	DPRC3239	0	2
CIS 790057	DPRC3244	2	5
CIS 790058	DPRC3259	2	5
	Head Grade Assay	(Avg Au g/t)	0.93

Table 13.35 Kekeda Oxide Low Grade Variability Composite Details (VC16)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4749	6	7
	Head Grade Assay	(Avg Au g/t)	3.60

Table 13.36 Kekeda Oxide High Grade Variability Composite Details (VC17)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4744	7	14
	DPDD4747	19	22
	Head Grade Assay	(Avg Au g/t)	0.76

Table 13.37 Kekeda Transitional Low Grade Variability Composite Details (VC18)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4746	4	7
	DPDD4753	21	22
CIS739213	DPDD1557	6	16
	Head Grade Assay	(Avg Au g/t)	2.65

Table 13.38 Kekeda Transitional High Grade Variability Composite Details (VC19)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4749	23	25
CIS 790026	DPDD1436	39	43
CIS 790031	DPDD1600	18	29
CIS 790062	DPRC3250	38	42
CIS 790063	DPRC3255	28	35
	Head Grade Assay	(Avg Au g/t)	0.97

Table 13.39 Kekeda Fresh Low Grade Variability Composite Details (VC20)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4745	25	26
	DPDD4754	52	56
CIS 790069	DPRC3319	79	92
CIS 790030	DPDD1599	67	77
	Head Grade Assay	(Avg Au g/t)	4.39

Table 13.40 Kekeda Fresh High Grade Variability Composite Details (VC21)

Enioda

Three (3) master composites (MC13 - M15) were prepared for the Enioda pit for fresh, transitional and oxide material. Six (6) variability composites were prepared (VC22 - VC27) with all composite information presented in Table 13.41 to Table 13.49.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4768	5	12
	DPDD4769	25	31
	DPDD4770	4	14
	DPDD4771	0	7
	DPDD4772	5	8
	DPDD4777	18	19
	Head Grade Assay	(Avg Au g/t)	1.53
	Resource	Au (g/t)	1.40

Table 13.41 Enioda Oxide Master Composite Details (MC13)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4767	28	36
	DPDD4773	32-42, 46-48	
	DPDD4775	45	49
	DPDD4776	19	23
	Head Grade Assay	(Avg Au g/t)	1.33
	Resource	Au (g/t)	1.37

Table 13.42 Enioda Transitional Master Composite Details (MC14)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4775	54	56
	DPDD4778	53	68
	DPDD4780	71-72, 74-77, 81-84	
	DPDD4781	67	72
	Head Grade Assay	(Avg Au g/t)	1.82
	Resource	Au (g/t)	1.82

Table 13.43 Enioda Fresh Master Composite Details (MC15)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4767	22	24
	DPDD4768	3	5
	Head Grade Assay	(Avg Au g/t)	0.87

Table 13.44 Enioda Oxide Low Grade Variability Composite Details (VC22)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4777	19	22
	Head Grade Assay	(Avg Au g/t)	2.43

Table 13.45 Enioda Oxide High Grade Variability Composite Details (VC23)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS739251	DPDD1571	27	32
	Head Grade Assay	(Avg Au g/t)	0.45

Table 13.46 Enioda Transitional Low Grade Variability Composite Details (VC24)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS 790032	DPDD1604	47	60
	Head Grade Assay	(Avg Au g/t)	3.09

Table 13.47 Enioda Transitional High Grade Variability Composite Details (VC25)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4781	72	77
	Head Grade Assay	(Avg Au g/t)	0.69

Table 13.48 Enioda Fresh Low Grade Variability Composite Details (VC26)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS739241	DPDD1567	69	79
	Head Grade Assay	(Avg Au g/t)	4.17

Table 13.49 Enioda Fresh High Grade Variability Composite (VC27)

Han

Three (3) master composites (MC16 - M18) were prepared for the Han pit for fresh, transitional and oxide material. Four (4) variability composites were prepared (VC28 - VC31) with all composite information presented in Table 13.50 to Table 13.56.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4756	0	2
	Head Grade Assay	(Avg Au g/t)	1.35
	Resource	Au (g/t)	0

Table 13.50 Han Oxide Master Composite Details (MC16)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4756	9-14, 16-18	
	DPDD4757	17	20
	DPDD4758	5	10
	DPDD4759	10-12, 15-17	
	DPDD4760	13	17
	Head Grade Assay	(Avg Au g/t)	2.04
	Resource	Au (g/t)	2.15

Table 13.51 Han Transitional Master Composite Details (MC17)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4761	76-77, 78-85, 88-92	
	DPDD4762	45	55
	DPDD4763	48-52, 53-58	
	DPDD4764	20-21, 22-24	
	DPDD4766	29	31
	Head Grade Assay	(Avg Au g/t)	1.96
	Resource	Au (g/t)	0

Table 13.52 Han Fresh Master Composite Details (MC18)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS739239	DPDD1566	21	31
	Head Grade Assay	(Avg Au g/t)	

Table 13.53 Han Transitional Low Grade Variability Composite Details (VC28)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS739239	DPDD1566	22	31
	Head Grade Assay	(Avg Au g/t)	23.55

Table 13.54 Han Transitional High Grade Variability Composite Details (VC29)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4766	31	37
CIS739235	DPDD1564	65	75.00
	Head Grade Assay	(Avg Au g/t)	0.97

Table 13.55 Han Fresh Low Grade Variability Composite Details (VC30)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS 790073	DPRC3612	24	30
CIS 790076	DPRC3591	29	33
CIS 790077	DPRC3584	21	27
	Head Grade Assay	(Avg Au g/t)	5.04

Table 13.56 Han Fresh High Grade Variability Composite Details (VC31)

Chegue Main

Three (3) master composites (MC19 - MC21) were prepared for the Chegue Main pit for fresh, transitional and oxide material. Six (6) variability composites were prepared (VC32 - VC37) with all composite information presented in Table 13.57 to Table 13.65.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4725	8	10
	DPRC4636	4	4
CIS339826	DPDD1541	0	5
CIS339827	DPDD1541	5	11
	Head Grade Assay	(Avg Au g/t)	1.11
	Resource	Au (g/t)	0.93

Table 13.57 Chegue Main Oxide Master Composite Details (MC19)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4725	31	36
	DPDD4726	13-18, 19-20, 33-36	
	DPDD4728	13-15, 18-20, 26-28	
	DPDD4729	24-37, 42-48	
	DPDD4733	49	50
	DPRC4637	4	5
	Head Grade Assay	(Avg Au g/t)	1.25
	Resource	Au (g/t)	1.01

Table 13.58 Chegue Main Traditional Master Composite Details (MC20)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4730	54	63
	DPDD4731	85	93
	DPDD4732	53	57
	DPDD4733	96	103
	DPDD4734	84	94
	DPDD4735	100-106, 108-110	
	Head Grade Assay	(Avg Au g/t)	1.59
	Resource	Au (g/t)	1.58

Table 13.59 Chegue Main Fresh Master Composite Details (MC21)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS 790014	DPDD1592	0	10
	Head Grade Assay	(Avg Au g/t)	1.26

Table 13.60 Chegue Main Oxide Low Grade Variability Composite Details (VC32)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPRC4636	4	4
	Head Grade Assay	(Avg Au g/t)	5.67

Table 13.61 Chegue Main Oxide High Grade Variability Composite Details (VC33)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS339831	DPDD1541	11	16
CIS 790038	DPDD1591	26	34
	Head Grade Assay	(Avg Au g/t)	0.74

Table 13.62 Chegue Main Transitional Low Grade Composite Details (VC34)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4733	50	51
	DPDD4733	61	63
	Head Grade Assay	(Avg Au g/t)	3.14

Table 13.63 Chegue Main Transitional High Grade Variability Composite Details (VC35)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4733	64	68
	DPDD4733	91	96
CIS339810	DPDD1537	57	67
	Head Grade Assay	(Avg Au g/t)	0.83

Table 13.64 Chegue Main Fresh Low Grade Variability Composite Details (VC36)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4727	15	21
CIS 790012	DPDD1501	84	97
CIS 790037	DPDD1594	45	52
CIS 790053	DPRC3441	60	71
	Head Grade Assay	(Avg Au g/t)	4.10

Table 13.65 Chegue Main Fresh High Grade Variability Composite Details (VC37)

Chegue South

Three (3) master composites (MC22 - MC24) were prepared for the Nokpa pit for fresh, transitional and oxide material. Four (4) variability composites were prepared (VC38 - VC41) with all composite information presented in Table 13.66 to Table 13.72.

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS 790044	DPRC3527	0	6
	Head Grade Assay	(Avg Au g/t)	3.16
	Resource	Au (g/t)	0.93

Table 13.66 Chegue South Oxide Master Composite Details (MC22)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4736	9-11, 22-29	
	DPDD4739	28	30
	DPDD4740	40-41, 42-43	
CIS339842	DPDD1548	35	40
	Head Grade Assay	(Avg Au g/t)	1.63
	Resource	Au (g/t)	1.04

Table 13.67 Chegue South Transitional Master Composite Details (MC23)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4737	61-64, 73-74, 85-87, 90-92	
	DPDD4738	46-47, 49-50	
	DPDD4739	30	33
	DPDD4740	68	75
	DPDD4741	81-82, 83-89	
	Head Grade Assay	(Avg Au g/t)	1.41
	Resource	Au (g/t)	1.34

Table 13.68 Chegue South Fresh Master Composite Details (MC24)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
CIS339838	DPDD1545	20	30
	Head Grade Assay	(Avg Au g/t)	0.74

Table 13.69 Chegue South Transitional Low Grade Variability Composite Details (VC38)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4740	41	42
CIS 790016	DPDD1595	7	20
	Head Grade Assay	(Avg Au g/t)	5.13

Table 13.70 Chegue South Transitional High Grade Variability Composite Details (VC39)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4739	33	35
	DPDD4740	63	68
CIS 790035	DPDD1487	70	72
CIS 790036	DPDD1597	66	68
	Head Grade Assay	(Avg Au g/t)	0.72

Table 13.71 Chegue South Fresh Low Grade Variability Composite Details (VC40)

ALS ID	Drill Hole	Interval	
		From (m)	To (m)
	DPDD4738	52	54
	DPDD4738	81	83
CIS339837	DPDD1544	64.5	74.5
	Head Grade Assay	(Avg Au g/t)	2.90

Table 13.72 Chegue South Fresh High Grade Variability Composite Details (VC41)

13.4.2 Sample Spatial Distribution Indication

The spatial distribution of all the feasibility study metallurgical sampling is shown in Figure 13.2 to Figure 13.12 in relation to the proposed open pits.

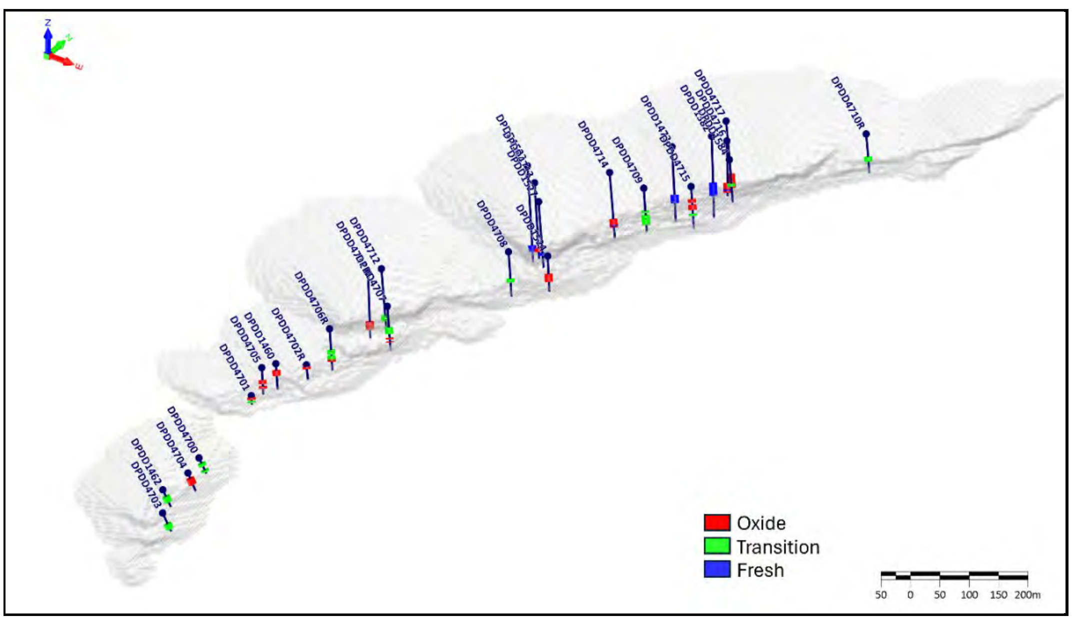


Figure 13.2 Provenance of Lithology Composites from the Souwa Open Pit

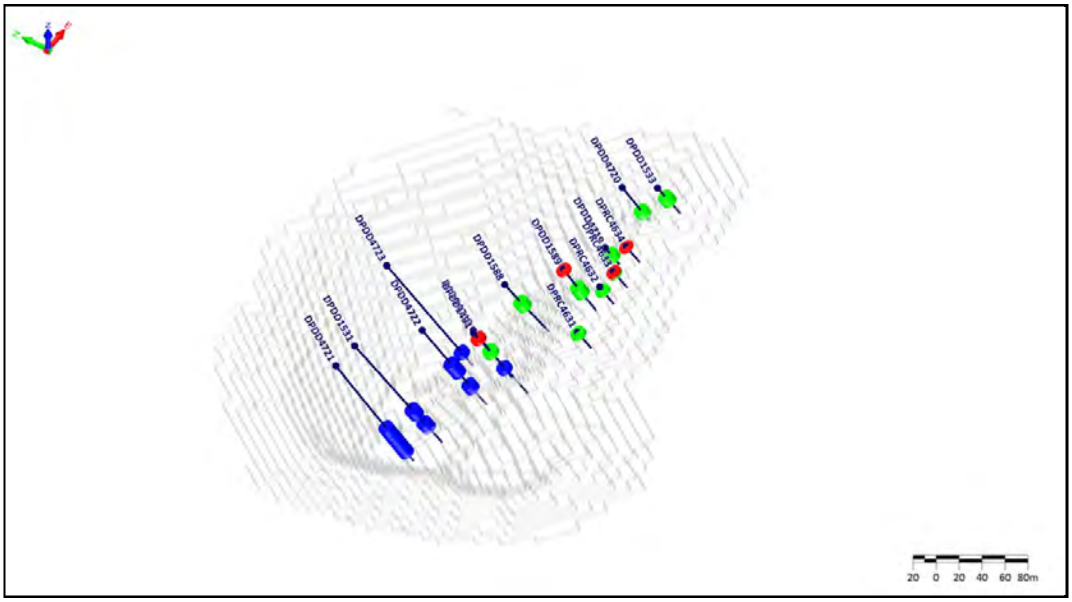


Figure 13.3 Provenance of Lithology Composites from the Nokpa Open Pit

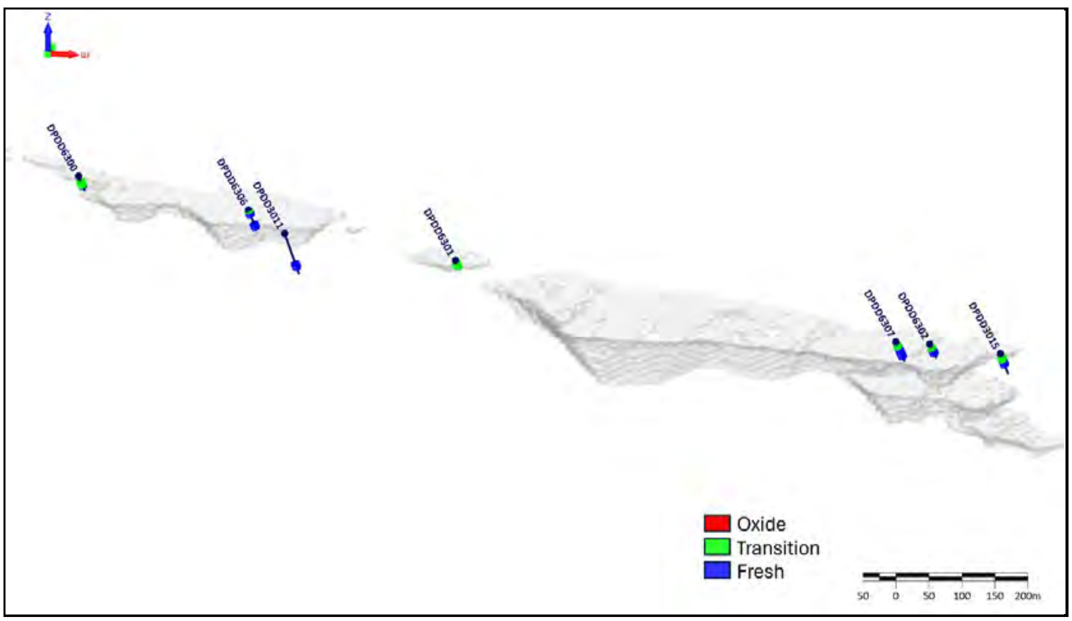


Figure 13.4 Provenance of Lithology Composites from the Kilosegui Open Pit (NW Section)

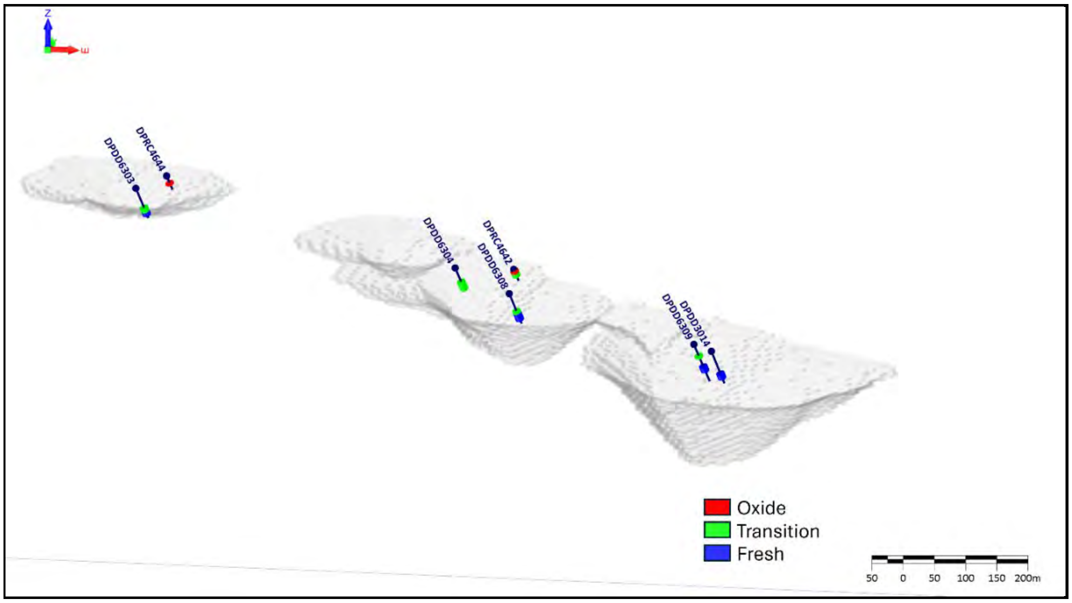


Figure 13.5 Provenance of Lithology Composites from the Kilosegui Open Pit (Central Section)

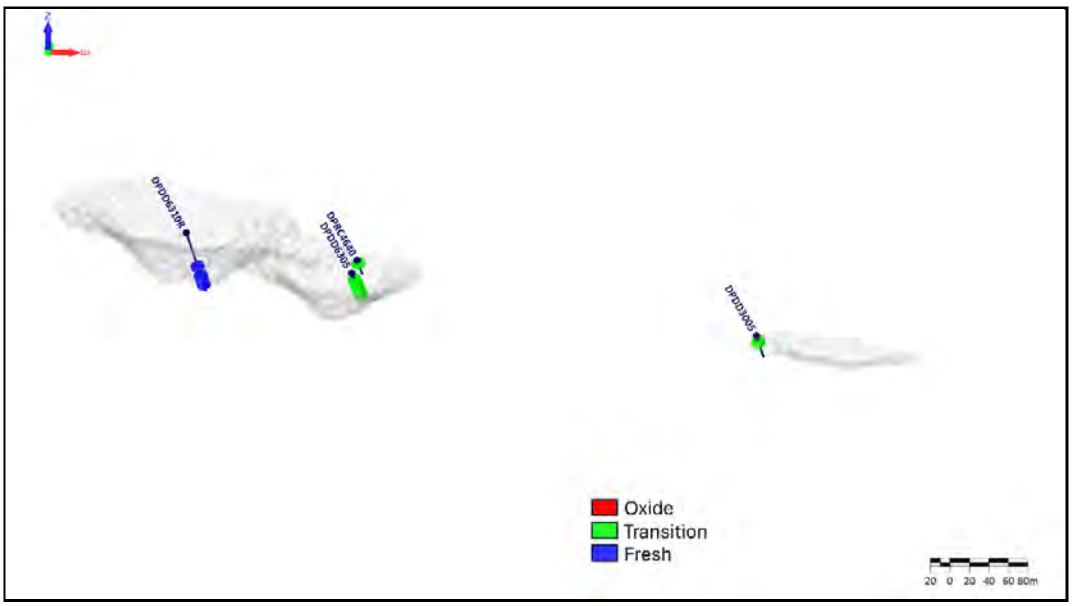


Figure 13.6 Provenance of Lithology Composites from the Kilosegui Open Pit (SE Section)

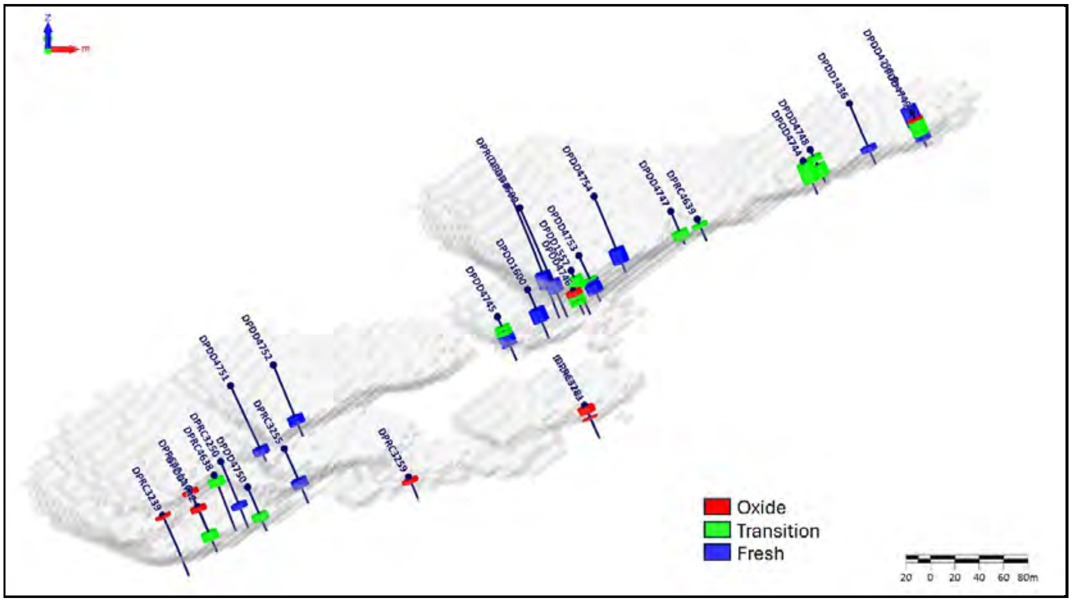


Figure 13.7 Provenance of Lithology Composites from the Kekeda Open Pit

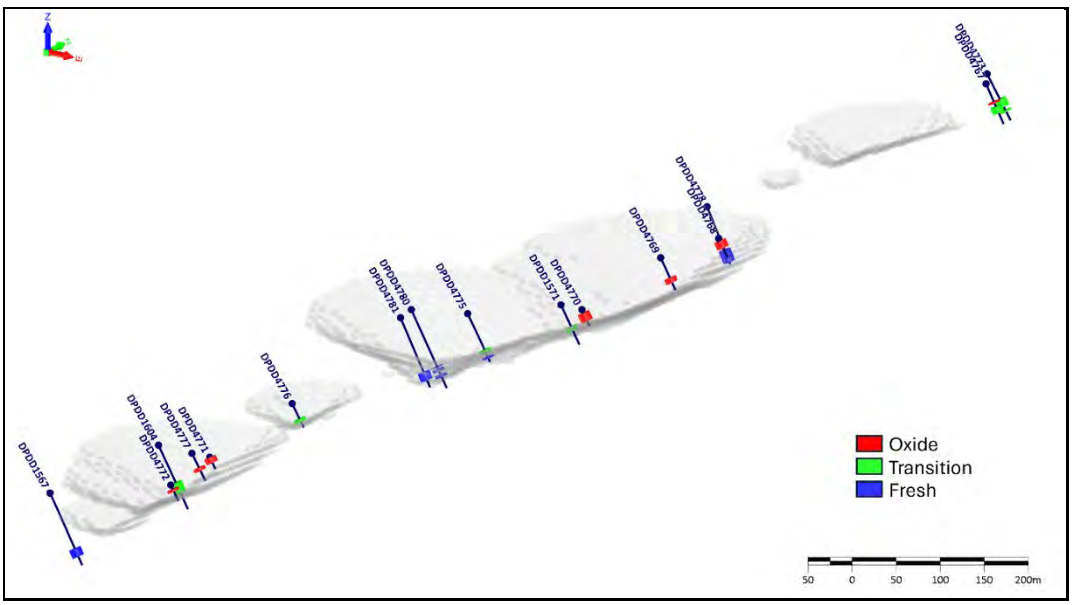


Figure 13.8 Provenance of Lithology Composites from the Enioda Open Pit

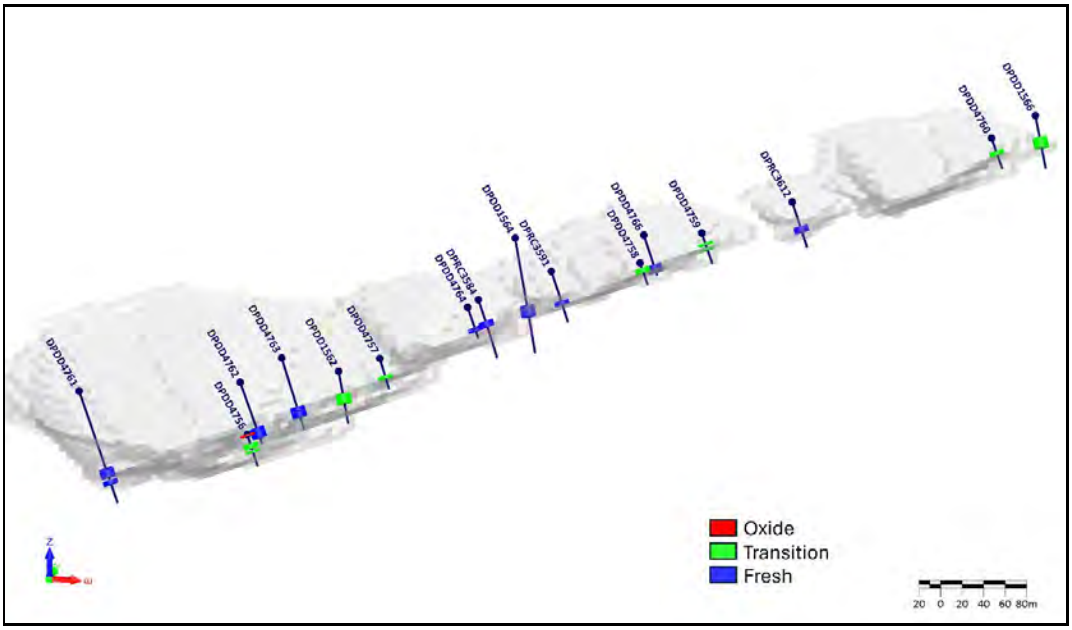


Figure 13.9 Provenance of Lithology Composites from the Han Open Pit

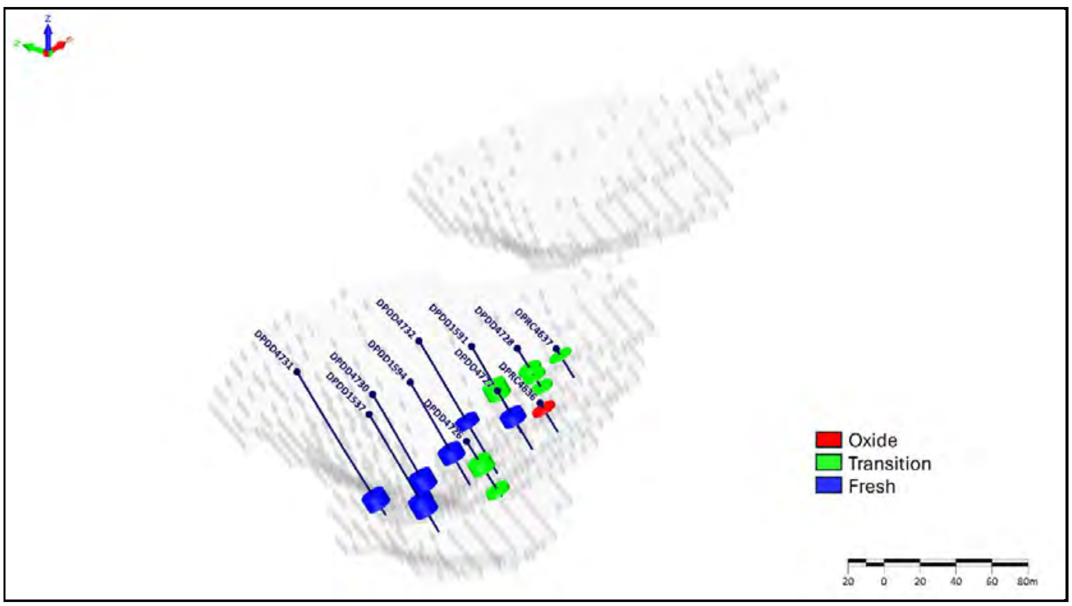


Figure 13.10 Provenance of Lithology Composites from the Chegue Main

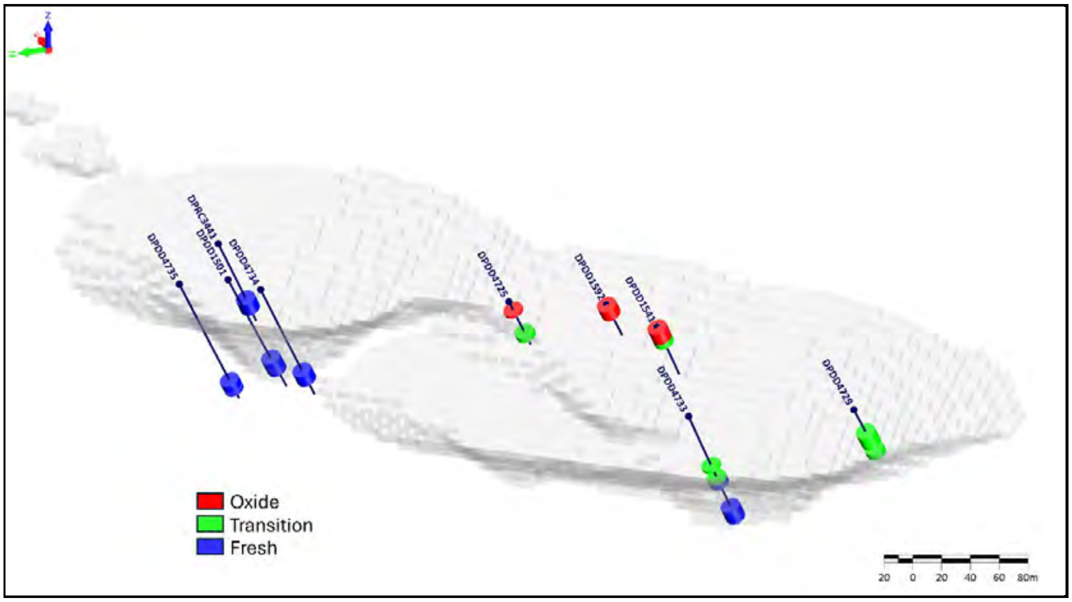


Figure 13.11 Provenance of Lithology Composites from the Chegue Main (North)

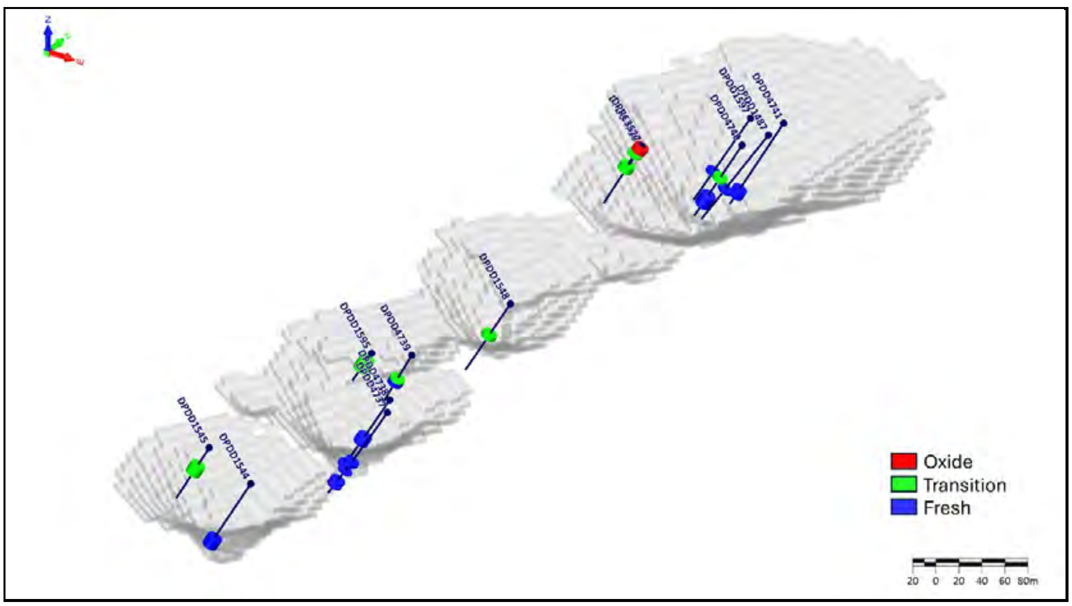


Figure 13.12 Provenance of Lithology Composites from the Chegue South

13.5 Head Assay Analysis

13.5.1 Master Composite Analysis

The master composite samples were analysed using the following methods:

- Duplicate fire assay;
- Total sulphur and carbon by LECO;
- 1 kg bottle roll via LeachWELL©;
- Multi-element ICP assay; and
- S.G via Pycnometer.

Replicate gold fire assay, screen fire assay and multi-element head analyses were completed on 24 master composite samples of oxide, transition and fresh material from the eight deposits.

Key analytes are presented in Table 13.73 to Table 13.76 and summarised based on the following observations:

- Samples reported gold head grades via fire assay in a range from 0.74 g/t to 3.70 g/t Au and averaged 1.63 g/t Au. The replicate gold assays for a number of composites varied significantly which is consistent with the PFS Master composites supporting the presence of coarse gold;
- The samples were assayed using LeachWELL©, which require 1kg charges, and reported significant variability when compared to average duplicate fire assay results:
  - Souwa (MC1) Oxide sample at 6.36 g/t Au by LeachWELL© versus 3.70 g/t Au by FA;
  - Nokpa (MC6) Fresh sample 1.35 g/t Au by LeachWELL© versus 2.37 g/t Au by FA;
  - Kilosegui (MC8) Transitional sample 1.19 g/t Au by LeachWELL© versus 1.80 g/t Au by FA;
  - Han (MC16) Oxide sample 2.66 g/t Au by Leach WELL versus 1.96 g/t by FA;
- To improve the reproducibility of the test work and minimise the impact of the coarse gold a gravity concentration step was adopted for all test work in the FS programme;
- The LeachWELL© analyses were conducted using a cyanide strength of 10 kg/t and reported the following average cyanide solubilities for the three lithology types across the eight (8) open pits:
  - Oxide composites - 97.2%;
  - Trans composites - 97.2%;
  - Fresh composites - 94.3%;
- Souwa (MC1) Oxide composite reported the highest gold head grade by FA with an average of 3.70 g/t Au. The Kilosegui (MC7) and Chegue Main (MC19) Oxide composites reported the lowest head assays at 0.68 g/t Au & 0.74 g/t Au respectively;

- The silver head grades reported were consistent with the PFS composite head assays and reported low grades at or under the 2.0 g/t Ag detection limit. The Han (MC16) and Enioda (MC13) Oxide composites were exceptions reporting elevated silver head grades at 12 g/t Ag and 6 g/t Ag respectively;
- Deleterious elements mercury, arsenic and antimony are low in the composites and should not present an environmental or occupational health risk in the elution or electrowinning circuits;
- The Oxide and Transition composites reported very low or below detection sulphide sulphur assays. Fresh composites reported sulphide sulphur levels in a range of 0.24% - 0.62%;
- The Nokpa (MC4) and Han (MC16) Oxide composites and Chegue Main (MC21) and Chegue South (MC24) Fresh composites reported elevated organic carbon levels at 0.42%, 0.78%, 0.32% and 0.24% respectively. All other composites reported low levels (<0.03% C_{org}) and preg-robbing due to the presence of organic carbon is not expected to occur;
- Cyanide consuming elements, Cu, Zn, Pb and Ni and Sb were reported at similar ranges to the PFS composites and are not expected to impact on the circuit;
- All Master composites reported varying grades of Tellurium at an average 2.54 g/t. The presence of Tellurides at these grades warranted monitoring during subsequent test work. However, there was no measurable correlation observed between Te grades and Au cyanide solubility and therefore is not expected to be a significant metallurgical issue.

Analyte	Unit	Souwa			Nokpa		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC1	MC2	MC3	MC4	MC5	MC6
Au_1	g/t	3.97	1.51	1.15	1.41	2.68	2.32
Au_2	g/t	3.43	1.24	1.21	1.17	2.93	2.41
Au_Av	g/t	3.70	1.38	1.18	1.29	2.81	2.37
Au LeachWELL®	g/t	6.36	1.68	1.28	1.27	2.75	1.35
CN soluble %	%	96.8	97.7	93.4	96.2	96.8	96.4
Ag	g/t	<2	<2	4	<2	<2	<2
As	ppm	<10	<10	<10	<10	<10	<10
C Total	%	<0.03	0.06	0.66	0.42	0.15	0.48
C Organic	%	<0.03	<0.03	<0.03	0.42	0.12	<0.03
Cu	ppm	30	18	26	8	22	30
Hg	ppm	0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	45	15	10	15	15	10
Pb	ppm	15	20	15	5	<5	20
S Total	%	0.02	0.08	0.66	<0.02	<0.02	0.42
S Sulphide	%	<0.02	0.04	0.60	<0.02	<0.02	0.28
Sb	ppm	0.20	<0.1	0.30	0.10	<0.1	<0.1
Te	ppm	5.4	2.4	3.0	0.6	4.0	3.4
Zn	ppm	48	50	48	20	44	38
SG	kg/m^3	2,684	2,710	2,732	2,642	2,679	2,722

Table 13.73 Head Assay Analysis MC1 - MC6

Analyte	Unit	Kilosegui			Kekeda		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC7	MC8	MC9	MC10	MC11	MC12
Au_1	g/t	0.65	1.70	1.19	1.21	0.90	1.88
Au_2	g/t	0.71	1.89	1.21	1.47	0.80	1.87
Au_Av	g/t	0.68	1.80	1.20	1.34	0.85	1.88
Au LeachWELL®	g/t	0.69	1.19	1.19	1.13	1.09	1.92
CN soluble %	%	98.6	96.0	86.9	93.8	97.3	96.5
Ag	g/t	<2	<2	<2	4	<2	<2
As	ppm	<10	<10	10	<10	<10	<10
C Total	%	<0.03	0.15	0.45	0.06	<0.03	0.36
C Organic	%	<0.03	<0.03	<0.03	0.06	<0.03	<0.03
Cu	ppm	18	20	12	54	26	16
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	25	10	5	40	10	<5
Pb	ppm	<5	5	<5	10	95	30
S Total	%	<0.02	0.1	0.34	<0.02	0.02	0.68
S Sulphide	%	<0.02	0.06	0.30	<0.02	<0.02	0.62
Sb	ppm	<0.1	0.30	0.10	0.10	<0.1	<0.1
Te	ppm	0.4	0.6	0.4	4.0	2.2	3.6
Zn	ppm	42	46	38	28	36	40
SG	kg/m^3	2,693	2,690	2,695	2,688	2,696	2,726

Table 13.74 Head Assay Analysis MC7 - MC12

Analyte	Unit	Enioda			Han		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC13	MC14	MC15	MC16	MC17	MC18
Au_1	g/t	1.39	1.00	1.34	1.79	2.34	1.27
Au_2	g/t	1.73	0.95	1.26	2.12	1.34	1.30
Au_Av	g/t	1.56	0.98	1.30	1.96	1.84	1.29
Au LeachWELL®	g/t	1.53	1.37	1.36	2.66	1.44	1.76
CN soluble %	%	98.7	96.8	91.9	98.9	98.0	96.7
Ag	g/t	6	2	<2	12	2	4
As	ppm	<10	<10	<10	<10	<10	<10
C Total	%	<0.03	0.48	0.90	0.99	0.03	0.9
C Organic	%	<0.03	<0.03	0.06	0.78	<0.03	0.06
Cu	ppm	98	78	28	40	16	12
Hg	ppm	<0.1	0.2	0.2	0.3	0.2	0.2
Ni	ppm	25	45	5	10	<5	10
Pb	ppm	20	30	10	15	80	30
S Total	%	<0.02	0.2	0.52	<0.02	0.14	0.66
S Sulphide	%	<0.02	0.12	0.44	<0.02	0.08	0.50
Sb	ppm	0.10	0.40	0.20	0.20	0.20	0.30
Te	ppm	2.2	5.0	1.0	2.4	3.0	3.6
Zn	ppm	56	84	40	36	30	46
SG	kg/m^3	2,711	2,760	2,781	2,660	2,753	2,807

Table 13.75 Head Assay Analysis MC13 - MC18

Analyte	Unit	Chegue Main			Chegue South		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
		MC19	MC20	MC21	MC22	MC23	MC24
Au_1	g/t	0.64	0.86	1.75	2.56	1.62	1.60
Au_2	g/t	0.83	0.98	1.46	3.89	1.45	1.76
Au_Av	g/t	0.74	0.92	1.61	3.23	1.54	1.68
Au LeachWELL®	g/t	0.81	1.17	1.35	2.55	1.18	1.41
CN soluble %	%	97.6	96.7	95.1	98.3	98.3	97.2
Ag	g/t	2	<2	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10	<10	<10
C Total	%	<0.02	0.06	0.42	<0.02	0.08	0.30
C Organic	%	<0.02	0.04	0.32	<0.02	0.04	0.24
Cu	ppm	14	10	22	12	14	12
Hg	ppm	0.1	<0.1	0.1	0.2	<0.1	<0.1
Ni	ppm	10	<5	<5	5	<5	<5
Pb	ppm	35	95	60	10	<5	15
S Total	%	<0.02	0.06	0.42	<0.02	0.08	0.3
S Sulphide	%	<0.02	0.04	0.32	<0.02	0.04	0.24
Sb	ppm	0.20	<0.1	0.10	0.20	<0.1	<0.1
Te	ppm	1.4	2.6	3.2	2.0	2.4	2.2
Zn	ppm	28	42	28	38	48	46
SG	kg/m^3	2,781	2,751	2,603	2,773	2,782	2,811

Table 13.76 Head Assay Analysis MC19 - MC24

13.5.2 Variability Composite Analysis

The variability composite samples were analysed using the following methods:

- Duplicate fire assay;
- Total sulphur and carbon by LECO;
- 1 kg bottle roll via LeachWELL©;
- Multi-element ICP assay; and
- S.G via Pycnometer.

Replicate gold fire assay, screen fire assay and multi-element head analyses were completed on 41 variability composite samples of oxide, transition and fresh material from the eight deposits are presented in Table 13.77 to Table 13.84.

The following comments can be made in reference to these results:

- The twenty (20) low-grade variability composites assayed for gold via fire assay reported an average gold head grade of 0.78 g/t Au;
- Five (5) low-grade variability composites reported gold head grades above 1.0 g/t Au:
  - Souwa Trans 1.02 g/t Au;
  - Nokpa Oxide 1.06 g/t Au;
  - Nokpa Trans 1.37 g/t Au;
  - Kekeda Fresh 1.24 g/t Au;
  - Enioda Oxide 1.08 g/t Au;
- The remaining fifteen (15) low-grade composites reported an average gold head grade at 0.64 g/t Au;
- The twenty-one (21) high-grade variability composites assayed for gold via fire assay reported an average gold head grade of 4.66 g/t Au;
- The silver head grades reported were consistent with the Master composite head assays and reported low grades at or under the 2.0 g/t Ag detection limit, with the exception of the Low Grade Enioda Oxide variability composite, which reported elevated silver and copper grades; 24 g/t Ag & 102 g/t or 0.102% Cu. Silver extraction and copper cyanide solubility was monitored in the subsequent test work however, because the head grades were not high, silver is not expected to be a significant metallurgical issue; and
- All Variability composites reported varying grades of Tellurium at an average 3.93 g/t. The presence of Tellurides at these grades was monitored in the subsequent test work however, because the head grades were not high, silver is not expected to be a significant metallurgical issue.

Analyte	Unit	Souwa					
		Oxide		Trans		Fresh	
		LG	HG	LG	HG	LG	HG
		VC1	VC2	VC3	VC4	VC5	VC6
Au_1	g/t	0.72	3.83	1.02	3.98	0.76	2.31
Au_2	g/t	0.65	3.18	1.01	3.64	0.94	2.57
Au_Av	g/t	0.69	3.51	1.02	3.81	0.85	2.44
Au LeachWELL®	g/t	0.71	1.97	1.41	3.93	0.13	2.46
CNsolAu	ppm	0.63	1.94	1.34	3.77	0.07	2.31
CNresAu	g/t	0.08	0.03	0.07	0.16	0.06	0.15
CN soluble %	%	88.7	98.5	95.0	95.9	52.7	93.9
Ag	g/t	<2	2	<2	<2	<2	<2
As	ppm	<10	<10	10	<10	<10	<10
C Total	%	<0.03	<0.03	0.51	<0.03	0.63	0.39
C Organic	%	0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Cu	ppm	40	30	16	46	30	20
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	10	10	10	10	10	10
Pb	ppm	<5	80	<5	<5	<5	<5
S Total	%	<0.02	<0.02	0.56	<0.02	0.78	1.66
S Sulphide	%	<0.02	<0.02	0.42	<0.02	0.68	1.44
Sb	ppm	1.7	0.7	0.5	0.5	0.4	0.3
Te	ppm	0.60	4.40	1.60	5.00	1.40	3.80
Zn	ppm	60	12	34	60	32	20
SG	kg/m^3	2592	2675	2399	2687	2720	2743

Table 13.77 Souwa Variability Head Assay Analysis VC1 - VC6

Analyte	Unit	Nokpa				
		Oxide	Trans		Fresh	
		LG	LG	HG	LG	HG
		VC7	VC8	VC9	VC10	VC11
Au_1	g/t	1.02	1.38	1.92	0.85	3.11
Au_2	g/t	1.09	1.35	2.12	0.85	3.12
Au_Av	g/t	1.06	1.37	2.02	0.85	3.12
Au LeachWELL®	g/t	1.09	0.87	1.68	0.86	3.53
CNsolAu	ppm	1.09	0.87	1.68	0.86	3.53
CNresAu	g/t	0.08	0.02	0.05	0.02	0.14
CN soluble %	%	93.1	97.8	97.1	97.7	96.2
Ag	g/t	<2	<2	<2	<2	2
As	ppm	<10	<10	<10	<10	<10
C Total	%	0.12	<0.03	<0.03	0.45	0.39
C Organic	%	0.09	<0.03	<0.03	<0.03	<0.03
Cu	ppm	18	30	24	40	8
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	15	15	30	25	10
Pb	ppm	<5	<5	15	<5	<5
S Total	%	<0.02	0.02	0.1	0.26	0.44
S Sulphide	%	<0.02	<0.02	0.06	0.16	0.38
Sb	ppm	0.4	0.2	0.2	0.3	1.7
Te	ppm	0.80	1.40	3.80	1.40	5.60
Zn	ppm	22	38	20	44	62
SG	kg/m^3	2639	2670	2673	2732	2734

Table 13.78 Nokpa Variability Head Assay Analysis VC7 - VC11

Analyte	Unit	Kilosegui			
		Trans		Fresh	
		LG	HG	LG	HG
		VC12	VC13	VC14	VC15
Au_1	g/t	0.52	1.80	0.79	2.44
Au_2	g/t	0.6	2.19	0.82	2.38
Au_Av	g/t	0.56	2.00	0.81	2.41
Au LeachWELL®	g/t	0.54	1.90	0.76	2.58
CNsolAu	ppm	0.54	1.83	0.70	2.45
CNresAu	g/t	0.03	0.07	0.06	0.13
CN soluble %	%	94.7	96.3	92.1	95.0
Ag	g/t	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10
C Total	%	0.21	0.03	0.39	0.57
C Organic	%	<0.03	<0.03	<0.03	<0.03
Cu	ppm	14	12	6	8
Hg	ppm	<0.1	<0.1	<0.1	<0.1
Ni	ppm	5	10	<5	10
Pb	ppm	<5	<5	<5	<5
S Total	%	<0.02	0.04	0.26	0.32
S Sulphide	%	<0.02	<0.02	0.16	0.20
Sb	ppm	3.7	0.2	0.3	0.3
Te	ppm	0.20	0.60	0.20	0.60
Zn	ppm	46	28	34	22
SG	kg/m³	2696	2524	2696	2738

Table 13.79 Kilosegui Variability Head Assay Analysis VC12 - VC15

Analyte	Unit	Kekeda				
		Oxide	Trans		Fresh	
		HG	LG	HG	LG	HG
		VC17	VC18	VC19	VC20	VC21
Au_1	g/t	10.40	0.48	1.66	1.47	3.84
Au_2	g/t	8.82	0.48	1.81	1.01	3.80
Au_Av	g/t	9.61	0.48	1.74	1.24	3.82
Au LeachWELL®	g/t	9.30	0.49	2.07	0.71	4.41
CNsolAu	ppm	8.93	0.47	2.03	0.69	4.38
CNresAu	g/t	0.37	0.02	0.04	0.02	0.03
CN soluble %	%	96.0	95.9	98.1	97.2	99.3
Ag	g/t	<2	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10	<10
C Total	%	<0.03	<0.03	<0.03	0.60	0.27
C Organic	%	<0.03	<0.03	<0.03	0.03	<0.03
Cu	ppm	14	10	20	12	8.0
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	5	<5	10	5	<5
Pb	ppm	70	<5	295	<5	75
S Total	%	<0.02	0.04	<0.02	0.24	0.36
S Sulphide	%	<0.02	0.02	<0.02	0.16	0.32
Sb	ppm	0.3	0.3	0.2	0.20	0.20
Te	ppm	10.20	1.40	6.40	1.80	5.80
Zn	ppm	50	16	40	62	52
SG	kg/m^3	2738	2686	2689	2720	2703

Table 13.80 Kekeda Variability Head Assay Analysis VC17 - VC21

Analyte	Unit	Enioda					
		Oxide		Trans		Fresh	
		LG	HG	LG	HG	LG	HG
		VC22	VC23	VC24	VC25	VC26	VC27
Au_1	g/t	0.99	0.83	0.48	4.76	0.93	7.43
Au_2	g/t	1.16	0.90	0.54	5.25	0.64	13.00
Au_Av	g/t	1.08	0.76	0.51	5.01	0.79	10.22
Au LeachWELL®	g/t	0.85	2.14	0.57	4.89	0.74	6.52
CNsolAu	ppm	0.83	2.12	0.53	4.82	0.66	6.36
CNresAu	g/t	0.02	0.02	0.04	0.07	0.08	0.16
CN soluble %	%	97.7	99.1	92.9	98.6	89.2	97.5
Ag	g/t	24	<2	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10	10.0	<10
C Total	%	0.09	<0.03	0.03	<0.03	0.75	0.87
C Organic	%	0.06	<0.03	<0.03	<0.03	<0.03	<0.03
Cu	ppm	102	14	28	44	24	18
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ni	ppm	30	5	15	25	15	10
Pb	ppm	<5	480	<5	190	<5	35
S Total	%	<0.02	<0.02	<0.02	0.06	0.68	0.50
S Sulphide	%	<0.02	<0.02	<0.02	<0.02	0.52	0.40
Sb	ppm	0.3	0.10	0.40	0.10	0.20	0.30
Te	ppm	1.80	1.40	1.00	8.80	1.20	5.80
Zn	ppm	72	34	38	66	30	62
SG	kg/m^3	2687	2678	2704	2680	2710	2739

Table 13.81 Enioda Variability Head Assay Analysis VC22 - VC27

Analyte	Unit	Han			
		Trans		Fresh	
		LG	HG	LG	HG
		VC28	VC29	VC30	VC31
Au_1	g/t	0.62	20.30	0.45	5.03
Au_2	g/t	0.53	23.70	0.45	7.81
Au_Av	g/t	0.58	22.00	0.45	6.42
Au LeachWELL®	g/t	0.66	22.60	0.55	5.62
CNsolAu	ppm	0.63	22.50	0.52	5.52
CNresAu	g/t	0.03	0.10	0.03	0.10
CN soluble %	%	95.5	99.6	94.5	98.2
Ag	g/t	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10
C Total	%	<0.03	<0.03	0.03	0.57
C Organic	%	<0.03	<0.03	<0.03	<0.03
Cu	ppm	14	32	10	114.00
Hg	ppm	<0.1	<0.1	<0.1	0.10
Ni	ppm	<5	<5	<5	10.00
Pb	ppm	20	370	<5	220.00
S Total	%	0.48	0.32	0.62	0.52
S Sulphide	%	0.38	0.18	0.30	0.44
Sb	ppm	0.20	0.20	0.10	0.90
Te	ppm	1.60	29.40	1.20	7.60
Zn	ppm	42	26	42	54.00
SG	kg/m^3	2720	2702	2709	2728

Table 13.82 Han Variability Head Assay Analysis VC28 - VC31

Analyte	Unit	Chegue Main				
		Oxide	Trans		Fresh	
		HG	LG	HG	LG	HG
		VC33	VC34	VC35	VC36	VC37
Au_1	g/t	3.45	0.92	2.31	0.46	1.96
Au_2	g/t	4.54	0.79	3.10	0.51	2.45
Au_Av	g/t	4.00	0.86	2.71	0.49	2.21
Au LeachWELL®	g/t	4.75	0.74	2.03	0.51	3.41
CNsolAu	ppm	4.68	0.72	2.00	0.47	3.39
CNresAu	g/t	0.07	0.02	0.03	0.04	0.02
CN soluble %	%	98.5	97.3	98.5	92.1	99.4
Ag	g/t	<2	<2	<2	<2	<2
As	ppm	<10	<10	<10	<10	<10
C Total	%	0.03	<0.03	0.27	0.99	<0.03
C Organic	%	0.03	<0.03	<0.03	<0.03	<0.03
Cu	ppm	26.00	22.00	16.00	14.00	10.00
Hg	ppm	0.10	0.40	<0.1	0.10	<0.1
Ni	ppm	20.00	5.00	5.00	<5	<5
Pb	ppm	45.00	20.00	20.00	10.00	30.00
S Total	%	<0.02	0.04	0.08	0.32	<0.02
S Sulphide	%	<0.02	0.02	0.06	0.32	<0.02
Sb	ppm	0.30	116.00	0.30	0.20	0.10
Te	ppm	2.20	16.40	2.80	1.00	3.60
Zn	ppm	14.00	28.00	30.00	34.00	16.00
SG	kg/m^3	2683	2717	2539	2728	2687

Table 13.83 Chegue Main Variability Head Assay Analysis VC32 - VC37

Analyte	Unit	Chegue South			
		Trans		Fresh	
		LG	HG	LG	HG
		VC38	VC39	VC40	VC41
Au_1	g/t	0.39	1.88	0.67	3.60
Au_2	g/t	0.45	2.04	0.69	3.40
Au_Av	g/t	0.42	1.96	0.68	3.50
Au LeachWELL®	g/t	0.51	3.73	0.82	2.99
CNsolAu	ppm	0.50	3.69	0.78	2.93
CNresAu	g/t	0.01	0.04	0.04	0.06
CN soluble %	%	98.0	98.9	95.1	98.0
Ag	g/t	<2	<2	<2	4.00
As	ppm	<10	<10	<10	<10
C Total	%	0.18	0.21	0.63	0.42
C Organic	%	<0.03	0.03	<0.03	<0.03
Cu	ppm	8.00	14.00	8.00	14.00
Hg	ppm	<0.1	0.10	<0.1	<0.1
Ni	ppm	5.00	5.00	5.00	<5
Pb	ppm	<5	5.00	<5	<5
S Total	%	0.06	0.06	0.32	0.36
S Sulphide	%	0.04	0.04	0.32	0.36
Sb	ppm	0.30	0.10	<0.1	<0.1
Te	ppm	0.80	3.20	1.20	5.20
Zn	ppm	42.00	34.00	54.00	38.00
SG	kg/m^3	2706	2737	2736	2719

Table 13.84 Chegue South Variability Head Assay Analysis VC38 - VC41

13.6 Mineralogy

13.6.1 Master Composite Mineralogy

Master composite mineralogy was characterised by QEMSCAN and XRD; in each case samples were ground to a P_{80} size of 50 µm. Twenty-four samples were selected for mineralogical analysis by QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) and XRD (X-ray diffraction, for mineral speciation only).

Master composite QEMSCAN model mineralogy is presented below in Figure 13.13 and in Table 13.85 to Table 13.88 and summarised as follows:

- Relatively high carbonate content was detected in the eight 'Fresh' samples (MC-3, MC-6, MC-9, MC-12, MC-15, MC-18, MC-21, and MC-24). Calcite was observed in all the 'Fresh' composites and intermediate ankerite-dolomite was most abundant in MC-15 and MC-18. Of the Transition samples MC-14 was the only composite containing a significant proportion of carbonate, largely as 'ankerite-dolomite';
- Pyrite was detected reporting between 0.9% and 1.63% by mass in the Fresh samples and measured less than 0.5% in the 'Transition' samples, and less than 0.1% in 'Oxide' samples. Trace amounts of complex, poorly characterised Fe-sulphate phases that occur in the 'Oxide' and 'Transition' samples, are likely to include partially oxidised pyrite and the products of more extensive oxidation such as melanterite;
- A higher proportion of clay minerals was evident in the eight 'Oxide' samples (MC-1, MC-4, MC-7, MC-10, MC-13, MC-16, MC-19 and MC 22). The XRD data indicates that these are predominately smectite-type minerals with variable Fe composition, as indicated by the QEMSCAN spectra. Three main types of clay minerals were defined, and the Fe-rich clay minerals were observed to be higher in some samples (for example, MC-11 and MC-22) than in others (for example, MC-7 and MC-19);
- The model chart below illustrates that the goethite/limonite content is low, with an average of 1% across the 24 samples to a maximum of 3%;
- Higher levels of K-Feldspar between 5 and 8% were identified in the Kilosegui deposit samples (MC-7, MC-8, and MC-9, compared to the other composites containing less than less than 0.5% K-feldspar;
- Quartz was reported between 24 and 71% of the sample mass with an average of 43%;
- The Na-rich plagioclase (albite) on average, accounted for 28% of the sample mass and was lower in the 'Oxide' samples compared to the 'Transition' samples, probably due to the alteration of plagioclase to clay minerals in the more weathered zones;
- No native gold particles were detected.

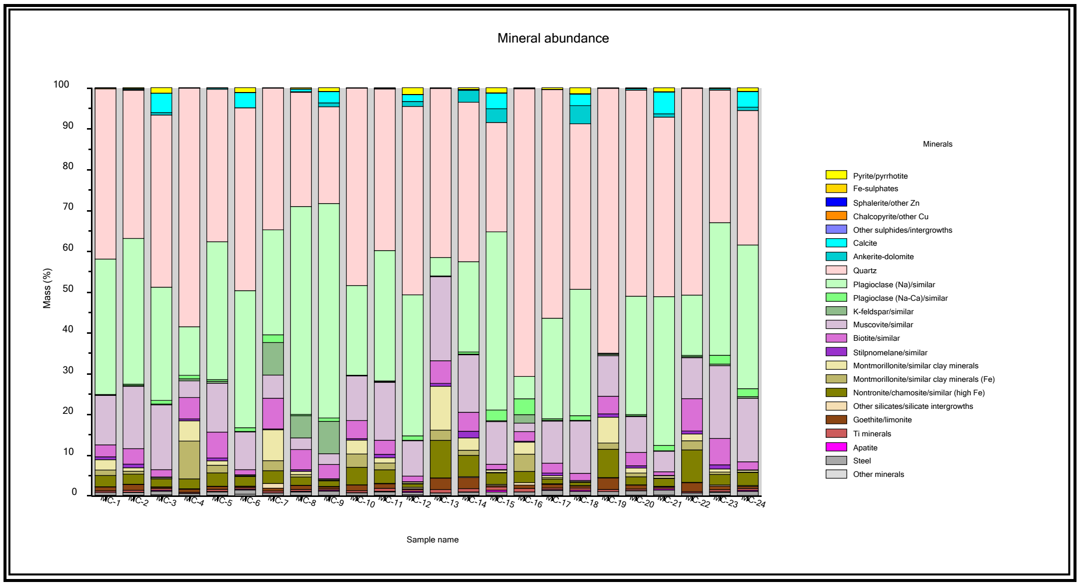


Figure 13.13 Master Composite - Model Mineralogy

Mineral group	Composite Model Mass %					
	Souwa			Nokpa		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-1	MC-2	MC-3	MC-4	MC-5	MC-6
Pyrite/pyrrhotite	0.10	0.20	1.34	0.05	0.11	1.15
Fe-sulphates	0.14	0.08	0.02	0.01	0.01	0.02
Sphalerite/other Zn	0.01	0.01	0.01	0.00	0.01	0.01
Calcite	0.02	0.26	4.74	0.01	0.25	3.72
Quartz	41.7	36.3	42.2	58.5	37.4	44.8
Plagioclase (Na)/similar	33.2	35.7	27.7	11.9	33.8	33.6
Plagioclase (Na-Ca)/similar	0.11	0.32	0.89	0.86	0.42	0.91
K-feldspar/similar	0.13	0.22	0.19	0.49	0.47	0.14
Muscovite/similar	12.1	15.3	15.9	4.09	12.0	9.23
Biotite/similar	2.93	3.80	1.79	5.24	6.35	1.26
Stilpnomelane/similar	0.70	0.88	0.09	0.45	0.68	0.11
Montmorillonite/similar clay minerals	2.54	0.85	0.28	4.98	1.09	0.21
Montmorillonite/similar clay minerals (Fe)	1.33	0.71	0.07	9.26	1.88	0.08
Nontronite/chamosite/similar (high Fe)	2.77	2.48	1.98	2.42	3.27	2.38
Goethite/limonite	0.97	1.40	0.31	0.75	0.67	0.27
Other minerals	1.25	1.53	2.45	0.98	1.69	2.14
TOTAL	100	100	100	100	100	100

Table 13.85 Master Composites Souwa & Nokpa - Model Mineralogy

Mineral group	Composite Model Mass %					
	Kilosegui			Kekeda		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-7	MC-8	MC-9	MC-10	MC-11	MC-12
Pyrite/pyrrhotite	0.05	0.27	0.90	0.06	0.12	1.63
Fe-sulphates	0.01	0.02	0.01	0.03	0.09	0.01
Sphalerite/other Zn	0.01	0.01	0.01	0.01	0.01	0.02
Calcite	0.03	0.68	2.78	0.01	0.05	1.68
Quartz	34.7	28.0	23.7	48.4	39.6	46.2
Plagioclase (Na)/similar	25.7	50.9	52.6	22.0	31.9	34.6
Plagioclase (Na-Ca)/similar	1.88	0.31	0.83	0.08	0.17	1.07
K-feldspar/similar	7.98	5.46	7.91	0.10	0.17	0.11
Muscovite/similar	5.68	2.85	2.64	10.9	14.2	8.67
Biotite/similar	7.52	4.93	3.50	4.44	3.46	1.37
Stilpnomelane/similar	0.19	0.39	0.11	0.36	0.82	0.09
Montmorillonite/similar clay minerals	7.58	0.80	0.25	3.40	1.30	0.34
Montmorillonite/similar clay minerals (Fe)	2.45	0.61	0.12	3.26	1.65	0.11
Nontronite/chamosite/similar (high Fe)	3.14	2.02	1.40	4.30	3.33	0.83
Goethite/limonite	0.71	0.96	0.48	1.38	1.12	0.16
Other minerals	2.32	1.77	2.79	1.33	1.96	3.16
TOTAL	100	100	100	100	100	100

Table 13.86 Master Composites Kilosegui & Kekeda - Model Mineralogy

Mineral group	Composite Model Mass %					
	Enioda			Han		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-13	MC-14	MC-15	MC-16	MC-17	MC-18
Pyrite/pyrrhotite	0.08	0.40	1.24	0.07	0.39	1.45
Fe-sulphates	0.04	0.11	0.02	0.02	0.03	0.02
Sphalerite/other Zn	0.01	0.01	0.01	0.01	0.01	0.01
Calcite	0.02	0.09	3.88	0.10	0.03	2.83
Quartz	41.4	39.1	26.8	70.5	56.0	40.6
Plagioclase (Na)/similar	4.52	22.1	43.7	5.47	24.6	31.0
Plagioclase (Na-Ca)/similar	0.05	0.51	2.62	3.89	0.38	1.14
K-feldspar/similar	0.12	0.16	0.21	2.07	0.20	0.17
Muscovite/similar	20.6	14.1	10.5	2.08	10.3	12.8
Biotite/similar	5.56	4.65	1.31	2.35	2.43	1.75
Stilpnomelane/similar	0.69	1.62	0.12	0.23	0.63	0.10
Montmorillonite/similar clay minerals	10.8	3.02	0.56	2.95	0.49	0.26
Montmorillonite/similar clay minerals (Fe)	2.50	1.26	0.10	4.23	0.36	0.10
Nontronite/chamosite/similar (high Fe)	9.21	5.22	2.85	2.74	1.13	0.78
Goethite/limonite	2.80	2.71	0.34	0.82	0.92	0.51
Other minerals	1.60	4.92	5.83	2.48	2.06	6.55
TOTAL	100	100	100	100	100	100

Table 13.87 Master Composites Enioda & Han - Model Mineralogy

Mineral group	Composite Model Mass %					
	Chegue Main			Chegue South		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-19	MC-20	MC-21	MC-22	MC-23	MC-24
Pyrite/pyrrhotite	0.09	0.16	0.95	0.07	0.18	0.86
Fe-sulphates	0.02	0.03	0.01	0.01	0.02	0.01
Sphalerite/other Zn	0.01	0.01	0.02	0.01	0.01	0.01
Calcite	0.02	0.31	5.35	0.02	0.30	3.86
Quartz	64.9	50.5	44.0	50.7	32.5	32.9
Plagioclase (Na)/similar	0.20	29.0	36.5	14.8	32.5	35.2
Plagioclase (Na-Ca)/similar	0.05	0.32	1.31	0.30	2.15	1.95
K-feldspar/similar	0.35	0.18	0.10	0.26	0.38	0.40
Muscovite/similar	9.95	8.78	5.03	10.0	17.8	15.6
Biotite/similar	4.32	3.26	0.90	7.94	6.46	1.99
Stilpnomelane/similar	0.80	0.56	0.10	0.72	1.01	0.08
Montmorillonite/similar clay minerals	6.34	1.22	0.55	1.67	0.77	0.46
Montmorillonite/similar clay minerals (Fe)	1.52	0.92	0.12	2.24	0.59	0.09
Nontronite/chamosite/similar (high Fe)	6.94	1.97	1.91	7.94	2.57	3.19
Goethite/limonite	2.82	0.96	0.16	2.19	0.87	0.53
Other minerals	1.68	1.78	3.03	1.14	1.83	2.85
TOTAL	100	100	100	100	100	100

Table 13.88 Master Composites Chegue Main & Chegue South - Model Mineralogy

Sulphur Deportment

The QEMSCAN reported sulphur at an average of 0.05% in the 'Oxide' samples, 0.14% in the Transition' samples and 0.67% in the 'Fresh' samples. Details of the sulphur deportment in the master composites is presented in Table 13.89 to Table 13.92.

Mineral group	Composite Model Mass %					
	Souwa			Nokpa		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-1	MC-2	MC-3	MC-4	MC-5	MC-6
Pyrite/pyrrhotite	58.1	79.1	94.0	68.3	80.0	90.7
Fe-sulphates	21.1	7.30	0.25	4.24	2.53	0.26
Sphalerite/other Zn	4.58	2.20	0.55	4.60	2.96	0.53
Chalcopyrite/other Cu	0.55	1.24	0.14	2.84	0.86	0.06
Other sulphides/intergrowths	2.11	1.11	0.48	2.11	2.38	0.60
Biotite/similar	1.84	1.58	0.74	6.08	2.35	0.47
Nontronite/chamosite/similar (high Fe)	9.00	5.21	0.79	7.10	5.39	0.65
Other minerals	2.74	2.00	2.85	4.45	3.04	6.59
TOTAL	100	100	100	100	100	100

Table 13.89 Master Composites Souwa & Nokpa - Sulphur Deportment

Mineral group	Composite Model Mass %					
	Kilosegui			Kekeda		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-7	MC-8	MC-9	MC-10	MC-11	MC-12
Pyrite/pyrrhotite	66.2	91.6	95.7	64.9	68.2	96.9
Fe-sulphates	3.94	1.42	0.34	7.93	12.7	0.18
Sphalerite/other Zn	6.67	1.58	0.45	4.80	3.03	0.66
Chalcopyrite/other Cu	0.82	0.17	0.08	4.02	0.56	0.13
Other sulphides/intergrowths	2.64	0.56	0.50	1.46	0.79	1.07
Biotite/similar	6.67	0.97	0.83	3.65	1.79	0.29
Nontronite/chamosite/similar (high Fe)	9.49	2.67	1.09	10.8	9.09	0.37
Other minerals	3.22	0.88	0.74	2.27	3.27	0.30
TOTAL	100	100	100	100	100	100

Table 13.90 Master Composites Kiloegui & Kekeda - Sulphur Deportment

Mineral group	Composite Model Mass %					
	Enioda			Han		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-13	MC-14	MC-15	MC-16	MC-17	MC-18
Pyrite/pyrrhotite	56.6	82.9	96.1	64.0	88.9	93.3
Fe-sulphates	8.12	5.44	0.26	3.86	1.58	0.24
Sphalerite/other Zn	4.84	1.78	0.68	7.18	1.42	0.43
Chalcopyrite/other Cu	1.58	1.25	0.25	2.97	0.41	0.52
Other sulphides/intergrowths	3.03	0.70	0.18	3.01	0.95	2.83
Biotite/similar	6.09	1.54	0.83	2.99	0.66	0.57
Nontronite/chamosite/similar (high Fe)	16.0	5.10	1.10	8.56	1.69	0.53
Other minerals	3.33	0.97	0.40	3.90	4.30	1.20
TOTAL	100	100	100	100	100	100

Table 13.91 Master Composites Enioda & Han - Sulphur Deportment

Mineral group	Composite Model Mass %					
	Chegue Main			Chegue South		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
	MC-19	MC-20	MC-21	MC-22	MC-23	MC-24
Pyrite/pyrrhotite	69.9	72.6	89.9	66.2	83.4	95.9
Fe-sulphates	3.67	3.77	0.29	3.00	2.68	0.29
Sphalerite/other Zn	4.72	4.51	1.09	7.39	3.43	0.63
Chalcopyrite/other Cu	0.88	0.96	0.37	0.49	0.82	0.29
Other sulphides/intergrowths	1.60	3.50	2.70	2.71	0.93	0.61
Biotite/similar	3.87	1.47	0.71	4.83	2.54	0.82
Nontronite/chamosite/similar (high Fe)	10.4	4.21	1.33	11.1	4.59	0.98
Other minerals	4.74	8.78	3.22	3.96	1.44	0.29
TOTAL	100	100	100	100	100	100

Table 13.92 Master Composites Chegue Main & Chegue South - Sulphur Deportment

13.7 Grind and Cyanide Optimization Test Work

13.7.1 Grind Optimization Test Work

Grind versus recovery direct cyanidation tests with removal of gravity gold prior to leaching were performed on the master composites at grind size P_{80}'s of 125, 106 and 75 µm to determine the optimum size for all future leach tests. This phase of test work also allowed an assessment of the cyanidation leach time likely to be needed to ensure high dissolution of gold for all weathering types and deposits.

For the gravity test a 3 kg sub sample was ground to 80% passing 125, 106 and 75 µm and processed as a single pass through a 3" standard Knelson concentrator. The Knelson concentrate was subsequently intensively leached to duplicate industry standard gravity concentrate leach conditions.

The grind versus cyanidation test work was conducted as bottle rolls under the following conditions:

- Gravity concentration by Knelson concentrator, amalgamation and leaching of gravity tailings;
- Bottle roll leaches at pulp density of 45% solids in Perth tap water;
- pH 10.0 to 10.5 adjusted with commercial lime (60% available CaO);
- Initial cyanide dosage of 1.22 kg/t NaCN with residual cyanide levels maintained at or above 500 ppm;
- Oxygen sparging to achieve dissolved oxygen level of 20 ppm or greater; and
- 40 hour leach duration with samples at 2, 4, 8, 12, 24, and 40 hours.

Souwa

Grind optimization leach test work results for the Souwa deposit are presented in Table 13.93 and leach curves are illustrated in Figure 13.14. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations were made:

- The Souwa oxide composite reported elevated calculated head grades ranging from 4.82 g/t to 5.81 g/t Au when compared to the assayed head at 3.70 g/t Au;
- 40 hour gold extractions ranged from 85.7% to 95.4%;
- Residue grades ranged from 0.07 g/t to 0.37 g/t;
- Gravity extractions ranged from 25.4% to 39.1%;
- Cyanide consumptions ranged from 0.21 kg/t to 0.39 kg/t;
- Both the Oxide and Transitional master composites had fast leach kinetics with >90% gold extraction after 8 hours. Only marginal improvements in recovery were reported at the P_{80}-75 µm, and at the extended 40 hour leach residence time; and
- The Souwa Fresh master composite sample reported lower gold recovery at 86.7% and marginal improvements in recovery at the finer P_{80}-75 µm grind. The slower leach kinetics reported a recovery delta of +3.5% from the extended 40 hour leach residence time.

Sample ID	Units	SOUWA OXIDE (MC-1)			SOUWA TRANS (MC-2)			SOUWA FRESH (MC-3)		
Test Number		PW_7612	PW_7613	PW_7614	PW_7615	PW_7616	PW_7617	PW_7618	PW_7619	PW_7620
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	39.1	38.3	38.0	25.4	28.0	26.8	36.4	33.1	27.2
2 Hour	%	81.1	83.0	86.6	80.2	81.8	82.7	73.0	73.9	76.0
4 Hour	%	86.9	88.5	91.6	88.6	90.5	90.6	76.3	76.0	77.4
8 Hour	%	90.7	90.8	93.3	91.2	91.6	91.1	79.6	79.6	80.5
24 Hour	%	92.6	93.3	94.3	93.3	92.6	93.0	81.6	83.6	83.2
40 Hour	%	93.5	93.8	95.4	92.2	94.1	93.5	85.7	86.0	86.7
Residue Grade	g/t	0.37	0.30	0.27	0.09	0.07	0.08	0.21	0.21	0.18
Gravity Recovery	%	39.1	38.3	38.0	25.4	28.0	26.8	36.4	33.1	27.2
Leach Recovery	%	54.4	55.4	57.3	66.8	66.1	66.8	49.3	52.9	59.5
Overall Recovery	%	93.5	93.8	95.4	92.2	94.1	93.5	85.7	86.0	86.7
Calc'd Head	g/t	5.67	4.82	5.81	1.16	1.18	1.23	1.47	1.50	1.35
Lime Cons.	kg/t	0.52	0.57	0.56	0.29	0.32	0.31	0.22	0.18	0.23
Cyanide Cons.	kg/t	0.39	0.36	0.39	0.33	0.33	0.36	0.21	0.30	0.27

Table 13.93 Souwa Grind Optimization Leach Results

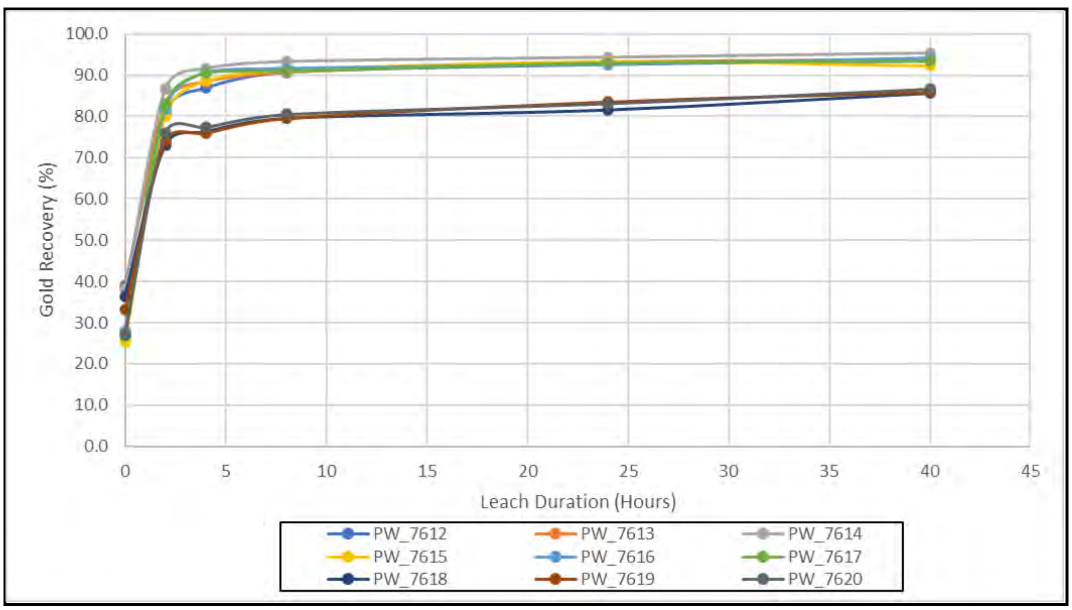


Figure 13.14 Souwa Grind Optimization Leach Kinetic Curves

Nokpa

Grind optimization leach test work results for the Nokpa deposit are presented in Table 13.94 and leach curves are illustrated in Figure 13.15.

Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 86.7% to 96.3%;
- Residue grades ranged from 0.07 g/t to 0.21 g/t;
- Gravity extractions ranged from 12.9% to 29.4%;
- Cyanide consumptions ranged from 0.21 kg/t to 0.39 kg/t;
- Both the Oxide and Transitional master composites had fast leach kinetics with >80% gold extraction after 8 hours;
- The Oxide composite reported a recovery delta of +4.4% after 24 hour and +2.7% after 40 hour leach residence times at the finer P_{80}-75 µm grind;
- Only marginal improvements in recovery were reported in the Transitional composite at the P_{80}-75 µm, and at the extended 40 hour leach residence time; and
- The Fresh master composite sample reported lower gold recovery at 88.8% and a recovery delta of +2.0% at the finer P_{80}-75 µm grind. The slower leach kinetics reported a recovery delta of +6.3% after the additional 16 hour leach residence time.

Sample ID	Units	NOKPA OXIDE (MC-4)			NOKPA TRANS (MC-5)			NOKPA FRESH (MC-6)		
Test Number		PW_7621	PW_7622	PW_7623	PW_7624	PW_7625	PW_7626	PW_7627	PW_7628	PW_7629
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	15.6	15.8	24.5	12.9	14.3	14.2	27.7	29.4	29.3
2 Hour	%	81.8	81.4	86.8	90.1	88.4	91.5	67.3	69.2	70.5
4 Hour	%	86.5	86.5	92.1	92.5	91.5	92.4	71.3	71.1	73.3
8 Hour	%	89.9	88.7	92.1	93.4	93.4	95.0	74.4	76.0	77.3
24 Hour	%	91.6	89.8	94.3	95.3	94.7	96.3	79.3	81.3	82.5
40 Hour	%	93.3	92.5	95.2	94.9	95.1	96.3	86.7	86.8	88.8
Residue Grade	g/t	0.07	0.08	0.07	0.13	0.13	0.10	0.18	0.21	0.17
Gravity Recovery	%	15.6	15.8	24.5	12.9	14.3	14.2	27.7	29.4	29.3
Leach Recovery	%	77.7	76.7	70.7	82.0	80.8	82.1	59.1	57.4	59.4
Overall Recovery	%	93.3	92.5	95.2	94.9	95.1	96.3	86.7	86.8	88.8
Calc'd Head	g/t	1.04	1.07	1.35	2.53	2.67	2.72	1.36	1.60	1.51
Lime Cons.	kg/t	1.31	0.73	0.75	0.68	0.75	0.73	0.22	0.25	0.24
Cyanide Cons.	kg/t	0.24	0.39	0.37	0.30	0.30	0.27	0.24	0.21	0.21

Table 13.94 Nokpa Grind Optimization Leach Results

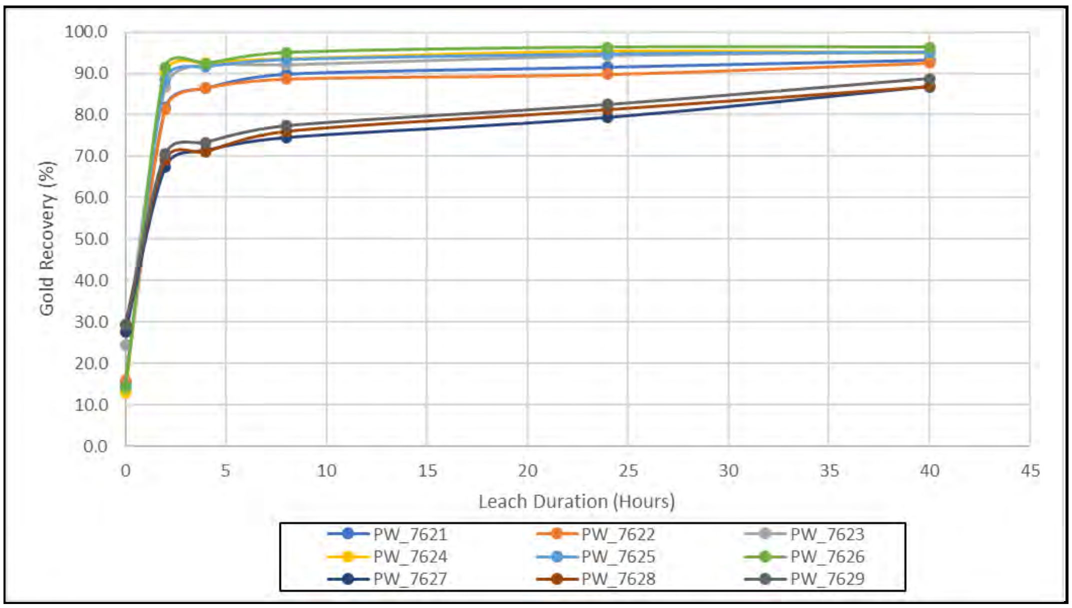


Figure 13.15 Nokpa Grind Optimization Leach Kinetic Curves

Kilosegui

Leach test work results for the Kilosegui deposit are presented in Table 13.95 and leach curves are illustrated in Figure 13.16.

Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 81.3% to 97.7%;
- Residue grades ranged from 0.02 g/t to 0.27 g/t;
- Gravity extractions ranged from 34.2% to 55.2%;
- Cyanide consumptions ranged from 0.18 kg/t to 0.36 kg/t;
- The Oxide master composites had fast leach kinetics with >90% gold extraction after 2 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the oxide material is not grind sensitive;
- The transitional composite reported a recovery delta of +2.5% at the finer P_{80}-75 µm grind, however no improvement from the extended 40 hour leach residence time; and
- The Fresh master composite sample reported maximum gold recovery at 84.7% and a recovery delta of +3.2% at the finer P_{80}-75 µm grind.

Sample ID	Units	KILOSEGUI OXIDE (MC-7)			KILOSEGUI TRANS (MC-8)			KILOSEGUI FRESH (MC-9)		
Test Number		PW_7630	PW_7631	PW_7632	PW_7633	PW_7634	PW_7635	PW_7636	PW_7637	PW_7638
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	50.3	44.1	34.9	55.2	34.2	37.7	42.6	41.1	37.7
2 Hour	%	91.3	90.6	90.0	83.7	76.0	82.9	73.8	75.0	77.7
4 Hour	%	96.3	94.7	95.8	90.5	87.3	90.3	78.4	79.4	83.0
8 Hour	%	96.3	96.4	96.8	92.2	88.8	91.2	80.5	80.2	83.0
24 Hour	%	96.3	97.2	97.7	93.9	90.2	92.7	81.3	81.1	83.9
40 Hour	%	97.7	97.2	96.8	93.2	90.2	92.7	81.3	81.5	84.7
Residue Grade	g/t	0.02	0.02	0.02	0.12	0.12	0.09	0.27	0.26	0.21
Gravity Recovery	%	50.3	44.1	34.9	55.2	34.2	37.7	42.6	41.1	37.7
Leach Recovery	%	47.3	53.1	61.8	38.1	56.0	55.0	38.8	40.4	47.1
Overall Recovery	%	97.7	97.2	96.8	93.2	90.2	92.7	81.3	81.5	84.7
Calc'd Head	g/t	0.85	0.72	0.62	1.78	1.23	1.23	1.45	1.40	1.37
Lime Cons.	kg/t	1.72	1.95	1.88	0.41	0.44	0.41	0.30	0.25	0.22
Cyanide Cons.	kg/t	0.18	0.18	0.21	0.27	0.24	0.27	0.27	0.36	0.36

Table 13.95 Kilosegui Grind Optimization Leach Results

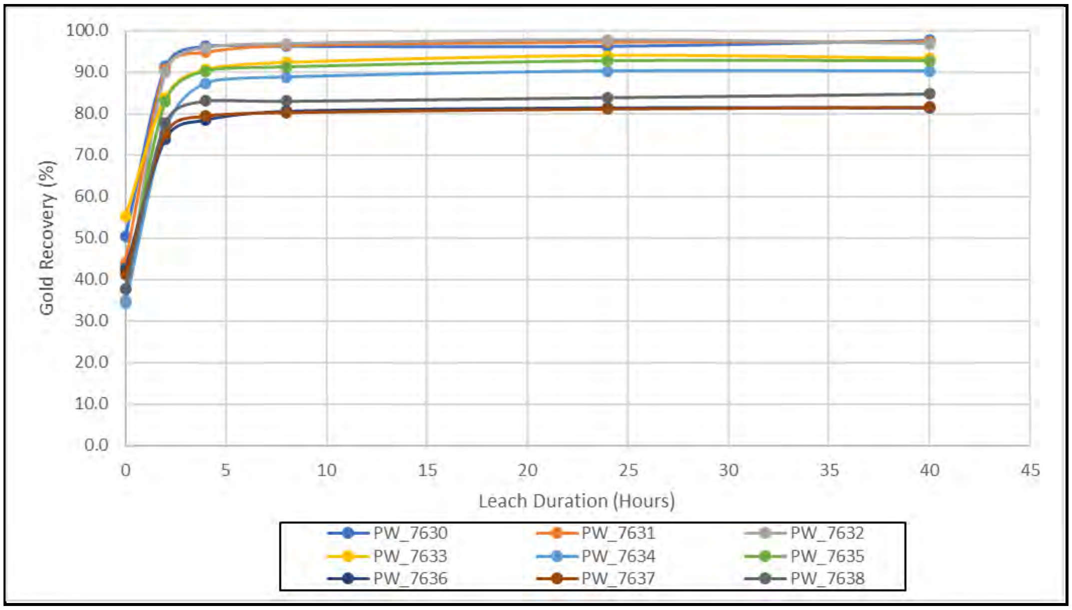


Figure 13.16 Kilosegui Grind Optimization Leach Kinetic Curves

Kekeda

Leach test work results for the Kekeda deposit are presented in Table 13.96 and leach curves are illustrated in Figure 13.17. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 88.1% to 95.8%;
- Residue grades ranged from 0.03 g/t to 0.20 g/t;
- Gravity extractions ranged from 15.8% to 38.4%;
- Cyanide consumptions ranged from 0.21 kg/t to 0.37 kg/t;
- The Oxide master composites had fast leach kinetics with >90% gold extraction after 4 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the oxide material is not grind sensitive;
- The Transitional master composites had fast leach kinetics averaging at 90% gold extraction after 4 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the oxide material is not grind sensitive;
- The Fresh master composite sample reported maximum gold recovery at 89.9%. No improvement in recovery was observed at the finer P_{80}-75 µm grind.

Sample ID	Units	KEKEDA OXIDE (MC-10)			KEKEDA TRANS (MC-11)			KEKEDA FRESH (MC-12)		
Test Number		PW_7639	PW_7640	PW_7641	PW_7642	PW_7643	PW_7644	PW_7645	PW_7646	PW_7647
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	20.2	24.0	25.2	24.8	15.8	20.5	35.3	38.4	32.2
2 Hour	%	85.1	88.2	87.1	88.6	87.6	89.5	73.1	73.2	70.4
4 Hour	%	91.1	94.1	92.5	89.6	89.4	90.0	75.7	77.0	75.2
8 Hour	%	91.9	94.1	94.8	92.0	90.0	92.2	79.2	81.4	79.5
24 Hour	%	94.4	95.8	94.8	93.9	91.7	93.2	83.9	86.5	84.2
40 Hour	%	94.4	95.8	95.6	94.4	94.1	93.8	88.1	89.8	89.6
Residue Grade	g/t	0.04	0.03	0.04	0.07	0.06	0.07	0.20	0.18	0.16
Gravity Recovery	%	20.2	24.0	25.2	24.8	15.8	20.5	35.3	38.4	32.2
Leach Recovery	%	74.2	71.8	70.3	69.5	78.3	73.3	52.8	51.4	57.4
Overall Recovery	%	94.4	95.8	95.6	94.4	94.1	93.8	88.1	89.8	89.6
Calc'd Head	g/t	0.72	0.71	0.79	1.25	1.01	1.12	1.68	1.77	1.54
Lime Cons.	kg/t	1.05	1.08	1.20	0.62	0.58	0.61	0.28	0.23	0.25
Cyanide Cons.	kg/t	0.39	0.43	0.37	0.36	0.33	0.36	0.21	0.27	0.30

Table 13.96 Kekeda Grind Optimization Leach Results

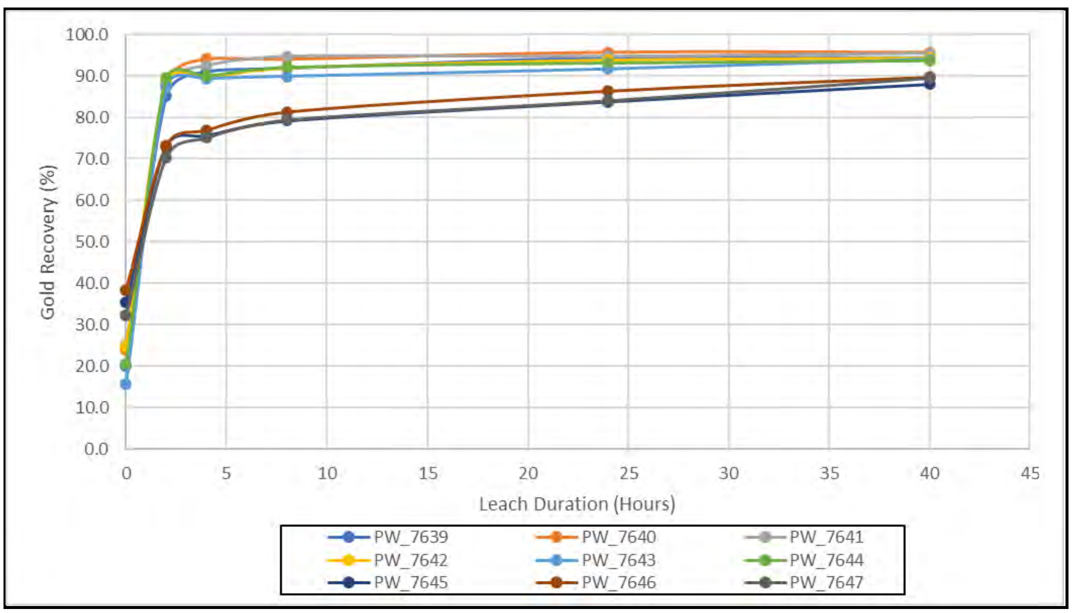


Figure 13.17 Kekeda Grind Optimization Leach Kinetic Curves

Enioda

Leach test work results for the Enioda deposit are presented in Table 13.97 and leach curves are illustrated in Figure 13.18.

Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 84.4% to 97.8%;
- Residue grades ranged from 0.03 g/t to 0.20 g/t;
- Gravity extractions ranged from 30.2% to 43.3%;
- Cyanide consumptions ranged from 0.33 kg/t to 0.48 kg/t;
- The Oxide and Transitional master composites reported fast leach kinetics with >90% gold extraction after 4 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the Oxide and Transitional material are not grind sensitive; and
- The Fresh master composite sample reported maximum gold recovery at 89.1% after 24 hour leach residence time and a recovery delta of +4.8% at the finer P_{80}-75 µm grind.

Sample ID	Units	ENIODA OXIDE (MC-13)			ENIODA TRANS (MC-14)			ENIODA FRESH (MC-15)		
Test Number		PW_7696	PW_7697	PW_7698	PW_7699	PW_7700	PW_7701	PW_7702	PW_7703	PW_7704
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	34.9	39.0	30.2	36.5	43.3	33.5	38.0	38.4	42.0
2 Hour	%	86.1	90.3	87.9	82.8	88.0	89.7	71.8	74.4	80.8
4 Hour	%	91.6	95.1	93.5	89.7	90.9	92.1	78.4	79.4	84.5
8 Hour	%	94.6	94.7	95.7	91.2	91.8	93.9	80.7	81.7	86.8
24 Hour	%	96.2	97.3	95.7	91.7	91.8	93.9	82.6	84.3	89.1
40 Hour	%	96.2	96.9	97.8	92.7	93.2	93.9	84.4	86.6	87.8
Residue Grade	g/t	0.06	0.05	0.03	0.09	0.09	0.06	0.20	0.18	0.16
Gravity Recovery	%	34.9	39.0	30.2	36.5	43.3	33.5	38.0	38.4	42.0
Leach Recovery	%	61.2	57.9	67.6	56.2	49.9	60.4	46.4	48.2	45.8
Overall Recovery	%	96.2	96.9	97.8	92.7	93.2	93.9	84.4	86.6	87.8
Calc'd Head	g/t	1.56	1.63	1.39	1.23	1.26	0.99	1.29	1.34	1.31
Lime Cons.	kg/t	1.00	1.05	1.08	0.67	0.62	0.69	0.27	0.26	0.25
Cyanide Cons.	kg/t	0.43	0.45	0.33	0.43	0.43	0.39	0.45	0.42	0.48

Table 13.97 Enioda Grind Optimization Leach Results

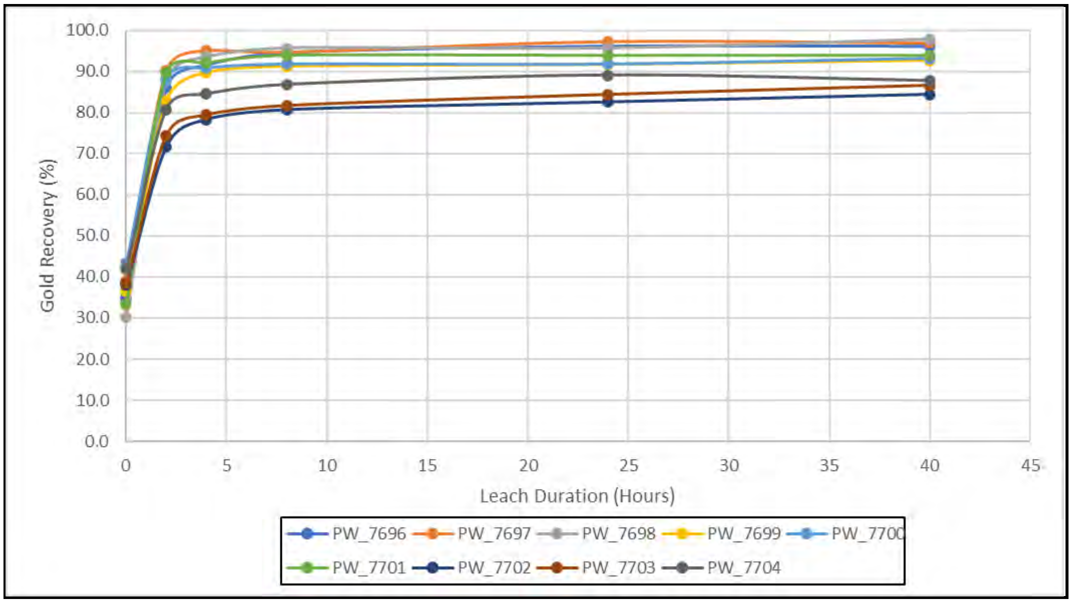


Figure 13.18 Enioda Grind Optimization Kinetic Leach Curves

Han

Leach test work results for the Han deposit are presented in Table 13.98 and leach curves are illustrated in Figure 13.19. The Han deposit did not have any oxide material available for leach test work. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 84.4% to 96.3%;
- Residue grades ranged from 0.06 g/t to 0.19 g/t;
- Gravity extractions ranged from 32.1% to 52.9%;
- Cyanide consumptions ranged from 0.30 kg/t to 0.39 kg/t;
- The Transitional master composite reported maximum gold recovery at 96.3% after 40 hour leach residence time. The three different grind sizes reported similar gold recoveries and residue grades indicating the Transitional material is not grind sensitive;
- The Fresh master composite sample reported maximum gold recovery at 88.7% after 40 hour leach residence time and a recovery delta of +3.2% at the finer P_{80}-75 µm grind.

Sample ID	Units	HAN TRANS (MC-17)			HAN FRESH (MC-18)		
Test Number		PW_7705	PW_7706	PW_7707	PW_7708	PW_7709	PW_7710
Grind Size (P_{80})	µm	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	52.9	40.9	41.9	32.1	35.9	45.0
2 Hour	%	87.3	82.7	85.0	69.7	71.3	76.5
4 Hour	%	90.5	87.6	88.4	73.7	75.0	80.4
8 Hour	%	93.4	90.5	90.7	77.1	79.2	82.7
24 Hour	%	95.2	93.9	93.7	80.5	82.4	84.9
40 Hour	%	95.2	95.7	96.3	84.4	85.5	88.7
Residue Grade	g/t	0.08	0.07	0.06	0.19	0.19	0.18
Gravity Recovery	%	52.9	40.9	41.9	32.1	35.9	45.0
Leach Recovery	%	42.3	54.8	54.3	52.3	49.6	43.7
Overall Recovery	%	95.2	95.7	96.3	84.4	85.5	88.7
Calc'd Head	g/t	1.67	1.62	1.60	1.22	1.31	1.59
Lime Cons.	kg/t	0.35	0.34	0.32	0.29	0.27	0.29
Cyanide Cons.	kg/t	0.36	0.33	0.39	0.33	0.30	0.33

Table 13.98 Han Grind Optimization Leach Results

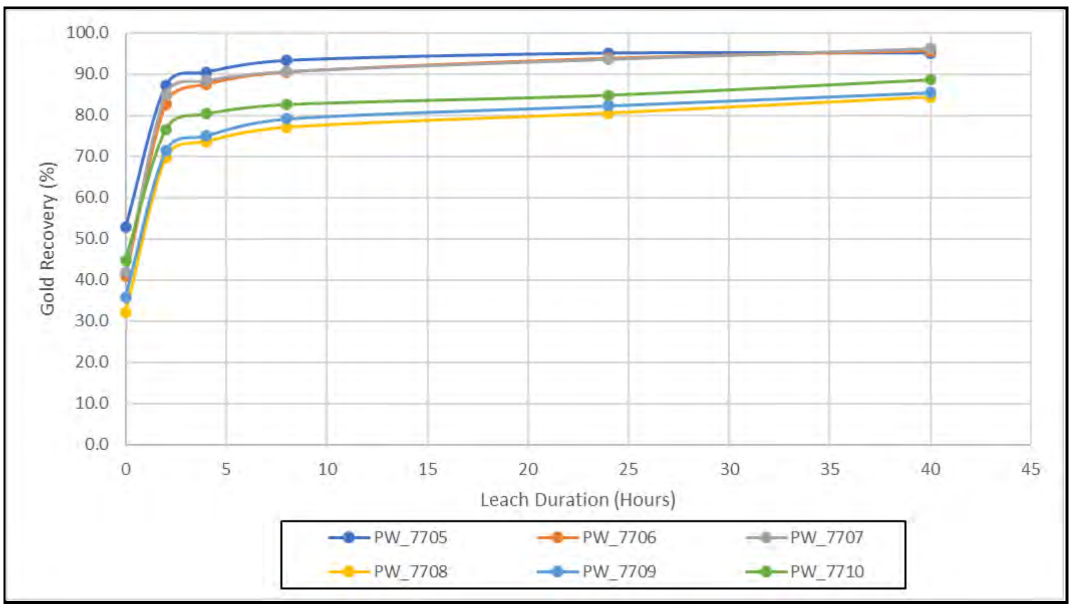


Figure 13.19 Han Grind Optimization Leach Kinetic Curves

Chegue Main

Leach test work results for the Chegue Main deposit are presented in Table 13.99 and leach recovery curves are illustrated in Figure 13.20.

Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 88.7% to 96.6%;
- Residue grades ranged from 0.04 g/t to 0.16 g/t;
- Gravity extractions ranged from 25.5% to 44.0%;
- Cyanide consumptions ranged from 0.18 kg/t to 0.31 kg/t;
- The Oxide and Transitional master composites reported fast leach kinetics with >90% gold extraction after 4 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the material is not grind sensitive;
- The Fresh master composite sample reported lower gold recovery at 89.8%. The three different grind sizes reported similar gold recoveries and residue grades indicating the material is not grind sensitive. The slower leach kinetics reported a recovery delta of +2.9% after the additional 16 hour leach residence time.

Sample ID	Units	CHEGUE MAIN OXIDE (MC-19)			CHEGUE MAIN TRANS (MC-20)			CHEGUE MAIN FRESH (MC-21)		
Test Number		PW_7711	PW_7712	PW_7713	PW_7714	PW_7715	PW_7716	PW_7717	PW_7718	PW_7719
Grind Size (P_{80})	µm	125	106	75	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	42.7	32.0	37.0	30.0	25.5	25.9	42.6	44.0	38.1
2 Hour	%	87.8	87.1	89.3	86.5	87.3	89.0	75.5	77.8	75.8
4 Hour	%	92.9	92.2	94.5	91.3	90.2	90.1	80.6	80.9	80.2
8 Hour	%	95.2	92.2	94.0	91.8	91.3	93.3	82.8	82.7	82.6
24 Hour	%	95.2	94.4	95.6	94.4	92.9	93.3	86.6	85.8	85.9
40 Hour	%	96.3	95.1	96.6	93.9	93.5	93.8	88.7	89.7	88.8
Residue Grade	g/t	0.04	0.04	0.04	0.07	0.07	0.07	0.16	0.14	0.14
Gravity Recovery	%	42.7	32.0	37.0	30.0	25.5	25.9	42.6	44.0	38.1
Leach Recovery	%	53.6	63.1	59.6	63.9	68.0	67.9	46.1	45.7	50.7
Overall Recovery	%	96.3	95.1	96.6	93.9	93.5	93.8	88.7	89.7	88.8
Calc'd Head	g/t	1.07	0.82	1.17	1.15	1.08	1.13	1.41	1.36	1.25
Lime Cons.	kg/t	1.36	1.43	1.53	0.46	0.44	0.46	0.22	0.24	0.23
Cyanide Cons.	kg/t	0.18	0.24	0.30	0.30	0.31	0.31	0.27	0.27	0.30

Table 13.99 Chegue Main Grind Optimization Leach Results

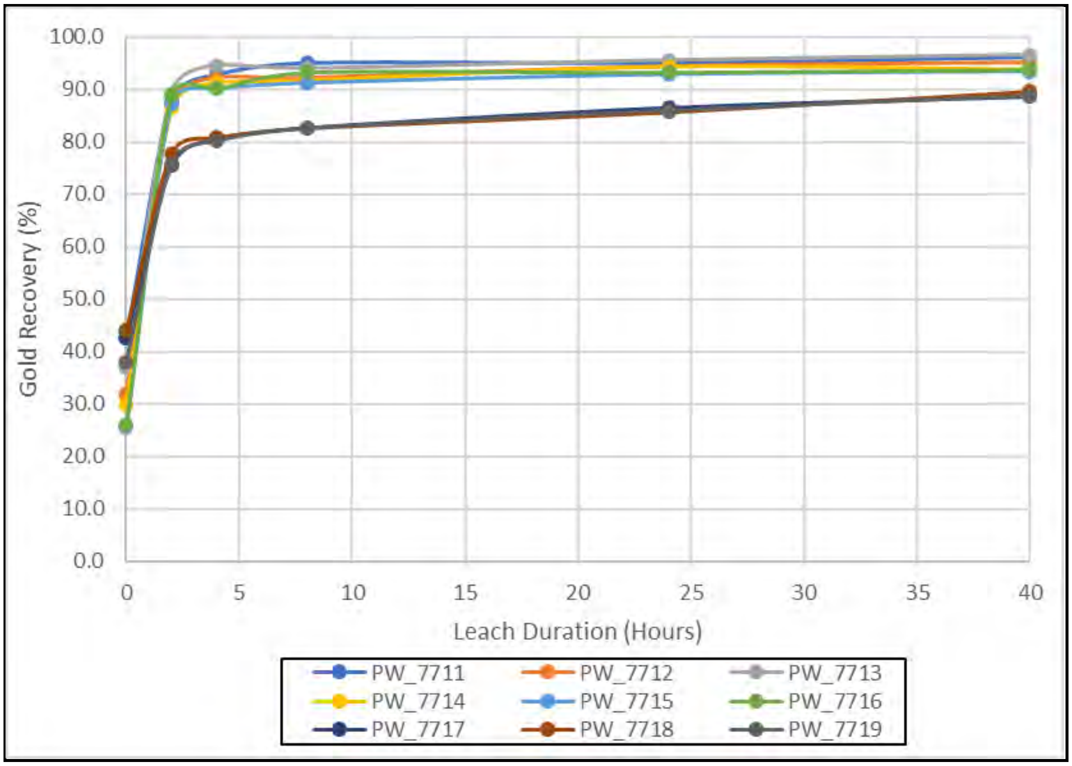


Figure 13.20 Chegue Main Grind Optimization Leach Kinetic Curves

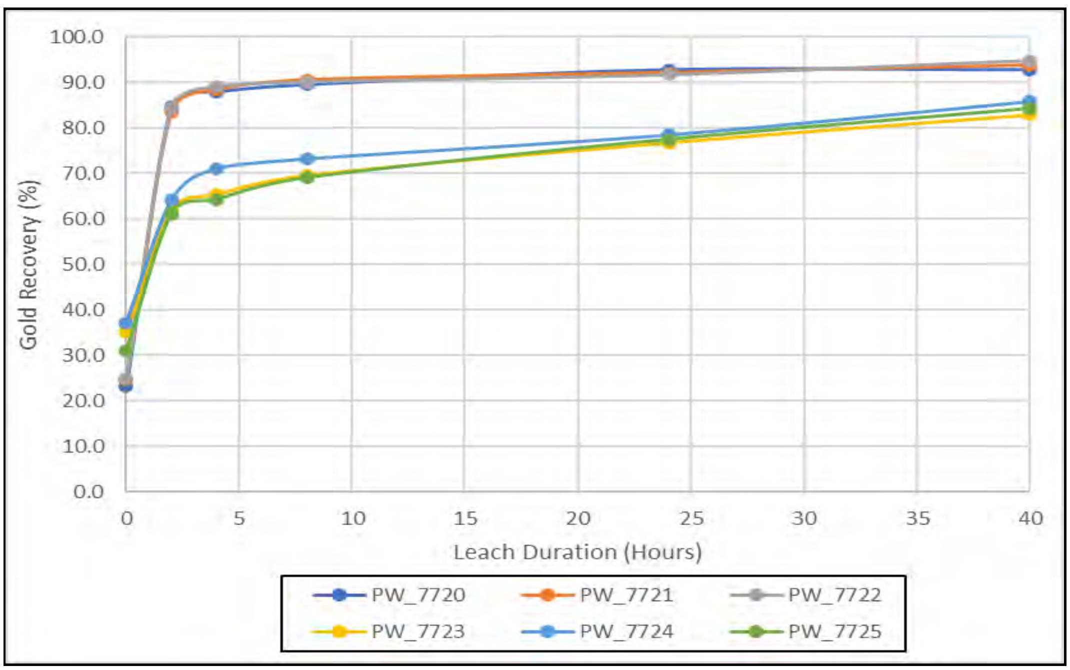
Chegue South

Grind optimization leach test work results for Chegue South are presented in Table 13.100 and leach kinetic curves are illustrated in Figure 13.21. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24). The following observations can be made:

- 40 hour gold extractions ranged from 82.9% to 94.8%;
- Residue grades ranged from 0.06 g/t to 0.20 g/t;
- Gravity extractions ranged from 23.2% to 37.2%;
- Cyanide consumptions ranged from 0.20 kg/t to 0.30 kg/t;
- The Transitional master composites reported fast leach kinetics with >90% gold extraction after 8 hours. The three different grind sizes reported similar gold recoveries and residue grades indicating the material is not grind sensitive;
- The Fresh master composite sample reported lower gold recovery at 85.9%. The three different grind sizes reported similar gold recoveries and residue grades indicating the material is not grind sensitive. The slower leach kinetics reported a recovery delta of +7.4% after the additional 16 hour leach residence time.

Sample ID	Units	CHEGUE SOUTH TRANS (MC-23)			CHEGUE SOUTH FRESH (MC-24)		
Test Number		PW_7720	PW_7721	PW_7722	PW_7723	PW_7724	PW_7725
Grind Size (P_{80})	µm	125	106	75	125	106	75
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10
(Maintain)	%	0.05	0.05	0.05	0.05	0.05	0.05
O_2/Air Sparge		O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	23.2	24.8	24.9	35.1	37.2	31.1
2 Hour	%	84.6	83.5	84.3	61.8	64.2	61.1
4 Hour	%	87.9	88.4	89.1	65.5	71.1	64.3
8 Hour	%	89.5	90.5	90.1	69.6	73.2	69.1
24 Hour	%	92.8	92.1	91.7	76.8	78.5	77.5
40 Hour	%	92.8	93.7	94.8	82.9	85.9	84.2
Residue Grade	g/t	0.08	0.07	0.06	0.20	0.16	0.18
Gravity Recovery	%	23.2	24.8	24.9	35.1	37.2	31.1
Leach Recovery	%	69.6	68.9	69.9	47.7	48.7	53.1
Overall Recovery	%	92.8	93.7	94.8	82.9	85.9	84.2
Calc'd Head	g/t	1.11	1.12	1.15	1.17	1.13	1.14
Lime Cons.	kg/t	0.54	0.56	0.56	0.30	0.31	0.29
Cyanide Cons.	kg/t	0.24	0.24	0.20	0.26	0.27	0.30

Table 13.100 Chegue South Grind Optimization Leach Results



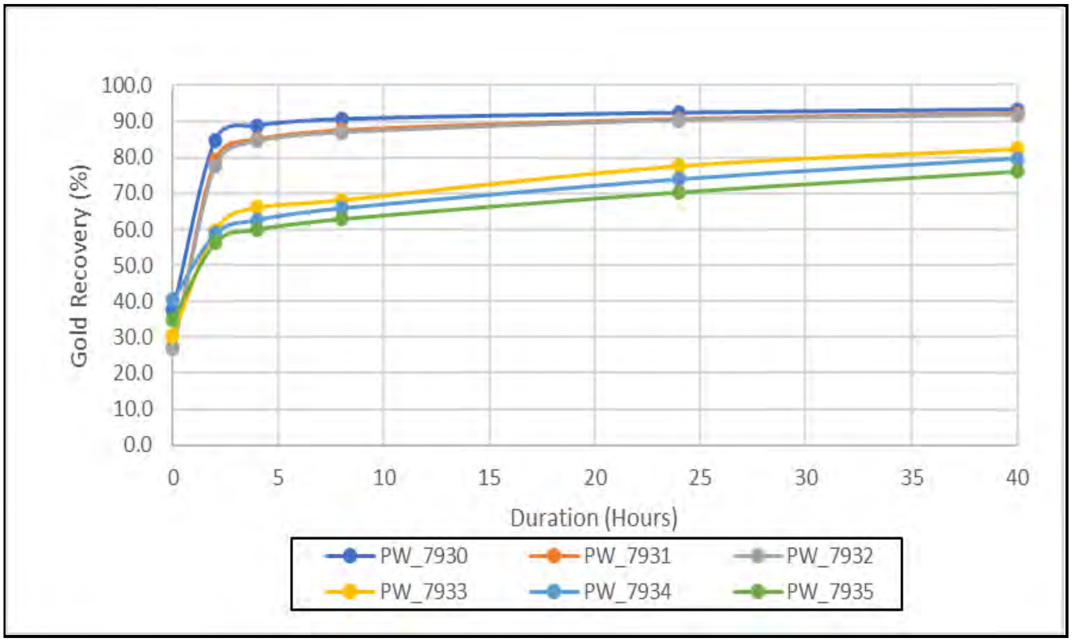


Figure 13.21 Chegue South Grind Optimization Leach Kinetic Curves

13.8 Oxygen Versus Air Direct Cyanidation Leach Tests

A series of oxygen versus air sparging direct cyanidation leach tests, with removal of gravity gold prior to leaching, were conducted on lithology composites from each of the eight (8) open pits. The aim of the tests was to measure the effects on gold recovery of using oxygen compared to air sparging.

For the gravity test a 3 kg sub sample was ground to 80% passing 106 µm (Oxide & Trans Composites) or 75 µm (Fresh composites) and processed as a single pass through a 3" standard Knelson concentrator. The Knelson concentrate was subsequently intensively leached to duplicate industry standard gravity concentrate leach conditions.

The oxygen versus air sparged direct cyanidation leach test work was conducted in bottle rolls under the following conditions:

- Gravity concentration by Knelson concentrator, amalgamation and leaching of gravity tailings;
- Bottle roll leaches at pulp density of 45% solids in Perth tap water;
- pH 10.0 to 10.5 adjusted with commercial lime (60% available CaO);
- Initial cyanide dosage of 0.05% CN with residual cyanide levels maintained at or above 250 ppm;
- Oxygen sparging to achieve dissolved oxygen level of 20 ppm or greater;
- Air sparging to provide a minimum of 3 ppm dissolved oxygen; and
- 40 hour leach duration with samples at 2, 4, 8, 12, 24, and 40 hours.

13.8.1 *Oxygen Versus Air Leach Results*

Souwa

The oxygen versus air cyanidation leach test work was conducted on the Souwa Oxide, Transitional and Fresh master composites with results presented in Table 13.101 The following observations can be made:

- Gold dissolution kinetics and overall recoveries were higher in all tests with the use of oxygen;
- The rate of reaction with the use of oxygen was higher for the period up to 8 hours, however for the proceeding 16 hours the rates were similar, and the final 14 hours recoveries plateaued with the oxygen sparged pulps reporting significantly higher overall recoveries; and
- Final recovery deltas after 24 hours were as follows:
  - Souwa Oxide - 92.2% with O_2 versus 90.0% with air, delta 2.2%;
  - Souwa Trans - 89.7% with O_2 versus 86.7% with air, delta 3.0%;
  - Souwa Fresh - 82.5% with O_2 versus 78.8% with air, delta 3.7%.

Sample ID	Units	SOUWA OXIDE (MC-1)		SOUWA TRANS (MC-2)		SOUWA FRESH (MC-3)	
Test Number		PW8058	PW8063	PW8059	PW8064	PW8060	PW8065
Grind Size (P_{80})	µm	106	106	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
24 Hour	%	92.2	90.0	89.7	86.7	82.5	78.8
40 Hour	%	93.9	91.7	91.7	90.8	85.9	81.1
Residue Grade	g/t	0.34	0.50	0.10	0.12	0.20	0.29
Gravity Recovery	%	33.4	38.4	24.1	23.8	27.0	31.3
Leach Recovery	%	60.5	53.3	67.6	67.0	58.9	49.8
Overall Recovery	%	93.9	91.7	91.7	90.8	85.9	81.1
Calc'd Head	g/t	5.56	6.05	1.20	1.31	1.41	1.51
Lime Cons.	kg/t	0.75	0.73	0.54	0.45	0.25	0.27
Cyanide Cons.	kg/t	0.21	0.18	0.21	0.12	0.21	0.14

Table 13.101 Souwa Master Composites - Oxygen Versus Air Leach Results

Nokpa

The oxygen versus air cyanidation leach test work was conducted on the Nokpa Oxide, Transitional and Fresh master composites with results presented in Table 13.102 The following observations can be made:

- Gold dissolution kinetics and overall recoveries were marginally higher in the Transitional sample and significantly higher in the Fresh sample with the use of oxygen;
- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 8 hours, however for the proceeding 16 hours the rates were similar, and the final 14 hours recoveries plateaued with the oxygen sparged Fresh sample reporting the larger recovery delta at +9.6%; and
- Final recovery deltas after 24 hours were as follows:
  - Nokpa Trans - 93.7% with O_2 versus 92.3% with air, delta 1.4%;
  - Nokpa Fresh - 84.8% with O_2 versus 75.9% with air, delta 8.9%.

Sample ID	Units	NOKPA TRANS (MC-5)		NOKPA FRESH (MC-6)	
Test Number		PW8061	PW8066	PW8062	PW8067
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
24 Hour	%	93.7	92.3	84.8	75.9
40 Hour	%	94.5	94.0	89.8	80.2
Residue Grade	g/t	0.15	0.16	0.16	0.25
Gravity Recovery	%	13.0	13.6	28.9	25.8
Leach Recovery	%	81.5	80.4	60.9	54.4
Overall Recovery	%	94.5	94.0	89.8	80.2
Calc'd Head	g/t	2.74	2.68	1.57	1.26
Lime Cons.	kg/t	0.78	0.74	0.43	0.22
Cyanide Cons.	kg/t	0.15	0.14	0.18	0.08

Table 13.102 Nokpa Master Composites - Oxygen Versus Air Leach Results

Kilosegui

The oxygen versus air cyanidation leach test work was conducted on the Kilosegui Transitional and Fresh Master composites with results presented in Table 13.103 The following observations can be made:

- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 4 hours, however for the proceeding 20 hours the rates were similar, and the final 14 hours recoveries plateaued with the air sparged Transitional sample reporting marginally higher recovery, and the oxygen sparged Fresh sample reporting a higher recovery; and
- Final recovery deltas after 24 hours were as follows:
  - Kilosegui Trans - 87.0% with O_2 versus 88.7% with air, delta -1.7%;
  - Kilosegui Fresh - 84.5% with O_2 versus 83.3% with air, delta 2.7%.

Sample ID	Units	KILOSEGUI TRANS (MC-8)		KILOSEGUI FRESH (MC-9)	
Test Number		PW8170	PW8183	PW8171	PW8184
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
24 Hour	%	87.0	88.7	84.5	83.3
40 Hour	%	89.5	89.8	85.0	82.3
Residue Grade	g/t	0.13	0.12	0.21	0.21
Gravity Recovery	%	33.6	32.1	39.1	35.1
Leach Recovery	%	55.9	57.6	45.9	47.2
Overall Recovery	%	89.5	89.8	85.0	82.3
Calc'd Head	g/t	1.23	1.17	1.40	1.19
Lime Cons.	kg/t	0.42	0.41	0.42	0.29
Cyanide Cons.	kg/t	0.12	0.15	0.06	0.15

Table 13.103 Kilosegui Master Composites - Oxygen Versus Air Leach Results

Kekeda

The oxygen versus air cyanidation leach test work was conducted on the Kekeda Transitional and Fresh Master composites with results presented in Table 13.104. The following observations can be made:

- Gold dissolution kinetics and overall recoveries were marginally higher in the Transitional sample and significantly higher in the Fresh sample with the use of oxygen;
- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 24 hours, and the final 24 hours recoveries plateaued with the oxygen sparged Fresh sample reporting the larger recovery delta at +7.7%; and
- Final recovery deltas after 24 hours were as follows:
  - Kekeda Trans - 92.6% with O_2 versus 90.4% with air, delta 2.2%;
  - Kekeda Fresh - 92.6% with O_2 versus 84.9% with air, delta 7.7%.

Sample ID	Units	KEKEDA TRANS (MC-11)		KEKEDA FRESH (MC-12)	
Test Number		PW8172	PW8185	PW8173	PW8186
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
24 Hour	%	92.6	90.4	92.6	84.9
40 Hour	%	92.2	91.5	92.9	87.6
Residue Grade	g/t	0.11	0.09	0.11	0.22
Gravity Recovery	%	30.0	21.5	33.2	44.7
Leach Recovery	%	62.2	70.0	59.8	42.9
Overall Recovery	%	92.2	91.5	92.9	87.6
Calc'd Head	g/t	1.40	1.06	1.56	1.77
Lime Cons.	kg/t	0.60	0.64	0.67	0.30
Cyanide Cons.	kg/t	0.14	0.14	0.02	0.15

Table 13.104 Kekeda Master Composites - Oxygen Versus Air Leach Results

Enioda

The oxygen versus air cyanidation leach test work was conducted on the Enioda Oxide, Transitional and Fresh Master composites with results presented in Table 13.105.

The following observations can be made:

- Gold dissolution kinetics and overall recoveries were higher in both the transitional and fresh tests with the use of oxygen;
- The rate of reaction with the use of oxygen was higher for the period up to 8 hours, however for the proceeding 16 hours the rates were similar, and the final 14 hours recoveries plateaued with the oxygen sparged pulps reporting lower overall recoveries except in the transitional sample; and
- Final recovery deltas after 24 hours were as follows:
  - Enioda Oxide - 95.8% with O_2 versus 97.1% with air, delta -1.3%;
  - Enioda Trans - 94.2% with O_2 versus 92.9% with air, delta 1.3%;
  - Enioda Fresh - 84.2% with O_2 versus 81.6% with air, delta 2.6%.

Sample ID	Units	ENIODA OXIDE (MC-13)		ENIODA TRANS (MC-14)		ENIODA FRESH (MC-15)	
Test Number		PW8174	PW8187	PW8175	PW8188	PW8176	PW8189
Grind Size (P_{80})	µm	106	106	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
24 Hour	%	95.8	97.1	94.2	92.9	84.2	81.6
40 Hour	%	95.4	95.9	94.2	92.4	86.0	86.7
Residue Grade	g/t	0.07	0.06	0.07	0.09	0.18	0.17
Gravity Recovery	%	29.8	29.6	36.9	28.3	29.3	33.1
Leach Recovery	%	65.6	66.3	57.3	64.1	56.8	53.6
Overall Recovery	%	95.4	95.9	94.2	92.4	86.0	86.7
Calc'd Head	g/t	1.52	1.45	1.20	1.19	1.29	1.28
Lime Cons.	kg/t	1.42	1.21	0.88	0.58	0.32	0.26
Cyanide Cons.	kg/t	0.12	0.15	0.15	0.21	0.14	0.18

Table 13.105 Enioda Master Composites - Oxygen Versus Air Leach Results

Han

The oxygen versus air cyanidation leach test work was conducted on the Han Transitional and Fresh Master composites with results presented in Table 13.106. The following observations can be made:

- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 24 hours, and the final 16 hours recoveries plateaued with the oxygen sparged Transitional sample reporting marginally higher recovery, and the oxygen sparged Fresh sample reporting a higher recovery; and
- Final recovery deltas after 24 hours were as follows:
  - Kilosegui Trans - 94.8% with O_2 versus 90.0% with air, delta 4.8%;
  - Kilosegui Fresh - 87.1% with O_2 versus 81.5% with air, delta 5.6%.

Sample ID	Units	HAN TRANS (MC-17)		HAN FRESH (MC-18)	
Test Number		PW8177	PW8190	PW8178	PW8191
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
24 Hour	%	94.8	90.0	87.1	81.5
40 Hour	%	95.5	93.3	88.1	84.1
Residue Grade	g/t	0.07	0.11	0.21	0.26
Gravity Recovery	%	41.0	40.7	47.7	46.1
Leach Recovery	%	54.6	52.5	40.3	38.0
Overall Recovery	%	95.5	93.3	88.1	84.1
Calc'd Head	g/t	1.57	1.63	1.76	1.63
Lime Cons.	kg/t	0.41	0.52	0.31	0.21
Cyanide Cons.	kg/t	0.12	0.12	0.15	0.18

Table 13.106 Han Master Composites - Oxygen Versus Air Leach Results

Chegue Main

The oxygen versus air cyanidation leach test work was conducted on the Chegue Main Transitional and Fresh Master composites with results presented in Table 13.107 the following observations can be made:

- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 4 hours, however for the proceeding 20 hours the rates were similar, and the final 14 hours recoveries plateaued with the air sparged Transitional sample reporting marginally higher recovery, and the oxygen sparged Fresh sample reporting a higher recovery; and
- Final recovery deltas after 24 hours were as follows:
  - Chegue Main Trans - 94.5% with O_2 versus 91.7% with air, delta 2.8%;
  - Chegue Main Fresh - 85.8% with O_2 versus 82.8% with air, delta 3.0%.

Sample ID	Units	CHEGUE MAIN TRANS (MC-20)		CHEGUE MAIN FRESH (MC-21)	
Test Number		PW8179	PW8192	PW8180	PW8193
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
24 Hour	%	94.5	91.7	85.8	82.8
40 Hour	%	94.0	91.1	88.7	83.7
Residue Grade	g/t	0.07	0.10	0.16	0.22
Gravity Recovery	%	28.2	24.6	44.5	41.7
Leach Recovery	%	65.8	66.5	44.2	42.0
Overall Recovery	%	94.0	91.1	88.7	83.7
Calc'd Head	g/t	1.17	1.07	1.42	1.35
Lime Cons.	kg/t	0.50	0.35	0.29	0.22
Cyanide Cons.	kg/t	0.14	0.20	0.08	0.20

Table 13.107 Chegue Main Master Composites - Oxygen Versus Air Leach Results

Chegue South

The oxygen versus air cyanidation leach test work was conducted on the Chegue South Transitional and Fresh Master composites with results presented in Table 13.108 the following observations can be made:

- In both Transitional and Fresh samples, the rate of reaction with the use of oxygen was higher for the period up to 4 hours, however for the proceeding 20 hours the rates were similar, and the final 14 hours recoveries plateaued with the air sparged Transitional sample reporting marginally higher recovery, and the oxygen sparged Fresh sample reporting a higher recovery; and
- Final recovery deltas after 24 hours were as follows:
  - Chegue South Trans - 87.7% with O_2 versus 84.6% with air, delta 3.1%;
  - Chegue South Fresh - 75.9% with O_2 versus 68.6% with air, delta 7.3%.

Sample ID	Units	CHEGUE SOUTH TRANS (MC-23)		CHEGUE SOUTH FRESH (MC-24)	
Test Number		PW8181	PW8194	PW8182	PW8195
Grind Size (P_{80})	µm	106	106	75	75
% NaCN (Init/Mtn)	%	0.05/0.025	0.05/0.025	0.05/0.025	0.05/0.025
% Solids	%	45	45	45	45
O_2/Air Sparge		O_2	Air	O_2	Air
DCN or CIL		DCN	DCN	DCN	DCN
0 Hour	%	27.4	23.7	32.5	35.2
2 Hour	%	76.8	71.8	58.8	56.1
4 Hour	%	83.0	78.9	66.0	60.6
8 Hour	%	84.7	82.3	68.8	61.6
24 Hour	%	87.7	84.6	75.9	68.6
40 Hour	%	89.9	89.8	80.8	74.6
Residue Grade	g/t	0.14	0.13	0.21	0.31
Gravity Recovery	%	27.4	23.7	32.5	35.2
Leach Recovery	%	62.5	66.1	48.3	39.4
Overall Recovery	%	89.9	89.8	80.8	74.6
Calc'd Head	g/t	0.50	0.50	0.57	0.62
Lime Cons.	kg/t	0.54	0.37	0.32	0.24
Cyanide Cons.	kg/t	0.09	0.27	0.12	0.24

Table 13.108 Chegue South Master Composites - Oxygen Versus Air Leach Results

13.9 Cyanide Optimization Test Work

13.9.1 Cyanide Optimization Test Work

Leach test work was conducted at three (3) cyanide addition rates, 0.07/0.035, 0.05/0.025 and 0.03/0.015 %NaCN (Initial/Maintained) to determine the optimum cyanide addition rate for all future leach tests. 106 µm was identified as the optimum grind size for oxide and transitional ore types and 75 µm for fresh. For the gravity test a 3 kg sub sample was ground to 80% passing the target grind size and passed as a single pass through a 3" standard Knelson concentrator. The Knelson concentrate was subsequently intensively leached to simulate industry standard gravity concentrate leach conditions.

Souwa

Cyanide optimization leach test work results for Souwa are presented in Table 13.109 and leach kinetic curves are illustrated in Figure 13.22. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 83.3% to 94.9%;
- Residue grades ranged from 0.08 g/t to 0.50 g/t;
- Gravity extractions ranged from 26.4% to 42.4%;
- Cyanide consumptions ranged from 0.09 kg/t to 0.21 kg/t;
- The Oxide master composites reported similar gold recoveries at the high and medium cyanide concentrations, 0.07/0.035% CN & 0.05/0.025% CN after 40 hours. A recovery delta at -2.1% was reported at the low concentration (0.03/0.015% CN);
- The Transitional master composites reported similar gold recoveries at all three cyanide concentrations;
- The Fresh master composites reported similar gold recoveries at the high and medium cyanide concentrations, 0.07/0.035% CN and 0.05/0.025% CN after 40 hours. A recovery delta at -3.9% was reported at the low concentration (0.03/0.015% CN); and
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	SOUWA OXIDE (MC-1)			SOUWA TRANS (MC2)			SOUWA FRESH (MC3)		
Test Number		PW_7873	PW_7874	PW_7875	PW_7876	PW_7877	PW_7878	PW_7879	PW_7880	PW_7881
Grid Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	36.3	39.8	42.4	29.0	34.6	26.4	32.6	32.0	28.2
2 Hour	%	84.8	87.1	83.9	86.5	82.1	83.0	72.6	72.1	65.8
4 Hour	%	91.2	91.8	88.8	89.6	86.0	89.6	77.3	77.6	71.9
8 Hour	%	90.7	93.0	90.7	90.9	86.8	89.6	79.4	80.4	76.0
24 Hour	%	92.1	94.0	91.8	92.2	88.3	91.7	84.4	84.9	80.5
40 Hour	%	94.0	94.9	92.8	92.7	89.8	92.8	87.3	87.2	83.3
Residue Grade	g/t	0.31	0.26	0.50	0.10	0.16	0.08	0.18	0.17	0.22
Gravity Recovery	%	36.3	39.8	42.4	29.0	34.6	26.4	32.6	32.0	28.2
Leach Recovery	%	57.7	55.1	50.4	63.6	55.2	66.4	54.7	55.2	55.1
Overall Recovery	%	94.0	94.9	92.8	92.7	89.8	92.8	87.3	87.2	83.3
Calc'd Head Grade	g/t	5.10	5.09	6.94	1.36	1.57	1.11	1.42	1.32	1.31
Lime Cons.	kg/t	0.81	0.80	0.78	0.51	0.52	0.55	0.38	0.36	0.31
Cyanide Cons.	kg/t	0.21	0.15	0.09	0.21	0.15	0.08	0.18	0.12	0.09

Table 13.109 Souwa Cyanide Optimization Leach Results

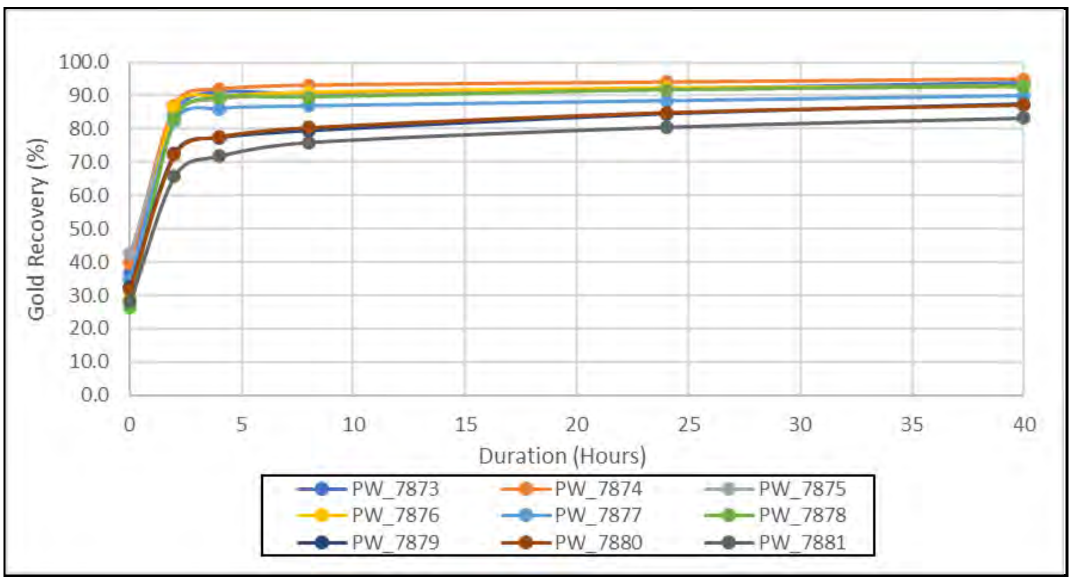


Figure 13.22 Souwa Cyanide Optimization Leach Kinetic Curves

Nokpa

Cyanide optimization leach test work results for Nokpa are presented in Table 13.110 and leach kinetic curves are illustrated in Figure 13.23. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 86.2% to 95.0%;
- Residue grades ranged from 0.14 g/t to 0.19 g/t;
- Gravity extractions ranged from 14.4% to 31.4%;
- Cyanide consumptions ranged from 0.09 kg/t to 0.30 kg/t;
- The Transitional master composites reported similar gold recoveries at the high and medium cyanide concentrations, 0.07/0.035% CN & 0.05/0.025% CN after 40 hours. A recovery delta at -0.5% was reported at the low concentration (0.03/0.015% CN) after 40 hours;
- The Fresh master composites reported similar gold recoveries at all three cyanide concentrations after 40 hours;
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	NOKPA TRANS (MC5)			NOKPA FRESH (MC6)		
Test Number		PW_7882	PW_7883	PW_7884	PW_7885	PW_7886	PW_7887
Grid Size (P_{80})	µm	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	15.5	15.7	14.4	23.3	29.1	31.4
2 Hour	%	87.2	85.1	82.7	69.9	71.6	67.8
4 Hour	%	90.6	89.2	88.2	70.9	72.1	72.5
8 Hour	%	92.8	92.7	90.9	74.2	73.5	74.2
24 Hour	%	94.1	93.2	93.6	81.6	78.5	79.6
40 Hour	%	95.0	94.5	94.0	86.7	86.2	86.7
Residue Grade	g/t	0.14	0.15	0.16	0.17	0.18	0.19
Gravity Recovery	%	15.5	15.7	14.4	23.3	29.1	31.4
Leach Recovery	%	79.5	78.7	79.6	63.5	57.1	55.3
Overall Recovery	%	95.0	94.5	94.0	86.7	86.2	86.7
Calc'd Head Grade	g/t	2.78	2.71	2.67	1.28	1.31	1.43
Lime Cons.	kg/t	0.68	0.72	0.66	0.23	0.22	0.23
Cyanide Cons.	kg/t	0.30	0.15	0.12	0.20	0.14	0.09

Table 13.110 Nokpa Cyanide Optimization Leach Results

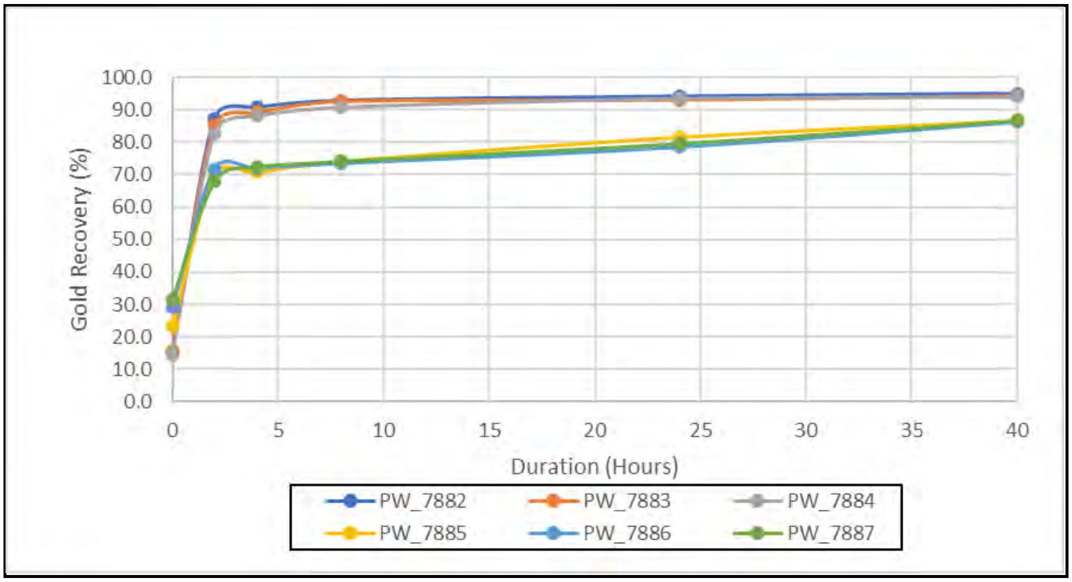


Figure 13.23 Nokpa Cyanide Optimization Leach Kinetic Curves

Kilosegui

Cyanide optimization leach test work results for Kilosegui are presented in Table 13.111 and leach kinetic curves are illustrated in Figure 13.24. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 83.4% to 98.7%;
- Residue grades ranged from 0.01 g/t to 0.23 g/t;
- Gravity extractions ranged from 35.3% to 51.5%;
- Cyanide consumptions ranged from 0.08 kg/t to 0.27 kg/t;
- The Oxide master composites reported marginally higher gold recovery at the high cyanide concentrations at 0.07/0.035% CN after 40 hours. A recovery delta at -1.6% was reported at the medium concentration (0.05/0.025% CN);
- The Transitional master composites reported similar gold recoveries at all three cyanide concentrations;
- The Fresh master composites reported marginally higher gold recovery at the high cyanide concentrations at 0.07/0.035% CN after 40 hours. A recovery delta at -0.6% was reported at the medium concentration (0.05/0.03% CN) and a recovery delta at -1.2% at the low concentration (0.03/0.015% CN);
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	KILOSEGUI OXIDE (MC7)			KILOSEGUI TRANS (MC8)			KILOSEGUI FRESH (MC9)		
Test Number		PW_7888	PW_7889	PW_7890	PW_7891	PW_7892	PW_7893	PW_7894	PW_7895	PW_7896
Grid Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	45.5	37.8	51.5	38.7	35.3	41.4	38.5	40.3	36.4
2 Hour	%	97.2	93.5	94.2	78.3	80.8	76.2	82.2	79.0	77.5
4 Hour	%	97.2	95.3	96.3	85.7	87.5	85.8	83.4	82.3	81.1
8 Hour	%	97.2	97.1	95.6	89.4	89.5	89.6	83.8	84.0	82.5
24 Hour	%	97.2	97.1	97.0	91.2	92.0	92.0	83.8	84.0	83.9
40 Hour	%	98.7	97.1	97.7	91.6	91.5	92.9	84.6	84.0	83.4
Residue Grade	g/t	0.01	0.02	0.02	0.11	0.10	0.09	0.23	0.23	0.22
Gravity Recovery	%	45.5	37.8	51.5	38.7	35.3	41.4	38.5	40.3	36.4
Leach Recovery	%	53.2	59.2	46.2	52.9	56.2	51.4	46.1	0.0	47.0
Overall Recovery	%	98.7	97.1	97.7	91.6	91.5	92.9	84.6	40.3	83.4
Calc'd Head Grade	g/t	0.77	0.68	0.86	1.31	1.18	1.27	1.50	1.44	1.33
Lime Cons.	kg/t	0.75	0.75	0.75	0.50	0.45	0.50	0.30	0.31	0.30
Cyanide Cons.	kg/t	0.27	0.21	0.12	0.18	0.18	0.08	0.24	0.14	0.12

Table 13.111 Kilosegui Cyanide Optimization Leach Results

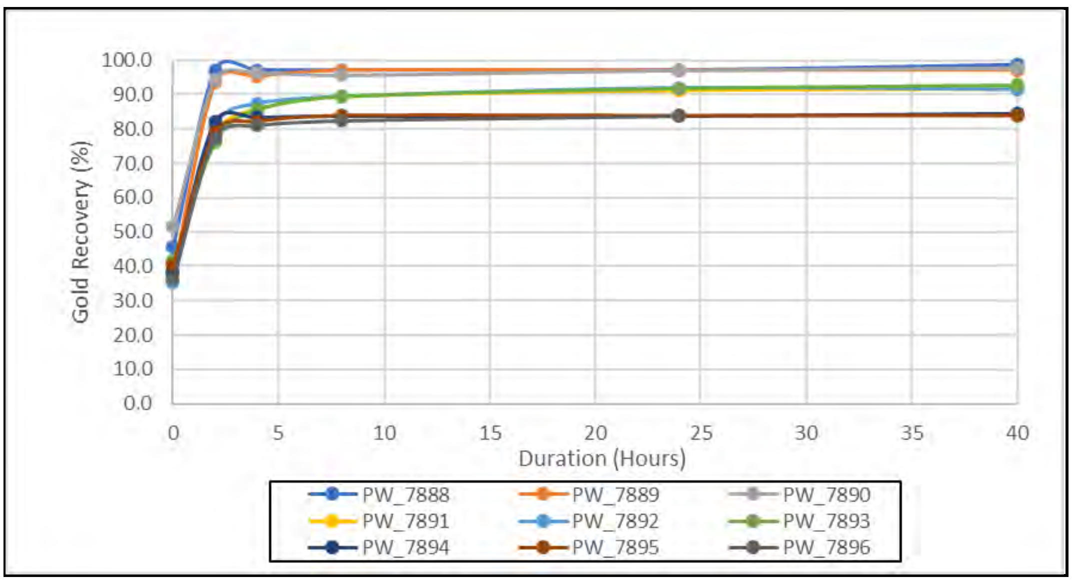


Figure 13.24 Kilosegui Cyanide Optimization Leach Kinetic Curves

Kekeda

Cyanide optimization leach test work results for Kekeda are presented in Table 13.112 and leach kinetic curves are illustrated in Figure 13.25. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 88.0% to 96.0%;
- Residue grades ranged from 0.03 g/t to 0.20 g/t;
- Gravity extractions ranged from 20.8% to 43.1%;
- Cyanide consumptions ranged from 0.12 kg/t to 0.27 kg/t;
- The Oxide and Transitional master composites reported similar gold recoveries at all three cyanide concentrations;
- The Fresh master composites reported marginally higher gold recovery at the high cyanide concentrations at 0.07/0.035% CN after 40 hours. A recovery delta at -0.5% was reported at the medium concentration (0.05/0.03% CN) and a recovery delta at -2.0% at the low concentration (0.03/0.015% CN);
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	KEKEDA OXIDE (MC10)			KEKEDA TRANS (MC11)			KEKEDA FRESH (MC12)		
Test Number		PW_7897	PW_7898	PW_7899	PW_7900	PW_7901	PW_7902	PW_7903	PW_7904	PW_7905
Grid Size (P_{80})	µm	106	106	106	106	106	106	106	106	106
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	27.0	25.5	43.1	30.6	20.8	27.0	34.7	37.0	35.5
2 Hour	%	88.0	87.1	88.5	90.2	84.3	86.7	73.1	70.9	64.7
4 Hour	%	92.7	89.6	92.9	91.8	88.1	90.1	77.6	74.8	70.5
8 Hour	%	92.7	90.4	95.3	92.4	88.1	91.7	81.6	78.7	74.8
24 Hour	%	93.6	91.2	95.9	93.4	90.7	92.3	86.8	86.4	82.7
40 Hour	%	95.4	96.0	95.9	94.5	93.8	94.0	90.0	89.5	88.0
Residue Grade	g/t	0.03	0.03	0.04	0.06	0.07	0.07	0.15	0.18	0.20
Gravity Recovery	%	27.0	25.5	43.1	30.6	20.8	27.0	34.7	37.0	35.5
Leach Recovery	%	68.4	70.5	52.8	64.0	73.0	66.9	55.3	52.5	52.6
Overall Recovery	%	95.4	96.0	95.9	94.5	93.8	94.0	90.0	89.5	88.0
Calc'd Head Grade	g/t	0.65	0.74	0.98	1.10	1.14	1.08	1.50	1.71	1.67
Lime Cons.	kg/t	1.09	1.11	1.11	0.59	0.58	0.61	0.28	0.29	0.33
Cyanide Cons.	kg/t	0.24	0.21	0.12	0.27	0.18	0.12	0.27	0.18	0.12

Table 13.112 Kekeda Cyanide Optimization Leach Results

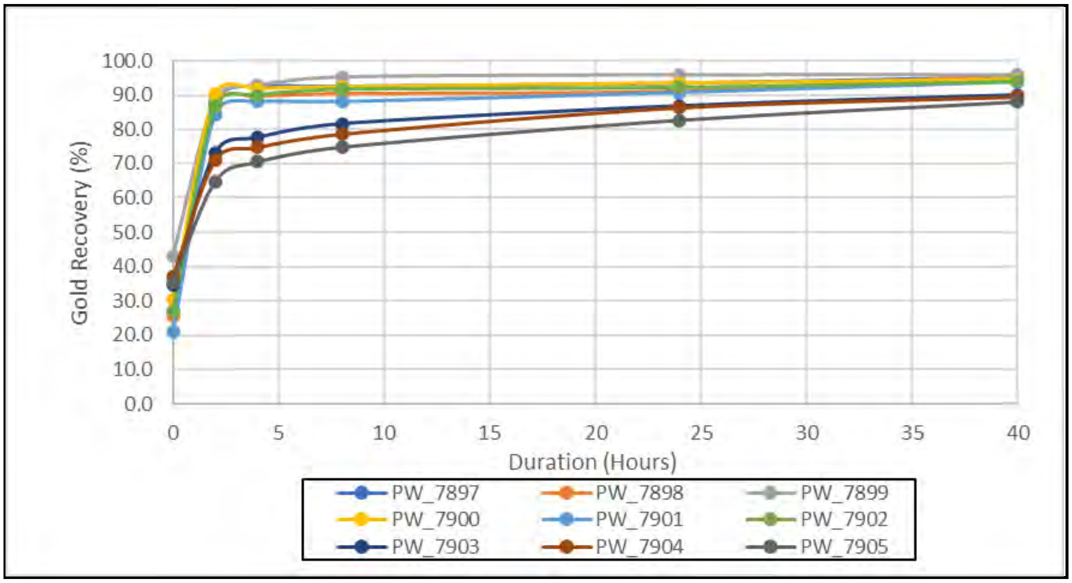


Figure 13.25 Kekeda Cyanide Optimization Kinetic Leach Curves

Enioda

Cyanide optimization leach test work results for Enioda are presented in Table 13.113 and leach kinetic curves are illustrated in Figure 13.26. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 86.0% to 96.3%;
- Residue grades ranged from 0.06 g/t to 0.20 g/t;
- Gravity extractions ranged from 27.5% to 41.6%;
- Cyanide consumptions ranged from 0.09 kg/t to 0.37 kg/t;
- The Oxide composite reported similar gold recoveries at all three cyanide concentrations;
- The Transitional master composites reported marginally higher gold recovery at the medium cyanide concentrations at 0.05/0.025% CN after 40 hours. A recovery delta at -3.5% was reported at the low cyanide concentration (0.03/0.015% CN);
- The Fresh master composites reported inconsistent recoveries with the high and low cyanide concentrations reporting higher recoveries than the medium cyanide concentration leach test. A recovery delta at -4.6% was reported at the medium concentration (0.05/0.03% CN) and a recovery delta at -1.7% at the low concentration (0.03/0.015% CN);
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	ENIODA OXIDE (MC13)			ENIODA TRANS (MC14)			ENIODA FRESH (MC15)		
Test Number		PW_7906	PW_7907	PW_7908	PW_7909	PW_7910	PW_7911	PW_7912	PW_7913	PW_7914
Grid Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	33.0	28.2	27.5	41.6	36.7	48.7	34.4	32.9	36.2
2 Hour	%	89.2	91.0	83.1	84.4	87.0	83.2	81.1	75.8	79.3
4 Hour	%	93.0	93.1	89.7	89.5	93.2	88.2	83.0	79.7	82.9
8 Hour	%	94.8	94.7	93.8	90.0	92.6	90.2	86.8	80.5	85.2
24 Hour	%	95.9	95.9	95.1	92.7	94.8	91.0	88.2	83.9	88.3
40 Hour	%	96.3	95.9	96.3	93.1	95.4	91.8	90.5	86.0	88.8
Residue Grade	g/t	0.06	0.06	0.06	0.09	0.05	0.12	0.12	0.20	0.15
Gravity Recovery	%	33.0	28.2	27.5	41.6	36.7	48.7	34.4	32.9	36.2
Leach Recovery	%	63.3	67.7	68.8	51.5	58.7	43.1	56.2	53.1	52.5
Overall Recovery	%	96.3	95.9	96.3	93.1	95.4	91.8	90.5	86.0	88.8
Calc'd Head Grade	g/t	1.62	1.48	1.47	1.31	1.08	1.47	1.27	1.42	1.33
Lime Cons.	kg/t	1.19	1.17	1.13	0.67	0.69	0.78	0.31	0.34	0.35
Cyanide Cons.	kg/t	0.37	0.27	0.18	0.36	0.24	0.13	0.30	0.27	0.09

Table 13.113 Enioda Cyanide Optimization Leach Results

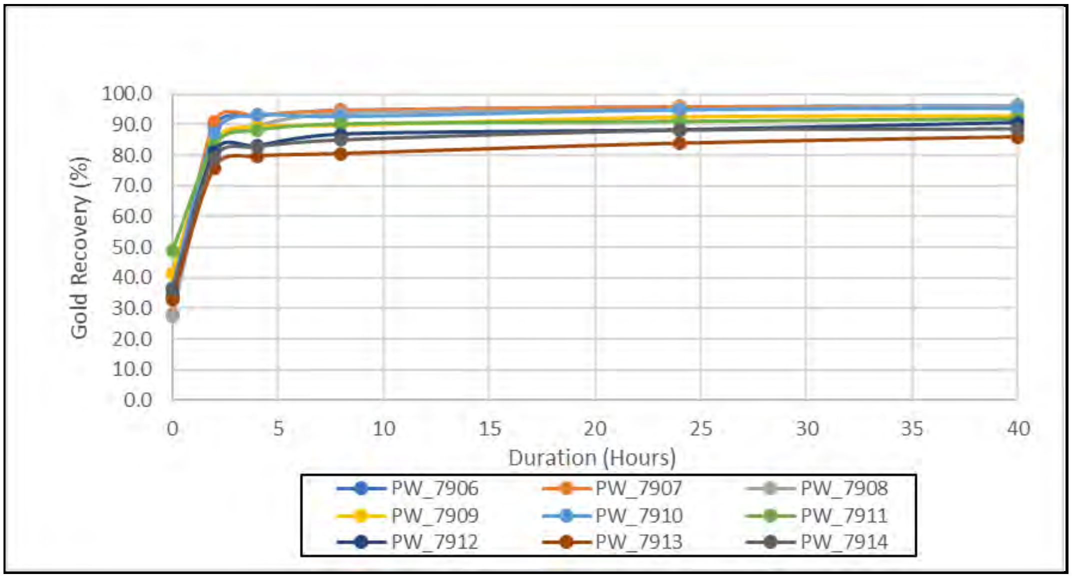


Figure 13.26 Enioda Cyanide Optimization Kinetic Leach Curves

Han

Cyanide optimization leach test work results for Han are presented in Table 13.114 and leach kinetic curves are illustrated in Figure 13.27. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24). The following observations can be made:

- 40 hour gold extractions ranged from 86.9% to 94.3%;
- Residue grades ranged from 0.09 g/t to 0.20 g/t;
- Gravity extractions ranged from 29.6% to 52.5%;
- Cyanide consumptions ranged from 0.12 kg/t to 0.24 kg/t;
- The Han Transitional and Fresh master composites reported high gravity gold components averaging at 45%, which is the likely source of the inconsistent gold recoveries reported at the three cyanide concentrations;
- The gold dissolution kinetics observed were similar which suggest the medium cyanide concentration will be adequate. The optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	HAN TRANS (MC17)			HAN FRESH (MC18)		
Test Number		PW_7915	PW_7916	PW_7917	PW_7918	PW_7919	PW_7920
Grid Size (P_{80})	µm	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	29.6	43.7	52.5	48.6	44.7	49.8
2 Hour	%	79.7	82.2	83.8	80.0	77.5	78.2
4 Hour	%	80.8	83.8	85.0	80.7	78.3	80.1
8 Hour	%	84.5	86.4	87.8	84.2	81.8	82.6
24 Hour	%	87.5	89.8	90.2	87.2	85.0	86.6
40 Hour	%	90.1	94.3	91.4	89.9	86.9	89.1
Residue Grade	g/t	0.16	0.09	0.17	0.16	0.20	0.18
Gravity Recovery	%	29.6	43.7	52.5	48.6	44.7	49.8
Leach Recovery	%	60.5	50.6	38.9	41.3	42.2	39.3
Overall Recovery	%	90.1	94.3	91.4	89.9	86.9	89.1
Calc'd Head Grade	g/t	1.61	1.59	1.97	1.58	1.53	1.65
Lime Cons.	kg/t	0.40	0.42	0.45	0.24	0.26	0.30
Cyanide Cons.	kg/t	0.18	0.18	0.12	0.24	0.20	0.12

Table 13.114 Han Cyanide Optimization Leach Results

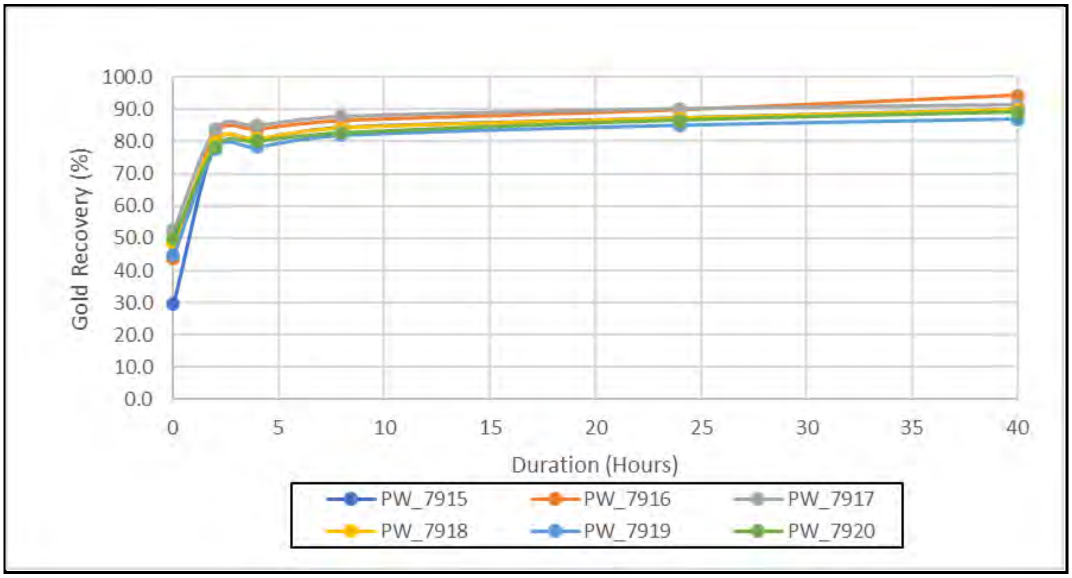


Figure 13.27 Han Cyanide Optimization Kinetic Leach Curves

Chegue Main

Cyanide optimization leach test work results for Chegue Main are presented in Table 13.115 and leach kinetic curves are illustrated in Figure 13.28. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24).

The following observations can be made:

- 40 hour gold extractions ranged from 87.2% to 96.4%;
- Residue grades ranged from 0.03 g/t to 0.18 g/t;
- Gravity extractions ranged from 28.0% to 44.3%;
- Cyanide consumptions ranged from 0.12 kg/t to 0.33 kg/t;
- The Oxide master composite reported similar gold recovery at the three different cyanide concentrations after 40 hours. A recovery delta at -1.8% was reported for the medium cyanide concentration (0.05/0.025% CN) test and a recovery delta at 0.8% for the low cyanide concentration (0.03/0.015% CN) versus the high cyanide concentration (0.07/0.035% CN);
- The Transitional master composite reported similar gold recoveries at the high and medium cyanide concentrations, 0.07/0.035% CN & 0.05/0.025% CN after 40 hours. A recovery delta at -2.1% was reported at the low concentration (0.03/0.015% CN);
- The Fresh master composites reported similar gold recoveries at the three different cyanide concentrations after 40 hours;
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	CHEGUE MAIN OXIDE (MC19)			CHEGUE MAIN TRANS (MC20)			CHEGUE MAIN FRESH (MC21)		
Test Number		PW_7921	PW_7922	PW_7923	PW_7924	PW_7925	PW_7926	PW_7927	PW_7928	PW_7929
Grid Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	34.8	35.4	36.7	28.6	28.0	28.2	43.3	43.8	44.3
2 Hour	%	89.8	86.8	82.2	84.7	84.1	81.5	75.6	78.0	74.5
4 Hour	%	93.5	88.8	90.2	88.9	88.8	85.7	78.6	80.8	78.9
8 Hour	%	93.5	93.3	92.3	89.5	89.9	88.3	80.8	83.1	81.1
24 Hour	%	94.9	94.6	93.6	91.5	93.0	90.9	84.6	86.4	84.6
40 Hour	%	96.4	94.6	95.6	93.1	93.0	90.9	87.2	89.1	88.0
Residue Grade	g/t	0.03	0.05	0.04	0.08	0.08	0.11	0.18	0.14	0.17
Gravity Recovery	%	34.8	35.4	36.7	28.6	28.0	28.2	43.3	43.8	44.3
Leach Recovery	%	61.5	59.2	58.8	64.5	65.0	62.8	43.8	45.3	43.7
Overall Recovery	%	96.4	94.6	95.6	93.1	93.0	90.9	87.2	89.1	88.0
Calc'd Head Grade	g/t	0.82	0.93	0.90	1.15	1.14	1.16	1.40	1.29	1.38
Lime Cons.	kg/t	0.89	0.99	1.02	0.42	0.44	0.49	0.26	0.26	0.31
Cyanide Cons.	kg/t	0.33	0.21	0.12	0.31	0.18	0.12	0.18	0.15	0.09

Table 13.115 Chegue Main Cyanide Optimization Leach Results

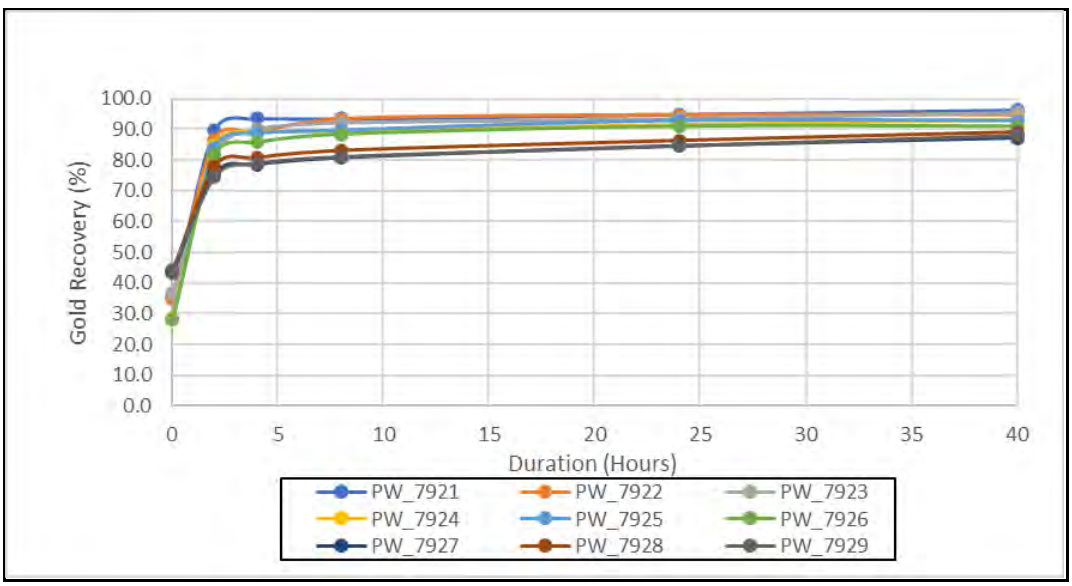


Figure 13.28 Chegue Main Cyanide Optimization Kinetic Leach Curves

Chegue South

Cyanide optimization leach test work results for Chegue South are presented in Table 13.116 and leach kinetic curves are illustrated in Figure 13.29. Comprehensive leach log sheets are provided in Report 6599 (IMO 2024, Ref. 24). The following observations can be made:

- 40 hour gold extractions ranged from 76.1% to 93.2%;
- Residue grades ranged from 0.09 g/t to 0.38 g/t;
- Gravity extractions ranged from 26.8% to 40.5%;
- Cyanide consumptions ranged from 0.06 kg/t to 0.20 kg/t;
- The Transitional master composites reported marginally higher gold recovery at the high cyanide concentration of 0.07/0.035% CN after 40 hours. A recovery delta at -1.1% was reported at the medium concentration (0.05/0.025% CN) after 40 hours;
- The Fresh master composites reported marginally higher gold recovery at the high cyanide concentration of 0.07/0.035% CN after 40 hours. A recovery delta at -2.8% was reported at the medium concentration (0.05/0.025% CN) after 40 hours and -6.3% was reported at the low cyanide concentration 0.03/0.015% CN;
- An optimum cyanide concentration of 0.05/0.025% CN (Initial/maintained) was selected for future leach tests.

Sample ID	Units	CHEGUE SOUTH TRANS (MC23)			CHEGUE SOUTH FRESH (MC24)		
Test Number		PW_7930	PW_7931	PW_7932	PW_7933	PW_7934	PW_7935
Grid Size (P_{80})	µm	106	106	106	75	75	75
% NaCN	%	0.07/0.035	0.05/0.025	0.03/0.015	0.07/0.035	0.05/0.025	0.03/0.015
O_2/Air		O_2	O_2	O_2	O_2	O_2	O_2
DCN or CIL		DCN	DCN	DCN	DCN	DCN	DCN
0 Hour	%	37.5	27.0	26.8	30.7	40.4	35.1
2 Hour	%	84.8	79.2	77.6	59.7	58.8	56.2
4 Hour	%	88.8	85.0	84.5	66.1	62.7	60.0
8 Hour	%	90.6	87.6	87.0	68.2	65.9	62.9
24 Hour	%	92.4	90.7	90.4	77.7	73.9	70.4
40 Hour	%	93.3	92.2	91.8	82.4	79.6	76.1
Residue Grade	g/t	0.09	0.09	0.10	0.20	0.38	0.35
Gravity Recovery	%	37.5	27.0	26.8	30.7	40.4	35.1
Leach Recovery	%	55.8	65.2	65.1	51.7	39.2	41.1
Overall Recovery	%	93.3	92.2	91.8	82.4	79.6	76.1
Calc'd Head Grade	g/t	1.34	1.16	1.23	1.14	1.87	1.44
Lime Cons.	kg/t	0.50	0.50	0.50	0.26	0.30	0.30
Cyanide Cons.	kg/t	0.20	0.15	0.06	0.20	0.12	0.06

Table 13.116 Chegue South Cyanide Optimization Leach Results

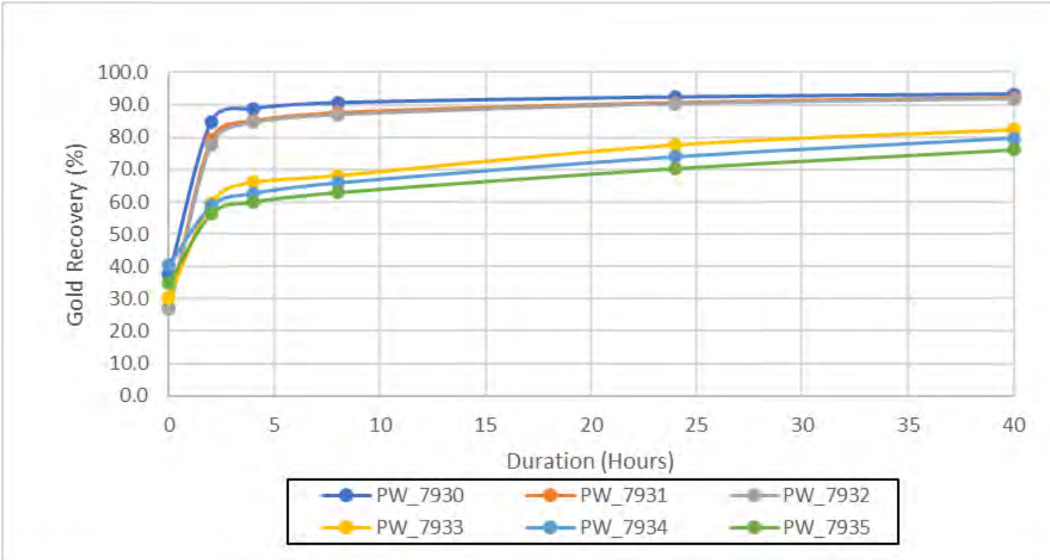


Figure 13.29 Chegue South Cyanide Optimization Kinetic Leach Curves

13.10 Pre-Oxidation Shear Reactor Test Work (Mach©)

A series of cyanidation agitation tests were conducted on four (4) fresh master composites using a MACH© shear reactor to assess the effect the reactor has on gold recovery and reagent consumption.

The tests were conducted at four numbers of passes (0, 1, 2 & 5 passes) through the shear reactor followed by a 6 hour period of agitation denoted as the pre-oxidation step.

During the pre-oxidation step oxygen sparging was applied to maximise dissolved oxygen levels and lime to maintain pH at 10.5.

A base line test as conducted with no passes through the shear reactor.

Two tests were conducted at the same number of passes through the shear reactor.

1. No cyanide added during the 6 hour pre-oxidation step and denoted the Pre-oxidation Test.
2. Cyanide added at an initial concentration at 0.10% CN and denoted the Booster Test.

Following the 6 hour pre-oxidation steps carbon was added and a standard CIL test was completed with cyanide levels maintained at 0.05% CN.

The base line test conducted without the shear reactor was also conducted with and without cyanide addition during the 6 hour pre-oxidation step.

Souwa Fresh Master Composite

Mach reactor test work was conducted on the Souwa Fresh master composite with results presented in Table 13.117. The following observations can be made:

- Similar gold residue grades and overall gold recoveries were reported in all eight (8) CIL cyanidation tests demonstrating no improvement with the use of the shear reactor;
- It is important to note the tests were not conducted at optimum cyanide levels and a different result may have been reported should the tests be conducted at lower cyanide strengths;
- Cyanide consumption was reported 20% higher in the Boosted leach tests compared to the tests where no cyanide was added during the 6 hour pre-oxidation stage. Higher consumption rates are likely due to the additional 6 hours cyanide leach time provided in these tests;
- Lime consumption was reported in the baseline test (PW_7948) with no pre-oxidation or direct cyanide leach stage at 0.61 kg/t. It subsequently decreased to an average of 0.36 kg/t in the shear reactor tests (PW_7949 to 51) and further to 0.22 kg/t in the Boosted leach tests (PW_7953 to 56). A similar relationship but less evident was observed in the Kekeda Fresh composite tests and very little variances were reported in the testing of the Nokpa and Kilosegui Fresh master composites;
- Increasing the number of passes though the shear reactor had no impact on the gold recovery.

Sample ID	Units	Souwa Fresh (MC3)							
		Pre-Oxidation Test				Boosted Test			
Test Number		PW_7948	PW_7949	PW_7950	PW_7951	PW_7953	PW_7954	PW_7955	PW_7956
Grind Size (P_{80})	µm	75	75	75	75	75	75	75	75
6 Hour Pre-Ox with CN		No	No	No	No	Yes	Yes	Yes	Yes
6 Hour Pre-Ox with MACH		No	Yes	Yes	Yes	No	Yes	Yes	Yes
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
NaCN (Maint)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Passes		0 Pass	1 Pass	2 Pass	5 Pass	0 Pass	1 Pass	2 Pass	5 Pass
40 Hour Rec	%	89.0	89.7	90.3	89.7	89.2	89.4	89.2	89.9
Residue Grade	g/t	0.15	0.14	0.14	0.14	0.15	0.16	0.15	0.15
Gravity Recovery	%	25.3	25.4	24.0	25.4	24.3	22.3	24.2	22.7
Leach Recovery	%	63.8	64.2	66.3	64.2	64.9	67.1	64.9	67.2
Overall Recovery	%	89.0	89.7	90.3	89.7	89.2	89.4	89.2	89.9
Calc'd Head Grade	g/t	1.37	1.36	1.44	1.36	1.39	1.51	1.39	1.48
Lime Cons.	kg/t	0.61	0.39	0.36	0.34	0.22	0.22	0.22	0.22
Cyanide Cons.	kg/t	0.97	0.92	0.94	0.97	1.16	1.19	1.16	1.22

Table 13.117 Souwa Fresh Master Composite - Mach Reactor Leach Results

Nokpa Fresh Master Composite

Mach reactor test work was conducted on the Nokpa Fresh master composite with results presented in Table 13.118. The following observations can be made:

- Marginally lower gold residue grades were reported in the Boosted CIL cyanidation tests demonstrating an improvement in gold recovery in the tests conducted with a 6 hour period of cyanidation prior to addition of carbon;
- The highest overall recovery reported was in the Booster CIL cyanidation test (PW_7964) at 94.5% with one pass through the shear reactor. The next highest was reported in the base line test conducted without the shear reactor with cyanide addition during the 6 hour pre-oxidation step (PW_7963) at 94.0%;
- Similar overall gold recoveries were reported following 5 passes through the shear reactor with and without cyanide addition in the 6 hours prior to addition of carbon. Both tests reported lower recoveries at 93.1% and 93.3%, lower than that achieved without use of the shear reactor at 94.0%;
- It is important to note the tests were not conducted at optimum cyanide levels and a different result may have been reported should the tests be conducted at lower cyanide strengths;
- Cyanide consumption was reported at an average of 17% higher in the Boosted leach tests compared to the tests where no cyanide was added during the 6 hour pre-oxidation stage. Higher consumption rates are likely due to the additional 6 hours cyanide leach time provided in these tests;
- Lime consumption was reported marginally lower in the Boosted leach tests (PW_7963 to 66);
- Increasing the number of passes though the shear reactor had no impact on the gold recovery.

Sample ID	Units	Nokpa Fresh (MC6)							
		Pre-Oxidation Test				Boosted Test			
Test Number		PW_7958	PW_7959	PW_7960	PW_7961	PW_7963	PW_7964	PW_7965	PW_7966
Grind Size (P_{80})	µm	75	75	75	75	75	75	75	75
6 Hour Pre-Ox with CN		No	No	No	No	Yes	Yes	Yes	Yes
6 Hour Pre-Ox with MACH		No	Yes	Yes	Yes	No	Yes	Yes	Yes
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maint)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Passes		0 Pass	1 Pass	2 Pass	5 Pass	0 Pass	1 Pass	2 Pass	5 Pass
40 Hour Rec	%	92.8	91.6	91.6	93.1	94.0	94.5	93.3	93.3
Residue Grade	g/t	0.10	0.12	0.11	0.09	0.08	0.08	0.09	0.09
Gravity Recovery	%	24.2	23.5	25.4	25.8	27.7	25.5	27.4	27.4
Leach Recovery	%	68.6	68.0	66.2	67.2	66.3	68.9	65.9	65.9
Overall Recovery	%	92.8	91.6	91.6	93.1	94.0	94.5	93.3	93.3
Calc'd Head Grade	g/t	1.39	1.42	1.32	1.30	1.33	1.44	1.34	1.34
Lime Cons.	kg/t	0.41	0.41	0.42	0.35	0.35	0.35	0.35	0.35
Cyanide Cons.	kg/t	0.85	0.73	0.86	0.92	0.96	0.99	1.06	1.05

Table 13.118 Nokpa Fresh Master Composite - Mach Reactor Leach Results

Kilosegui Fresh Master Composite

Mach reactor test work was conducted on the Kilosegui Fresh master composite with results presented in Table 13.119. The following observations can be made:

- Similar gold residue grades and overall gold recoveries were reported in all eight (8) CIL cyanidation tests demonstrating no improvement with the use of the shear reactor;
- Increasing the number of passes though the shear reactor had no impact on the gold recovery;
- It is important to note the tests were not conducted at optimum cyanide levels and a different result may have been reported should the tests be conducted at lower cyanide strengths;
- Marginally higher average gold recoveries (1.2%) were reported in the Pre-oxidation tests which were undertaken without cyanide addition in the first 6 hour period when compared to the Booster CIL cyanidation tests. The variance may be attributed to a marginally higher average calculated head grade;
- Minor variances were reported in cyanide and lime consumption across the eight (8) CIL cyanidation tests conducted on the Kilosegui Fresh master composite.

Sample ID	Units	Kilosegui Fresh (MC9)							
		Pre-Oxidation Test				Boosted Test			
Test Number		PW_7968	PW_7969	PW_7970	PW_7971	PW_7973	PW_7974	PW_7975	PW_7976
Grind Size (P_{80})	µm	75	75	75	75	75	75	75	75
6 Hour Pre-Ox with CN		No	No	No	No	Yes	Yes	Yes	Yes
6 Hour Pre-Ox with MACH		No	Yes	Yes	Yes	No	Yes	Yes	Yes
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maint)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Passes		0 Pass	1 Pass	2 Pass	5 Pass	0 Pass	1 Pass	2 Pass	5 Pass
40 Hour Rec	%	84.2	86.0	83.5	86.0	83.3	83.3	83.3	84.9
Residue Grade	g/t	0.22	0.18	0.23	0.19	0.21	0.21	0.21	0.20
Gravity Recovery	%	36.1	39.0	36.0	37.1	35.1	35.1	35.1	33.3
Leach Recovery	%	48.1	47.1	47.5	48.9	48.2	48.2	48.2	51.7
Overall Recovery	%	84.2	86.0	83.5	86.0	83.3	83.3	83.3	84.9
Calc'd Head Grade	g/t	1.39	1.29	1.40	1.36	1.26	1.26	1.26	1.33
Lime Cons.	kg/t	0.47	0.56	0.46	0.49	0.41	0.40	0.41	0.40
Cyanide Cons.	kg/t	0.86	0.89	0.83	0.83	0.84	0.78	0.65	0.83

Table 13.119 Kilosegui Fresh Master Composite - Mach Reactor Leach Results

Kekeda Fresh Master Composite

Mach reactor test work was conducted on the Kekeda Fresh master composite with results presented in Table 13.120. The following observations can be made:

- Marginally lower average gold residue grades (0.02 g/t) were reported in the Boosted CIL cyanidation tests demonstrating an improvement in gold recovery in the tests conducted with a 6 hour period of cyanidation prior to addition of carbon;
- The highest overall recovery reported was in the Booster CIL cyanidation test (PW_7985) at 94.3% with one pass through the shear reactor;
- The next highest was reported in the base line test conducted without the shear reactor with cyanide addition during the 6 hour pre-oxidation step (PW_7983) at 94.1% and was 1.6% higher than reported in the Pre-oxidation CIL test (PW_7978), without carbon or cyanide in the first 6 hour period;
- It is important to note the tests were not conducted at optimum cyanide levels and a different result may have been reported should the tests be conducted at lower cyanide strengths;
- Cyanide consumption was reported at an average of 18% higher in the Boosted leach tests compared to the tests where no cyanide was added during the 6 hour pre-oxidation stage. Higher consumption rates are likely due to the additional 6 hours cyanide leach time provided in these tests;
- Lime consumption was reported marginally lower in the Boosted leach tests (PW_7983 to 86);
- Increasing the number of passes though the shear reactor had no impact on the gold recovery.

Sample ID	Units	Kekeda Fresh (MC12)							
		Baseline Test				Boosted Test			
Test Number		PW_7978	PW_7979	PW_7980	PW_7981	PW_7983	PW_7984	PW_7985	PW_7986
Grind Size (P_{80})	µm	75	75	75	75	75	75	75	75
6 Hour Pre-Ox with CN		No	No	No	No	Yes	Yes	Yes	Yes
6 Hour Pre-Ox with MACH		No	Yes	Yes	Yes	No	Yes	Yes	Yes
NaCN (Initial)	%	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
(Maint)	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Passes		0 Pass	1 Pass	2 Pass	5 Pass	0 Pass	1 Pass	2 Pass	5 Pass
40 Hour Rec	%	92.5	92.0	92.0	93.1	94.1	93.8	94.3	93.6
Residue Grade	g/t	0.12	0.13	0.13	0.11	0.09	0.10	0.09	0.11
Gravity Recovery	%	32.8	32.3	32.3	33.0	33.7	31.8	32.4	31.4
Leach Recovery	%	59.7	59.7	59.7	60.1	60.4	62.0	62.0	62.2
Overall Recovery	%	92.5	92.0	92.0	93.1	94.1	93.8	94.3	93.6
Calc'd Head Grade	g/t	1.60	1.62	1.62	1.59	1.52	1.61	1.59	1.64
Lime Cons.	kg/t	0.53	0.44	0.42	0.43	0.33	0.31	0.32	0.32
Cyanide Cons.	kg/t	0.95	1.03	0.98	0.95	1.13	1.19	1.25	1.24

Table 13.120 Kekeda Fresh Master Composite - Mach Reactor Leach Results

13.11 Optimised CIL and Pre-Oxidation CIL Test Work

A set of three (3) Carbon-in-Leach (CIL) cyanidation leach tests, with removal of gravity gold prior to leaching, were conducted on lithology composites from each of the eight (8) open pits. The aim of the tests was to measure and compare the gold recovery reported under previously optimised test conditions, with the recovery achieved following a 6 hour pre-oxidation step (no cyanide), and with and without addition of lead nitrate.

For the gravity test a 3 kg sub sample was ground to 80% passing 106 µm (Oxide & Trans Composites) and 80% passing 75 µm (Fresh composites) and processed as a single pass through a 3" standard Knelson concentrator. The Knelson concentrate was subsequently intensively leached to duplicate industry standard gravity concentrate leach conditions.

The Optimised CIL cyanidation leach test work was conducted in bottle rolls under the following conditions:

- Gravity concentration by Knelson concentrator, amalgamation and leaching of gravity tailings;
- Bottle roll leaches at pulp density of 45% solids in Perth tap water;
- pH 10.0 to 10.5 adjusted with commercial lime (60% available CaO);
- Initial cyanide dosage of 0.05% CN with residual cyanide levels maintained at or above 250 ppm;
- Oxygen sparging to achieve dissolved oxygen level of 20 ppm or greater;
- Activated carbon added at a density of 20 g/L pulp; and
- 40 hour leach duration.

The two Pre-oxidation CIL cyanidation tests were conducted in bottle rolls under the same conditions as the Optimized CIL cyanidation leach tests, however prior to the addition of cyanide and carbon each included a 6 hour pre-oxidation step, with one test dosed with 500 g/t $PbNO_3$.

13.11.1 Souwa Master Composites - MC-1, MC-2 and MC-3

Optimised and Pre-oxidation test work results are presented in Table 13.121.

Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Souwa Oxide Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime consumption was marginally higher in the Pre-oxidation with $PbNO_3$ CIL cyanidation test (PW_8090).

Souwa Transitional Composite

- A notable improvement in gold recovery (3.5%) was reported in the Pre-oxidation - no $PbNO_3$ CIL cyanidation test compared to the baseline test with no 6 hour pre-oxidation step; and
- Lime or cyanide consumption were consistent in the three tests.

Souwa Fresh Composite

- A marginal increase in gold recovery (1.5%) was reported in the Pre-oxidation with $PbNO_3$ CIL cyanidation test compared to the test with no 6 hour Pre-oxidation step; and
- Lime consumption was marginally higher in the Pre-oxidation with $PbNO_3$ CIL cyanidation test (PW_8092).

Sample ID	Units	SOUWA OXIDE (MC-1)			SOUWA TRANS (MC-2)			SOUWA FRESH (MC-3)		
Test Number		PW8074	PW8090	PW8095	PW8075	PW8091	PW8096	PW8076	PW8092	PW8097
Grind Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
Pre-oxidation	hours		6	6		6	6		6	6
$PbNO_3$	ppm		500			500			500	
Residue Grade	g/t	0.40	0.46	0.42	0.17	0.13	0.09	0.16	0.14	0.17
Gravity Recovery	%	37.0	39.4	37.6	38.1	25.4	36.6	29.8	30.7	29.9
Leach Recovery	%	54.7	52.7	54.9	52.6	65.8	57.5	58.1	58.8	57.9
Overall Recovery	%	91.7	92.0	92.5	90.7	91.2	94.2	88.0	89.5	87.8
Calc'd Head	g/t	1.92	2.12	2.04	0.93	1.09	1.00	1.33	1.33	1.40
Lime Cons.	kg/t	0.55	0.97	0.77	0.78	0.62	0.67	0.27	0.42	0.29
Cyanide Cons.	kg/t	0.65	0.65	0.57	0.48	0.60	0.53	0.57	0.55	0.53

Table 13.121 Souwa Composite Pre-Oxidation Leach Results

13.11.2 Nokpa Master Composites - MC-5 and MC-6

Optimised and Pre-Oxidation test work results are presented in Table 13.122. Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Nokpa Transitional Composite

- The highest recovery achieved was in the CIL test however this was largely due to the high gravity recovery (62.0%) compared to 13.5% and 15.2% in the pre-oxidation tests;
- There was no significant improvement between the pre-oxidation tests with and without lead nitrate;
- Lime and cyanide consumptions were consistent in the baseline pre-oxidation and CIL test with elevated lime consumption in the lead nitrate test.

Nokpa Fresh Composite

- A recovery comparison was not possible due to the high variability in head grades; however, it is notable that all three tests reported similar gold residue grades; and
- Lime or cyanide consumption were consistent in the three tests.

Sample ID	Units	NOKPA TRANS (MC-5)			NOKPA FRESH (MC-6)		
Test Number		PW8077	PW8093	PW8098	PW8079	PW8094	PW8099
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours		6	6		6	6
$PbNO_3$	ppm		500			500	
Residue Grade	g/t	0.13	0.17	0.18	0.11	0.12	0.12
Gravity Recovery	%	62.0	13.5	15.2	17.8	31.5	31.6
Leach Recovery	%	33.1	79.8	78.4	77.3	59.0	60.9
Overall Recovery	%	95.1	93.2	93.6	95.1	90.5	92.5
Calc'd Head	g/t	2.64	2.44	2.74	2.26	1.26	1.61
Lime Cons.	kg/t	0.80	1.02	0.80	0.25	0.37	0.33
Cyanide Cons.	kg/t	0.71	0.59	0.65	0.45	0.55	0.48

Table 13.122 Nokpa Composites Pre-Oxidation Leach Results

13.11.3 Kilosegui Master Composites - MC-8 and MC-9

Optimised and Pre-Oxidation test work results are presented in Table 13.123. Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Kilosegui Transitional Composite

- A notable improvement in gold recovery (3.2%) was reported in the pre-oxidation - $PbNO_3$ CIL cyanidation test compared to the pre-oxidation without $PbNO_3$ however this may be due to the increased gravity recovery +15.1%; and
- Lime and cyanide consumptions were different across all three tests with the pre-oxidation baseline test having a significantly lower lime consumption 0.08 g/t compared to 0.73 g/t and 0.62 g/t.

Kilosegui Fresh Composite

- A notable improvement in gold recovery (7.52%) was observed in the Pre-oxidation without $PbNO_3$ CIL cyanidation test compared to the test with $PbNO_3$. The test without the 6 hour Pre-oxidation step obtained the highest recovery (95.1%); and
- Lime or cyanide consumption were consistent in the three tests.

Sample ID	Units	KILOSEGUI TRANS (MC-8)			KILOSEGUI FRESH (MC-9)		
Test Number		PW8355	PW8398	PW8411	PW8348	PW8392	PW8405
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours		6	6		6	6
$PbNO_3$	ppm		500			500	0
Residue Grade	g/t	0.13	0.13	0.15	0.25	0.28	0.23
Gravity Recovery	%	41.5	45.1	30.0	43.2	36.9	49.8
Leach Recovery	%	47.9	46.2	58.2	46.8	43.4	37.8
Overall Recovery	%	89.4	91.4	88.2	90.0	80.3	87.5
Calc'd Head	g/t	1.23	1.51	1.27	2.51	1.42	1.81
Lime Cons.	kg/t	0.73	0.62	0.08	0.25	0.36	0.31
Cyanide Cons.	kg/t	0.36	0.51	0.51	0.59	0.51	0.51

Table 13.123 Kilosegui Composites Pre-Oxidation Leach Results

13.11.4 Kekeda Master Composites - MC-11 and MC-12

Optimised and Pre-Oxidation test work results are presented in Table 13.124. Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Kekeda Transitional Composite

- A notable improvement in gold recovery (3.5%) was reported in the pre-oxidation - $PbNO_3$ CIL cyanidation test compared to the pre-oxidation without $PbNO_3$; and
- Lime and cyanide consumptions were consistent across all three tests.

Kekeda Fresh Composite

- Recoveries were consistent across all three tests;
- The pre-oxidation without $PbNO_3$ reported an elevated gravity gold component (47.2%), however, all three tests reported similar residue gold grades; and
- Lime or cyanide consumption were consistent in the three tests.

Sample ID	Units	KEKEDA TRANS (MC-11)			KEKEDA FRESH (MC-12)		
Test Number		PW8357	PW8399	PW8412	PW8349	PW8393	PW8406
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours	0	6	6	0	6	6
$PbNO_3$	ppm	0	500	0	0	500	0
Residue Grade	g/t	0.08	0.05	0.10	0.11	0.12	0.10
Gravity Recovery	%	37.0	21.3	23.5	39.3	41.7	47.2
Leach Recovery	%	57.7	74.0	68.3	54.0	51.5	47.4
Overall Recovery	%	94.8	95.3	91.8	93.3	93.2	94.6
Calc'd Head	g/t	1.53	2.15	1.22	1.64	1.76	1.86
Lime Cons.	kg/t	0.61	0.87	0.66	0.26	0.36	0.41
Cyanide Cons.	kg/t	0.60	0.57	0.53	0.57	0.55	0.51

Table 13.124 Kekeda Composites Pre-Oxidation Leach Results

13.11.5 Enioda Master Composites - MC-13, MC-14 and MC-15

Optimised and Pre-Oxidation test work results are presented in Table 13.125. Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Enioda Oxide Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime consumption was significantly lower in the Pre-oxidation without $PbNO_3$ CIL cyanidation test (PW_8413).

Enioda Transitional Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime consumption was significantly lower in the Pre-oxidation without $PbNO_3$ CIL cyanidation test (PW_8414).

Enioda Fresh Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime and cyanide consumptions were similar across all three tests.

Sample ID	Units	ENIODA OXIDE (MC13)			ENIODA TRANS (MC14)			ENIODA FRESH (MC15)		
Test Number		PW8358	PW8400	PW8413	PW8359	PW8401	PW8414	PW8350	PW8394	PW8407
Grind Size (P_{80})	µm	106	106	106	106	106	106	75	75	75
Pre-oxidation	hours	0	6	6	0	6	6	0	6	6
$PbNO_3$	ppm	0	500	0	0	500	0	0	500	0
Residue Grade	g/t	0.06	0.04	0.05	0.09	0.07	0.08	0.17	0.17	0.18
Gravity Recovery	%	28.8	26.1	28.4	46.8	40.5	42.1	47.1	45.6	44.1
Leach Recovery	%	67.7	71.2	68.0	46.9	54.0	51.7	42.4	44.1	44.3
Overall Recovery	%	96.5	97.3	96.4	93.8	94.5	93.8	89.5	89.7	88.4
Calc'd Head	g/t	1.57	1.51	1.39	1.44	1.28	1.28	1.63	1.61	1.56
Lime Cons.	kg/t	1.00	1.37	0.35	0.59	0.89	0.21	0.27	0.36	0.36
Cyanide Cons.	kg/t	0.72	0.63	0.69	0.72	0.63	0.65	0.57	0.57	0.53

Table 13.125 Enioda Composites Pre-Oxidation Leach Results

13.11.6 Han Master Composites - MC-17 & MC-18

Optimised and Pre-Oxidation test work results are presented in Table 13.126. Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Han Transitional Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime consumption was higher in both Pre-oxidation tests, 0.55 g/t and 0.59 g/t compared to 0.33 kg/t in the baseline CIL test; and
- Cyanide consumption was consistent across all three tests.

Han Fresh Composite

- Both the Pre-oxidation test with $PbNO_3$ (92.4%) and the baseline CIL test (93.3%) reported similar recoveries, and the Pre-oxidation test without $PbNO_3$ reported lower recovery at 88.4%. Residue gold grades were similar in all three tests; and
- Lime and cyanide consumptions were similar across all three tests with the exception of the baseline CIL cyanide consumption which was higher (0.76 kg/t compared to 0.55 kg/t and 0.51 kg/t).

Sample ID	Units	HAN TRANS (MC-17)			HAN FRESH (MC-18)		
Test Number		PW8361	PW8402	PW8361	PW8351	PW8395	PW8408
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours	0	6	6	0	6	6
$PbNO_3$	ppm	0	500	0	0	500	0
Residue Grade	g/t	0.14	0.08	0.14	0.16	0.17	0.16
Gravity Recovery	%	33.2	39.1	33.2	60.5	59.1	42.1
Leach Recovery	%	60.1	55.8	60.1	32.8	33.3	46.3
Overall Recovery	%	93.3	94.9	93.3	93.3	92.4	88.4
Calc'd Head	g/t	2.09	1.57	2.09	2.40	2.24	1.38
Lime Cons.	kg/t	0.33	0.55	0.33	0.24	0.35	0.33
Cyanide Cons.	kg/t	0.53	0.49	0.53	0.00	0.00	0.51

Table 13.126 Han Composites Pre-Oxidation Leach Results

13.11.7 Chegue Main Master Composites - MC-20 and MC-21

Optimised and Pre-Oxidation test work results are presented in Table 13.127 Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Chegue Main Transitional Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime or cyanide consumption were consistent in the three tests.

Chegue Main Fresh Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime or cyanide consumption were consistent in the three tests.

Sample ID	Units	CHEGUE MAIN TRANS (MC-20)			CHEGUE MAIN FRESH (MC-21)		
Test Number		PW8363	PW8403	PW8416	PW8352	PW8396	PW8409
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours	0	6	6	0	6	6
$PbNO_3$	ppm	0	500	0	0	500	0
Residue Grade	g/t	0.09	0.09	0.09	0.15	0.14	0.12
Gravity Recovery	%	26.2	26.4	28.92	47.8	50.2	48.3
Leach Recovery	%	65.5	65.2	64.0	42.8	40.2	44.8
Overall Recovery	%	91.6	91.6	93.0	90.7	90.4	93.2
Calc'd Head	g/t	1.07	1.07	1.28	1.61	1.46	1.76
Lime Cons.	kg/t	0.54	0.58	0.50	0.23	0.34	0.33
Cyanide Cons.	kg/t	0.48	0.51	0.51	0.53	0.55	0.57

Table 13.127 Chegue Main Composites Pre-Oxidation Leach Results

13.11.8 Chegue South Master Composites - MC23 and MC24

Optimised and Pre-Oxidation test work results are presented in Table 13.128 Comprehensive log sheets are provided in Report 6599 (IMO 2024, Ref. 24) and the following observations can be made:

Chegue South Fresh Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime or cyanide consumption were consistent in the three tests except the lime consumption in the Pre-oxidation test without lead nitrate at 0.59 kg/t compared to 0.25 kg/t and 0.34 kg/t.

Chegue South Transitional Composite

- A marginal improvement in gold recovery (1.6%) was reported in the pre-oxidation - $PbNO_3$ CIL cyanidation test compared to the pre-oxidation without $PbNO_3$; and
- Lime or cyanide consumption were consistent in the three tests except the lime consumption in the Pre-oxidation test with lead nitrate at 0.83 kg/t compared to 0.42 kg/t and 0.50 kg/t.

Chegue South Fresh Composite

- The three CIL cyanidation tests reported similar gold recoveries demonstrating no benefit from the 6 hour pre-oxidation step, with or without the addition of $PbNO_3$; and
- Lime or cyanide consumption were consistent in the three tests except the lime consumption in the Pre-oxidation test without lead nitrate at 0.59 kg/t compared to 0.25 kg/t and 0.34 kg/t.

Sample ID	Units	CHEGUE SOUTH TRANS (MC23)			CHEGUE SOUTH FRESH (MC24)		
Test Number		PW8365	PW8404	PW8417	PW8353	PW8397	PW8410
Grind Size (P_{80})	µm	106	106	106	75	75	75
Pre-oxidation	hours	0	6	6	0	6	6
$PbNO_3$	ppm	0	500	0	0	500	0
Residue Grade	g/t	0.13	0.08	0.12	0.25	0.22	0.24
Gravity Recovery	%	29.4	29.3	30.2	38.5	37.3	36.4
Leach Recovery	%	60.8	64.2	61.7	45.3	47.1	45.1
Overall Recovery	%	90.2	93.5	91.9	83.8	84.4	81.5
Calc'd Head	g/t	1.27	1.24	1.48	1.51	1.41	1.30
Lime Cons.	kg/t	0.42	0.83	0.50	0.25	0.34	0.59
Cyanide Cons.	kg/t	0.53	0.39	0.54	0.60	0.57	0.45

Table 13.128 Chegue South Pre-Oxidation Leach Results

13.12 Thickener Test Work

A series of dynamic thickener tests were conducted on the bulk leach tails from Souwa and Nokpa Master Composites with the aim to determine flux rates to size the thickeners and identify optimum flocculant dose rates for overflow clarity. The thickener test work was conducted by Metso at their Perth Technology Centre Bayswater, Australia. A comprehensive analysis is available as part of the Metso report provided in Report 6599 (IMO 2024, Ref. 24). A summary of the thickening test work results is provided in Table 13.129 and the following observations can be made:

- Solids loadings were reported in the 0.75 to 1.0 t/m²/h range indicating a thickener diameter of between 30 - 35 m will be required;
- Underflow slurry densities at or above 55% solids were achieved, except for the Nokpa Transitional sample reporting at 48% solids which also required 300 g/t of coagulant to meet the <100 mg/L TSS overflow clarity guidelines; and
- The Souwa Oxide Master composite dynamic thickener test achieved an underflow slurry density of 55% solids, however required high flocculant dose rates at 120 g/t and reported poor overflow clarity more than 1,000 mg/L.

Process Stream	Units	SOUWA Oxide (MC-1)	SOUWA Trans (MC-2)	SOUWA Fresh (MC-3)	NOKPA Trans (MC-5)	NOKPA Fresh (MC-6)
Solids Feed Rate	t/h	675	675	500	675	500
Solids Loading	t/(m²/h)	0.75	1.0	1.0	0.75	1.0
Feed Slurry Density	% Solids (w/w)	40	40	40	40	40
Diluted Feedwell Slurry Density	% Solids (w/w)	18	18	18	12	12
Slurry pH		8.6	9.1	9.1	9.4	8.9
Coagulant Dosage	g/t	-	-	-	300	-

Process Stream	Units	SOUWA Oxide (MC-1)	SOUWA Trans (MC-2)	SOUWA Fresh (MC-3)	NOKPA Trans (MC-5)	NOKPA Fresh (MC-6)
Flocculant Dosage	g/t	120	40	10	30	10
Underflow Density	% Solids (w/w)	55	60	62	48	67
Yield Stress	Pa	243	38	43	36	30
Overflow Clarity	mg/L	1060	<100	<100	<100	<100
Required Thickener Diameter	m	34	30	26	34	26

Table 13.129 Thickener Test Work Results

13.13 Cyanide Detox Test Work

Cyanide detox test work was conducted with as a continuous and bulk test. Results are presented in Table 13.130 to Table 13.140 Conditions are presented in Table 13.130.

The cyanide detoxification tests conducted on the bulk Souwa thickener test samples reported residence times in the range 80 - 95 minutes from the three lithology types. The tests conducted on the Nokpa Transitional and Fresh composites however, reported contrasting residence time requirements at 50 minutes and 150 minutes respectively.

Conditions	Units	PW8103-D1	PW8103-B1	PW8104-D1	PW8104-B1	PW8100-D1	PW8100-B2	PW8110-D1	PW8110-B1	PW8108-D1	PW8108-B1
		SOUWA Oxide (MC1)		SOUWA Trans (MC2)		SOUWA Fresh (MC3)		NOKPA Trans (MC5)		NOKPA Fresh (MC6)	
pH		8.51	8.51	8.51	8.50	8.51	8.50	8.53	8.50	8.51	8.51
D.O.	ppm	6.13	8.31	6.35	8.41	7.25	13.00	7.08	8.13	6.75	6.76
ISE CN	mV	-12.5	-15.0	-59.0	20.0	28.9	+30	27.2	19.0	-172	10.0
Temp.	°C	23.0	22.6	21.7	24.0	24.1	26.5	25.4	24.0	23.5	27.0
SMBS	mL/hr	10.2	95.2	10.0	95.3	10.0	77.4	10.3	89.4	10.3	94.6
SMBS	kg/t	1.58	0.63	2.35	3.05	2.29	1.39	0.96	1.56	3.13	2.52
$CuSO_4.5H_2O$	mL/hr	9.67	97.8	9.50	96.8	9.50	84.6	9.63	94.8	9.75	119
$CuSO_4.5H_2O$	kg/t	0.19	0.08	0.12	0.26	0.09	0.09	0.13	0.33	0.13	0.11
Feed	mL/hr	1,600	23,055	1,200	23,806	600	25,048	1,200	23,877	1,200	16,883
Lime	mL/hr	26.7	71.7	25.0	118.1	22.5	293.7	6.3	18.0	21.3	79.0
CN Load	g/hr	0.09	1.37	0.09	2.17	0.03	1.28	0.04	1.34	0.14	2.39
SO_2 Ratio	g SO_2/g CN	8.79	8.77	9.12	7.54	13.40	13.01	8.01	3.60	8.00	7.80
$CuSO_4.5H_2O$	mg/L	110	142	72.2	118	41.3	109	78.4	90.1	74.1	178
Lime	/ g SO_2	0.69	0.46	0.58	0.48	1.03	0.56	0.36	0.16	0.37	0.53
Retention	min	29.2	92.0	38.6	80.0	74.8	95.0	39.1	50.0	38.7	150
Solution	L/hr	1.23	18.0	0.92	18.5	0.46	19.5	0.92	18.5	0.92	13.5

Table 13.130 Cyanide Detox Test Work Conditions

Souwa Oxide

The Souwa Oxide detox test work results for continuous and bulk are presented in Table 13.131 and Table 13.132.

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.013	71.1	2	2	<0.5	1.2	76.7	0
DISP 1A	0-40	N/A	0.17	<0.2	<1.0	<0.5	<0.2	1.57	99.8
DISP 1B	40-80	N/A	0.39	<0.2	<1.0	<0.5	<0.2	1.79	99.5
DISP 2A	80-120	N/A	0.33	<0.2	<1.0	<0.5	<0.2	1.72	99.5
DISP 2B	120-160	N/A	0.20	<0.2	<1.0	<0.5	<0.2	1.59	99.7
DISP 3A	160-200	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.9
DISP 3B	200-240	N/A	0.20	<0.2	<1.0	<0.5	<0.2	1.60	99.7

Table 13.131 Souwa Oxide Continuous Cyanide Detoxification Results

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.01	77.4	2.00	12.0	<0.5	1.2	111	0
FINAL	95	N/A	0.30	<0.2	<1.0	<0.5	<0.2	1.70	99.6

Table 13.132 Souwa Oxide Bulk Cyanide Detoxification Results

Souwa Trans

The Souwa Trans detox test work results for continuous and bulk are presented in Table 13.133 and Table 13.134.

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.020	102	1.20	<1.0	<0.5	1.200	103	0.0
DISP 1A	0-20	N/A	0.48	0.60	<1.0	<0.5	<0.2	1.87	99.5
DISP 1B	20-40	N/A	0.19	0.40	<1.0	<0.5	<0.2	1.58	99.8
DISP 2A	40-60	N/A	0.29	0.20	<1.0	<0.5	<0.2	1.69	99.7
DISP 2B	60-80	N/A	0.17	0.40	<1.0	<0.5	<0.2	1.56	99.8
DISP 3A	80-100	N/A	0.07	0.20	<1.0	<0.5	<0.2	1.47	99.9
DISP 3B	100-120	N/A	0.14	<0.2	<1.0	<0.5	<0.2	1.54	99.9

Table 13.133 Souwa Trans Continuous Cyanide Detoxification Results

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.02	119	0.80	1.00	<0.5	1.00	121	0
FINAL	80	N/A	0.00	0.40	<1.0	<0.5	<0.2	1.0	100

Table 13.134 Souwa Trans Bulk Cyanide Detoxification Results

Souwa Fresh

The Souwa Fresh detox test work results for continuous and bulk are presented in Table 13.135 and Table 13.136.

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%
FEED	0	0.019	71	4	1	<0.5	0.6	73.8	0
DISP 1A	0-40	N/A	0.17	<0.2	<1.0	<0.5	<0.2	1.57	99.8
DISP 1B	40-80	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.9
DISP 2A	80-120	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.9
DISP 2B	120-160	N/A	0.20	<0.2	<1.0	<0.5	<0.2	1.60	99.7
DISP 3A	160-200	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.9
DISP 3B	200-240	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.9

Table 13.135 Souwa Fresh Continuous Cyanide Detoxification Results

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.02	66.7	4.00	2.00	<0.5	0.60	72.3	0.0
FINAL	95	N/A	2.00	3.00	<1.0	<0.5	<0.2	3.40	97.0

Table 13.136 Souwa Fresh Bulk Cyanide Detoxification Results

Nokpa Trans

The Nokpa Trans detox test work results for continuous and bulk are presented in Table 13.137 and Table 13.138.

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%
FEED	0	0.004	47.5	0.40	<1.0	<0.5	<0.2	48.9	0
DISP 1A	0-20	N/A	0.26	<0.2	<1.0	<0.5	<0.2	1.66	99.4
DISP 1B	20-40	N/A	0.28	<0.2	<1.0	<0.5	<0.2	1.67	99.4
DISP 2A	40-60	N/A	0.11	<0.2	<1.0	<0.5	<0.2	1.50	99.8
DISP 2B	60-80	N/A	0.14	<0.2	<1.0	<0.5	<0.2	1.54	99.7
DISP 3A	80-100	N/A	0.10	<0.2	<1.0	<0.5	<0.2	1.50	99.8
DISP 3B	100-120	N/A	0.22	<0.2	<1.0	<0.5	<0.2	1.62	99.5

Table 13.137 Nokpa Trans Continuous Cyanide Detoxification Results

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%
FEED	0	0.01	73.0	0.20	<1.0	<0.5	0.20	74.4	0.0
FINAL	50	N/A	0.50	<0.2	<1.0	<0.5	<0.2	1.90	99.3

Table 13.138 Nokpa Trans Bulk Cyanide Detoxification Results

Nokpa Fresh

The Nokpa Fresh detox test work results for continuous and bulk are presented in Table 13.139 and Table 13.140.

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.025	155	1.80	<1.0	<0.5	0.40	156	0.0
DISP 1A	0-20	N/A	0.24	1.80	<1.0	<0.5	<0.2	1.64	99.8
DISP 1B	20-40	N/A	1.56	5.80	<1.0	<0.5	<0.2	2.96	99.0
DISP 2A	40-60	N/A	1.14	3.00	<1.0	<0.5	<0.2	2.54	99.3
DISP 2B	60-80	N/A	1.32	5.00	<1.0	<0.5	<0.2	2.72	99.1
DISP 3A	80-100	N/A	4.41	8.20	<1.0	<0.5	<0.2	5.81	97.2
DISP 3B	100-120	N/A	2.23	5.80	<1.0	<0.5	<0.2	3.63	98.6

Table 13.139 Nokpa Fresh Continuous Cyanide Detoxification Results

Sample ID	Period	CN_{FREE}	CN_p	Cu	Fe	Ni	Zn	Calculated	WAD CN
Units	Mins	%	mg/L	mg/L	mg/L	mg/L	mg/L	CN Total	Reduction (%)
FEED	0	0.02	185	1.80	7.00	<0.5	0.60	204	0.0
FINAL	150	N/A	2.93	8.00	<1.0	<0.5	<0.2	4.33	98.4

Table 13.140 Nokpa Fresh Bulk Cyanide Detoxification Results

13.14 Gold Recovery - Regression Analysis

A series of regression analyses were conducted to assess the impact of leach feed head gold grade on (40-hr) leach residue grade and extraction. The regression format is presented in Table 13.141 and coefficients in Table 13.142 for each lithology type in the eight (8) deposits. The comparison to actual are depicted in Figure 13.30 - Figure 13.37 and reported a high level of correlation albeit it was from a limited data set.

A modified fixed tail grade recovery model may fit with low grade deposits however, the head grades vary significantly throughout the Doropo gold deposits which limits the accuracy of this type of model.

The leach test work data supporting the regression analyses has been provided in Report 6599 (IMO 2024, Ref. 24).

Y = a*(head grade) + b		
Y	*	b
Leach Residue Au Grade (g/t)	Leach Feed Au Grade (g/t	Intercept

Table 13.141 Leach Recovery Regression Format

Composite	Slope	Intercept
Coefficient	a	b
Souwa Oxide	0.935	0.019
Souwa Trans	0.995	-0.077
Souwa Fresh	0.921	-0.077
Nokpa Trans	0.932	0.030
Nokpa Fresh	0.958	-0.121
Kilosegui Oxide	1.006	-0.022
Kilosegui Trans	0.954	-0.036
Kilosegui Fresh	0.865	-0.013
Kekeda Oxide	0.967	-0.007
Kekeda Trans	0.995	-0.065
Kekeda Fresh	0.954	-0.086
Enioda Oxide	1.004	-0.050
Enioda Trans	0.994	-0.075
Enioda Fresh	0.994	-0.122
Han Trans	0.902	0.042
Han Fresh	0.966	-0.147
Chegue Main Oxide	0.994	-0.040
Chegue Main Trans	0.917	0.010
Chegue Main Fresh	0.922	-0.038
Chegue South Trans	0.891	0.008
Chegue South Fresh	0.717	0.136

Table 13.142 Leach Recovery Regression Co-efficients

13.14.1 Souwa Recovery Regression

Oxide

The data selected for the regression analysis on Souwa Oxide ores was derived from a series of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised and from composites reporting high and low head grades. Two LeachWELL soluble Au assays results were included from the FS test programme. The results from a total of 18 leach tests were used and is presented in Figure 13.30 below.

Transitional

The data selected for the regression analysis on Souwa Transitional ores was derived from a limited number of leach tests conducted in the FS. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from tests in the PFS test work programme were not used in the regression analysis due to significantly lower head grades. A single LeachWELL soluble Au assay was included from the FS test work programme. The results from a total of 5 leach tests were used and are presented in Figure 13.30 below.

Fresh

The data selected for the regression analysis on Souwa Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. A single LeachWELL soluble Au assay was included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.30 below.

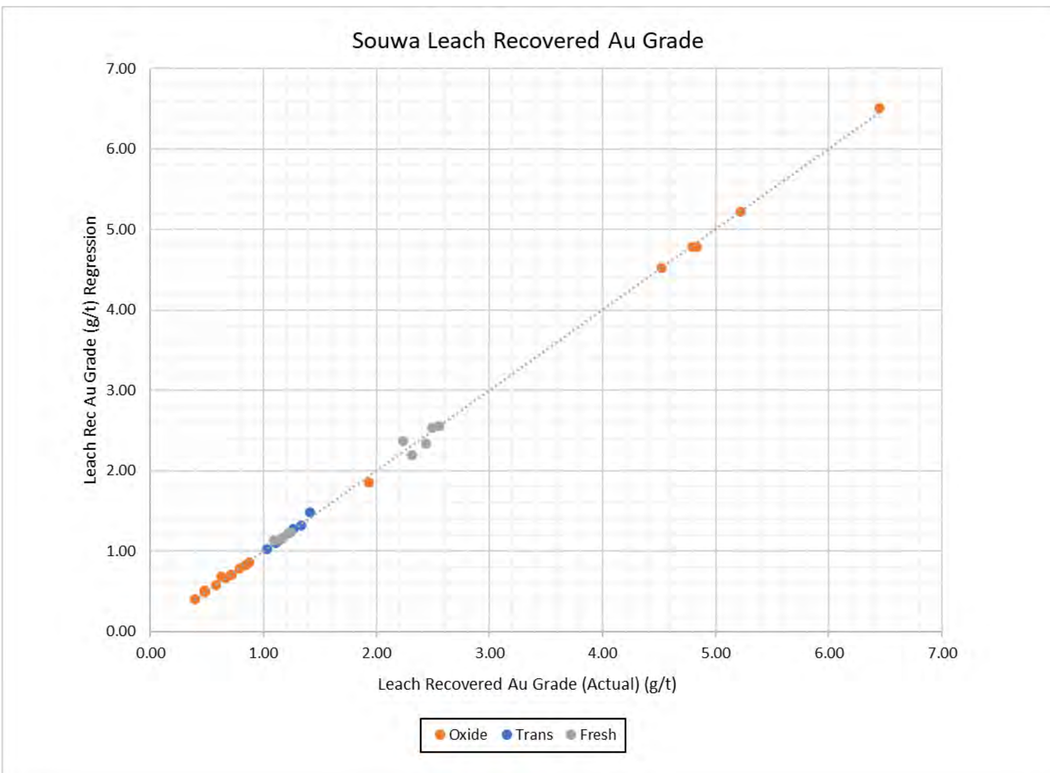


Figure 13.30 Souwa Regression Curve

13.14.2 Nokpa Recovery Regression

Transitional

The data selected for the regression analysis on Nokpa Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assays were included from the FS test work programme. The results from a total of 11 leach tests were used and are presented in Figure 13.31 below.

Fresh

The data selected for the regression analysis on Nokpa Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 11 leach tests were used and are presented in Figure 13.31 below.

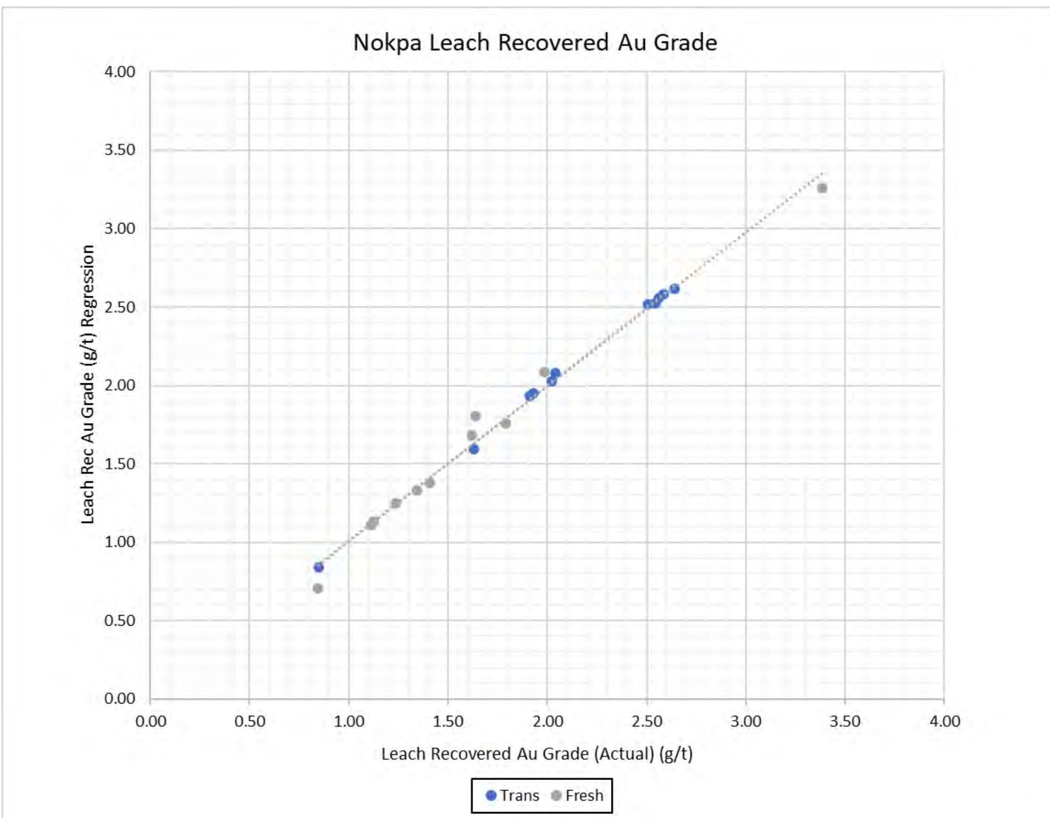


Figure 13.31 Nokpa Regression Curve

13.14.3 Kilosegui Recovery Regression

Oxide

The data selected for the regression analysis on Kilosegui Oxide ores was derived from a limited number of leach tests conducted in the FS test work programme. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Furthermore, they were from composites reporting low head grades at an average of 0.76 g/t Au. The low residue grades reported from this data set reported an elevated regression recovery of 99.1%. This was viewed as unrealistic and an average of the four leach test recoveries at 97.7% was selected for the predicted recovery at the reserve grade of 1.42 g/t Au. The results from a total of 4 leach tests were used and is presented in Figure 13.32 below.

Transitional

The data selected for the regression analysis on Kilosegui Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.32 below.

Fresh

The data selected for the regression analysis on Kilosegui Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from a total of 8 leach tests were used and are presented in Figure 13.32 below.

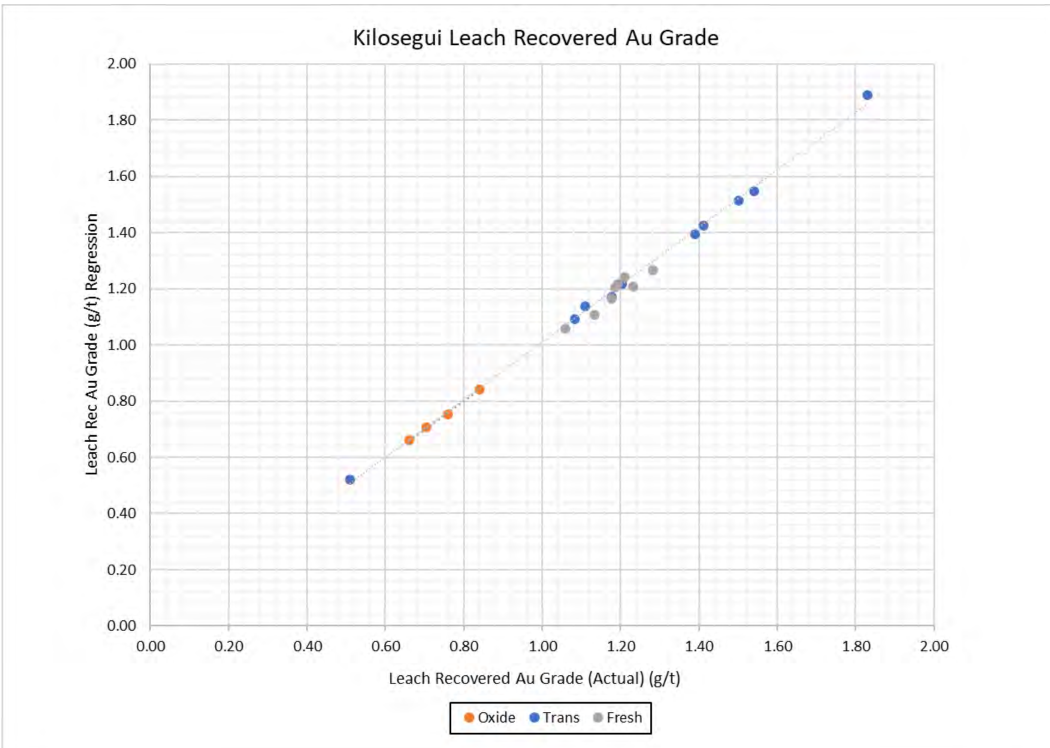


Figure 13.32 Kilosegui Regression Curve

13.14.4 Kekeda Recovery Regression

Oxide

The data selected for the regression analysis on Kekeda Oxide ores was derived from a limited number of conducted in the FS test work programme. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from a total of 4 leach tests were used and is presented in Figure 13.33 below.

Transitional

The data selected for the regression analysis on Kekada Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.33 below.

Fresh

The data selected for the regression analysis on Kekeda Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.33 below.

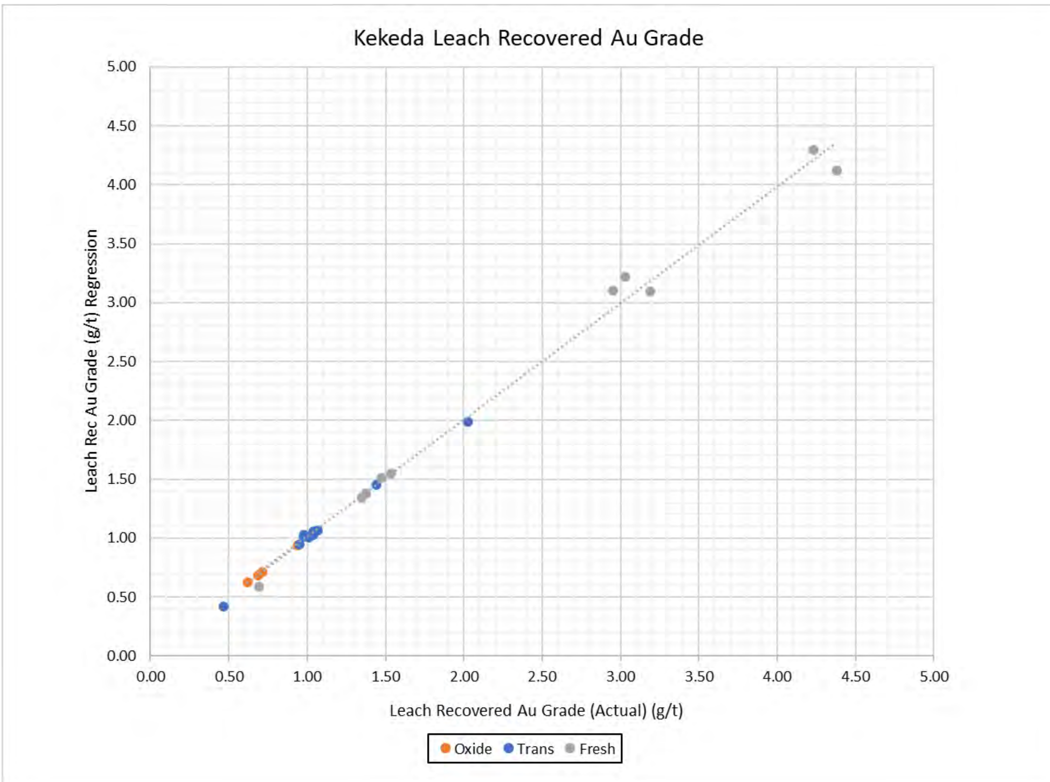


Figure 13.33 Kekeda Regression Curve

13.14.5 Enioda Recovery Regression

Oxide

The data selected for the regression analysis on Enioda Oxide ores was derived from a limited number of conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and is presented in Figure 13.34 below.

Transitional

The data selected for the regression analysis on Enioda Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.34 below.

Fresh

The data selected for the regression analysis on Enioda Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 10 leach tests were used and are presented in Figure 13.34 below.

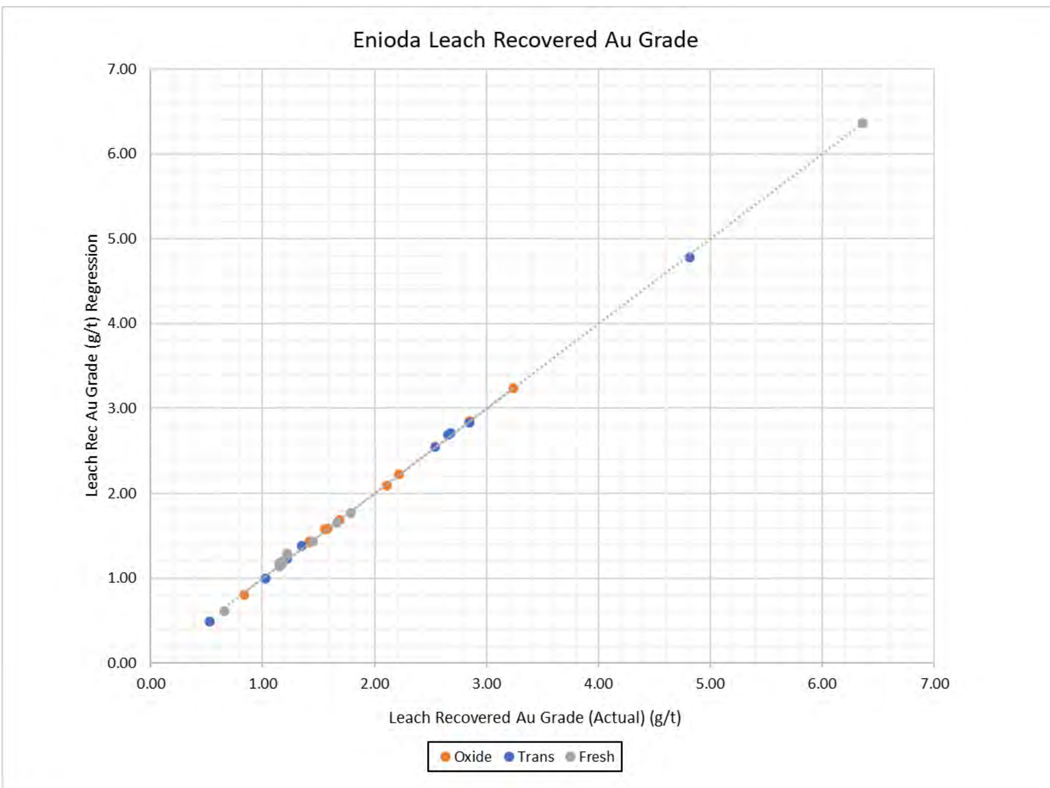


Figure 13.34 Enioda Regression Curve

13.14.6 Han Recovery Regression

Transitional

The data selected for the regression analysis on Han Transitional ores was derived from a limited number of conducted in the FS test work programme. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. A single LeachWELL soluble Au assay was included from the FS test work programme. The results from a total of 5 leach tests were used and is presented in Figure 13.35 below.

Fresh

The data selected for the regression analysis on Han Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. Two LeachWELL soluble Au assay were included from the FS test work programme. The results from a total of 9 leach tests were used and are presented in Figure 13.35 below.

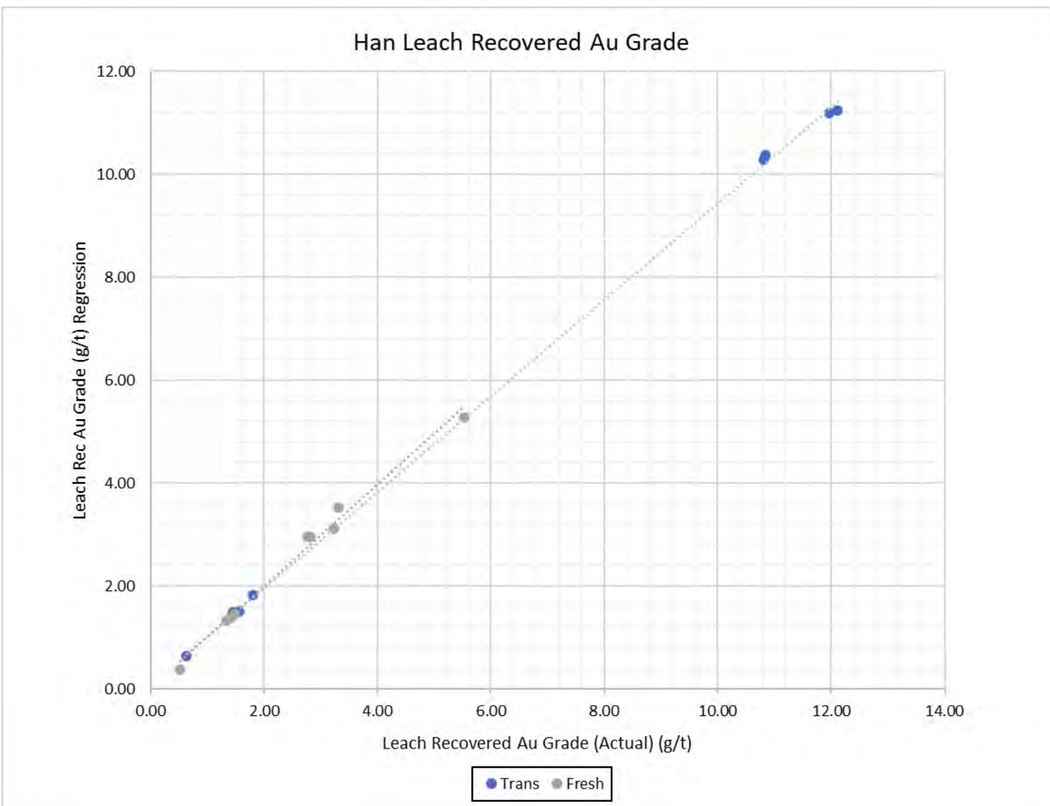


Figure 13.35 Chegue Main Regression Curve

13.14.7 Chegue Main Recovery Regression

Transitional

The data selected for the regression analysis on Chegue Main Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. A single LeachWELL soluble Au assay was included from the FS test work programme The results from a total of 13 leach tests were used and are presented in Figure 13.36 below.

Fresh

The data selected for the regression analysis on Chegue Main Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from a total of 8 leach tests were used and are presented in Figure 13.36 below.

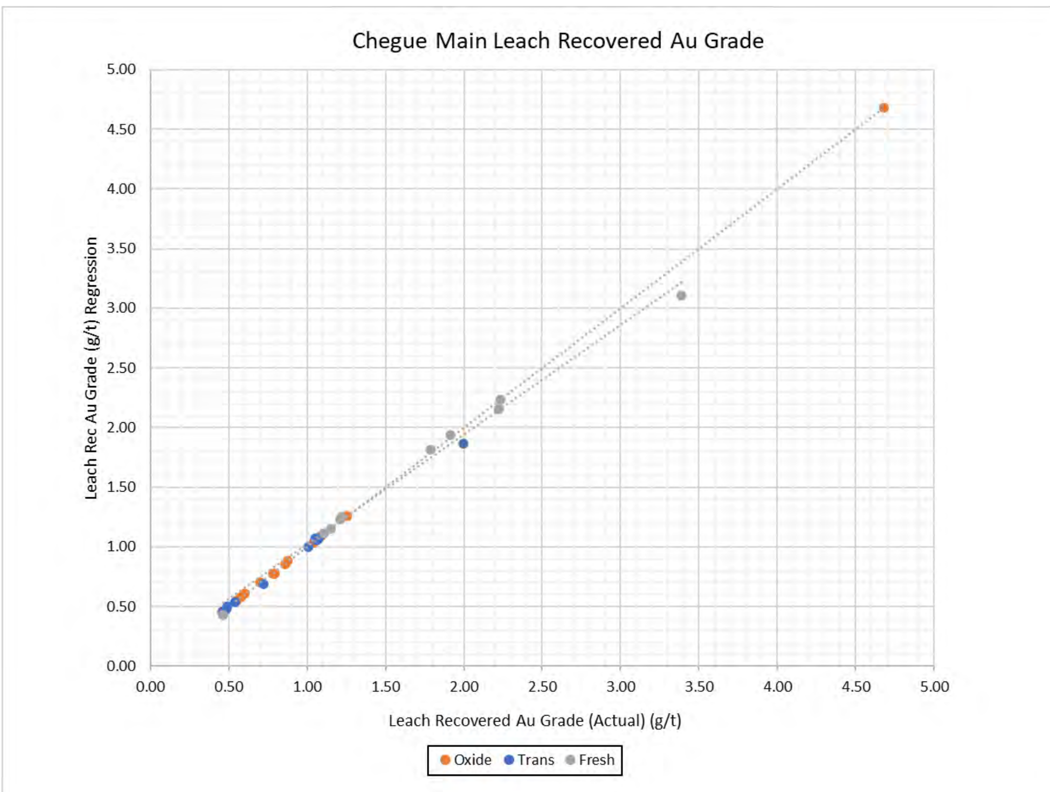


Figure 13.36 Chegue Main Regression Curve

13.14.8 Chegue South Recovery Regression

Transitional

The data selected for the regression analysis on Chegue South Transitional ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from a total of 11 leach tests were used and are presented in Figure 13.37 below.

Fresh

The data selected for the regression analysis on Chegue South Fresh ores was derived from a limited number of leach tests conducted in the PFS and FS test work programmes. The data selected consisted of tests which used a range of cyanide strengths which had not been optimised. The results from a total of 10 leach tests were used and are presented in Figure 13.37 below.

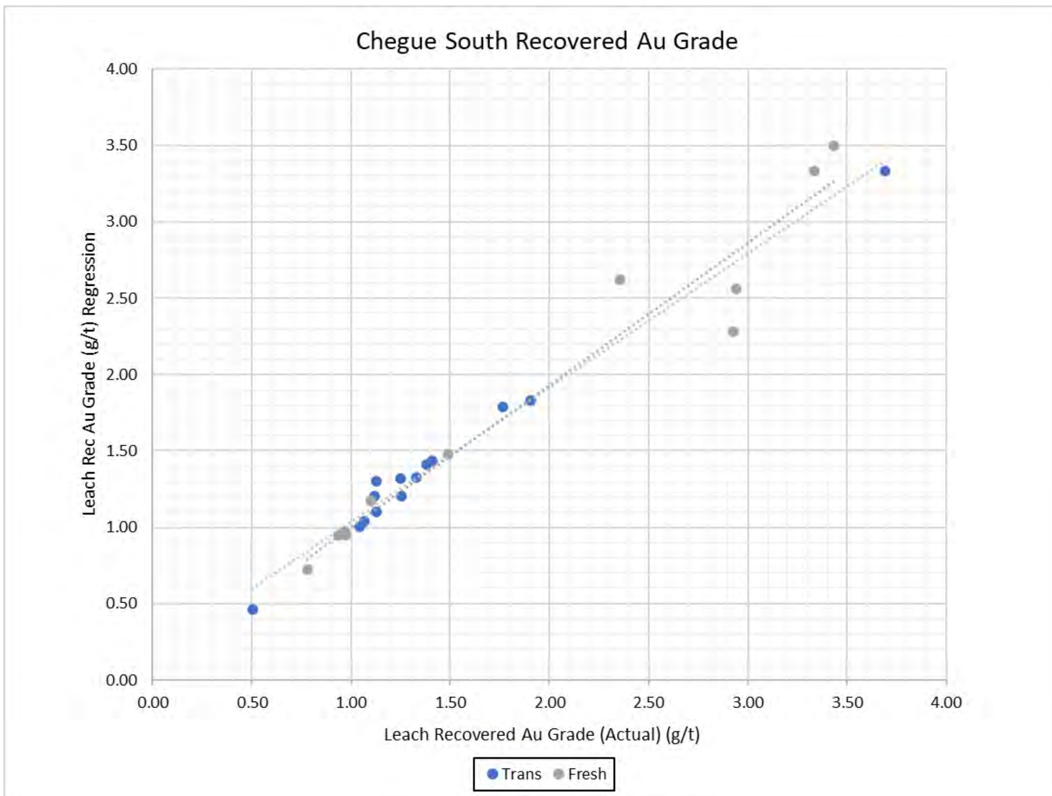


Figure 13.37 Chegue South Regression Curve

TABLE OF CONTENTS

14. MINERAL RESOURCE ESTIMATES 1
14.1 Introduction 1
14.2 Software 1
14.3 Data Used for Estimation 2
14.4 Drill Database 2
14.5 Actions on Undefined/Null and Below Detection Limit Samples 2
14.6 Wireframe Models and Data Coding 2
14.6.1 Mineralisation 2
14.6.2 Weathering, Oxidation and Regolith 16
14.7 Sample Compositing 17
14.8 Boundary Analysis on Weathering Surfaces 19
14.9 Basic Statistics 20
14.10 Gold Grade Caps and Distance Limiting 32
14.11 Diffusivity Tests 34
14.12 Variography 36
14.13 Block Model Definition 40
14.14 Ordinary Kriging Interpolation 42
14.15 Localised Uniform Conditioning 47
14.16 Density Assignments 48
14.17 Mining Depletion 49
14.18 Mineral Resource Estimate Validation 49
14.19 Mineral Resource Estimate Classification 51
14.19.1 Reasonable Prospects for Eventual Economic Extraction 51
14.19.2 Classification Criteria 52
14.20 Mineral Resource Statement 65
14.20.1 Reporting 65
14.20.2 Pit Optimisation Parameters 65
14.20.3 Results 66
14.21 Factors that May Affect the Mineral Resource 94
14.22 Comparison to Previous Mineral Resource 94
14.23 Audits and Reviews 96

LIST OF FIGURES

Figure 14.1	Log Probability Plots per Prospect for Gold Grade Assays, Showing Grade Inflexions at 0.2 g/t to 0.3 g/t, the Approximate Threshold Used to Define the Mineralised Zone Boundaries	3
Figure 14.2	ATI Interpreted Mineralised Lodes with Drill Holes	4
Figure 14.3	CHG and NOK Interpreted Mineralised lodes with Drill Holes	5
Figure 14.4	ENI Interpreted Mineralised Lodes with Drill Holes	6
Figure 14.5	HAN Interpreted Mineralised Lodes with Drill Holes	7
Figure 14.6	HND and HNDS Interpreted Mineralised Lodes with Drill Holes	8
Figure 14.7	KEK Interpreted Mineralised Lodes with Drill Holes	9
Figure 14.8	KILO Interpreted Mineralised Lodes with Drill Holes	11
Figure 14.9	NAR Interpreted Mineralised Lodes with Drill Holes	11
Figure 14.10	SAN Interpreted Mineralised Lodes with Drill Holes	12
Figure 14.11	SOL Interpreted Mineralised Lodes with Drill Holes	13
Figure 14.12	SWA Interpreted Mineralised Lodes with Drill Holes	14
Figure 14.13	THN Interpreted Mineralised Lodes with Drill Holes	15
Figure 14.14	VAKO Interpreted Mineralised Lodes with Drill Holes	16
Figure 14.15	Histogram of Raw Assay Length (m)	18
Figure 14.16	Scatter Plot of Raw Assay Length Versus Gold Grade	18
Figure 14.17	Boundary Analysis Between Oxide (200 Series) Vs Fresh (100 series) at Enioda Prospect	20
Figure 14.18	CHG Domain 5001 Log-probability Plot with Population Break at 0.2 g/t Au	21
Figure 14.19	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ATI Domains	24
Figure 14.20	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - CHG Domains	25
Figure 14.21	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ENI Domains	25
Figure 14.22	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HAN Domains	26
Figure 14.23	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HND Domains	26
Figure 14.24	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HNDS Domains	27
Figure 14.25	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KEK Domains	27
Figure 14.26	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KLG Major Domains	28
Figure 14.27	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NAR Minor Domains	28
Figure 14.28	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NOK Domains	29
Figure 14.29	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SAN Domains	29
Figure 14.30	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SOL Domains	30
Figure 14.31	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SWA Domains	30
Figure 14.32	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - THN Domains	31
Figure 14.33	Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - VAKO Domains	31
Figure 14.34	Diffusion Test for Gold Grade Based on 1 m Composites (2022 MRE)	35

Figure 14.35 Example Gaussian Variogram (left) and Back-Transformed Variogram (right) for Domain 2001 at VAKO 36
Figure 14.36 KNA Plot Using the Kriging Slope of Regression and Kriging Efficiency as Criteria for Selection of Minimum and Optimum Informing Composites - CHG domain 50041 (Ore domain 5004) 43
Figure 14.37 Example Swath Plots for Gold Grade in Domain 12001 at ATI....50
Figure 14.38 Cross-section View Looking SE at ATI. Block Grades and Drill Assay Grades 51
Figure 14.39 Plan View of the Mineral Resource Classification at ATI 54
Figure 14.40 Isometric View Looking SW of the Mineral Resource Classification at ATI....55
Figure 14.41 Plan View of the Mineral Resource Classification at CHG-NOK 55
Figure 14.42 Plan View of the Mineral Resource Classification at ENI 56
Figure 14.43 Isometric View Looking E of the Mineral Resource Classification at ENI....56
Figure 14.44 Plan View of the Mineral Resource Classification at HAN 57
Figure 14.45 Isometric View Looking SE of the Mineral Resource Classification at HAN....58
Figure 14.46 Plan View of the Mineral Resource Classification at HND 58
Figure 14.47 Plan View of the Mineral Resource Classification at KEK....59
Figure 14.48 Plan View of the Mineral Resource Classification at KILO....60
Figure 14.49 Zoom View of the Mineral Resource Classification at KILO....60
Figure 14.50 Plan View of the Mineral Resource Classification at NAR 61
Figure 14.51 Plan View of the Mineral Resource Classification at SAN 62
Figure 14.52 Plan View of the Mineral Resource Classification at SOL....62
Figure 14.53 Plan View of the Mineral Resource Classification at SWA....63
Figure 14.54 Isometric View Looking ESE of the Mineral Resource Classification at SWA....63
Figure 14.55 Plan View of the Mineral Resource Classification at THN 64
Figure 14.56 Plan View of the Mineral Resource Classification at VAKO....65
Figure 14.57 Grade and Tonnage Curves for In-pit Classified Material - ATI 86
Figure 14.58 Grade and Tonnage Curves for In-pit Classified Material - CHG Main 86
Figure 14.59 Grade and Tonnage Curves for In-pit Classified Material - CHG South....87
Figure 14.60 Grade and Tonnage Curves for In-pit Classified Material - ENI 87
Figure 14.61 Grade and Tonnage Curves for In-pit Classified Material - HAN 88
Figure 14.62 Grade and Tonnage Curves for In-pit Classified Material - HND 88
Figure 14.63 Grade and Tonnage Curves for In-pit Classified Material - HNDS....89
Figure 14.64 Grade and Tonnage Curves for In-pit Classified Material - KEK....89
Figure 14.65 Grade and Tonnage Curves for In-pit Classified Material - KILO....90
Figure 14.66 Grade and Tonnage Curves for In-pit Classified Material - NAR 90
Figure 14.67 Grade and Tonnage Curves for In-pit Classified Material - NOK 91
Figure 14.68 Grade and Tonnage Curves for In-pit Classified Material - SAN....91
Figure 14.69 Grade and Tonnage Curves for In-pit Classified Material - SOL....92
Figure 14.70 Grade and Tonnage Curves for in-pit Classified Material - SWA....92

Figure 14.71 Grade and Tonnage Curves for In-pit Classified Material - THN....93
Figure 14.72 Grade and Tonnage Curves for In-pit Classified Material - VAKO....93

LIST OF TABLES

Table 14.1 Estimation Databases and Prospects....2
Table 14.2 Weathering Zones Modelled and the Logging Codes ('WI' Field) Used....17
Table 14.3 Regolith Zones Modelled and the Logging Codes ('Lith1' Field) Used....17
Table 14.4 Oxidation Zones Modelled and the Logging Codes ('WI' Field) Used....17
Table 14.5 Basic Statistics of the Composites (naïve) all Domains in the Various Updated Prospect Areas....24
Table 14.6 Listing of Gold Grade Caps and Distance Limiting Parameters per Mineralisation/Estimation Domain....34
Table 14.7 Variogram Model Parameters per Mineralisation/Estimation Domain - Sills Normalised to 100%....39
Table 14.8 Block Model Definition per Prospect Area....41
Table 14.9 OK Panel Block Sizes Used for Gold Interpolation per Prospect....42
Table 14.10 OK Panel Block Search Parameters for Gold Grades....46
Table 14.11 Basic Statistics for Dry Density (t/m^3) per Prospect....49
Table 14.12 Dry Density (t/m^3) Assignments in the Block Models for the Updated Mineral Resource....49
Table 14.13 Comparison Between 2022 MRE (drilling at 25 m x 25 m) and 2023 MRE (Drilling at 10 m x 10 m) Within the Grade Control Drilled Areas....53
Table 14.14 Doropo Updated Measured Mineral Resource Estimate (CIM Definition Standards), 31 October 2023....67
Table 14.15 Doropo Updated Indicated Mineral Resource Estimate (CIM Definition Standards), 31 October 2023....68
Table 14.16 Doropo Updated Inferred Mineral Resource Estimate (CIM Definition Standards), 31 October 2023....69
Table 14.17 Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au....70
Table 14.18 Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au....70
Table 14.19 Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au....71
Table 14.20 Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au....71
Table 14.21 Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au....72

Table 14.22 Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 72
Table 14.23 Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 73
Table 14.24 Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 73
Table 14.25 Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 74
Table 14.26 Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 74
Table 14.27 Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 75
Table 14.28 Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 75
Table 14.29 Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 76
Table 14.30 Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 76
Table 14.31 Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 77
Table 14.32 Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 77
Table 14.33 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 78
Table 14.34 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 78
Table 14.35 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 79
Table 14.36 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 79
Table 14.37 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 80
Table 14.38 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 80
Table 14.39 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 81
Table 14.40 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 81

Table 14.41 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 82
Table 14.42 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 82
Table 14.43 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 83
Table 14.44 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 83
Table 14.45 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade cut-off of 0.3 g/t Au 84
Table 14.46 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 84
Table 14.47 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au 85
Table 14.48 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au 85
Table 14.49 Reported Measured Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE....... 95
Table 14.50 Reported Indicated Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE........ 96
Table 14.51 Reported Inferred Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE 96

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

This Doropo updated Mineral Resource Estimate (MRE) has an effective date of 31 October 2023, with drill hole data cut-off dates from 22 May 2023 to 27 August 2023. The MRE has been reported in accordance with the CIM Definition Standards (CIM Council, 2014).

The updated Doropo MRE has been prepared by Flavie Isatelle, Patrick Adams and Michael Millad, all of Cube Consulting. Mr Millad, Cube Director and Principal Geologist/Geostatistician and Ms Isatelle, Principal Geostatistician assume Qualified Person responsibility for the updated MRE and the relevant sections of this report. Both Mr Millad and Ms Isatelle meet the definition of a Qualified Person by virtue of their graduate and post-graduate degree qualifications in the fields of geology and geostatistics and their relevant experience in the mining industry in a range of capacities, including more than five years of experience in the estimation of precious metal deposits of the type represented in the Doropo Project.

The reported Mineral Resources are not Mineral Reserves since they do not have demonstrated economic viability. There is no guarantee that all or part of the reported Mineral Resources will be converted to Mineral Reserves.

The Doropo Gold Project comprises sixteen prospects, namely Attire, Enioda, Chegue Main, Chegue South, Han, Hinda, Hinda South, Kekeda, Kilosegui, Nare, Nokpa, Sanboyoro, Solo, Souwa, Tchouahinin, and Vako. All of these prospects were updated in this study following the collection of the additional DFS phase drill data. The primary objective of the infill drilling was to raise the Mineral Resource confidence classification rating of the prospects for input into the upcoming DFS.

Of the sixteen prospects listed, Attire (ATI), Enioda (ENI), Chegue Main (CHG), Chegue South (CHS), Han (HAN), Kekeda (KEK), Kilosegui (KLG), Nokpa (NOK) and Souwa (SWA) were previously updated in 2022 for the PFS. The remainder were either last updated in 2020 or estimated for the first time in this update.

Previously, a cut-off grade of 0.5 g/t Au was used to report Mineral Resources. However, the PFS demonstrated that a proportion of the oxide Mineral Resources were economically viable at grades lower than 0.5 g/t au. As a result, the Mineral Resource reporting cut-off grade for this update has been lowered to 0.3 g/t.

14.2 Software

Cube made use of Geoaccess Professional, Supervisor, Leapfrog Geo, Surpac and Isatis v2018.5 software to undertake the MRE update.

14.3 Data Used for Estimation

14.4 Drill Database

Data from the site-maintained drill database was provided to Cube as .csv format export files on a regular basis as each prospect's DFS drilling program was completed. Ultimately, several files have been provided over the course of five months and several final MS Access estimation databases were compiled. Details are given in Table 14.1.

Database Name	Prospects
Doropo_Cube_01062023.accdb	CHG, CHS, NOK, SWA
Doropo_Cube_10072023.accdb	KEK, KLG
Doropo_Cube_31072023.accdb	ENI, ATI, HAN
Doropo_Cube_27082023.accdb	HND, HNDS, NAR, SAN, SOL, THN, VAKO

Table 14.1 Estimation Databases and Prospects

The table types used for estimation included collar, survey, assay, lithology (includes weathering/oxidation codes) and density.

14.5 Actions on Undefined/Null and Below Detection Limit Samples

Undefined or null sample intervals predominantly represent hanging wall intervals that were not sampled due to being above the mineralised lodes; these assay intervals were ignored for modelling purposes. Below detection limit value sample intervals were set at half of the detection limit for the specific assay method.

14.6 Wireframe Models and Data Coding

14.6.1 Mineralisation

Log probability plots of the raw data indicate a 0.2 g/t to 0.3 g/t natural low threshold (Figure 14.1) is suitable for defining the zones of mineralisation. This threshold is often also coincident with zones of silica-sericite alteration and quartz veining, along with the presence of sulphides.

The mineralised domain wireframes generated during the 2020 and 2022 MRE updates were used as a guide, with original interpretations from 2020 also taking into account hand-drawn sections provided by the field geologists. The infill drilling demonstrated that, broadly speaking, the 2020 and 2022 mineralisation interpretations were robust. Aside from local variations in width at new drill hole locations, some lode extensions in certain areas, and a few additional secondary lodes, the geological and mineralisation continuity has remained largely unchanged.

The intercept codes for each lode domain were defined in Leapfrog Geo, where they were used to generate wireframe volumes using the vein modelling method. Each solid was then coded in the database and exported to Surpac to generate the equal length composites for grade interpolation. The wireframes were clipped at 50 m beyond the last drill hole both down dip and along strike.

The hanging wall and footwall of each shear lode could be confidently defined in most cases, thus making interpretation relatively uncomplicated and supporting the view that the modelled volume and continuity of mineralisation at the lode scale is of high confidence.

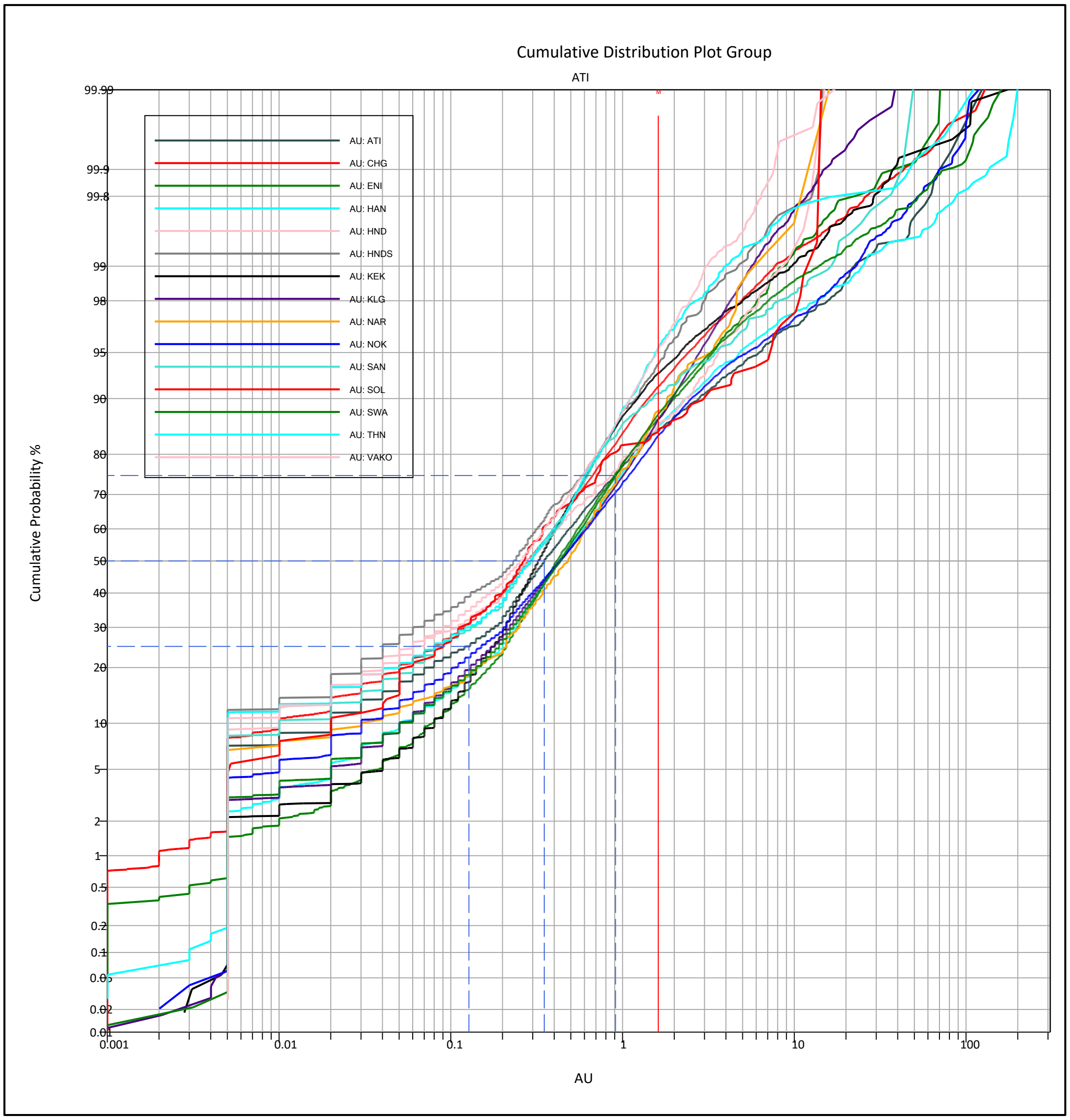
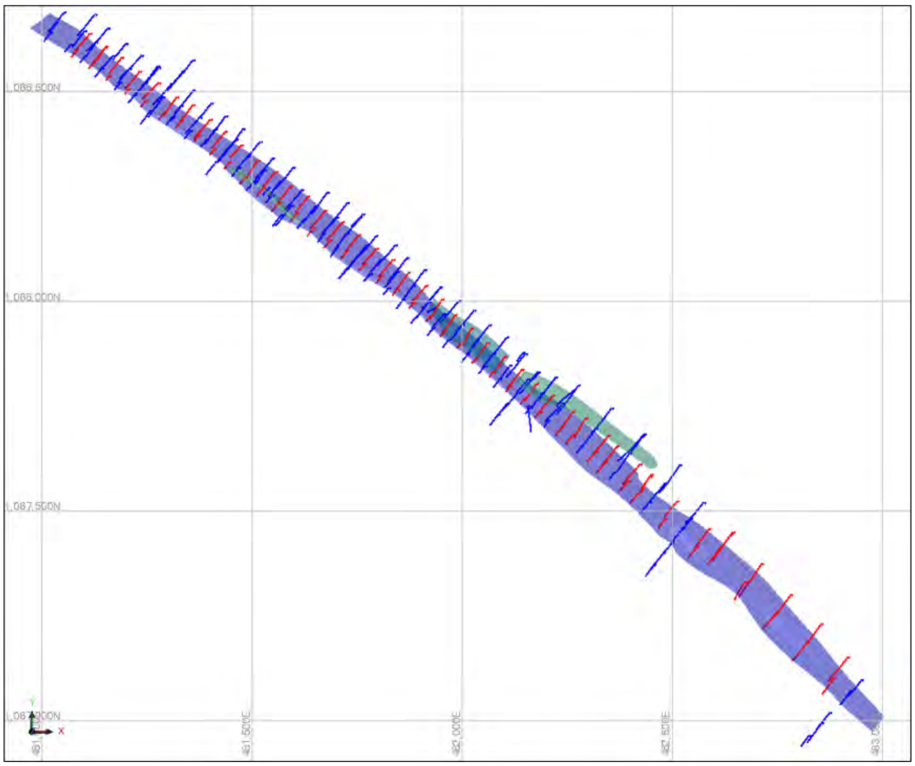


Figure 14.1 Log Probability Plots per Prospect for Gold Grade Assays, Showing Grade Inflexions at 0.2 g/t to 0.3 g/t, the Approximate Threshold Used to Define the Mineralised Zone Boundaries

Attire (ATI)

The infill drilling at Attire targeted the south-eastern extension of the lode and extended the main shear (Domain 12001) another 550 m in this direction (Figure 14.2), as well as infilling pre-existing holes on a 25 m spacing. Drill hole spacing varies from 25 m to 100 m. The lodes strike NW-SE and dip steeply to the north-east at approximately 70°, with thickness typically ranging between 5 m and 8 m. Mineralisation is open at depth and in the south.



(Blue=Existing, Red=New DFS Holes)

Figure 14.2 ATI Interpreted Mineralised Lodes with Drill Holes

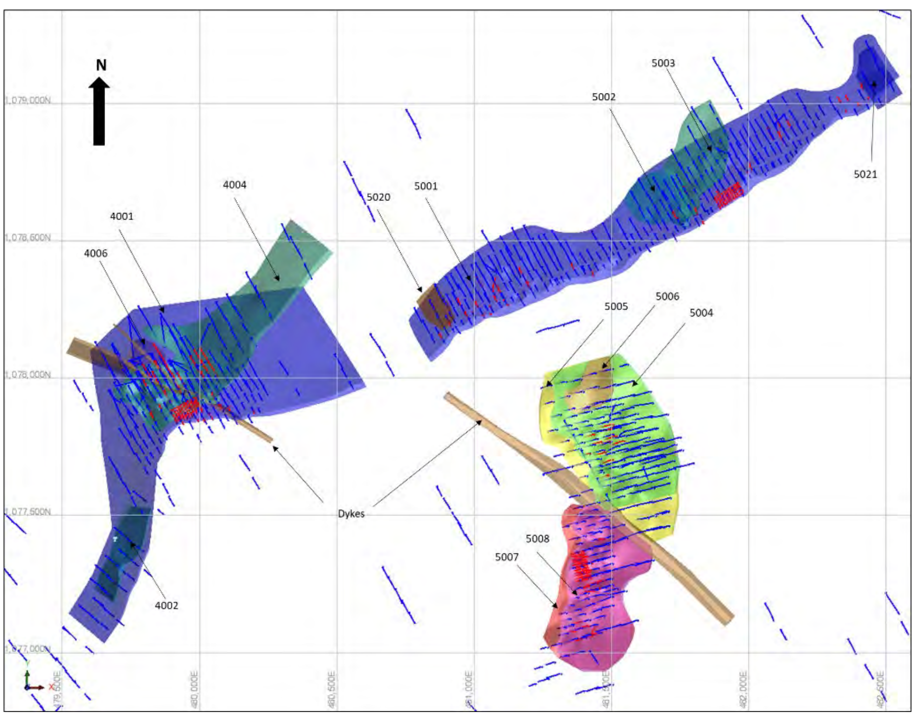
Chegue (CHG) and Nokpa (NOK)

Chegue Main, Chegue South and Nokpa are in close proximity to one another (Figure 14.3). Three closed spaced drilling programs at 10 m by 10 m were completed for the DFS; one for each of the prospects Nokpa, Chegue Main and South. The close spaced drilling confirms the continuity and general thickness of the previously interpreted mineralised zones.

The main mineralised lode at NOK is Domain 4001 and consists of an anticline with fold axis plunging to the north-northwest Figure 14.3. The mineralisation is concentrated on the fold axis and is open at depth. The limbs of the fold are not as well mineralised. Domain 4004 is a splay off the northern limb of Domain 4001. Domain 4002 consists of a hanging wall lode off the southern limb of Domain 4001. A new domain 4006 was digitised on the footwall of 4001 in the fold hinge. A NW-SE barren dyke cross-cuts the NOK anticline. The lodes at NOK dip between 30° and 35° to the northwest and north. The thickness of the main lode around the fold axis varies between 20 m and 40 m. The northern and southern limbs are thinner with an average thickness around 4 m to 5 m.

At CHG Main, information from the DFS infill drilling confirmed the 2022 PFS interpretation of the area. Two new domains 5020 and 5021 have been digitised on the hanging wall at the southern end (5020) and on the footwall at the northern end (4021) of 4001.

The mineralised lodes at CHG South are cross-cut by a NW-SE trending barren dyke. Three lodes are interpreted north of the dyke (Domains 5004 to 5006). South of the dyke, two mineralised lodes are interpreted (Domains 5007 and 5008). North of the dyke the lodes have an average thickness varying between 15 m and 20 m and dip gently to the east at 25° to 30°. South of the dyke the thickness for Domain 5008 varies between 10 m and 16 m and for Domain 5007 between 5 m and 10 m. Most of the gold metal for CHG South is located within Domain 5005.

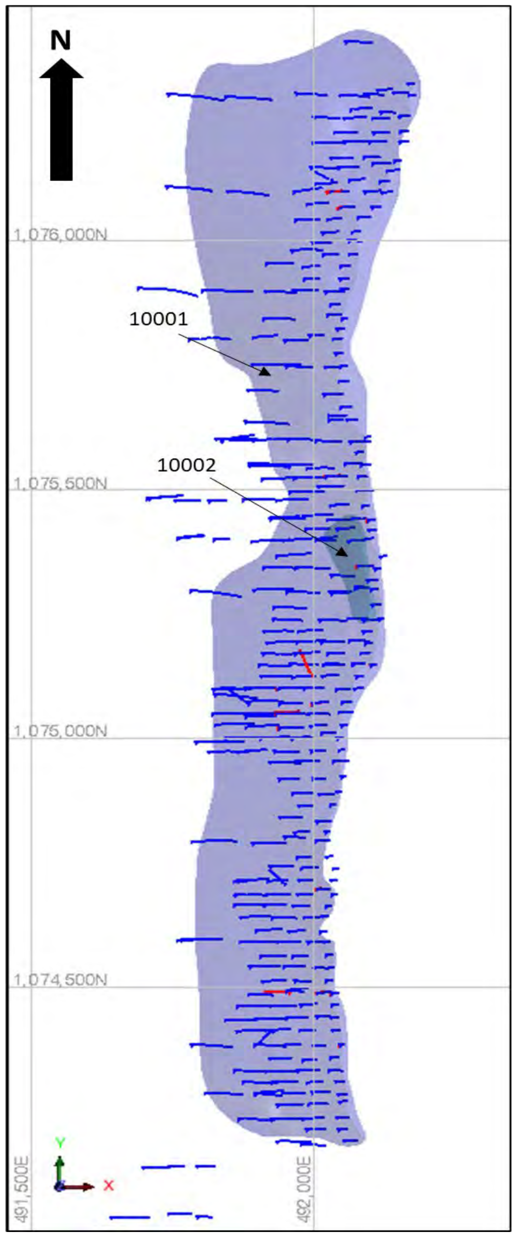


(Blue=Existing, Red=New DFS Holes)

Figure 14.3 CHG and NOK Interpreted Mineralised lodes with Drill Holes

Enioda (ENI)

The main lode interpreted at ENI is Domain 10001, which strikes N-S, dipping gently at ~30° to the west. It covers a strike distance of 2.2 km (Figure 14.4) with thickness varying between 10 m and 20 m. There was limited drilling during the DFS and the interpretation is very similar to the previous MRE in 2022. Mineralisation remains open at depth.

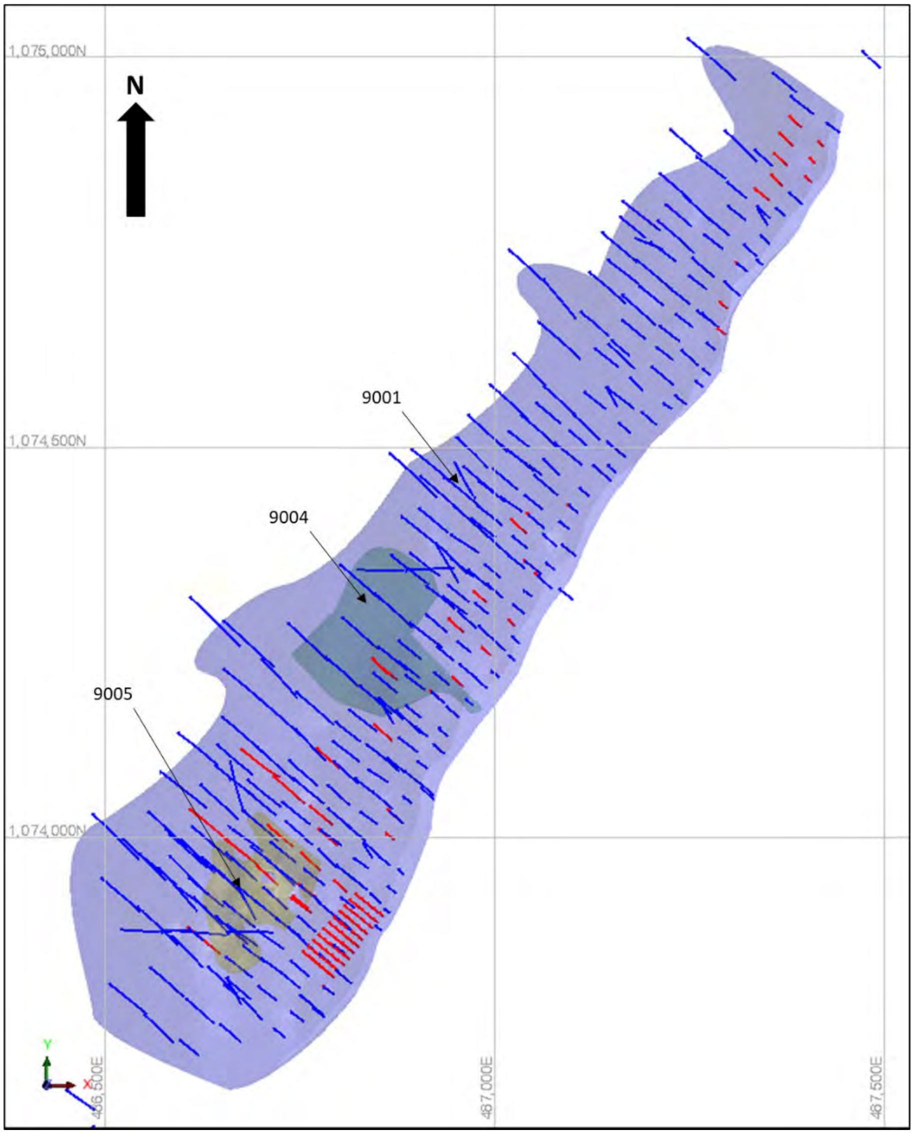


(Blue=Existing, Red=New DFS Holes)

Figure 14.4 ENI Interpreted Mineralised Lodes with Drill Holes

Han (HAN)

Domain 9001 is the main mineralised lode interpreted at HAN, containing most of the gold metal, with a thickness varying between 10 m to 15 m. The DFS infill drilling at HAN reduced the spacing to 25 m, with an area of closed spaced drilling at 10 m by 10 m (Figure 14.5). The infill drilling confirms the thickness and continuity of the mineralisation.

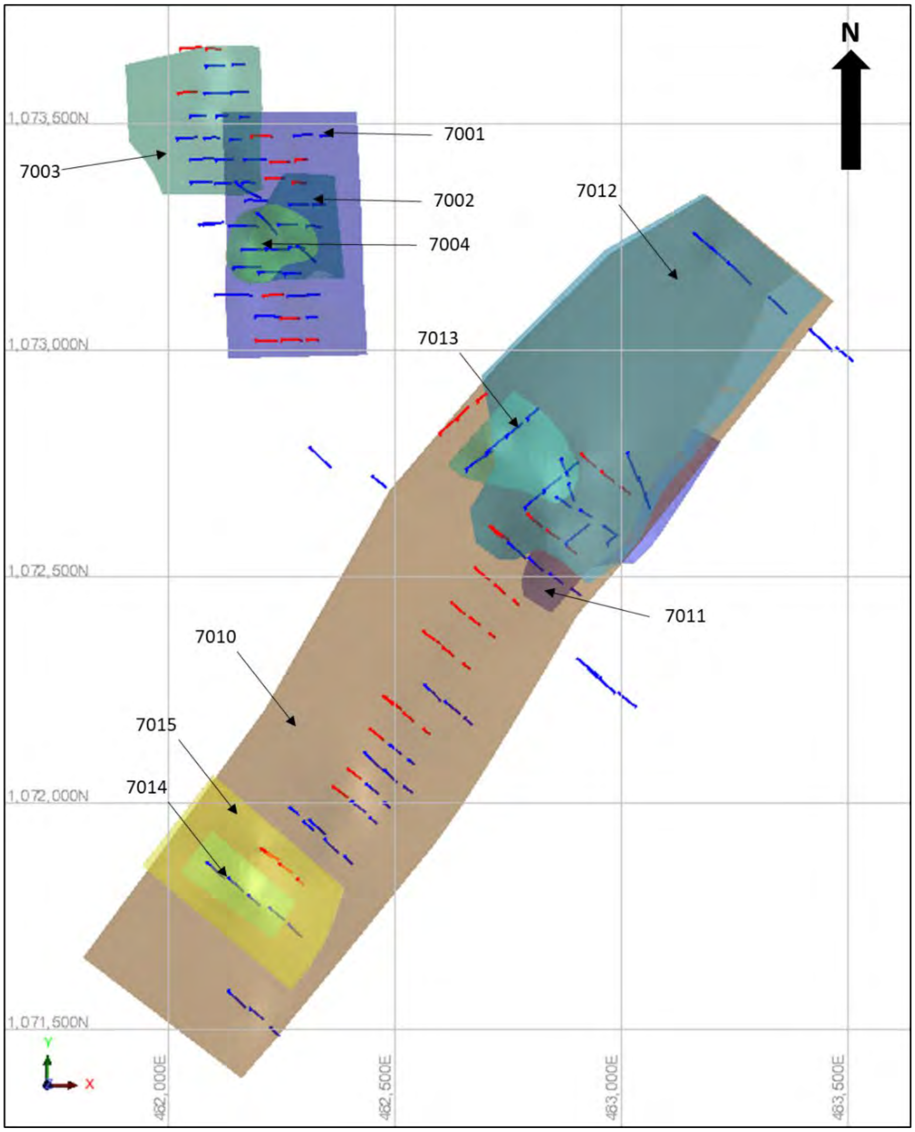


(Blue=Existing, Red=New DFS Holes)

Figure 14.5 HAN Interpreted Mineralised Lodes with Drill Holes

Hinda (HND) and Hinda South (HNDS)

Mineralisation at Hinda is split into two areas: Hinda (HND) comprising four lodes (7001 to 7004) striking N and shallow dipping to the west and Hinda South (HNDS) comprising six lodes (7010 to 7015) striking NE-SW and shallow dipping to the NW (Figure 14.6). The thickness of the lodes varies from 3 m to 23 m with an average of 10 m to15 m. Drill spacing varies from 50 m in HND to 100 m in HNDS. Infill drilling from the PFS-DFS period confirmed the thickness and consistency of the lodes in HND, with the interpretation of one additional lode (7004). HNDS was never interpreted nor estimated previously.

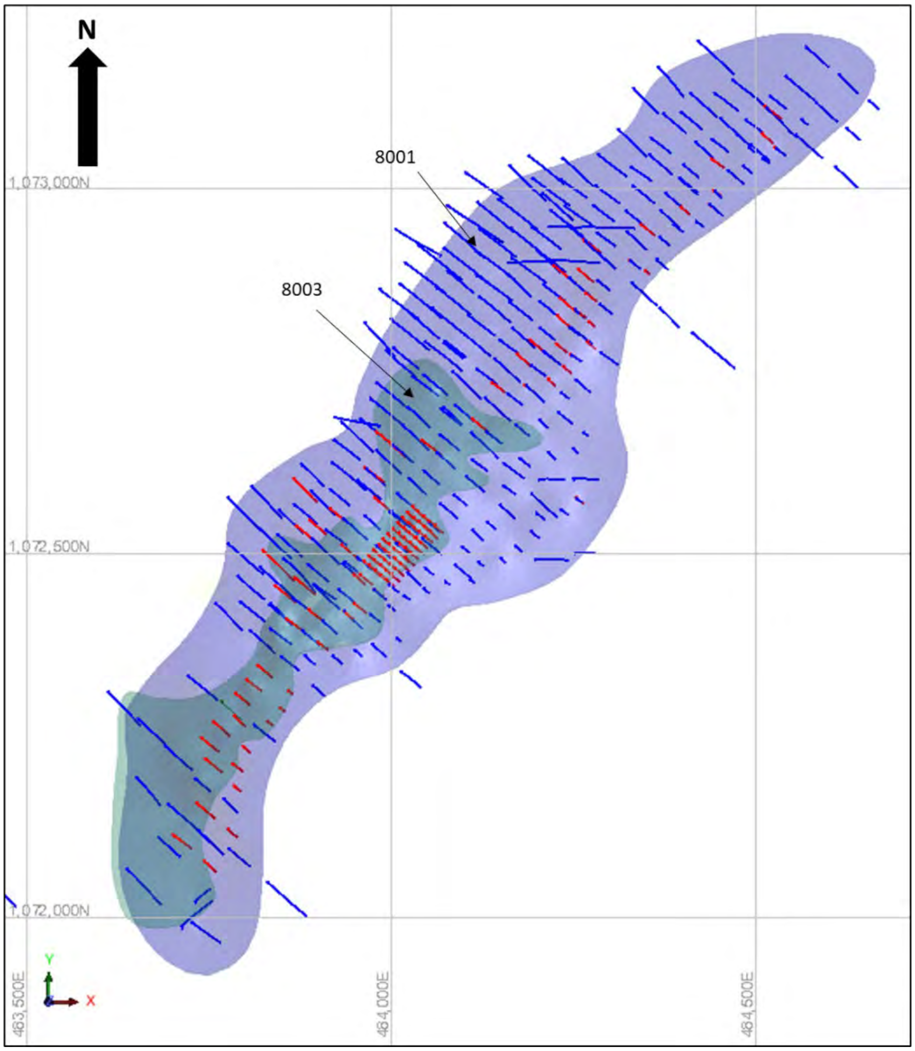


(Blue=Existing, Red=New DFS Holes)

Figure 14.6 HND and HNDS Interpreted Mineralised Lodes with Drill Holes

Kekeda (KEK)

The KEK mineralised lodes strike NE-SW with the main lode Domain 8001 extending over 1.5 km. Domain 8003 is a small splay off the hanging wall of Domain 8001. The main lode has a thickness varying between 10 m to 20 m. The DFS infill drilling at KEK reduced the spacing to 25 m and an area of closed spaced drilling at 10 m by 10 m was completed as well (Figure 14.7). Infill drilling at KEK has confirmed the thickness and consistency of the mineralisation relative to the 2022 update and it remains open down dip.

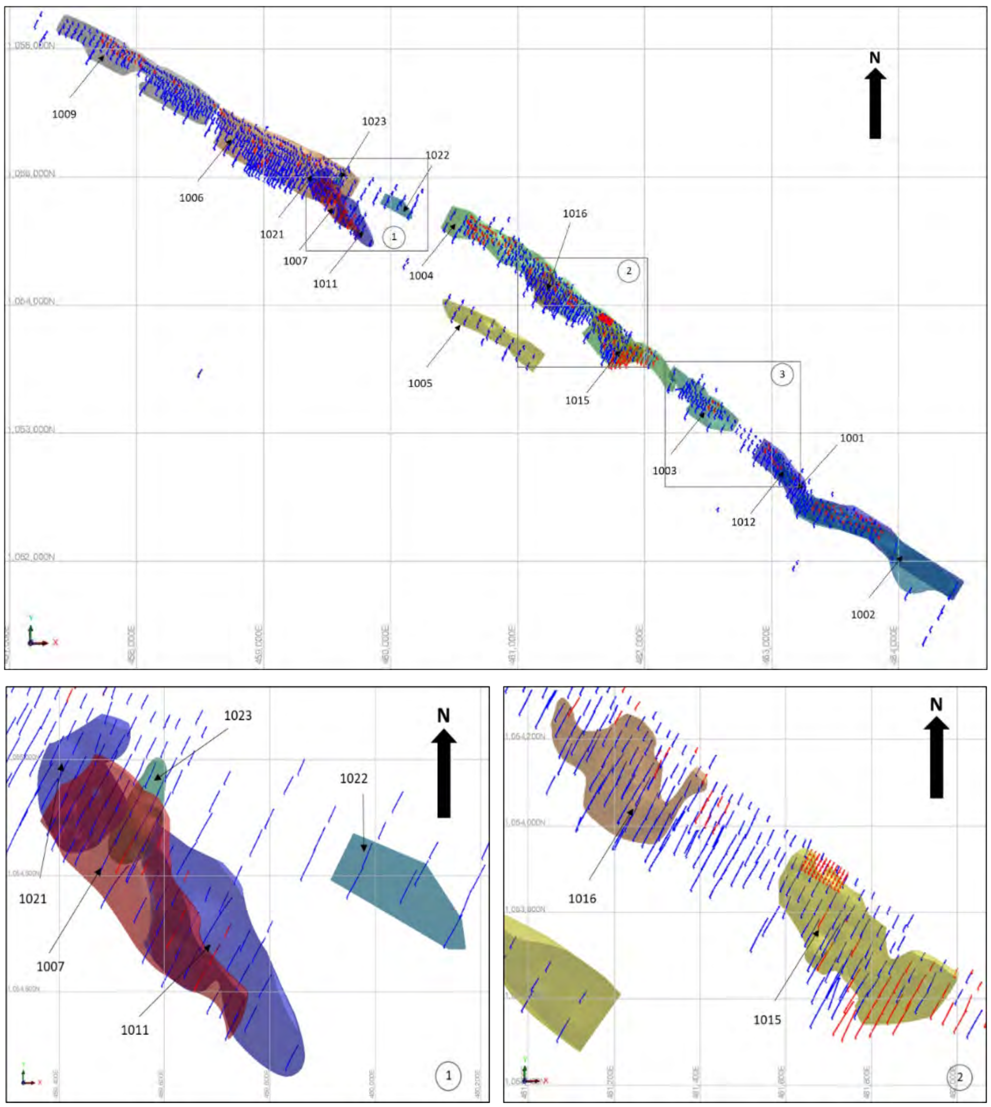


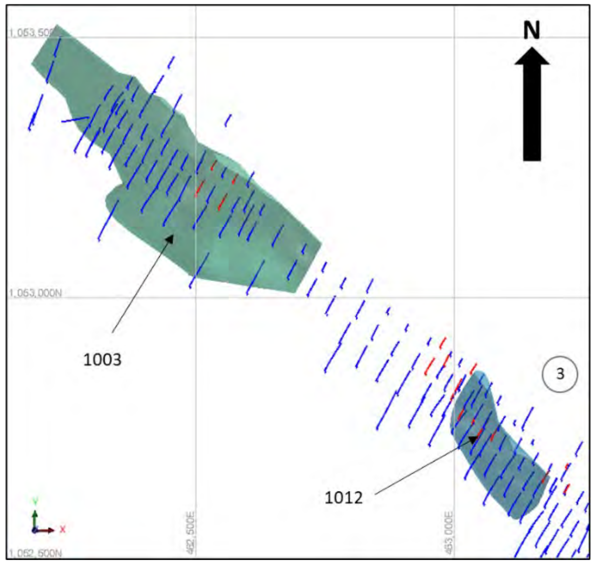
(Blue=Existing, Red=New DFS Holes)

Figure 14.7 KEK Interpreted Mineralised Lodes with Drill Holes

Kilosegui (KILO)

The lodes at KILO extend over an 8 km strike length and dip gently to the south-west at ~30°. The DFS program focused on infilling most of the area at 25 m with closely spaced drilling of 10 m by 10 m in a test area (Figure 14.8). Information from the infill drilling confirmed the extent and thickness of the mineralised domains as determined in the previous MRE in 2022. Fifteen mineralised lodes were updated, which include Domains 1001-1007, 1009, 1011-1012, 1015-1016 and 1021-1023. The wireframe interpretations were limited to 50 m beyond the last drill hole.



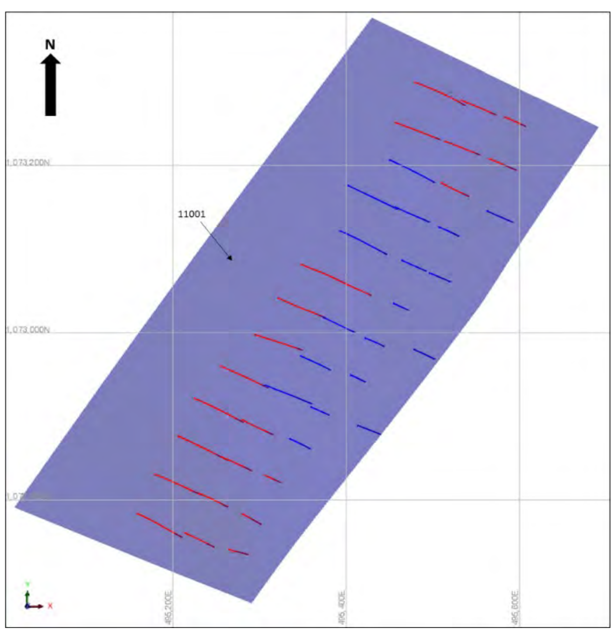


(Blue=Existing, Red=New DFS Holes)

Figure 14.8 KILO Interpreted Mineralised Lodes with Drill Holes

Nare (NAR)

Domain 11001 interpreted at Nare extends over 450 m and strikes NE-SW, dipping 40° to the west. Drill holes that were used to define the mineralised lode are located on a 50 m x 50 m spacing. New drilling from the DFS program extended the mineralisation in the south and in the north (Figure 14.10). Mineralisation is open at depth.

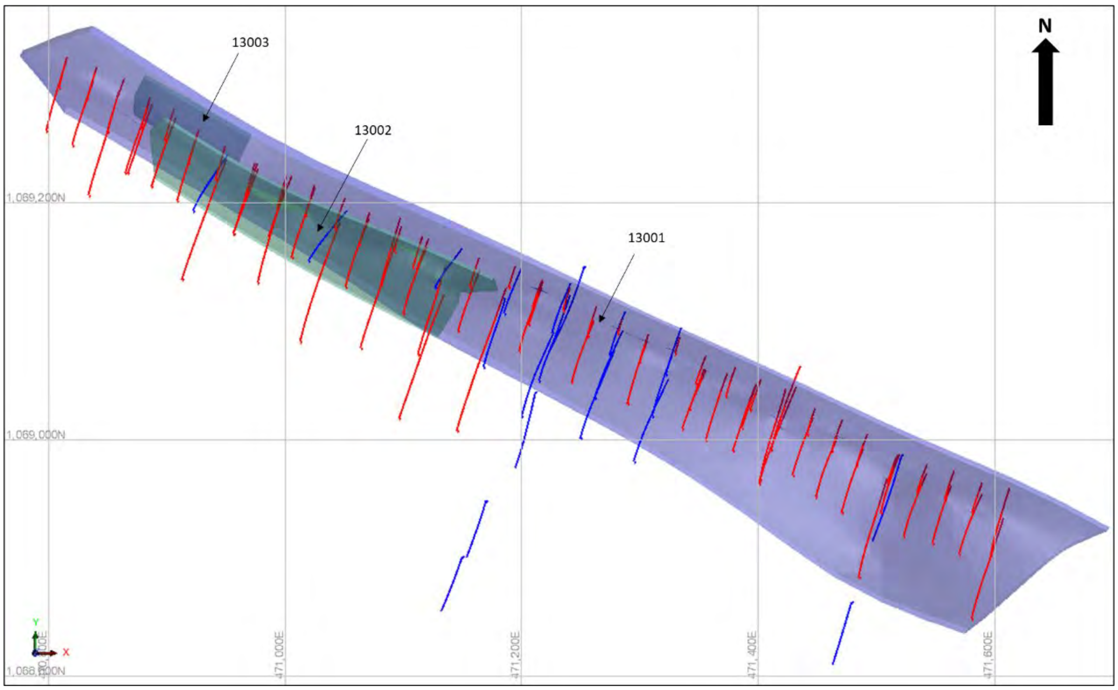


(Blue=Existing, Red=New DFS Holes)

Figure 14.9 NAR Interpreted Mineralised Lodes with Drill Holes

Sanboyoro (SAN)

Three domains have been interpreted at Sanboyoro (13001 to 13003). The main domain 13001 extends over 950 m in length. The mineralisation strikes WNW and dips steeply to the SW. Mineralisation remains open at depth and along strike. PFS-DFS program focused on homogenising the drill spacing to 25 m and extending to the north and the south (Figure 14.10).

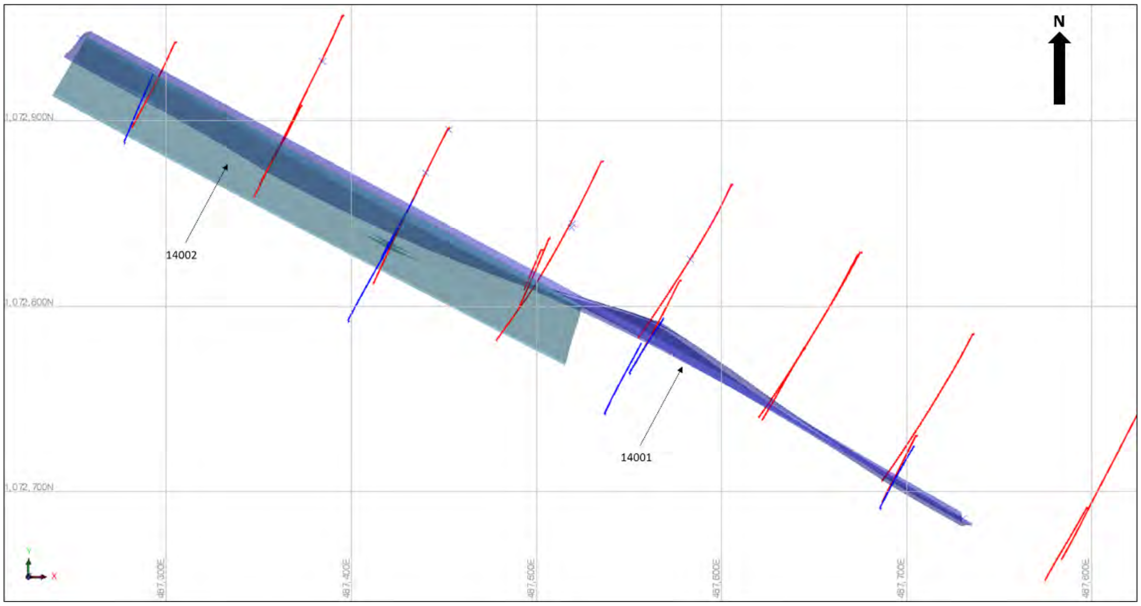


(Blue=Existing, Red=New DFS Holes)

Figure 14.10 SAN Interpreted Mineralised Lodes with Drill Holes

Solo (SOL)

The mineralisation at Solo strikes NW and dips steeply to the SW. The main domain 14001 extends over 550 m in length and remains open to the north and at depth. Two domains (14001 and 14002) have been interpreted. The drill spacing is approximately 75 m to 85 m (Figure 14.11).

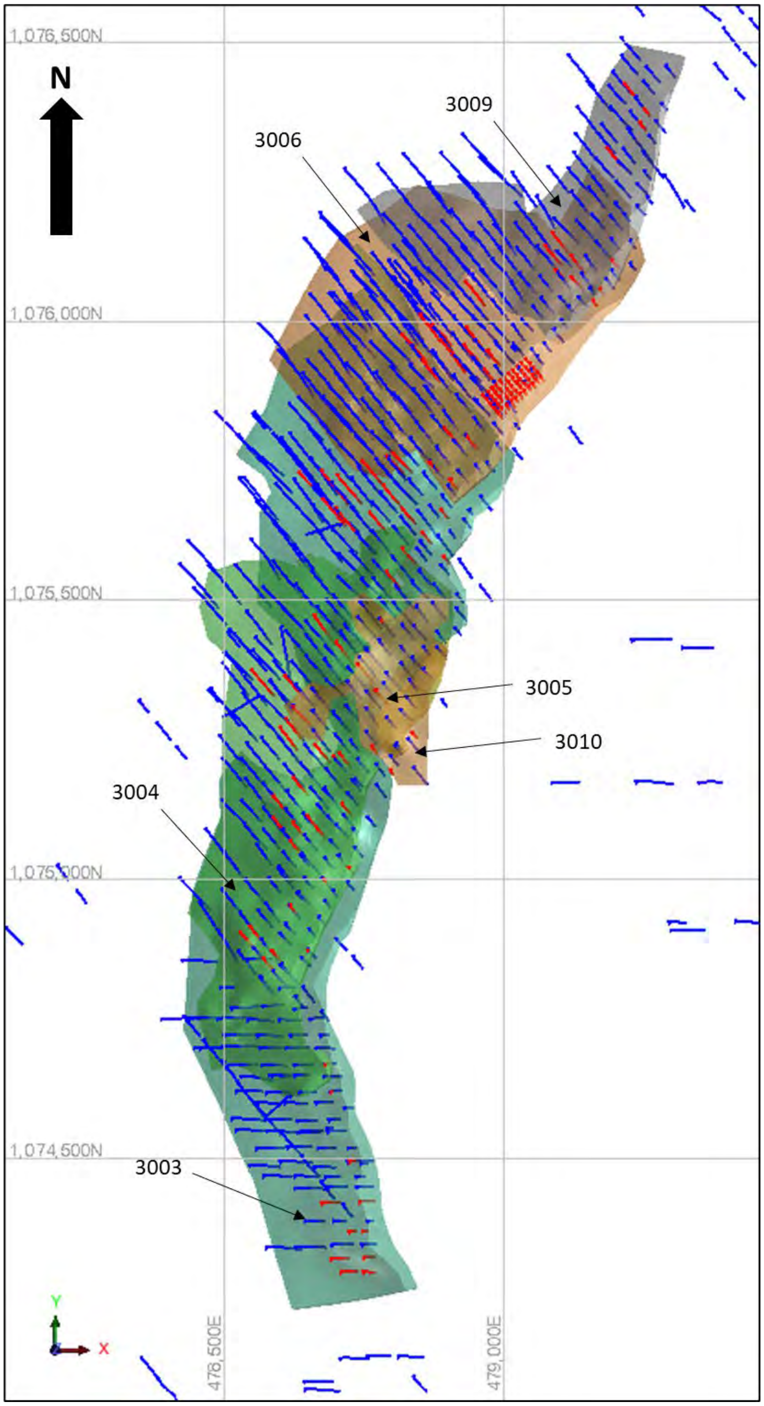


(Blue=Existing, Red=New PFS-DFS Holes)

Figure 14.11 SOL Interpreted Mineralised Lodes with Drill Holes

Souwa (SWA)

The drilling and mineralised lodes at SWA cover a strike extent of 2.4 km. The lodes strike NNE-SSW and dip at ~35° to the west. The average drill hole spacing has been reduced to a consistent 20 m, with a closed spaced drilling area at 10 m by 10 m as a result of the DFS infill drilling program. Six mineralised lodes were updated and consist of Domains 3003-3006, 3009 and 3010. The infill drilling has again confirmed the thickness and continuity of the mineralisation with respect to the previous MRE update in 2022. The wireframe interpretations were limited to 50 m beyond the last drill hole (Figure 14.12).



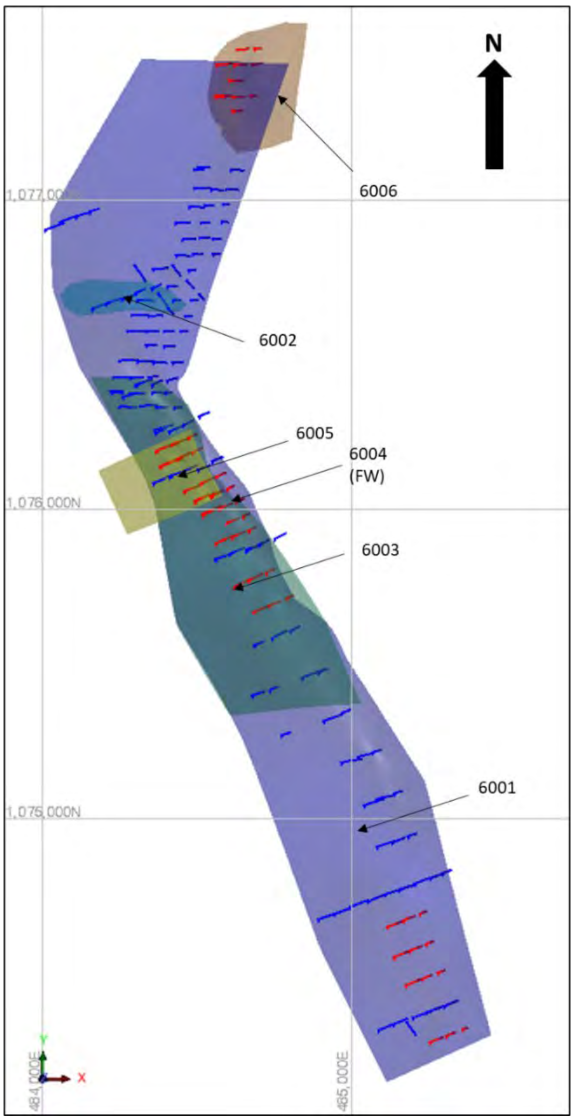
(Blue=Existing, Red=New DFS Holes)

Figure 14.12 SWA Interpreted Mineralised Lodes with Drill Holes

Tchouahinin (THN)

Tchouahinin extends over a strike length of 3.2 km. The main mineralised lode interpreted is Domain 6001, which strikes NW-SE in the south and swings to NE-SW to the north, via the intersection of two structures (Figure 14.13). Domain 6001 dips shallowly at 25° to the SW and W, with an average thickness between 5 and 15 m thick. Domain 6003 is a small splay off the hanging wall of the southern limb of the domain. Domains 6002, 6004, 6005 and 6006 are small splays from main Domains 6001 and 6003.

The drill spacing varies from 50 m in the north and centre, with the south being more sparsely drilled at 100 m to 150 m. Mineralisation remains open at depth and long strike.

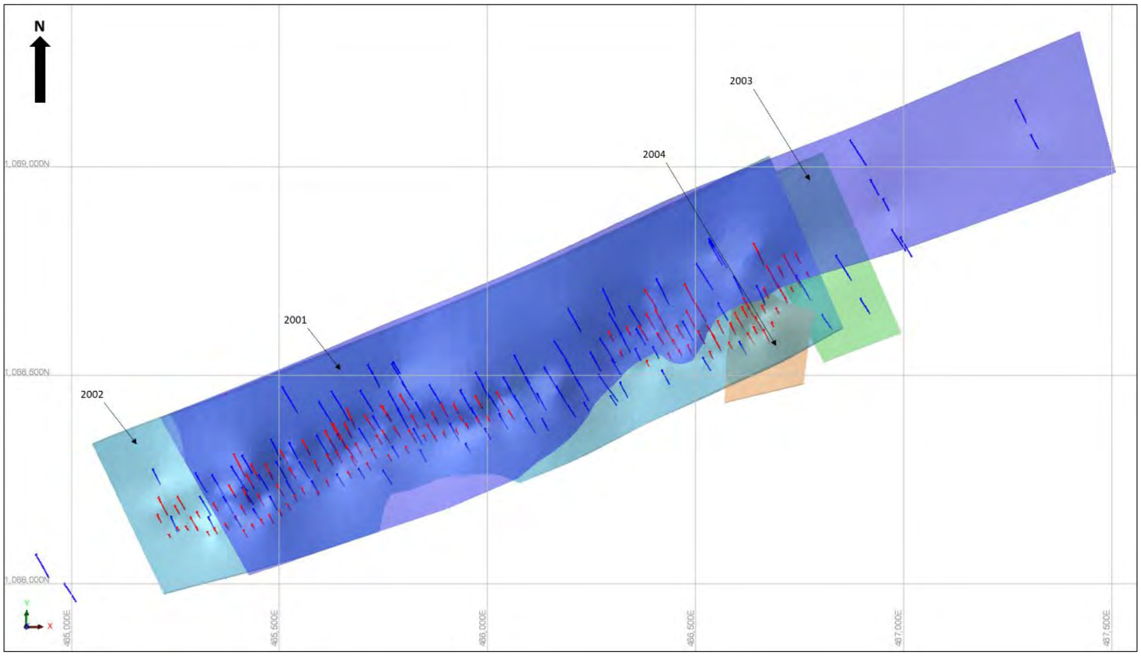


(Blue=Existing, Red=New PFS-DFS Holes)

Figure 14.13 THN Interpreted Mineralised Lodes with Drill Holes

Vako (VAKO)

Two main mineralised lodes (Domains 2001 and 2002) were updated at Vako and two additional lodes (domains 2003 and 2004) were interpreted. The main lodes strike ENE - WSW and dips 25° to 30° to the NW. Domain 2002 extends over a strike length of 1.7 km and Domain 2001 over a strike length of 2.4 km. Drill hole spacing at Vako averages around 25 m in the main area; the northern extent is drilled at a spacing of 150 m to 400 m. The average thickness of the mineralisation varies from 10 m to 25 m for Domain 2002 and 10 m to 15 m for Domain 2001. Mineralisation remains open at depth and to the north east (Figure 14.14).



(Blue=Existing, Red=New DFS Holes)

Figure 14.14 VAKO Interpreted Mineralised Lodes with Drill Holes

14.6.2 Weathering, Oxidation and Regolith

The weathering, oxidation and regolith characteristics were modelled separately. Weathering intensity was logged as a separate item to primary lithology. Surface DTM models were created in Leapfrog Geo using the codes displayed in Table 14.2, Table 14.3 and Table 14.4. The weathering and oxidation zones were based on the weathering intensity logging ('WI' database field) while the regolith was modelled using the 'Lith1' logging code in the supplied database.

WI Logging Code	Weathering Zone
CW	Completely Weathered
HW	Highly Weathered
MW	Moderately Weathered
SW	Slightly Weathered
FR	Fresh Rock

Table 14.2 Weathering Zones Modelled and the Logging Codes ('WI' Field) Used

Lith1 Logging Code/s	Regolith Zone
LAT, TCLY, TLAT, TPLT, TSND and TSOL	Transported Cover
SAP	Saprolite
SAPR	Saprock
All other Lithology Codes	Fresh Rock

Table 14.3 Regolith Zones Modelled and the Logging Codes ('Lith1' Field) Used

WI Logging Code/s	Oxidation Zone
CW, HW	Oxide
MW, SW	Transitional
FR	Fresh Rock

Table 14.4 Oxidation Zones Modelled and the Logging Codes ('WI' Field) Used

14.7 Sample Compositing

Cube carried out an assessment of the raw assay interval length and raw gold assays in order to determine the most appropriate length for compositing of the samples. The estimate was carried out using RC, DD and RCD samples only. A sample length of 1 m is the most common as shown in Figure 14.15, with almost 100% of the raw samples being exactly 1 m in length. There is no apparent relationship between sample length and assay grade (Figure 14.16).

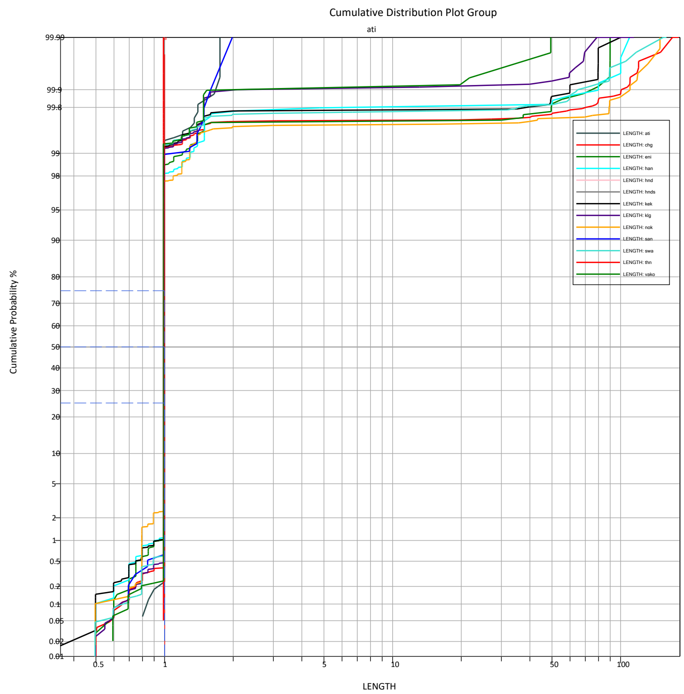


Figure 14.15 Histogram of Raw Assay Length (m)

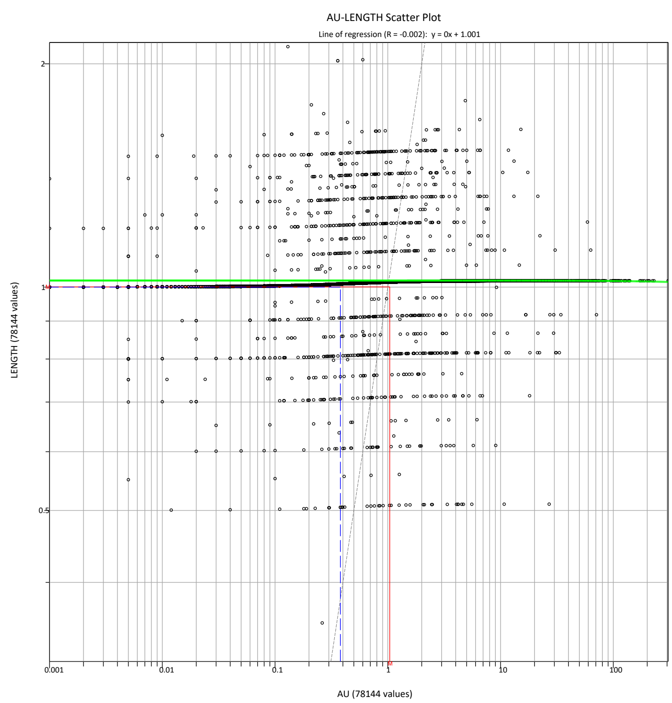


Figure 14.16 Scatter Plot of Raw Assay Length Versus Gold Grade

A compositing target length of 1 m was selected for the following reasons:

- Since almost 100% of the raw assay and density intervals are 1 m or less in length, the use of a 1 m target length would result in minimal sample splitting;
- The composite length was considered suitable for the range of estimation block sizes under consideration;
- The 1 m target length provides an opportunity for maximum resolution to be obtained on local estimates.

Samples were composited downhole per mineralised domain to 1 m using the best fit methodology in Surpac with a minimum coverage length of 0.25 m flagged.

14.8 Boundary Analysis on Weathering Surfaces

A boundary analysis was undertaken on the weathering zone transitions as part of the 2022 MRE (Centamin, 2021) and it was not considered necessary to repeat the analysis since the prospect areas had sufficient samples in each weathering domain at the time for a meaningful analysis.

An example of the boundary analysis chart is shown in Figure 14.17. The analysis shows that there is not a significant difference in the mean grade between the oxide and fresh domain and the gold grade changes gradationally across the boundary. Therefore, there is no need to separate the two weathering domains and the mineralised lode domains will serve as the only hard boundary in the estimation process.

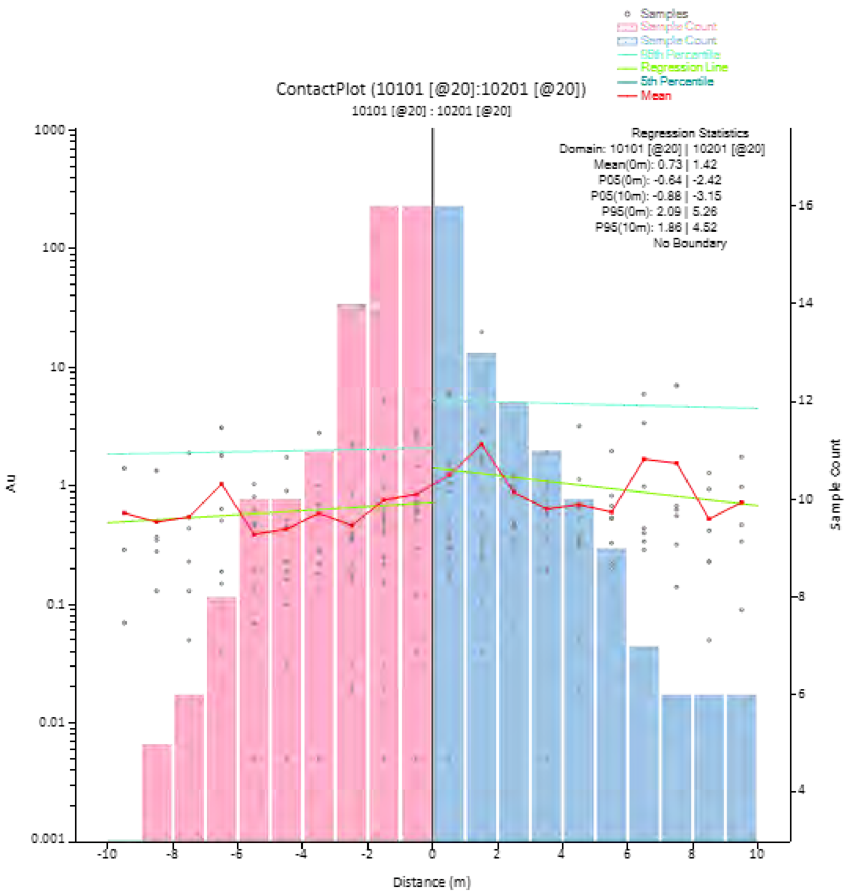


Figure 14.17 Boundary Analysis Between Oxide (200 Series) Vs Fresh (100 series) at Enioda Prospect

14.9 Basic Statistics

Basic statistics for the 1 m gold composites are summarised in Table 14.5. Log-probability plots for gold grade are displayed in Figure 14.19 to Figure 14.33.

The gold grade is predominantly highly variable and shows clear evidence of multi-modality. This demonstrates that there are at least two sub-populations of gold within the lode envelopes. However, explicit sub-domaining of the higher-grade sub-population, which typically manifests at or above the 90th percentile of the grade distribution, is considered to be a risky approach, due to the limited continuity at these grade thresholds and the attendant risk that would be introduced in terms of defining the high-grade volume. It was therefore decided to use a non-linear grade interpolation method at Doropo, which is well suited to dealing with high grade variability and also provides the advantage of being able to predict the recoverable grade-tonnage relationship at a smaller Selective Mining Unit (SMU) block size. Localised Uniform Conditioning (LUC), which is a non-linear method suited to open pit mining methods, was used to interpolate gold grade.

For the Chegue prospect, a large volume of internal waste has been included in the construction of the domains due to the discontinuous nature of the mineralisation. In order to constrain the extrapolation of grades into unmineralised volume, the use of an indicator interpolation to separate mineralised and unmineralised material was implemented for the main domains (5001, 5003, 5004, 5005, 5007, and 5008). The choice of the indicator threshold was based on the analysis of the log-probability plots for the domains. They all display a population break around 0.2 g/t Au. An example is given in Figure 14.18.

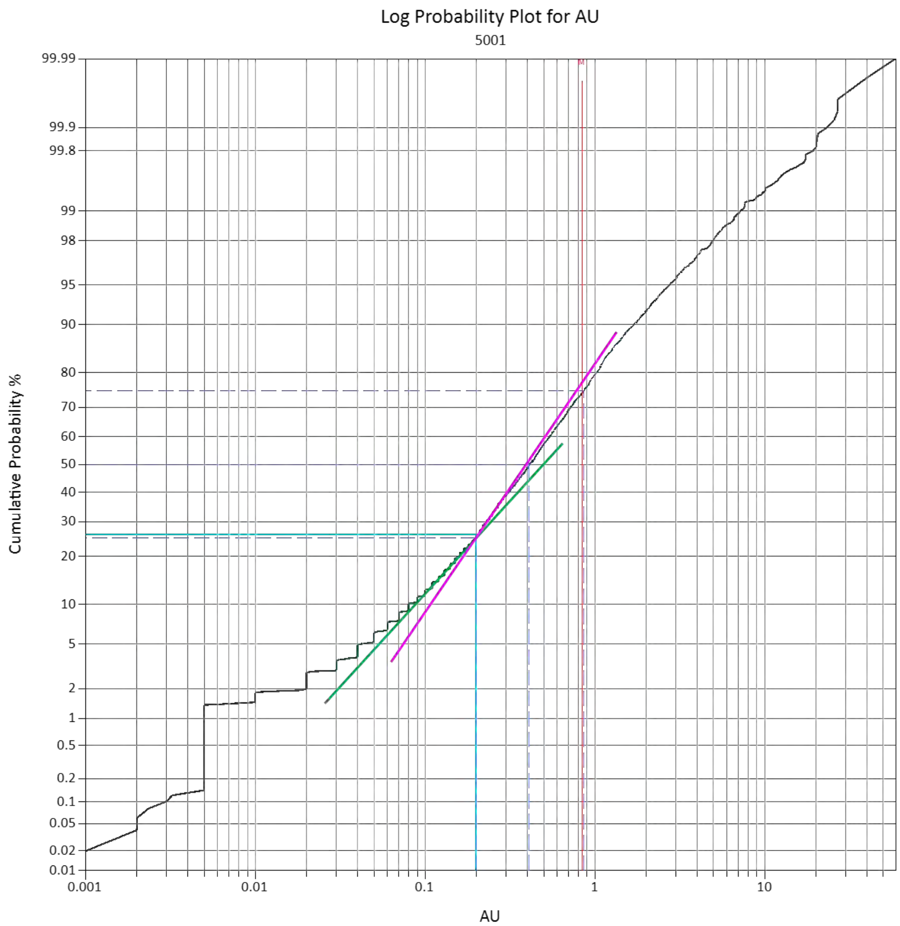


Figure 14.18 CHG Domain 5001 Log-probability Plot with Population Break at 0.2 g/t Au

Prospect	Domain	N	Min	Max	Mean	Median	Std Dev	CoV
ATI	12001	1420	0.005	123.80	1.75	0.35	6.38	3.643
	12002	70	0.005	4.64	0.51	0.29	0.81	1.575
	12003	60	0.005	6.07	0.56	0.32	0.92	1.646
	12004	53	0.005	4.01	0.62	0.32	0.82	1.327
CHG	50011 (Ore)	4162	0.002	59.00	0.95	0.49	2.00	2.108
	50012 (Waste)	890	0.001	20.47	0.35	0.13	1.11	3.176
	5002	186	0.002	18.84	0.88	0.40	1.83	2.072
	50031 (Ore)	326	0.01	21.06	0.93	0.38	1.88	2.025
	50032 (Waste)	51	0.005	2.67	0.21	0.06	0.42	2.004
	50041 (Ore)	747	0.002	79.21	1.18	0.41	4.50	3.817
	50042 (Waste)	1082	0.001	229.77	0.49	0.06	7.03	14.475
	50051 (Ore)	1451	0.001	139.70	1.34	0.40	6.58	4.914
	50052 (Waste)	1066	0.001	23.83	0.28	0.06	0.99	3.518
	5006	177	0.005	5.22	0.21	0.03	0.53	2.486
	50071 (Ore)	484	0.001	66.20	0.97	0.33	3.89	4.030
	50072 (Waste)	682	0.001	4.44	0.20	0.05	0.39	1.930
	50081 (Ore)	1352	0.002	74.25	1.16	0.42	3.99	3.441
	50082 (Waste)	982	0.001	24.16	0.38	0.06	1.53	3.991
	5020	24	0.005	4.37	0.96	0.32	1.21	1.255
	5021	63	0.005	2.69	0.30	0.21	0.37	1.241
ENI	10001	3342	0.001	70.60	1.01	0.42	2.80	2.770
	10002	52	0.055	5.83	0.60	0.30	0.85	1.417
HAN	9001	3681	0.001	199.60	1.90	0.46	8.98	4.740
	9004	107	0.003	7.44	0.37	0.13	0.82	2.226
	9005	138	0.001	3.29	0.44	0.24	0.54	1.215
HND	7001	157	0.005	6.21	0.50	0.23	0.82	1.652
	7002	91	0.005	11.30	1.43	0.64	1.96	1.367
	7003	144	0.005	15.02	1.04	0.34	2.14	2.060
	7004	30	0.005	6.00	0.59	0.20	1.20	2.032
HNDS	7010	480	0.005	12.48	0.56	0.27	1.00	1.798
	7011	97	0.005	2.89	0.37	0.21	0.50	1.348
	7012	118	0.005	14.55	0.71	0.16	1.99	2.797
	7013	101	0.005	5.83	0.58	0.28	0.84	1.445
	7014	14	0.005	1.86	0.36	0.20	0.49	1.347
	7015	60	0.005	1.30	0.18	0.13	0.21	1.137
KEK	8001	4849	0.003	204.70	0.87	0.33	4.36	5.024
	8003	1214	0.005	55.11	0.80	0.32	2.79	3.504

Prospect	Domain	N	Min	Max	Mean	Median	Std Dev	CoV
KLG	1001	2504	0.005	8.73	0.71	0.46	0.80	1.126
	1002	964	0.005	3.85	0.48	0.34	0.48	1.009
	1003	516	0.005	6.74	0.92	0.58	1.00	1.094
	1004	5605	0.005	53.83	1.17	0.57	2.06	1.763
	1005	169	0.005	9.10	0.85	0.44	1.21	1.421
	1006	5548	0.005	105.70	0.88	0.50	1.84	2.078
	1007	539	0.005	16.34	1.04	0.43	1.81	1.742
	1009	3028	0.001	38.94	0.78	0.42	1.36	1.759
	1011	384	0.005	5.59	0.64	0.29	0.86	1.348
	1012	211	0.005	3.57	0.50	0.36	0.48	0.947
	1015	587	0.005	13.71	0.51	0.26	0.96	1.866
	1016	345	0.005	3.83	0.51	0.29	0.63	1.232
	1021	194	0.005	13.71	0.50	0.25	1.12	2.229
	1022	23	0.005	1.20	0.24	0.21	0.25	1.052
	1023	71	0.03	1.27	0.34	0.24	0.29	0.846
NAR	11001	226	0.005	16.16	0.92	0.50	1.56	1.689
NOK	4001	4103	0.005	127.80	1.91	0.59	5.88	3.071
	4002	81	0.005	2.91	0.41	0.24	0.49	1.206
	4004	1484	0.002	103.90	1.11	0.23	4.90	4.409
	4006	163	0.005	10.33	0.75	0.32	1.23	1.645
SAN	13001	727	0.005	49.51	1.20	0.35	3.76	3.138
	13002	121	0.005	10.38	0.37	0.09	1.03	2.799
	13003	36	0.005	2.36	0.42	0.22	0.56	1.330
SOL	14001	95	0.005	14.27	1.48	0.28	2.95	1.996
	14002	48	0.005	8.38	0.78	0.24	1.63	2.084
SWA	3003	5132	0.005	105.56	0.95	0.38	3.78	3.980
	3004	2406	0.005	134.37	1.37	0.44	6.00	4.391
	3005	387	0.005	18.87	0.50	0.21	1.35	2.694
	3006	4507	0.001	182.39	1.72	0.65	6.28	3.642
	3009	927	0.005	126.40	1.56	0.57	6.55	4.213
	3010	435	0.005	306.13	1.91	0.37	16.62	8.683
THN	6001	1292	0.005	49.31	0.60	0.33	1.97	3.287
	6002	68	0.02	3.46	0.48	0.34	0.54	1.123
	6003	287	0.005	117.90	1.34	0.27	8.50	6.357
	6004	52	0.005	3.84	0.80	0.41	0.93	1.155
	6005	132	0.005	8.43	0.49	0.20	0.99	2.026
	6006	138	0.005	16.03	0.58	0.24	1.56	2.692

Prospect	Domain	N	Min	Max	Mean	Median	Std Dev	CoV
VAKO	2001	943	0.005	7.91	0.38	0.20	0.62	1.620
	2002	3784	0.005	19.81	0.54	0.30	0.85	1.577
	2004	47	0.005	5.25	0.54	0.27	0.87	1.605
	2006	119	0.005	8.11	0.50	0.26	0.89	1.795

Table 14.5 Basic Statistics of the Composites (naïve) all Domains in the Various Updated Prospect Areas

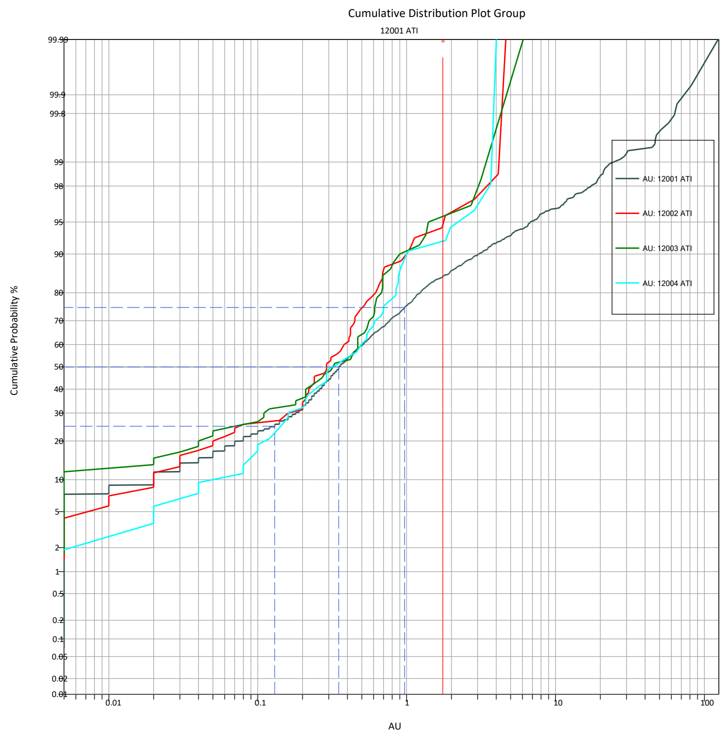


Figure 14.19 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ATI Domains

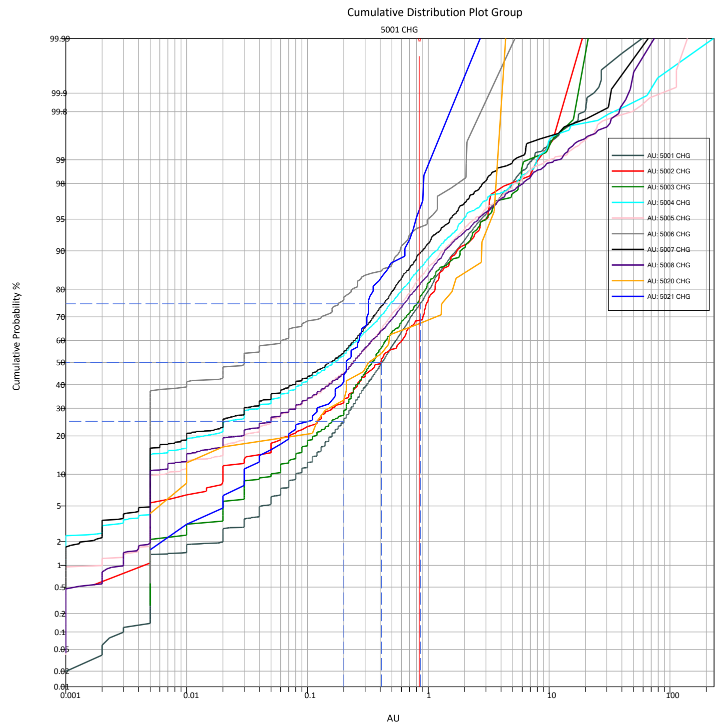


Figure 14.20 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - CHG Domains

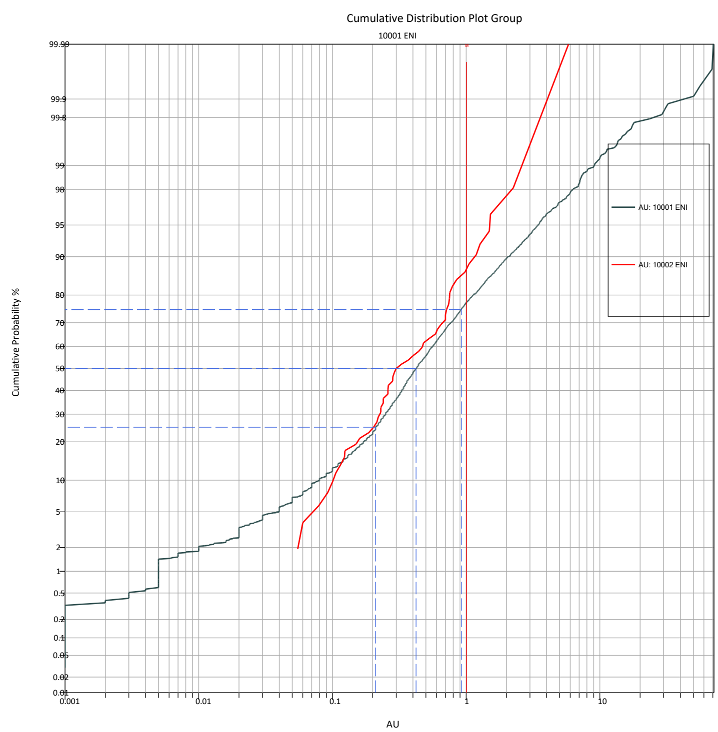


Figure 14.21 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - ENI Domains

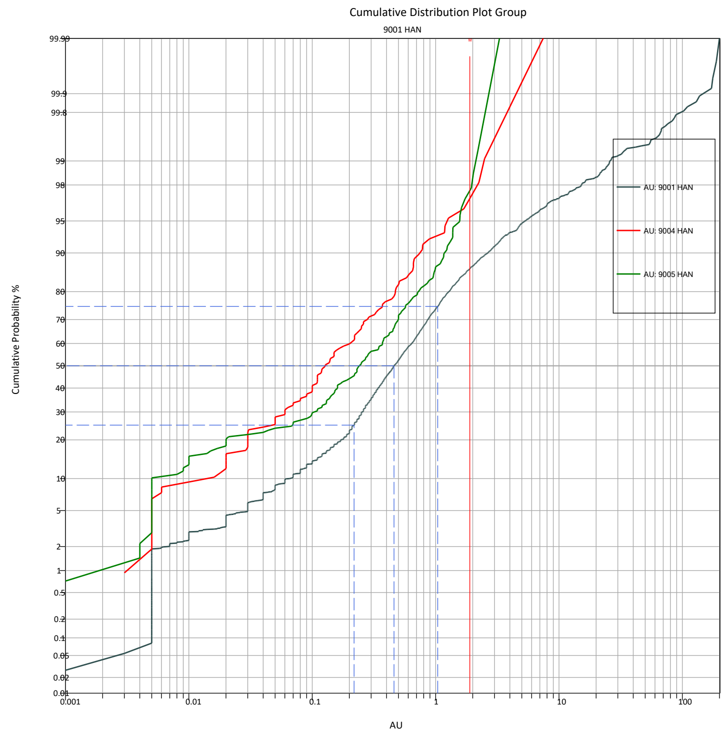


Figure 14.22 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HAN Domains

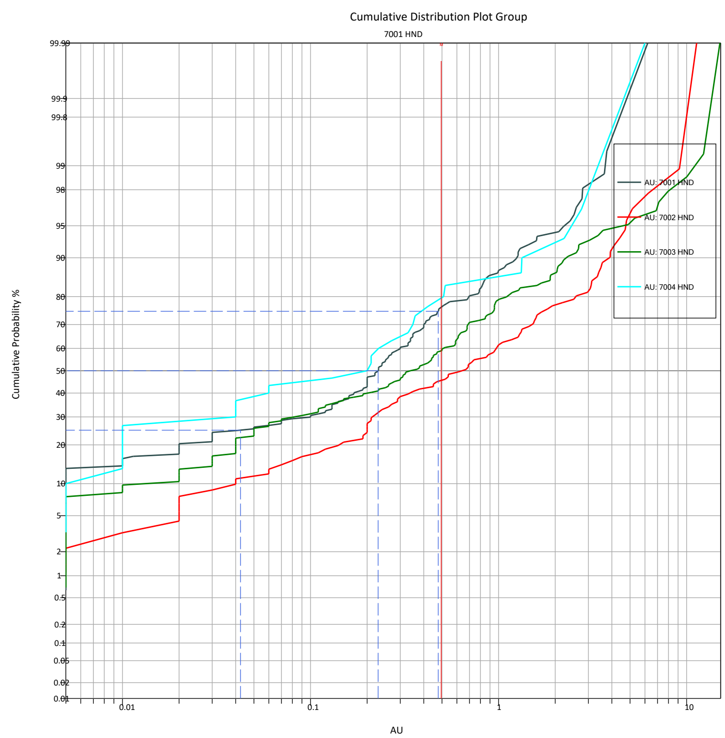


Figure 14.23 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HND Domains

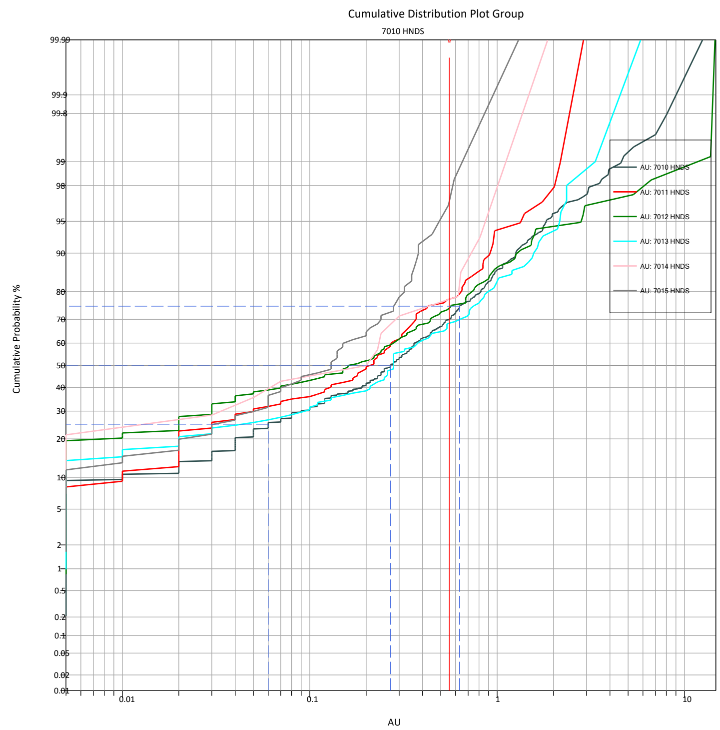


Figure 14.24 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - HNDS Domains

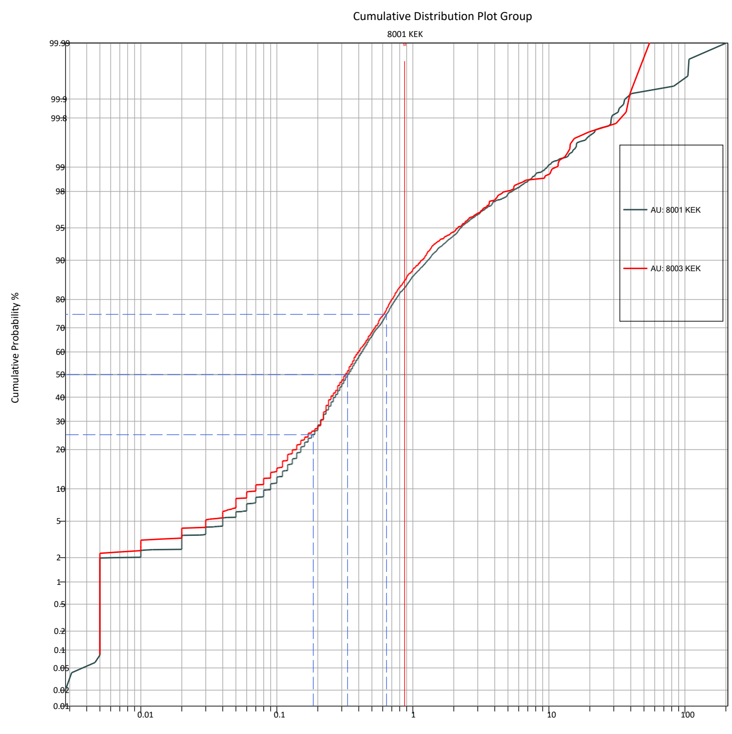


Figure 14.25 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KEK Domains

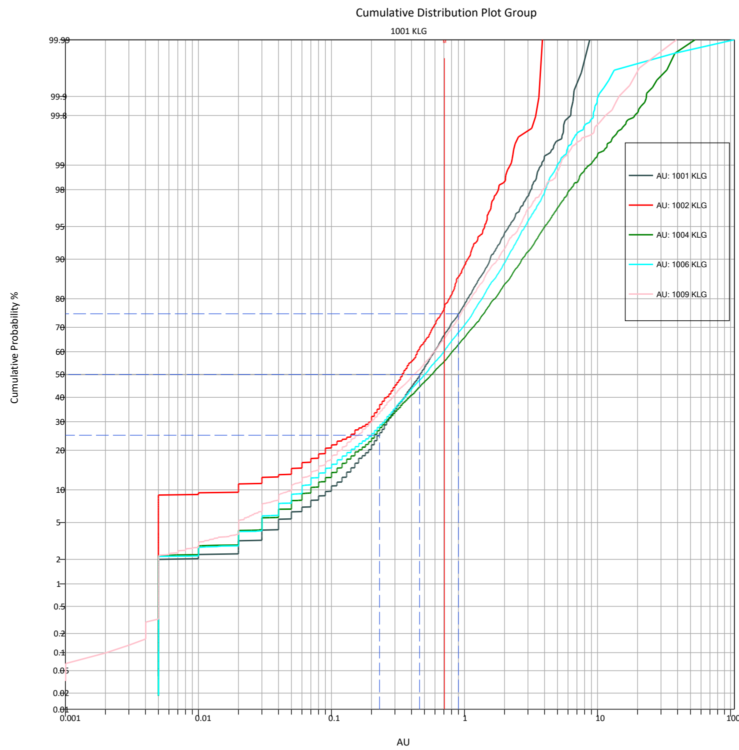


Figure 14.26 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - KLG Major Domains

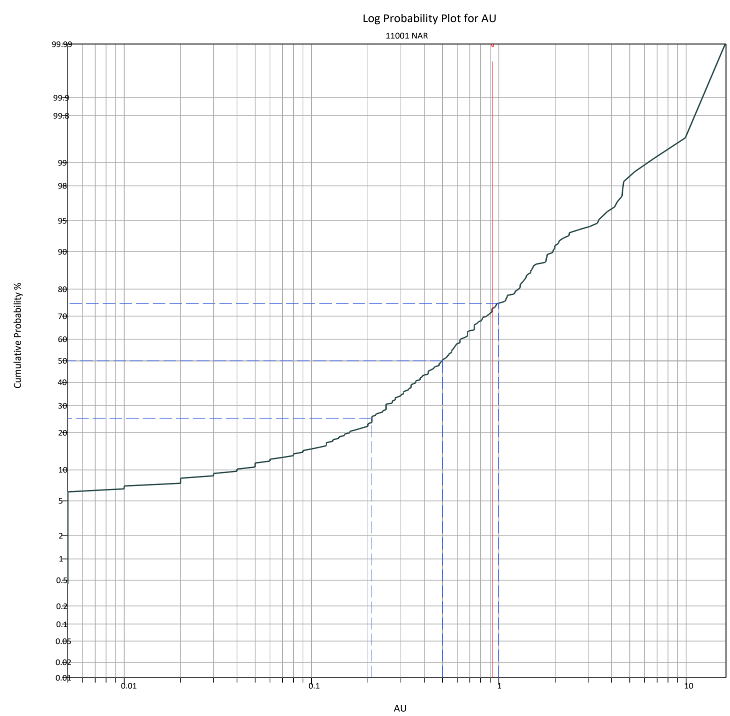


Figure 14.27 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NAR Minor Domains

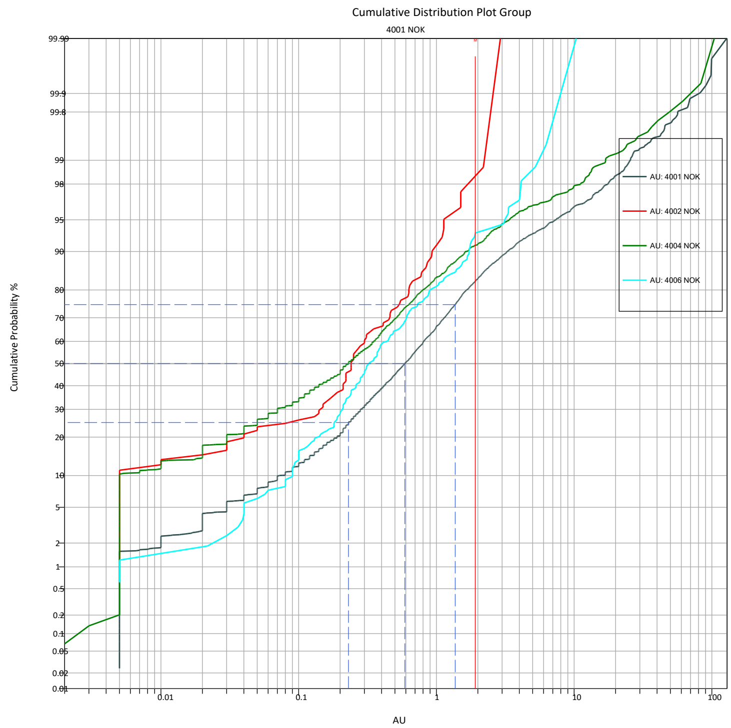


Figure 14.28 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - NOK Domains

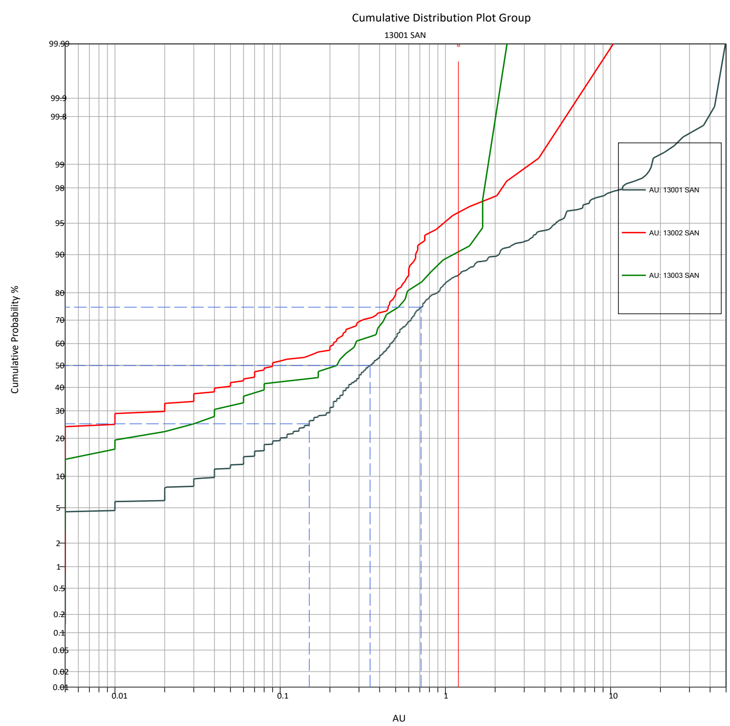


Figure 14.29 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SAN Domains

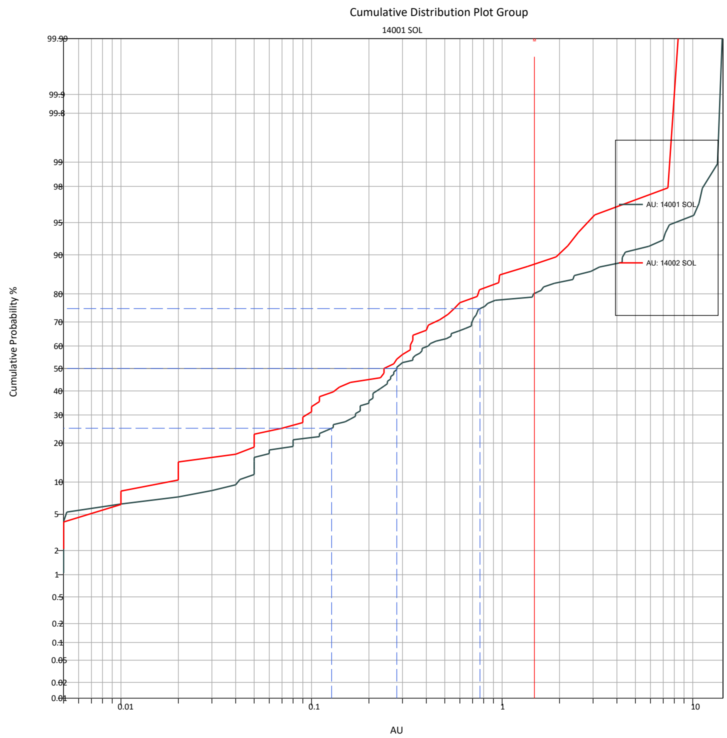


Figure 14.30 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SOL Domains

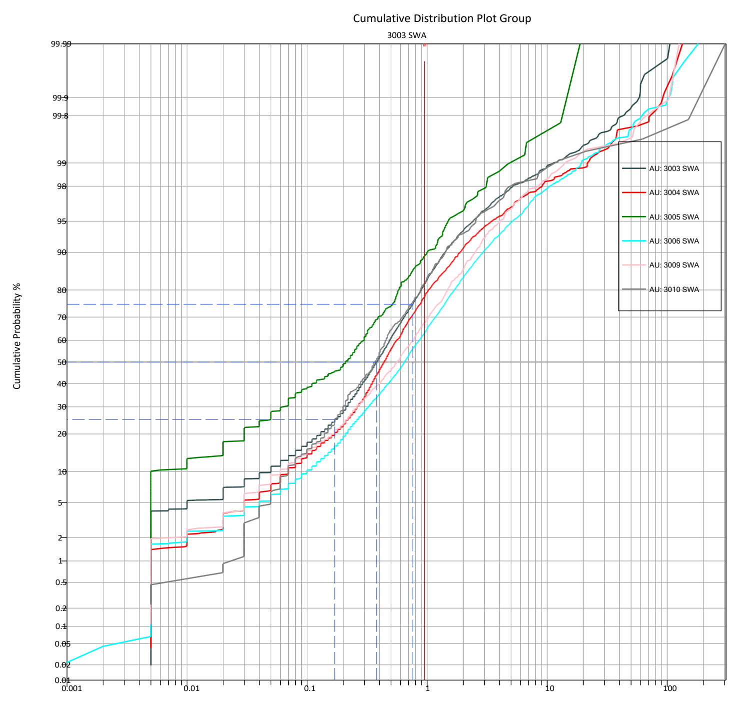


Figure 14.31 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - SWA Domains

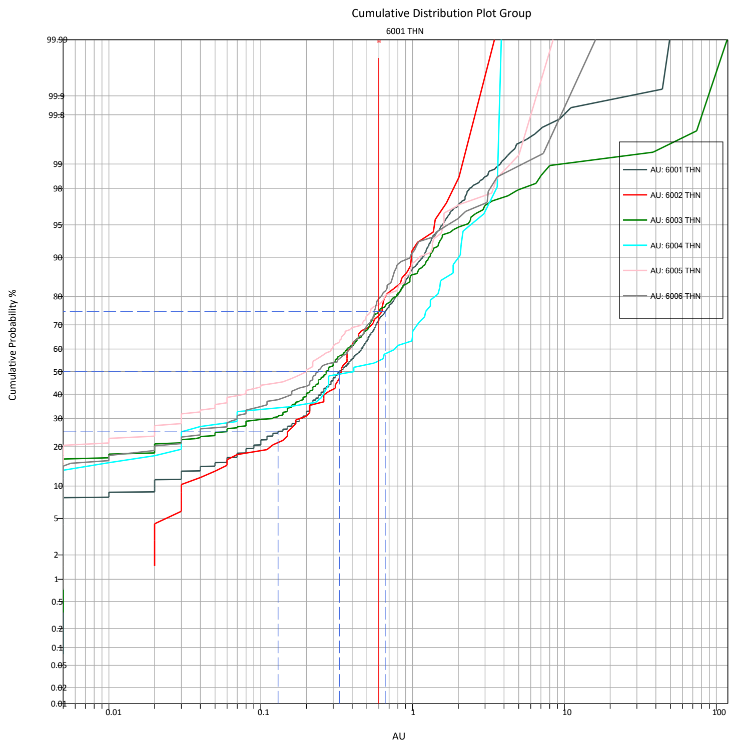


Figure 14.32 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - THN Domains

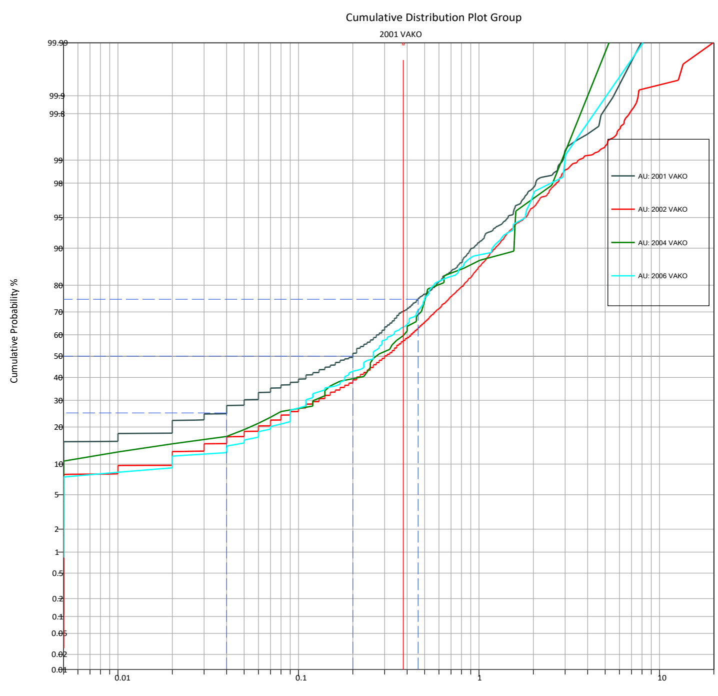


Figure 14.33 Log-probability Plots - 1 m Composite Gold Grade (Au g/t) - VAKO Domains

14.10 Gold Grade Caps and Distance Limiting

Gold grade caps were chosen per estimation domain, based on the following criteria:

- Examination of the grade distribution for each domain, noting the point at which the upper tail of the distribution loses support;
- By taking into account the variability and mineralisation continuity of the domain in question. More punitive caps were applied to areas of lesser continuity (chiefly in smaller, more poorly informed domains);
- By visual examination of the position of outlier values relative to surrounding samples and their values.

Generally speaking, the gold grade caps had only a minor impact on global mean values for the economically most important estimation domains, hence the risk to the MRE due to these actions is considered to be low.

An additional distance limiting constraint was applied to high grade samples in most domains. The grade threshold for distance limiting was chosen based on identification of prominent inflexion points in the log-probability plots of the grade distribution per estimation domain that are believed to represent the onset of the anomalously high-grade sub-population. Distance thresholds were generally set between 25 m and 60 m, dependent on nominal drill spacing and the observed continuity of higher-grade zones internal to the lode estimation domains.

The grade caps, distance limiting parameters and density truncations used for estimation are listed in Table 14.6.

Prospect	Domain	Top Cap (Au g/t)	No. Capped	Uncapped Mean (Au g/t)	Capped Mean (Au g/t)	%Diff Mean	Dist Lim Threshold (Au g/t)	Distance Limit (m)
ATI	12001	32.00	10	1.75	1.53	-12%	5.00	50
	12002	-	-	0.51	0.51	0%	0.70	25
	12003	4.00	1	0.56	0.52	-6%	1.00	25
	12004	-	-	0.62	0.62	0%	1.20	25
CHG	50011 (Ore)	28.00	2	0.95	0.94	-1%	-	-
	50012 (Waste)	4.50	6	0.35	0.31	-13%	-	-
	5002	8.00	2	0.88	0.81	-8%	-	-
	50031 (Ore)	12.00	2	0.93	0.89	-4%	-	-
	50032 (Waste	-	0	0.21	0.21	0%	-	-
	50041 (Ore)	18.00	5	1.18	0.97	-17%	-	-

Prospect	Domain	Top Cap (Au g/t)	No. Capped	Uncapped Mean (Au g/t)	Capped Mean (Au g/t)	%Diff Mean	Dist Lim Threshold (Au g/t)	Distance Limit (m)
CHG Cont'd	50042 (waste)	10.00	2	0.49	0.28	-43%	-	-
	50051 (Ore)	30.00	7	1.34	1.09	-19%	-	-
	50052 (Waste)	5.00	6	0.28	0.25	-10%	-	-
	5006	3.00	1	0.21	0.20	-6%	-	-
	50071 (Ore)	22.00	3	0.97	0.83	-14%	-	-
	50072 (Waste)	4.00	1	0.20	0.20	0%	-	-
	50081 (Ore)	51.00	1	1.16	1.14	-1%	4.00	36
	50082 (Waste)	10.00	5	0.38	0.34	-10%	1.50	25
	5020	-	0	0.96	0.96	0%	0.50	25
	5021	-	0	0.30	0.30	0%	-	-
ENI	10001	20.00	3	1.01	0.95	-6%	7.00	50
	10002	8.00	1	0.60	0.55	-9%	1.50	25
HAN	9001	20.00	60	1.90	1.35	-29%	20.00	25
	9004	3.00	1	0.37	0.33	-11%	0.80	25
	9005	2.50	1	0.44	0.44	-1%	0.60	30
HND	7001	-	-	0.50	0.50	1%	0.50	50
	7002	7.50	2	1.43	1.16	-19%	0.80	25
	7003	-	-	1.04	0.93	-11%	1.00	40
	7004	-	-	0.59	0.58	-2%	0.70	25
HNDS	7010	6.00	3	0.56	0.54	-3%	2.00	50
	7011	-	-	0.37	0.37	0%	-	-
	7012	8.00	2	0.71	0.61	-15%	2.00	50
	7013	4.00	1	0.58	0.56	-3%	-	-
	7014	-	-	0.36	0.36	0%	-	-
	7015	-	-	0.18	0.18	0%	-	-
KEK	8001	40.00	5	0.87	0.80	-8%	5.00	35
	8003	20.00	4	0.80	0.73	-9%	4.00	30
KLG	1001	7.00	2	0.71	0.71	0%	3.00	60
	1002	2.50	5	0.48	0.47	-1%	1.60	60
	1003	5.00	4	0.92	0.91	0%	3.00	35
	1004	20.00	11	1.17	1.15	-1%	7.00	35
	1005	6.00	2	0.85	0.83	-3%	2.00	35
	1006	13.00	3	0.88	0.86	-2%	5.00	60
	1007	15.00	1	1.04	1.04	0%	5.00	35
	1009	15.00	3	0.78	0.76	-1%	5.00	60
	1011	4.00	3	0.64	0.63	-1%	3.00	35
	1012	2.00	2	0.50	0.49	-2%	1.50	60
	1015	7.00	2	0.51	0.50	-3%	2.50	35

Prospect	Domain	Top Cap (Au g/t)	No. Capped	Uncapped Mean (Au g/t)	Capped Mean (Au g/t)	%Diff Mean	Dist Lim Threshold (Au g/t)	Distance Limit (m)
KLG Cont'd	1016	2.50	6	0.51	0.50	-2%	1.40	35
	1021	3.00	2	0.50	0.44	-13%	2.00	35
	1022	-	0	0.24	0.24	0%	-	-
	1023	-	0	0.34	0.34	0%	-	-
NAR	11001	8.00	2	0.92	0.84	-9%	1.70	50
NOK	4001	60.00	8	1.91	1.86	-3%	15.00	30
	4002	-	0	0.41	0.41	0%	1.50	30
	4004	45.00	4	1.11	1.03	-7%	10.00	30
	4006	8.00	1	0.75	0.73	-2%	2.00	30
SAN	13001	30.00	3	1.20	1.27	6%	1.25	40
	13002	5.00	1	0.37	0.33	-11%	1.00	50
	13003	-	-	0.42	0.46	11%	-	-
SOL	14001	-	0	1.48	0.86	-42%	1.50	50
	14002	4.00	2	0.78	0.58	-26%	1.50	50
SWA	3003	60.00	4	0.95	0.93	-2%	4.00	40
	3004	45.00	8	1.37	1.22	-11%	3.00	40
	3005	8.00	2	0.50	0.46	-8%	1.50	40
	3006	80.00	7	1.72	1.66	-4%	7.50	40
	3009	30.00	6	1.56	1.28	-18%	7.00	40
	3010	20.00	4	1.91	0.85	-55%	1.50	40
THN	6001	13.00	2	0.60	0.52	-13%	2.00	50
	6002	-	-	0.48	0.54	11%	-	-
	6003	15.00	3	1.34	0.57	-57%	3.00	50
	6004	-	-	0.80	0.61	-24%	-	-
	6005	4.00	2	0.49	0.41	-16%	1.50	50
	6006	5.00	2	0.58	0.53	-9%	-	-
VAKO	2001	5.00	2	0.38	0.33	-14%	1.40	50
	2002	8.00	3	0.54	0.54	-1%	1.50	25
	2004	3.00	1	0.54	0.50	-8%	1.00	50
	2006	4.00	1	0.50	0.43	-13%	-	-

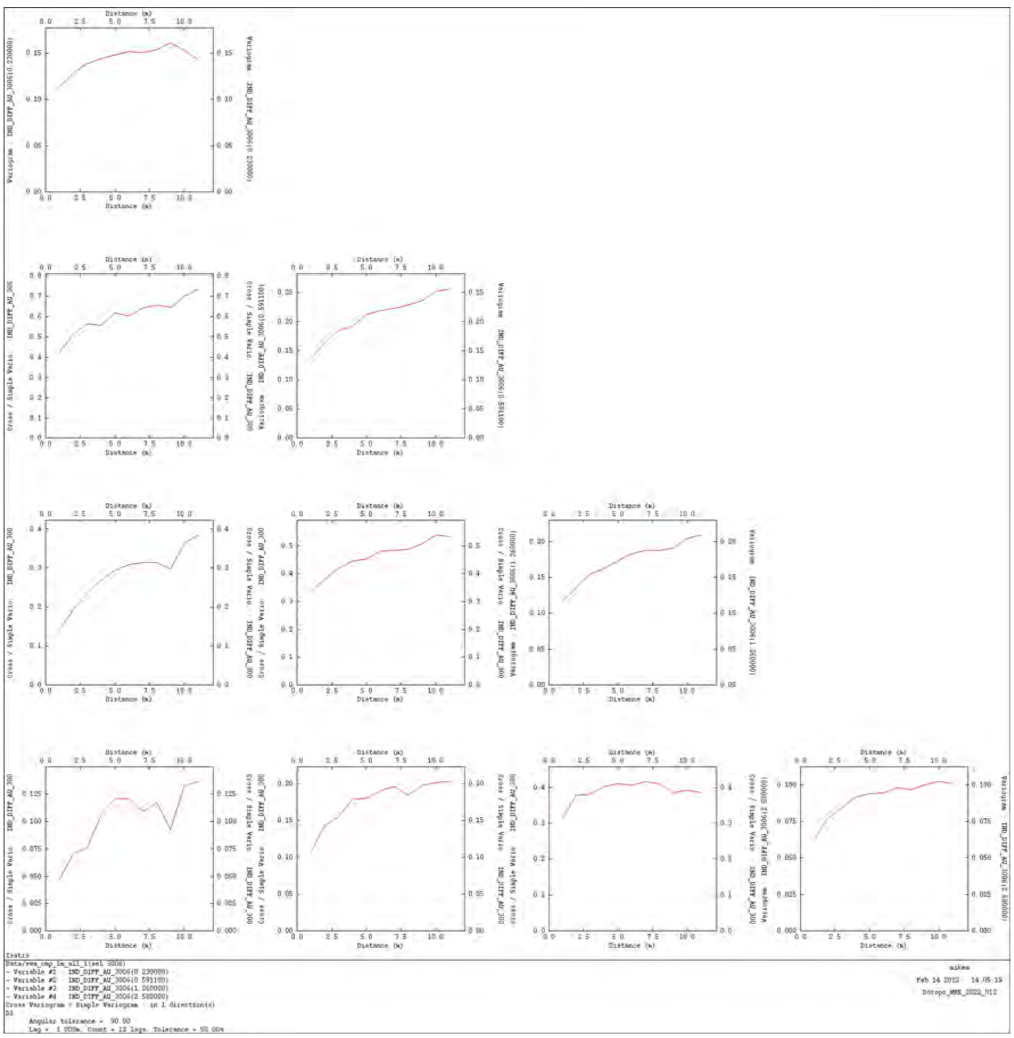
Table 14.6 Listing of Gold Grade Caps and Distance Limiting Parameters per Mineralisation/Estimation Domain

14.11 Diffusivity Tests

Diffusivity tests aim to test the grade architecture. The grade architecture will indicate which non-linear estimation method is preferred. When the architecture is diffusive, a Gaussian-method is recommended (such as Uniform Conditioning - or UC - and LUC); when the architecture is mosaic, Indicator-based methods are preferred (such as Multiple Indicator Kriging - or MIK - and LMIK).

Diffusivity tests were undertaken for the previous MRE in 2022 on gold grade, for a selection of the larger domains at various prospects (Figure 14.34). The diffusivity test is based on indicator variable variograms, defined at a range of threshold grades across the grade distribution. When the cross-variogram of two of the indicators is divided by the variogram of the lower grade indicator, a measure of the diffusivity between the thresholds is measured. If the quotient variogram exhibits some structure (i.e. not pure noise or a straight line), as is the case for all the tests undertaken at Doropo (Figure 14.34), this indicates that the grade is transitioning in a gradational fashion (diffusive). This result justifies the use of non-linear Gaussian methods for estimation (such as the LUC method used here).

The diffusivity tests were not repeated for this 2023 MRE update, as this characteristic of the mineralisation is considered to be well established in previous work.



Note: The right diagonal shows the indicator variograms at various threshold grades, and the remainder of the matrix shows the quotient of the indicator cross-variograms and variograms.

Figure 14.34 Diffusion Test for Gold Grade Based on 1 m Composites (2022 MRE)

14.12 Variography

Variogram models for the gold grade variable, per mineralisation/estimation domain, were produced by transforming the capped composite data to Gaussian space, modelling the spatial structure, and then back-transforming the model to real space for use in estimation. This process reduces the impact of outliers on the experimental variogram calculation, allowing for elucidation of the true underlying spatial structure.

In all cases, the major and semi-major directions were modelled with equal ranges. This is because there was, in most cases, no compelling evidence of significant anisotropy within the plane of maximum continuity, although there is sometimes visual evidence of plunging high-grade shoots, which would need to be further corroborated in the future by more dense drilling.

Variograms were modelled for the domains where robust experimental variograms could be obtained, typically larger domains containing more samples, and substitutions or grouping were made for the remainder of the domains based on proximity and statistical similarity.

The variogram model parameters used for interpolation are summarised in Table 14.7, with sills normalised to 1. An example of a gold variogram is in Figure 14.35.

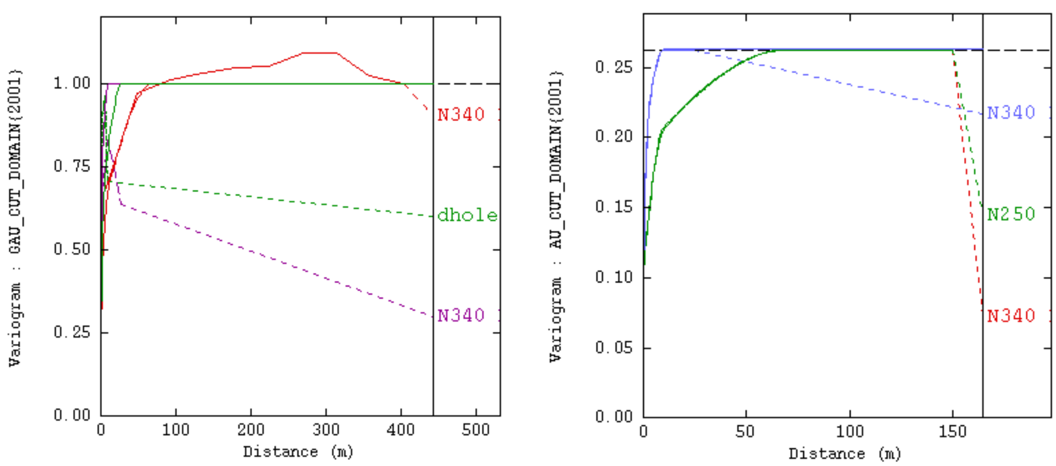


Figure 14.35 Example Gaussian Variogram (left) and Back-Transformed Variogram (right) for Domain 2001 at VAKO

Area	Domain	Nugget	Spherical 1				Spherical 2				Spherical 2				Isatis Rotation (Geol. Pl.)		
			Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	A	+X	-Z
ATI	12001	19.23%	62.86%	11	11	4	10.77%	48	48	10	7.14%	120	120	12	-60	70	90
	12002	41.13%	27.42%	5	5	3	31.45%	80	80	6					-50	50	90
	12003	37.09%	45.01%	6	6	3	17.90%	35	35	5					-55	70	90
	12004	26.13%	45.73%	7	7	4	28.14%	45	45	8					-55	80	90
CHG	50011 (Ore)	52.91%	22.56%	8	8	3	14.59%	37	37	7	9.93%	120	120	13	245	35	90
	50012 (Waste)	48.86%	24.92%	8	8	3	12.06%	35	35	7	14.16%	150	150	13	245	35	90
	5002	34.13%	40.80%	7	7	2	25.07%	54	54	6	0.00%				220	25	90
	50031 (Ore)	41.40%	39.63%	14	14	3	6.09%	43	43	7	12.89%	160	160	7	195	25	90
	50032 (Waste)	42.42%	37.83%	14	14	3	6.73%	43	43	7	13.02%	160	160	7	195	25	90
	50041 (Ore)	49.41%	33.37%	9	9	2	17.22%	70	70	8					150	160	90
	50042 (Waste)	53.45%	32.36%	9	9	2	14.20%	70	70	8					150	160	90
	50051 (Ore)	58.96%	28.41%	5	5	2	12.62%	30	30	5					160	155	90
	50052 (Waste)	49.69%	28.95%	5	5	2	21.36%	45	45	5					160	155	90
	5006	26.56%	50.68%	8	8	4	22.76%	48	48	10					205	135	90
	50071 (Ore)	73.95%	18.55%	10	10	2	7.50%	35	35	5					190	150	90
	50072 (Waste)	43.94%	33.12%	10	10	2	22.94%	35	35	5					190	150	90
	50081 (Ore)	59.07%	33.11%	12	12	3	7.83%	51	51	7					190	150	90
	50082 (Waste)	48.30%	40.99%	12	12	3	10.71%	80	80	7					190	150	90
	5020	32.77%	23.61%	7	7	4	43.61%	65	65	10					245	40	90
	5021	45.76%	22.34%	7	7	4	31.90%	65	65	10					245	40	90
ENI	10001	35.00%	49.06%	9	9	3	15.94%	44	44	7					180	30	130
	10002	31.43%	48.98%	9	9	3	19.59%	47	47	8					180	30	130
HAN	9001	53.79%	37.41%	7.5	7.5	2.5	8.80%	38	38	7.5					210	25	0
	9004	34.14%	50.93%	5.5	5.5	2.5	14.93%	38	38	9					210	25	0
	9005	24.62%	53.90%	13	13	5	21.49%	53	53	10					210	25	0

Area	Domain	Nugget	Spherical 1				Spherical 2				Spherical 2				Isatis Rotation (Geol. Pl.)		
			Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	A	+X	-Z
HND	7001	19.94%	57.21%	18	18	3	22.85%	150	150	6					175	25	90
	7002	30.66%	27.25%	15	15	3	42.09%	95	95	6					155	30	90
	7003	44.43%	38.35%	13	13	3	17.22%	105	105	10					150	25	90
	7004	41.17%	38.54%	15	15	3	20.29%	50	50	8					180	25	90
HNDS	7010	40.85%	32.37%	10	10	3	14.29%	32	32	8	12.49%	105	105	10	215	25	90
	7011	43.04%	22.92%	14	14	3	34.04%	125	125	8					220	20	90
	7012	32.08%	48.00%	12	12	3	19.92%	60	60	6					220	15	90
	7013	18.36%	45.37%	12	12	3	36.27%	60	60	9					-40	165	90
	7014	23.17%	33.89%	10	10	3	42.94%	105	105	10					200	20	90
	7015	28.23%	35.38%	15	15	3	36.39%	60	60	8					210	5	90
KEK	8001	44.02%	43.83%	9	8	2	12.15%	58	57	10					225	30	0
	8003	35.91%	48.92%	10	10	3	15.17%	45	45	7					225	30	0
KLG	1001	20.32%	51.80%	9	9	4.5	27.88%	62	62	10					125	30	0
	1002	17.99%	50.89%	10	10	4	31.12%	74	74	10					115	30	0
	1003	13.42%	44.18%	9	9	3.5	42.41%	34	34	10					130	35	0
	1004	31.63%	38.62%	7	7	3	29.75%	44	44	9					125	35	0
	1005	27.89%	36.34%	8	8	3.5	35.77%	46	46	10					115	35	0
	1006	23.49%	49.21%	11	11	4.5	27.30%	89	89	12					115	30	0
	1007	27.40%	49.59%	7	7	2.5	23.01%	29	29	6					140	30	0
	1009	31.57%	45.09%	11	11	3.5	23.35%	84	84	10					120	35	0
	1011	14.62%	47.95%	7.5	7.5	3.5	37.43%	41	41	10					140	30	0
	1012	18.42%	50.74%	10	10	4.5	30.84%	64	64	10					150	35	0
	1015	17.05%	57.21%	9	9	3.5	25.74%	37	37	8					125	30	0
	1016	20.24%	50.52%	7.5	7.5	2.5	29.24%	29	29	7					130	30	0
	1021	19.21%	38.12%	7.5	7.5	3.5	42.66%	36	36	8					115	25	0
	1022	27.26%	35.03%	7.5	7.5	3.5	37.71%	36	36	8					115	30	0
	1023	19.62%	37.10%	8	8	3.5	43.28%	36	36	8					115	35	0
NAR	11001	50.93%	26.28%	15	15	3	22.79%	200	200	8					215	40	90

Area	Domain	Nugget	Spherical 1				Spherical 2				Spherical 2				Isatis Rotation (Geol. Pl.)		
			Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	Sill	Major (m)	Semi (m)	Minor (m)	A	+X	-Z
NOK	4001	24.13%	57.12%	7	7	4	18.76%	26	26	7					260	40	0
	4002	14.80%	52.55%	7.5	7.5	4.5	32.65%	29	29	8					210	55	0
	4004	32.08%	57.90%	6	6	3	10.02%	25	25	6					220	40	0
	4006	16.90%	53.74%	7	7	4	29.36%	28	28	8					270	40	0
SAN	13001	46.82%	37.88%	10	10	5	3.84%	40	40	8	11.46%	120	120	8	120	70	90
	13002	54.23%	26.21%	8	8	3	6.48%	50	50	8	13.08%	125	125	10	120	80	90
	13003	43.99%	34.90%	10	10	3	21.12%	42	42	5					120	75	90
SOL	14001	34.93%	40.88%	12	12	3	24.19%	80	80	8					120	95	90
	14002	38.84%	29.54%	12	12	3	31.62%	80	80	8					120	95	90
SWA	3003	37.64%	53.37%	15	15	2.5	8.99%	110	110	8.5					200	27	0
	3004	38.24%	51.65%	12	12	3	10.11%	70	70	7					200	27	0
	3005	36.73%	45.58%	7	7	2.5	17.69%	42	42	7					200	27	0
	3006	45.91%	43.97%	15	15	3.5	10.12%	125	125	13					225	30	0
	3009	39.62%	40.40%	14	14	2	19.98%	60	60	5.5					215	30	0
	3010	40.39%	43.20%	7	7	3	16.41%	55	55	8					0	0	0
THN	6001	31.67%	46.37%	25	25	4	9.72%	75	75	5	12.25%	300	300	8	155	40	90
	6002	31.45%	39.45%	15	15	4	29.10%	110	110	5					100	35	90
	6003	36.36%	35.65%	15	15	4	27.99%	115	115	5					155	30	90
	6004	23.39%	44.44%	12	12	4	32.17%	67	67	8					145	45	90
	6005	24.30%	49.23%	10	10	4	26.47%	75	75	9					150	30	90
	6006	30.61%	40.53%	18	18	4	14.82%	65	65	9	14.03%	150	150	9	160	35	90
VAKO	2001	31.39%	26.42%	10	10	3	21.10%	70	70	10	21.10%	70	70	10	250	20	90
	2002	36.14%	32.52%	10	10	3	17.87%	25	25	8	13.48%	80	80	10	250	25	90
	2004	34.24%	30.05%	10	10	3	20.52%	25	25	8	15.19%	80	80	10	260	30	90
	2006	34.47%	31.91%	10	10	3	19.27%	25	25	8	14.35%	80	80	10	-90	25	90

Table 14.7 Variogram Model Parameters per Mineralisation/Estimation Domain - Sills Normalised to 100%

14.13 Block Model Definition

A block model was defined per prospect area, except for Nokpa and Chegue which were combined into one block model as they are within proximity of one another. Some of the block models were rotated to conform to the orientation of the lodes. The details for each block model definition are listed in Table 14.8.

Prospect	Item	X	Y	Z
ATI	Minimum Coordinate	483,000	1,066,700	-50
	Maximum Coordinate	483,500	1,069,700	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	-50	0	0
CHG-NOK	Minimum Coordinate	478,370	1,077,200	-200
	Maximum Coordinate	481,690	1,081,670	450
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	35	0	0
ENI	Minimum Coordinate	491,250	1,074,000	-50
	Maximum Coordinate	492,510	1,076,600	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	0	0	0
HAN	Minimum Coordinate	485,550	1,073,780	-50
	Maximum Coordinate	486,820	1,076,080	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	35	0	0
HND	Minimum Coordinate	481,700	1,072,900	-50
	Maximum Coordinate	482,620	1,073,700	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	0	0	0
HNDS	Minimum Coordinate	481,200	1,071,600	-50
	Maximum Coordinate	482,500	1,074,600	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	35	0	0
KEK	Minimum Coordinate	483,000	1,072,030	-50
	Maximum Coordinate	484,120	1,074,080	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	35	0	0

Prospect	Item	X	Y	Z
KLG	Minimum Coordinate	464,260	1,050,710	0
	Maximum Coordinate	466,850	1,060,000	450
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	-60	0	0
NAR	Minimum Coordinate	494,900	1,072,800	-50
	Maximum Coordinate	495,450	1,073,620	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	30	0	0
SAN	Minimum Coordinate	470,500	1,069,200	-50
	Maximum Coordinate	471,850	1,069,650	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	25	0	0
SOL	Minimum Coordinate	487,200	1,073,300	-50
	Maximum Coordinate	487,800	1,074,300	400
	Block Size	5	5	2.5
	Minimum Sub-block Size	1.25	1.25	0.625
	Rotation	120	0	0
SWA	Minimum Coordinate	478,000	1,074,000	-50
	Maximum Coordinate	479,550	1,076,700	450
	Block Size	5	5	2.5
	Minimum Sub-block Size	5	5	2.5
	Rotation	0	0	0
THN	Minimum Coordinate	483,850	1,074,000	-50
	Maximum Coordinate	485,890	1,077,720	450
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	0	0	0
VAKO	Minimum Coordinate	464,750	1,068,620	-50
	Maximum Coordinate	465,850	1,071,520	450
	Block Size	5	5	2.5
	Minimum Sub-block Size	2.5	2.5	1.25
	Rotation	70	0	0

Table 14.8 Block Model Definition per Prospect Area

14.14 Ordinary Kriging Interpolation

Ordinary Kriging (OK) interpolation of gold grades was undertaken into relatively large 'Panel' blocks, for which the dimensions are listed in Table 14.9. The Panel blocks were rotated in exactly the same manner as the ultimate SMU block size of 5 mX x 5 mY x 2.5 mRL, as per the rotation parameters listed in Table 14.8.

Dynamic Anisotropy (DA) was implemented to locally orient variogram models and search ellipsoids during OK interpolation, using the reference surfaces of the mineralisation/estimation domain wireframes produced in Leapfrog Geo to populate the block model with local rotation parameters.

One to two search pass interpolations were undertaken to fill all blocks within the lode estimation domains.

Isotropic search ellipsoids were used in accordance with the maximum distance and variogram anisotropy ratios. In some cases (HAN, KEK, KLG, NOK and SWA), the search ellipsoid was divided into four sectors (quadrants), with an optimum number of samples per quadrant set in order to produce a more optimal spread of informing samples for interpolation. The choice of optimum number of samples was selected by Kriging Neighbourhood Analysis (KNA), with an example given in Figure 14.36. Distance limiting parameters for the high-grade sub-population were implemented in order to mitigate against over-propagation of outlier values. The relevant interpolation parameters used are listed in Table 14.10.

Prospect	Panel Size (m)		
	X	Y	Z
ATI	20	20	10
CHG	20	20	5
ENI	20	20	10
HAN	20	20	5
HND	20	20	5
HNDS	20	20	5
KEK	20	20	5
KLG	20	20	5
NAR	20	20	5
NOK	10	20	5
SAN	20	20	5
SOL	20	20	5
SWA	10	20	5
THN	20	20	5
VAKO	20	20	5

Table 14.9 OK Panel Block Sizes Used for Gold Interpolation per Prospect

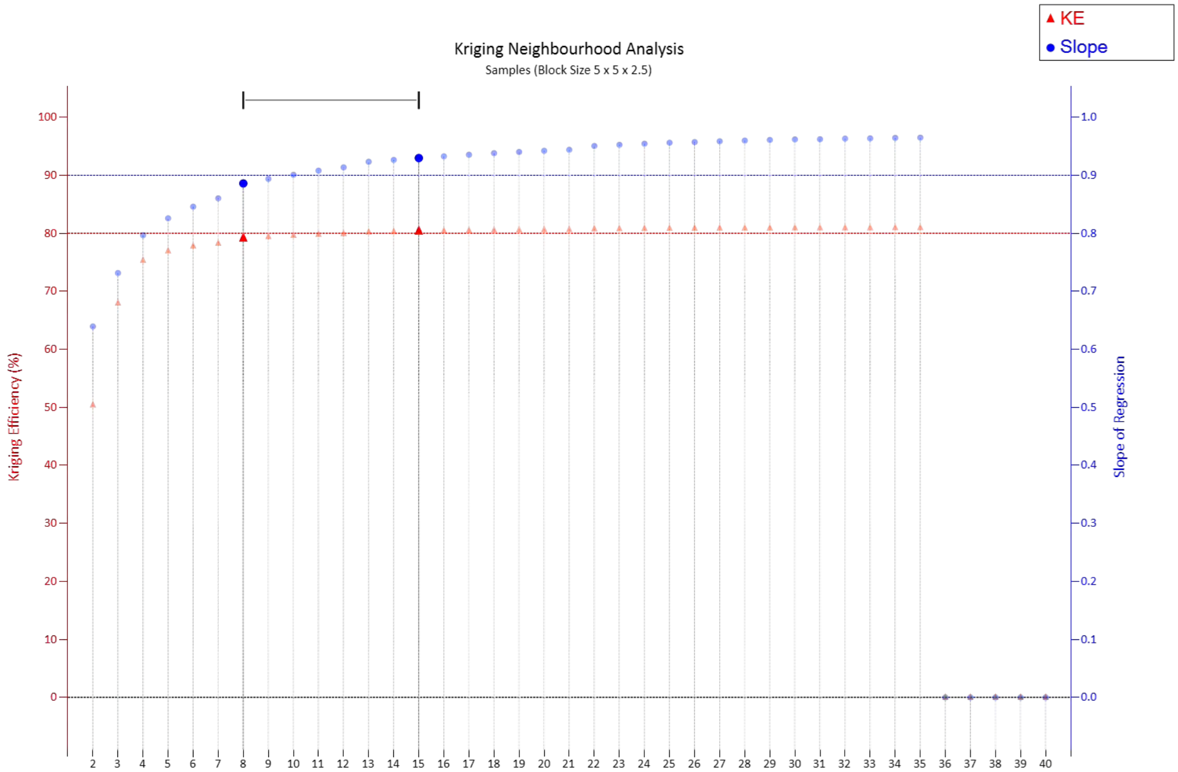


Figure 14.36 KNA Plot Using the Kriging Slope of Regression and Kriging Efficiency as Criteria for Selection of Minimum and Optimum Informing Composites - CHG domain 50041 (Ore domain 5004)

Area	Domain	1st Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	2nd Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	Dist Lim Threshold (Au g/t)	Dist Limit (m)
		Major	Semi	Minor				Major	Semi	Minor					
ATI	12001	100	100	8	11	1	16	400	400	100	5	1	18	5.00	50
	12002	100	100	15	13	1	20	400	400	100	5	1	20	0.70	25
	12003	100	100	15	13	1	20	400	400	100	5	1	20	1.00	25
	12004	60	60	5	8	1	16	400	400	100	5	1	20	1.20	25
CHG	50011 (Ore)	45	45	10	10	1	20	160	160	30	5	1	20	-	-
	50012 (Waste)	75	75	10	10	1	16	450	450	60	5	1	16	-	-
	5002	50	50	5	11	1	21	160	160	30	5	1	20	-	-
	50031 (Ore)	45	45	8	10	1	16	140	140	25	5	1	16	-	-
	50032 (Waste)	45	45	8	10	1	16	140	140	25	5	1	16	-	-
	50041 (Ore)	40	40	5	8	1	15	160	160	25	5	1	15	-	-
	50042 (Waste)	40	40	5	10	1	19	140	140	20	5	1	19	-	-
	50051 (Ore)	45	45	8	10	1	20	140	140	20	5	1	20	-	-
	50052 (Waste)	45	45	8	10	1	20	140	140	20	5	1	20	-	-
	5006	50	50	5	12	1	22	160	160	30	5	1	20	-	-
	50071 (Ore)	40	40	8	9	1	21	190	190	50	5	1	20	-	-
	50072 (Waste)	40	40	8	9	1	18	250	250	100	5	1	18	-	-
	50081 (Ore)	45	45	8	5	1	16	180	180	30	5	1	16	4.00	36
	50082 (Waste)	40	40	5	11	1	18	180	180	35	5	1	18	1.50	25
	5020	50	50	5	10	1	20	160	160	30	5	1	20	0.50	25
	5021	50	50	5	10	1	20	160	160	30	5	1	20	-	-
ENI	10001	165	165	55	8	1	5	200	200	45	5	1	20	7.00	50
	10002	75	75	25	8	1	5	-	-	-	-	-	-	1.50	25
HAN	9001	180	180	60	8	4	5	-	-	-	-	-	-	20.00	25
	9004	90	90	30	8	4	4	-	-	-	-	-	-	0.80	25
	9005	75	75	25	8	4	4	-	-	-	-	-	-	0.60	30

Area	Domain	1st Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	2nd Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	Dist Lim Threshold (Au g/t)	Dist Limit (m)
		Major	Semi	Minor				Major	Semi	Minor					
HND	7001	120	120	12	10	1	20	300	300	15	5	-	20	0.50	50
	7002	140	140	15	10	1	17	300	300	12	5	-	17	0.80	25
	7003	100	100	8	10	1	17	300	300	15	5	-	17	1.00	40
	7004	250	250	20	10	1	20	320	320	15	5	-	20	0.70	25
HNDS	7010	150	150	12	11	1	16	400	400	30	5	-	18	2.00	50
	7011	140	140	10	7	1	15	400	400	30	5	-	18	-	-
	7012	175	175	20	10	1	17	400	400	30	5	-	20	2.00	50
	7013	175	175	20	9	1	18	400	400	30	5	-	20	-	-
	7014	175	175	35	9	1	18	400	400	50	5	-	20	-	-
	7015	175	175	25	9	1	18	400	400	30	5	-	20	-	-
KEK	8001	105	105	35	8	4	6	-	-	-	-	-	-	5.00	35
	8003	120	120	40	8	4	6	-	-	-	-	-	-	4.00	30
KLG	1001	300	300	100	8	4	5	-	-	-	-	-	-	3.00	60
	1002	360	360	120	8	4	5	-	-	-	-	-	-	1.60	60
	1003	120	120	40	8	4	6	-	-	-	-	-	-	3.00	35
	1004	150	150	50	8	4	6	-	-	-	-	-	-	7.00	35
	1005	180	180	60	8	4	6	-	-	-	-	-	-	2.00	35
	1006	120	120	40	8	4	5	-	-	-	-	-	-	5.00	60
	1007	90	90	30	8	4	6	-	-	-	-	-	-	5.00	35
	1009	150	150	50	8	4	5	-	-	-	-	-	-	5.00	60
	1011	90	90	30	8	4	6	-	-	-	-	-	-	3.00	35
	1012	75	75	25	8	4	5	-	-	-	-	-	-	1.50	60
	1015	75	75	25	8	4	6	-	-	-	-	-	-	2.50	35
	1016	120	120	40	8	4	6	-	-	-	-	-	-	1.40	35
	1021	75	75	25	8	4	6	-	-	-	-	-	-	2.00	35
	1022	180	180	60	8	4	6	-	-	-	-	-	-	-	-
	1023	90	90	30	8	4	6	-	-	-	-	-	-	-	-
NAR	11001	140	140	10	10	1	17	350	350	40	5	-	20	1.70	50

Area	Domain	1st Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	2nd Search radii (m)			Min Comp	No. Sectors	Opt/ Sector	Dist Lim Threshold (Au g/t)	Dist Limit (m)
		Major	Semi	Minor				Major	Semi	Minor					
NOK	4001	390	390	130	8	4	4	-	-	-	-	-	-	15.00	30
	4002	180	180	60	8	4	4	-	-	-	-	-	-	1.50	30
	4004	150	150	50	8	4	4	-	-	-	-	-	-	10.00	30
	4006	60	60	20	8	4	4	-	-	-	-	-	-	2.00	30
SAN	13001	100	100	10	10	1	17	350	350	25	5	-	20	1.25	40
	13002	80	80	8	10	1	17	350	350	25	5	-	20	1.00	50
	13003	80	80	10	10	1	17	350	350	25	5	-	20	-	-
SOL	14001	110	110	8	10	1	17	350	350	25	5	-	20	1.50	50
	14002	125	125	10	10	1	17	350	350	35	5	-	20	1.50	50
SWA	3003	150	150	50	8	4	5	-	-	-	-	-	-	4.00	40
	3004	120	120	40	8	4	5	-	-	-	-	-	-	3.00	40
	3005	90	90	30	8	4	5	-	-	-	-	-	-	1.50	40
	3006	120	120	40	8	4	5	-	-	-	-	-	-	7.50	40
	3009	120	120	40	8	4	5	-	-	-	-	-	-	7.00	40
	3010	75	75	25	8	4	5	-	-	-	-	-	-	1.50	40
THN	6001	120	120	8	10	1	16	500	500	50	5	-	18	2.00	50
	6002	120	120	8	10	1	16	500	500	50	5	-	18	-	-
	6003	120	120	8	10	1	16	500	500	50	5	-	18	3.00	50
	6004	120	120	15	10	1	16	500	500	50	5	-	18	-	-
	6005	120	120	15	10	1	16	500	500	50	5	-	18	1.50	50
	6006	120	120	15	10	1	16	500	500	50	5	-	18	-	-
VAKO	2001	110	110	12	10	1	17	400	400	50	5	-	20	1.40	50
	2002	110	110	12	10	1	17	450	450	40	5	-	20	1.50	25
	2004	110	110	12	10	1	17	450	450	40	5	-	20	1.00	50
	2006	110	110	12	10	1	17	450	450	40	5	-	20	-	-

Table 14.10 OK Panel Block Search Parameters for Gold Grades

14.15 Localised Uniform Conditioning

LUC was undertaken for a target SMU block size of 5 mX x 5 mY x 2.5 mRL within the designated estimation domains for gold grade. The 'Panel' OK estimates for these variables constitute the precursor step to LUC.

The LUC estimation process is preceded by a standard Uniform Conditioning (UC) estimate of recoverable resources. The UC resource estimation process attempts to estimate the recoverable tonnage and grade based on the dimensions of the SMU, which is regarded as being practically achievable during actual mining. UC post-processing of the OK results was implemented for 5 mX x 5 mY x 2.5 mRL sized SMUs and incorporated an Information Effect correction based on an assumed ultimate grade control drill hole spacing of 10 mX x 10 mY x 1 mZ.

The Information Effect is a theoretical 'penalty' adjustment to the SMU grade tonnage distribution to account for anticipated smoothing and ore loss and dilution incurred when making mining selectivity decisions based on likely grade control spaced data - the impact of this correction is typically small and immaterial. The result of the UC process is an estimate, per Panel, of the recoverable metal, tonnes and grade at various grade cut-offs, assuming that SMU sized blocks are ultimately selected during mining. However, the reader should note that the UC process does not assign grade estimates to individual SMU's within a panel. The UC grade estimates at a cut-off of 0 g/t conform exactly to the OK estimates per Panel, and this property was used to validate the UC block model.

The UC process applies a Change of Support correction based on the composite sample distribution and variogram model, conditioned to the Panel grade estimate, to predict the likely grade tonnage distribution at the SMU selectivity.

LUC is a post-processing step that can be applied to UC estimates to provide indicative SMU scale estimates within each panel (Abzalov, 2006). The results of the LUC are consistent with the underlying UC estimate on a per-panel basis. LUC requires initially that local "ranking" SMU estimates are made by some chosen estimation method. In this case OK was used to generate the SMU ranks within each Panel.

The local SMU estimates within each panel are ranked in order of increasing grade. A quantile-quantile type matching of the SMU grade distribution, as determined by UC, with the ranked SMU's is then made. The OK 'ranking' SMU grades are finally replaced by the corresponding grades from the UC grade distribution. This yields the LUC SMU-scale grade estimates, which conform exactly to the SMU grade distribution predicted by the UC in panels. Gold grades within the barren dykes and the transported cover were reset to background values.

It should be clearly noted that the LUC estimates are typically based on relatively wide spaced data and are therefore of low confidence at the local scale. They should be considered to be indicative of the SMU grade variability that will eventuate when the deposit is grade controlled and mined. The individual SMU grade estimates are simply a probabilistic realisation of the grade and tonnage at this selectivity scale and provide a result which simplifies the mining studies, while providing a more realistic estimation of the grade-tonnage relationship that will eventuate over the life-of-mine when production commences. It would, however, be highly inadvisable to rely on the LUC estimates for short term mine planning purposes. LUC estimates are generally only applicable to open pit scenarios, such as those currently envisioned for Doropo, and are typically unsuitable for underground mining where opportunities for local block selectivity are limited.

14.16 Density Assignments

Density assignment is based on a total of 19,587 measurements at the updated prospects that were taken using the immersion method on drill core. Table 14.11 shows the dry density reading statistics per prospect.

The density variability is low, even when aggregated across weathering zones, and is observed to be multi-modal. When split by weathering intensity, the density distributions become unimodal and variability is reduced to even lower levels, sometimes markedly so. Since the bedrock lithology at Doropo is essentially granodiorite at all the prospects, no significant differentiation of density is observed when split by lithological sub-types.

As a result of the density analysis, it was decided to assign density on the basis of weathering intensity at all prospects, with the exception of the transported cover unit, which was treated separately because it has a somewhat elevated mean density; this is probably due to the fact that the cover is sometimes lateritised or contains a relatively high proportion of unweathered quartz detritus. The final density values were determined on the basis of this subdivision, with the mean density per category being assigned following a process of eliminating both low and high outlier values by inspection of density histograms (Table 14.12).

Prospect	N	Min	Max	Mean	Median	Std Dev	CoV
ATI	876	1.60	2.85	2.51	2.70	0.29	0.12
CHG	3,561	1.52	3.07	2.48	2.68	0.33	0.13
ENI	1,461	-0.21	3.12	2.30	2.38	0.47	0.20
HAN	1,811	1.16	2.84	2.67	2.70	0.13	0.05
KEK	1,491	1.54	6.96	2.56	2.70	0.29	0.11
KLG	4,277	1.19	4.38	2.61	2.67	0.21	0.08
NOK	2,303	1.50	3.79	2.63	2.69	0.18	0.07
SAN	182	1.85	2.73	2.52	2.59	0.19	0.08
SWA	3,165	1.50	5.89	2.41	2.68	0.42	0.17
VAKO	460	1.55	3.16	2.52	2.64	0.28	0.11
All Prospects	19,587	-0.21	6.96	2.53	2.68	0.32	0.13

Table 14.11 Basic Statistics for Dry Density (t/m^3) per Prospect

Zone	ATI	CHG-NOK	ENI	HAN	HND	KEK	KLG	NAR	SAN-VAKO	SOL	SWA	THN
Transported Cover	1.91	2.00	2.11	1.92	2.00	2.00	1.94	2.11	2.07	1.92	2.03	2.00
Completely Weathered	1.91	1.90	1.77	2.00	1.93	1.93	1.81	1.77	2.07	2.00	1.79	1.90
Highly Weathered	1.91	1.90	1.77	2.00	1.93	1.93	1.81	1.77	1.93	2.00	1.79	1.90
Moderately Weathered	2.22	2.16	2.15	2.39	2.20	2.20	2.33	2.15	2.39	2.39	2.16	2.16
Slightly Weathered	2.54	2.51	2.57	2.57	2.50	2.50	2.57	2.57	2.61	2.57	2.58	2.51
Fresh	2.72	2.70	2.72	2.70	2.70	2.70	2.67	2.72	2.66	2.70	2.70	2.70

Table 14.12 Dry Density (t/m^3) Assignments in the Block Models for the Updated Mineral Resource

14.17 Mining Depletion

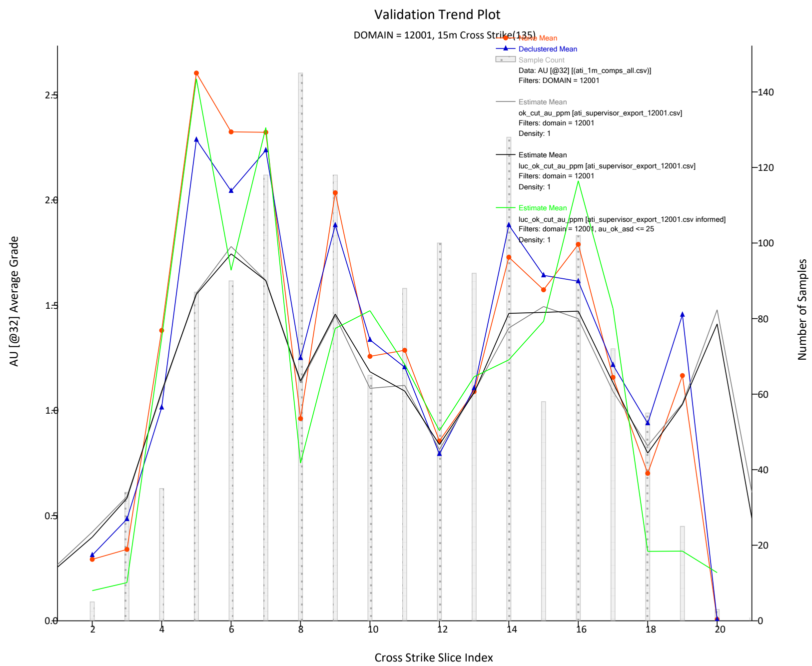
No mining has taken place at the Project to date and hence no depletion is recorded in the block model. Artisinal workings do occur in the area, but these have had, at most, a negligible impact on the modelled Mineral Resource.

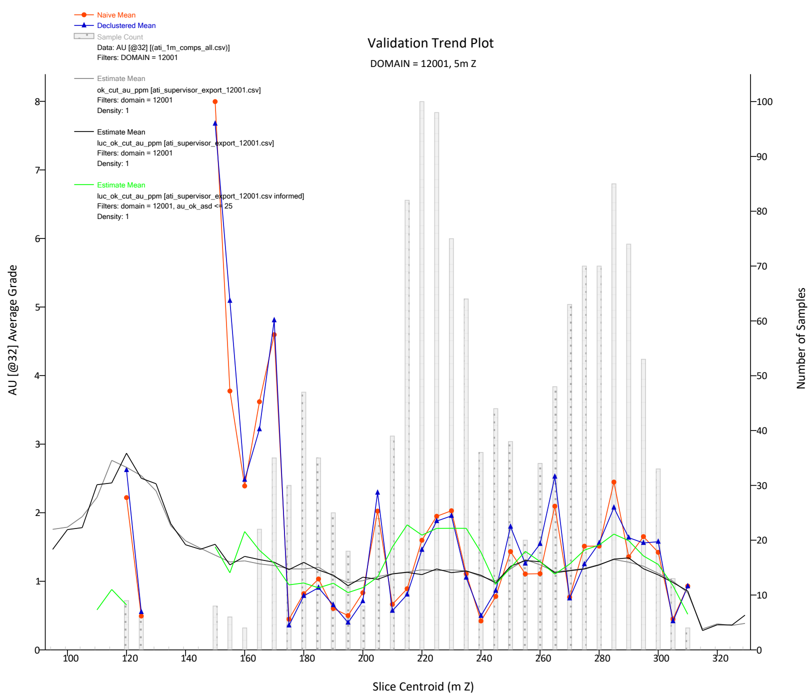
14.18 Mineral Resource Estimate Validation

Mineral Resource estimate validation has been undertaken by the following means:

- Global and local statistical comparisons of the mean grades of estimated blocks to the declustered mean grades of composites. The results of Inverse Distance Squared (ID2) and Moving Window Average (MWA) check estimates were also considered. The global mean of the estimated gold grades were found to match the informing composite data satisfactorily. Any larger variances could be readily explained by extrapolation of sample grades into relatively poorly informed volumes, such as around the periphery of the mineralised domains. Local estimated informed cells mean was found to match the MWA estimates satisfactorily as well;
- Using swath plots to compare estimated block grades to the informing composite grades (see examples in Figure 14.37 and Figure 14.20);
- By visual validation, both in cross-section and 3D isometric views, of the estimated block grades overlaid on drill assay data (Figure 14.38 and Figure 14.21).

The block estimates were considered to be an accurate reflection of the input sample data.





Composites naïve (red) and declustered (blue) mean, estimated OK (grey), LUC (black) and Informed Cells LUC (green)

Figure 14.37 Example Swath Plots for Gold Grade in Domain 12001 at ATI

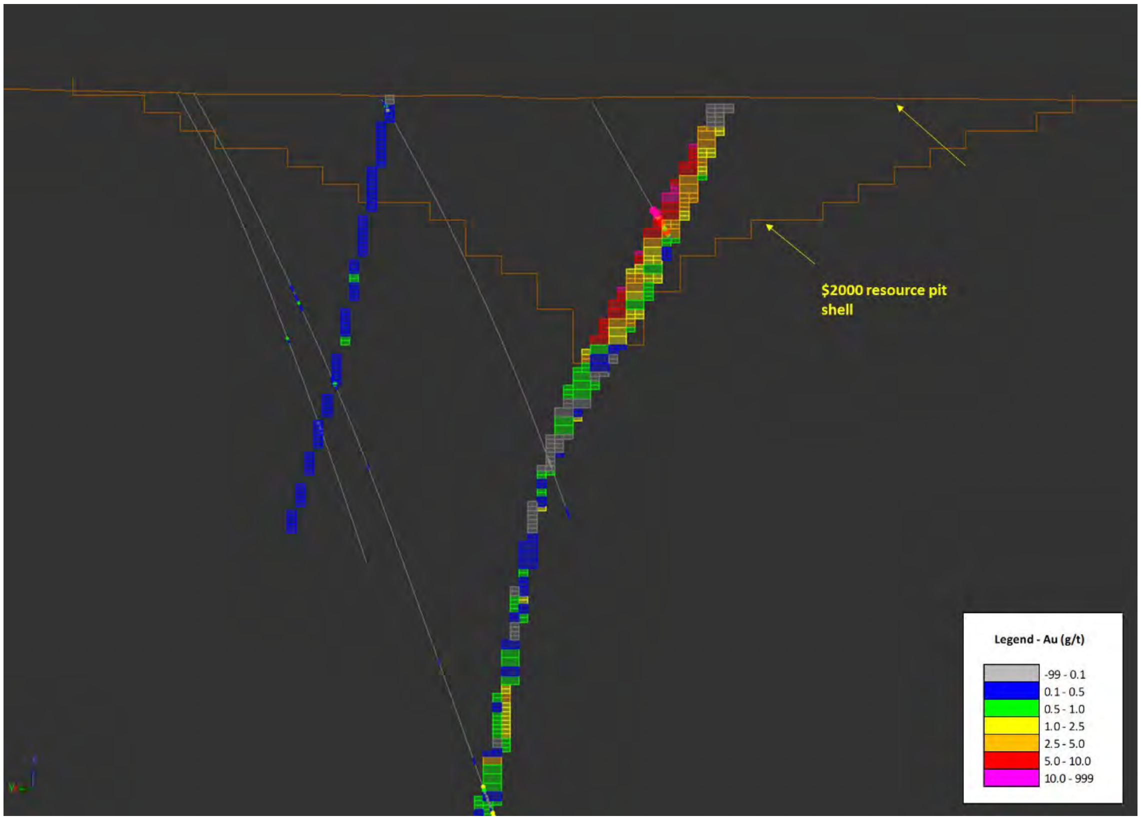


Figure 14.38 Cross-section View Looking SE at ATI. Block Grades and Drill Assay Grades

14.19 Mineral Resource Estimate Classification

14.19.1 Reasonable Prospects for Eventual Economic Extraction

The Mineral Resource is considered to have Reasonable Prospects for Eventual Economic Extraction (RPEEE) on the following basis:

- The deposit is located in a mining jurisdiction with multiple operating gold mines, with no known impediments to land access or tenure status;
- The volume, orientation and grade of the Resource is amenable to mining extraction via traditional open pit methods;
- Current metallurgical recovery based on available metallurgical test work was used in a pit optimisation to generate the resource pit shells.

The Mineral Resource is reported within a series of pit shells using a gold metal price assumption of USD2,000/oz Au. The gold spot price at the time of public release of this MRE update (October 2023) was ~USD1,900/oz Au.

Cube believes this is a reasonable approach, considering the potential mine life and considerations for reporting Mineral Resources in accordance with the CIM Definitions and Standards (CIM Council, 2014).

14.19.2 Classification Criteria

The Doropo Mineral Resource as at 31 October 2023 is intended for public reporting and represents an update to the previous public release dated 29 November 2022. The Mineral Resource has been classified and reported in accordance with the CIM Definition Standards (CIM Council, 2014).

Cube considers the following points to be material in the classification of the Doropo Mineral Resource:

- Database Integrity - The QAQC data have demonstrated that the sampling precision and accuracy is within an acceptable range for the estimation of a Mineral Resource;
- Geological Interpretation - The current geological interpretations including mineralisation, structure, weathering and lithology are considered to be of a high standard and that the geometry of and factors controlling the mineralisation are sufficiently well understood to classify Mineral Resources;
- Drill Hole Spacing and Sampling Density - Mineralisation interpretations and attribute interpolations are based on a variable drill hole spacing. Where infill drilling has been undertaken as part of the DFS phase, the nominal drill spacing is 25 m x 25 m. Elsewhere the spacing widens to a nominal 50 m x 50 m to 100 m x 100 m. Localised grade control drilling at 10 m x 10 m have been performed as part of the DFS;
- Estimation Method - Cube has undertaken LUC interpolation of gold grades, is considered appropriate per the statistical and geostatistical characteristics of the gold mineralisation and the anticipated open pit mining methods;
- Interpolation Quality - OK interpolation quality parameters, which rely primarily on data spacing and the continuity model (i.e. the variogram), have been considered in the classification of Mineral Resources.

It is Cube's conclusion that the Doropo mineralisation domains are sufficiently drilled to allow classification. As with any non-rigidly defined classification there will always be some blocks within categories that depart from defined criteria. It is Cube's view that the final outcome must reflect a practical combination of data quality, geological knowledge, and interpolation quality parameters that may be more numerical in nature. This approach to classification aims to avoid creating a complex, numerically derived 'mosaic'. Cube has considered all criteria and has classified the resource as Measured, Indicated or Inferred.

Indicated Mineral Resources are defined where the nominal drill spacing is 25-30 m or tighter. Measured Mineral Resources are defined where there is grade control drilling (10 m x 10 m spacing).

The following factors played a major role in the definition of Measured Mineral Resources:

- Gold grade variogram ranges always exceed 25 m and often exceed 50 m;
- Interpolation quality is observed to drop off as one moves out of the zone of 25-30 m drilling towards areas drilled out at 50 m;
- Thickness, grade tenor and variability between mineralisation with a drill spacing of 25-30m and 10 m x 10 m shows little variation in aggregate across the various deposits (Table 14.13) which confirm the confidence in the grade and geological continuity;
- Data quality and integrity, as well as the confidence in the geological and mineralisation models, especially at a 10 m x 10 m drill spacing, is considered by the Qualified Person to be sufficient to satisfy the codified criteria for a Measured classification;
- The Qualified Persons are satisfied that a 10 m x 10 m drill spacing is sufficient to allow for the estimation of deposit physical characteristics with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Prospects	2022			2023			Rel. Diff. %		
	Tonnes	Au	Oz	Tonnes	Au	Oz	Tonnes	Au	Oz
CHG	187,330	1.05	6,334	187,581	1.09	6,556	0%	3%	4%
CHS	252,246	1.46	11,818	238,842	1.06	8,142	-5%	-27%	-31%
HAN	107,653	1.61	5,576	114,191	2.02	7,400	6%	25%	33%
KEK	274,570	1.09	9,657	207,886	0.80	5,318	-24%	-27%	-45%
KLG	216,176	1.00	6,966	212,014	1.10	7,515	-2%	10%	8%
NOK	350,136	2.27	25,521	343,512	2.48	27,379	-2%	9%	7%
SWA	252,674	1.67	13,559	274,418	1.85	16,338	9%	11%	20%
Total	1,640,786	1.51	79,430	1,578,443	1.55	78,647	-4%	3%	-1%

Table 14.13 Comparison Between 2022 MRE (drilling at 25 m x 25 m) and 2023 MRE (Drilling at 10 m x 10 m) Within the Grade Control Drilled Areas

The following factors played a major role in the definition of Indicated Mineral Resources:

- Gold grade variogram ranges always exceed 25 m and often exceed 50 m;
- Interpolation quality is observed to drop off as one moves out of the zone of 25-30 m drilling towards areas drilled out at wider spacing of 50 m x 50 m;
- Data quality and integrity, as well as the confidence in the geological and mineralisation models, especially at a 25-30 m drill spacing, is considered by the Qualified Person to be sufficient to satisfy the codified criteria for an Indicated classification;
- The Qualified Persons are satisfied that a 25-30 m drill spacing is sufficient to allow for the estimation of deposit physical characteristics with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Areas drilled at a wider spacing than 25-30 m and less than or equal to 50 m x 50 m were classified as Inferred Mineral Resources inside the mineralisation domain envelopes, except in the transported cover and around the periphery of the drilling at a distance greater than 50 m beyond the last drill hole, where no Mineral Resource has been defined and classified.

Areas that have been "grade control" drilled at 10 m x 10 m are classified as Measured Mineral Resources inside the mineralisation domain envelopes, except in the transported cover. This applies at the following prospects: Chegue Main and South, Han, Kekeda, Kilosegui, Nokpa and Souwa.

Prospects Hinda, Hinda South, Nare, Solo, and Tchouahinin are classified as Inferred Mineral Resources only in their totality as the drill spacing exceeds 25-30 m.

Only the volume contained within the mineralisation/estimation domain were considered eligible for classification as Mineral Resources.

The Mineral Resource estimate appropriately reflects the Qualified Persons' view of the deposit (see Figure 14.39 to Figure 14.56).

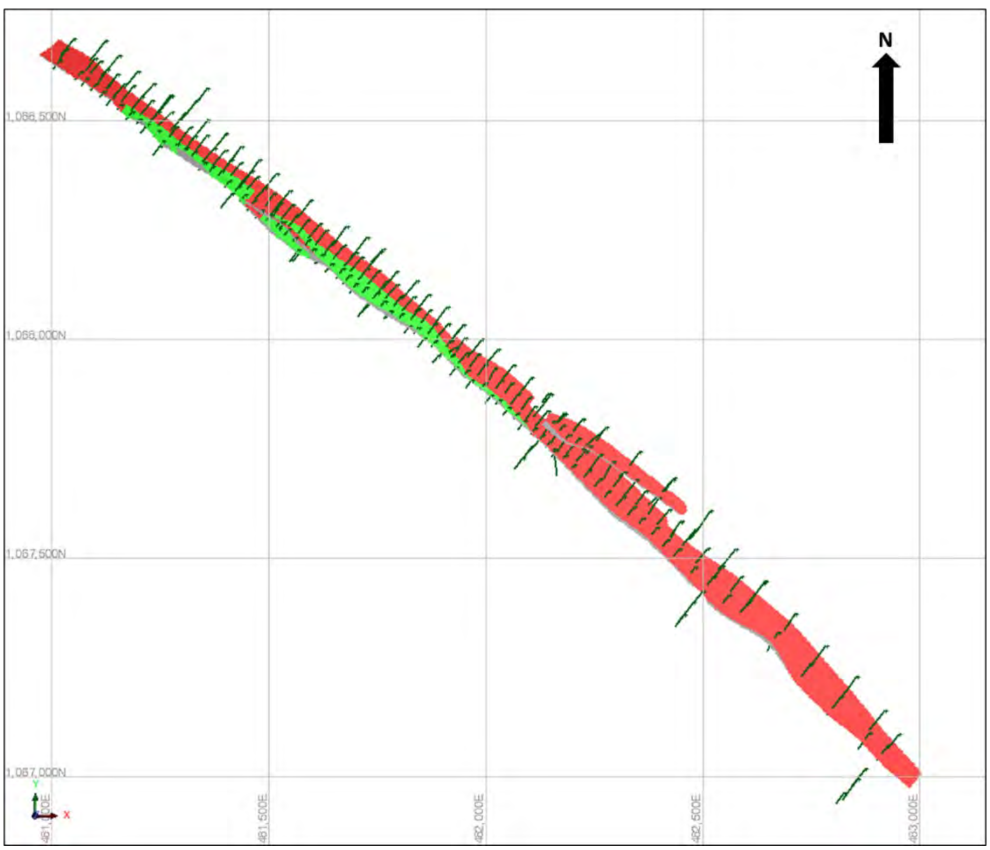


Figure 14.39 Plan View of the Mineral Resource Classification at ATI

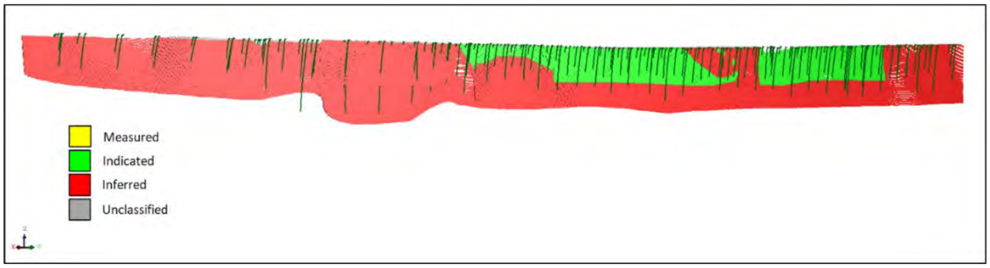


Figure 14.40 Isometric View Looking SW of the Mineral Resource Classification at ATI

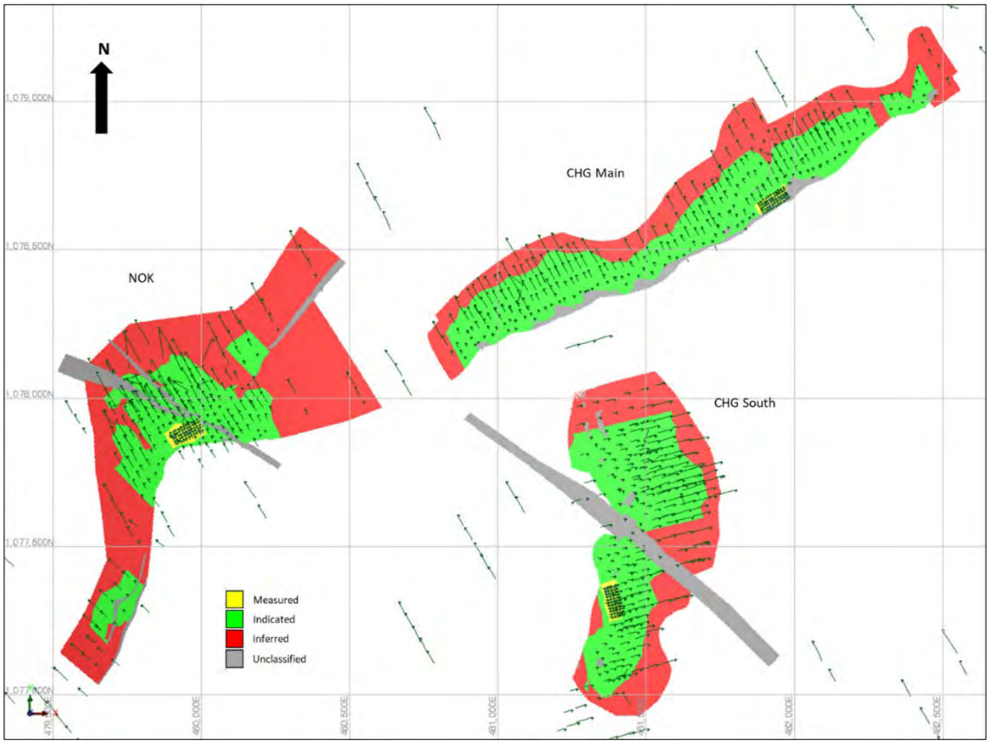


Figure 14.41 Plan View of the Mineral Resource Classification at CHG-NOK

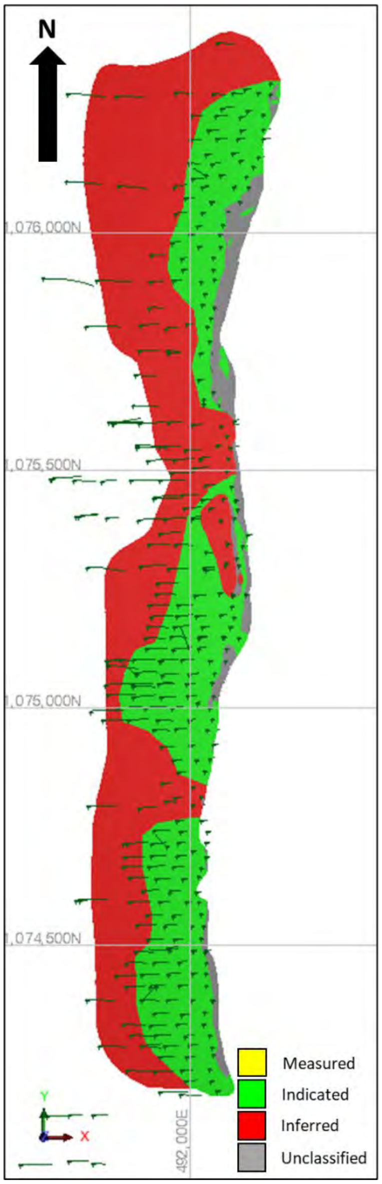


Figure 14.42 Plan View of the Mineral Resource Classification at ENI

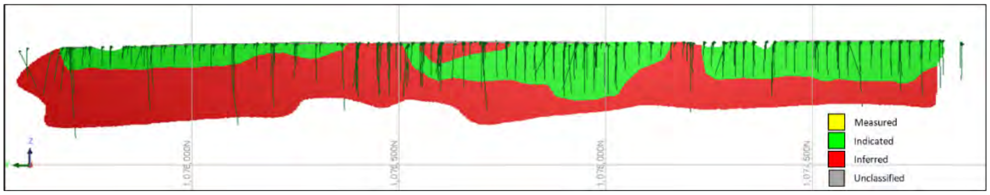


Figure 14.43 Isometric View Looking E of the Mineral Resource Classification at ENI

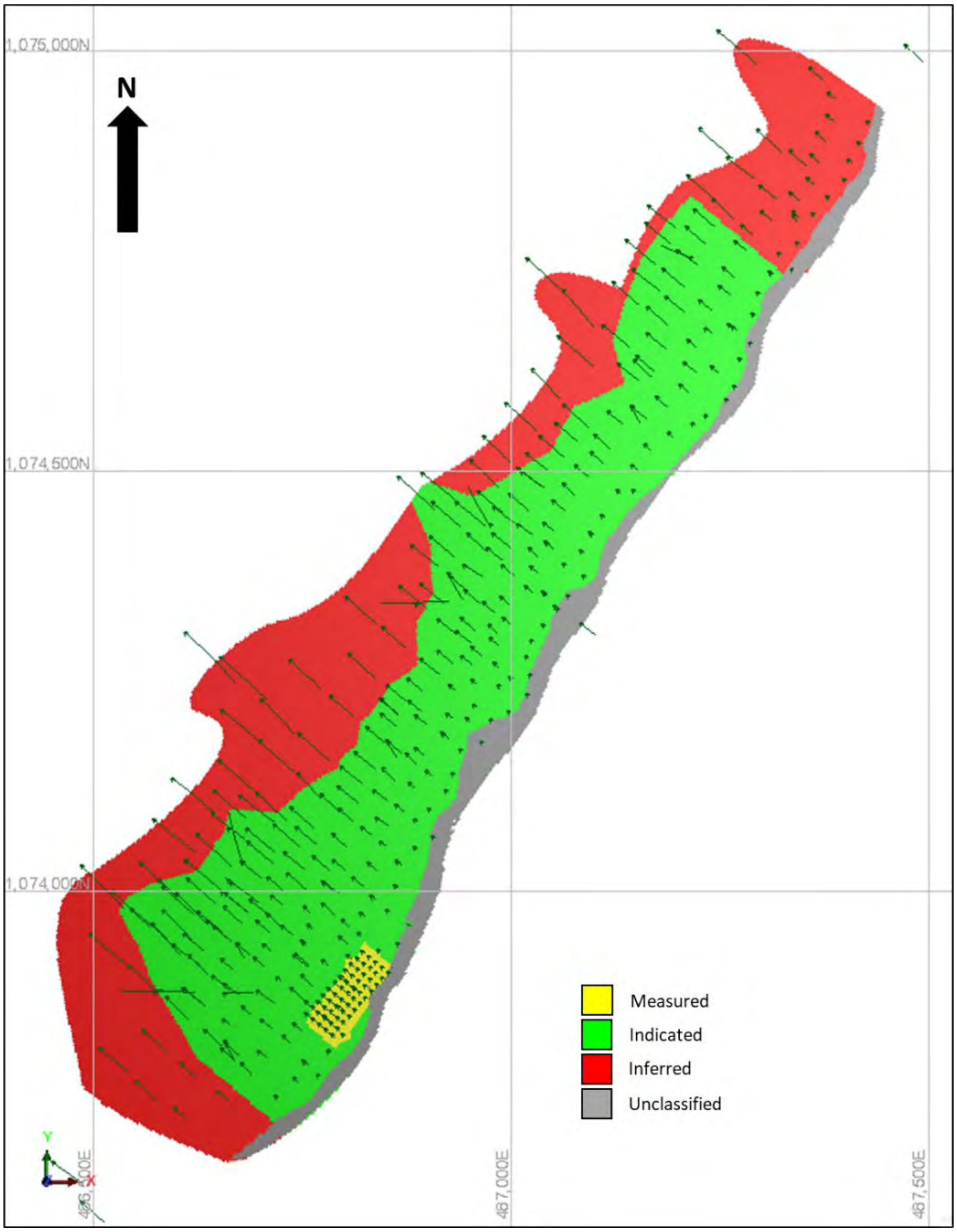


Figure 14.44 Plan View of the Mineral Resource Classification at HAN

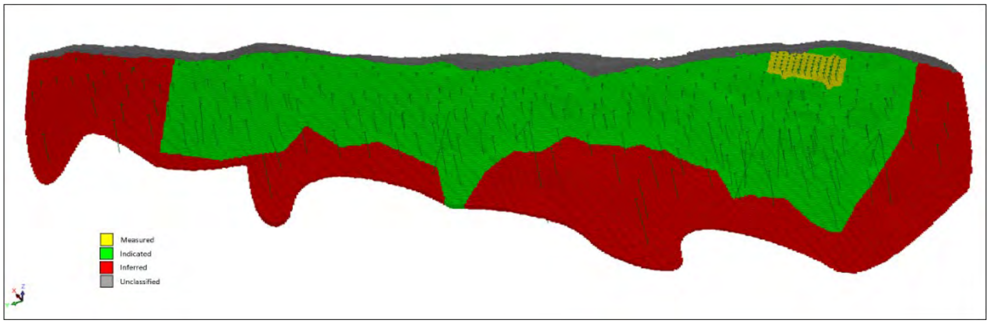


Figure 14.45 Isometric View Looking SE of the Mineral Resource Classification at HAN

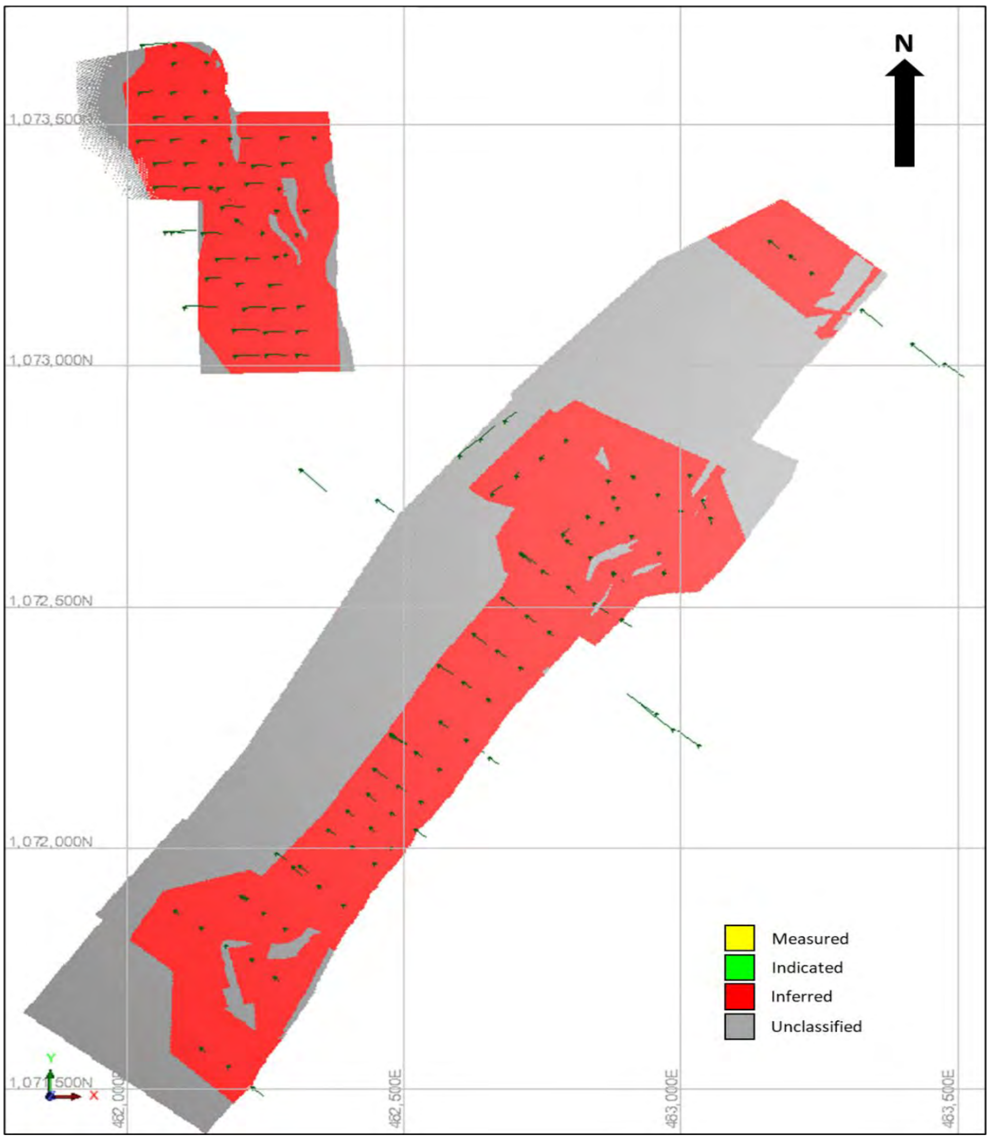


Figure 14.46 Plan View of the Mineral Resource Classification at HND

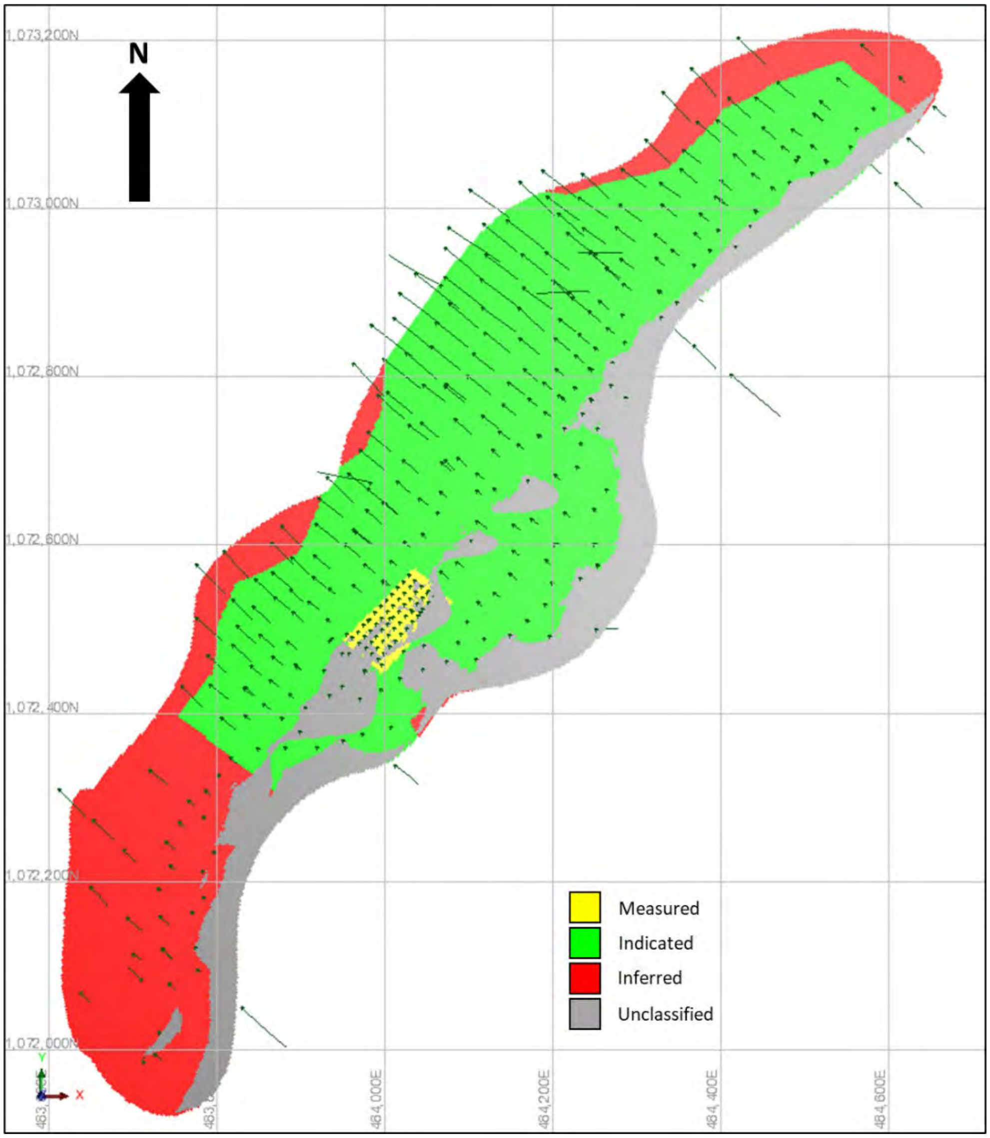


Figure 14.47 Plan View of the Mineral Resource Classification at KEK

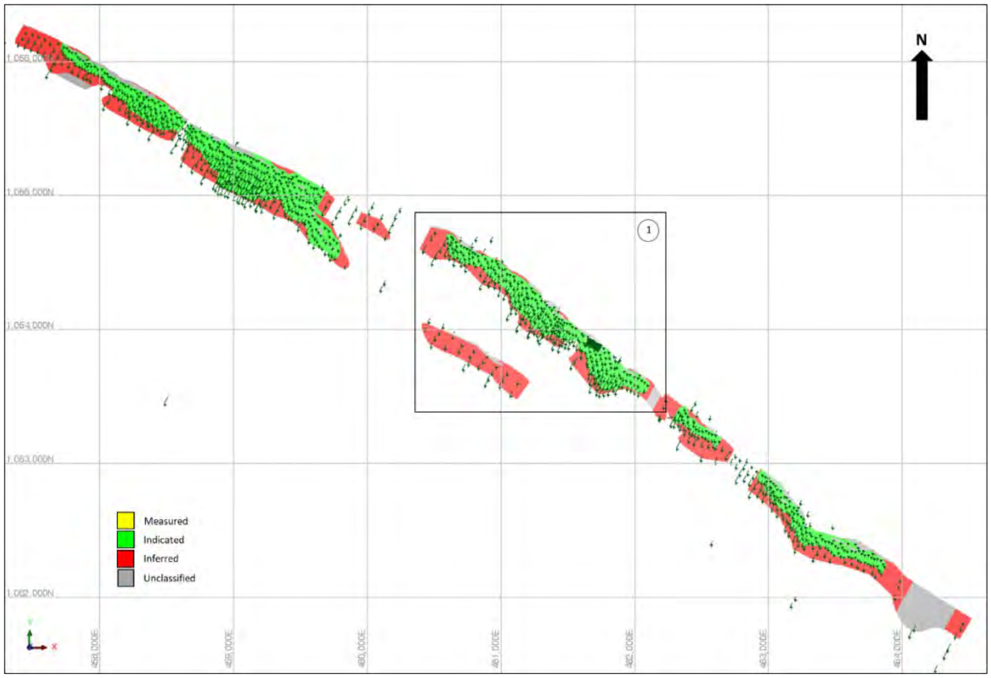


Figure 14.48 Plan View of the Mineral Resource Classification at KILO

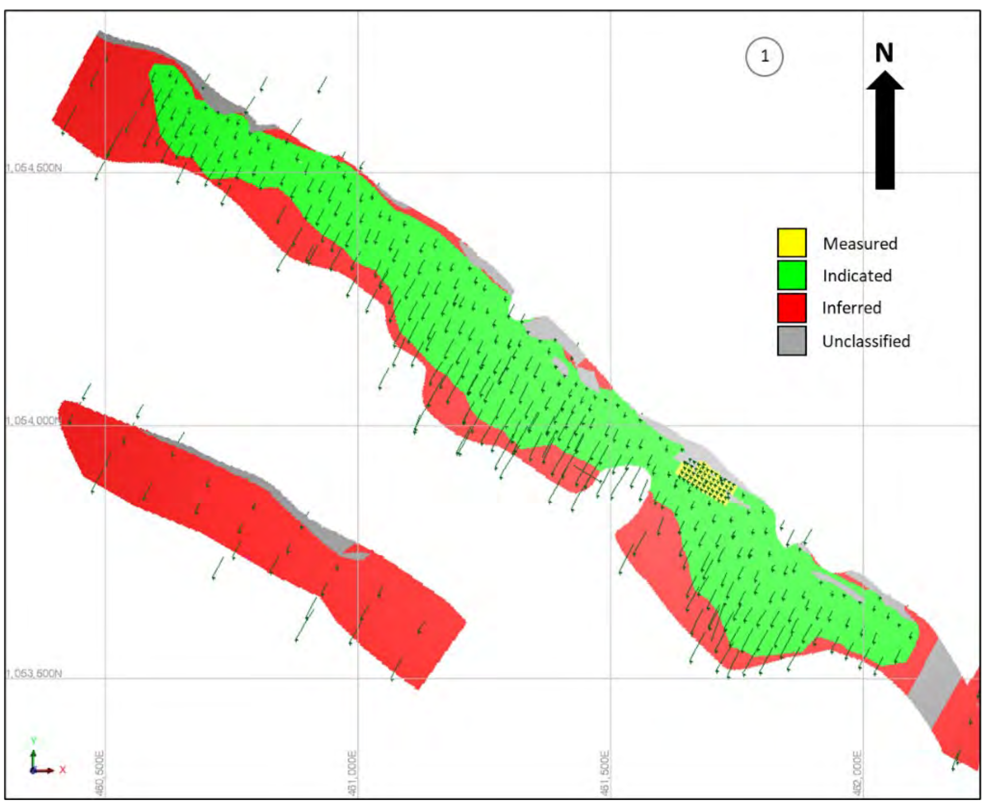


Figure 14.49 Zoom View of the Mineral Resource Classification at KILO

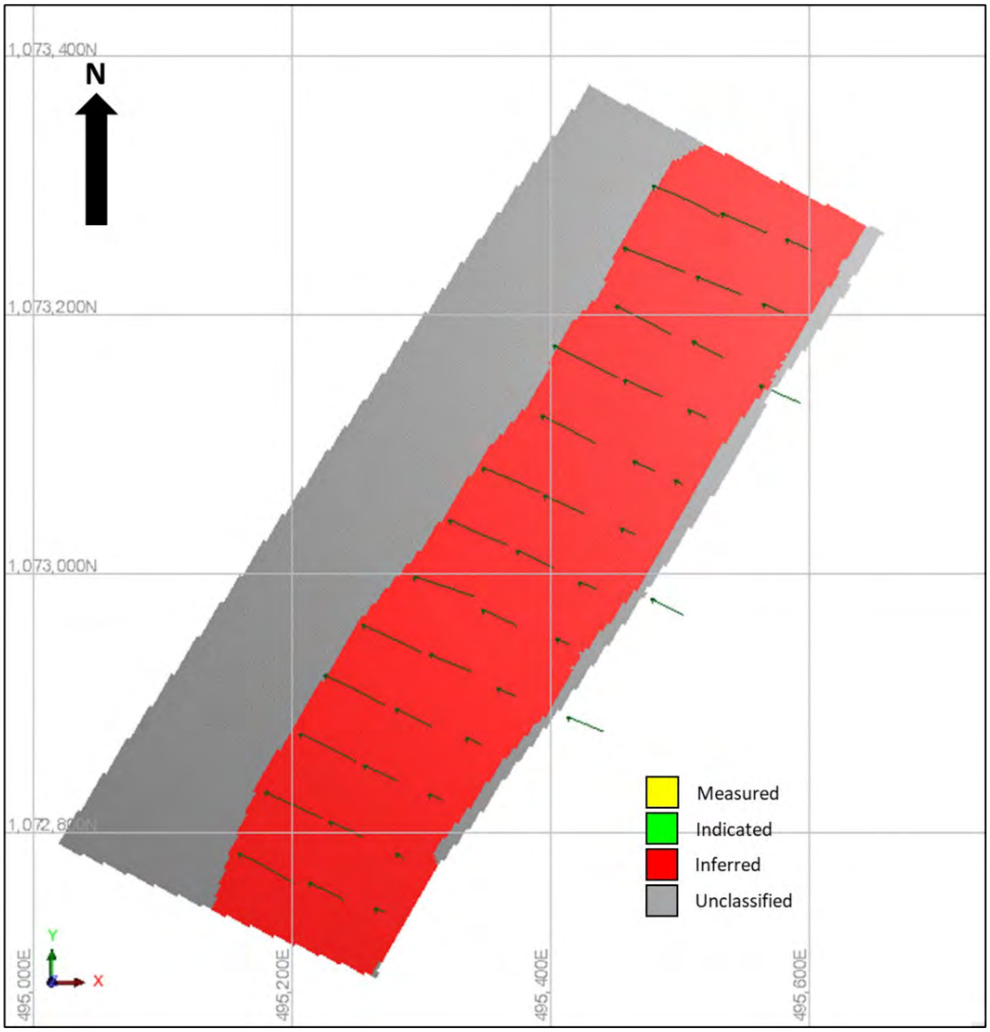


Figure 14.50 Plan View of the Mineral Resource Classification at NAR

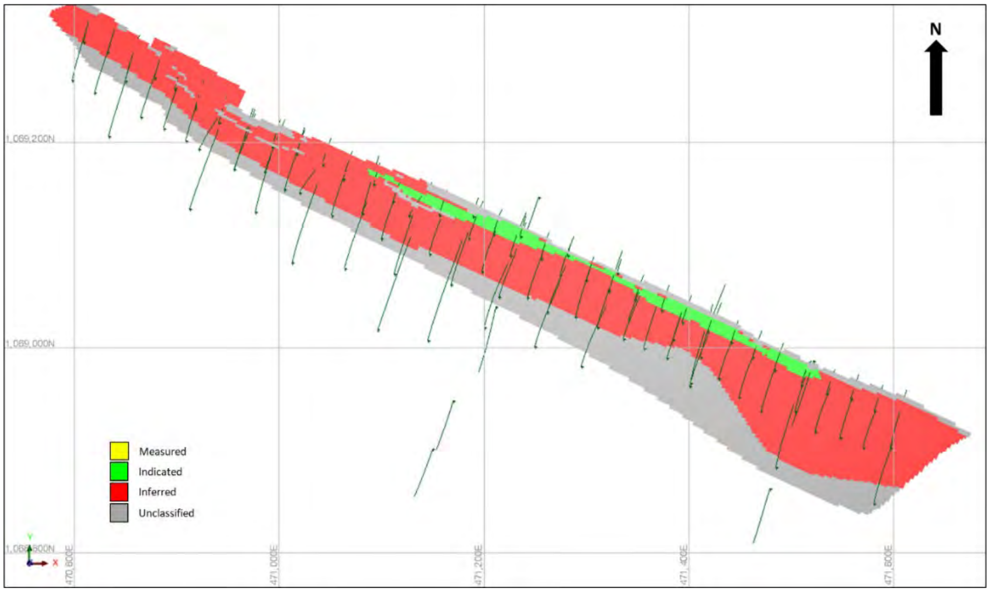


Figure 14.51 Plan View of the Mineral Resource Classification at SAN

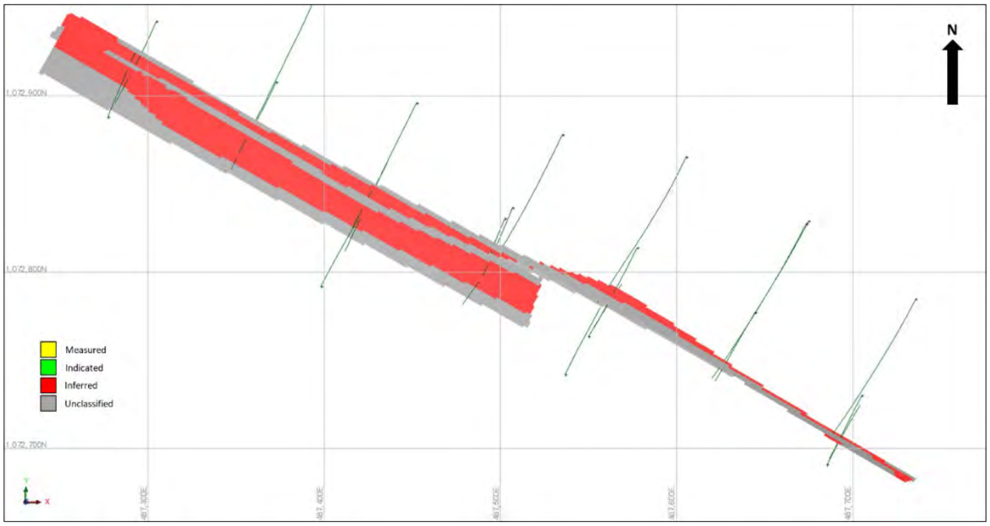


Figure 14.52 Plan View of the Mineral Resource Classification at SOL

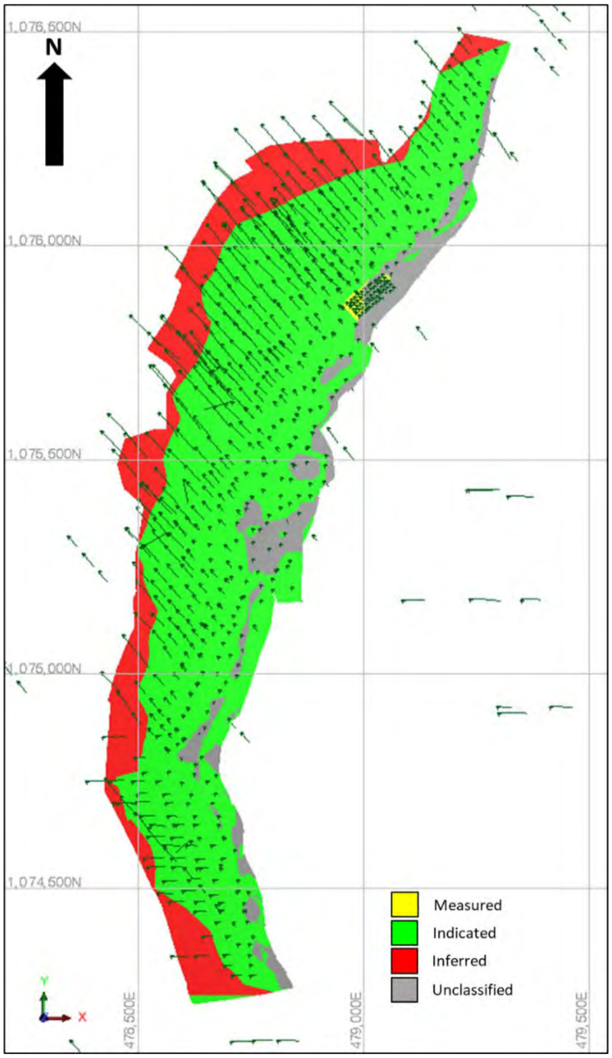


Figure 14.53 Plan View of the Mineral Resource Classification at SWA

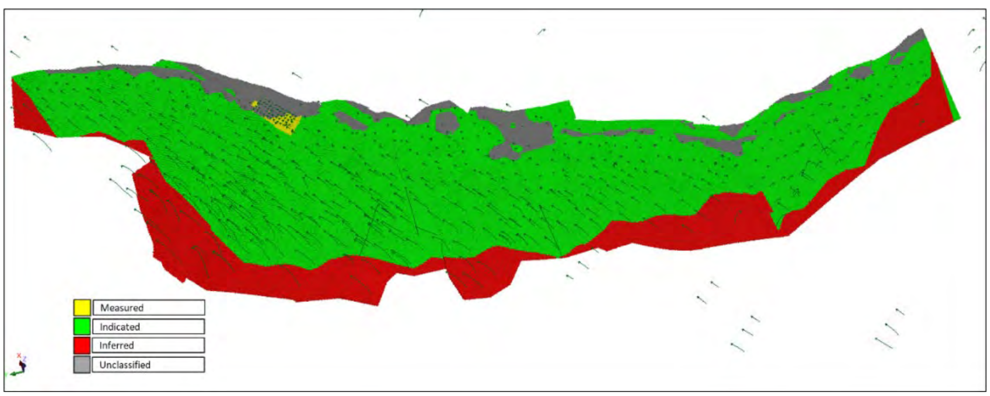


Figure 14.54 Isometric View Looking ESE of the Mineral Resource Classification at SWA

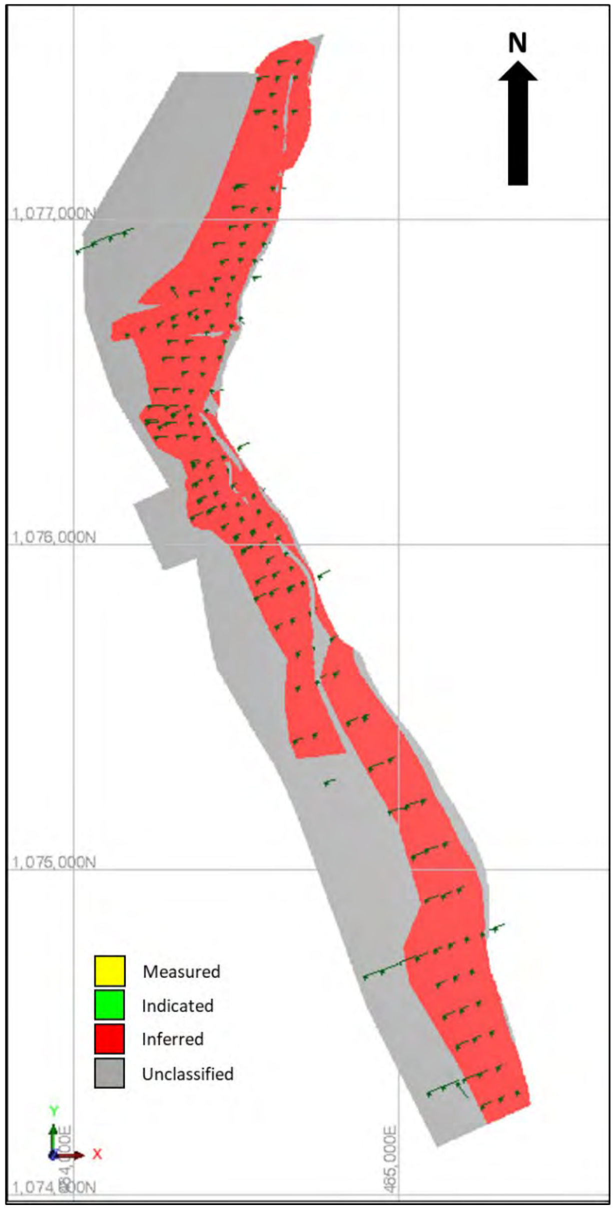


Figure 14.55 Plan View of the Mineral Resource Classification at THN

Figure 14.56 Plan View of the Mineral Resource Classification at VAKO

14.20 Mineral Resource Statement

14.20.1 Reporting

As a designated reporting issuer in Canada, Centamin is subject to certain Canadian disclosure requirements and standards, including the requirements of NI 43-101. The Doropo Gold Project is a material mineral project for the purposes of NI 43-101. The confidence categories are assigned according to the CIM Definition Standards - for Mineral Resources and Mineral Reserves May 2014.

14.20.2 Pit Optimisation Parameters

The pit optimisations used to constrain the Mineral Resource were generated by Orelogy Mine Consultants. Key input parameters used to produce the reporting pit shells are as follows:

- All models have been re-blocked to 10 mX x 10 mY x 5 mRL;
- Gold price assumption of USD2,000 per troy ounce;
- Overall pit wall slope angles used are (in the range of):
  - 24° in oxide;
  - 28° in transitional;
  - 48° in fresh;
- Mining Recovery of 92% (8% ore loss);
- Mining Dilution of 14%;
- Process Recovery:
  - Oxide: 93.5%;

- Transitional: 94.2%;
- Fresh: 92.5%;

- Discount rate: 5%;
- Mill Throughput: 5.2 Mt/a (for oxide and trans material);
- Mill Throughput: 4.0 Mt/a (for fresh rock).

14.20.3 Results

The updated Doropo MRE, as constrained to the optimised pit shells, is summarised in Table 14.14 to Table 14.16. A more detailed breakdown of the Mineral Resources by oxidation is shown in Table 14.17 through Table 14.48. Grade-tonnage curves per updated prospect are presented in Figure 14.57 through Figure 14.72. The tonnage and grade curves show a relatively slow change in both these parameters for all prospects except CHG South and KEK, which display steeper transitions through the range of cut-offs considered.

Measured Mineral Resources (0.3 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Attire	-	-	-
Chegue Main	0.19	1.09	0.007
Chegue South	0.21	1.12	0.008
Enioda	-	-	-
Han	0.11	2.06	0.007
Hinda	-	-	-
Hinda South	-	-	-
Kekeda	0.19	0.82	0.005
Kilosegui	0.21	1.11	0.007
Nare	-	-	-
Nokpa	0.34	2.49	0.027
Sanboyoro	-	-	-
Solo	-	-	-
Souwa	0.26	1.95	0.016
Tchouahinin	-	-	-
Vako	-	-	-
Total	1.51	1.60	0.077

Measured Mineral Resources (0.5 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Attire	-	-	-
Chegue Main	0.17	1.13	0.006
Chegue South	0.16	1.34	0.007
Enioda	-	-	-
Han	0.10	2.19	0.007

Measured Mineral Resources (0.5 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Hinda	-	-	-
Hinda South	-	-	-
Kekeda	0.09	1.29	0.004
Kilosegui	0.16	1.30	0.007
Nare	-	-	-
Nokpa	0.29	2.83	0.027
Sanboyoro	-	-	-
Solo	-	-	-
Souwa	0.22	2.23	0.016
Tchouahinin	-	-	-
Vako	-	-	-
Total	1.21	1.89	0.074

Notes:

- *Some numerical differences may occur due to rounding;*
- *RPEEE is defined by optimised pit shells based on a gold price of USD2,000/oz;*
- *Reported at a gold grade cut-off of 0.3 and 0.5 g/t Au (as indicated);*
- *Includes drill holes up to and including 27 August 2023;*
- *Includes Mineral Reserves.*

Table 14.14 Doropo Updated Measured Mineral Resource Estimate (CIM Definition Standards), 31 October 2023

Indicated Mineral Resources (0.3 g/t Au COG)			
Prospect	Mt	Au g/t	Au Koz
Attire	0.42	1.86	0.025
Chegue Main	6.52	1.02	0.213
Chegue South	4.85	1.11	0.174
Enioda	3.31	1.31	0.139
Han	4.33	1.82	0.253
Hinda	-	-	-
Hinda South	-	-	-
Kekeda	3.99	1.04	0.133
Kilosegui	29.66	1.14	1.090
Nare	-	-	-
Nokpa	3.80	1.74	0.213
Sanboyoro	0.01	1.33	0.001
Solo	-	-	-
Souwa	15.98	1.42	0.729
Tchouahinin	-	-	-
Vako	2.48	0.73	0.058
Total	75.34	1.25	3.027

Indicated Mineral Resources (0.5 g/t Au COG)			
Prospect	Mt	Au g/t	Au Koz
Attire	0.36	2.08	0.024
Chegue Main	5.20	1.17	0.196
Chegue South	3.37	1.43	0.155
Enioda	2.93	1.43	0.135
Han	3.80	2.02	0.246
Hinda	-	-	-
Hinda South	-	-	-
Kekeda	3.04	1.23	0.120
Kilosegui	24.21	1.31	1.018
Nare	-	-	-
Nokpa	3.12	2.03	0.204
Sanboyoro	0.01	1.41	0.001
Solo	-	-	-
Souwa	12.15	1.74	0.679
Tchouahinin	-	-	-
Vako	1.78	0.85	0.049
Total	59.98	1.47	2.826

Notes:

- *Some numerical differences may occur due to rounding;*
- *RPEEE is defined by optimised pit shells based on a gold price of USD2,000/oz;*
- *Reported at a gold grade cut-off of 0.3 and 0.5 g/t Au (as indicated);*
- *Includes drill holes up to and including 27 August 2023;*
- *Includes Mineral Reserves.*

Table 14.15 Doropo Updated Indicated Mineral Resource Estimate (CIM Definition Standards), 31 October 2023

Inferred Mineral Resources (0.3 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Attire	0.71	2.43	0.055
Chegue Main	0.64	1.12	0.023
Chegue South	0.12	0.74	0.003
Enioda	1.20	1.12	0.043
Han	0.25	1.79	0.014
Hinda	0.15	1.54	0.007
Hinda South	0.84	0.78	0.021
Kekeda	0.13	0.87	0.004
Kilosegui	1.77	1.13	0.064
Nare	0.05	0.95	0.002
Nokpa	0.05	0.65	0.001
Sanboyoro	0.11	1.61	0.006
Solo	0.16	2.43	0.013

Inferred Mineral Resources (0.3 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Souwa	-	-	-
Tchouahinin	1.06	0.96	0.033
Vako	0.12	0.71	0.003
Total	7.37	1.23	0.292

Inferred Mineral Resources (0.5 g/t Au COG)			
Prospect	Mt	Au g/t	Au Moz
Attire	0.54	3.08	0.053
Chegue Main	0.51	1.31	0.021
Chegue South	0.07	0.97	0.002
Enioda	1.03	1.24	0.041
Han	0.24	1.87	0.014
Hinda	0.12	1.78	0.007
Hinda South	0.63	0.90	0.018
Kekeda	0.09	1.08	0.003
Kilosegui	1.48	1.27	0.060
Nare	0.05	0.95	0.002
Nokpa	0.03	0.76	0.001
Sanboyoro	0.09	1.99	0.006
Solo	0.14	2.75	0.012
Souwa	-	-	-
Tchouahinin	0.79	1.15	0.029
Vako	0.09	0.81	0.002
Total	5.90	1.44	0.272

Notes:

- *Some numerical differences may occur due to rounding;*
- *RPEEE is defined by optimised pit shells based on a gold price of USD2,000/oz;*
- *Reported at a gold grade cut-off of 0.3 and 0.5 g/t Au (as indicated);*
- *Includes drill holes up to and including 27 August 2023;*
- *No Mineral Reserves included.*

Table 14.16 Doropo Updated Inferred Mineral Resource Estimate (CIM Definition Standards), 31 October 2023

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	33	1.51	2
	Transitional	269	1.71	15
	Fresh	117	2.32	9
	Total Indicated	419	1.86	25
Inferred	Oxide	54	2.28	4
	Transitional	244	1.58	12
	Fresh	411	2.94	39
	Total Inferred	708	2.43	55

Table 14.17 Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	28	1.72	2
	Transitional	227	1.95	14
	Fresh	108	2.46	9
	Total Indicated	364	2.08	24
Inferred	Oxide	40	2.94	4
	Transitional	165	2.15	11
	Fresh	330	3.57	38
	Total Inferred	535	3.08	53

Table 14.18 Breakdown of the ATI Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	187	1.09	7
	Transitional	0	0.87	0
	Fresh	-	-	-
	Total Measured	188	1.09	7
Indicated	Oxide	2,228	0.86	62
	Transitional	559	1.00	18
	Fresh	3,728	1.11	133
	Total Indicated	6,515	1.02	213
Inferred	Oxide	121	0.79	3
	Transitional	35	1.38	2
	Fresh	488	1.18	18
	Total Inferred	644	1.12	23

Table 14.19 Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	175	1.13	6
	Transitional	0	0.87	0
	Fresh	-	-	-
	Total Measured	175	1.13	6
Indicated	Oxide	1,641	1.02	54
	Transitional	430	1.17	16
	Fresh	3,134	1.25	126
	Total Indicated	5,204	1.17	196
Inferred	Oxide	88	0.93	3
	Transitional	30	1.52	1
	Fresh	389	1.38	17
	Total Inferred	507	1.31	21

Table 14.20 Breakdown of the CHG Main Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	65	1.12	2
	Transitional	36	1.58	2
	Fresh	113	0.98	4
	Total Measured	214	1.12	8
Indicated	Oxide	855	0.84	23
	Transitional	320	0.86	9
	Fresh	3,672	1.20	142
	Total Indicated	4,847	1.11	174
Inferred	Oxide	101	0.63	2
	Transitional	2	0.72	0
	Fresh	16	1.41	1
	Total Inferred	119	0.74	3

Table 14.21 Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	46	1.40	2
	Transitional	32	1.71	2
	Fresh	86	1.16	3
	Total Measured	164	1.34	7
Indicated	Oxide	551	1.09	19
	Transitional	195	1.15	7
	Fresh	2,626	1.52	128
	Total Indicated	3,373	1.43	155
Inferred	Oxide	58	0.81	2
	Transitional	1	0.90	0
	Fresh	13	1.65	1
	Total Inferred	72	0.97	2

Table 14.22 Breakdown of the CHG South Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	805	1.26	33
	Transitional	1,514	1.29	63
	Fresh	990	1.39	44
	Total Indicated	3,309	1.31	139
Inferred	Oxide	821	1.22	32
	Transitional	119	0.67	3
	Fresh	263	0.99	8
	Total Inferred	1,202	1.12	43

Table 14.23 Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	672	1.43	31
	Transitional	1,346	1.40	60
	Fresh	908	1.48	43
	Total Indicated	2,927	1.43	135
Inferred	Oxide	76	0.82	2
	Transitional	212	1.13	8
	Fresh	742	1.31	31
	Total Inferred	1,030	1.24	41

Table 14.24 Breakdown of the ENI Mineral Resource by Oxidation State. Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	Mt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	49	1.83	3
	Fresh	62	2.23	4
	Total Indicated	110	2.06	7
Indicated	Oxide	3	1.56	0
	Transitional	398	1.91	24
	Fresh	3,927	1.81	229
	Total Indicated	4,327	1.82	253
Inferred	Oxide	0	1.61	0
	Transitional	105	2.06	7
	Fresh	145	1.59	7
	Total Inferred	250	1.79	14

Table 14.25 Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	45	1.95	3
	Fresh	57	2.37	4
	Total Measured	102	2.19	7
Indicated	Oxide	2	1.98	0
	Transitional	345	2.14	24
	Fresh	3,456	2.00	223
	Total Indicated	3,803	2.02	246
Inferred	Oxide	0	1.61	0
	Transitional	100	2.13	7
	Fresh	135	1.68	7
	Total Inferred	235	1.87	14

Table 14.26 Breakdown of the HAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	109	1.35	5
	Transitional	39	2.06	3
	Fresh	-	-	-
	Total Inferred	149	1.54	7

Table 14.27 Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	85	1.62	4
	Transitional	38	2.12	3
	Fresh	-	-	-
	Total Inferred	123	1.78	7

Table 14.28 Breakdown of the HND Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	251	0.75	6
	Transitional	503	0.80	13
	Fresh	86	0.81	2
	Total Inferred	840	0.78	21

Table 14.29 Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	177	0.89	5
	Transitional	382	0.92	11
	Fresh	74	0.87	2
	Total Inferred	634	0.90	18

Table 14.30 Breakdown of the HNDS Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	4	1.10	0
	Transitional	56	0.73	1
	Fresh	128	0.86	4
	Total Measured	188	0.82	5
Indicated	Oxide	159	0.76	4
	Transitional	1,811	0.96	56
	Fresh	2,016	1.12	73
	Total Indicated	3,986	1.04	133
Inferred	Oxide	74	0.92	2
	Transitional	58	0.81	2
	Fresh	-	-	-
	Total Inferred	132	0.87	4

Table 14.31 Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	3	1.30	0
	Transitional	26	1.12	1
	Fresh	63	1.36	3
	Total Measured	92	1.29	4
Indicated	Oxide	92	1.02	3
	Transitional	1,321	1.17	50
	Fresh	1,624	1.29	67
	Total Indicated	3,036	1.23	120
Inferred	Oxide	49	1.18	2
	Transitional	41	0.97	1
	Fresh	-	-	-
	Total Inferred	90	1.08	3

Table 14.32 Breakdown of the KEK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	1	0.74	0
	Transitional	118	1.04	4
	Fresh	90	1.20	3
	Total Measured	209	1.11	7
Indicated	Oxide	152	0.96	5
	Transitional	5,071	0.98	159
	Fresh	24,438	1.18	926
	Total Indicated	29,661	1.14	1,090
Inferred	Oxide	81	0.93	2
	Transitional	402	1.04	13
	Fresh	1,283	1.17	48
	Total Inferred	1,766	1.13	64

Table 14.33 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	1	0.89	0
	Transitional	88	1.26	4
	Fresh	76	1.35	3
	Total Measured	165	1.30	7
Indicated	Oxide	115	1.14	4
	Transitional	3,793	1.17	143
	Fresh	20,301	1.33	871
	Total Indicated	24,209	1.31	1,018
Inferred	Oxide	62	1.09	2
	Transitional	337	1.16	13
	Fresh	1,082	1.31	46
	Total Inferred	1,481	1.27	60

Table 14.34 Breakdown of the KILO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	23	0.94	1
	Transitional	31	0.95	1
	Fresh	-	-	-
	Total Inferred	54	0.95	2

Table 14.35 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	23	0.94	1
	Transitional	31	0.96	1
	Fresh	-	-	-
	Total Inferred	54	0.95	2

Table 14.36 Breakdown of the NAR Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	163	2.12	11
	Transitional	65	2.48	5
	Fresh	114	3.01	11
	Total Measured	342	2.49	27
Indicated	Oxide	303	1.40	14
	Transitional	193	1.33	8
	Fresh	3,302	1.80	191
	Total Indicated	3,798	1.74	213
Inferred	Oxide	0	0.36	0
	Transitional	0	0.55	0
	Fresh	47	0.65	1
	Total Inferred	48	0.65	1

Table 14.37 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	139	2.42	11
	Transitional	51	3.03	5
	Fresh	104	3.28	11
	Total Measured	294	2.83	27
Indicated	Oxide	229	1.72	13
	Transitional	139	1.69	8
	Fresh	2,749	2.08	183
	Total Indicated	3,117	2.03	204
Inferred	Oxide	-	-	-
	Transitional	0	0.61	0
	Fresh	32	0.76	1
	Total Inferred	32	0.76	1

Table 14.38 Breakdown of the NOK Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	10	1.21	0
	Fresh	3	1.78	0
	Total Indicated	12	1.33	1
Inferred	Oxide	7	0.93	0
	Transitional	83	1.42	4
	Fresh	24	2.47	2
	Total Inferred	114	1.61	6

Table 14.39 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	9	1.28	0
	Fresh	2	1.88	0
	Total Indicated	11	1.41	1
Inferred	Oxide	5	1.15	0
	Transitional	61	1.80	4
	Fresh	21	2.73	2
	Total Inferred	87	1.99	6

Table 14.40 Breakdown of the SAN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	1	0.70	0
	Transitional	102	1.92	6
	Fresh	58	3.37	6
	Total Inferred	161	2.43	13

Table 14.41 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	1	0.88	0
	Transitional	83	2.27	6
	Fresh	55	3.51	6
	Total Inferred	139	2.75	12

Table 14.42 Breakdown of the SOL Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	253	1.96	16
	Transitional	2	0.96	0
	Fresh	1	0.61	0
	Total Measured	255	1.95	16
Indicated	Oxide	5,996	1.35	260
	Transitional	1,049	1.46	49
	Fresh	8,933	1.46	420
	Total Indicated	15,978	1.42	729
Inferred	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Inferred	-	-	-

Table 14.43 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	214	2.24	15
	Transitional	2	1.06	0
	Fresh	1	0.65	0
	Total Measured	217	2.23	16
Indicated	Oxide	4,399	1.69	239
	Transitional	776	1.83	46
	Fresh	6,972	1.76	395
	Total Indicated	12,147	1.74	679
Inferred	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Inferred	-	-	-

Table 14.44 Breakdown of the SWA Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	433	0.77	11
	Transitional	575	1.04	19
	Fresh	48	1.70	3
	Total Inferred	1,056	0.96	33

Table 14.45 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Indicated	-	-	-
Inferred	Oxide	292	0.95	9
	Transitional	447	1.22	18
	Fresh	46	1.74	3
	Total Inferred	786	1.15	29

Table 14.46 Breakdown of the THN Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	233	0.61	5
	Transitional	1,744	0.69	39
	Fresh	507	0.91	15
	Total Indicated	2,484	0.73	58
Inferred	Oxide	75	0.70	2
	Transitional	45	0.70	1
	Fresh	4	1.04	0
	Total Inferred	124	0.71	3

Table 14.47 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.3 g/t Au

Classification	Oxidation	kt	Au g/t	Au koz
Measured	Oxide	-	-	-
	Transitional	-	-	-
	Fresh	-	-	-
	Total Measured	-	-	-
Indicated	Oxide	142	0.75	3
	Transitional	1,215	0.81	32
	Fresh	427	1.00	14
	Total Indicated	1,784	0.85	49
Inferred	Oxide	55	0.80	1
	Transitional	34	0.79	1
	Fresh	4	1.12	0
	Total Inferred	9	0.81	2

Table 14.48 Breakdown of the VAKO Mineral Resource by Oxidation State Reported within a USD2,000 Optimised Pit Shell at a Gold Grade Cut-off of 0.5 g/t Au

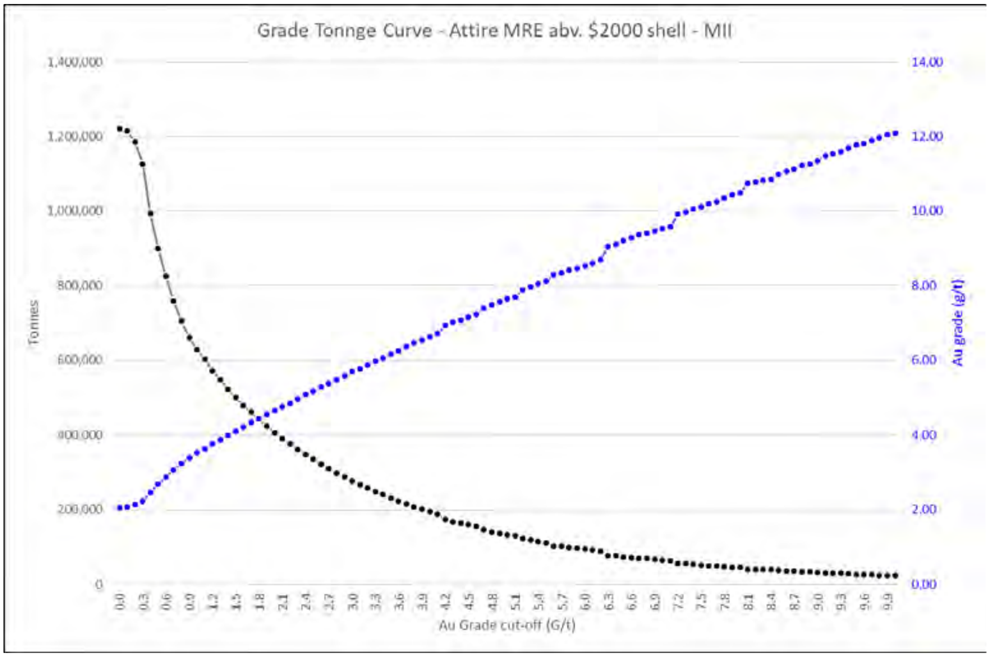


Figure 14.57 Grade and Tonnage Curves for In-pit Classified Material - ATI

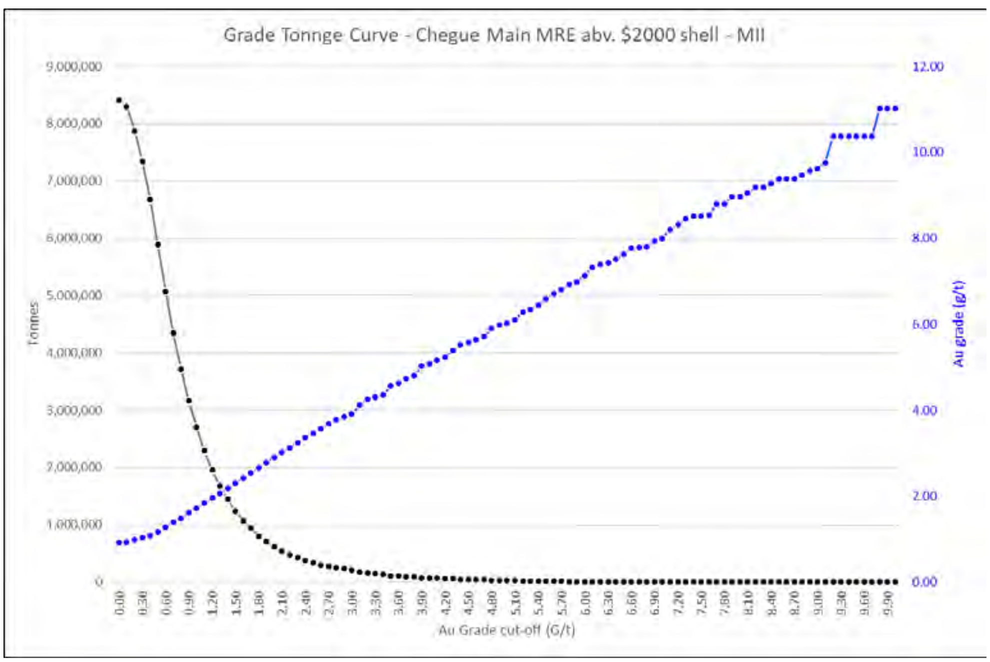


Figure 14.58 Grade and Tonnage Curves for In-pit Classified Material - CHG Main

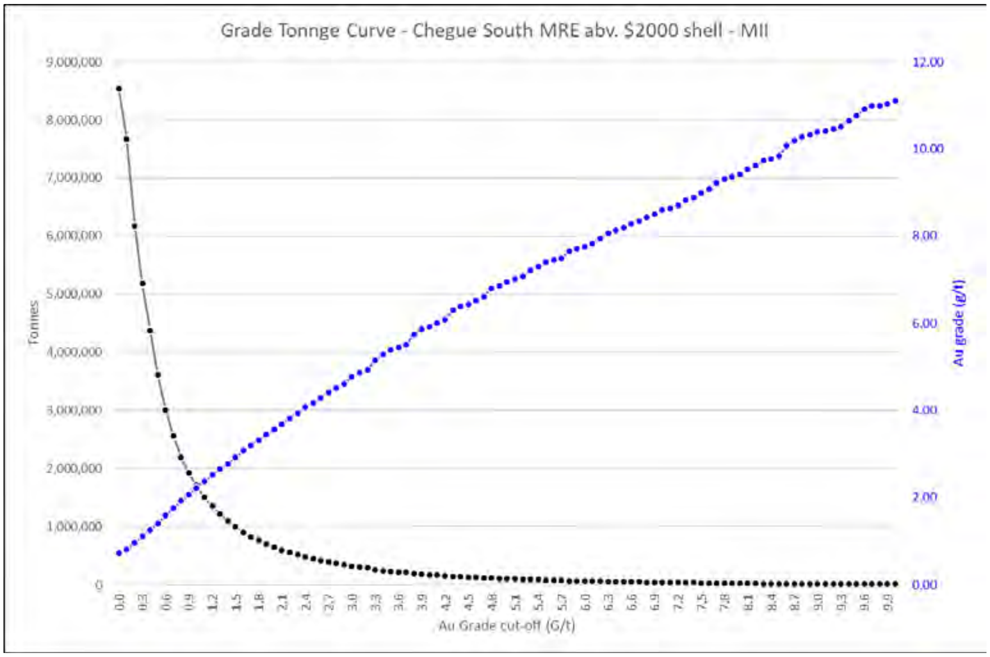


Figure 14.59 Grade and Tonnage Curves for In-pit Classified Material - CHG South

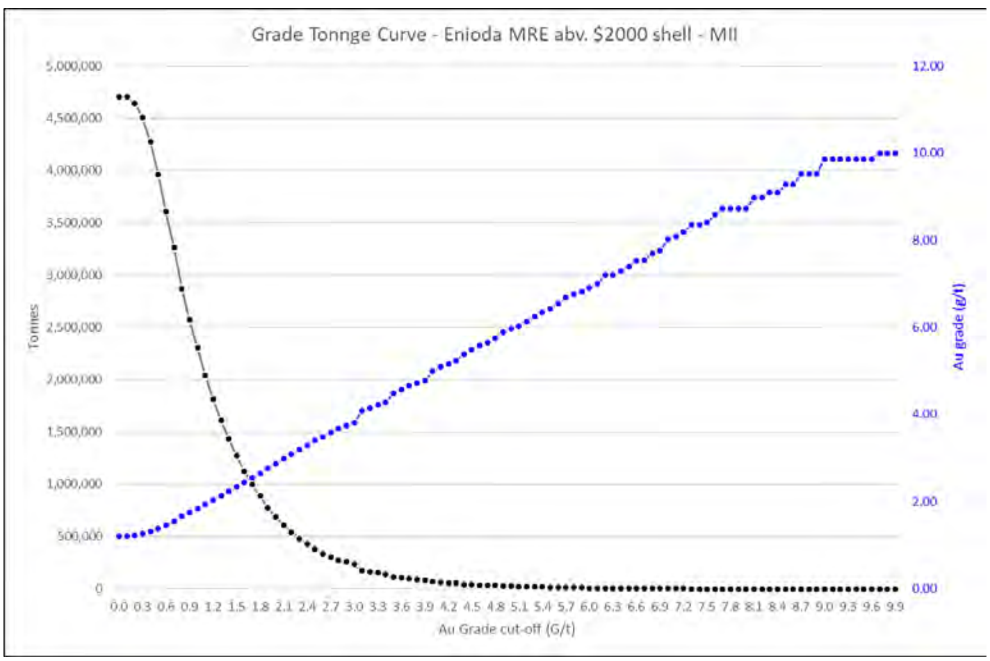


Figure 14.60 Grade and Tonnage Curves for In-pit Classified Material - ENI

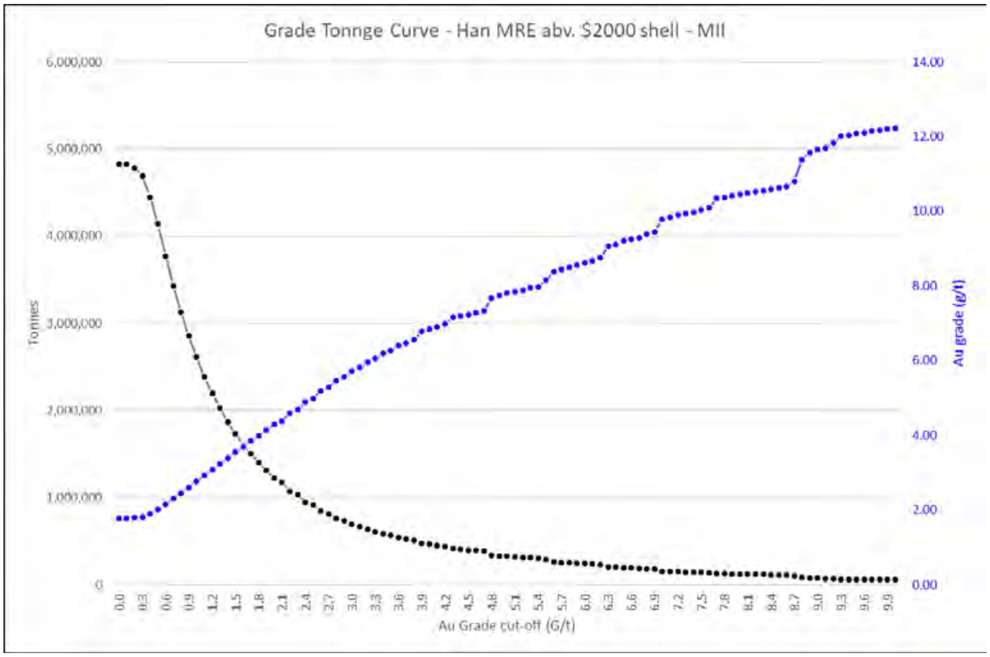


Figure 14.61 Grade and Tonnage Curves for In-pit Classified Material - HAN

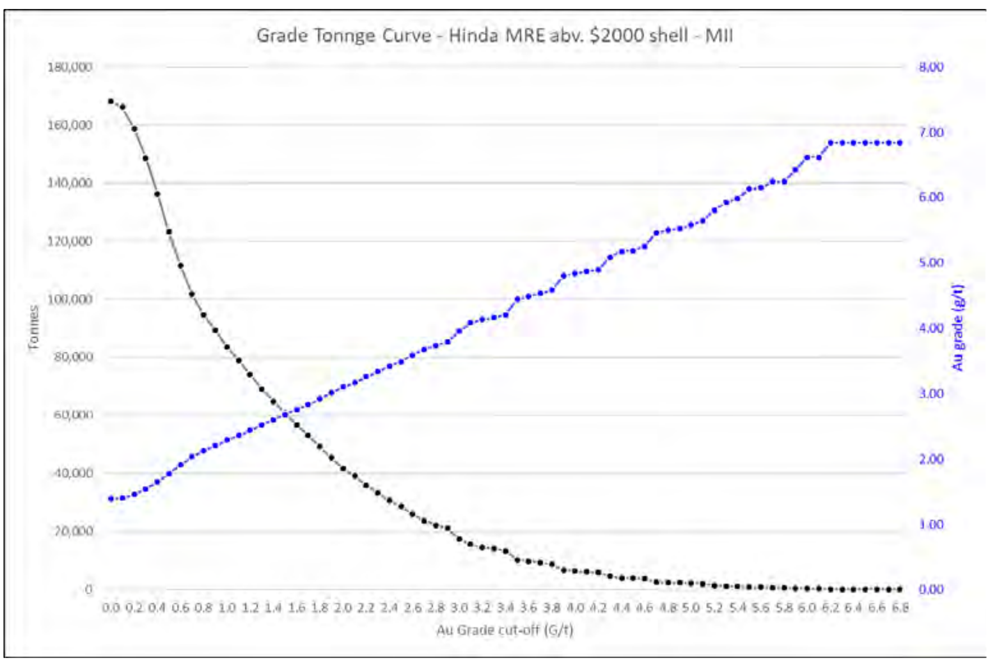


Figure 14.62 Grade and Tonnage Curves for In-pit Classified Material - HND

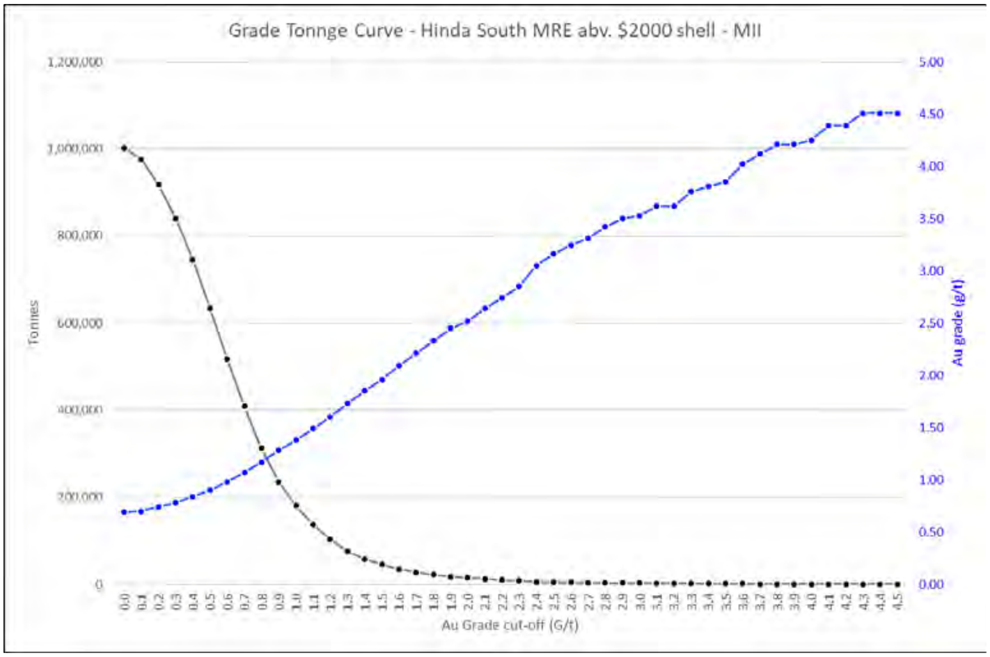


Figure 14.63 Grade and Tonnage Curves for In-pit Classified Material - HNDS

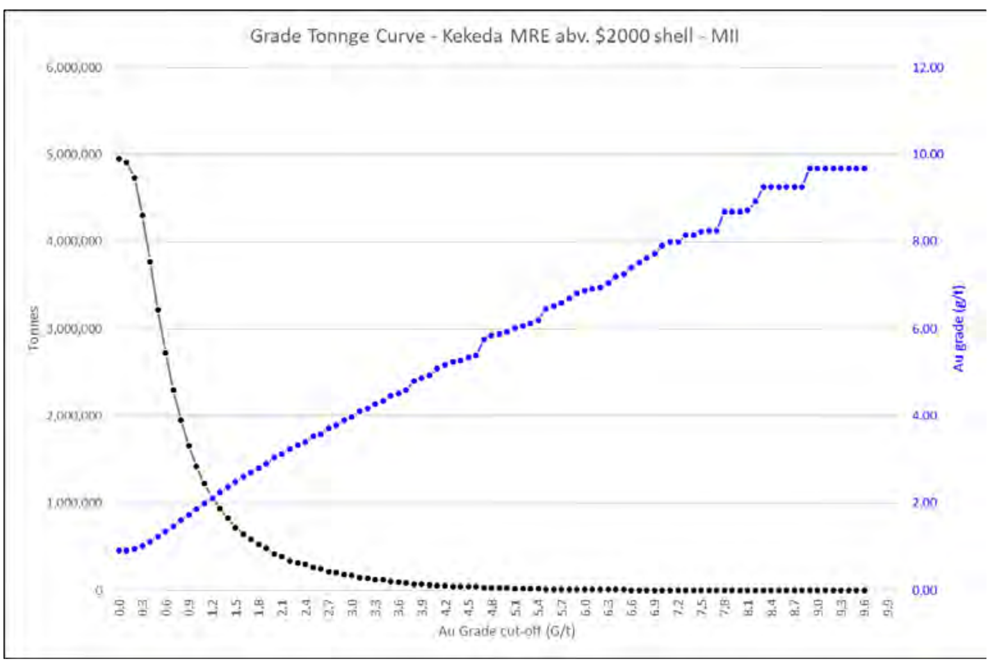


Figure 14.64 Grade and Tonnage Curves for In-pit Classified Material - KEK

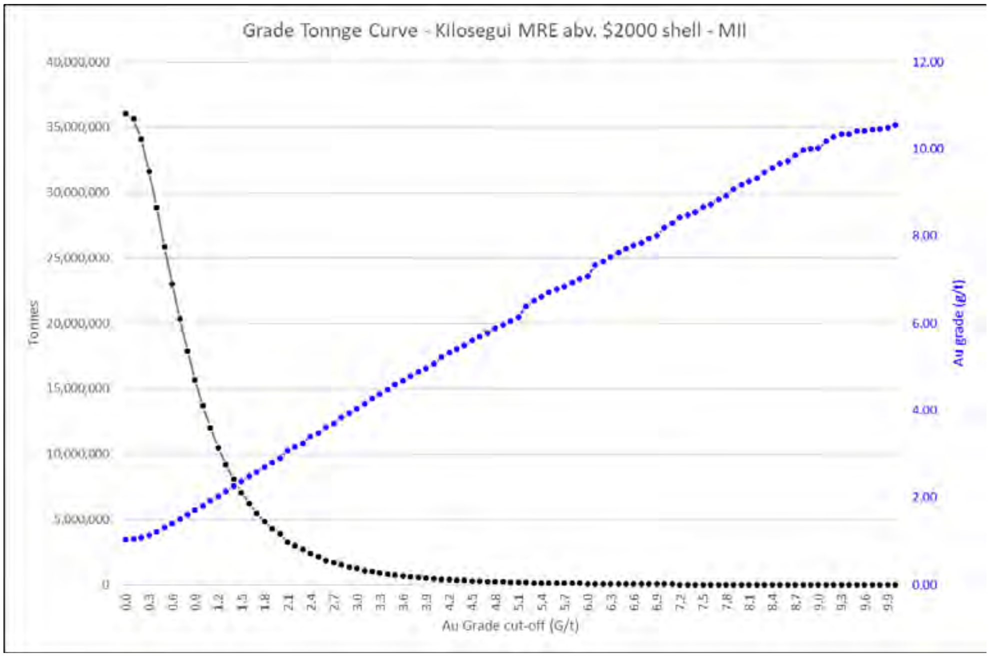


Figure 14.65 Grade and Tonnage Curves for In-pit Classified Material - KILO

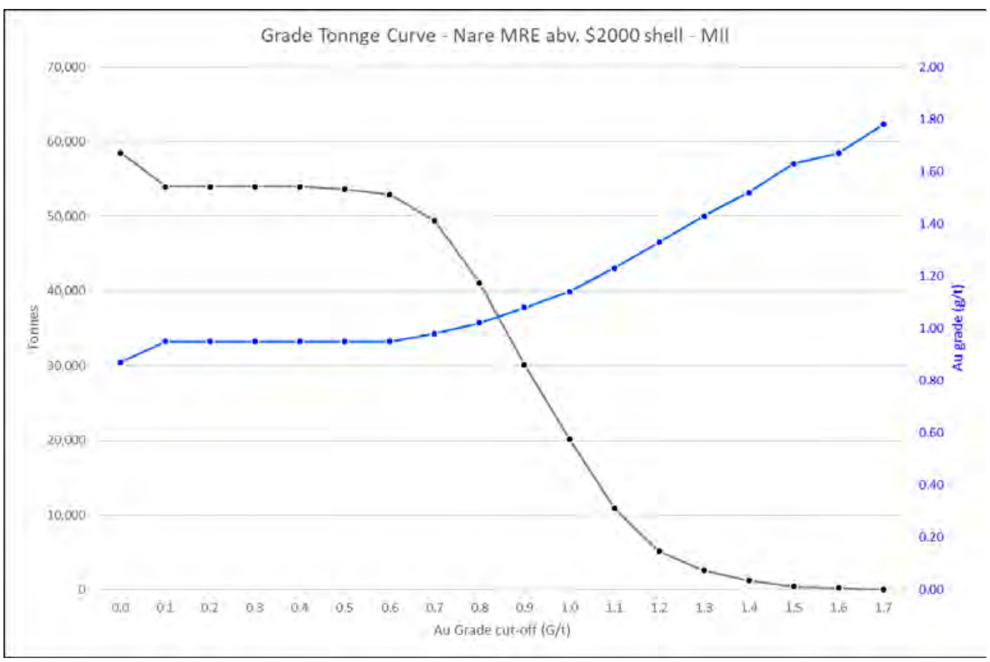


Figure 14.66 Grade and Tonnage Curves for In-pit Classified Material - NAR

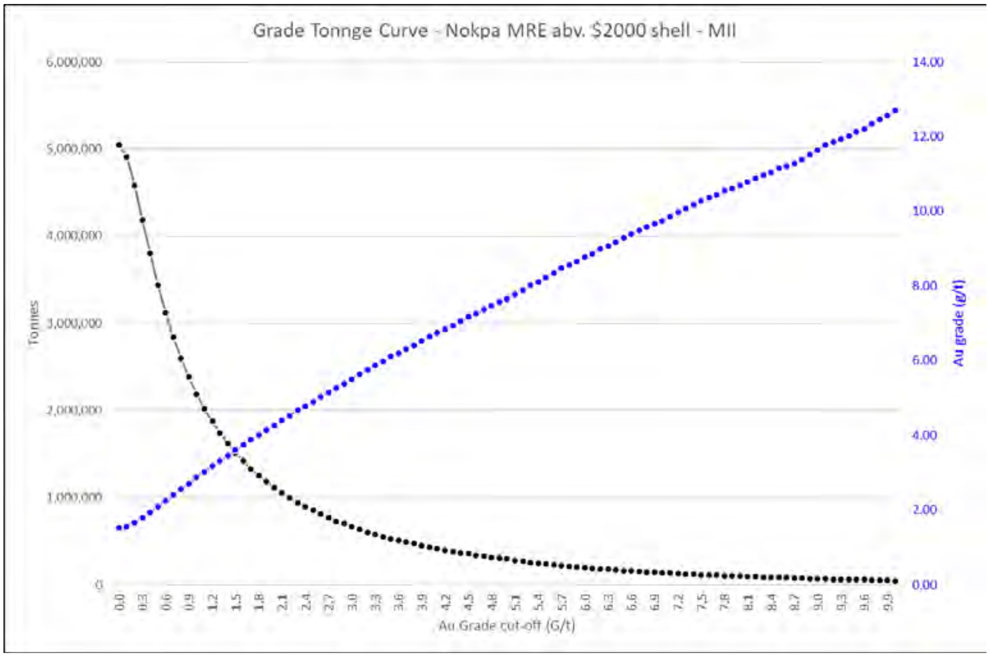


Figure 14.67 Grade and Tonnage Curves for In-pit Classified Material - NOK

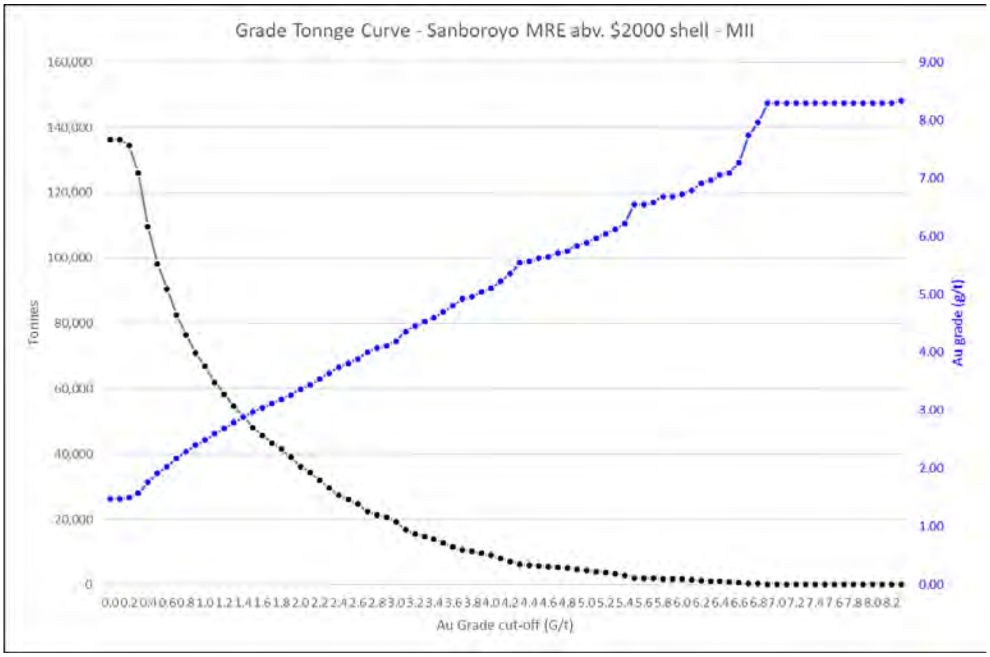


Figure 14.68 Grade and Tonnage Curves for In-pit Classified Material - SAN

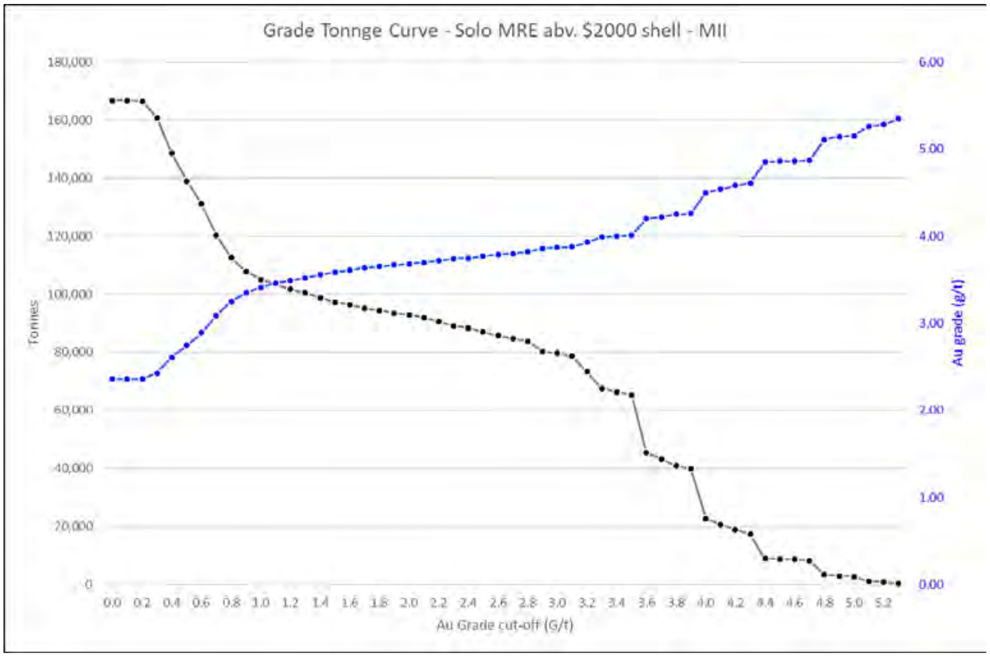


Figure 14.69 Grade and Tonnage Curves for In-pit Classified Material - SOL

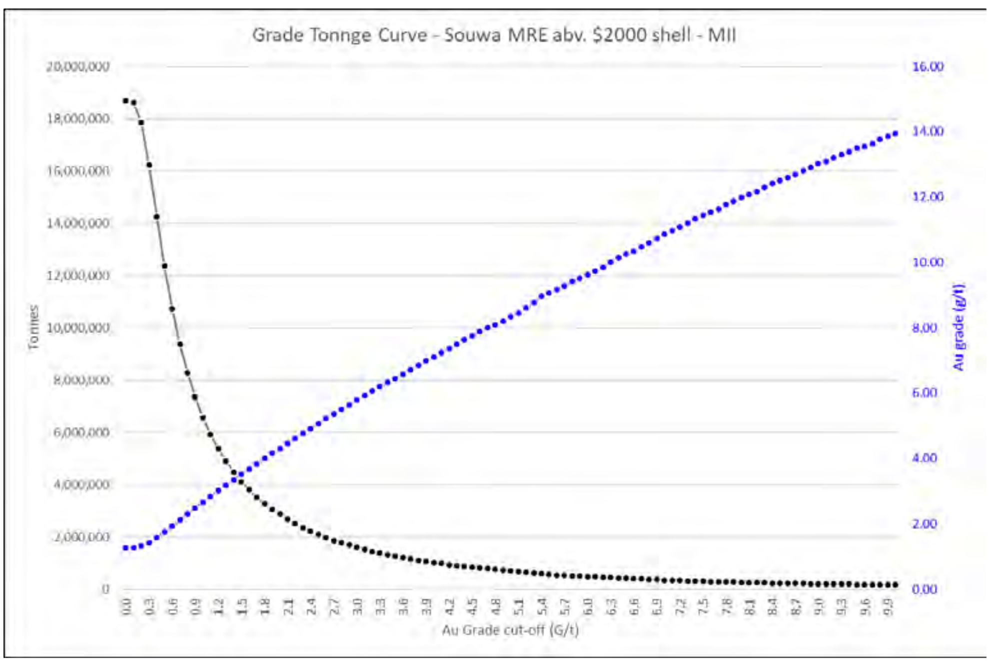


Figure 14.70 Grade and Tonnage Curves for in-pit Classified Material - SWA

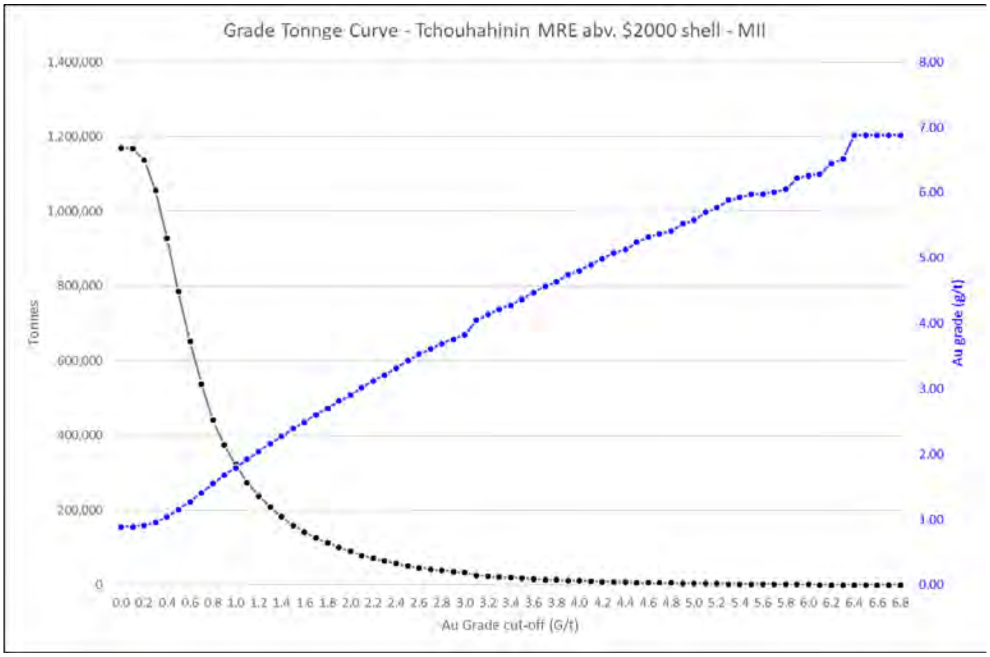


Figure 14.71 Grade and Tonnage Curves for In-pit Classified Material - THN

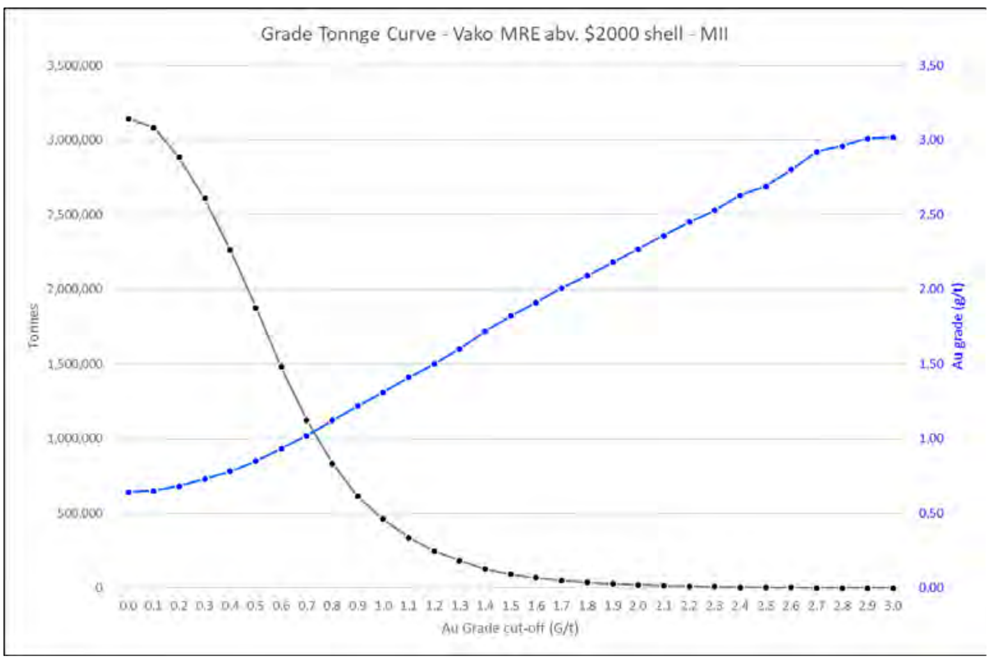


Figure 14.72 Grade and Tonnage Curves for In-pit Classified Material - VAKO

14.21 Factors that May Affect the Mineral Resource

The drilling, sampling and analytical methods utilised by Centamin are considered appropriate for Mineral Resource modelling and the input data have been found to be of sufficient quality.

Most of the Doropo prospects remain open, predominantly at depth. Additional drilling would be required to fully delineate the mineralisation extents.

The estimation method employed is considered to be appropriate for open pit mining studies.

There are no current known environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors that could materially impact the MRE.

14.22 Comparison to Previous Mineral Resource

Reported Mineral Resources for this 2023 update MRE and the 2022 MRE are compared in Table 14.49 through Table 14.51.

The following points are pertinent:

- The increase in Mineral Resource tonnage from 2022 to 2023 is due predominantly to the introduction of new prospects: Sanboyoro, Solo, and Hinda South;
- A small proportion of the Indicated resources has been converted to Measured after grade control drilling took place;
- A large proportion of Inferred Resources (10 Mt) has been converted to Indicated after infill drilling at 25 - 30 m spacing;
- Overall, at 0.5 g/t Au cut-off, there is an increase in tonnes (1%) and grade (3%) for 4% more ounces;
- Comparison has been done at 0.5 g/t Au due to the 2022 MRE being reported at this cut-off;
- Comparison has been done above $2,000 pit shells created for the DFS (October 2023).

Prospect	Measured Mineral Resources - 2023			Measured Mineral Resources - 2022		
	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Attire	-	-	-			-
Chegue Main	0.17	1.13	0.006			-
Chegue South	0.16	1.34	0.007			-
Enioda	-	-	-			-
Han	0.10	2.19	0.007			-
Hinda*	-	-	-			-
Hinda South**	-	-	-			-
Kekeda	0.09	1.29	0.004			-
Kilosegui	0.16	1.30	0.007			-
Nare*	-	-	-			-
Nokpa	0.29	2.83	0.027			-
Sanboyoro**	-	-	-			-
Solo**	-	-	-			-
Souwa	0.22	2.23	0.016			-
Tchouahinin*	-	-	-			-
Vako*	-	-	-			-
Total	1.21	1.89	0.074	0	0	0

Table 14.49 Reported Measured Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE

Prospect	Indicated Mineral Resources - 2023			Indicated Mineral Resources - 2022		
	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Attire	0.36	2.08	0.024	-		-
Chegue Main	5.20	1.17	0.196	4.94	1.22	0.193
Chegue South	3.37	1.43	0.155	3.14	1.34	0.136
Enioda	2.93	1.43	0.135	2.22	1.46	0.104
Han	3.80	2.02	0.246	3.55	2.10	0.240
Hinda*	-	-	-	-		-
Hinda South**	-	-	-	-		-
Kekeda	3.04	1.23	0.120	3.14	1.42	0.143
Kilosegui	24.21	1.31	1.018	19.57	1.27	0.801
Nare*	-	-	0.000	-		0.000
Nokpa	3.12	2.03	0.204	3.69	1.96	0.232
Sanboyoro**	0.01	1.41	0.001	-		-
Solo**	-	-	-	-		-
Souwa	12.15	1.74	0.679	13.24	1.73	0.736
Tchouahinin*	-	-	-	-		0.000

Prospect	Indicated Mineral Resources - 2023			Indicated Mineral Resources - 2022		
	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Vako*	1.78	0.85	0.049	-		-
Total	59.98	1.47	2.826	53.48	1.50	2.584

Table 14.50 Reported Indicated Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE

Prospect	Inferred Mineral Resources - 2023			Inferred Mineral Resources - 2022		
	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Attire	0.54	3.08	0.053	0.84	2.04	0.055
Chegue Main	0.51	1.31	0.021	0.47	1.30	0.019
Chegue South	0.07	0.97	0.002	0.06	0.93	0.002
Enioda	1.03	1.24	0.041	2.14	1.19	0.082
Han	0.24	1.87	0.014	0.31	1.53	0.015
Hinda*	0.12	1.78	0.007	0.11	1.42	0.005
Hinda South**	0.63	0.90	0.018			-
Kekeda	0.09	1.08	0.003	0.05	0.82	0.001
Kilosegui	1.48	1.27	0.060	6.66	1.06	0.226
Nare*	0.05	0.95	0.002	0.04	1.16	0.002
Nokpa	0.03	0.76	0.001	0.03	0.79	0.001
Sanboyoro**	0.09	1.99	0.006			-
Solo**	0.14	2.75	0.012			-
Souwa	-	-	0.000	0.02	0.87	0.001
Tchouahinin*	0.79	1.15	0.029	0.60	1.39	
Vako*	0.09	0.81	0.002	1.64	0.94	0.049
Total	5.90	1.44	0.272	12.95	1.10	0.457

Table 14.51 Reported Inferred Mineral Resources @ 0.5 g/t Au - 2023 MRE Update Versus 2022 MRE

14.23 Audits and Reviews

No independent audit has been completed on the updated Doropo MRE.

Cube undertook regular internal peer reviews during the course of the MRE work.

TABLE OF CONTENTS

15. MINERAL RESERVES ESTIMATES 1
15.1 Introduction 1
15.1.1 Basis of Mineral Reserve 1
15.1.2 Statement of Independence 1
15.1.3 Risks and Forward-Looking Statements 1
15.1.4 Use of the Term 'Ore' in this Feasibility Study 2
15.2 Reliance On Other Experts 2
15.3 Mineral Reserve Estimate 3
15.3.1 Introduction 3
15.3.2 Mining Approach 4
15.3.3 Cut-off Grade 5
15.3.4 Mineral Reserve Estimate 7

LIST OF TABLES

Table 15.1 Breakeven Cut-Off Grade - Modifying Factors 6
Table 15.2 Breakeven Cut-Off Grades 6
Table 15.3 Doropo Gold Project FS Mineral Reserve Estimate 7

15. MINERAL RESERVES ESTIMATES

15.1 Introduction

15.1.1 Basis of Mineral Reserve

This Mineral Reserve estimate was produced for public reporting under Canadian National Instrument ("NI") 43-101 in Canada (NI 43-101, 2014). It was prepared in accordance with the requirements of:

- Disclosure and reporting requirements of the Toronto Stock Exchange ("TSX") as stipulated in TSX (2010);
- Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects', Form NI 43-101F1 and Companion Policy 43-101CP (NI 43-101, 2014); and
- Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards (CIM, 2014).

The Mineral Reserves reported herein exclude Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as a Mineral Reserve, so any Inferred Resources within the pit shell are considered 'waste' material for the purposes of this Mineral Reserve Estimate.

15.1.2 Statement of Independence

Orelogy is an independent consulting company contracted by Centamin to carry out the mining component of the Doropo FS. Neither Orelogy, nor the authors of this Part 15, currently have or previously have had, any material interest in Centamin or the mineral properties in which Centamin has an interest. Orelogy's relationship with Centamin is solely one of professional association between client and independent consultant.

This Part 15 was prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is not contingent on the results of this Part 15. No member or employee of Orelogy is, or is intended to be, a director, officer or other direct employee of Centamin.

In the preparation of this Part 15, Orelogy has used information provided by Centamin and other third-party experts appointed by Centamin. Orelogy has verified this information, making due enquiry of all material issues that are required in order to comply with NI 43-101 requirements.

15.1.3 Risks and Forward-Looking Statements

Mining and mineral exploration, development and production is, by its nature, a business with significant risk. Profitability and asset values can be affected by unforeseen operating and technical issues, the majority of which are, and will be, beyond the control of Centamin or any other operating entity.

This Part 15 contains forward-looking statements. These forward-looking statements are based on the opinions and estimates of Centamin, Orelogy and other specialist consultants at the date the statements were made. Readers are cautioned not to place undue reliance on forward-looking information or statements. By their nature, forward-looking statements involve numerous assumptions, inherent risks and uncertainties, both general and specific, which contribute to the possibility that actual results may differ materially from those anticipated in the forward-looking statements. Events or circumstances beyond Centamin's control could also cause actual results to differ materially from those estimated or projected and expressed in, or implied by, these forward-looking statements.

Orelogy considers the basis, and associated outcomes, of the forward-looking assumptions applied to their component of work to be reasonable. However, Orelogy does not guarantee future results, levels of activity, performance or achievements of the Project as outlined in the Feasibility Study.

15.1.4 Use of the Term 'Ore' in this Feasibility Study

The Canadian National Instrument Companion Policy 43-101 (Section 2.3) indicates that, in the context of Mineral Resource estimates, the term 'ore' implies technical feasibility and economic viability that should only be attributed to 'Mineral Reserves'. In compliance with Section 2.3 of the Companion Policy, the term ore is not used in the Mineral Resource context of this FS.

The term ore is used in the mining and processing sections of this FS in a generic way to describe the 'mineable' part of the resource estimate that will be extracted from the mine and fed to the process plant. Where appropriate this is referred to as the 'Mineral Reserve' after investigation and application of all relevant Modifying Factors as discussed in Section 15.3.3, in conformance with the definitions in CIM (2014).

15.2 Reliance On Other Experts

This report was prepared by Orelogy for Centamin as a component of the Doropo FS for the purposes of Public Reporting. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Orelogy at the time of preparation of this report;
- Assumptions, conditions, and qualifications discussed in this report;
- Data, reports, and other information supplied by Centamin and other third parties, as documented and referenced in this FS Report.

For the purpose of this Part 15 Orelogy has relied on ownership information and other local knowledge provided by Centamin. Orelogy has no independent information regarding property title or mineral rights for the Doropo Gold Project and expresses no opinion as to the ownership status of the property.

Except for the purposes legislated under Canadian or other securities laws, any use of this report by any third party is at that party's sole risk.

The major components of this Mineral Reserve estimate comprise:

- Resource modelling based on available data;
- Mine design and production scheduling;
- Mining cost estimation;
- Metallurgical test work;
- Process design and process plant cost estimation;
- Environmental assessment;
- Financial analysis;
- Other supporting studies on geology, hydrogeology, hydrology, rock mechanics for pit slope design and geotechnical engineering.

In addition, this Mineral Reserve estimate is based on the specialist consultant studies identified in Part 3 of this NI 43-101 Technical Report and information from studies conducted on the behalf of the project owner, by independent specialist consultants.

15.3 Mineral Reserve Estimate

15.3.1 Introduction

Centamin engaged Orelogy to estimate a Mineral Reserve for the Project based on the latest available resource model as part of the Doropo FS.

The Mineral Reserve developed by Orelogy as part of the Doropo FS was announced by Centamin on 18 July 2024. Centamin and Orelogy confirm that they are not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Mineral Reserve estimates in the market announcement continue to apply and have not materially changed.

The Mineral Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors available at the time. The Proven Mineral Reserve estimate is based on Mineral Resources classified as Measured. The Probable Mineral Reserve estimate is based on Indicated Mineral Resources. No Inferred Mineral Resource was included in the Mineral Reserve. The Mineral Reserve estimate represents the economically mineable part of the Indicated Mineral Resources.

At this time, there are no known environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Mineral Reserve of the Doropo Gold Project.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely utilized in the gold industry in West Africa. Financial modelling completed as part of the FS shows that the Project is economically viable under current assumptions. Material Modifying Factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Mineral Reserve estimation process.

15.3.2 Mining Approach

The Mineral Resource Models (MRMs) produced by Cube (2023), as discussed in Part 14 of the Technical Report, were used to develop a Mining Block Model (MBM) for each deposit.

A total of seven (7) deposits were evaluated:

- Souwa (SWA);
- Nokpa (NOK);
- Chegue Main and Chegue South (CHG-MAIN and CHG-STH);
- Enioda (ENI);
- Han (HAN);
- Kekeda (KEK);
- Kilosegui (KLG).

The MRMs have a consistent minimum block size (sub-cell size) of 5 m x 5 m x 2.5 m (X,Y,Z), with a parent size ranging from 20 m x 20 m x 5 m (X,Y,Z) to 20 m x 40 m x 10 m (X,Y,Z). The MBMs were based on the associated MRM but were regularised to a consistent block model size of 10 m x 10 m x 5 m (X,Y,Z), maintaining the granularity of the MRM sub-cells by retaining an ore parcel and a waste parcel within the regularised block. An Orelogy proprietary routine was then run across the MBM that applies dilution on a block-by-block basis, by transferring a proportion of the waste parcel in any given block into the ore parcel. In this way a location “edge effect” type dilution is generated whereby blocks at the ore/waste get more heavily diluted than blocks in the middle of orebody. As a result, the global dilution and ore loss across all the deposits approximated 5% and 6% respectively but varied from deposit to deposit by 2%-11% and 2%-9% respectively. These levels of dilution and ore loss are considered appropriate for:

- The proposed 5 m blast height and excavating ore in a 2.5 m flitch height;
- The proposed equipment size (150 tonne excavator for selective mining);
- The orebody geometry of each deposit.

Open pit optimizations were run in Whittle 4X using a US$1,450/oz gold price to define the geometry of the economic open pit shapes. Mining costs were derived from submissions from mining contractors to a Request for Budget Pricing. Other modifying factors such as processing operating costs and performance, general and administrative overheads, project capital and royalties were provided by Centamin.

Optimization shells were selected, and pit designs were then completed based on these shells, along with associated waste storage landforms, mine haul roads and other mining infrastructure. Some internal mining stages were developed for Souwa.

Mining of the Doropo Project will be undertaken utilising medium-scale open pit mining equipment (90 t haul trucks, 150 t to 200 t hydraulic excavators) typical for West Africa. The mining process will be based on a conventional drill, blast, load and haul operation. Mining areas of disturbance (i.e., pits, dumps, roads etc.) will be cleared of vegetation and topsoil which will be stockpiled for subsequent rehabilitation purposes. Centamin will have a management and technical team which will supervise the mining contract and undertake all technical activities such as grade control, surveying and mine planning.

A Life of Mine (LOM) mining schedule was generated detailing the movement of ore and waste on 5 m benches for a mine life of approximately 10-years. This includes haulage of the ore from the satellite pits (Han, Kekeda, Enioda and Kilosegui) to the process plant located at Souwa. The schedule indicates that the design process feed rate can be met for the entire mine life.

Approximately 38.2 Mt of ore is processed, of which approximately 80% is direct fed to the plant and 5% is stockpiled as low-grade material.

The FS is based on mining using a 5 m blast height and excavating in 2.5 m flitches. Waste will be excavated at the 5 m blast height where the larger production excavator is used.

For the purposes of the mining study, it was assumed that a ‘dry pit’ will be operated and that sufficient dewatering will be undertaken to comply with the geotechnical design criteria.

15.3.3 Cut-off Grade

The estimated breakeven cut-off grade for the Doropo FS Mineral Reserve was based on the calculation detailed below:

$$COG = \frac{(Ore\ Related\ Mining\ Cost + Processing\ Cost)\ \times (1 + Dilution)}{(Net\ Price\ \times Process\ Recovery)}$$

The modifying factors used to develop the cut-off grade were those available at the time of the LOM production scheduling and are detailed in the Table 15.1.

Item		Unit	Value
Ore Related Mining Costs	ROM Rehandle	$US/dmt	$0.62
	Grade Control	$US/dmt	$0.50
	Owners Team	$US/dmt weathered	$0.63
		$US/dmt fresh	$0.85
Processing Cost		$US/dmt oxide	$11.43
		$US/dmt trans	$11.26
		$US/dmt fresh	$14.35
G & A Cost		$US/dmt oxide	$3.81
		$US/dmt trans	$3.81
		$US/dmt fresh	$5.14
Ore Haulage	Enioda	$US/dmt	$2.95
	Han	$US/dmt	$2.50
	Kekeda	$US/dmt	$2.13
	Kilosegui	$US/dmt	$5.50
Process Recovery	Weathered	%	Refer to Table 16.13 (Part 16)
	Fresh	%	
Gold Price	Base Gold Price	$US/oz	$1,450.00
	Govt. Royalty	%	4.00%
	Tenement Royalty	%	0.50%
	Social Fund Royalty	%	0.50%
	Gold Loss	%	0.05%
	Charge	$US/oz	$4.00
	Net Gold Price	$US/oz	$1,372.78
		$US/gram	$44.14

Table 15.1 Breakeven Cut-Off Grade - Modifying Factors

As costs and process recovery vary by weathering and/or location, the calculated cut-off grade also varies as detailed in the table below.

Mining Area	Unit	Oxide	Trans	Fresh
Souwa	g/t	0.43	0.45	0.62
Kilosegui	g/t	0.56	0.58	0.77
Nokpa	g/t	1.00[1]	0.42	0.62
Cheque Main	g/t	0.43	0.45	0.62
Cheque South	g/t	N/A	0.45	0.62
Kekeda	g/t	0.49	0.55	0.68
Han	g/t	N/A	0.50	0.69
Enioda	g/t	0.52	0.57	0.70

[1] Nokpa oxide only has recovery for >1.0 g/t material (refer to Table 16.16, in Part 16)

Table 15.2 Breakeven Cut-Off Grades

15.3.4 Mineral Reserve Estimate

The Mineral Reserves were estimated for the Doropo Gold Project as part of this FS by Orelogy Mine Consulting. The total Proven and Probable Mineral Reserve is estimated at 38.2 Mt at 1.53 g/t Au with a contained gold content of 1,876 koz.

The Mineral Reserve for the Project is reported according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The Mineral Resource was converted by applying Modifying Factors. The Proven Mineral Reserve estimate is based on the Mineral Resource classified as Measured. The Probable Mineral Reserve estimate is based on the Mineral Resource classified as Indicated. Table 15.3 presents a summary of the Mineral Reserves on a 100% Project basis at a gold price of US$1,450/oz.

Classification	Mt	Grade (g/t Au)	Contained Ounces
Proven	1.3	1.73	70,100
Probable	37.0	1.52	1,805,559
Total	**38.2**	**1.53**	**1,875,659**

Notes:

- *The Mineral Reserve conforms with and uses the CIM (2014) definitions.*
- *The Mineral Reserve was evaluated using a gold price of US$1,450 per ounce.*
- *The Mineral Reserve was evaluated using variable cut-off grades as described in Table 15.2.*
- *Block grade and tonnage dilution was incorporated into the model.*
- *All figures are rounded to reflect appropriate levels of confidence.*
- *Apparent differences may occur due to rounding.*

Table 15.3 Doropo Gold Project FS Mineral Reserve Estimate

All figures are rounded to reflect appropriate levels of confidence. Apparent differences may occur as a result of rounding. Based on the information presented in this FS, the Mineral Reserve estimation process has converted approximately 85% of the Measured Mineral Resources to Proven Mineral Reserve and 50% of the Indicated Mineral Resources to Probable Mineral Reserves.

TABLE OF CONTENTS

16. MINING METHODS ... 1
16.1 Introduction ... 1
16.2 Mining Method Selection ... 1
16.3 Geotechnical Considerations ... 4
16.3.1 Modelling ... 4
16.3.2 Slope Stability Assessment ... 6
16.3.3 Saprolite Design ... 7
16.3.4 Saprock and Fresh Rock Design ... 8
16.3.5 Operational Considerations ... 9
16.4 Mining Contractor Request for Budget Pricing ... 10
16.5 Mining Dilution and Recovery Factors ... 13
16.5.1 Mining Block Model ... 13
16.5.2 Dilution and Ore Loss ... 15
16.6 Open Pit Optimisation ... 17
16.6.1 Open Pit Optimisation Approach ... 17
16.6.2 Optimisation Modifying Factors ... 20
16.6.3 Optimisation Results and Shell Selection ... 25
16.7 Mine Design ... 37
16.7.1 Pit Design Criteria ... 37
16.7.2 Final Pit Design ... 39
16.7.3 Pit Design Reconciliation to Whittle Shells ... 58
16.7.4 Waste Rock Dump Design ... 59
16.7.5 General Site Arrangement ... 60
16.8 Life of Mine Production Schedule ... 66
16.8.1 Scheduling Objective and Constraints ... 66
16.8.2 LOM Production Schedule Results ... 76
16.9 Mining Cost Rates and Assumptions ... 84
16.9.1 Estimation Approach and Accuracy ... 84
16.9.2 Site Establishment and Mobilisation/Demobilisation ... 84
16.9.3 Primary Production Rates ... 86
16.9.4 Other Mining Activities and Fixed Costs ... 89
16.9.5 Ore Handling ... 89
16.9.6 Dayworks ... 90
16.10 LOM Mining Contractor Cost Estimate ... 90

LIST OF FIGURES

Figure 16.1 Location of Mining Areas ... 2
Figure 16.2 Geological Modelling of Weathering Zones ... 5
Figure 16.3 Pit Slope Design Parameters ... 7
Figure 16.4 Comparison of November 2023 RBP Submission ($/t Mined Basis) ... 13
Figure 16.5 Best & Worst Case DCF Mining Sequence (Cross-Section View) ... 19
Figure 16.6 Souwa Pit Optimisation - Tonne/Value Chart ... 27
Figure 16.7 Souwa Pit Optimisation - Strip Ratio and Cost Per Oz ... 28
Figure 16.8 Kilosegui Pit Optimisation - Tonne/Value Chart ... 29
Figure 16.9 Kilosegui Pit Optimisation -Strip Ratio and Cost Per Oz ... 29
Figure 16.10 Han Pit Optimisation - Tonne/Value Chart ... 30
Figure 16.11 Han Pit Optimisation - Strip Ratio and Cost Per Oz ... 30
Figure 16.12 Kekeda Pit Optimisation - Tonne/Value Chart ... 31
Figure 16.13 Kekeda Pit Optimisation - Strip Ratio and Cost Per Oz ... 32
Figure 16.14 Nokpa Pit Optimisation - Tonne/Value Chart ... 33
Figure 16.15 Nokpa Pit Optimisation - Strip Ratio and Cost Per Oz ... 33
Figure 16.16 Chegue Main Pit Optimisation - Tonne/Value Chart ... 34
Figure 16.17 Chegue Main Pit Optimisation - Strip Ratio and Cost per oz ... 34
Figure 16.18 Chegue South Pit Optimisation - Tonne/Value Chart ... 35
Figure 16.19 Chegue South Pit Optimisation - Strip Ratio and Cost Per Oz ... 36
Figure 16.20 Enioda Pit Optimisation - Tonne/Value Chart ... 36
Figure 16.21 Enioda Pit Optimisation - Strip Ratio and Cost Per Oz ... 37
Figure 16.22 Two-way Ramp Layout ... 38
Figure 16.23 One-way Ramp Layout ... 38
Figure 16.24 General Site Arrangement ... 41
Figure 16.25 Pit Design - Souwa Borrow Pit ... 42
Figure 16.26 Pit Design - Souwa Stage 1 ... 43
Figure 16.27 Pit Design - Souwa Stage 1A ... 44
Figure 16.28 Pit Design - Souwa Stage 2 ... 45
Figure 16.29 Pit Design - Souwa Stage 3 ... 46
Figure 16.30 Pit Design - Souwa Stage 4 ... 47
Figure 16.31 Pit Design - Nokpa Stage 1 ... 49
Figure 16.32 Pit Design - Nokpa Stage 2 ... 50
Figure 16.33 Pit Design - Kilosegui East ... 52
Figure 16.34 Pit Design - Kilosegui West ... 53
Figure 16.35 Pit Design - Enioda ... 54
Figure 16.36 Pit Design - Han ... 55
Figure 16.37 Pit Design - Kekeda ... 56

Figure 16.38 Pit Design - Chegue Main and South 57
Figure 16.39 General Site Arrangement - Hub Mining Area 61
Figure 16.40 General Site Arrangement - Kekeda, Han, Enioda Satellite Mining Area 62
Figure 16.41 General Site Arrangement - Kilosegui Mining Area 63
Figure 16.42 ROM Pad Layout Design 65
Figure 16.43 Process Plant Ramp-Up (% of Nominal Throughput) 68
Figure 16.44 Pit Stage Layout for LOM Production Schedule (Hub and Satellite Pits) 74
Figure 16.45 Pit Stage Layout for LOM Production Schedule (Kilosegui) 75
Figure 16.46 LOM Total Mining by Mining Area (Annual) 77
Figure 16.47 Total Mining by Mining Area (Pre-Production to End Year 2) 77
Figure 16.48 Mining Sequence by Pit Stage 78
Figure 16.49 Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade 80
Figure 16.50 Utilisation of Plant Capacity (hours) 80
Figure 16.51 Feed Ounces, Recovered Ounce and Recovery 81
Figure 16.52 Stockpile Balance and Grade by Weathering 81
Figure 16.53 Annual Mining Costs 90
Figure 16.54 Mining Contractor Cost Estimate Breakdown 91

LIST OF TABLES

Table 16.1 Description of Geological Weathering Domains 6
Table 16.2 Saprolite (CW & HW) Design Recommendations 8
Table 16.3 Saprock and Fresh Rock (MW, SW & FR) Design Recommendations 9
Table 16.4 Request for Budget Pricing Items 12
Table 16.5 Average of Competitive Contractor RBP Submission ($/t Mined Basis) 13
Table 16.6 Resource Block Model Parameters 14
Table 16.7 Dilution Parameters by Orebody 16
Table 16.8 Approximate Ore Loss and Dilution 17
Table 16.9 Saprolite (Oxide) Weathering Zones Geotechnical Parameters 20
Table 16.10 Saprock (Transition) and Fresh Weathering Zones Geotechnical Parameters 21
Table 16.11 Slope Angles by Mining Area used in the Whittle Optimisation Process 21
Table 16.12 Average Waste Mining Cost by Deposit 22
Table 16.13 Process Recoveries 23
Table 16.14 Ore Cost 24
Table 16.15 Satellite Ore Haulage Costs 24
Table 16.16 Product Revenue Parameters 25
Table 16.17 Optimisation Results by Mining Area 26
Table 16.18 Pit Design Inventory and % Difference from Optimisation Shells 58

Table 16.19 Waste Rock Dump Design Criteria 59
Table 16.20 Waste Rock Dump Designed Capacity Versus Required Capacity 59
Table 16.21 LOM Production Schedule Periods 67
Table 16.22 Schedule Production Rate Cat 6015 69
Table 16.23 Schedule Production Rate Cat 6020 70
Table 16.24 Digging Fleet and Mining Capacity by Period 72
Table 16.25 LOM Production Schedule Physicals by Period (Pre-Production to Year 3) 82
Table 16.26 LOM Production Schedule Physicals by Period (Year 4 to Year 9) 83
Table 16.27 Selected RFQ Submission - Mobilisation and Demobilisation 85
Table 16.28 Selected RFQ Submission - Site Establishment and Infrastructure 85
Table 16.29 Selected RFQ Submission - Drill and Blast Rates ($/BCM) 86
Table 16.30 Selected RFQ Submission - Waste Load and Haul Rates ($/BCM) 87
Table 16.31 Selected RFQ Submission - Ore Load and Haul Rates ($/BCM) 88
Table 16.32 Selected RBP Submission - Fixed Fees and Unit Rates 89
Table 16.33 Selected RBQ Submission - Ore Reclaim Rates 89
Table 16.34 Selected RFQ Submission - Satellite Pit Ore Haulage Rates 90
Table 16.35 Total Mining Contract Costs to Year 3 by Quarterr 93
Table 16.36 Total Mining Contract Costs to Year 4 - Year 9 by Quarter/Year 94

16. MINING METHODS

16.1 Introduction

The mining component of this Feasibility Study was prepared by Orelogy Mine Consulting (“Orelogy”). This comprised the following key activities:

- Distribution of a mining contract Request for Budget Pricing (RBP) for the Doropo Gold Project to a range of suitably qualified mining contractors in West Africa, and evaluation of the submissions received. The RBP was based on mine plans developed utilising the outcomes from the previous Preliminary Feasibility Study (PFS) for the Project;
- Pit optimisations were completed for the seven (7) deposits of the project utilising the October 2023 Mineral Resource Models for each deposit inclusive of allowances for dilution and ore loss. Appropriate Modifying Factors were applied including mining costs based on the RBP submissions and the most up-to-date parameters provided by Centamin (i.e. processing costs and recoveries, G&A costs, gold price, royalties etc.);
- Optimisation shells were selected from the optimisation results for each deposit, and pit designs were completed along with associated designs for waste rock dumps, mine roads, ore haulage roads etc.;
- A Life of Mine (LOM) Production Schedule was developed utilising the pit designs across months for the first two (2) years, quarters for the following three (3) years of production and then annually thereafter. There was also one quarter of preproduction mining;
- A detailed mining cost estimate was then developed utilising the various physicals derived from the LOM schedule and the rates from a selected RBP submission. This did not include Centamin’s mining team and related costs which were developed by Centamin.

The outcomes of the mining component of the FS were then provided to Centamin and the other contributing third-party consultants to be utilised in the development of the final Doropo FS financial model and subsequent economic evaluation of the Project.

16.2 Mining Method Selection

The Doropo Gold Project is a multi-pit project that consists of seven (7) deposits as shown in Figure 16.1.

They can be considered as:

- Three (3) deposits adjacent to the proposed processing facility comprising Souwa, Nokpa and Chegue Main/Chegue South. These are referred to as the “hub deposits” in this section of the report;

- Four (4) satellite pits, being Kekeda, Han, Enioda and Kilosegui, located respectively 6 km SE, 7 km E, 11.5 km E and 28 km SW of the processing facility. These are referred to as the “satellite deposits” in this section of the report.

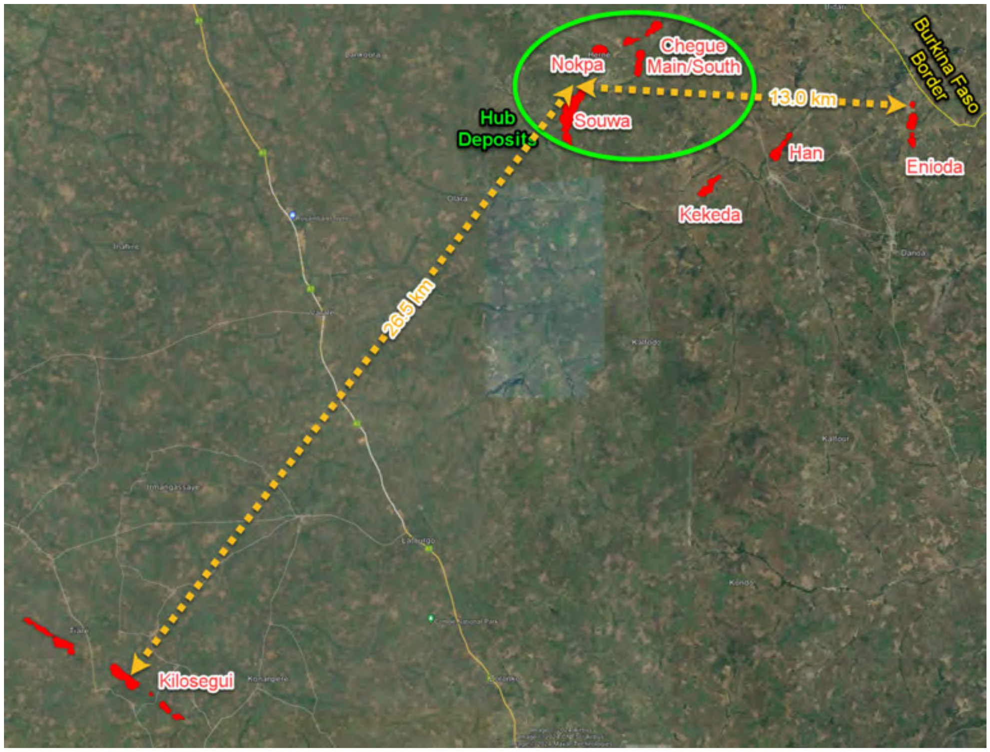


Figure 16.1 Location of Mining Areas

All open pits will be mined utilising a conventional truck and excavator approach undertaking a typical drill, blast, load and haul production cycle. Mining is based on all production related activities being undertaken by a suitably qualified and experienced mining contractor. These activities include, but are not limited to:

- Construction of all mining infrastructure required for undertaking mining operations such as workshops, warehousing, fuel and lubricants area, washdown facilities, administration building, crib rooms and ablutions etc.;
- Construction of all mine haul roads and ore haulage roads from the satellite pits to the processing plant;
- Site preparation for pits, WRD’s, roads etc. (i.e., clearing and grubbing of vegetation, removal and stockpiling of topsoil material);
- Primary production drilling and blasting;
- Primary production loading and hauling;
- Support of primary production activities with a suitable sized fleet of support equipment (i.e., bulldozers, graders, water trucks etc.);

- Pit-dewatering via in pit sump pumps;
- Establishment and maintenance of suitable surface water management infrastructure;
- Site rehabilitation and closure works such as re-profiling WRDs to final landform, rehandling and spreading stockpiled topsoil material etc.

Centamin will provide oversight and management of the mining contractors and undertake all technical requirements for the mining operation such as grade control, mine planning, mine surveying etc.

Mining of both ore and waste will be undertaken using an excavator (backhoe configuration) to load rigid body dump trucks. Drill and blast will be carried out on 5 m bench heights. It was assumed that 20% of the highly weathered oxide material will be free-dig (i.e., not require blasting) with a small amount of dozer ripping. The ore will be excavated in 2.5 m flitches to minimise dilution and ore loss. Waste will be mined in either 2.5 m flitches or at the 5 m blast height depending on the size of the excavator. The configuration for the mining fleet of the contractor submission on which the FS is based consists of:

- Up to 2 x 150-tonne class excavators;
- Up to 2 x 300-tonne class excavators;
- Up to 24 x 90-tonne payload rigid bodied dump trucks.

Waste will be placed in designated waste rock dumps (WRD's) adjacent to the open pits. Ore mined at the hub deposits will be transported directly to the central processing plant with the mining dump trucks. In the case of the satellite deposits, ore will be dumped to a Run-of-Mine (ROM) stockpile area immediately adjacent to the pit ramp exit for each satellite pit. From here it will be rehandled by a front end-loader to road type tipper trucks for transport to the process plant.

Ore arriving at the process plant from the hub pits will be either:

- Direct tipped into the primary crusher bin (80% of plant feed); or
- Placed on stockpile fingers on top of the primary crusher ROM pad (15% of plant feed); or
- Dumped to a low-grade stockpile (5% of plant feed) on natural ground level adjacent to the primary crusher ROM pad (designated stockpile material).

All ore arriving at the process plant from the satellite pits will be placed in a "satellite ore stockpile" on natural ground level adjacent to the low-grade stockpile.

It is assumed that all ore from the satellite ore and low-grade stockpiles will be reclaimed by front-end loader (or excavator) and hauled by mine trucks to the ROM Pad, where it will be direct tipped into the primary crusher bin.

Mining is scheduled such that the hub pits and up to two satellite pits can be in production concurrently.

It was assumed that the construction of other infrastructure such as process plant ROM stockpile pad, Tailings Storage Facility (TSF) embankments etc. is carried out by others. However, an allowance has been made for a "borrow" pit within the Souwa pit design for sourcing the quantity of bulk fill material required for this construction. This borrow pit has then been depleted from the Souwa mining inventory on the assumption it is excavated to completion prior to mining commencing.

Rehabilitation works will be carried out on an ongoing basis as areas of WRD's are finalised or infrastructure such as roads, stockpiles etc. are no longer required.

No consideration has been made for underground extensions of the operation in this FS.

16.3 Geotechnical Considerations

The FS geotechnical evaluation for the open pits was undertaken by SRK Consulting (UK) Ltd (Ref. 20).

As part of the feasibility assessment, a total of 8 holes of approximately 820 m total length were completed. These drillholes were designed for two purposes:

1. To infill geotechnical drilling and inform the geotechnical model; and
2. To collect samples for laboratory testing.

These are in addition to 33 drillholes completed for the 2023 PFS and 23 drillholes completed for the Scoping Study in 2018.

16.3.1 Modelling

Open pit stability will be controlled by shear strength of the rock mass within the weathered horizons and structurally controlled within the fresh rock. The geotechnical domain model considers the weathering model from a rock mass failure perspective. The objective of the geotechnical domain model is to provide a representation of the variability of the rock properties across the deposit area. The rock mass is variable by nature, and as such, a simplification of the parameters with similar units is required for analysis and design purposes.

16.3.2 Lithological Domains

Based on the regional geology context, the deposits have been sub-domained into three major areas: Doropo, Kilosegui, and Enioda (Figure 16.2). For the Doropo area, laboratory testing was completed at Chegue and Souwa. The lithologies considered for the geotechnical domain model include granodiorite, the dolerite dykes in Chegue South and Nokpa, and the mafic units at Enioda. Each domain was individually assessed for rock mass quality and rock strength as part of the geotechnical characterization work.

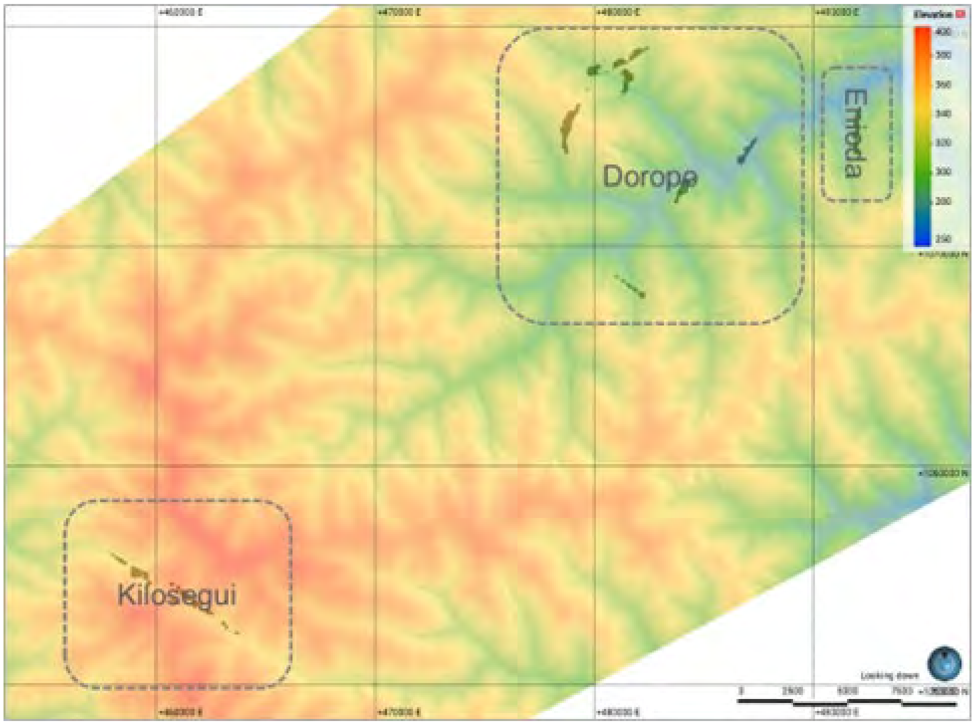


Figure 16.2 Major Lithological Domains

16.3.3 Weathering Domains

The geotechnical model primarily comprises the weathering domains modelled for each pit. There are six modelled weathering horizons (refer to Figure 16.3).

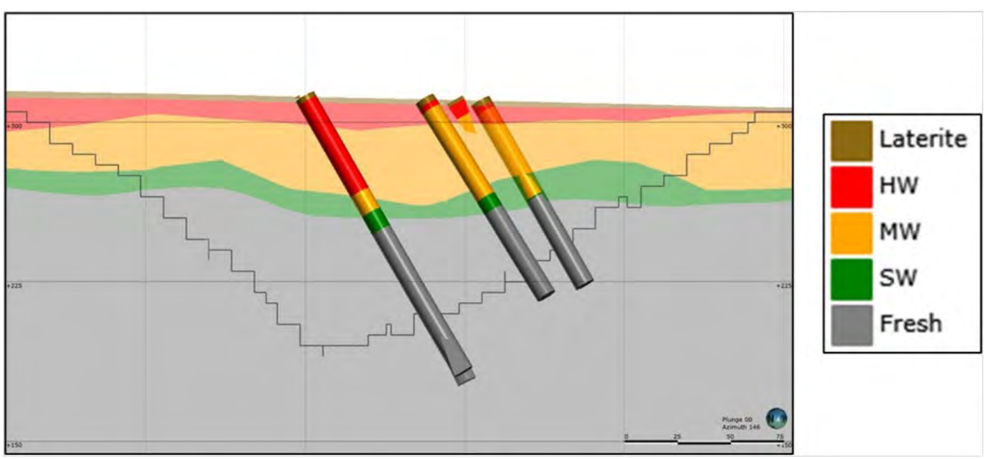


Figure 16.3 Geological Modelling of Weathering Zones

These have been modelled into three distinct weathering domains for design purposes as described in Table 16.1.

Weathering Horizons	Weathering Domain	Description	Rock Strength	Fracture Frequency
Completely Weathered (CW)	Saprolite/ Laterite	No constituent minerals are recognizable, except in part quartz grain (in the case of Doropo).	R0	40
Highly Weathered (HW)		Unconsolidated, some constituent minerals are visible including quartz which is abundant and some rare feldspars	R0 - R1	40
Moderately Weathered (MW)	Saprock	Consolidated although crumbly, only few minerals are recognizable, quartz and feldspars are little or no altered, but micas are almost completely altered (all rusty).	R0 - R2	20-40
Slightly Weathered (SW)		Consolidated, most of the minerals are recognizable, quartz and feldspars are not altered, but micas are partially altered (rusty).	R3 - R5	0 - 5
Fresh (FR)	Fresh	None of the constituents are altered.	R3 - R5	0 - 5

Table 16.1 Description of Geological Weathering Domains

Deposit-scale structures typically emulate regional-scale structures. Structural data, such as the orientation of faults, fractures and joints in drill holes, geotechnical logging data (RQD/recoveries) and gold grade interpolants, allow for the modelling of mineralised structures and structural discontinuities in the deposit, that can be linked to shear zones, faults, and cross-cutting dykes.

The orientations of the deposit-scale mineralised structures, interpreted as brittle-ductile shear zones and veins in the Doropo deposits, show variable trends from NE-SW, NW-SE, E-W to N-S. The dominant trends in Chegue, Enioda, Han, Kekeda, Nokpa and Souwa is N-S and NE-SW, whereas Kilosequi and Attire show NW-SE and WNW-ESE trends.

16.3.4 Slope Stability Assessment

Potential rock slope failure mechanisms that can influence the stability of the pit slopes need to be understood to develop safe and practical design recommendations. Rock slopes can generally be classified according to two principal failure mechanisms:

- Kinematically Controlled Failure Mechanisms: Structurally controlled failure in rock occurs as the result of sliding along pre-existing structures or discontinuities (e.g., sliding along foliation);
- Rock Mass Strength Failure: Slopes excavated in weak or heavily fractured rock masses, or very high slopes, can be susceptible to overall rock mass failure, which involves the development of step-path or pseudo-circular type failure zones through intact rock.

The bench and inter-ramp design for the pits were based on the following considerations:

- Saprolite stability will be controlled by the slope angles and influence of groundwater; and
- Slopes within the saprock and fresh rock domains will be structurally controlled, and the achievable slope geometries will be driven by the bench scale kinematics.

Two-dimensional limit equilibrium (LE) slope stability analysis was used to evaluate the expected rock and saprolite slope stability conditions. The LE analysis was conducted using Slide2D™. The stability analysis considered the potential for overall non-circular failures through the various units. Stability modelling was focused on evaluating the designs in the saprolite and saprock domains, as these will be the most sensitive to groundwater.

Figure 16.4 provides a diagrammatic layout of a pit wall slope with the design parameters defined.

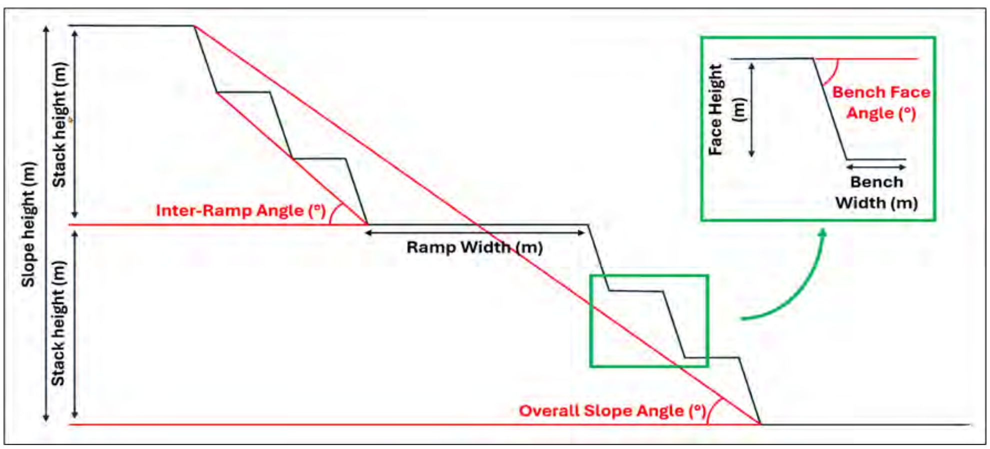


Figure 16.4 Pit Slope Design Parameters

16.3.5 Saprolite Design

Bench face design philosophy in saprolites represents a compromise between minimising erosion from surface water run-off (steeper bench faces more favourable) and maintaining stability (where shallower bench faces are more stable). It is assumed benches in saprolite will not require blasting (i.e.," free dig") and can be cut back to the desired bench face angle (BFA).

Bench and inter-ramp design in the Saprolite will be dependent on the overall thickness of the saprolite horizons and will be managed by increasing bench widths to reduce the Inter-Ramp Angle (IRA) as the thickness of the weathered domain increases. Bench face angles for the Saprolite will be set at 60° to encourage drainage of the faces. Berms should be graded at 2% to 3% into the pit to encourage sheet flow and surface water run-off, thus reducing erosion. The design recommendations for the saprolite domain are summarised in Table 16.2. Kilosegui and Enioda have their own slope geometries due to variations in thickness and orientation.

Domain	Thickness	Face Height	Berm Width	BFA	IRA	Stack Height	Factor of Safety	
	(m)	(m)	(m)	(°)	(°)	(m)	Hu = 0.8	Hu = 0.6
Doropo	< 20 m	5	4	60	36	20	1.6	1.6
	< 40 m	5	5	60	32.4	20	1.26	1.3
Enioda	All	5	5.5	60	30.8	20	1.3	1.36
Kilosegui	All	5	4.5	60	34.1	20	1.4	1.37

Table 16.2 Saprolite (CW & HW) Design Recommendations

Mining in the saprolite will be unpredictable at times, with local changes in material properties, groundwater conditions, or relict structures that may cause instabilities. Inter-ramp stack heights should be limited to 20 m, with a 15 m geotechnical berm placed between stacks. The base of the saprolite should also have a 15 m geotechnical berm before entering the saprock and fresh rock domains. These geotechnical berms serve two purposes:

1. To decouple the slopes and promote local depressurisation of the stacks; and
2. To enable access to the slopes for remediation and surface water management.

16.3.6 Saprock and Fresh Rock Design

The structural data was grouped into structural domains based on similar identified joint sets, with the Doropo area being split into two domains defined by a large, NW striking dolerite dyke:

- North (comprising Chegue Main, Chegue South and Han);
- South (comprising Souwa, Nokpa and Kekeda).

Enioda and Kilosegui remain independent domains, reflecting the different geological context of the deposit.

Bench design for the saprock and fresh rock domains is driven by kinematic instabilities and increasing bench widths to maintain suitable catchment. The achievable inter-ramp angles are a function of the bench geometries. The contact between the saprock and fresh rock is expected to be variable in terms of depths around the pits, and some flexibility will be required during implementation. The saprock designs will be limited to a bench height of 10 m to mitigate potential kinematic failures exacerbated by weathered fractures. The fresh rock will optimize to a 20 m bench height based on improved geotechnical conditions. The design recommendations for the saprock and fresh rock domain are summarised in Table 16.3.

There will be no geotechnical berms required for the current design configurations in the saprock and fresh rock domains. Pit depths within these units are less than a practical geotechnical berm spacing of 120 m. There is a requirement for a geotechnical berm at the top of the Saprock/base of saprolite of 15 m.

Domain	Weathering Domain	Azimuth From (°)	Azimuth To (°)	Bench Height (m)	Bench Width (m)	BFA (°)	IRA (°)	Stack Height (m)
Enioda	Saprock (MW/SW)	140	220	10	10	75	38.3	120
		220	140	10	6.5	75	47.4	120
	Fresh	100	140	20	10	75	52.5	120
		140	220	20	12	75	49	120
		220	100	20	8.5	75	55.3	120
Chegue Main/Chegue South/Han	Saprock (MW/SW)	40	320	10	6.5	75	47.4	120
		320	40	10	10	75	38.3	120
	Fresh	60	320	20	8.5	75	55.3	120
		320	60	20	13	75	47.4	120
Souwa/Nokpa/Kekeda	Saprock (MW/SW)	20	300	10	6.5	75	47.4	120
		300	20	10	8	75	43.1	120
	Fresh	20	300	20	8.5	75	55.3	120
		300	20	20	11	75	50.7	120
Kilosegui	Saprock (MW/SW)	100	340	10	6.5	75	47.4	120
		340	100	10	8	75	43.1	120
	Fresh	0	60	20	12	75	49	120
		60	120	20	10	75	52.5	120
		120	360	20	8.5	75	55.3	120

Table 16.3 Saprock and Fresh Rock (MW, SW & FR) Design Recommendations

16.3.7 Operational Considerations

The slope design guidelines are presented to global best-practice design standards.

Operational recommendations in regard to final wall slopes include:

- Saprolite cut slopes and partially blasted saprock slopes should be excavated utilising batter boards and guidance from the survey team to ensure design compliance;
- Convex slopes of one or more stack heights should be minimised in final designs to minimize the effects of unconfinement;
- Pits designed with single-ramp access should trigger a review of geotechnical hazards, slope and bench performance, and evaluation of possible risks to access.

Opportunities remain during mining to optimise the bench heights and berm widths. Specifically, the requirements for additional catchment in certain aspects of the design domains may be reduced should design compliance be maintained during implementation.

16.4 Mining Contractor Request for Budget Pricing

A Request for Budget Pricing (RBP) was developed in Q3 2023 based around the outcomes from the 2022 Preliminary Feasibility Study for the Doropo Gold Project.

The RBP was distributed to eight (8) reputable mining contacting groups with capability to undertake a project of this scale in Côte d'Ivoire, being:

- BCM International Ltd;
- Corica Mining Services - part of Diallo and Sons Group (DSG);
- Capdrill Mining;
- De Simone Group;
- EPSA Group;
- Mota-Engil Group;
- PW Mining International;
- Aertssen Group nv.

The scope of services for the mining contractor as detailed in the RBP was as follows:

- Site Establishment and supply of mining infrastructure required to undertake the scope of services, including:
  - Workshops for all mining and ancillary equipment and the Contractor light vehicle fleet;
  - Facilities for maintenance stores, offices, lubrication station, training facilities and ablutions including vehicle go-line area;
  - HV/LV wash-bay;
  - Bulk fuel storage facilities which will include the high flow (heavy vehicles) and low flow (light vehicles) refuelling bowsers;
  - Drainage and sumps for maintenance workshop, fuel storage and washdown facility;
  - Explosives magazine complying to relevant Côte d'Ivoire regulations;
  - Bulk explosives ANFO and emulsion facility (either fixed or mobile) complying with Côte d'Ivoire regulations;
  - Oil and waste oil separator/storage;
  - Any bulk earthworks required for the site establishment and construction of the mining areas;
  - Turkey's nests;
- Mobilisation and demobilisation of all plant, equipment and personnel;
- Clearing of vegetation and topsoil stripping of all mining areas of disturbance and storage of material in suitable stockpiles for later reclamation purposes;
- Construction of all ex-pit haul roads as follows:

- Souwa/Nokpa - roads for mine haul trucks to the Process Plant ROM pad and adjacent waste dumps;
- Han/Kekeda/Enioda/Kilosegui - roads for mine haul trucks to adjacent ROM stockpile and waste dumps. Road from local ROM stockpile to the Process Plant Rom Pad;

▪ Construction of Process Plant Rom Pad and LG S/P pads;
▪ Construction of the local ROM stockpiles where required;
▪ Sheeting of ore stockpile bases with suitable mineralised low-grade material;
▪ Drilling and blasting operations (including presplitting as required);
▪ Excavation of waste from open pits and hauling to designated waste locations;
▪ Excavation of ore from open pits and hauling to either the process plant ROM pad/crusher or the relevant local designated stockpiles;
▪ Rock breaking of oversize ore to a size suitable for loading to mine trucks and road haulage trucks;
▪ Rehandling by front end loader from the Process Plant ROM Pad stockpile fingers to the primary crusher;
▪ Reclaim by loader/road truck from the local ROM stockpiles to the Process Plant ROM Pad stockpile;
▪ Reclaim by loader/mine truck from the Low Grade (LG) stockpiles and satellite pit ore discharge area adjacent to the Process Plant ROM Pad stockpile direct to the crusher;
▪ Ongoing pit dewatering as required;
▪ Progressive mine closure:
  - Shaping and rehabilitation of waste rock dumps (WRD's);
  - Removal and rehabilitation mine roads and stockpile pads;
  - Demobilisation and site disestablishment of all infrastructure, equipment and personnel at mine closure;
▪ Côte D' Ivoire registration requirements;
▪ Relevant insurances (i.e., PL, WC, Equipment, MV etc.).

The following items were either excluded from the scope or to be provided by Centamin:

▪ Centamin will supply all fuel required for the purposes of fulfilling the mining scope at a cost of $1.00/l;
▪ Centamin will provide a demarcated area for the construction of the mining contractor's facilities adjacent to the process plant. Centamin will provide off-take points for power and potable water at a specified location. These utilities will be provided at no cost to the contractor;
▪ Accommodation for the Contractor personnel (including OEM personnel) will be provided by Centamin in the mine camp;

- The Contractor's submission should be based on Centamin providing on-site catering for the Contractors employees within Centamin's mine mess facility;
- Submission to exclude allowances for all relevant taxes and duties (i.e., WHT, VAT, importation costs etc.).

The RBP requested rates for the items detailed in Table 16.4.

Activity		Rate
Site establishment and infrastructure		$ (lump sum)
Mobilisation and demobilisation of equipment and plant		$ per unit
Management Fee		$/month
Fixed Costs and Overheads		$/month
Site Preparation	Clear & Grub	$/Ha
	Topsoil Stripping	$/bcm
	Topsoil Haulage	$/bcm/km
Rehabilitation	Topsoil placement	$/bcm
	Reprofiling	$/Ha
	Contouring/ripping & topsoil spreading (includes pads roads)	$/Ha
Haulroad Construction	Mine Truck	$/km
	Road Truck	$/km
ROM Crusher Feeding	FEL only, assume average 75 m from stockpile finger to crusher bin	$/t
Low Grade Stockpile Reclaim	FEL and Mine Truck, assume 300 m horiz. and 10 m vert. to crusher bin	$/t
Satellite ROM Stockpile Reclaim	Loading by FEL & Hauling (assumes tipper truck)	$/t
Drill and Blast	Oxide	$/bcm
	Transition	$/bcm
	Fresh	$/bcm
	Pre-splitting	$/LM drilled
Load and Haul	Ore by pit by bench to crusher or local ROM stockpile as appropriate	$/bcm
	Waste by pit by bench to local WRD	$/bcm

Table 16.4 Request for Budget Pricing Items

All groups approached provided submissions which afforded Centamin a useful dataset of costs to assess and determine suitable rates for the FS. The overall costs and associated $/t rates are mostly comparable to final costs for the FS as the RBP schedule was based on the final PFS schedule.

Figure 16.5 shows the spread of the total mining costs received on a $/t mined basis. The average of the costs received was $4.08/t mined. However, Contractors D & G appeared to be high cost outliers when compared to the other six (6) submissions. Therefore, when these two submissions were excluded, the average cost reduced to $3.71/t mined as shown in Table 16.5.

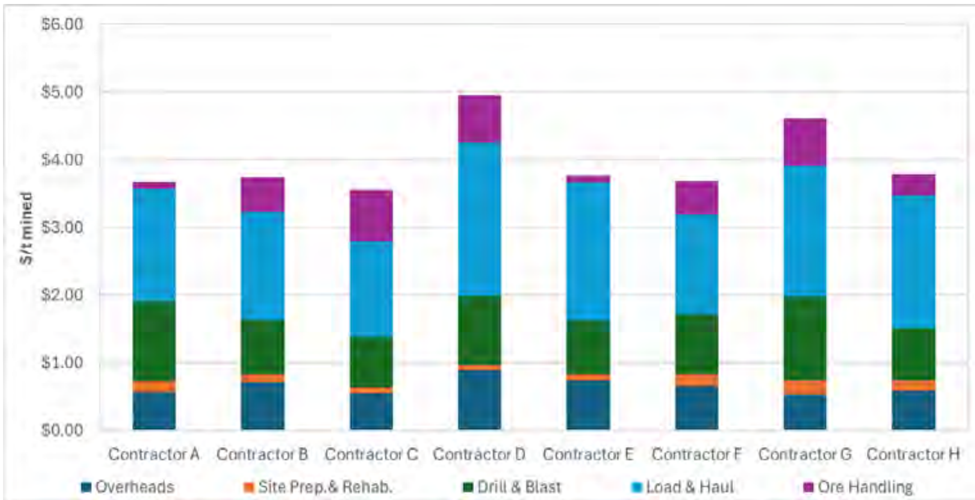


Figure 16.5 Comparison of November 2023 RBP Submission ($/t Mined Basis)

Cost Centre	Unit Cost ($/t mined)
Overheads (Fixed Fee, Mob/demob, Establishment)	$0.64
Site Prep & Rehabilitation (Clearing, topsoil stripping and replacement, dump profiling)	$0.13
Drill & Blast (incl Presplit)	$0.86
Load & Haul (ore & waste)	$1.69
Ore Handling (satellite pit haulage, LG S/P reclaim, crusher feed)	$0.39
Total	$3.71

Table 16.5 Average of Competitive Contractor RBP Submission ($/t Mined Basis)

16.5 Mining Dilution and Recovery Factors

16.5.1 Mining Block Model

The mine design and Mineral Reserve estimate for this FS was based on mineral resource models (MRM's) developed by Cube Consulting Pty Ltd (Cube) for their Doropo Mineral Resource Estimate (MRE) with an effective date of 31 October 2023. A total of six (6) models were provided in 2023, all in Surpac™ (.mdl) format. The Chegue (Main and South) and Nokpa deposits were provided in one model. Key block model parameters are outlined in Table 16.6.

As can be seen most of the MRM's are rotated to align the grid of the block model with the direction of strike of the orebody. While this is a commonly accepted approach to give better resolution of the boundary of the orebody, it should be noted that:

- In these cases, the X direction does not relate to East/West or the Y direction to North/South;
- In the combined Nokpa/Chegue model, neither deposit is aligned with model orientation.

Deposit	Range	Easting (m)	Northing (m)	Elevation (m)	X size (m)	Y size (m)	Z size (m)	Rotation (°)
Enioda (bm_eni_smu_202206)	Min	491,250	1,074,000	-50	5	5	2.5	0
	Max	492,510	1,076,600	400	20	40	10	
Han (bm_han_smu_20220628)	Min	485,550	1,073,780	-50	5	5	2.5	35
	Max	486,820	1,076,080	400	20	20	5	
Kekeda (bm_kek_smu_20220601)	Min	483,000	1,072,030	-50	5	5	2.5	35
	Max	484,120	1,074,080	400	20	20	5	
Kilosegui (bm_klg_smu_202207)	Min	464,260	1,050,710	0	5	5	2.5	-60
	Max	466,850	1,060,000	450	20	20	5	
Nokpa (bm_nok_chg_smu_20220601)	Min	478,370	1,077,200	-200	5	5	2.5	35
	Max	481,690	1,081,670	450	20	20	5	
Souwa (bm_swa_smu_20220601)	Min	478,000	1,074,000	-50	5	5	2.5	0
	Max	479,550	1,076,700	450	20	40	10	

Table 16.6 Resource Block Model Parameters

Orelogy used an in-house procedure to convert the MRM's to a Mining Block Model (MBM) in a format suitable for Hexagon MineSight™, the general mining package (GMP) used by Orelogy:

- Export MRM to a comma separated value (csv) text file and clean-up by replacing null, non-number, and negative numbers with zero;
- Convert to a regularised block model suitable for mine planning purposes. MineSight™ preserves an ore parcel inside each regularised block to maintain the granularity of the underlying resource model. An approximate economic cut-off grade was used for each deposit to define which resource blocks were considered as ore parcels. The regular block size selected was 10 m (X) by 10 m (Y) by 5 m (Z) with a single ore parcel. This was considered an appropriate Selective Mining Unit (SMU) size for the mining model;
- Code classification and weathering flags on a majority volume basis. This can result in a degree of averaging or "smearing" of these items. Therefore, the classification and weathering fields of the re-blocked model were coded with the original classification where required to reconcile the proportional model to the original MRM's and eliminate this effect;

- Accumulate contained metal to ensure exact reconciliation with the original MRM's;
- Intersect RBM with the topography surface to assign a topo %. Only the proportion of the block below the surface was reported.

The MBM's reconciled back to the original resource models with a variance of less than 0.25% in tonnes and ounces after the re-blocking and reclassification process, with the exception being Kekeda with an increase of 0.5% in ore tonnes. The Nokpa, Chegue and Chegue South areas were individually coded in the combined block model to allow for optimisation and reporting of results on a per deposit basis.

16.5.2 Dilution and Ore Loss

Orebody dilution is the result of waste or sub-grade material being excavated with ore during the process of mining, increasing the tonnes but lowering the grade of the ore delivered as process feed.

Ore loss may result from a combination of:

- Inaccuracy in locating the ore/waste boundary from grade control;
- Errors in ore block set-out;
- Errors in ore control (ore spotting);
- Poor accuracy of excavation along the identified boundary;
- Ore being misdirected to the wrong destination;
- Material being diluted to below cut-off grade.

The Mining Block Model resulting from the process described in Section 16.5.1 does not include any allowance for either dilution or mining recovery, with the exception of internal dilution added during the sample compositing process used for resource estimation. As part of the development of a credible and robust mine plan, dilution and mining recovery must be considered.

Of the many methods that can be used to estimate mining dilution and recovery, the application of edge dilution is considered most suitable to these deposits, given the selective mining method proposed. Orelogy assessed this effect by applying a dilution/ore loss allowance along the "edge blocks" in the resource model, based upon an initial ore/waste cut-off assumption. Orelogy have in-house routines that can apply the edge effect in three possible ways:

- Dilution Only - assumes ore blocks are preferentially over-excavated and dilution is accepted to minimise ore loss;
- Ore Loss Only - assumes ore blocks are preferentially under-excavated and ore loss is accepted to minimise dilution;
- Mixed Ore Loss/Dilution (equal transfer of material in and out of ore parcel).

The dilution only option was selected as the preferred option for these deposits. This is due to the high negative impact of any ore loss in narrow, structurally controlled deposits.

A 1.0 m skin width was selected to represent the selectivity that is assumed to be achievable by the excavator. The dilution skin was applied to identified edge blocks assuming a zero-diluent grade (i.e., diluent material contributed no metal content).

The dilution script was applied to the regularised Mining Block Models (MBMs) and the following steps undertaken:

1. The code runs across the block model strike from both directions and identifies the edge blocks (i.e. blocks with an ore percent and an adjacent block that is 100% waste). It also separately identifies isolated blocks that have 100% waste blocks on both sides;
2. On a section-by-section basis the script added diluent material to the ore percent in any identified edge block or isolated block up to a maximum ore percent of 100 with a consequent reduction in waste percent. The contained metal remains unchanged and therefore the grade was reduced. The percentage of diluent applied was dependent on the apparent skin width [e.g. For a 10 m x 10 m block the assumed 1.0 m of dilution "skin" equates to 10% additional diluent material to be added to the ore percent, i.e. (1.0 x 10)/(10 x 10);
3. The MBMs were then re-reported at an appropriate cut-off grade which resulted in ore loss due to material being diluted below cut-off grade.

Table 16.7 summarises the parameters used in the dilution script for each MBM.

Area	Orebody dip (°)	Skin width (m)	Dilution Axis	Ore Definition (g/t)
Enioda	60-55	1.0	E-W	0.40
Han	60	1.0	E-W	0.40
Kekeda	60-55	1.0	E-W	0.40
Kilosegui	60	1.0	E-W	0.50
Nokpa	75-55	1.0	E-W	0.35
Souwa	65	1.0	E-W	0.35

Table 16.7 Dilution Parameters by Orebody

The dilution and ore loss from RBM and MBM are shown by area in Table 16.8, where it can be seen that dilution and ore loss vary widely. In general, the lower the deposit grade, the lower the dilution and the higher the ore loss. This is because a higher proportion of the edge diluted blocks are diluted below the cut-off grade and therefore "lost", whereas the higher-grade deposits can wear more dilution from losing ore below cut-off. However, as the dilution approach is locational, differences in geometries and grade continuity also has a considerable influence.

Table 16.8 shows that an overall average of approximately 5% dilution and 6% ore loss is generated.

Area	Undiluted Ore		Diluted Ore		Dilution	Ore Loss
	(Mt)	(g/t)	(Mt)	(g/t)	(%)	(%)
ENI	15.61	0.85	*15.75*	*0.80*	*6.5%*	*5.2%*
HAN	9.90	1.24	*9.89*	*1.19*	*3.9%*	*3.8%*
KEK	11.01	0.77	*10.65*	*0.74*	*4.1%*	*7.0%*
KLG	64.27	0.93	*59.74*	*0.92*	*1.3%*	*8.3%*
NOK	51.30	1.01	*54.63*	*0.92*	*9.7%*	*2.9%*
SWA	29.72	1.18	*32.47*	*1.06*	*10.9%*	*1.5%*
Total	181.80	0.99	183.12	0.94	4.8%	5.8%

Table 16.8 Approximate Ore Loss and Dilution

16.6 Open Pit Optimisation

16.6.1 Open Pit Optimisation Approach

The first stage of the conversion of a Mineral Resource into a mineable open pit Mineral Reserve is the open pit optimisation process. It is at this stage that all the latest physical, technical, and economic parameters are applied to the orebody to determine the "ideal" open pit excavation geometry.

If the economics of this "ideal" pit geometry (shell) are favourable, the shell can then be used as a guide in the subsequent pit design process.

The WHITTLE™ open pit optimisation software tool was used by Orelogy to undertake this component of the study. WHITTLE™ is an industry recognised pit optimisation package.

The term "ore" is used in the following sections describing the optimisation process. It is used to describe mineralised material that the optimisation considered potentially economic. It should not be confused with the stricter definition of economically extractable material as denoted by ore in a "Mineral Reserve".

In broad terms, the process that WHITTLE™ undertakes is to vary the base input price by a range of factors (referred to as the Revenue Factors), up and down from a base value of 1. For any given Revenue Factor WHITTLE™ then produces a three-dimensional shape, or "shell", that generates the maximum possible value for all the input parameters and the associated factored price.

Lower Revenue Factors (i.e., lower price) will produce smaller shells; the higher Revenue Factor, the larger the shell. This results in a set of "nested" shells, with each shell lying inside the shell of the next largest Revenue Factor.

The value of each shell is then reported at the original input price of the optimisation. The effect of this is:

- Shells with a Revenue Factor <1 are smaller and have less ore than the shell with Revenue Factor 1. This reduces the revenue generated and therefore they will have a lower value;
- Shells with a Revenue Factor >1 are bigger and have relatively more waste (i.e., higher strip ratio). This increases the costs and therefore they will have a lower value;
- The result is the classic WHITTLE™ cash flow curve that peaks at the base price (i.e., Revenue Factor 1). The robustness of this shell is reflected in how quickly the value falls away either side of this peak.

These nested shells are important for a number of reasons:

- The smaller shells indicate the areas of highest value in the ore body and therefore give a guide as to where mining should commence;
- The larger shells provide an indication of how much additional mineralisation may become economic, or alternatively what current ore may become unviable, should input parameters (assumptions) change in the future;
- They permit WHITTLE™ to develop a theoretical "schedule" for mining the deposit over time and therefore allow a Discounted Cash Flow (DCF) to be generated.

It is important to have some appreciation of how WHITTLE™ generates a DCF as these values are used in this study as a guide to both sensitivity and the shell selected as the basis for pit design. The WHITTLE™ cash flows differ from accounting cash flows which include the effects of capital debt servicing, depreciation, and taxation.

WHITTLE™ generates the following two standard DCF's:

Worst Case DCF assumes that, for any given shell, extraction is undertaken bench by bench sequentially from the top to bottom of the pit. This results in a significant amount of any overlying overburden being removed prior to presentation of ore (i.e. a large pre-strip), and there are no initial shells or stages that access higher value ore earlier in the schedule. This is clearly not a preferred pit extraction scenario from a value perspective.

Best Case DCF assumes that, for any given shell, extraction is undertaken sequentially from the smallest shell generated by the optimisation, through all the intervening shells out to the shell selected. This approach focuses on the highest value areas of the orebody and generates the least amount of waste stripping. As such it provides the highest DCF for the selected shell. However, this approach is constrained by practical considerations as it is effectively mining the orebody using an impractically high number of small incremental pushbacks that may not be achieved in reality as:

- The distances between successive shells are often too narrow to mine practically;

- The bench turnover rate required to mine the shell in the time frame is not achievable;
- The base of the shell is too small to allow for practical access or safe extraction.

Figure 16.6 outlines a schematic representation of both the Worst Case DCF mining sequence and the Best Case DCF mining sequence.

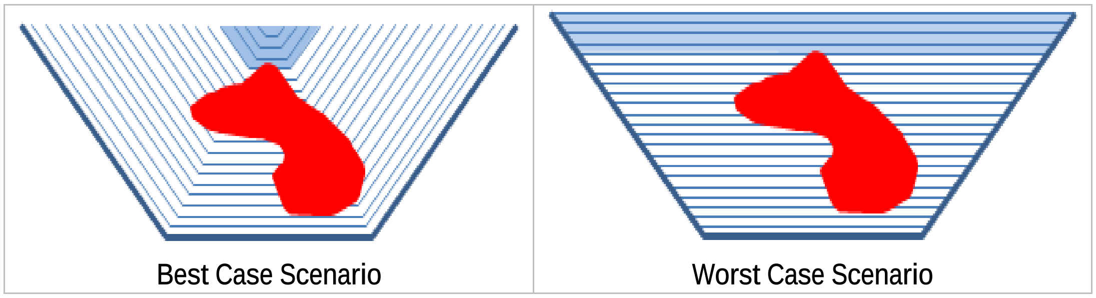


Figure 16.6 Best & Worst Case DCF Mining Sequence (Cross-Section View)

It clearly shows that:

- The Best Case scenario will provide the lowest strip ratio/highest value schedule but will also invariably be unachievable due to the shells being too close together;
- The Worst Case scenario will generate the lowest value due to not targeting the ore and therefore mining a significant amount of unnecessary waste.

Hence these approaches generate two divergent cash flows that provide the extremities of the theoretical value that can be generated from a deposit. The schedule and associated cashflow that represents what will happen in reality will in fact lie somewhere between these two endpoints.

It is important to note that the cashflows reported by WHITTLE™, both discounted and undiscounted, will differ from those in the project's financial model as the optimisation:

- In general, excludes capital and financing costs;
- Does not account for the effects of taxation;
- Is based on a high-level annual mining schedule.

The following sections detail the parameters used for the final shell selection optimisation on which subsequent pit designs were based.

16.6.2 Optimisation Modifying Factors

Wall Slopes

Wall slope parameters were derived from the parameters provided by SRK Consulting (SRK) in the document "32094 - Pit Designs Summary_Draft_20240326". (SRK Consulting (UK) Limited, 2024). The report categorised the geotechnical parameters into distinct domains for Saprolite (oxide), Saprock (transition) and Fresh material types. Saprolite (oxide) was classified in the model as being Completely Weathered (CW) or Heavily Weathered (HW) and Saprock (transition) was Moderately Weathered (MW) or Slightly Weathered (SW). Fresh rock remained classified as Fresh.

The geotechnical parameters for Saprolite (oxide) zones and Saprock (transition) and Fresh zones are outlined below in Table 16.9 and Table 16.10 respectively.

Domain	Azimuth (°)		Bench Height	Berm Width	Batter Angle	Inter-ramp Angle
	From	To	(m)	(m)	(°)	(°)
Enioda	0	360	5	5.5	60	30.8
Doropo < 20 m	0	360	5	4	60	36
Doropo < 40 m	0	360	5	5	60	32.4
Kilosegui	0	360	5	4.5	60	34.1

Table 16.9 Saprolite (Oxide) Weathering Zones Geotechnical Parameters

Domain	Weathering	Azimuth (°)		Bench Height	Berm Width	Batter Angle	Inter-ramp Angle
		From	To	(m)	(m)	(°)	(°)
Enioda	Saprock	140	220	10	10	75	38.3
Enioda	Saprock	220	140	10	6.5	75	47.4
Enioda	Fresh	100	140	20	10	75	52.5
Enioda	Fresh	140	220	20	12	75	49
Enioda	Fresh	220	100	20	8.5	75	55.3
North	Saprock	40	320	10	6.5	75	47.4
North	Saprock	320	40	10	10	75	38.3
North	Fresh	60	320	20	8.5	75	55.3
North	Fresh	320	60	20	13	75	47.4
South	Saprock	20	300	10	6.5	75	47.4
South	Saprock	300	20	10	8	75	43.1
South	Fresh	20	300	20	8.5	75	55.3
South	Fresh	300	20	20	11	75	50.7
Kilosegui	Saprock	100	340	10	6.5	75	47.4
Kilosegui	Saprock	340	100	10	8	75	43.1

Domain	Weathering	Azimuth (°)		Bench Height	Berm Width	Batter Angle	Inter-ramp Angle
		From	To	(m)	(m)	(°)	(°)
Kilosegui	Fresh	0	60	20	12	75	49
Kilosegui	Fresh	60	120	20	10	75	52.5
Kilosegui	Fresh	120	360	20	8.5	75	55.3

Table 16.10 Saprock (Transition) and Fresh Weathering Zones Geotechnical Parameters

In Table 16.10, the North domain includes the Chegue (Main and South) and Han pits. While the South domain covers the Nokpa, Souwa and Kekeda pits (refer to Section 16.3.6).

Geotech berms of 15 m width are included every 20 m in Saprolite (oxide) material and at the base of the Saprolite weathering zone. The ramp can serve as the geotechnical berm.

These Inter-ramp Angles are then used in Whittle, with an appropriate adjustment to slope angle for ramp locations, to run the optimisation. The pit designs from the PFS were used as a guide to ramp locations when developing the overall slopes. Souwa is a special case as a more detailed breakdown of ramps and slopes was conducted which included modelling overall slope angles in various sections of the pit.

Table 16.11 outlines the overall slope angles used for the optimisation in Whittle.

Mining Area	Orientation	Slope Angle (°)		
		Oxide	Trans	Fresh
Enioda	Footwall I	24.7	30.2	37
	Hanging wall (W)	24.7	25.8	49
Han	Footwall I	14.8	21.7	50.4
	Hanging wall (W)	14.8	28.2	48.1
Kekeda	Footwall I	36	34.6	39.2
	Hanging wall (W)	36	32	51.1
Kilosegui	Footwall I	17.5	34.6	35.5
	Hanging wall (W)	11.8	32	52.5
Nokpa	Footwall I	21.2	47.4	51
	Hanging wall (W)	21.2	23.8	46.9
Chegue South	Hanging wall I	24.7	29	46.2
	Footwall (W)	24.7	38.3	53.1
Chegue Main	Footwall I	21.2	25.8	40.2
	Hanging wall (W)	21.2	38.6	46.2
Souwa	Various	Various		

Table 16.11 Slope Angles by Mining Area used in the Whittle Optimisation Process

Mining Costs

The LG algorithm in Whittle determines whether a block is ore or waste at the pit crest based on the potential cash flow that can be generated from the block at that point. For this reason, the mining cost applied to a block is based on the assumption that each block is waste.

Mining costs used in the optimisation were derived from the first round of submissions (Oct 2023) to the Mining Contractor Request for Budget Pricing discussed in Section 16.4. The waste mining costs were variable by location (i.e. load and haul rates by bench by pit) and by material type (i.e., drill and blast costs by material type). They were then combined and coded into the individual block models for each mining area by bench and/or by material type as appropriate.

Other mining costs generated as part of the RBP submissions were applied as a global $/bcm cost. The resulting average mining costs by deposit are detailed in Table 16.12.

Deposit	$/t
Enioda	$3.07
Han	$3.24
Kekeda	$3.03
Kilosegui	$3.35
Nokpa	$3.23
Chegue South	$2.95
Chegue Main	$2.95
Souwa	$3.16

Table 16.12 Average Waste Mining Cost by Deposit

Process Recoveries

Updated processing recovery data was provided by Centamin. These were classified by:

- Location - i.e. deposit;
- Weathering - oxide, transition and fresh material types;
- Grade - three grade “bins” (≥0.25 g/t to < 0.5 g/t, ≥0.50 g/t to < 1.0 g/t and > 1.0 g/t).

There were still some areas with 0% recovery due to insufficient metallurgical testing, being:

- Oxide recovery for all grades in Han and Chegue South;
- Oxide recovery for grades below 1.0 g/t, in Nokpa.

Table 16.13 shows all the process recoveries used.

Mining Area	Material Type	Grade Range		
		≥0.25 g/t < 0.5 g/t	**≥0.50 g/t < 1.0 g/t**	> 1.0 g/t
Souwa	Oxide	92.0%	93.9%	95.0%
	Transition	88.0%	92.1%	94.4%
	Fresh	78.5%	80.7%	87.6%
Kilosegui	Oxide	90.0%	94.0%	97.7%
	Transition	88.0%	89.3%	92.0%
	Fresh	78.5%	80.7%	85.5%
Nokpa	Oxide			93.1%
	Transition	93.3%	94.0%	95.1%
	Fresh	78.5%	80.7%	89.0%
Chegue Main	Oxide	92.0%	94.0%	95.7%
	Transition	88.0%	89.3%	92.7%
	Fresh	78.5%	80.7%	89.7%
Cheque South	Oxide			
	Transition	88.0%	89.3%	89.9%
	Fresh	78.5%	80.7%	81.9%
Kekeda	Oxide	90.0%	94.0%	96.0%
	Transition	88.0%	89.3%	93.7%
	Fresh	78.5%	80.7%	88.3%
Han	Oxide			
	Transition	88.0%	89.3%	92.2%
	Fresh	78.5%	80.7%	88.8%
Enioda	Oxide	90.0%	94.0%	96.8%
	Transition	88.0%	89.3%	93.9%
	Fresh	78.5%	80.7%	92.7%

Table 16.13 Process Recoveries

Process Cost

Processing costs applied in Whittle are a combination of processing costs, fixed G&A costs and any ore related mining costs. As such it can be considered an all-in Ore Cost as opposed to purely a processing cost. The costs used are detailed in Table 16.14.

Parameter		Unit	Value
Production Rates	Oxide	Mt/a	5.4
	Transition	Mt/a	5.4
	Fresh	Mt/a	4
Ore Related Mining Costs	ROM Rehandle	US$/dmt	$0.62
	Grade Control	US$/dmt	$0.50
	Owners Team	M US$/year	$3.39
		US$/dmt Oxide	$0.63
		US$/dmt Trans	$0.63
		US$/dmt Fresh	$0.85
Processing Cost		US$/dmt Oxide	$11.43
		US$/dmt Trans	$11.26
		US$/dmt Fresh	$14.35
G&A Costs		US$/dmt Oxide	$3.81
		US$/dmt Trans	$3.81
		US$/dmt Fresh	$5.14
Total Processing Cost		US$/dmt Oxide	$15.24
		US$/dmt Trans	$15.07
		US$/dmt Fresh	$19.49

Table 16.14 Ore Cost

There is also an intrinsic cost differential between mining ore and mining waste which is generally related to either variations in haulage distance or drill and blast parameters for ore. The difference between mining ore and mining waste is referred to as the Ore Mining Premium (OMP) and was coded into the block model on a mining area and bench basis and was then also added to the Whittle “processing cost”.

Ore Haulage Costs

Ore from the satellite pits will be placed on ROM stockpiles adjacent to the pits, and then rehandled through to the process plant using road type tipper trucks. The cost for this rehandle component was also applied in Whittle as an additional ore cost for these mining areas. Based on a loading cost of $0.70/t and a haulage cost of $0.15/t/km, the total haulage costs by mining area are detailed in Table 16.15.

Mining Area	Haulage Distance (km)	Cost (US$/t)
Enioda	15.00	$2.95
Han	12.00	$2.50
Kekeda	9.50	$2.13
Kilosegui	32.00	$5.50

Table 16.15 Satellite Ore Haulage Costs

Capital Cost

Capital costs were not applied in Whittle. All the deposits were optimised separately and, as there was no definitive scheduling sequence at this point in time, the capital could not be correctly distributed across the different mining areas.

The capital component in Whittle is applied as an upfront cost and therefore does not attract discounting. Consequently, the effect on shell value from capital is the same across all shells in relative terms and therefore does not inherently affect the basis for shell selection.

Revenue and Financial

Revenue inputs are outlined in Table 16.16. Gold price was assumed at a relatively conservative $1,450 US$/oz to ensure it remained consistent with Centamin corporate reporting. Selling costs include government and tenement royalties of 4% and 0.5%, respectively. Transport and refining charges were $4.00 US$/oz with a 0.05% refining ore loss.

A 5% discount rate was used for NPV calculations, as directed by Centamin.

Parameter		Basis	Value
Revenue	Gold price	US$/oz	$1,450
Selling Cost	Govt. Royalty	4% of sales	$58.00
	Tenement Royalty	0.5% of sales	$7.25
	Social Fund Royalty	0.5% of sales	$7.25
	Gold Loss	0.05% of sales	$0.725
	Dore Transport Cost	US$/oz	$4.00
Net Price		US$/oz	$1,372.78
		US$/gram	$44.14

Table 16.16 Product Revenue Parameters

16.6.3 Optimisation Results and Shell Selection

Optimisation scenarios were run on Measured and Indicated material only.

For most of the mining areas, the shells are relatively small and provide less than one year of plant feed. As this feed rate is the basis of the schedule used for calculation of the Whittle DCF, there is effectively no discounting and therefore no difference between Best Case and Worst Case DCF's for these areas. Also, in many cases, the plot of the DCF is relatively flat across several revenue factors. Given the margin of error of the optimisation process, these optimisation shells are essentially the same and this allows for selection of a larger (or smaller) shell for, essentially, the same DCF result.

Table 16.17 outlines the overall inventory from the selected optimisation shells in each mining area.

Pit	Shell #	Revenue Factor	Equiv. Price	Physicals						Financials (Undiscounted)					DCF		Strip Ratio	Cost / Oz.
				Ore				Waste	Total	Mining Cost	Ore Cost	Selling Costs	Rev.	Cash flow	Worst Case	Best Case		
			US$/oz	Mt	g/t	Cont. koz	Rec. koz	Mt	Mt	($M)	($M)	($M)	($M)	($M)	($M)	($M)	W:O	($/Oz)
Enioda	28	1.04	1508	11.68	1.61	603.8	547.6	48.3	60.0	-$43	-$43	-$7	$126	$33	$32	$32	6.0	1,075
Han	30	1.08	1566	12.00	1.42	548.8	475.6	47.0	59.0	-$61	-$72	-$17	$296	$146	$141	$141	4.0	735
Kekeda	29	1.06	1537	3.16	2.19	222.6	204.0	15.6	18.8	-$31	-$46	-$7	$122	$38	$37	$37	3.5	999
Kilosegui	28	1.04	1508	2.29	1.25	91.7	84.3	7.9	10.2	-$198	-$299	-$40	$690	$153	$136	$143	3.9	1,128
Nokpa	27	1.02	1479	2.97	2.00	190.2	167.0	22.2	25.2	-$81	-$59	-$14	$242	$88	$85	$85	7.5	925
Chegue South	26	1.04	1508	2.82	1.12	101.3	92.9	12.8	15.7	-$27	-$23	-$4	$68	$15	$15	$15	6.8	1,137
Chegue Main	29	1.06	1537	1.18	1.41	53.5	47.2	8.0	9.2	-$46	-$50	-$8	$135	$30	$30	$30	4.6	1,123
Souwa	29	1.06	1537	2.01	1.42	91.6	86.6	12.0	14.0	-$190	-$214	-$46	$794	$343	$315	$326	4.1	823
Total				38.10	1.55	1.903.6	1.705.1	173.9	212.0	-$677	-$806	-$144	$2,472	$846	N/A	N/A	4.6	954

Table 16.17 Optimisation Results by Mining Area

Souwa

The Souwa optimisation results are presented graphically in Figure 16.7 and Figure 16.8. Figure 16.7 displays tonnes (ore and waste) and discounted cash flow (DCF). The DCF's shown are the Whittle Worst Case, Best Case and the Average Case (refer to Section 16.6.1). As can be seen, all the DCFs are virtually flat from the Max. Worst Case DCF (Shell 23) to Max. Best Case DCF (Shell 26). The Worst Case DCF only varies by 0.5% from Shell 20 to 29, which is effectively the same at the level of accuracy of the optimisation.

Figure 16.8 provides the strip ratio and cost per oz. for each shell, both as a total and as an incremental change from the previous shell. There are no significant hinge points or step changes in the total curves for each KPI, which reflects the very flat DCF curves.

Given the relatively conservative US$1,450/oz base price used, and the fact that the Worst Case DCF curve is so flat, it is considered justified to push the ultimate shell selection past Revenue Factor 1 in order to maximise resource conversion. Consequently Shell 29 (revenue factor 1.06) was selected as the basis for pit designs as there is little risk to overall DCF by selecting this shell.

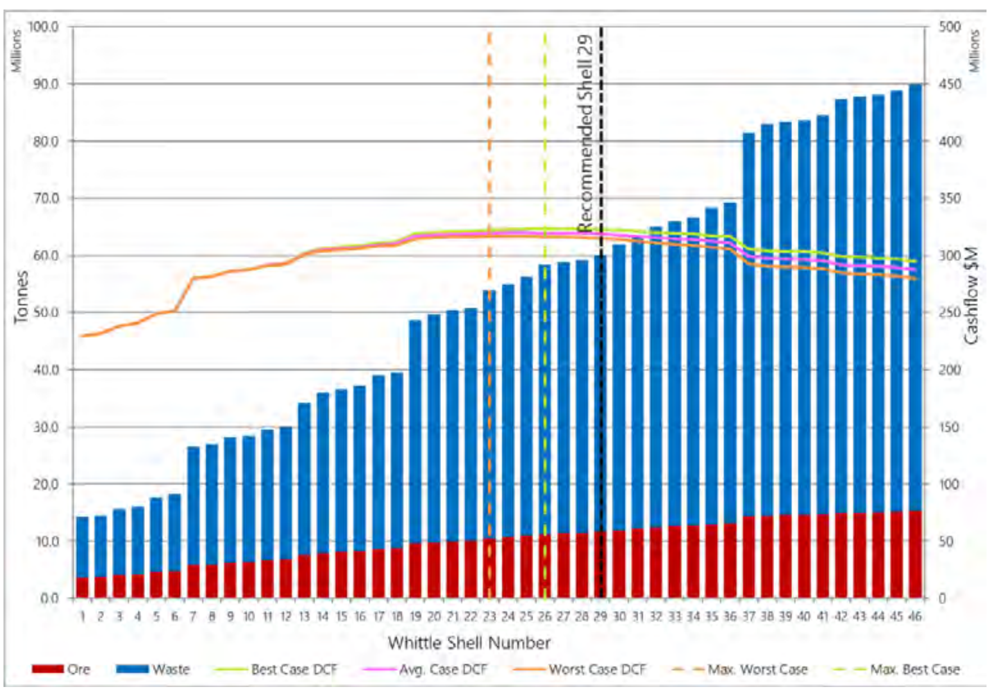


Figure 16.7 Souwa Pit Optimisation - Tonne/Value Chart

The Worst Case DCF for Shell 29 is only 0.5% lower than the maximum Worst Case DCF (Shell 23) and 2.5% lower than the maximum Best Case DCF (Shell 26), yet there is almost 6% additional recovered ounces generated.

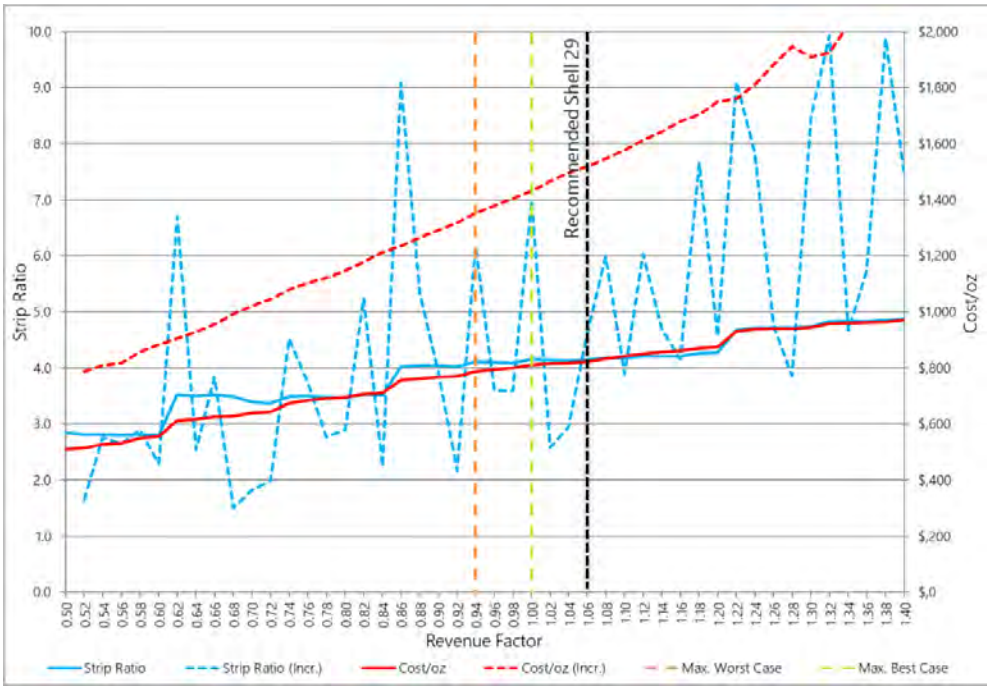


Figure 16.8 Souwa Pit Optimisation - Strip Ratio and Cost Per Oz

Kilosegui

The Kilosegui optimisation results are presented graphically Figure 16.9 and Figure 16.10. As can be seen, the DCF curves for Kilosegui are much steeper either side of the peak, with the Max. Best Case and Max. Worst Case DCF shells (25 and 26 respectively) separated by only a 0.02 Revenue Factor increment.

While the DCF curves are not as flat as those generated for Souwa, the low price does allow some flexibility in shell selection. Shell 28 at a revenue factor of 1.04, generates 7% additional recovered ounces than the Maximum Worst Case shell for only a 1.5% drop in Worst Case DCF and 0.5% drop in Best Case DCF. Therefore, Shell 28 was selected as appropriate for design purposes.

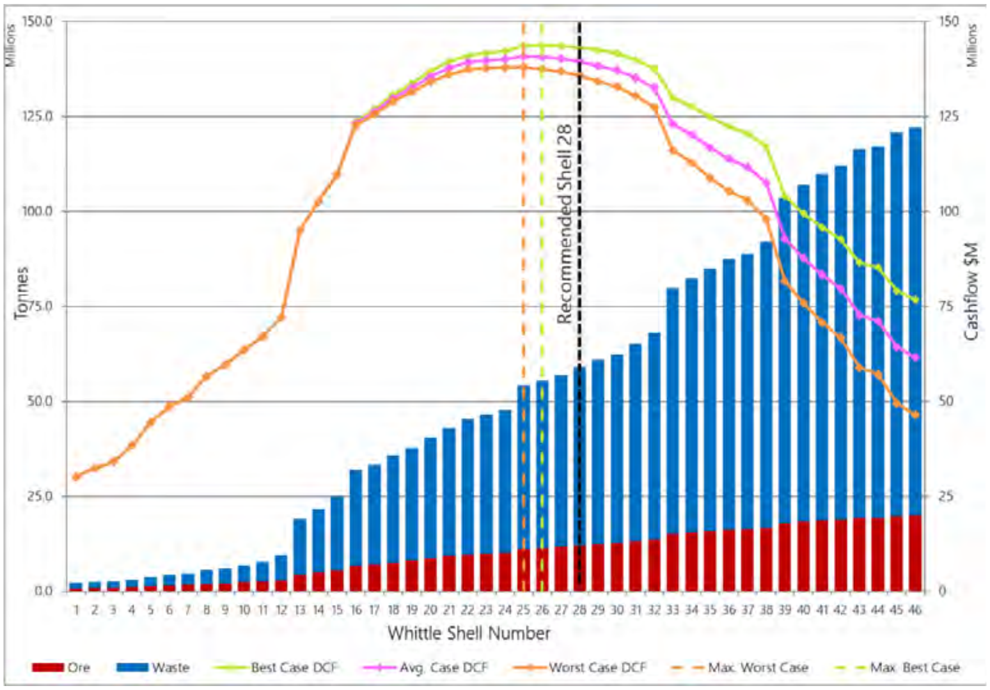


Figure 16.9 Kilosegui Pit Optimisation - Tonne/Value Chart

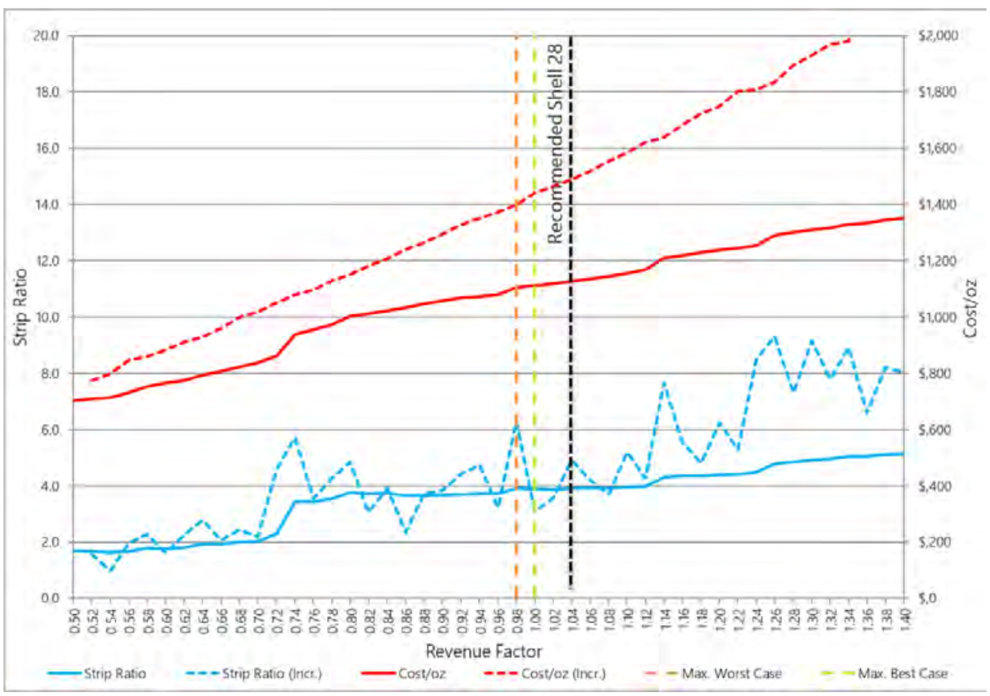


Figure 16.10 Kilosegui Pit Optimisation - Strip Ratio and Cost Per Oz

Han

The Han optimisation results are presented graphically in Figure 16.11 and Figure 16.12. The DCF curves for Han are very flat, with the variation in DCF varying by less than 0.5% from Shell 18 to Shell 30 (revenue factor 0.84 to 1.08 respectively). Selecting Shell 30 (Revenue Factor 1.08) is justified as it is within 0.5% of the maximum DCF and offers a 5% increase in recovered ounces.

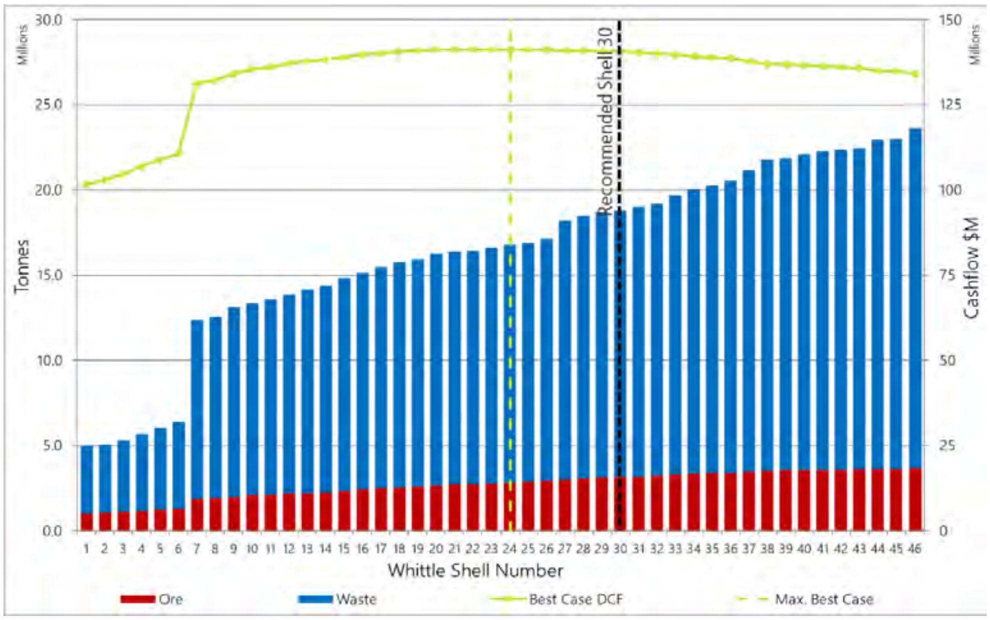


Figure 16.11 Han Pit Optimisation - Tonne/Value Chart

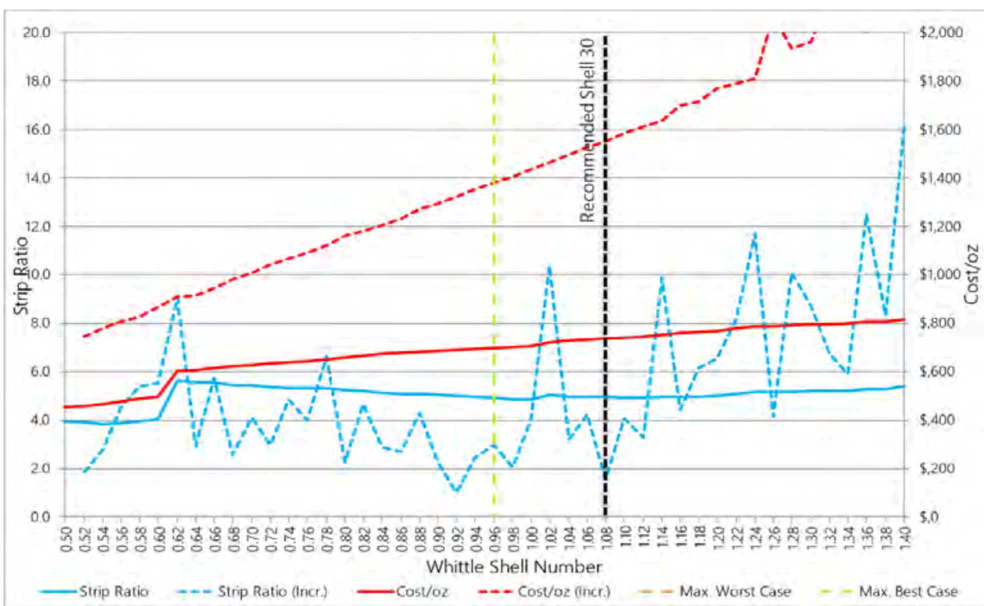


Figure 16.12 Han Pit Optimisation - Strip Ratio and Cost Per Oz

Kekeda

The Kekeda optimisation results are presented graphically in Figure 16.13 and Figure 16.14. Again, the DCF curve is very flat around the Max. Best Case DCF Shell 24 (Revenue Factor 0.98). Kekeda has lost the most ore and value on a by pit basis from the PFS outcome (approx. -45%).

In line with the basis for the preceding shell selections discussed, a Revenue Factor of 1.06 was used as the selection criteria for the Kekeda shell. This shell (29) generates just over 5% more recovered ounces than the Max. Worst Case DCF shell with only a 0.4% reduction in associated cashflow.

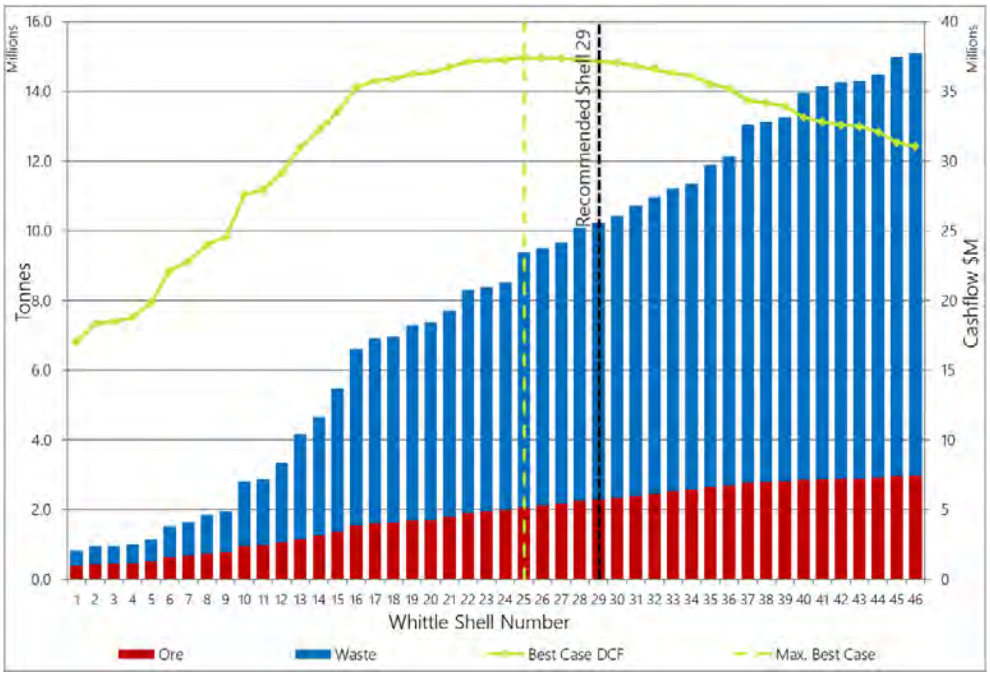


Figure 16.13 Kekeda Pit Optimisation - Tonne/Value Chart

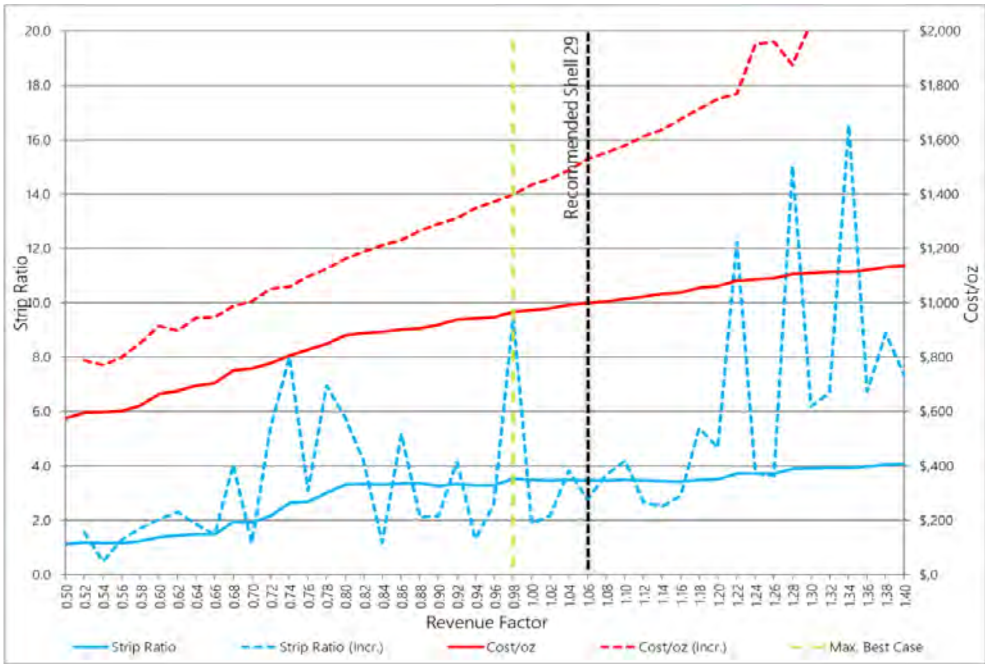


Figure 16.14 Kekeda Pit Optimisation - Strip Ratio and Cost Per Oz

Nokpa

The Nokpa optimisation results are presented graphically in Figure 16.15 and Figure 16.16. Shell 27 (Revenue Factor 1.02) was selected as it generates 20% more recovered ounces with a minimal 0.8% reduction in overall cashflow.

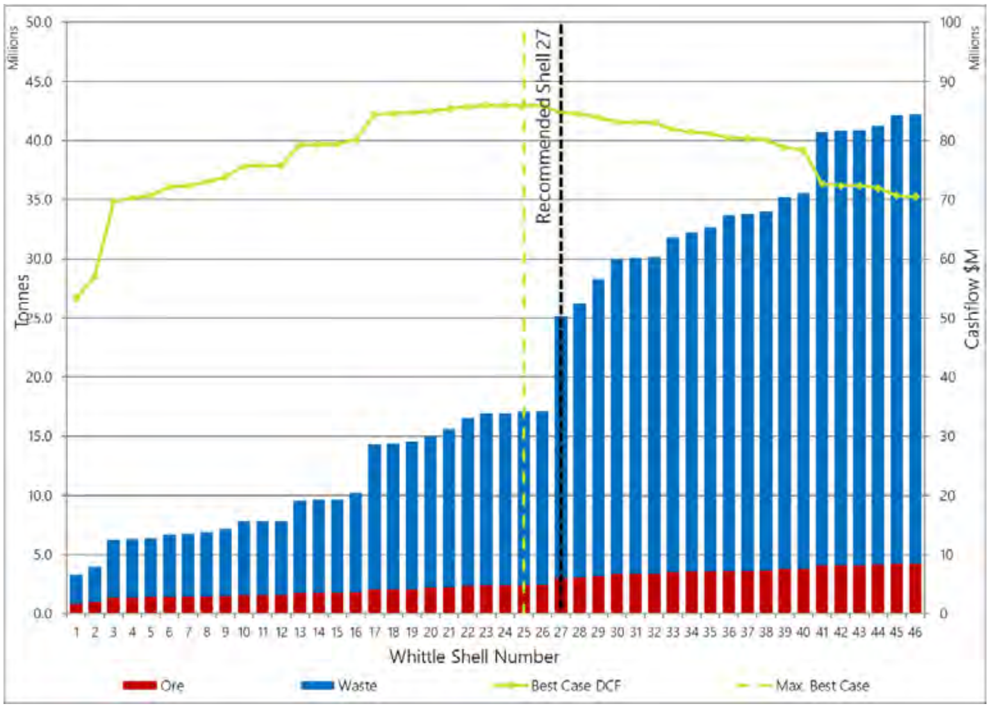


Figure 16.15 Nokpa Pit Optimisation - Tonne/Value Chart

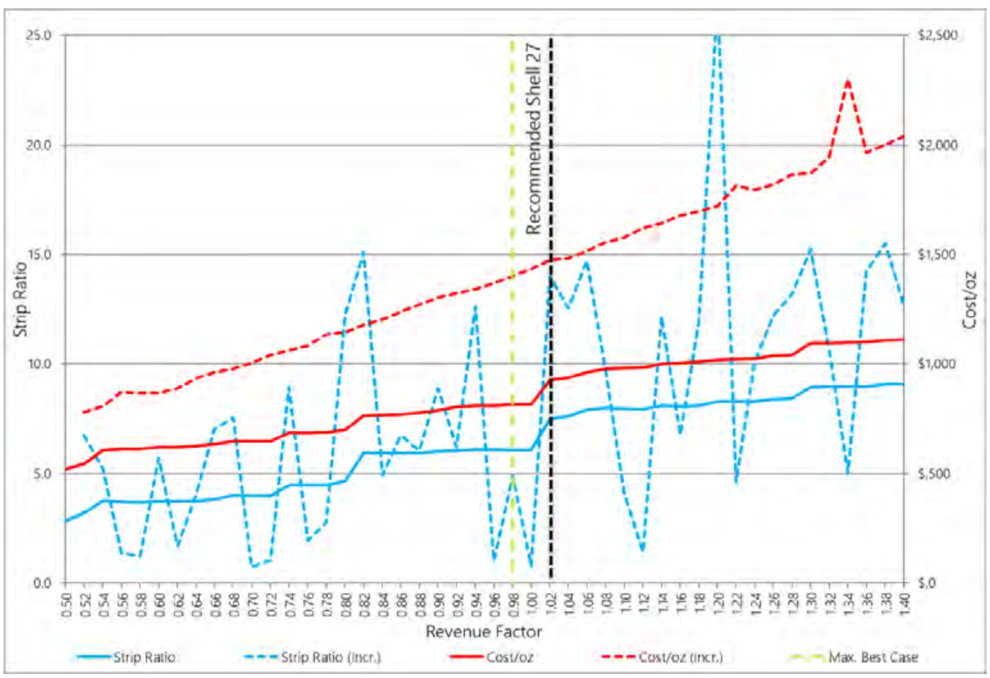


Figure 16.16 Nokpa Pit Optimisation - Strip Ratio and Cost Per Oz

Chegue Main

The Chegue Main optimisation results are presented graphically in Figure 16.17 and Figure 16.18. Shell 29 (Revenue Factor 1.06) was the selected shell as it generates 4% more recovered ounces with no change in overall cashflow.

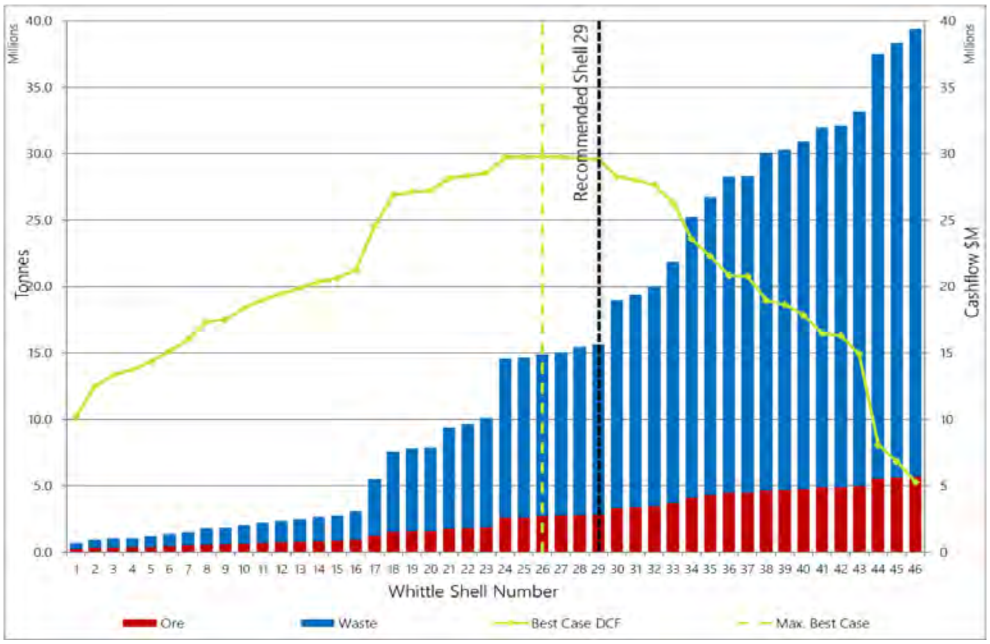


Figure 16.17 Chegue Main Pit Optimisation - Tonne/Value Chart

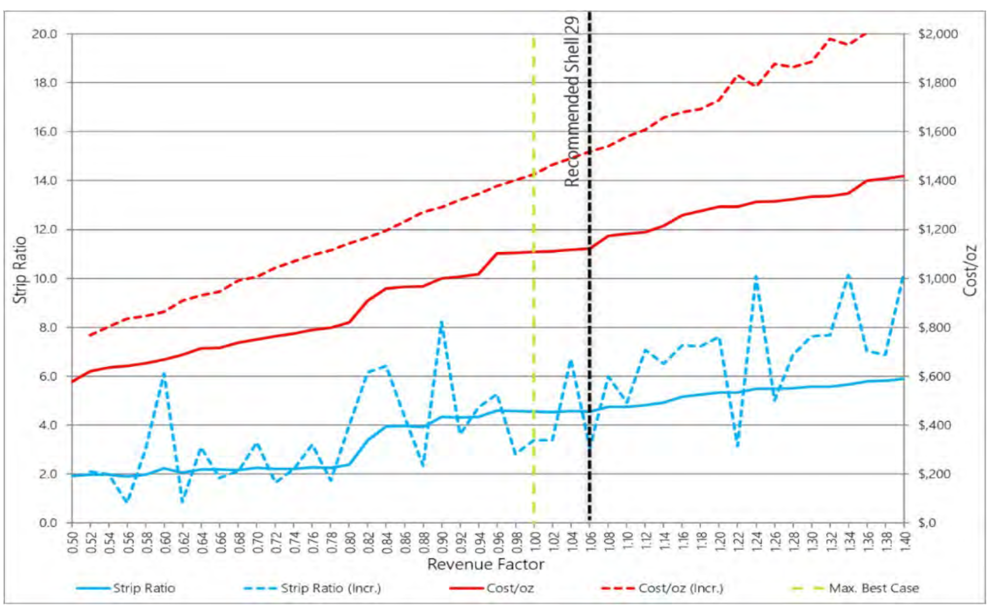


Figure 16.18 Chegue Main Pit Optimisation - Strip Ratio and Cost Per Oz

Chegue South

The Chegue South optimisation results are presented graphically in Figure 16.19 and Figure 16.20. As can be seen, Chegue South features a distinctive pattern of curves. The cashflow remains nearly level from Shell 18 (Revenue Factor 0.88) up to Shell 33 (Revenue Factor 1.18), but then there is a significant increase in the strip ratio (from 6.7 to 9.8), causing the shell value to drop to zero. Shell 26 (Revenue Factor 1.04) is the optimal choice, as it provides 3% more recovered ounces while only reducing overall cashflow by 0.4%.

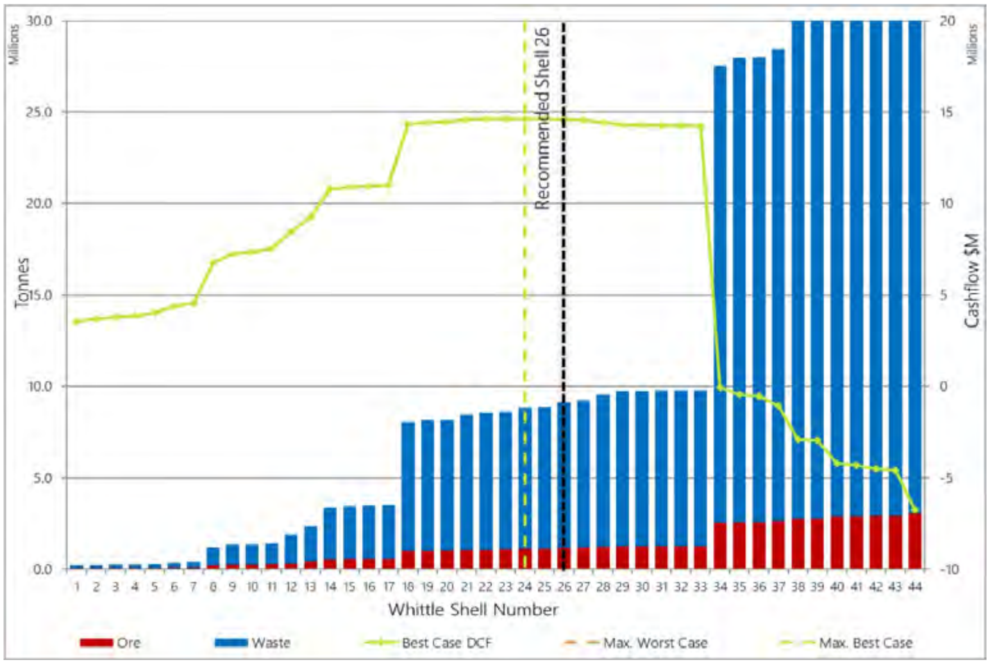


Figure 16.19 Chegue South Pit Optimisation - Tonne/Value Chart

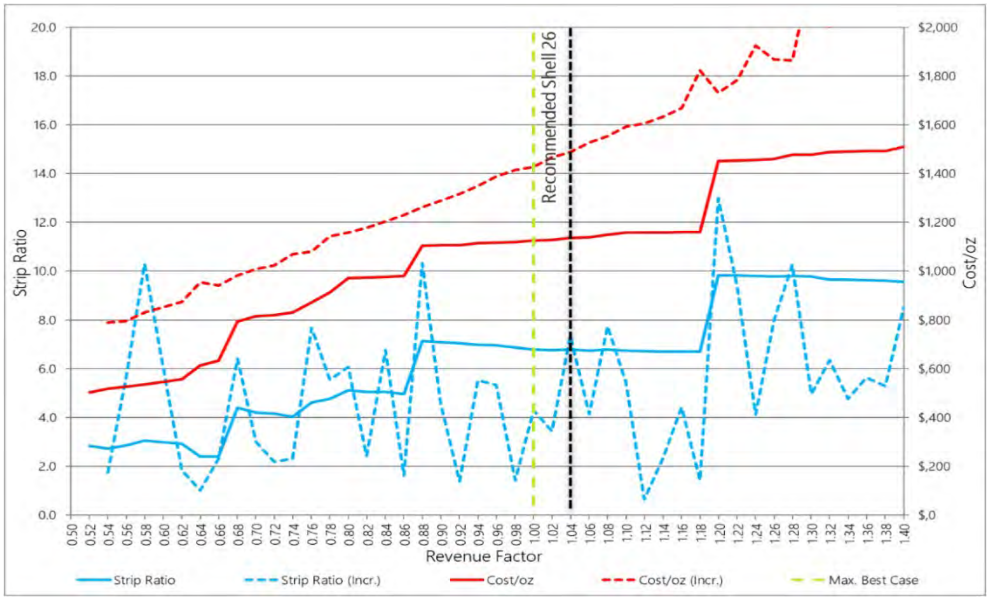


Figure 16.20 Chegue South Pit Optimisation - Strip Ratio and Cost Per Oz

Enioda

The Enioda optimisation results are presented graphically in Figure 16.21 and Figure 16.22. Shell 28 (Revenue Factor 1.04) was selected as it generates 8% more recovered ounces for a 0.5% drop in overall cashflow.

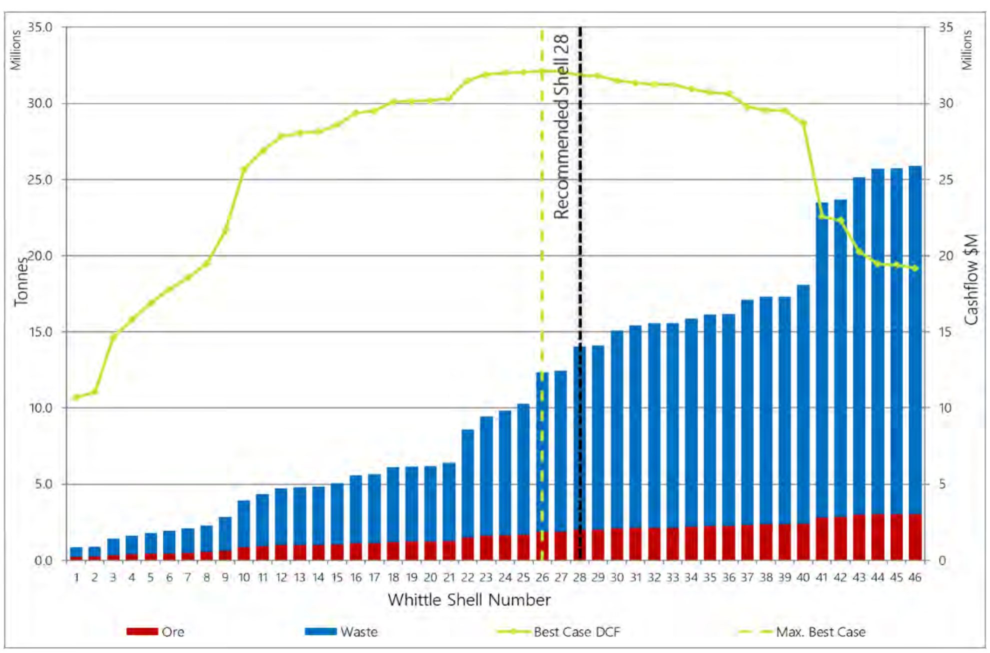


Figure 16.21 Enioda Pit Optimisation - Tonne/Value Chart

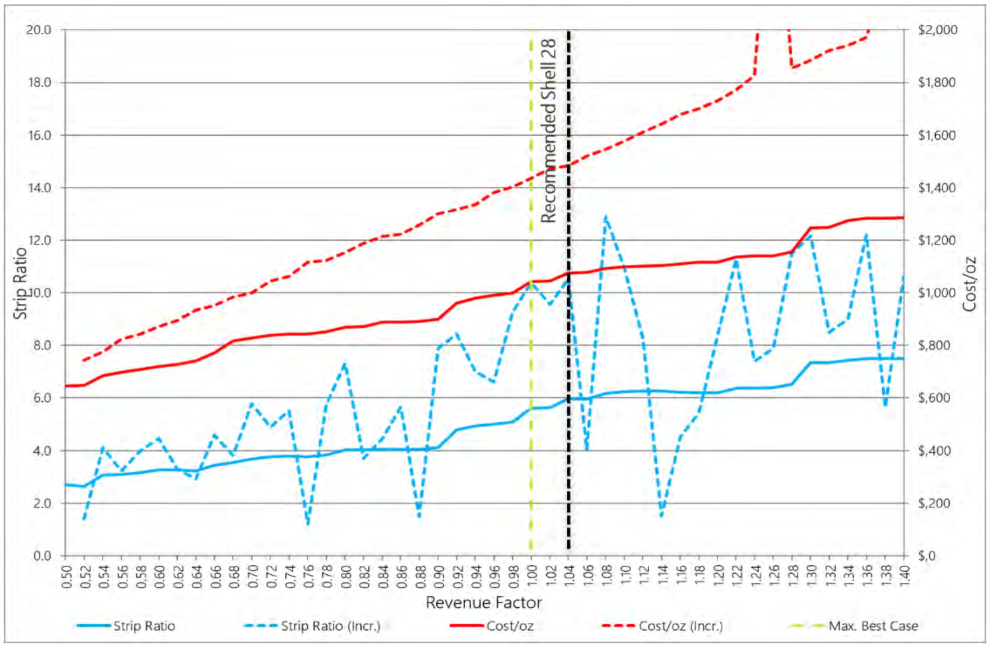


Figure 16.22 Enioda Pit Optimisation - Strip Ratio and Cost Per Oz

16.7 Mine Design

16.7.1 Pit Design Criteria

Pit Slope Design Criteria

Slope design criteria (i.e., berm widths, bench face height and bench face angles) were provided SRK Consulting (UK) Ltd and the parameters are described in detail in Section 16.3 Geotechnical Considerations.

The parameters were coded into a re-blocked "slope code" model with much larger block sizes in the X and Y directions to smooth out the variations in the weathered zone boundaries which required a specified catch berm width. In this way a more practical pit design could be generated that still honoured the overall slopes requirement of the SRK recommendations. The pit design tool in MineSight reads these parameters as the designer steps through the design by 5 m bench increments.

The designs of the WRD's are shown in the figures in the following Section 16.7.2 Final Pit Design.

Ramp Design Widths

The ramp widths based on the selected Caterpillar 777 dump truck were 16 m and 24 m for one-way and two-way ramps respectively. These layouts were provided by Centamin based on their operating experience in Africa and their geometry is shown in Figure 16.23 and Figure 16.24.

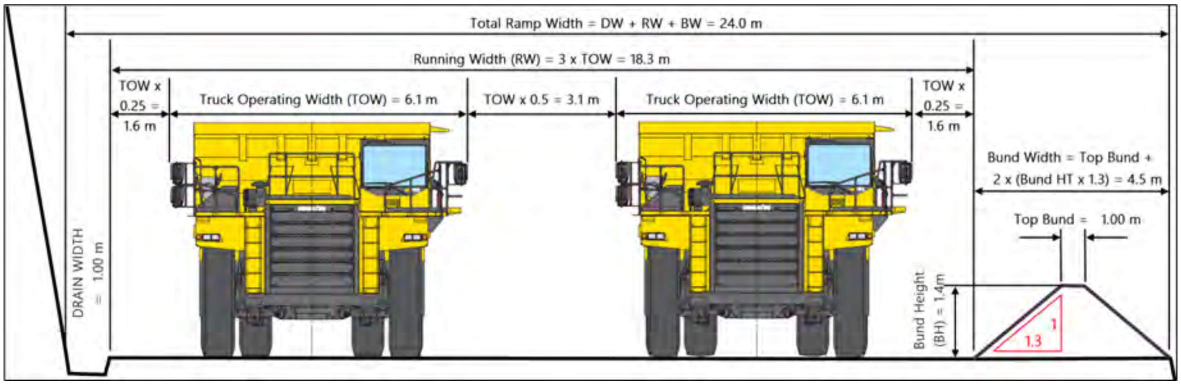


Figure 16.23 Two-way Ramp Layout

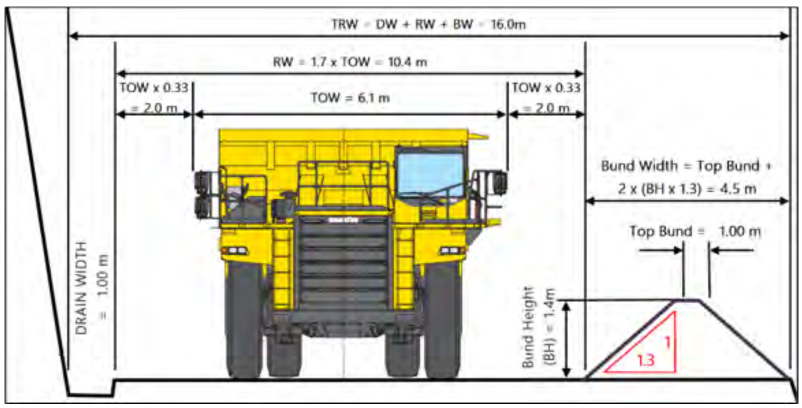


Figure 16.24 One-way Ramp Layout

Single lane ramp access is generally used for the benches at the base of the pits based on the lower traffic intensity in these areas of the pits. However, the small nature of some of the satellite pits has required relatively aggressive use of single lane ramps to minimise excess waste stripping.

Minimum Mining Width

Pit design objectives were the safe, efficient, and practical extraction of ore and waste. This meant a minimum mining width was applied at all stages to ensure that adequate space is available for equipment to access mining areas and avoid congestion. Based on a 150 t-class excavator and a 90 t-class truck, the following mining width constraints were applied.

Pit wall pushback - a minimum safe mining width of 50 m was used between any successive pit wall pushbacks for interim stages. This only applies to the Souwa and Nokpa mining areas as these were the only pits where internal pit stage designs were developed.

Base of pit - a mining width of 25 m was applied to the bottom benches of the pit designs to allow space for machine clearance and excavator slew. Final bench ‘goodbye’ cuts to a depth of up to 10 m were designed where there was sufficient length to allow temporary access to be developed with the excavator.

16.7.2 Final Pit Design

The optimisation shells detailed above were used as a guide for final pit designs. Souwa and Nokpa are the only deposits where internal stages were designed, targeting the high value areas indicated from shells of lower revenue factor. The designs were cognisant of design criteria and mining practicalities detailed in Section 16.7.1. The design objective was to produce practical designs that maximise ore tonnes. Every effort was made to minimise waste but not to the detriment of ore recovery.

Figure 16.25 provides a general site arrangement, providing context for the locations of all the pits. The following figures show the layout of the pit stages, associated waste rock dumps (WRD’s) and mine haul roads, as well as indicative locations of satellite ROM pads and offices/laydown areas.

Souwa

Souwa is the largest pit and primary value for the Project. Consequently, it recieved more design evaluation and has been divided into several stages. This is to reduce initial strip ratio and bring forward access to the higher grade material, thereby unlocking value earlier. The stages have been designed on the basis of optimisation shells at lower revenue factors which target higher margin areas.

- Construction Borrow Pit - A borrow pit has been designed to provide bulk fill material for initial construction works of the first lift of the tailings storage facility and initial ROM Pad construction (see Figure 16.26). The allowance equates to approximately 1,000 kbcm of weathered material, and it is assumed this will be removed at the project construction phase by others. Consequently, this pit was depleted from the mining inventory for the LOM Production Schedule (refer to Section 16.8);
- Stage 1 - The first mining stage (see Figure 16.27) targets low strip ratio oxide material for early mill feed, as well as higher grade transition and fresh material for grade, along the footwall of the deposit;
- Stage 1 A - The LOM Production Schedule indicated that the Stage 1 design was too large. Consequently Stage 1 was split into a north (Stage 1A) and South (Stage 1B) to assist with accessing early ore and balancing strip ratio and material movement. Stage 1A is shown in Figure 16.28;
- Stage 2 - This stage is a pushback in the northwest of the pit to access high value ore beneath Stage 1 (see Figure 16.29);
- Stage 3 - This is the southern end of the main pit (see Figure 16.30) which can be mined through a separate ramp access;

- **Stage 4** - This is a western pushback to the ultimate highwall and completes the Souwa pit (see Figure 16.31). As such, it has the highest strip ratio and lowest value of all the Souwa stages.

Waste rock from Stage 1 and 2 will be deposited in the eastern waste rock dump, with the majority of Stage 3 and 4 utilising the western waste rock dump.

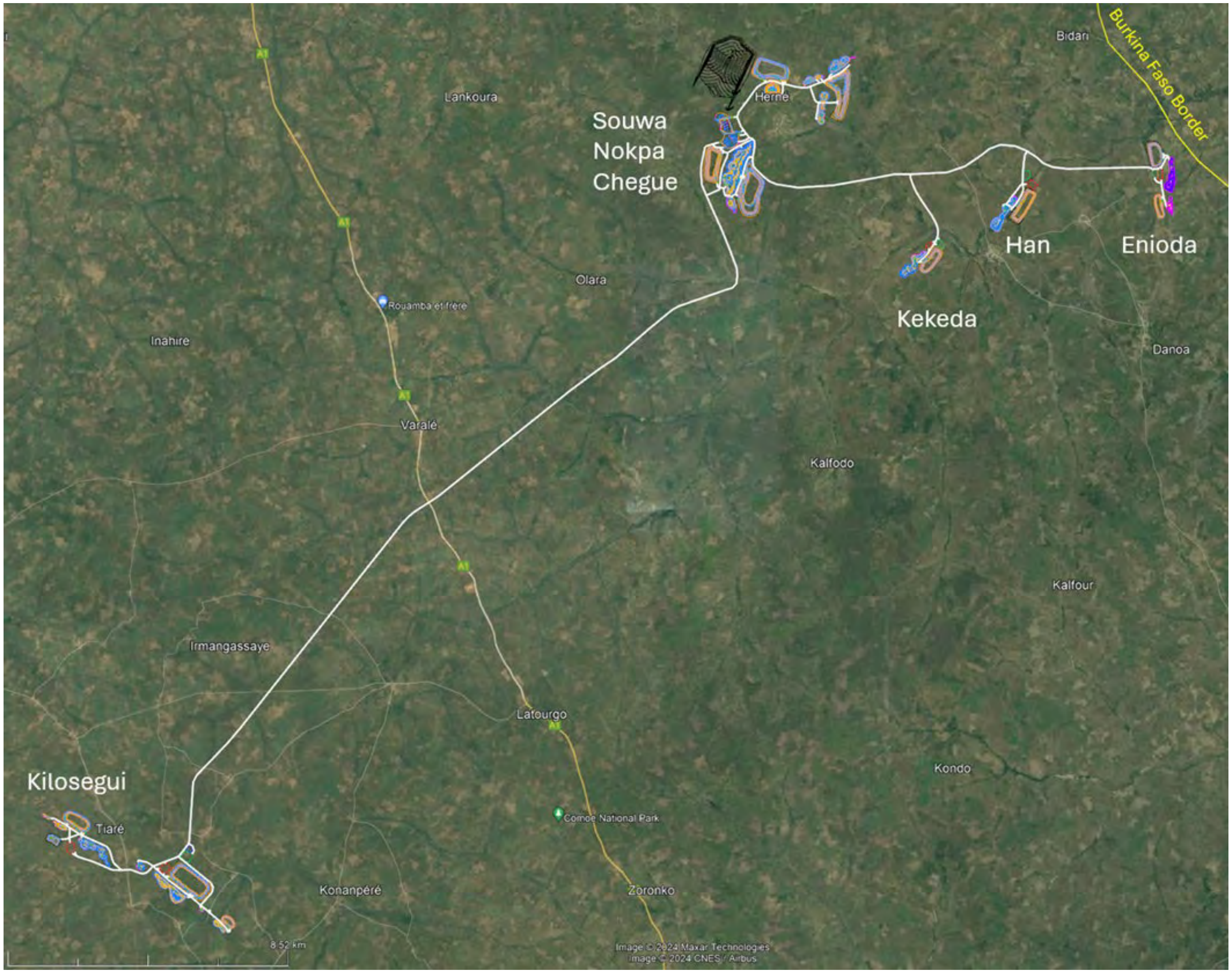


Figure 16.25 General Site Arrangement

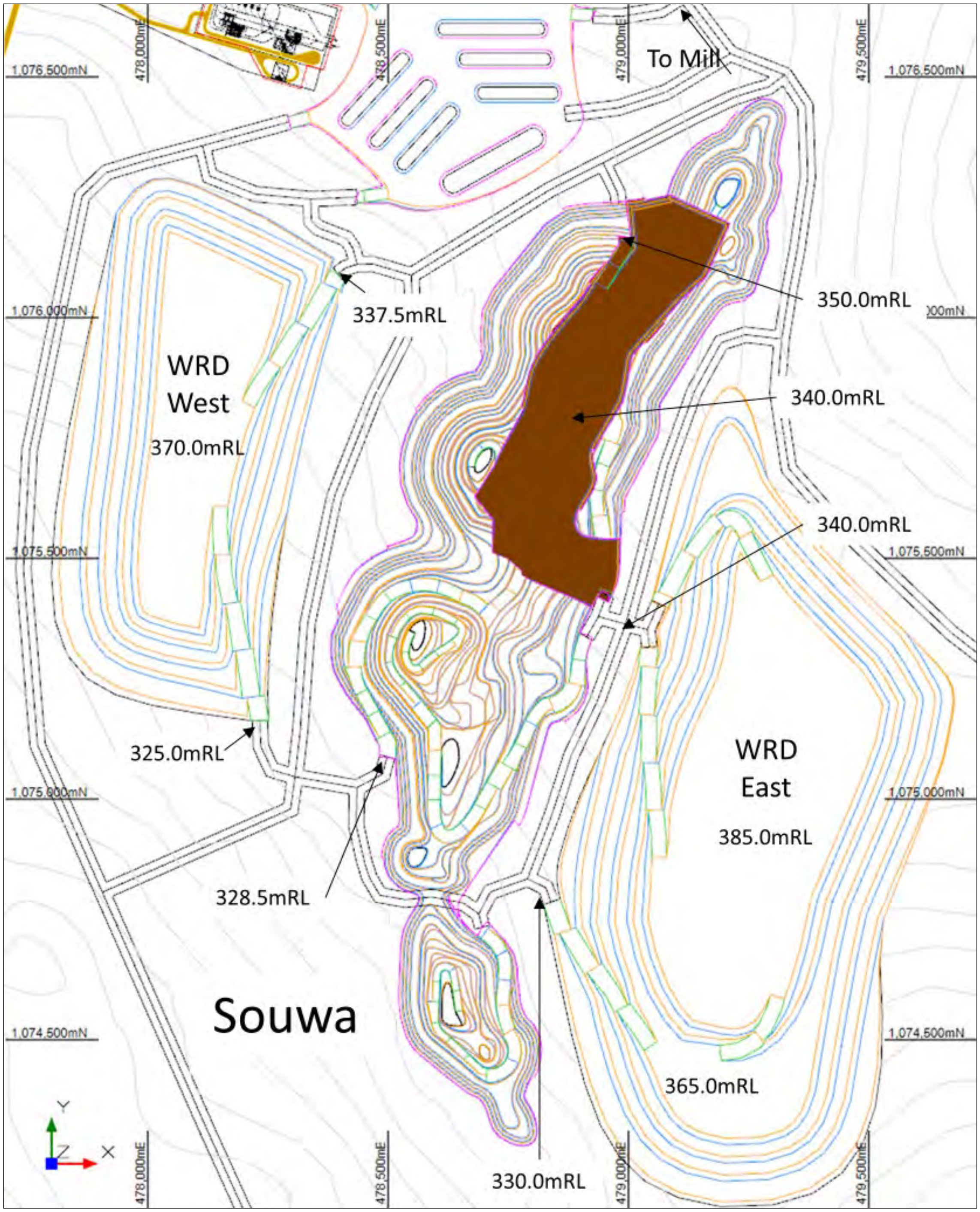


Figure 16.26 Pit Design - Souwa Borrow Pit

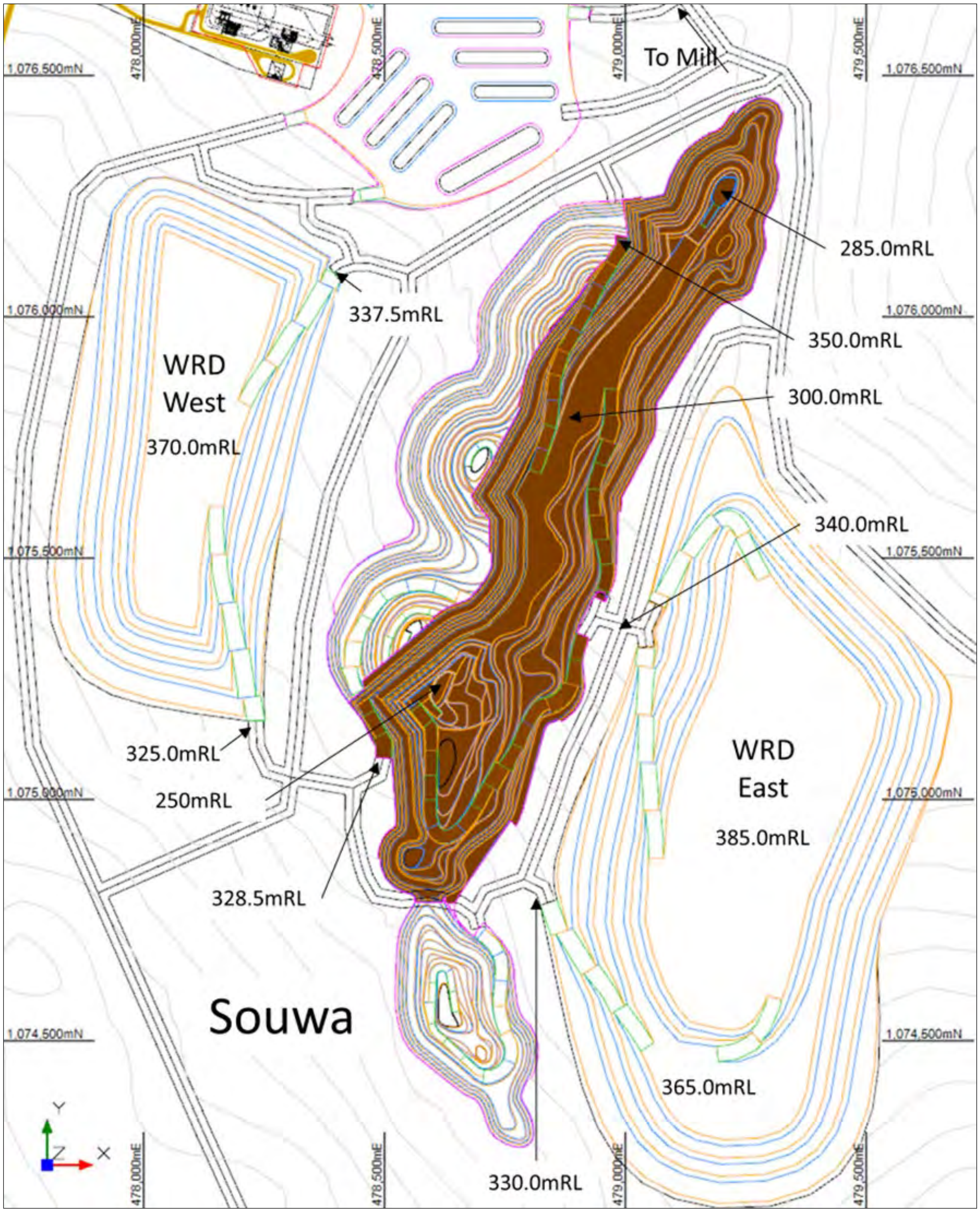


Figure 16.27 Pit Design - Souwa Stage 1

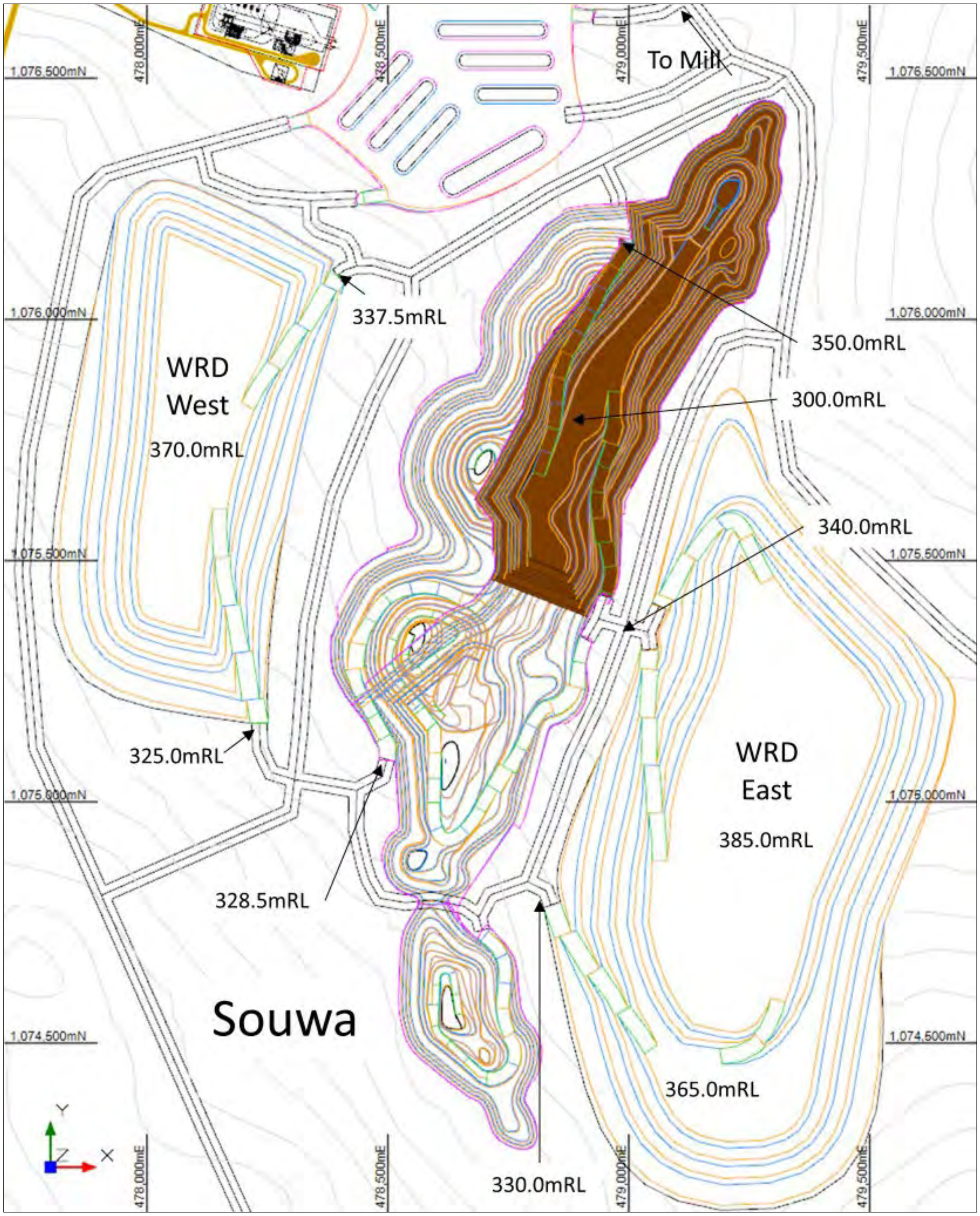


Figure 16.28 Pit Design - Souwa Stage 1A

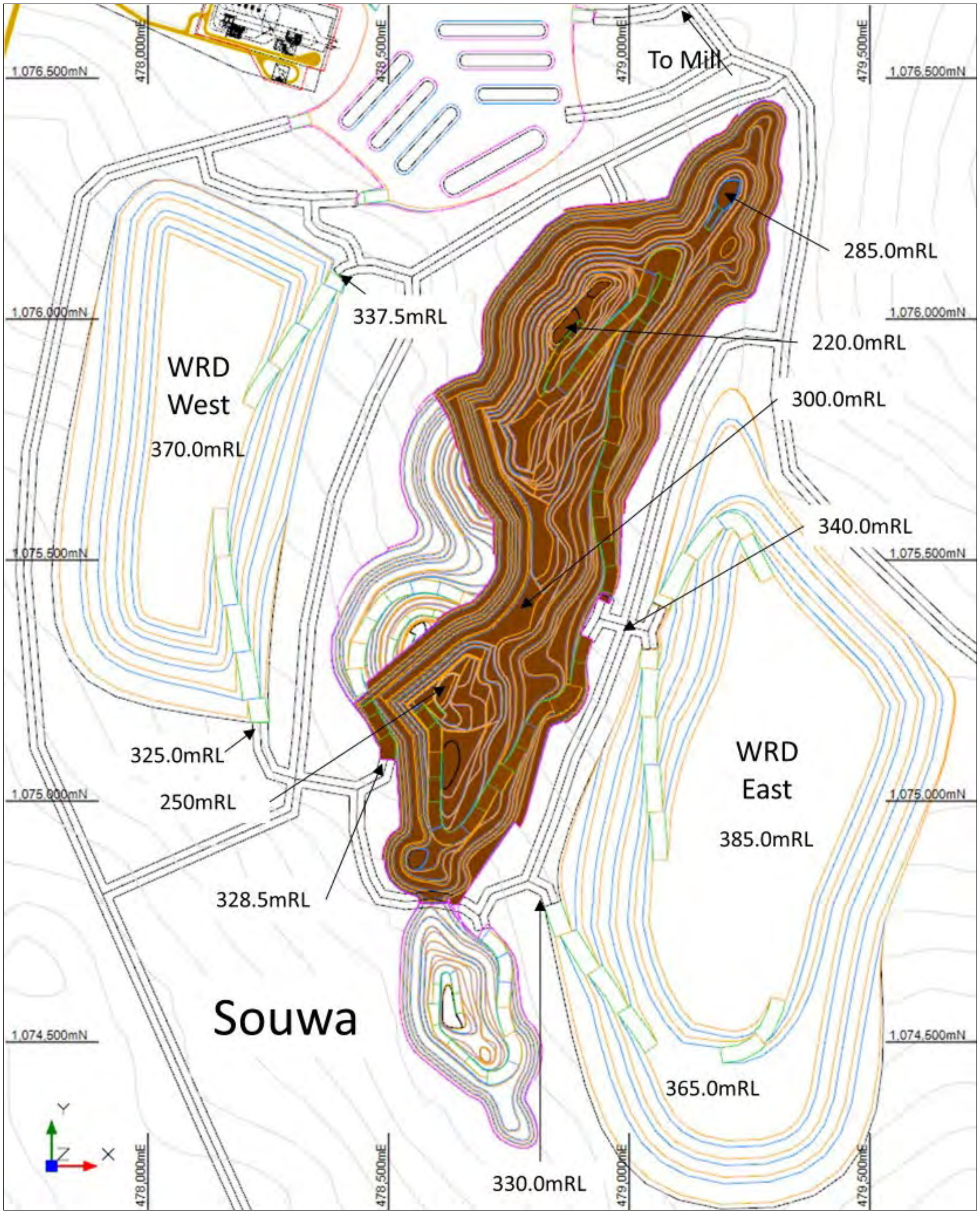


Figure 16.29 Pit Design - Souwa Stage 2

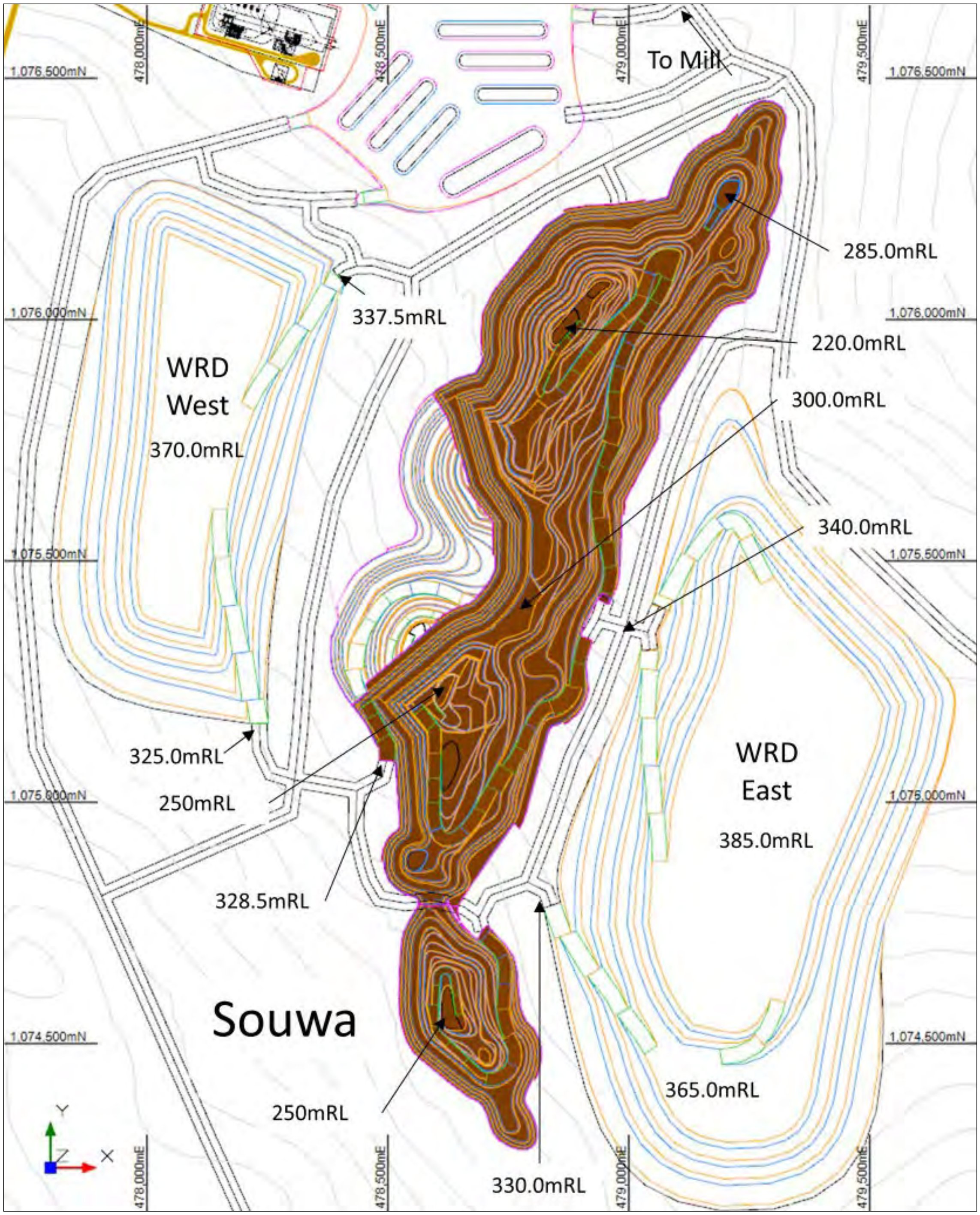


Figure 16.30 Pit Design - Souwa Stage 3

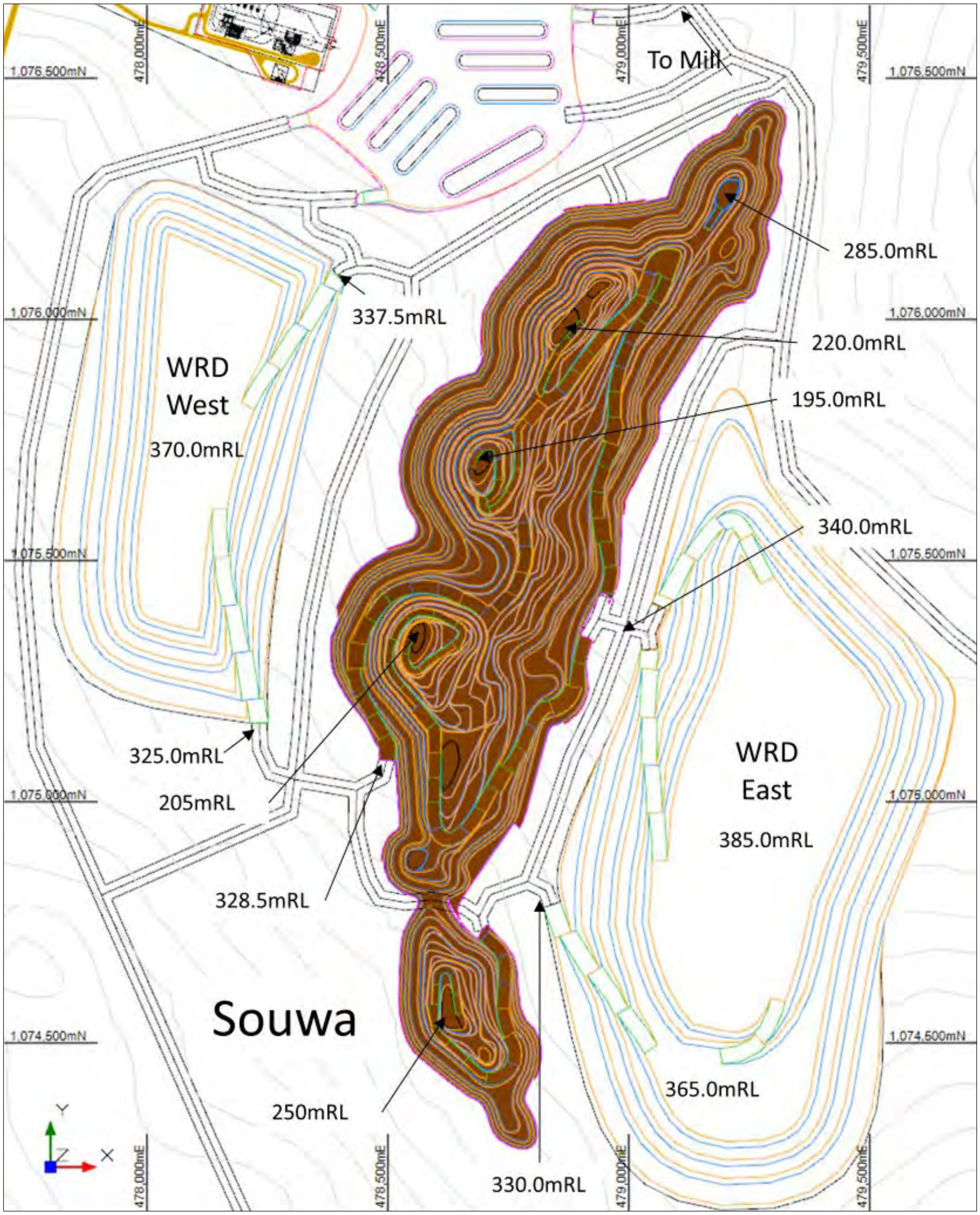


Figure 16.31 Pit Design - Souwa Stage 4

Nokpa

The Nokpa pit was initially designed as a single stage pit. However, to better enable the schedule to target higher value material, an internal stage was subsequently developed. This internal stage is shown in Figure 16.32, and will be mined utilising the existing final ramp design to a depth of 115 m.

The second stage is shown in Figure 16.33 and will require a temporary internal ramp to a depth of 50 m, which will then be mined on retreat and the final pit-ramp on the south wall will be utilised for the remainder of the second stage. The second stage extends to a depth of 175 m.

All waste material from Nokpa will be deposited on a waste dump located to the North. This waste dump takes into account nearby water flows and is constrained by the TSF to the west. Ore will be hauled directly to the adjacent Process Plant and ROM pad.

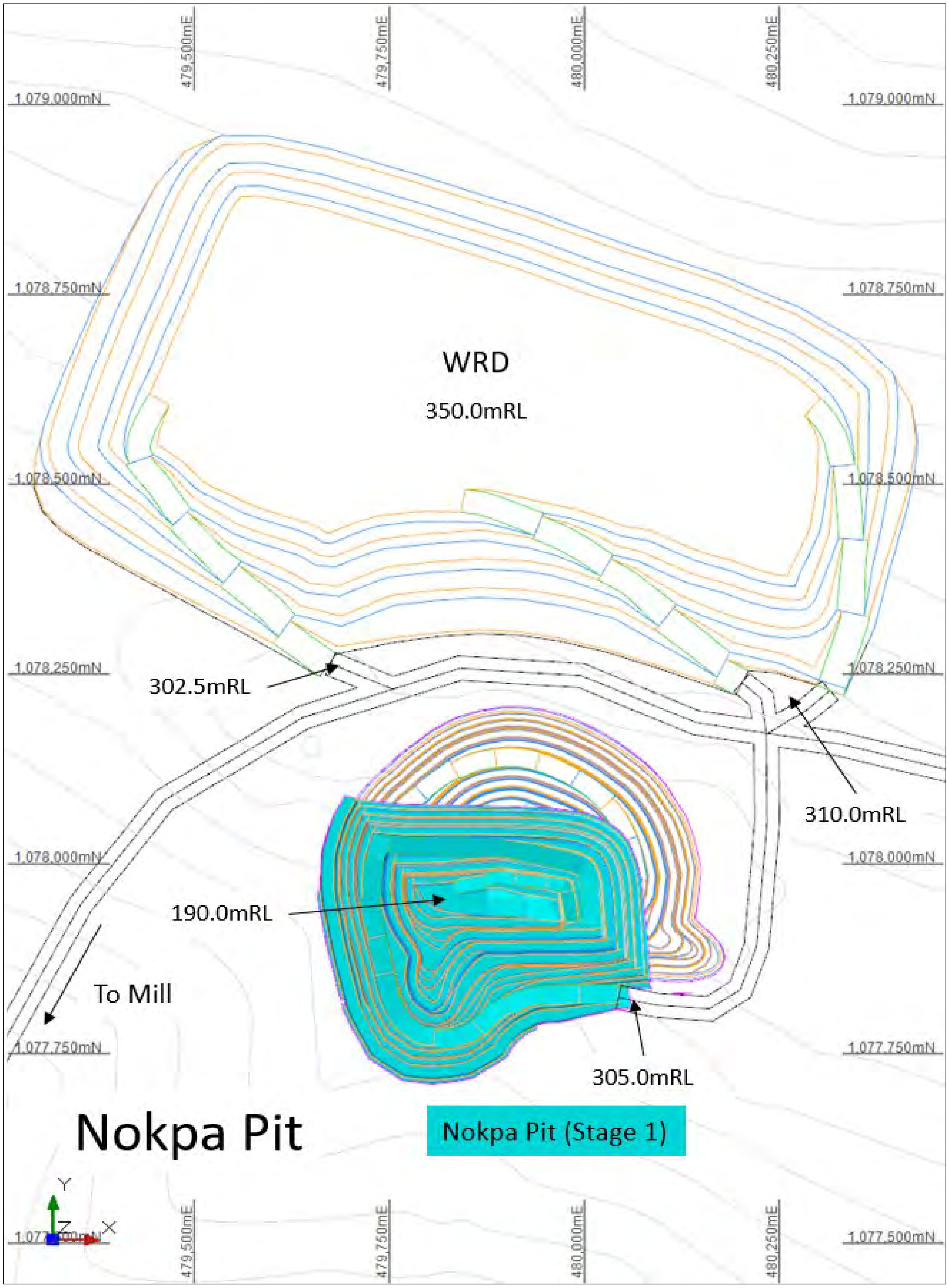


Figure 16.32 Pit Design - Nokpa Stage 1

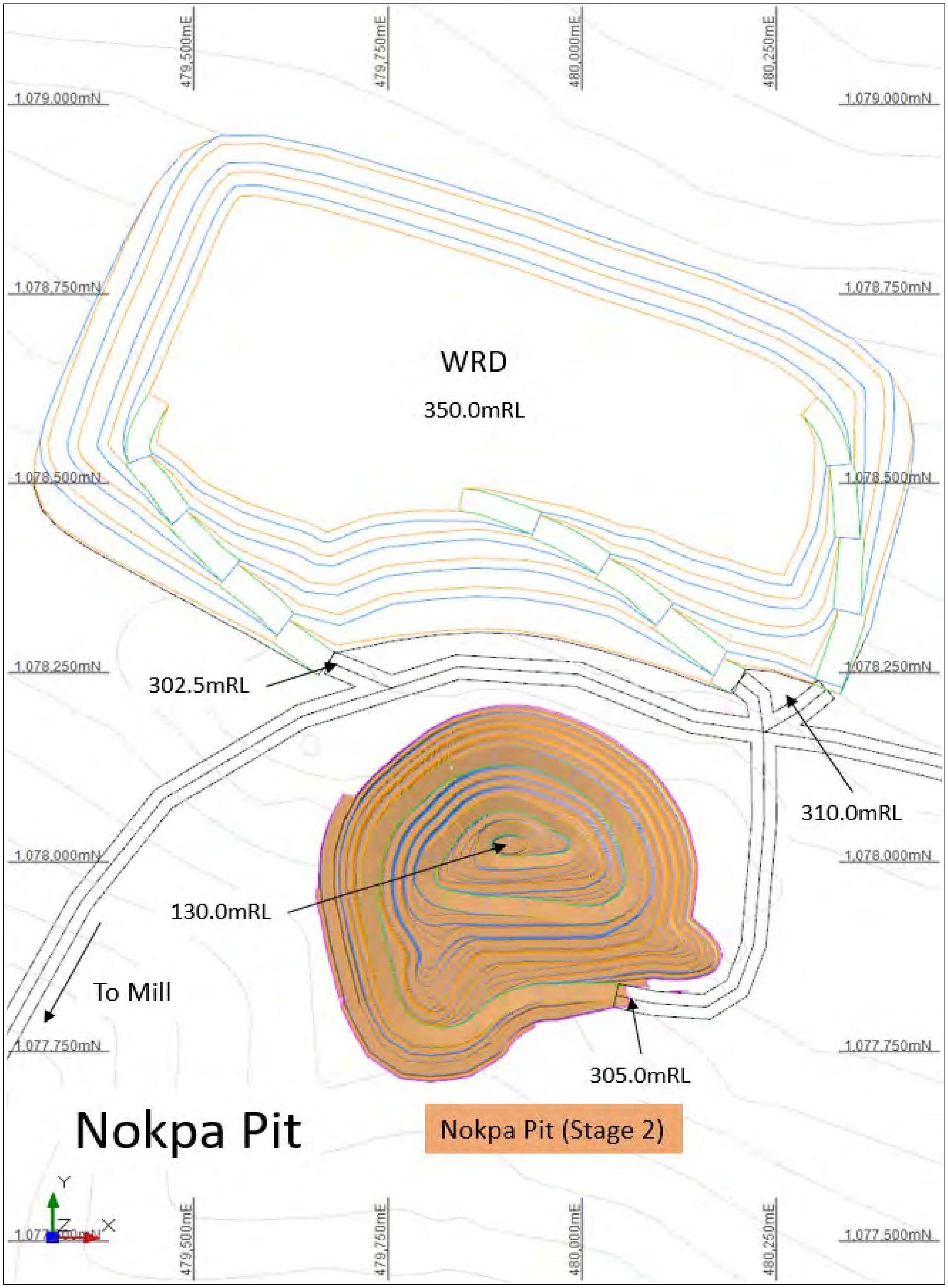


Figure 16.33 Pit Design - Nokpa Stage 2

Kilosegui

The Kilosegui pits are spread across a strike length of more than 6 km. They are located approximately 32 km from the Souwa ROM/Mill and, as such, require their own offices and laydown areas. The pits are effectively divided into three (3) East Pits (see) and four (4) West Pits (see)either side of the public road/village of Dupondius. Haul trucks will stockpile ore at one of two stockpiles (East and West) which will then be transported via road trucks to the road train stockpile at the Souwa ROM.

The main pit is in the middle of the Kilosegui pits and extends to a depth of 140 m. There is also a second large pit in the west which extends to a depth of 95 m.

Enioda

The Enioda pits sit approximately 15 km from the process plant ROM pad. As such, Enioda ore requires stockpiling and rehandle by road train. The primary pit at Enioda is approximately 90 m deep as shown in Figure 16.36. Waste dumps and infrastructure are located on the west of the pit to maintain a standoff distance of approx. 500 m from the Burkina Faso border.

Han

Two pits make up the Han deposit as shown in Figure 16.37. The larger pit is to the south and is approximately 100 m deep. A smaller pit to the north is approximately 50 m deep. The pit is approximately 12 km from the process plant ROM pad and will require ore to be stockpiled and hauled via road truck.

Kekeda

The Kekeda pit has an unusual geometry due to some shallow parallel deposits that WHITTLE™ targeted in the optimisation. The result is a very flat footwall, with a steep hangingwall as shown in Figure 16.38. The maximum pit depth is 90 m. The pit is approximately 9.5 km from the process plant ROM pad and will require ore to be stockpiled and hauled via road truck.

Chegue Main and South

The Chegue Main and South pits are shown in Figure 16.39. They are located close to the Mill and, as such, do not require a separate ROM pad or office.

Three pits make up the Chegue Main pits, with the deepest being about 95 m. The two smaller pits are 65 m and 40 m deep respectively.

Chegue South has a series of connected pits. The deepest pit being 85 m deep. Both the Chegue Main and South pits utilise a combined waste dump. This waste dump is designed with potential expansion of the Chegue South pit in mind, while also being offset from the seasonal waterways to the east and south.

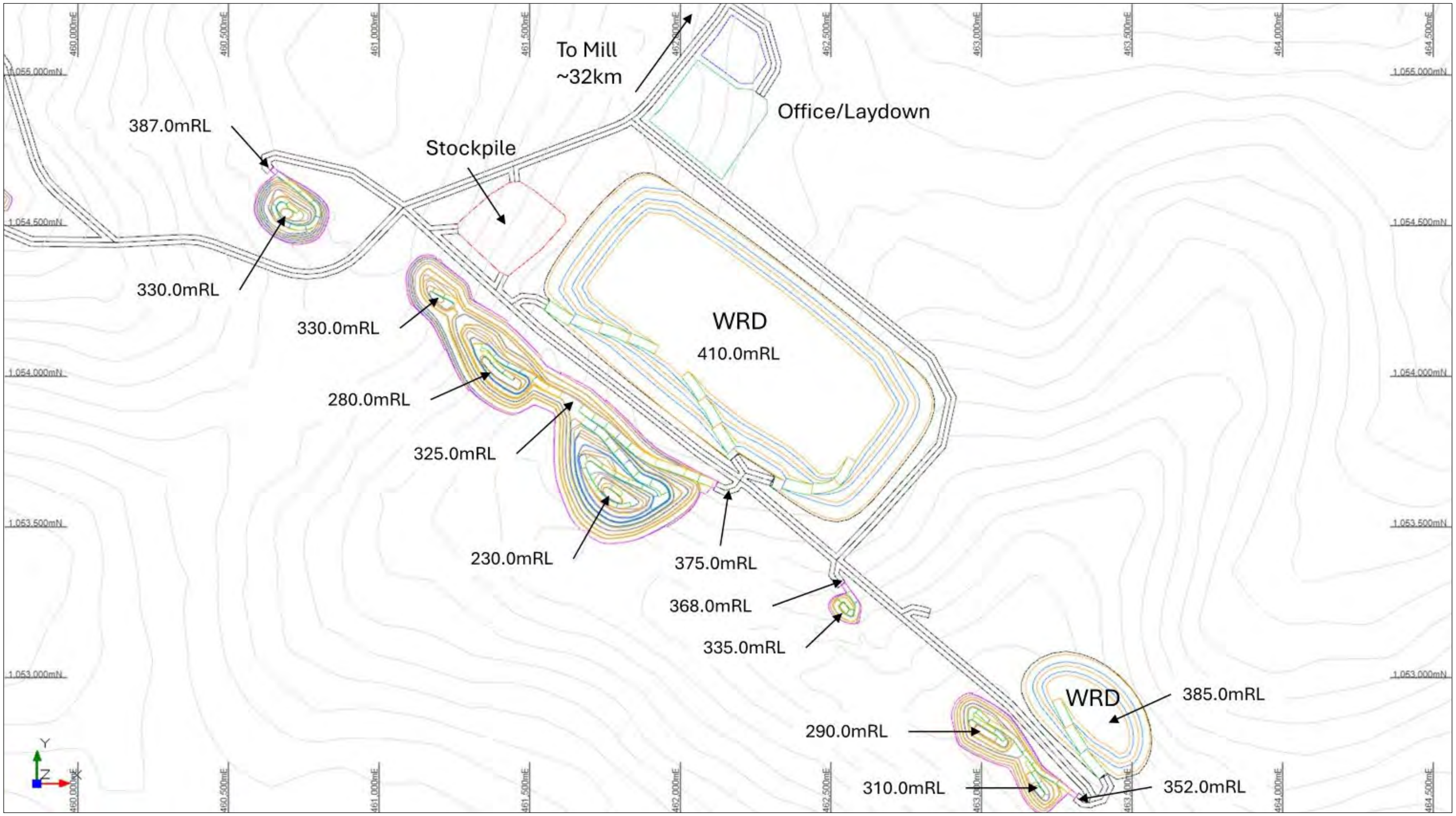


Figure 16.34 Pit Design - Kilosegui East

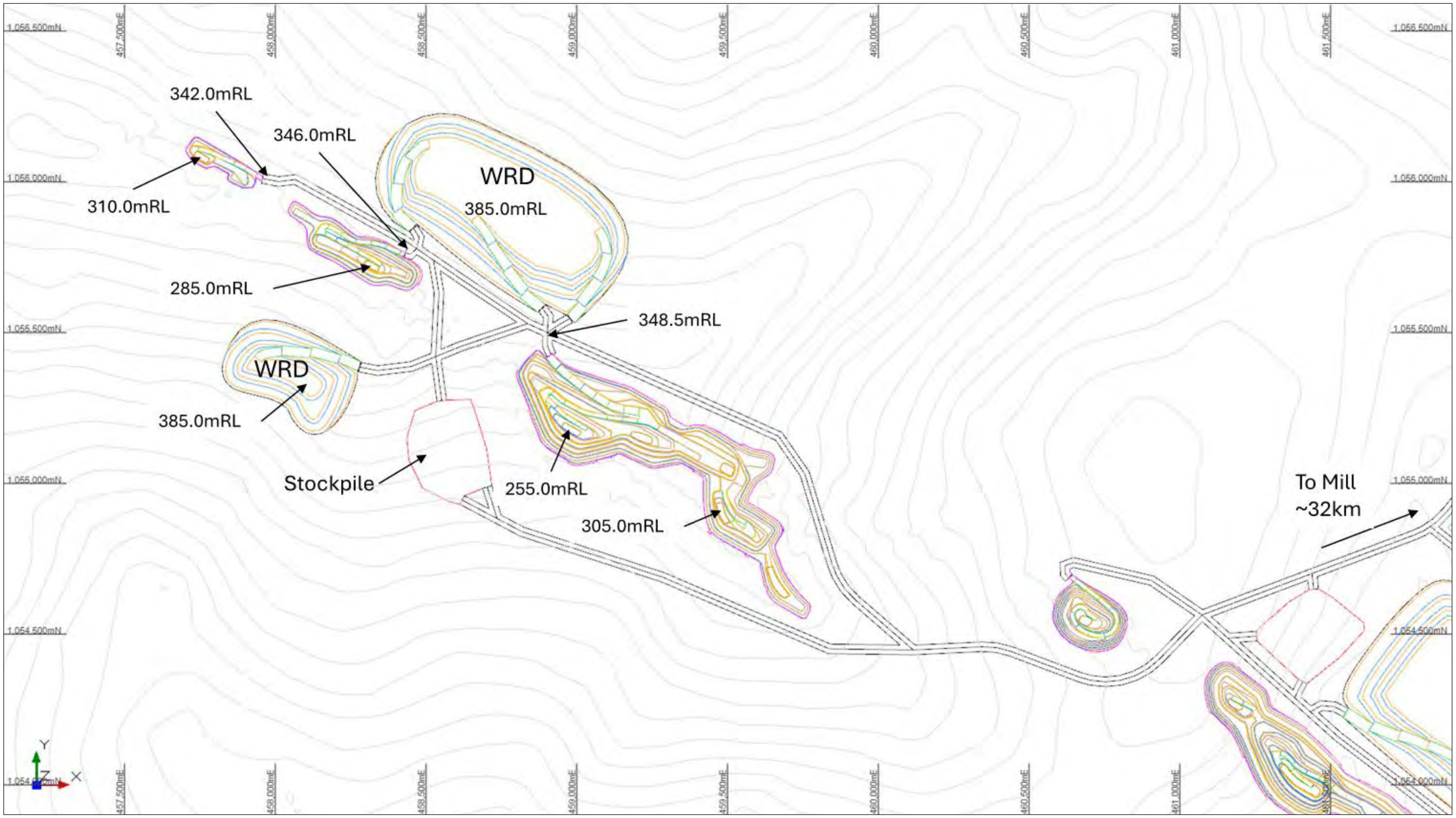


Figure 16.35 Pit Design - Kilosegui West

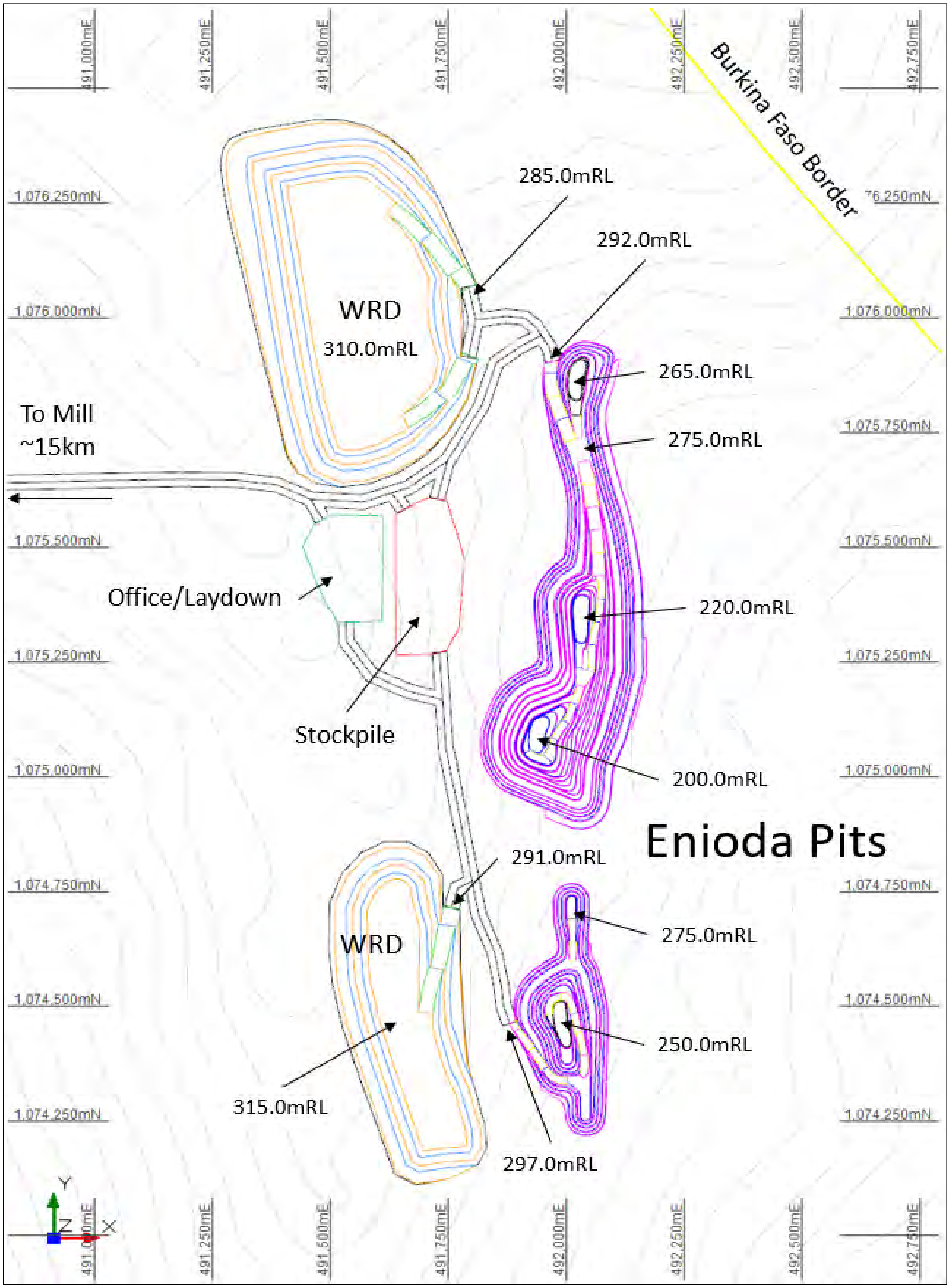


Figure 16.36 Pit Design - Enioda

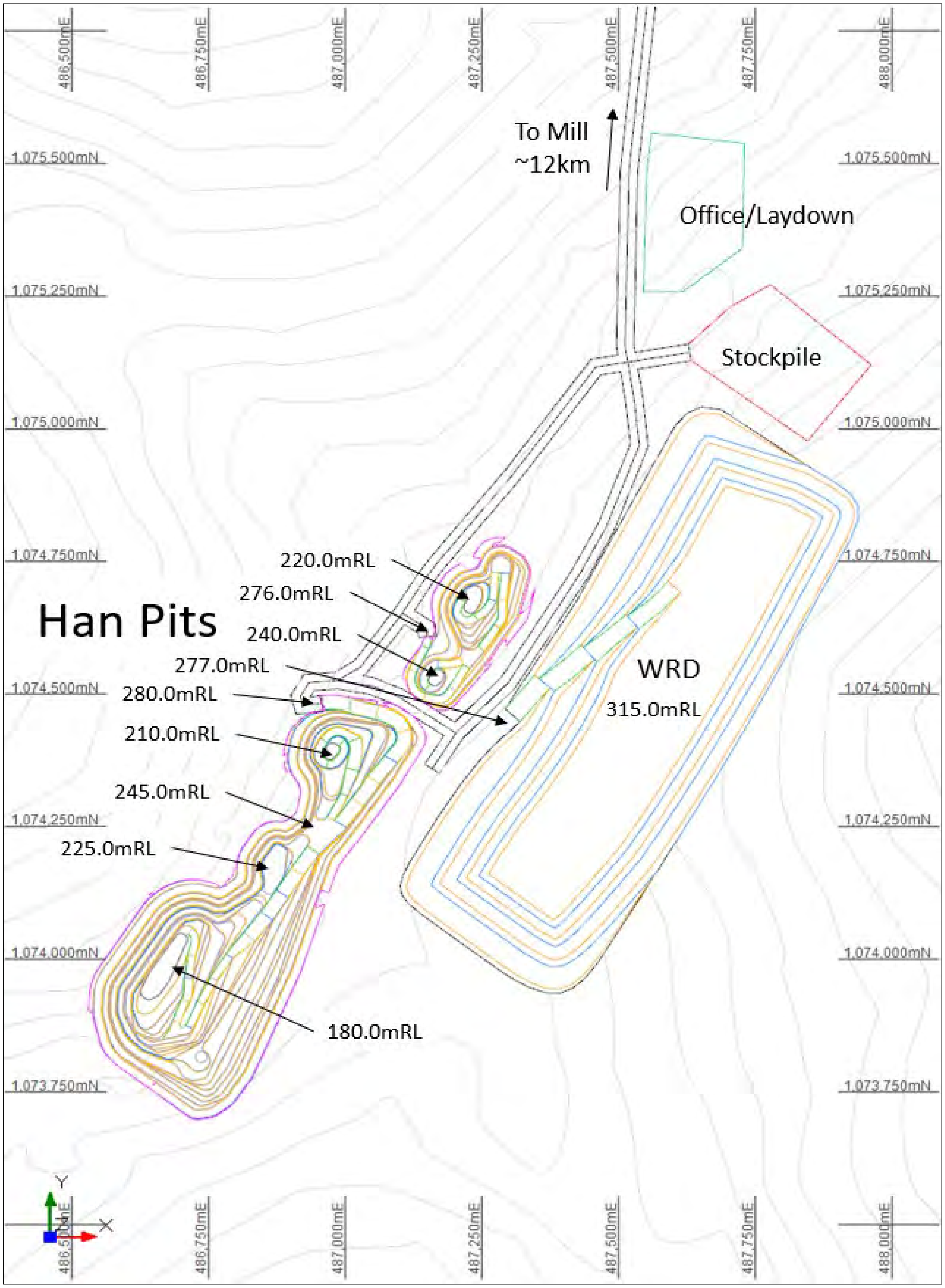


Figure 16.37 Pit Design - Han

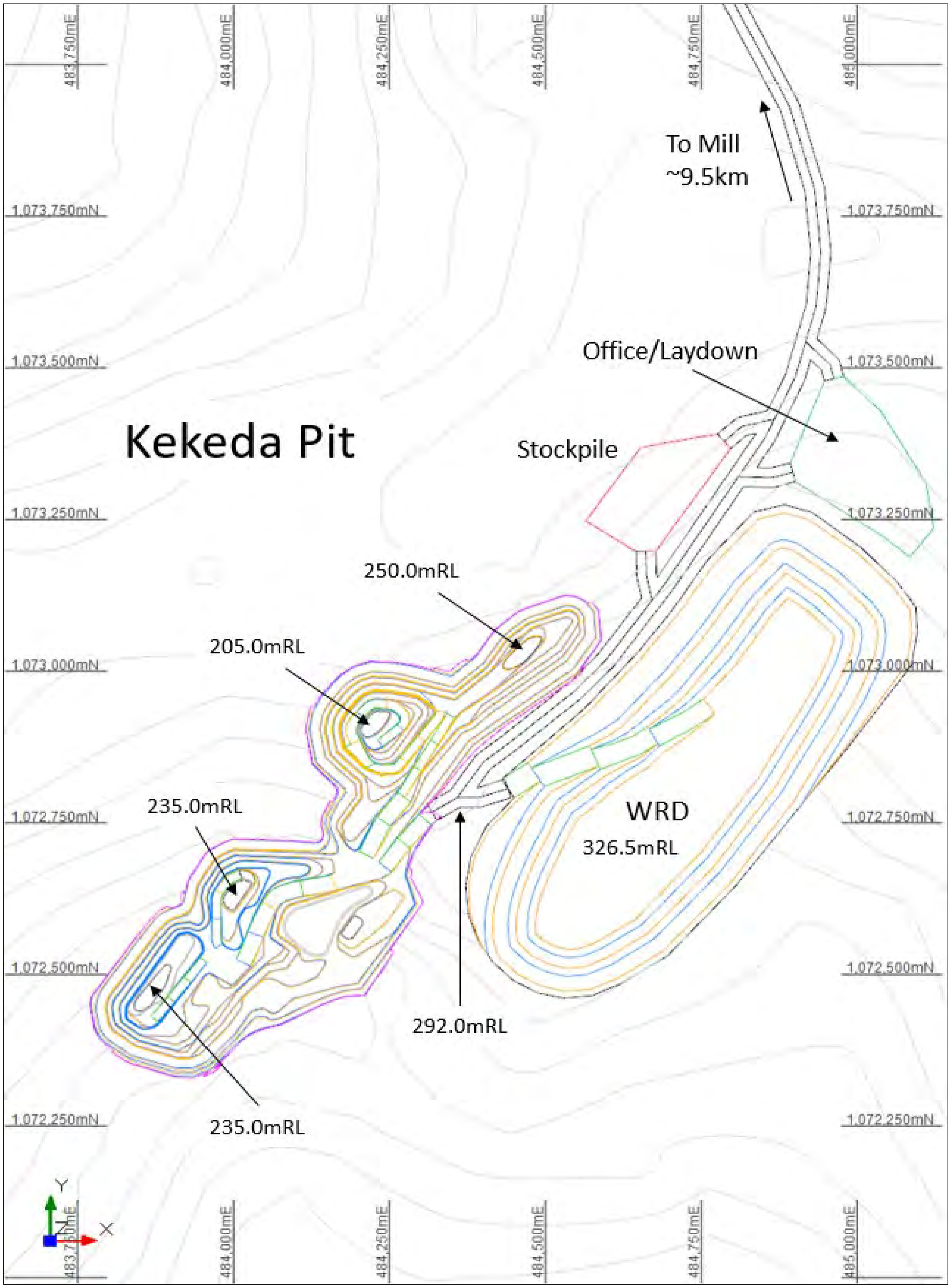


Figure 16.38 Pit Design - Kekeda

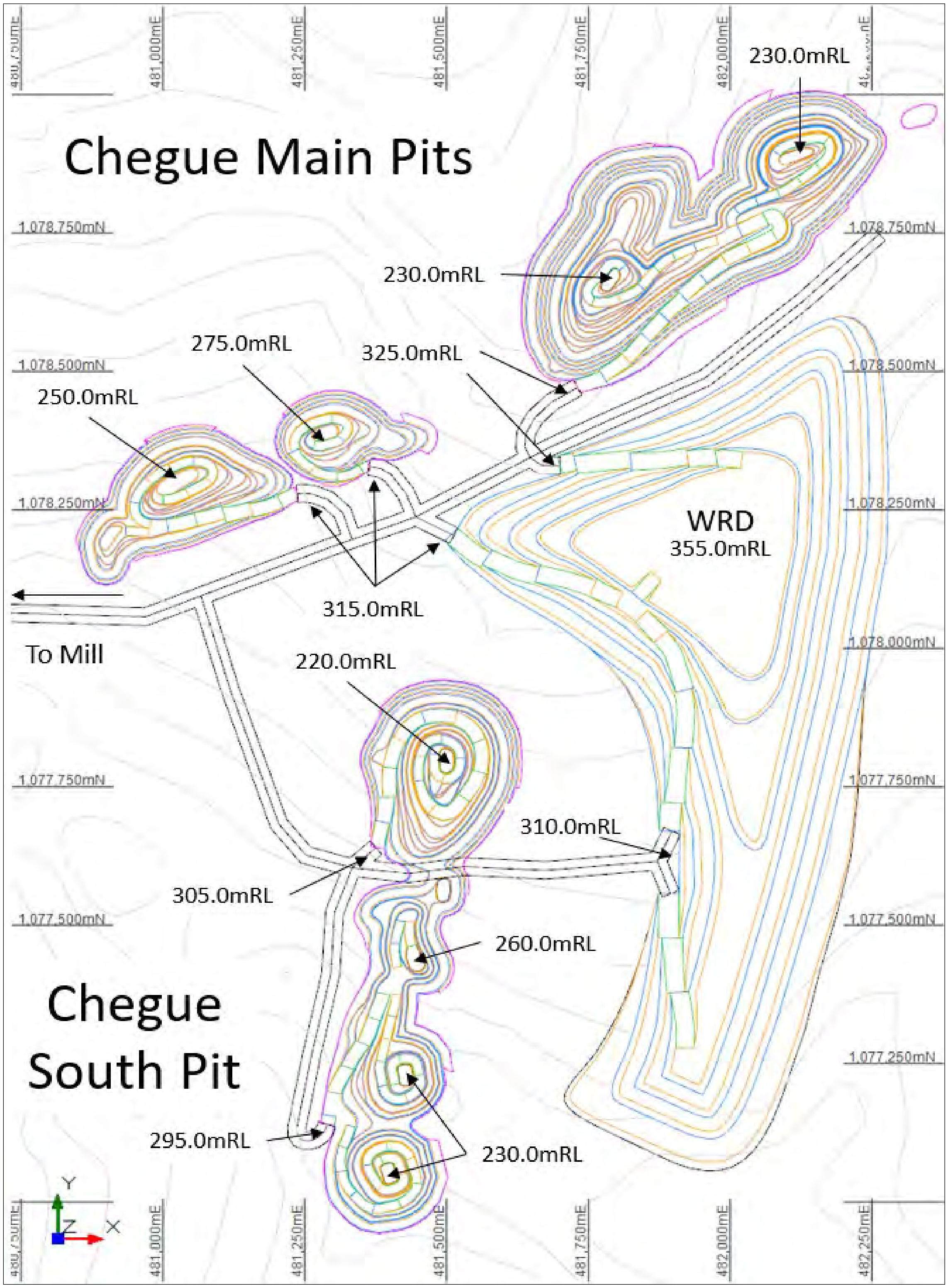


Figure 16.39 Pit Design - Chegue Main and South

16.7.3 *Pit Design Reconciliation to Whittle Shells*

Comparison of the WHITTLE™ optimised pit shells against the pit designs reveal that matching the design to the WHITTLE™ pit shell was challenging. This was a function of:

- The small nature of many of the pits, particularly the satellite pits;
- Complex geometry where a single pit would drive to multiple low points requiring separate access;
- The varying stack berm width in fully and partially weathered material was difficult to model due to the highly variable thickness of these zones.

Consequently, an overall increase in waste of 10% is considered a good outcome. However the outliers to this are Kekeda and Souwa, both of which show an approximate 20% increase in waste. In both cases this is due to complex shell geometry, and in the case of Souwa also multiple stages requiring additional independent ramp access. In both case there are also considerable increases in ore tonnes, resulting in a strip ratio increase of approximately 10%. Chegue South presents the most unfavourable outcome, with increasing waste and decreasing ore, further reducing the value of what was already a relatively low value pit. Consequently, this pit is scheduled late in the mine life.

Mining Area	Ore		Cont. Au	Waste	Total	Strip Ratio
	kt	g/t	Oz.	kt	kt	
Enioda	2,055	1.38	90,867	12,300	14,354	6.0
Han	3,190	2.16	221,202	16,743	19,932	5.2
Kekeda	2,417	1.21	93,811	9,302	11,719	3.8
Kilosegui	11,718	1.40	530,796	48,647	60,365	4.2
Nokpa	2,985	1.97	188,610	25,332	28,317	8.5
Chegue South	1,064	1.36	46,530	8,442	9,506	7.9
Chegue Main	2,900	1.08	100,897	12,918	15,818	4.5
Souwa	12,131	1.57	611,507	56,158	56,158	4.6
Total	38,459	1.52	1,884,220	189,843	216,171	4.9
% Difference from Optimisation Shell						
Enioda	3.7%	-3.6%	0.0%	2.8%	2.9%	-0.9%
Han	4.3%	-3.3%	0.6%	8.5%	7.8%	4.1%
Kekeda	11.4%	-4.5%	6.4%	24.1%	21.3%	11.4%
Kilosegui	2.4%	-3.1%	-0.5%	5.2%	4.6%	2.7%
Nokpa	-1.3%	-1.5%	-2.7%	9.5%	8.2%	10.8%
Chegue South	-7.2%	-3.6%	-10.4%	10.4%	8.1%	18.9%
Chegue Main	3.5%	-2.9%	0.3%	1.8%	2.1%	-1.6%
Souwa	8.2%	-4.7%	3.2%	17.8%	-4.7%	8.9%
Total	4.4%	-3.6%	0.7%	10.2%	3.4%	5.5%

Table 16.18 Pit Design Inventory and % Difference from Optimisation Shells

16.7.4 Waste Rock Dump Design

The designs of the WRD's were shown on the figures in Section 16.7.2. The Waste Rock Dump (WRD) designs have been developed based on the as-mined waste volumes and the following assumed swell factors:

- Oxide = 10%;
- Transition = 15%;
- Fresh = 25%.

WRD slope design criteria assumed a concave final rehabilitation surface and a maximum dump height of 40 m above ground level. The slope design assumptions are detailed in Table 16.19.

Height		Face Height	Face Slope	Bern Width	Overall Slope
From (m)	To (m)	(m)	(°)	(m)	(°)
0	10	10	37	43.4	10
10	20	10	37	24.1	15
23	30	10	37	17.5	18
30	40	10	37	N/A	N/A

Table 16.19 Waste Rock Dump Design Criteria

Some contingency capacity was included with each WRD design to account for changes to material mixes and assumed swell factors. Each deposit has individual WRD(s) positioned and designed to minimise haulage distance wherever possible.

Table 16.20 summarises the required capacity of each WRD, as well as the designed capacity. In general, WRD have been designed with 10% - 15% more capacity than required to allow for contingency. However as can be seen some of the smaller satellite pits are considerably more than this, which leaves some opportunity to optimise the dump footprints in subsequent mine planning works. Overall, the additional capacity is approximately 15%.

Mine Area	Required	Designed	Difference	Required
	LCM ('000)	LCM ('000)	LCM ('000)	LCM ('000)
Enioda	7.2	8.8	1.6	22%
Han	7.8	8.3	0.5	7%
Kekeda	4.9	6.0	1.1	23%
Kilosegui	23.9	27.2	3.3	14%
Souwa	31.9	37.1	5.1	16%
Nokpa	12.2	14.0	1.7	14%
Chegue & Chegue South	12.1	13.6	1.4	12%
Total	100.1	114.9	14.9	15%

Table 16.20 Waste Rock Dump Designed Capacity Versus Required Capacity

16.7.5 General Site Arrangement

Site Layout

Most of the mine infrastructure will be centred around the Nokpa/Souwa pits. Kekeda, Han and Enioda are the closest satellite ore bodies with haulage distances of 9.5 km, 12 km and 15 km respectively. These pits will be connected via haul roads to the processing facility stockpiling areas. Kilosegui is 32 km to the south-west and a major haul road will need to be constructed to allow haulage of ore to the processing facilities. This haul road will cross a major highway and this intersection will need to be managed with a flyover or similar crossing.

The main ROM pad and processing facility is located to the northwest of the Souwa pit between the pit and the proposed tailings storage facility (TSF). The ROM Pad and associated ore stockpiling areas are east and south east of the process facility respectively. This sits in the arc of the Souwa/Nokpa/Chegue hub area and adjacent to Souwa, the primary mining area for the first few years of mine life. Offices and workshops will also be located in this area.

Haul roads, waste dumps and infrastructure have all been designed with the intention to minimise the impact on the local villages. In the Souwa/Nokpa/Chegue hub area, the haul roads and waste dumps have been designed to maintain as much distance from the local village of Herewedouo as possible.

Figure 16.40 to Figure 16.42 provide the general arrangement of the different mining areas.

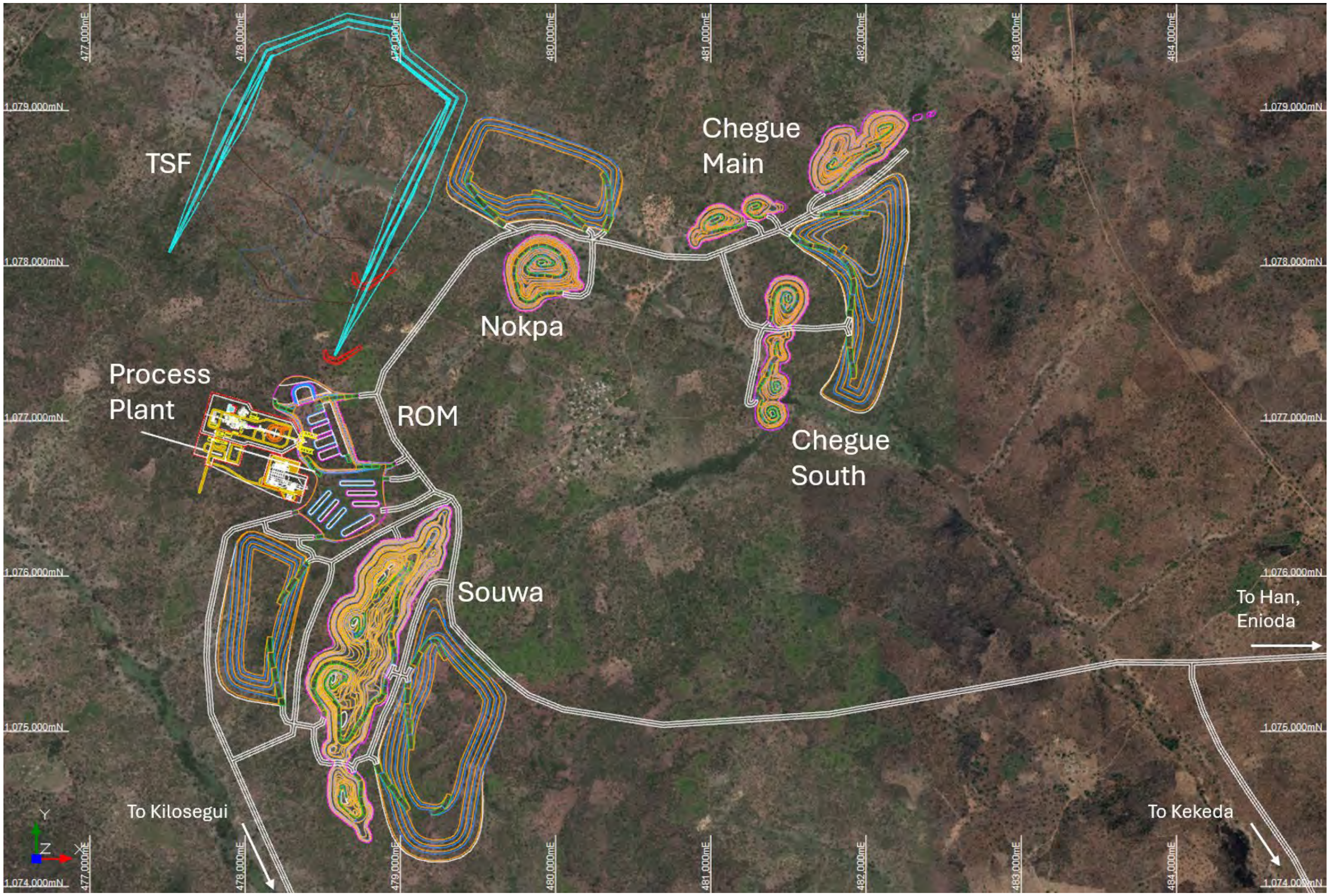


Figure 16.40 General Site Arrangement - Hub Mining Area

Figure 16.41 General Site Arrangement - Kekeda, Han, Enioda Satellite Mining Area

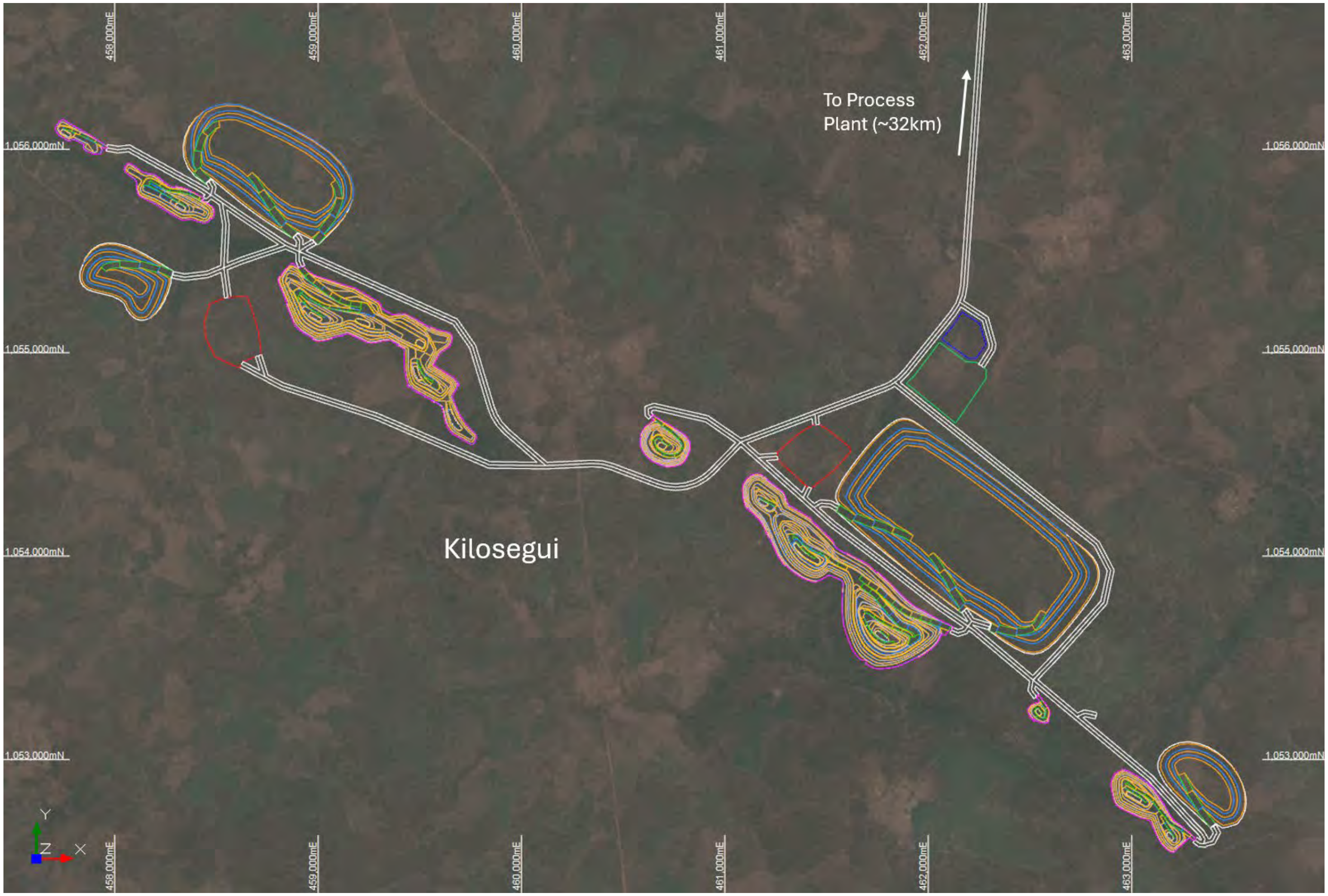


Figure 16.42 General Site Arrangement - Kilosegui Mining Area

ROM Pad Layout

The ROM pad has been designed to balance direct tip via haul trucks into the crusher, as well as providing sufficient storage capacity from the skyway. The pad is designed at 15 m above the natural topography to match the assumed height of the crusher feed bin. A 5 m skyway sits on top of the ROM pad for the haul trucks to dump on four separate fingers. Total storage capacity on the ROM pad is for approximately 250 kt of material (i.e. 3 - 3.5 weeks of plant feed depending on material type).

The ROM pad has been designed with double lane ramps to allow flexibility in vehicle movements. Operationally, these could run as single lane ramps to control vehicle movements across the skyway, depending on the location of mining and the direction of the haul trucks to the ROM.

To the north of the ROM an area has been set aside for long-term haul truck ore storage. This area will be filled by haul trucks tipping off the edge of the ROM pad. This has a nominal capacity of 400 kt. However, it can be expanded as required.

To the south is the road train stockpile. Material from Kilosegui, Han, Kekeda and Enioda will be transported here, before being rehandled into the crusher. The surface in this location is assumed to be roughly natural topography level with a small amount of fill or sheeting to ensure a level surface. There is capacity for around 300 kt of road train haulage and another 100 kt or more for long-term road train haulage.

The ROM design will require approximately 3,000,000 LCM of material for construction. It is anticipated that a much smaller ROM pad will be built during the pre-production period to provide access to the crusher with minimal upfront expenditure. The ROM Pad and associated stockpiling areas will then be progressively built over the first year of production. It is assumed that all fill material required will be sourced from Souwa waste. It is planned that this material will be sourced from suitable waste within the Souwa pit.

The ROM Pad layout is shown in Figure 16.43. It lies more than 750 m to the west of the current limit of the Herewedouo village.

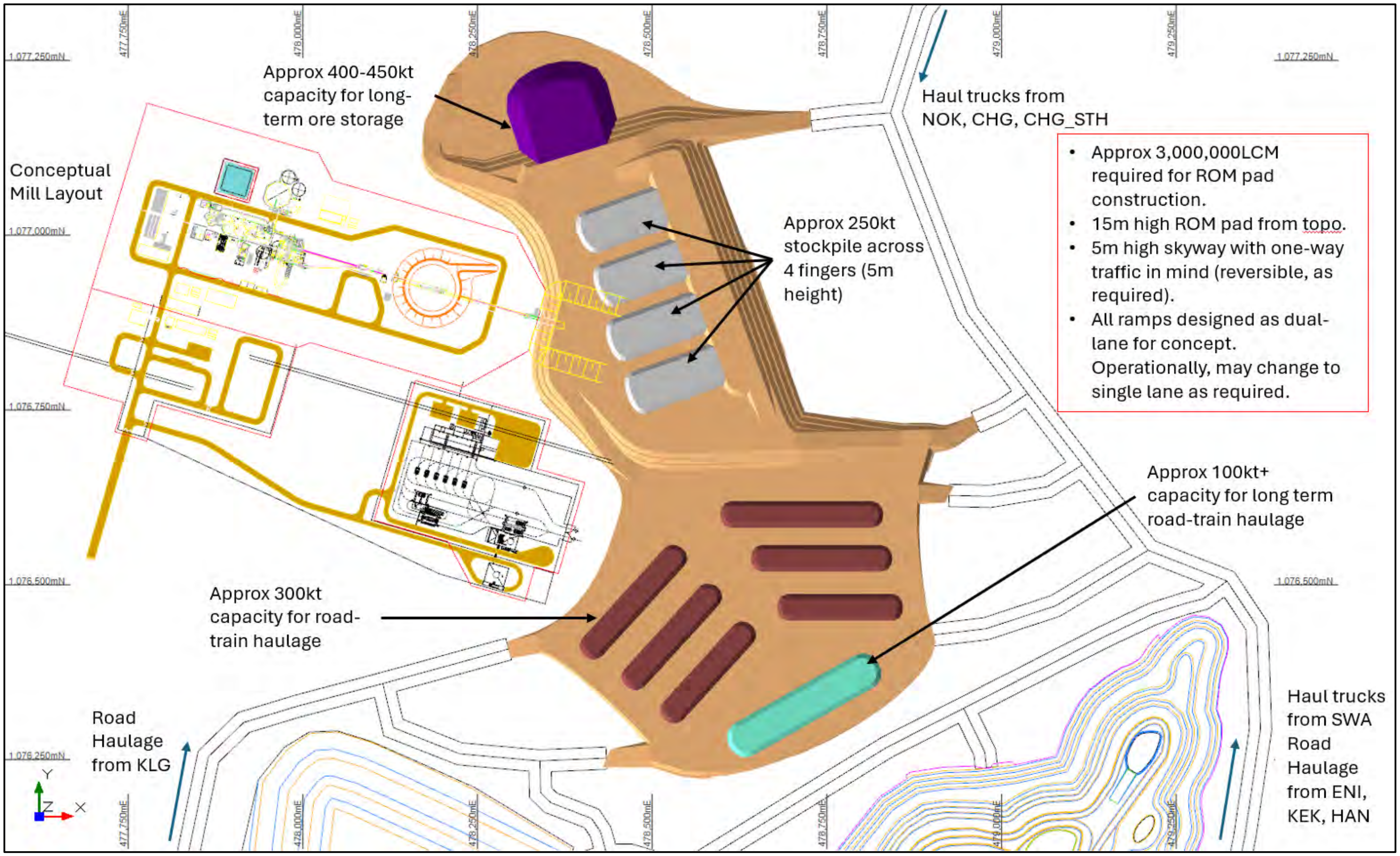


Figure 16.43 ROM Pad Layout Design

16.8 Life of Mine Production Schedule

The purpose of the FS Life of Mine (LOM) production schedule was to generate a practical, realistically achievable schedule that maximises project value within the given constraints and objectives. The FS LOM production schedule:

- Satisfies processing feed requirements;
- Incorporates a ramp-up of operations for mining and processing;
- Minimises any pre-strip and generally delays waste mining where possible;
- Avoids excessive vertical advance rates;
- Mines from a realistic number of areas per period;
- Schedules achievable material movements per area.

The inventories used for the FS LOM production schedule are the final Doropo Ore Reserves as detailed in Section 15.3.4.

The Doropo pits were scheduled using Hexagon MinePlan Schedule Optimizer™ (MPSO).

16.8.1 Scheduling Objective and Constraints

Scheduling Periods

The Life of Mine (LOM) production schedule was developed in periods detailed in Table 16.21. It is based on:

- 1 x quarter (pre-production);
- 24 x monthly periods (i.e. 2 years);
- 12 x quarterly periods (i.e. 3 years);
- Annual periods thereafter.

The schedule assumes mining contractor mobilisation during the two quarters before plant production commences, with initial stripping, clearing and pre-strip mining occurring in the quarter preceding plant production.

Period #	Period Name	Year	Duration
			Months
1	Y-1Q4	Y-1	3
2	Y1M1	Y1	1
3	Y1M2		1
4	Y1M3		1
5	Y1M4		1
6	Y1M5		1

Period #	Period Name	Year	Duration
			Months
7	Y1M6		1
8	Y1M7		1
9	Y1M8		1
10	Y1M9		1
11	Y1M10		1
12	Y1M11		1
13	Y1M12		1
14	Y2M1	Y2	1
15	Y2M2		1
16	Y2M3		1
17	Y2M4		1
18	Y2M5		1
19	Y2M6		1
20	Y2M7		1
21	Y2M8		1
22	Y2M9		1
23	Y2M10		1
24	Y2M11		1
25	Y2M12		1
26	Y3Q1	Y3	3
27	Y3Q2		3
28	Y3Q3		3
29	Y3Q4		3
30	Y4Q1	Y4	3
31	Y4Q2		3
32	Y4Q3		3
33	Y4Q4		3
34	Y5Q1	Y5	3
35	Y5Q2		3
36	Y5Q3		3
37	Y5Q4		3
38	Y6	Y6	12
39	Y7	Y7	12
40	Y8	Y8	12
41	Y9	Y9	12
42	Y10	Y10	12

Table 16.21 LOM Production Schedule Periods

Processing Targets and Constraints

A processing plant feed rate of 5.4 Mt/a for weathered material and 4.0 Mt/a for fresh material was utilised throughout the schedule. The schedule assumed 6,000 operating hrs per year for the crusher and converted the weathered and fresh throughput rates into an hourly rate of 900 t/h and 667 t/h respectively.

A plant ramp up was also utilised as shown in Figure 16.44.

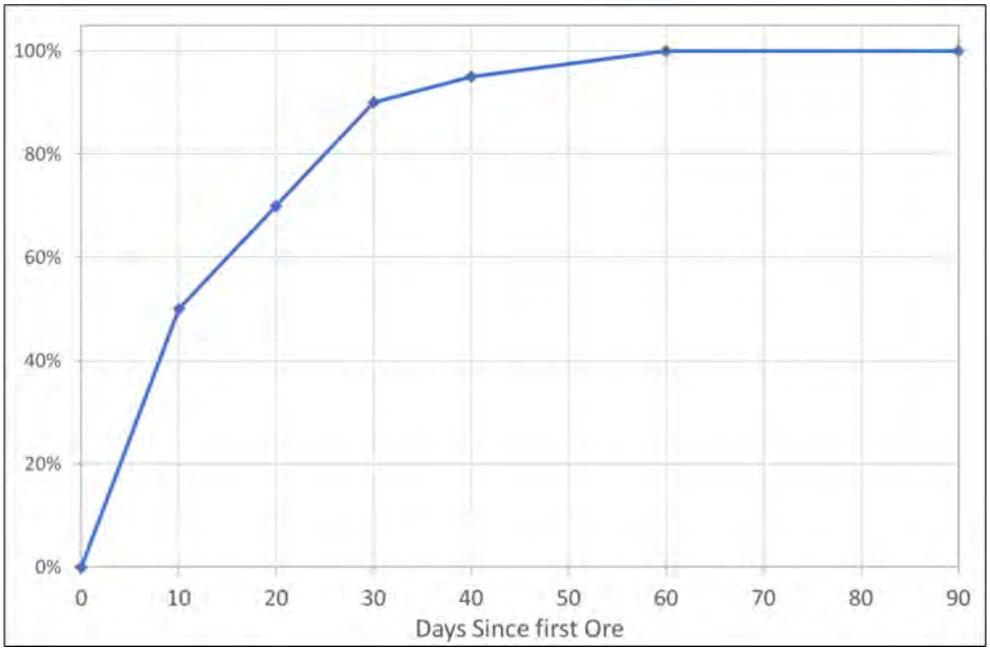


Figure 16.44 Process Plant Ramp-Up (% of Nominal Throughput)

This equated to the following ramp up over the first three months of the schedule:

- Y1M1 58%;
- Y1M2 96%;
- Y1M3 100%.

There were no requirements for feed blending to the process plant.

Mining Production Rates

The LOM Production Schedule was undertaken as a Total Material Movement (TMM) schedule, which assumes the loading unit utilisation is the constraint and that sufficient trucking is available to “service” the loader. Effective equipment management will be necessary to ensure that the truck fleet size is maintained at a reasonable level.

The digging fleet for the LOM production schedule was based on the contractor submissions to the RBP and assumed a combination of 150-tonne (Cat 6015) and 200-tonne (Cat 6020) excavators matched to a 92-tonne payload truck (Cat 777). Table 16.22 and Table 16.23 show the calculation of global productivity for the Cat 6015 and Cat 6020 respectively based on the LOM split of material types.

Description	Unit	Waste			Ore		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
Loading Unit bucket size	m³	8.1	8.1	8.1	8.1	8.1	8.1
Bucket fill factor	%	95%	95%	95%	95%	95%	95%
Calculated Bucket Capacity	m³	7.7	7.7	7.7	7.7	7.7	7.7
Dry Bulk Density	%	1.88	2.31	2.69	1.83	2.33	2.69
Swell	%	15%	20%	25%	15%	20%	25%
Moisture	%	3%	4%	5%	3%	4%	5%
Loose Wet Density	wmt/m³	1.68	2.00	2.26	1.64	2.02	2.26
Rated Lift	t	14.6	14.6	14.6	14.6	14.6	14.6
Calculated Lift	t	13.0	15.4	17.4	12.6	15.5	17.4
Actual Bucket Payload	wmt	13.0	14.6	14.6	12.6	14.6	14.6
	m³	7.7	7.3	6.5	7.7	7.2	6.5
Bucket Cycle Time	secs	33	33	33	33	33	33
Truck Tray Fill Factor	%	95%	95%	95%	95%	95%	95%
Truck Rated Capacity	m³	57.2	57.2	57.2	57.2	57.2	57.2
	t	92.2	92.2	92.2	92.2	92.2	92.2
Max. Dump Truck Capacity	wmt	92.2	92.2	92.2	92.2	92.2	92.2
Passes per truck theor.	#	7.1	6.3	6.3	7.3	6.3	6.3
Actual # Passes	#	7.0	6.0	6.0	7.0	6.0	6.0
Actual Truck Payload	wmt	90.7	87.5	87.5	88.3	87.5	87.5
Truck De-rating Factor	%	98%	95%	95%	96%	95%	95%
First Bucket Drop Time	secs	10	10	10	10	10	10
Loading spot time	minutes	25	25	25	25	25	25
Total load Time	secs	143	143	143	143	143	143
Working Time	Mins per Hr	50	50	50	45	45	45
Loading Unit Productivity	wmt/Op. Hr	1,168	1,312	1,312	1,023	1,181	1,181
	dmt/Op. Hr	1,133	1,260	1,247	992	1,134	1,122
	SMU Factor	88%	88%	88%	88%	88%	88%
	dmt/Eng. Hr	996	1,107	1,096	872	996	986
	Mdmt/year	6.29	6.99	6.92	5.51	6.30	6.23
Prop. Material	%	23%	27%	32%	3%	6%	10%
Average Production Rate	Mdmt/year	6.65					
Scheduled Production Rate	Mdmt/year	6.00					
Scheduled De-rating	%	-10%					

Table 16.22 Schedule Production Rate Cat 6015

Description	Unit	Waste			Ore		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh
Loading Unit bucket size	m³	12.0	12.0	12.0	12.0	12.0	12.0
Bucket fill factor	%	95%	95%	95%	95%	95%	95%
Calculated Bucket Capacity	m³	11.4	11.4	11.4	11.4	11.4	11.4
Dry Bulk Density	%	1.88	2.31	2.69	1.83	2.33	2.69
Swell	%	15%	20%	25%	15%	20%	25%
Moisture	%	3%	4%	5%	3%	4%	5%
Loose Wet Density	wmt/m³	1.68	2.00	2.26	1.64	2.02	2.26
Rated Lift	t	21.6	21.6	21.6	21.6	21.6	21.6
Calculated Lift	t	19.2	22.8	25.8	18.7	23.0	25.8
Actual Bucket Payload	wmt	19.2	21.6	21.6	18.7	21.6	21.6
	m³	11.4	10.8	9.6	11.4	10.7	9.6
Bucket Cycle Time	secs	36	36	36	36	36	36
Truck Tray Fill Factor	%	95%	95%	95%	95%	95%	95%
Truck Rated Capacity	m³	57.2	57.2	57.2	57.2	57.2	57.2
	t	92.2	92.2	92.2	92.2	92.2	92.2
Max. Dump Truck Capacity	wmt	92.2	92.2	92.2	92.2	92.2	92.2
Passes per truck theor.	#	4.8	4.3	4.3	4.9	4.3	4.3
Actual # Passes	#	4.0	4.0	4.0	4.0	4.0	4.0
Actual Truck Payload	wmt	76.8	86.4	86.4	74.7	86.4	86.4
Truck De-rating Factor	%	83%	94%	94%	81%	94%	94%
First Bucket Drop Time	secs	10	10	10	10	10	10
Loading spot time	minutes	25	25	25	25	25	25
Total load Time	secs	143	143	143	143	143	143
Working Time	Mins per Hr	50	50	50	45	45	45
Loading Unit Productivity	wmt/Op. Hr	1,611	1,813	1,813	1,411	1,631	1,631
	dmt/Op. Hr	1,562	1,740	1,722	1,369	1,566	1,550
	SMU Factor	88%	88%	88%	88%	88%	88%
	dmt/Eng. Hr	1,373	1,529	1,514	1,203	1,376	1,362
	Mdmt/year	8.68	9.66	9.56	7.60	8.70	8.61
Prop. Material	%	23%	27%	32%	3%	6%	10%
Average Production Rate	Mdmt/year	9.19					
Scheduled Production Rate	Mdmt/year	8.00					
Scheduled De-rating	%	-13%					

Table 16.23 Schedule Production Rate Cat 6020

The productivity for these units was then derated for scheduling purposes by 10% to 13% to allow for movement of the digging units between different working areas.

Table 16.24 shows the mining fleet build-up and associated mining capacity by period.

Period #	Cat 6015	Cat 6020	Mining Capacity	
			Mt/yr	Mt/period
Y-1Q4	1	1	14	3.50
Y1M1	2	1	20	1.67
Y1M2	2	1	20	1.67
Y1M3	2	2	28	2.33
Y1M4	2	2	28	2.33
Y1M5	2	2	28	2.33
Y1M6	2	2	28	2.33
Y1M7	2	2	28	2.33
Y1M8	2	2	28	2.33
Y1M9	2	2	28	2.33
Y1M10	2	2	28	2.33
Y1M11	2	2	28	2.33
Y1M12	2	2	28	2.33
Y2M1	2	2	28	2.33
Y2M2	2	2	28	2.33
Y2M3	2	2	28	2.33
Y2M4	2	2	28	2.33
Y2M5	2	2	28	2.33
Y2M6	2	2	28	2.33
Y2M7	2	2	28	2.33
Y2M8	2	2	28	2.33
Y2M9	2	2	28	2.33
Y2M10	2	2	28	2.33
Y2M11	2	2	28	2.33
Y2M12	2	2	28	2.33
Y3Q1	2	2	28	7
Y3Q2	2	2	28	7
Y3Q3	2	2	28	7
Y3Q4	2	2	28	7
Y4Q1	2	2	28	7
Y4Q2	2	2	28	7
Y4Q3	2	2	28	7
Y4Q4	2	2	28	7
Y5Q1	2	2	28	7
Y5Q2	2	2	28	7
Y5Q3	2	2	28	7
Y5Q4	2	2	28	7
Y6	2	2	28	28
Y7	2	1	20	28
Y8	2	1	20	28

Period #	Cat 6015	Cat 6020	Mining Capacity	
			Mt/yr	Mt/period
Y9	2	1	20	28
Y10	2	1	20	28

Table 16.24 Digging Fleet and Mining Capacity by Period

Mining Targets and Constraints

Mine scheduling is inherently a trade-off between conflicting objectives and constraints. For the Doropo Project, the primary objective was to maximise NPV while also ensuring practical mining constraints were applied. This included the following:

- Pre-production mining to generate approx. 800 kt of stockpiled ore feed;
- Endeavour to maximise grade in the early years;
- Consider grade against the higher throughput rate of weathered material when "bringing ounces forward";
- Concentrate initial mining in the Hub pits if possible;
- Do not mine more than one satellite pit at any given time, with the exception of Kilosegui. It is assumed that the smaller satellite pits are close enough to the Hub pits that a degree of fleet interchange can occur if required;
- Kilosegui is treated as a separate mining area with its own fleet that is "stranded" to some degree due to the distance. It is assumed a fleet of 1 x Cat 6015 and 1 x Cat 6020 are utilised here with a maximum mining capacity of 14 Mt/a. Kilosegui is also limited to a maximum of 3 Mt/a of ore feed to the process plant in order to limit the amount of truck haulage required to the process plant on top of haulage required from other satellite pits;
- A maximum bench turnover rate per pit of:
  - 2 per month;
  - 5 per quarter;
  - 14 per year.

Pit Staging

Figure 16.45 shows the staging of the pits for the Hub and Satellite pits utilised for the LOM Production Schedule. The following should be noted:

- Souwa is staged as shown previously in Figure 16.27 to Figure 16.31;
- Nokpa utilises the stages as shown in Figure 16.32 and Figure 16.33;
- The three Chegue Main pits are named Stage 1, 2 and 3 from west to east;
- Chegue South is a single stage;
- Kekeda is a single stage;

- The two Han pits are named Stage 1 and 2 from north to south;
- The two Enioda pits are named Stage 1 and 2 from south to north.

Figure 16.46 shows the staging of the pits for Kilosegui utilised for the LOM Production Schedule. It should be noted that the stage shown as KLG East Stage 3 was not mined as part of the LOM Production Schedule as it was considered too small and added no value.

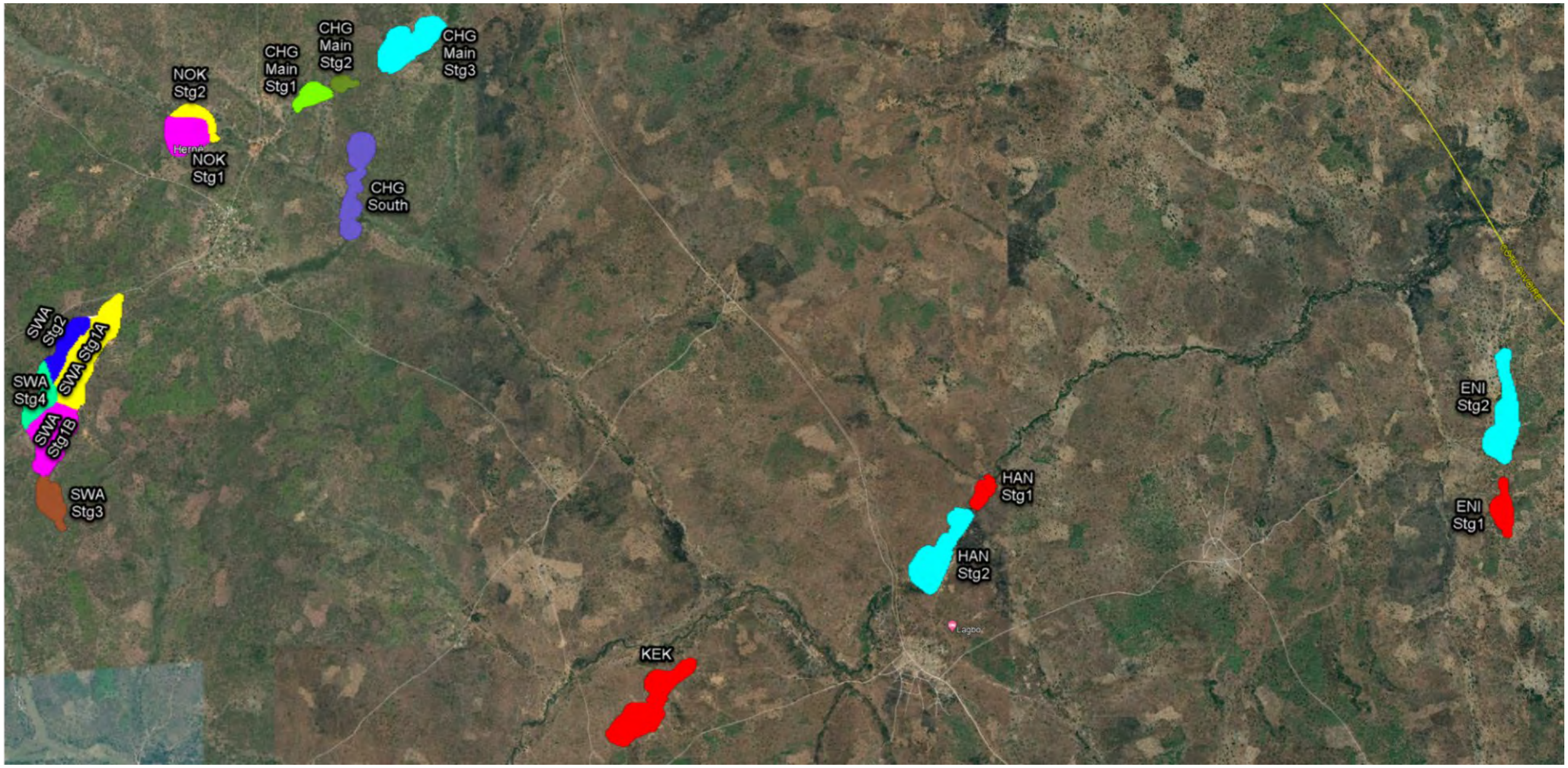


Figure 16.45 Pit Stage Layout for LOM Production Schedule (Hub and Satellite Pits)

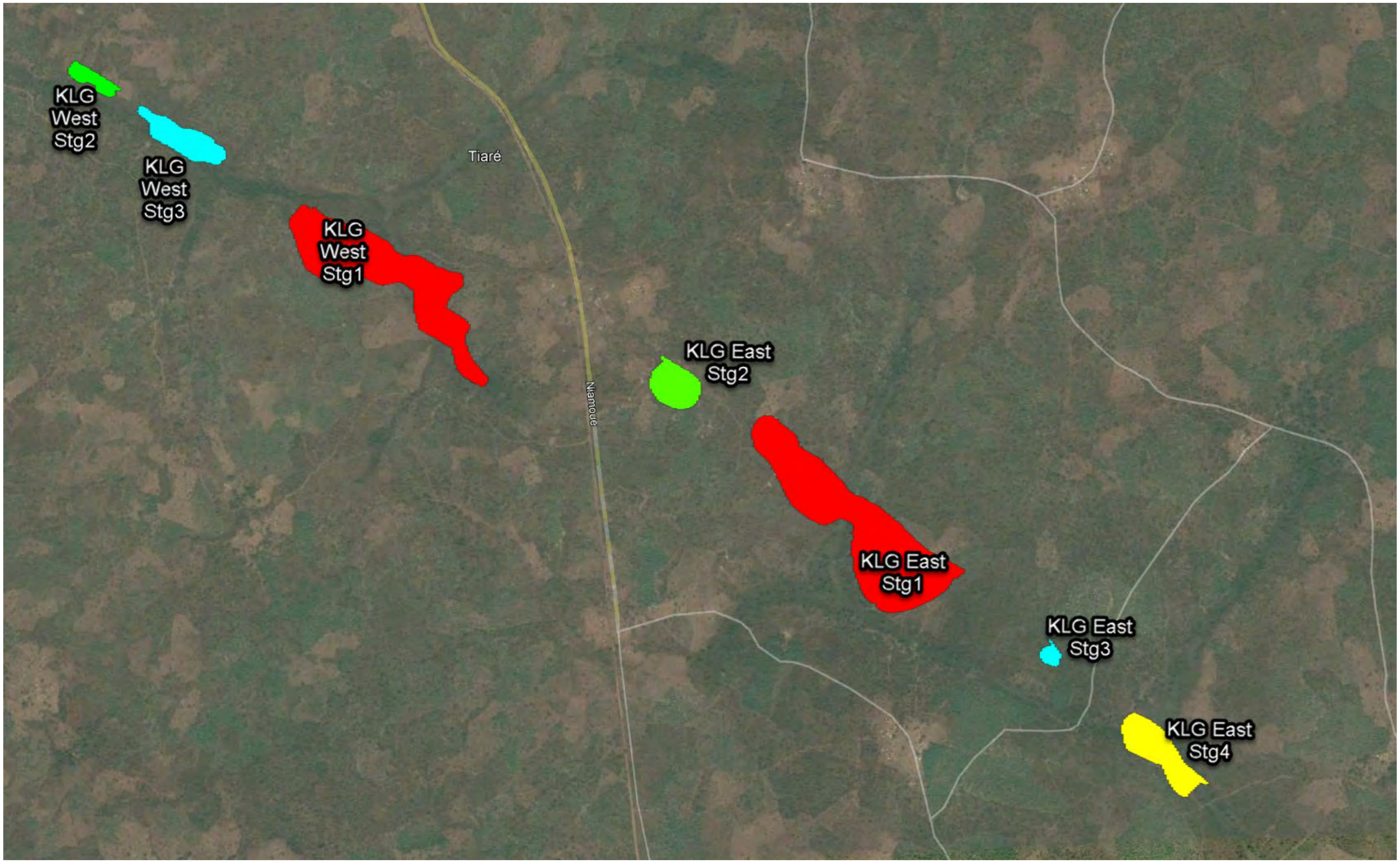


Figure 16.46 Pit Stage Layout for LOM Production Schedule (Kilosegui)

Stockpiling Strategy

An initial target of 800 kt of ore on stocks at the start of production was set for the pre-production period. An ongoing minimum stockpile level of approximately 400 kt was targeted through the mine-life, which equates to approx. 1 month of plant feed at the LOM average plant feed rate.

16.8.2 LOM Production Schedule Results

The LOM Production Schedule results are present in Figure 16.47 to Figure 16.53. They are presented in an annualised form for clearer presentation with the exception of Figure 16.48, which shows the first 2 Years at a monthly granularity. The physicals are then presented by quarters and years in Table 16.25 and Table 16.26.

Figure 16.47 LOM Total Mining by Mining Area (Annual) and Figure 16.48 Total Mining by Mining Area (Pre-Production to End Year 2).

The figures show the total mined material by area. The hub pits are shown as solid fill, with the satellite pit in hatched fill for clarity. As can be seen, mining concentrated in the hub deposits for the first 2 years on production. The first satellite pits mined are Han and Kilosegui, which both commence in the third quarter of Year 2. The schedule does not mine the maximum scheduled mining capacity if it is not reached, as can be seen in Y2-M9.

Figure 16.49 Mining Sequence by Pit Stage

This figure gives further granularity of the mining sequence by stage in quarters and then years. The following is noted:

- Han is completed by Year 4 with both stages being mined concurrently;
- Kilosegui is consistently mined for six (6) years from Y2Q4 through to Year 8. It was attempted to concentrate the mining on the eastern side only until close to completion and then progress to the western side. However, with the other constraints applied to Kilosegui, this was not possible, and the east and west are mined concurrently from Y5Q4 until Year 7;
- SWA Stage 1 and Stage 2, Chegue Main Stage 1 and Stage 2 and Nokpa Stage 1 are completed by Y3Q4;
- From Y3Q4 until Y5Q1 (i.e. 18 months) the Hub deposits are not mined. Kilosegui is mined consistently over this period, with Han finishing and Enioda starting in Y4Q2.

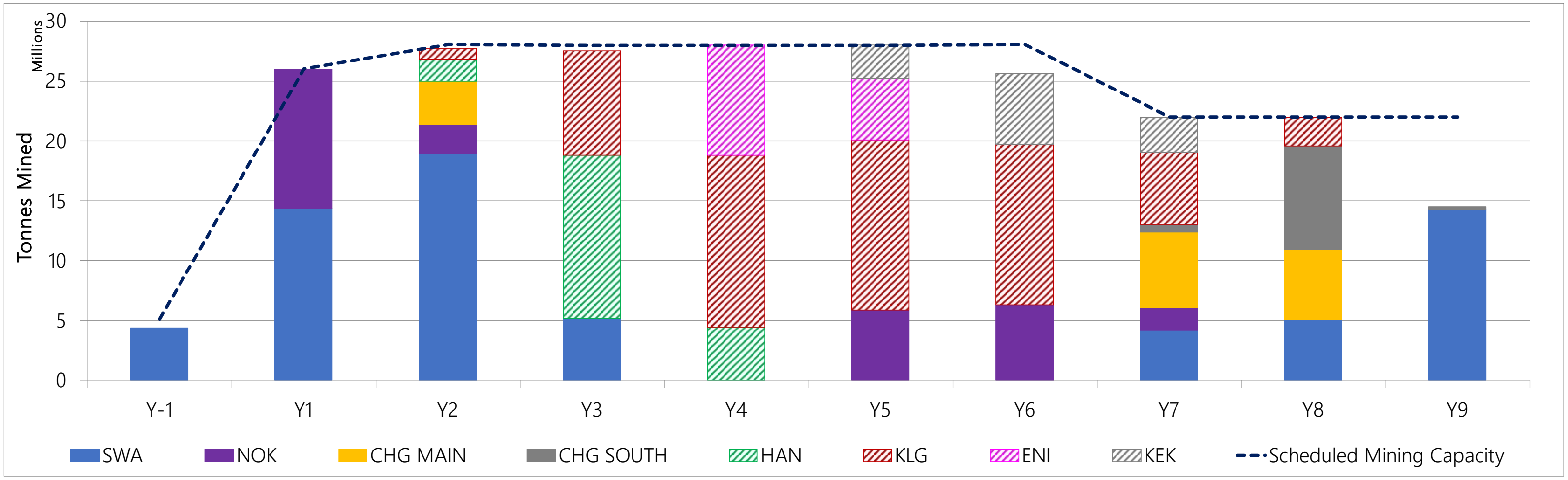


Figure 16.47 LOM Total Mining by Mining Area (Annual)

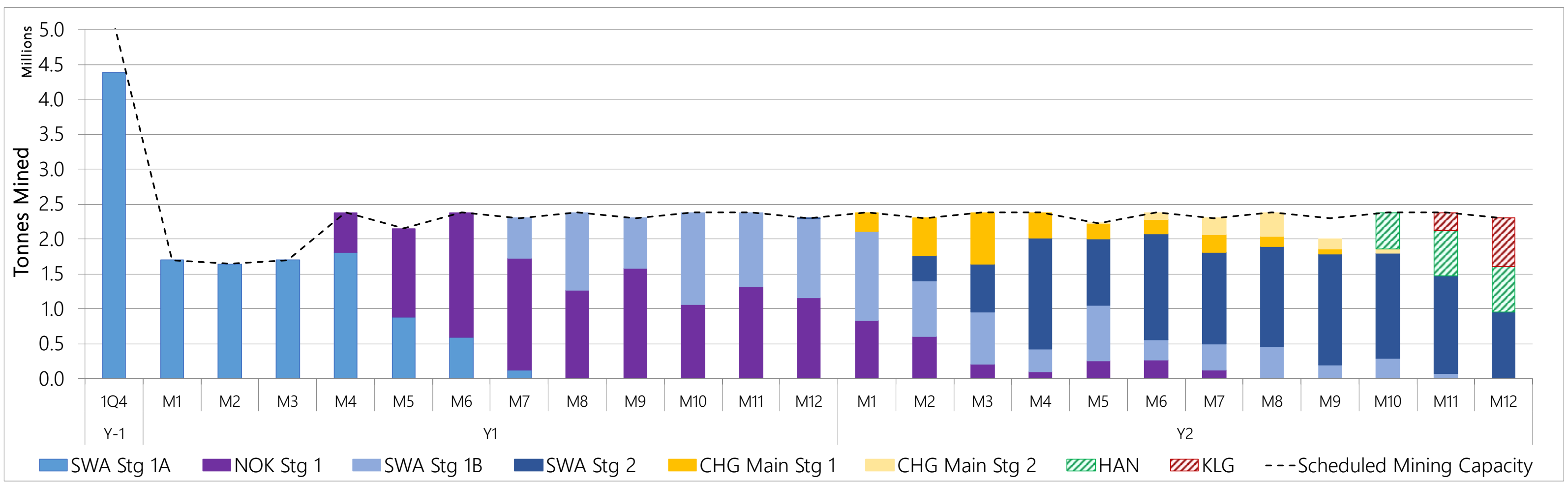


Figure 16.48 Total Mining by Mining Area (Pre-Production to End Year 2)

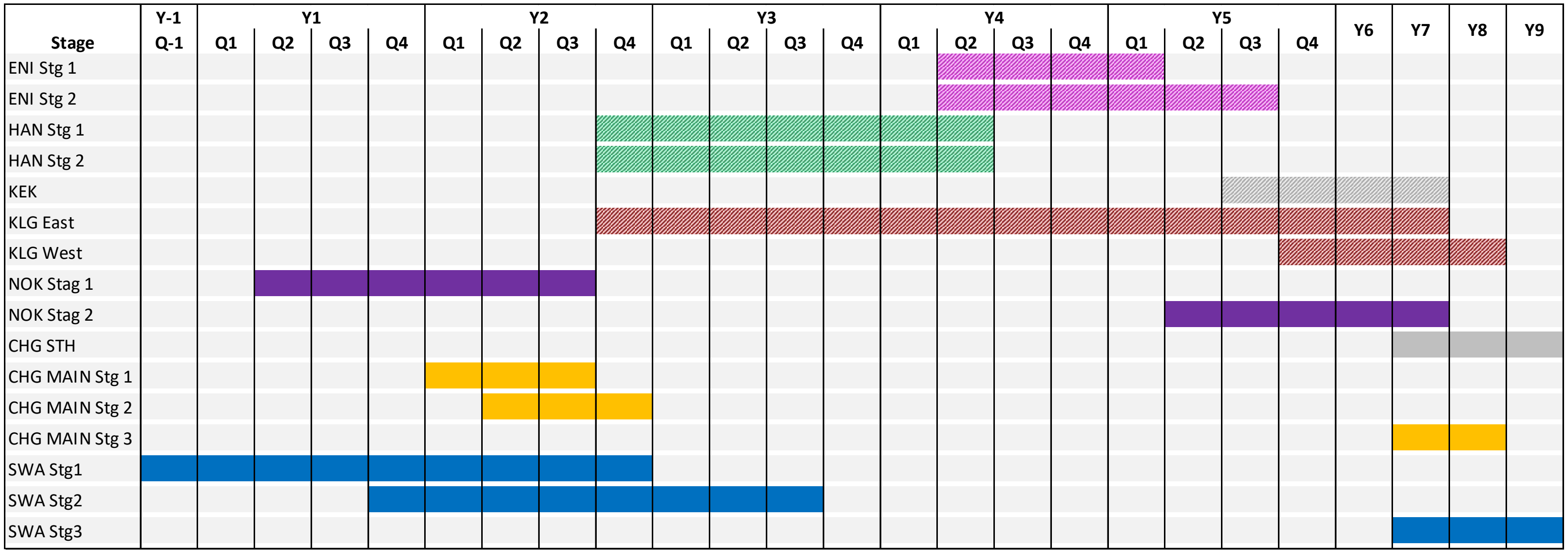


Figure 16.49 Mining Sequence by Pit Stage

Figure 16.50 Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade

As shown in this figure, there is a relatively small amount of “scheduled” stockpile reclaim as part of the LOM Production Schedule. The target of higher head grade in the early years has been meet, with the expected grade spike at the end of the minelife as the high-grade ore at the base of Souwa presents. As can be seen there is also a higher throughput achieved through the first two (2) years of production as the weathered ore from the Hub deposits is prioritised.

Figure 16.51 Utilisation of Plant Capacity (hours).

This figure shows that the plant is fully utilised on the basis of throughput and plant hours described *Processing Targets and Constraints* above.

Figure 16.52 Feed Ounces, Recovered Ounce and Recovery.

This figure also shows that recovery is higher over the first two (2) years of production, averaging 91.8%, as opposed to an average of 87.8% for the remainder of the mine life. This is another function of targeting the weathered ore in the Hub deposits.

Figure 16.53 Stockpile Balance and Grade by Weathering.

This chart shows the higher initial stockpiling balance of 800 kt, which is drawn down in Year 1 to approximately 400 kt. It is then maintained at 400-500 kt through to Year 4 until climbing back to c.800 kt+ before being drawn down at the end of the mine life. The stockpiled grade in the latter half of the mine life is generally much lower than the direct feed grade for the same period, with a comparison of 1.18 g/t versus 1.61 g/t for Fresh ore and 1.05 g/t versus 0.79 g/t for Weathered ore.

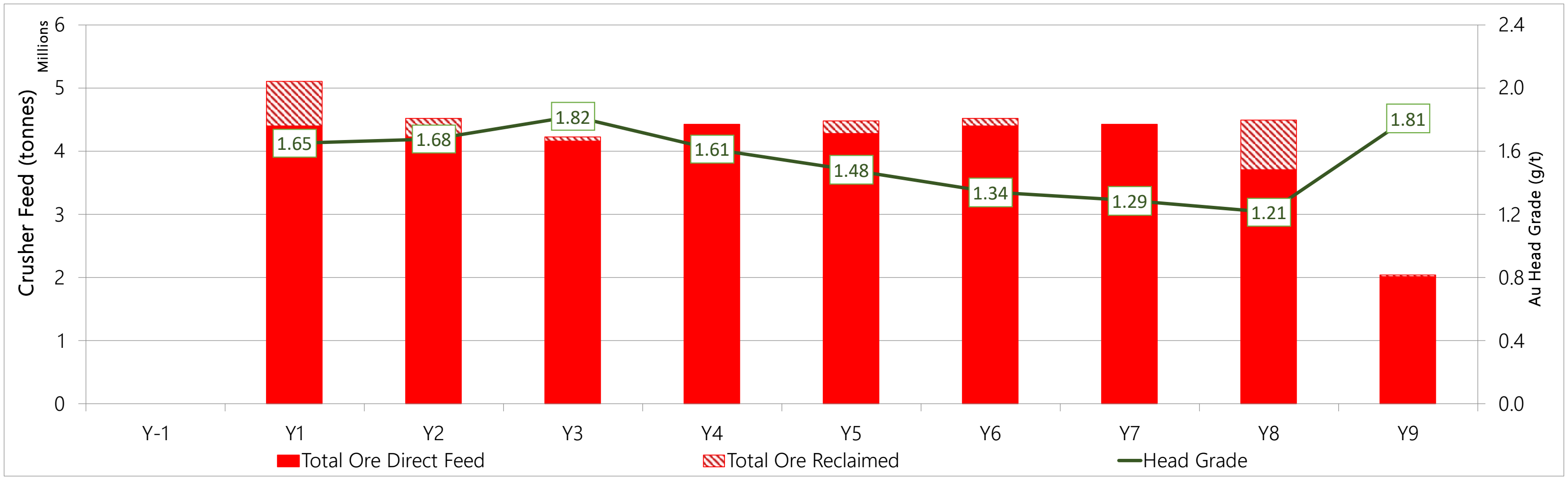


Figure 16.50 Process Plant Feed by Direct and Stockpile Reclaim and Feed Grade

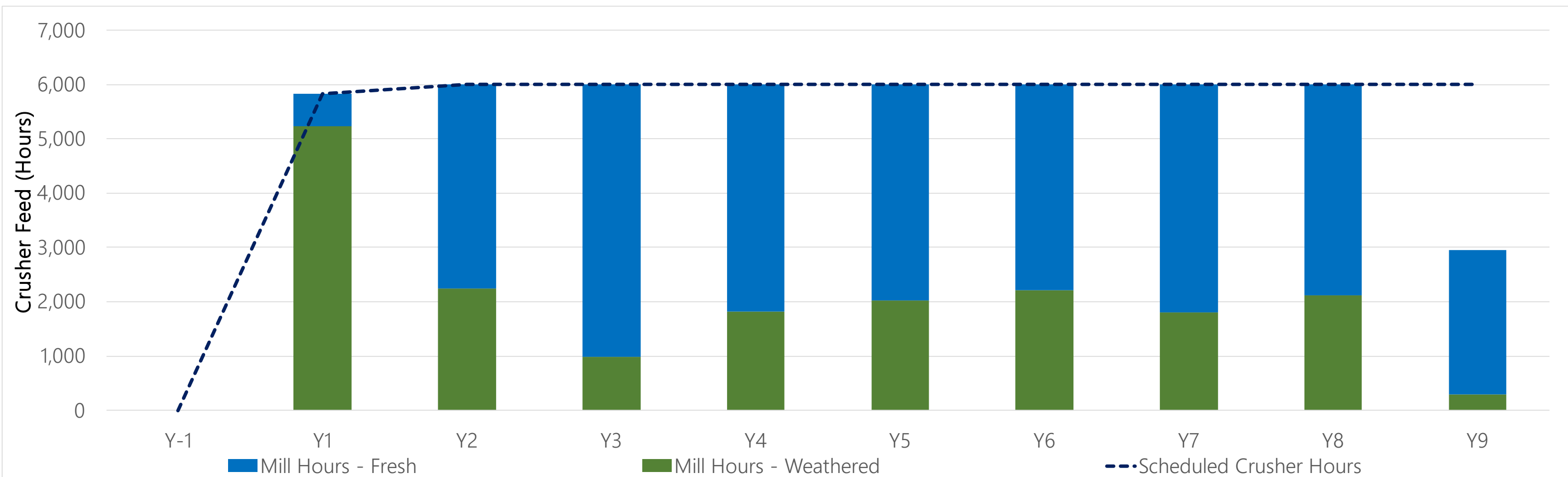


Figure 16.51 Utilisation of Plant Capacity (hours)

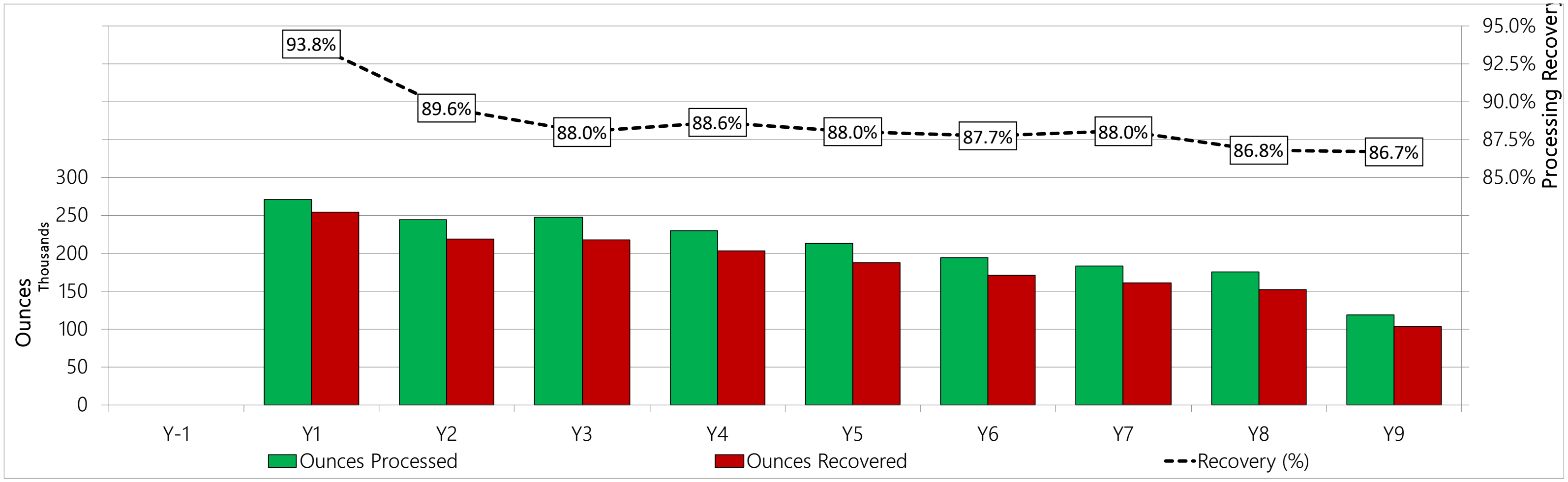


Figure 16.52 Feed Ounces, Recovered Ounce and Recovery

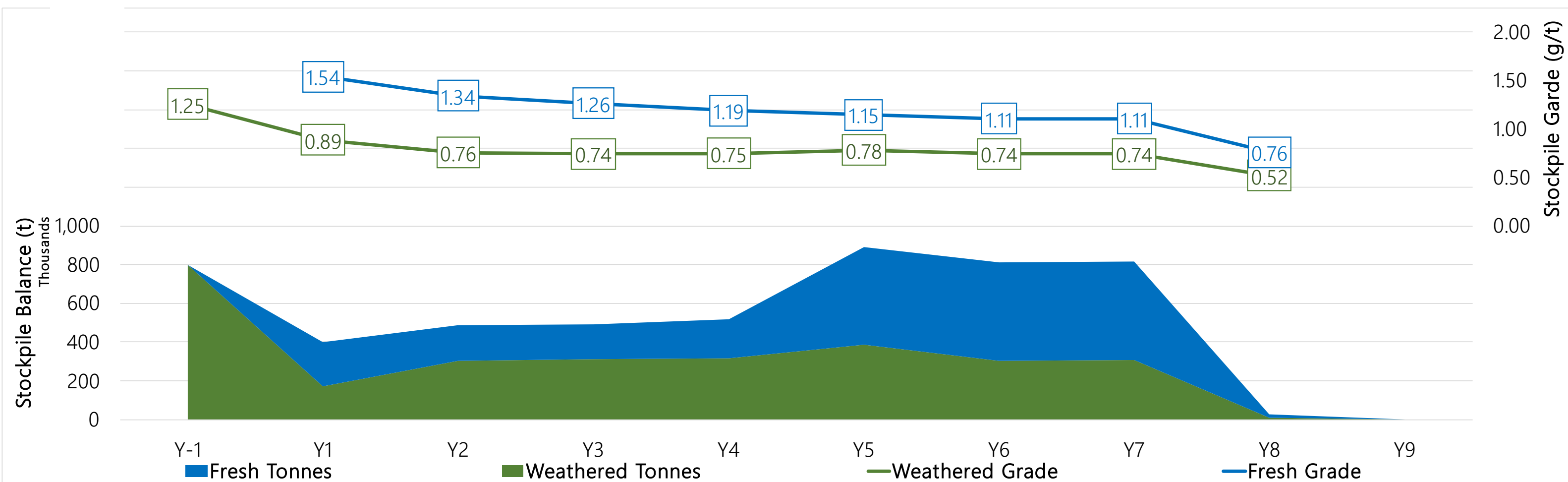


Figure 16.53 Stockpile Balance and Grade by Weathering

Items	Units	Total	Y-1	Y1				Y2				Y3			
			Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Total Material Mined	Mt	225.8	4.4	5.0	6.9	7.0	7.1	7.1	7.0	6.7	7.1	7.1	6.9	6.5	7.1
Total Material Moved	Mt	228.0	4.4	5.1	7.0	7.5	7.1	7.1	7.0	6.9	7.1	7.1	6.9	6.5	7.1
Ore Mined	Mt	38.2	0.80	1.18	1.28	0.85	1.39	1.19	1.18	1.09	1.15	1.06	1.08	1.05	1.04
	g/t	1.53	1.25	1.72	1.94	1.53	1.51	1.63	1.37	1.58	1.97	1.73	1.72	2.13	1.67
Waste Mined	Mt	187.6	3.6	3.9	5.6	6.1	5.7	5.9	5.8	5.6	5.9	6.0	5.8	5.5	6.0
Strip Ratio	W:O	4.9	4.5	3.3	4.4	7.2	4.1	4.9	4.9	5.1	5.2	5.6	5.4	5.2	5.8
Direct Feed Ore	Mt	36.0		1.2	1.3	0.8	1.2	1.2	1.2	0.8	1.0	1.0	1.0	1.0	1.0
	g/t	1.55		1.7	2.0	1.6	1.5	1.6	1.4	1.8	2.0	1.7	1.8	2.1	1.7
Stockpiled Ore	Mt	2.2	0.8	0.0	0.0	0.1	0.2	0.0	0.0	0.2	0.1	0.0	0.0	0.0	
	g/t	1.18	1.25	0.00	0.82	0.79	1.57	0.67	0.68	0.91	1.33	1.00	1.00	1.02	
Reclaimed Ore	Mt	2.2		0.0	0.1	0.6	0.1		0.1	0.2					0.1
	g/t	1.18		1.90	1.51	1.26	1.09		1.78	1.33					1.31
Ore Processed	Mt	38.2	0.0	1.2	1.3	1.3	1.2	1.2	1.2	1.1	1.0	1.0	1.0	1.0	1.1
	g/t	1.53	0.00	1.72	1.93	1.46	1.48	1.64	1.40	1.68	2.05	1.74	1.75	2.14	1.65
	koz	1,876	0.0	66.3	83.4	62.4	59.1	62.4	55.0	58.7	67.9	58.7	58.6	71.2	58.9
	koz rec.	1,667	0.0	62.7	78.5	58.7	54.5	56.6	50.1	52.2	59.6	51.3	51.4	62.7	52.2
	% rec.	88.9%	0.0%	94.5%	94.1%	94.1%	92.3%	90.8%	91.0%	89.0%	87.8%	87.5%	87.6%	88.1%	88.8%
TMM (Mt)	ENI	14.4													
	HAN	19.9									1.8	3.2	3.4	3.5	3.6
	KEK	11.7													
	KLG	60.1									1.0	1.2	1.6	2.5	3.5
	NOK	28.0			3.6	4.5	3.5	1.6	0.6	0.1					
	CHG STH	9.5													
	CHG MAIN	15.8						1.5	0.9	1.2	0.1				
	SWA	66.4	4.4	5.0	3.3	2.5	3.5	3.9	5.5	5.4	4.2	2.7	1.9	0.6	

Table 16.25 LOM Production Schedule Physicals by Period (Pre-Production to Year 3)

Items	Units	Y4				Y5				Y6	Y7	Y8	Y9
		Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4				
Total Material Mined	Mt	7.1	6.9	7.0	7.1	7.1	6.9	7.0	7.1	25.6	22.0	22.0	14.5
Total Material Moved	Mt	7.1	6.9	7.0	7.1	7.1	6.9	7.1	7.1	25.8	22.0	22.8	14.5
Ore Mined	Mt	1.13	1.07	1.13	1.11	1.19	1.13	1.28	1.24	4.44	4.42	3.71	2.01
	g/t	1.57	1.88	1.47	1.52	1.46	1.40	1.39	1.55	1.34	1.28	1.26	1.82
Waste Mined	Mt	5.9	5.8	5.8	5.9	5.9	5.8	5.7	5.8	21.2	17.6	18.3	12.5
Strip Ratio	W:O	5.2	5.4	5.2	5.4	4.9	5.1	4.4	4.7	4.8	4.0	4.9	6.2
Direct Feed Ore	Mt	1.1	1.1	1.1	1.1	1.1	1.1	1.0	1.0	4.4	4.4	3.7	2.0
	g/t	1.6	1.9	1.5	1.5	1.5	1.4	1.5	1.6	1.3	1.3	1.3	1.8
Stockpiled Ore	Mt	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.2	0.0	0.0	0.0	
	g/t	0.89	1.14	1.07	0.88	1.43	0.98	1.11	1.20	0.86	0.49	0.62	
Reclaimed Ore	Mt			0.0				0.1	0.1	0.1		0.8	0.0
	g/t			1.43				1.38	1.05	1.06		0.98	0.66
Ore Processed	Mt	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	4.5	4.4	4.5	2.0
	g/t	1.58	1.88	1.48	1.52	1.46	1.42	1.46	1.59	1.34	1.29	1.21	1.81
	koz	56.2	64.7	54.3	54.2	53.8	49.3	53.8	56.1	194.4	182.6	175.4	118.3
	koz rec.	49.8	57.4	48.3	47.7	47.7	43.4	47.9	48.5	170.5	160.8	152.2	102.6
	% rec.	88.6%	88.8%	89.0%	88.1%	88.6%	88.0%	89.1%	86.5%	87.7%	88.0%	86.8%	86.7%
TMM (Mt)	ENI		2.0	3.6	3.6	3.4	1.4	0.4					
	HAN	3.3	1.2										
	KEK							1.2	1.7	5.9	3.0		
	KLG	3.8	3.7	3.4	3.4	3.7	3.7	3.4	3.4	13.5	6.0	2.4	
	NOK						1.9	2.0	2.0	6.2	1.9		
	CHG STH										0.6	8.7	0.2
	CHG MAIN										6.3	5.8	
	SWA										4.2	5.1	14.3

Table 16.26 LOM Production Schedule Physicals by Period (Year 4 to Year 9)

16.9 Mining Cost Rates and Assumptions

16.9.1 Estimation Approach and Accuracy

The mining costs have been developed on the basis of a mining contractor operation. Two suitable contractors were selected from the submissions received in the November 2023 Request for Budget Pricing (RFB) as described in Section 16.4 to provide updated pricing in May 2024 based on an updated development sequence and production schedule which subsequently formed the basis of the FS schedule.

Owner's mining costs developed by Centamin related to management and technical personnel, grade control etc.

The following sections detail the assumptions and rates derived from the selected contractor submission.

The costs estimate was developed in three steps:

- Determining the physicals for each activity by period;
- Determining the unit rates for those activities;
- Calculation of the cost by period by:
  - Multiplying the scheduled quantity by the appropriate rate;
  - Apply any fixed costs on a time-based pro-rata basis;
  - The estimates assumed a fuel price of US$1.05/l.

The costs estimate has been developed to an accuracy of ±15%, in line with the requirements of a study to feasibility level.

16.9.2 Site Establishment and Mobilisation/Demobilisation

The mobilisation/demobilisation for the selected submission fleet is shown in Table 16.27.

Site establishment and associated infrastructure are detailed in Table 16.28.

Description		Make	Model	# Units	Mobilisation		Demobilisation	
					$'000/unit	$'000	$'000/unit	$'000
Mining Equipment	Excavator	Cat	6015B	4	$370.5	$1,482.0	$52.3	$209.2
		Cat	6020B	1	$328.7	$328.7	$26.4	$26.4
	Mine Truck	Cat	777E	24	$114.5	$2,748.0	$7.7	$184.8
	Road Truck	Volvo	FMX460	20	$55.7	$1,114.0	$6.9	$138.0
	Front End Loader	Cat	992K	3	$100.0	$302.0	$7.0	$21.00
		Cat	980	2	$7.5	$15.0	$0.0	$0.0
	Drill Type	Epiroc	D65	7	$55.7	$389.9	$6.9	$48.3
	Track Dozer	Cat	D9R	7	$64.9	$454.3	$9.8	$68.6

	Description	Make	Model	# Units	Mobilisation		Demobilisation	
					$'000/unit	$'000	$'000/unit	$'000
	Grader	Cat	16M	3	$70.7	$212.1	$6.9	$20.7
	Water Truck	Cat	777E	2	$114.5	$343.5	$7.7	$23.1
	Rockbreaker	Cat	349	1	$28.2	$28.2	$9.8	$9.8
	Support Dump Truck	Cat	777D/E	4	$114.5	$463.6	$7.7	$31.2
	Subtotal			78		$7,881.3		$781.1
Ancillary Equipment & Personnel	Vib. Roller	Cat	CS654	1	$7.5	$7.5	$0.0	$0.0
	Toolcarrier	Cat	IT28	1	$20.0	$20.0	$17.325	$17.3
	Fuel truck	Mercedes		3	$7.5	$22.5	$0.0	$0
	Pump	Sykes	CP150	2	$9.5	$19.0	$0.0	$0.0
	Pump	Sykes	HH200	11	$15.2	$167.2	$0.0	$0.0
	Service Truck	Mercedes		2	$7.5	$15.0	$0.0	$0.0
	Lighting Plant	Himoinsa		14	$11.7	$163.8	$0.0	$0.0
	Light Vehicle	Toyota		32	$3.7	$118.4	$0.0	$0.0
	Town Bus	Toyota		2	$7.5	$15.0	$0.0	$0.0
	Tyre Handler			1	$7.5	$7.5	$0.0	$0.0
	Road Haulage - Grader	Cat	14M	2	$39.2	$78.4	$0.0	$0.0
	Road Haulage - Watercart	Volvo	FMX460	2	$69.0	$138.0	$12.1	$24.2
	Subtotal					$772.3		$41.5
	Miscellaneous Items[1]			1	$297.4	$297.4	$172.8	$172.8
	Personnel			1	$157.8	$157.8	$0.0	$0.0
	Total					$8,950.6		$995.4

[1] *Minor Items, tooling, etc*

Table 16.27 Selected RFQ Submission - Mobilisation and Demobilisation

Description	$'000
Mining Contactor Office	$669
Ablutions Facility	$132
HME Workshop & Stores	$1,628
Wash Bay	$195
Explosives Supplier Infrastructure	$421
Turkey Nest - Souwa, Chegue/Nokpa	$89
Satellite Pit Infrastructure	$1,391
Miscellaneous	$319
Total	$4,844

Table 16.28 Selected RFQ Submission - Site Establishment and Infrastructure

16.9.3 Primary Production Rates

Table 16.29 details the drill and blast parameters and costs from the selected RBP submission. The contractor elected to utilise emulsion product across all material types at a density of 1.20 g/cm^3. There was a consistent pattern for oxide and transition material, with fresh material split 50/50 between two pattern sizes.

Material Type	Hole Diameter	Sub-Drill	Burden	Spacing	Powder Factor	Rate	Volume Split	Weighted Average
	mm	m	m	m	kg/bcm	$/bcm	%	$/bcm
Oxide	140	0.5	6.6	7.6	0.20	$0.66	15%	$0.89
		0.5	5.9	6.8	0.25	$0.82	25%	
		0.5	5.4	6.2	0.30	$0.98	60%	
Trans	127	0.5	5.1	5.9	0.30	$1.03	5%	$1.48
		0.5	4.5	5.2	0.40	$1.32	45%	
		0.7	4.1	4.7	0.50	$1.66	50%	
Fresh	127	0.8	3.8	4.4	0.60	$2.13	5%	$2.80
		0.9	3.6	4.1	0.70	$2.46	20%	
		1.0	3.4	3.9	0.80	$2.79	40%	
		1.2	3.3	3.8	0.90	$3.06	30%	
		1.3	3.2	3.7	1.00	$3.30	5%	

Table 16.29 Selected RFQ Submission - Drill and Blast Rates ($/BCM)

In addition, there was approximately 301,750 LM of presplit drilling (20 m vertical holes, 75 degree wall angle at 1.5 m spacing) included in the RBP, which was priced at $24.51/LM.

Load and haul rates were provided by deposit by bench for the three weathering zones. Table 16.30 and Table 16.31 below provide the load and haul by deposit by bench averaged across material types for waste and ore respectively.

Bench	Kekeda	Kilosegui	Souwa	Bench	Enioda	Han	Nokpa	Chegue
385		$2.49		325				$2.38
380		$2.24		320				$2.30
375		$2.71		315				$2.52
370		$2.82		310			$2.61	$2.57
365		$3.07		305			$2.68	$2.63
360		$3.38		300	$2.32		$2.79	$2.83
355		$3.39		295	$2.75		$2.96	$3.02
350		$3.30	$2.64	290	$2.59		$3.28	$3.11
345		$3.60	$2.70	285	$2.57		$3.25	$3.31
340		$3.67	$2.79	280	$2.65	$2.86	$3.56	$3.37
335		$3.90	$2.64	275	$2.78	$3.08	$3.90	$3.52
330		$4.06	$2.64	270	$2.96	$3.28	$4.18	$3.83
325		$4.22	$2.77	265	$3.13	$3.91	$4.29	$4.20
320		$4.29	$2.83	260	$3.42	$4.15	$4.33	$4.33
315		$4.32	$2.93	255	$3.55	$4.20	$4.40	$4.64
310		$4.32	$3.07	250	$3.79	$4.23	$4.55	$4.76
305		$4.46	$3.22	245	$4.01	$4.29	$4.65	$4.82
300	$2.68	$4.46	$3.43	240	$4.19	$4.38	$5.01	$5.01
295	$2.66	$4.80	$3.47	235	$4.26	$4.47	$5.04	$5.14
290	$2.63	$4.92	$3.70	230	$4.34	$4.49	$5.12	$5.26
285	$2.67	$4.98	$3.97	225	$4.37	$4.49	$5.15	$5.49
280	$2.74	$5.06	$4.28	220	$4.53	$4.43	$5.14	$5.65
275	$2.89	$5.18	$4.52	215	$4.72	$4.57	$5.18	
270	$3.06	$5.26	$5.03	210	$4.66	$4.94	$5.22	
265	$3.30	$5.49	$4.95	205	$5.32	$5.01	$5.36	
260	$3.92	$5.44	$4.95	200	$5.19	$5.08	$5.29	
255	$4.15	$5.77	$5.00	195		$5.14	$5.33	
250	$4.24	$5.78	$5.08	190		$5.24	$5.37	
245	$4.29	$5.75	$5.05	185		$5.45	$5.41	
240	$4.35	$6.20	$5.48	180		$5.43	$5.43	
235	$4.35	$6.44	$5.26	175			$5.89	
230	$4.49	$6.24	$5.28	170			$5.92	
225	$4.49		$5.34	165			$6.08	
220	$4.67		$5.36	160			$6.15	
215	$5.10		$5.72	155			$6.28	
210	$5.24		$5.92	150			$6.39	
205	$5.30		$5.99	145			$6.43	
200			$6.12	140			$6.34	
195				135			$7.25	
190				130			$6.82	

Table 16.30 Selected RFQ Submission - Waste Load and Haul Rates ($/BCM)

Bench	Kekeda	Kilosegui	Souwa	Bench	Enioda	Han	Nokpa	Chegue
385				325				
380		$1.95		320				$3.93
375		$3.47		315				$4.21
370		$3.78		310				$4.24
365		$3.96		305			$4.02	$4.34
360		$4.10		300	$2.15		$4.10	$4.58
355		$4.16		295	$2.97		$4.17	$4.73
350		$4.02		290	$2.81		$4.24	$4.77
345		$4.34	$2.32	285	$2.99		$4.07	$4.86
340		$4.61	$3.14	280	$3.07		$4.56	$5.07
335		$4.69	$3.22	275	$3.20		$4.86	$5.41
330		$4.97	$3.31	270	$3.34	$3.68	$5.31	$5.67
325		$5.16	$3.26	265	$4.29	$4.01	$5.46	$6.14
320		$5.10	$3.34	260	$3.77	$4.49	$5.53	$6.52
315		$5.19	$3.50	255	$3.91	$4.74	$5.66	$6.76
310		$5.22	$3.68	250	$4.07	$4.87	$5.80	$6.93
305		$5.30	$3.91	245	$4.32	$4.95	$5.90	$6.97
300	$3.19	$5.34	$4.29	240	$4.69	$4.99	$6.44	$7.07
295	$3.17	$5.62	$4.38	235	$4.84	$5.13	$6.48	$7.31
290	$3.11	$5.69	$4.66	230	$5.02	$5.14	$6.58	$7.27
285	$3.15	$5.86	$5.08	225	$5.06	$5.14	$6.62	$7.38
280	$3.32	$5.92	$5.41	220	$5.23	$5.08	$6.56	$7.55
275	$3.47	$5.97	$5.56	215	$5.31	$5.23	$6.60	
270	$3.70	$6.26	$5.61	210	$5.36	$5.27	$6.63	
265	$3.79	$6.36	$5.66	205	$6.08	$5.71	$6.67	
260	$4.37	$6.50	$5.66	200	$5.92	$5.79	$6.50	
255	$4.71	$6.75	$5.83	195		$5.85	$6.67	
250	$4.91	$6.81	$5.86	190		$5.95	$6.75	
245	$4.97	$6.77	$5.98	185		$6.06	$6.83	
240	$5.03	$7.11	$6.62	180		$6.14	$6.88	
235	$5.02	$7.37	$6.34	175			$7.07	
230	$5.16	$7.13	$6.33	170			$7.16	
225	$5.16		$6.42	165			$7.26	
220	$5.35		$6.40	160			$7.39	
215	$5.82		$6.46	155			$7.61	
210	$5.96		$6.68	150			$7.86	
205	$6.02		$7.06	145			$7.99	
200			$7.20	140			$7.94	
195			$7.58	135			$9.04	
190				130			$8.45	

Table 16.31 Selected RFQ Submission - Ore Load and Haul Rates ($/BCM)

16.9.4 Other Mining Activities and Fixed Costs

Other mining costs were generated as part of the RBP submissions on a rates basis as detailed in Table 16.32.

Category	Activity		Unit	Rate
Fixed Cost	Monthly Management Fee (105 months)		$/mth	$1,170,520
Site Preparation	Clear & Grub		$/Ha	$4,390
	Topsoil Stripping	Topsoil Depth	m	0.3
		Stripping	$/bcm	$1.72
			$/Ha	$5,163
Road Construction	Mine	Width (30 m)	$/km	$291,000
	Ore Haul	Width (20 m)	$/km	$134,000
Rehabilitation	Topsoil placement	Topsoil Depth	m	0.3
		Rate	$/bcm	$1.10
		Total	$/Ha	$3,300
	Reprofiling		$/Ha	$2,560
	Contouring/ripping/topsoil spreading		$/Ha	$2,110
	Total		$/Ha	$7,970

Table 16.32 Selected RFQ Submission - Fixed Fees and Unit Rates

16.9.5 Ore Handling

Ore rehandle rates were provided for reclaim from short term stockpiles on top of the process plant ROM pad, and FEL/truck rehandle from longer-term low-grade stockpiles and satellite pit haulage stockpiles adjacent to the ROM pad. These rates are shown in Table 16.33.

Activity		Unit	Value
ROM Pad Reclaim	ROM Crusher Feeding[1]	$/t	$0.61
	Proportion of Direct Tip	%	71%
	Weighted ROM Rehandle Rate	$/t	$0.18
Low Grade Stockpile Reclaim[2]		$/t	$1.06

[1] FEL only, assume average 75 m from stockpile finger to crusher bin

[2] FEL and Mine Truck, assume 300 m horz. and 10 m vert. to crusher bin (includes both satellite pit ore and LG S/P reclaim).

Table 16.33 Selected RFQ Submission - Ore Reclaim Rates

Ore haulage from the satellite pits were costed based on a combined loading and haulage rate, tipping onto a stockpile area adjacent to the process plant as detailed in Table 16.34.

Satellite Pit	Distance (km)	Rate ($/t)
Kilosegui	30.2	$5.87
Kekeda	8.4	$2.09
Enioda	14.8	$3.38
Han	8.5	$2.57

Table 16.34 Selected RFQ Submission - Satellite Pit Ore Haulage Rates

16.9.6 Dayworks

A dayworks rates of 3% was applied on top of the combined costs for drilling, blasting, loading, hauling and ore rehandle.

16.10 LOM Mining Contractor Cost Estimate

The annual Life of Mine contractor mining costs are detailed in Figure 16.54, Table 16.35 and Table 16.36.

The total mining cost for the LOM is estimated at US$845.5M which equates to US$3.74/t mined or US$22.12/tonne ore mined.

The costs on a US$/t mined basis are slightly lower over the first half of the mine life (i.e., Y-1 to Y4) at US$3.60/t compared to the second half at US$3.89/t. The same cost profile is achieved on a US$/t ore basis (US$21.75/t vs US$22.47/t).

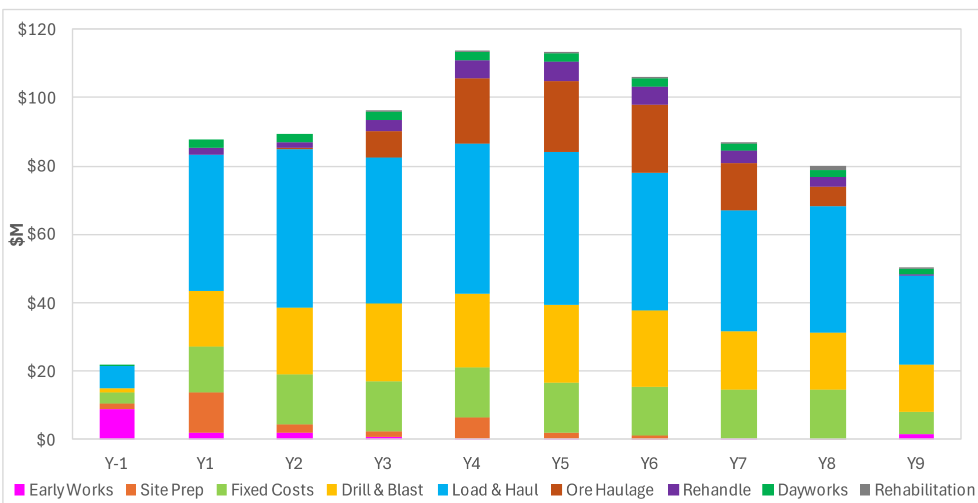


Figure 16.54 Annual Mining Costs

As can be seen in Figure 16.55, the drill, blast, load and haul costs account for 64% (⅔) of the total mining cost. Ore handling (ore haulage and reclaim) and the fixed contract costs account for approx. 15% each. The remaining 6% is attributed to the other mining activities.

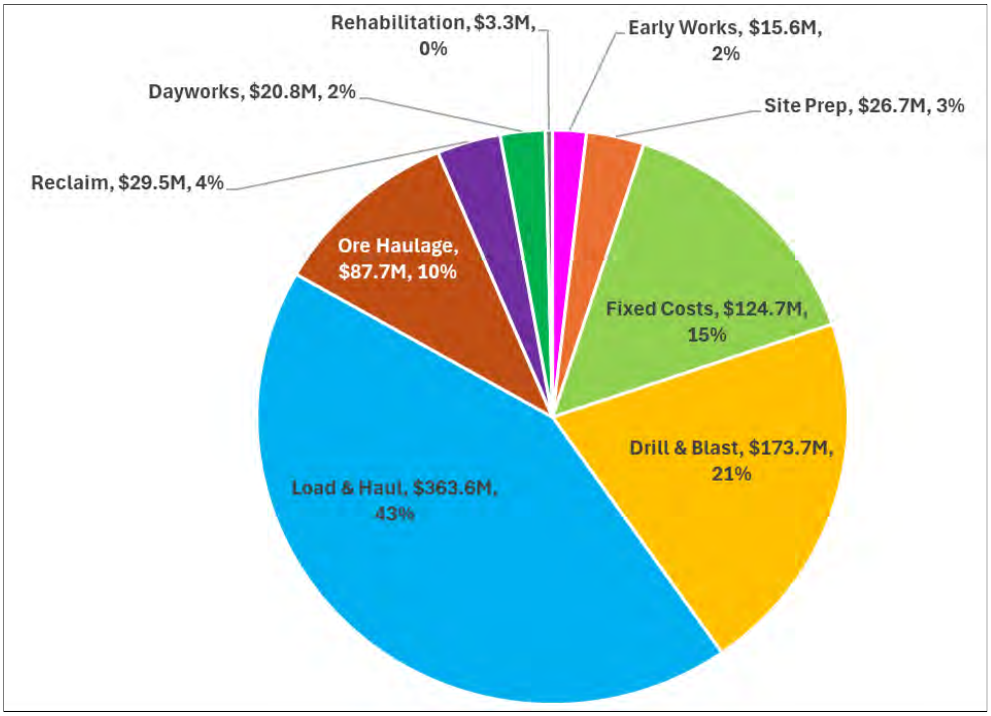


Figure 16.55 Mining Contractor Cost Estimate Breakdown

Activity		Unit	Total	Y-1		Y1				Y2				Y3			
				Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Early Works	Site Estab.	$M	$4.92	$3.46		$0.09				$1.28						$0.04	
	Mobilisation	$M	$9.08	$2.78	$2.45	$1.32	$0.58			$0.14		$0.39		$0.10		$0.68	
	Demobilisation	$M	$1.58						$0.01				$0.01				$0.04
	Subtotal	$M	$15.58	$6.24	$2.45	$1.41	$0.58		$0.01	$1.42		$0.39	$0.01	$0.10		$0.72	$0.04
Fixed Costs	Management Fee	$M	$49.54		$1.22	$1.22	$1.22	$1.22	$1.22	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47
	Fixed & Overheads	$M	$75.21		$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15
	Subtotal	$M	$124.75		$3.36	$3.36	$3.36	$3.36	$3.36	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61
Site Prep	Clear & Grub	$M	$4.71	$0.22	$0.18	$0.37	$0.30	$0.34	$0.59	$0.29	$0.15	$0.09	$0.09			$0.08	$0.24
	Topsoil Stripping	$M	$5.54	$0.26	$0.21	$0.44	$0.36	$0.40	$0.70	$0.34	$0.17	$0.11	$0.11			$0.09	$0.28
	Mine Roads	$M	$10.24	$0.89		$0.47	$0.47		$2.82	$1.37							$0.36
	Ore Haul Roads[1]	$M	$6.17			$1.23	$1.56	$1.63								$0.27	$0.26
	Subtotal	$M	$26.67	$1.37	$0.39	$2.51	$2.69	$2.37	$4.11	$1.99	$0.32	$0.19	$0.19			$0.45	$1.14
Drill & Blast	Ore	$M	$32.10		$0.23	$0.59	$0.84	$0.49	$1.02	$0.94	$0.89	$0.94	$1.14	$1.03	$1.05	$1.04	$0.92
	Waste	$M	$141.57		$0.80	$1.76	$3.10	$3.86	$4.50	$3.51	$3.17	$4.29	$4.82	$4.13	$5.11	$4.65	$5.09
	Subtotal	$M	$173.68		$1.02	$2.35	$3.94	$4.35	$5.51	$4.45	$4.06	$5.23	$5.96	$5.15	$6.16	$5.69	$6.01
Load & Haul	Ore	$M	$75.51		$1.42	$2.12	$2.31	$1.58	$2.72	$2.56	$2.47	$2.27	$2.28	$2.09	$2.19	$2.06	$1.88
	Waste	$M	$288.06		$5.03	$5.59	$8.15	$8.88	$8.82	$9.09	$8.90	$8.97	$9.83	$8.06	$9.06	$8.23	$9.13
	Subtotal	$M	$363.57		$6.45	$7.70	$10.46	$10.46	$11.54	$11.64	$11.37	$11.24	$12.11	$10.15	$11.25	$10.30	$11.00
Rehabilitation	Subtotal	$M	$3.27													$0.03	$0.03
Dayworks	Subtotal	$M	$20.76	$0.04	$0.34	$0.48	$0.61	$0.62	$0.74	$0.65	$0.58	$0.61	$0.66	$0.57	$0.63	$0.60	$0.65
Satellite Pit Ore Haulage	Kilosegui	$M	$67.68											$0.11	$0.26	$0.62	$2.37
	Kekeda	$M	$5.01														
	Enioda	$M	$6.88														
	Han	$M	$8.12										$0.37	$0.61	$0.65	$1.52	$1.62
	Subtotal	$M	$87.69										$0.37	$0.72	$0.91	$2.14	$3.98

Activity		Unit	Total	Y-1		Y1				Y2				Y3			
				Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Rehandle	ROM Pad	$M	$6.93			$0.30	$0.32	$0.26	$0.30	$0.30	$0.30	$0.24	$0.24	$0.23	$0.23	$0.17	$0.14
	Stockpile	$M	$22.57			$0.02	$0.08	$0.60	$0.07	$0.00	$0.06	$0.27	$0.15	$0.26	$0.28	$0.76	$1.20
	Subtotal	$M	$29.50			$0.31	$0.41	$0.86	$0.37	$0.30	$0.36	$0.51	$0.39	$0.49	$0.50	$0.93	$1.34
Grand Total		$M	$845.5	$7.7	$14.0	$18.1	$22.1	$22.0	$25.7	$24.1	$20.3	$21.8	$23.3	$20.8	$23.1	$24.5	$27.8
		$/t mined	$3.74		$3.19	$3.60	$3.20	$3.15	$3.63	$3.41	$2.91	$3.26	$3.30	$2.95	$3.34	$3.77	$3.94
		$/t ore	$22.12		$17.52	$15.34	$17.21	$25.90	$18.41	$20.23	$17.13	$20.01	$20.32	$19.56	$21.27	$23.41	$26.67

Table 16.35 Total Mining Contract Costs to Year 3 by Quarterr

Activity		Unit	Y4				Y5				Y6	Y7	Y8	Y9
			Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4				
Early Works	Site Estab.	$M					$0.04							
	Mobilisation	$M	$0.47				$0.02				$0.06	$0.07	$0.02	
	Demobilisation	$M				$0.00				$0.08	$0.01	$0.07	$0.02	$1.35
	Subtotal	$M	$0.47			$0.00	$0.06			$0.08	$0.07	$0.14	$0.04	$1.35
Fixed Costs	Management Fee	$M	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47	$1.47	$5.86	$5.86	$5.86	$2.43
	Fixed & Overheads	$M	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$2.15	$8.60	$8.60	$8.60	$4.30
	Subtotal	$M	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61	$3.61	$14.46	$14.46	$14.46	$6.73
Site Prep	Clear & Grub	$M	$0.16	$0.12	$0.27	$0.33	$0.20		$0.09	$0.15	$0.45			
	Topsoil Stripping	$M	$0.19	$0.15	$0.31	$0.39	$0.24		$0.11	$0.18	$0.53			
	Mine Roads	$M	$0.36	$1.48	$0.74	$0.39				$0.89				
	Ore Haul Roads[1]	$M			$0.82	$0.40								
	Subtotal	$M	$0.71	$1.75	$2.14	$1.51	$0.44		$0.19	$1.22	$0.97			
Drill & Blast	Ore	$M	$1.01	$0.97	$0.86	$0.98	$1.02	$1.02	$1.06	$1.16	$3.83	$3.83	$3.25	$2.00
	Waste	$M	$5.13	$4.16	$3.63	$4.73	$5.24	$4.46	$4.30	$4.64	$18.30	$13.05	$13.45	$11.72

Activity		Unit	Y4				Y5				Y6	Y7	Y8	Y9
			Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4				
	Subtotal	$M	$6.13	$5.12	$4.49	$5.71	$6.25	$5.48	$5.36	$5.80	$22.13	$16.88	$16.70	$13.72
Load & Haul	Ore	$M	$2.08	$2.13	$2.08	$2.17	$2.30	$2.24	$2.49	$2.56	$7.95	$8.85	$8.23	$4.48
	Waste	$M	$9.14	$8.74	$8.57	$9.06	$9.09	$8.92	$8.53	$8.76	$32.46	$26.74	$28.68	$21.62
	Subtotal	$M	$11.22	$10.87	$10.66	$11.23	$11.39	$11.16	$11.02	$11.32	$40.41	$35.60	$36.91	$26.10
Rehabilitation	Subtotal	$M	$0.06	$0.05	$0.14	$0.16	$0.08	$0.06	$0.09	$0.12	$0.48	$0.44	$0.89	$0.64
Dayworks	Subtotal	$M	$0.65	$0.64	$0.63	$0.67	$0.65	$0.61	$0.61	$0.66	$2.35	$2.02	$2.07	$1.42
Satellite Pit Ore Haulage	Kilosegui	$M	$2.65	$1.63	$4.07	$4.05	$3.78	$4.07	$4.32	$4.20	$17.49	$12.13	$5.95	
	Kekeda	$M							$0.23	$0.68	$2.41	$1.69		
	Enioda	$M		$0.51	$1.43	$1.38	$1.82	$1.21	$0.53					
	Han	$M	$1.66	$1.70										
	Subtotal	$M	$4.31	$3.85	$5.49	$5.43	$5.60	$5.27	$5.07	$4.88	$19.90	$13.82	$5.95	
Rehandle	ROM Pad	$M	$0.14	$0.13	$0.14	$0.14	$0.14	$0.14	$0.15	$0.14	$0.59	$0.80	$0.89	$0.50
	Stockpile	$M	$1.20	$1.16	$1.24	$1.20	$1.24	$1.15	$1.21	$1.17	$4.62	$2.65	$1.96	$0.02
	Subtotal	$M	$1.33	$1.29	$1.38	$1.34	$1.38	$1.29	$1.35	$1.31	$5.21	$3.45	$2.86	$0.52
Grand Total		$M	$28.5	$27.2	$28.5	$29.7	$29.5	$27.5	$27.3	$29.0	$106.0	$86.8	$79.9	$50.5
		$/t mined	$4.04	$3.94	$4.09	$4.20	$4.18	$3.98	$3.91	$4.11	$4.13	$3.95	$3.63	$3.49
		$/t ore	$25.12	$25.31	$25.22	$26.72	$24.79	$24.30	$21.29	$23.32	$23.89	$19.63	$21.54	$25.12

Table 16.36 Total Mining Contract Costs to Year 4 - Year 9 by Quarter/Year

TABLE OF CONTENTS

17. RECOVERY METHODS 1
17.1 Introduction 1
17.2 Summary Flowsheet 1
17.3 Process Description 4
17.3.1 Crushing 4
17.3.2 Ore Storage and Reclaim 4
17.3.3 Grinding and Classification 5
17.3.4 Gravity Gold Recovery 6
17.3.5 Leaching and Adsorption 7
17.3.6 Elution and Gold Recovery 8
17.3.7 Tailings and Cyanide Destruction 10
17.3.8 Reagents 11
17.3.9 Air Services 16
17.3.10 Water Services 17
17.3.11 Electrical 19
17.4 Process Design 20
17.4.1 Process Design Criteria 20
17.4.2 Reagent and Consumable Requirements 22
17.5 Plant Design 23
17.5.1 Crushing 23
17.5.2 Ore Storage and Reclaim 23
17.5.3 Grinding and Classification 23
17.5.4 Gravity Gold Recovery 25
17.5.5 Leaching and Adsorption 26
17.5.6 Elution and Gold Recovery 29
17.5.7 Carbon Regeneration 30
17.5.8 Tailings and Cyanide Destruction 31
17.6 Plant General 31
17.6.1 Metallurgical Accounting 31
17.6.2 Process Plant Control System 32
17.7 Equipment Selection 33
17.7.1 Crushers 33
17.7.2 Grinding Mills 33
17.7.3 Gravity Circuit 33
17.7.4 Thickener 34

LIST OF FIGURES

Figure 17.1 Process Flowsheet....3
Figure 17.2 Laboratory Scale Gravity Recovery by Master Composite....26
Figure 17.3 Gold Extraction Curves for Oxide/Transition and Fresh Ores....28

LIST OF TABLES

Table 17.1 Plant Power Requirements....19
Table 17.2 Ore Sources....20
Table 17.3 Metallurgical Tested Comminution Properties....21
Table 17.4 Circuit Design Criteria....22
Table 17.5 Processing Plant Annual Consumption Rates for Key Reagents and Consumables....22
Table 17.6 Comminution Design Parameters....24
Table 17.7 SAG Mill Operating Conditions....24
Table 17.8 Ball Mill Operating Conditions....25
Table 17.9 Leach Circuit Design Parameters....28
Table 17.10 Adsorption (CIL) Design Parameters....29
Table 17.11 Elution Circuit Design Parameters....30

17. RECOVERY METHODS

17.1 Introduction

The Doropo Gold Project is a 4.0 Mt/a (fresh feed) gold mining project in the Bouna region of Cote d'Ivoire. The processing plant will utilise industry standard comminution, leaching, adsorption, and gold recovery technologies to produce a saleable gold doré. The Doropo gold plant will process ore from nine different pits with varying quantities of fresh, transitional and oxide ores. The oxide and transitional ores are associated with saprolite and saprock, while the fresh ores are associated with granodiorite and tonalitic granite gneiss host rocks.

In design and feed material assessment, two different design points are used to account for the varying feed material types. A plant feed of 100% fresh ore is designed at a mill feed rate of 500 t/h, annualised at 4.0 Mt/a. Owing to softer properties of the ore, a plant feed of 100% oxide/transitional ore is designed at a mill feed rate 675 t/h, annualised at 5.4 Mt/a. The equipment and unit processes selected in the Doropo plant flowsheet are capable of handling each individual material type and all blend variants in between.

17.2 Summary Flowsheet

The plant, irrespectively of feed blend, will operate on a continuous 24 hour-a-day basis, via two 12 hour processing shifts. The key ore and project design criteria that the plant is designed upon include:

- 4.0 Mt/a fresh feed or 5.4 Mt/a on oxide/transitional feed;
- Crushing circuit utilisation of 80%;
- Wet plant utilisation of 91.3%, which includes plant auxiliaries and services;
- Instrumentation and process automation for efficient control and indication to supplement personnel operation;
- Standby/duty installations for process critical pumps.

The process design criteria are used to develop an oxide and fresh feed mass balance. The mass balances form the basis of the process plant equipment sizing. The plant design is based on the PFS and DFS metallurgical test work, circuit modelling, and selected vendor equipment, all sized from the mass balance.

The processing plant flowsheet is shown in Figure 17.1 and summarised as:

- Primary scalping of ROM via a vibrating grizzly feeder to bypass -100 mm material from the primary crusher;
- Primary crushing of ROM +100 mm to produce a crushed product with top size of 300 mm;
- Storage of crushing circuit product on the Coarse Ore Stockpile (COS) with live capacity of 12,000 t;

- Primary grinding via SAG milling and pebble crushing;
- Secondary grinding in closed circuit with hydrocyclone classification to produce an 80% passing 75 µm grind size on fresh ores and 106 µm on oxide/transitional ores;
- Gravity gold concentration and intensive leaching of gravity concentrates;
- Product trash removal and thickening prior to leaching;
- Gold leaching in two dedicated leach tanks preceding six Carbon-In-Leach (CIL) adsorption tanks with 8 g/L activated carbon;
- Gold recovery via 12 t batch elution processes using the pressure Zadra methodology with acid washing;
- Dedicated gravity and CIL electrowinning circuits;
- Carbon regeneration via a diesel fired horizontal kiln; and
- Leached tailings detoxification via SO_2/Air and disposal to the Tailings Storage Facility (TSF).

Various utility and plant infrastructure such as water supply, reagent mixing, storage and distribution, distribution of air and electrical energy supply/distribution, roads, communications, and site buildings support the Project.

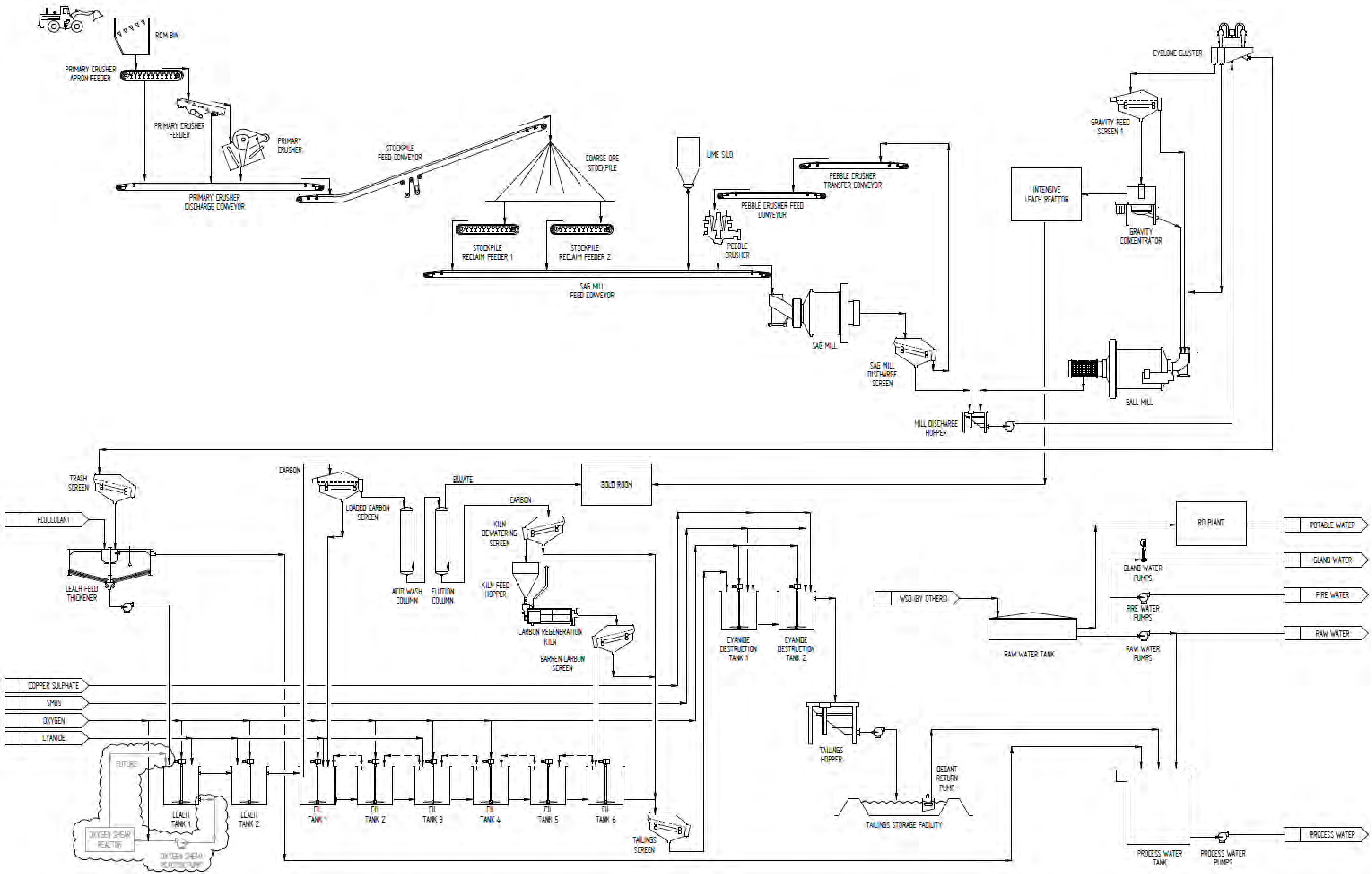


Figure 17.1 *Process Flowsheet*

17.3 Process Description

The process plant description should be read in conjunction with process flow diagrams.

17.3.1 Crushing

The open pit ore is either direct-tipped or fed by FEL into the 250 t capacity ROM bin. The ROM bin will have dual truck tipping capability with the ability to accept trucked feed from both sides. The ROM bin will be fitted with traffic lights to signal to the mining fleet when material can be safely dumped. It will also feature an 800 mm by 800 mm static grizzly for oversize rejection. Ore from the ROM bin will be reclaimed via the primary crusher apron feeder.

The primary crusher apron feeder will transport the ROM ore to the primary crusher vibrating grizzly. The primary crusher grizzly feeder will be a vibrating grizzly with 100 mm spaced fingers to divert undersize ROM material to bypass the primary crusher. The scalped grizzly oversize will report to the primary crusher for size reduction. The primary crusher is a jaw crusher designed to produce a product size distribution P_{80} of 118 mm for oxide/transitional feeds and 143 mm for fresh ores. The primary crushed product and grizzly undersize will report to and recombine on the primary crusher discharge conveyor.

A self-cleaning primary tramp metal magnet will be mounted at the primary discharge conveyor discharge prior to the transition of feed onto the stockpile feed conveyor. The crushed and scalped ore is then conveyed to the Coarse Ore Stockpile (COS) via the stockpile feed conveyor.

Dust generated via crushing and feed transition will be removed by dust collection and suppression systems. Suppression will be achieved through spray water addition onto the primary crusher discharge conveyor.

A dedicated drive-in sump and maintenance crane account for spillage and facilitate maintenance around the crusher.

The crushing circuit production rate will be monitored by the primary crusher weightometer installed along the primary crusher discharge conveyor.

17.3.2 Ore Storage and Reclaim

The stockpiled ore will be reclaimed by two apron feeders situated below the COS. The reclaimed ore will be fed onto the SAG mill feed conveyor. The SAG mill feed conveyor will be fitted with the SAG mill feed weightometer which will provide the instantaneous rate and totalised tonnages. The weightometer provided instantaneous rate, along with SAG mill operational parameters, will control the reclaim feeder speed and plant throughput.

Quicklime will be added to the SAG mill feed conveyor in addition to the reclaimed ore. The quicklime, used for pH control for the leaching circuit, and will be dosed onto the conveyor from a dedicated 250 t capacity storage silo situated over the SAG mill feed conveyor.

SAG mill grinding media will be periodically added to the SAG mill ball charging hopper from the dedicated SAG media storage bunker. The SAG mill ball charger will automatically control the ball addition to the SAG mill feed conveyor and SAG mill.

Dust generated during reclaim and feed transitions will be managed by the reclaim tunnel dust collector and suppressed via spray water addition onto the SAG mill feed conveyor.

The reclaim tunnel sump pumps will capture clean-up and spillage from within the reclaim tunnel and from the pebble crusher and lime silo areas.

17.3.3 Grinding and Classification

The SAG mill feed conveyor will transport reclaimed ore to the SAG mill for primary grinding. The reclaimed ore will be combined with crushed pebbles and quicklime, with process water added in the SAG mill feed chute. The 8.7 m (IS) diameter by 5.0 m EGL SAG mill will have a 7,000 kW variable speed drive. The primary ground ore will be discharged from the mill and screened by the 3.0 m wide by 6.0 m long SAG mill discharge screen, equipped with 12 mm aperture panels.

The SAG mill discharge screen will separate the oversize SAG produced pebbles from the primary ground slurry. The pebbles will be conveyed via the pebble crusher transfer conveyor and the pebble crusher feed conveyor to the pebble crusher. The pebbles will report to the pebble crusher surge bin and, together with the pebble crusher vibrating feeder, maintain choke feed to the pebble crusher. The pebble crusher will be a 220 kW cone crusher with an operating closed side setting of 15 mm and is expected to treat up to 100 t/h on oxide/transitional or fresh ore. The crushed pebbles will report back to the SAG mill feed conveyor to be reground in the SAG mill.

The SAG mill discharge screen undersize will gravitate to the mill discharge hopper. The mill discharge hopper will be common to both the SAG and ball mills and capture spillage from the reclaim and grinding areas in addition to the Intensive Leach Reactor (ILR) residue. The combined mill discharge hopper contents will be pumped via one of the two cyclone feed pumps (arranged in duty/standby configuration) to a cluster of hydrocyclones for classification.

The target grind size P_{80} will be 75 μm for fresh ore or 106 μm for oxide/transitional ores, which will be achieved by changing the vortex finders and spigots. The cyclone overflow will report to the trash screen and onto the leach feed thickening. The cyclone underflow will be split between the gravity circuit and the ball mill for secondary grinding.

The 6.8 m (IS) diameter by 10.2 m EGL overflow discharge ball mill will have 9,000 kW variable speed drive. The ball mill is anticipated to operate with a 30% ball charge with 60 mm make-up grinding media. The speed will be varied to cater for the differing power requirements of the ore being processed. The ground ore will discharge from the ball mill onto the ball mill trommel, with 15 mm aperture panels. The trommel oversize will report to the ball mill scats bunker, and the trommel undersize will gravitate into the mill discharge hopper. The ball mill grinding media will be added into the ball mill feed chute via a purpose-built kibble, lifted by the ball loading hoist. A dedicated ball mill media bunker will be situated within the grinding area for ball storage and kibble loading.

The mill area will be serviced by three sump pumps and has a dedicated drive-in sump for cleanup and spillage capture. A davit crane will be installed at the cyclone cluster to facilitate routine maintenance.

The SAG and Ball mills will have removable trolley feed spouts which, will be removed during relines to facilitate liner access and changeout. The SAG mill liner handler will facilitate reline component change out for the SAG mill.

17.3.4 Gravity Gold Recovery

Gravity recoverable gold will be captured by a dedicated gravity circuit. The gravity circuit will be fed a portion of the cyclone underflow diverted from the cyclone underflow launder. The cyclone underflow will gravitate to the 2.44 m wide by 4.88 m long gravity scalping screen where the plus 2 mm oversize material will be returned to the ball mill for further size reduction, and the undersize material will report to the gravity concentrator for coarse gold removal.

The centrifugal gravity concentrator will cycle between concentration and gold capture, to purging of captured coarse gold. The captured coarse gold will be periodically purged to the Intensive Leach Reactor (ILR) feed hopper. The gravity concentrator tailings will discharge into the ball mill feed chute. When the concentrator is purging the feed will be bypassed and feed directed to the ball mill feed chute. The feed to the concentrator will be reinstated once the concentrator purge is complete. The plant raw water distribution will supply the gravity concentrator with fluidisation water.

The collected gravity concentrates will report to the ILR concentrate feed hopper. The ILR will process the gravity concentrates from the previous 24 hours of production. The concentrate will transfer into the leach reactor where sodium cyanide, caustic and raw water will be added and the intensive cyanidation process will begin. The leach contents will recirculate within the reactor until the captured coarse gold is leached. The gold laden liquor will then be clarified and pumped to the gravity electrowinning tank for electrowinning. The leach residue will be pumped from the reactor into the mill discharge hopper completing the batch process. The ILR will then be able to receive concentrate and start leaching another batch.

The pregnant gravity gold solution within the gravity electrowinning tank will be electrowon in a dedicated gravity electrowinning cell, which will contain nine 1,000 mm x 1,000 mm stainless steel cathodes and ten anodes. The gold laden solution will be pumped from the gravity electrowinning tank to the gravity electrowinning cell via the variable speed gravity electrowinning pump. The solution will be recirculated until the gold concentrate is reduced to less than 10 ppm gold, which is anticipated to take 20 hours. The resulting gold sludge will be recovered by in-situ pressure cleaning of the cathodes.

The gravity area will be serviced by a sump pump for spillage and clean-up capture. The spillage will be pumped to the mill discharge hopper and incorporated back into the process.

The gravity concentration and intensive leach reactor will be enclosed within a secure compound which will have access control to reduce the risk of gold theft. A gold trap will be located within the intensive leach area to capture any free gold before it reports to the mill area bund.

17.3.5 Leaching and Adsorption

The cyclone overflow will be screened and thickened in the pre-leach thickener before reporting to the leaching and adsorption circuit. The cyclone overflow will gravitate to the 3.05 m wide by 7.32 m long trash screen with 0.8 mm aperture panels, where oversize grit and trash will be captured on the screen deck and report to the trash bunker for periodic removal.

The trash screen undersize will gravitate to the 35 m diameter hi-rate pre-leach thickener, where it will be combined with flocculant. The pre-leach thickener will thicken the slurry to 50% solids for fresh ores (48% solids for oxide/transitional ores). The oxide leach feed density is lower than fresh because some of the oxide materials exhibit high viscosity characteristics. The excess water recovered by thickening will report to the process water tank for reuse in the plant.

The pre-leach thickener underflow pumps will transport the thickened slurry to the leach and adsorption circuit via the leach feed autosampler. The leach-feed autosampler will be a two-stage sampling system to collect slurry composite samples for process optimisation and metallurgical accounting. The slurry not collected within the composite sample reports to the leach feed distribution box, which will be able to direct feed into either leach tank.

Sodium cyanide will be added into the leach feed distribution box and leaching will commence. Additional cyanide, if required, may also be added into CIL tank 1 and CIL tank 3 as staged cyanide addition. The dissolved oxygen level in the slurry will be maintained via down agitator shaft additions to each of the leach tanks and CIL tanks 1 to 4, with oxygen generated from onsite Pressure Swing Absorption (PSA) plants.

The leach circuit alkalinity will be maintained by the addition of quicklime onto the SAG mill feed conveyor. The leach circuit cyanide concentration, pH, and dissolved oxygen levels will be continuously monitored by instrumentation for process control and operation.

The leach and adsorption circuit will consist of two leach and six CIL adsorption tanks operated in series, equipped with pumped screens and inter-tank launders. The circuit is designed to provide a residence time of 27 hours for oxide/transitional material and 38 hours for fresh ore.

The leached gold in solution will be recovered by adsorption using activated carbon, which is suspended within the agitated CIL tanks. Carbon will be advanced in a counter-current direction to the slurry.

As leached gold sequentially loads onto the activated carbon, the carbon in slurry will be advanced within the CIL tanks by upstream pumping. The carbon transfer pumps will have recessed impellers to minimise the propagation of carbon fines. The carbon will be retained within each CIL tank by a 15 m^2 pumped inter-tank screen, which retains the carbon in tank while allowing slurry to launder into the next tank.

The slurry and carbon counter-current movement will produce a gold loaded carbon in CIL tank 1. The loaded carbon will be pumped from CIL tank 1 in batches for acid washing, elution and regeneration. The leached slurry exiting CIL tank 6 reports to the cyanide destruction and tails pumping.

The slurry will flow between the tanks via the interconnected launders. Launder gates will be installed between the tanks to facilitate bypassing and taking a tank offline if required.

The leaching and CIL tanks will be serviced by an overhead gantry crane installed above the leach and CIL tanks. The crane will be used to service and rotate the in-service intertank screen with the rotable intertank screen spare, housed within the dedicated CIL screen rebuild bay.

The leaching and CIL circuit area will be bunded and serviced by three (3) area sump pumps. The spillage or cleanup in the leach, CIL and tails hopper areas will be directed via the sloped floor into the sump pump catchments for return to the circuit.

17.3.6 Elution and Gold Recovery

The gold-loaded carbon will be removed in 12-tonne batches for acid washing and elution. The loaded carbon will be pumped from CIL tank 1 (or CIL tank 2 if the preceding tank is offline) to the 1.8 m wide by 3.6 m long loaded carbon screen. The loaded carbon screen will separate and wash the carbon from the slurry. The loaded carbon will discharge from the screen into the acid wash column. The slurry will report to the screen undersize and gravitate back into CIL tank 1. The loaded carbon transfer will continue until the acid wash column is filled, which is expected to take 6 hours.

The carbon will then be acid washed with 3% hydrochloric acid to remove inorganic foulants. The acid will be mixed with raw water and injected into the base of the acid wash column until the column is filled. The dilute acid will then soak in the carbon bed, reacting to remove the foulants. After the acid soak time expires, raw water will be injected into the base of the column and rinsing of the carbon bed will commence. The spent acid, reacted foulants, and rinse water during the acid wash rinse will report to the tailings hopper for disposal into the TSF. After the four bed volumes of rinse water have passed through the column, the column will be drained, and the carbon will be pressure educted into the elution column.

The elution process will recover the gold using the pressure Zadra methodology. The Zadra method will involve circulating a heated solution (135°C) of raw water, with 0.2% sodium cyanide and 2% caustic through the gold loaded carbon bed at elevated pressures and temperatures to desorb the gold and produce a gold laden eluate. The eluate will circulate from the eluate tank through the heat exchangers into the base of the elution column. It will exit the column through the elution filters and then pass through the secondary heat exchange and trim cooler before entering the CIL electrowinning cells. The elution process involves pre-heating, eluate production and cooling steps. To maintain desorption efficiency, every third strip will require fresh solution make-up.

The heat for the Zadra process will be provided by the elution heater. The elution heater will burn diesel to heat and recirculate hot oil, which in turn will heat the eluate in counter-current flow within the plate and frame primary heat exchanger. The hot eluate exiting the top of the column will pass through the recovery heat exchanger, pre-heating the eluate returning to the elution column. The eluate will be cooled to 90°C by the trim water cooler prior to electrowinning.

The heat exchanger plates will be constructed of titanium as this material provides superior corrosion resistance compared to stainless steel, mitigating risks associated with using raw water in the elution process.

The CIL electrowinning system will consist of two electrowinning cells operating in parallel, each equipped with a rectifier. Electrical current will pass from the anodes to the cathodes, depositing the gold as a gold sludge. The CIL elution and electrowinning cycle will take 16 hours and will operate 6 strips per week. The barren CIL electrowinning solution will be pumped via the eluate pump to the leach feed distribution box. The electrowinning cells will be fitted with ducting and extraction fans to remove fumes and off-gas, providing ventilation in the goldroom.

Some of the ore has shown varying levels of silver content. Therefore, the electrowinning cells have been conservatively sized to handle the design 2,000 g/t gold and 2,400 g/t silver loadings.

To facilitate the periodic cell cleaning and gold capture, the cell anodes will be removed from the cell using the anode lifting frame and goldroom gantry crane. The removed anodes will be stored on the anode rack in the goldroom, where there will be cleaned, stored, and returned to the electrowinning cells once the gold recovery is complete.

The gold sludge captured on the cathode mesh will be pressure washed in-situ, reporting to the sludge hopper below. The sludge hopper will feed a plate frame filter press, which will dewater the recovered gold sludge. The excess filtrate will report to the goldroom sump pump to be returned to the process. The filtered gold sludge will then be deposited into the oven trays from the filter when the filter press is depressurised and opened.

The filtered gold sludge will then be dried in the goldroom ovens. Once dried, the sludge will then be mixed with smelting fluxes and smelted in a diesel fired smelting furnace. The resulting doré will be stored in the goldroom safe and vault until transported from site and sold. The smelting furnace will be fitted with an exhaust fan and fume hood for exhaust extraction.

17.3.7 Tailings and Cyanide Destruction

The cyanide destruction and tailings circuit will receive leached tailings from exiting CIL tank 6. The slurry exiting CIL tank 6 will first pass the tailings autosampler, a two-stage sampling system collecting a slurry composite sample for process optimisation and metallurgical accounting. The slurry not collected within the composite sample will gravitate to the 3.05 m wide by 7.32 m long tailings screen. The tailings screen oversize will capture coarse carbon in a bulk bag, while the screen undersize will gravitate to the cyanide destruction feed box.

The cyanide destruction circuit will detoxify the residual WAD cyanide through the addition of oxygen and sulphur dioxide. The destruction method will follow the INCO process (Air/SO_2) to produce a plant tailings with WAD cyanide below 50 ppm. Sodium metabisulfite (SMBS) and copper sulphate will be dosed into the cyanide destruction feed box, commencing the detoxification process. The sodium metabisulphite provides the sulphur dioxide for the reaction, whilst the copper sulphate provides soluble copper needed to catalyse the reaction. Oxygen will be added via agitator downshaft addition and will be provided from the site PSA plants. There will be provision for caustic to be added into the cyanide destruction feed box to maintain detoxification pH.

The two (2) cyanide destruction tanks will each be 750 m^3 agitated slurry tanks operated in series. The cyanide destruction tanks will provide 1.5 hours of residence for WAD cyanide destruction when processing oxide/transition feed and 2.1 hours on fresh feeds. The detoxified slurry will gravitate from cyanide destruction tank 2 into the tailings hopper.

The tailings hopper will combine the detoxified tailings, sump pump spillage, RO reject and acid washing spent acid within the tailings hopper. The tailings will then be pumped to the TSF for disposal by one of two sets (duty/standby arrangement) of two-stage centrifugal slurry pumps. A flowmeter will be installed on the tailings pipeline to indicate the slurry flowrate to the TSF.

The water decanted from the deposited tailings will be captured within the TSF facility and will be returned to the process plant for reuse, via the decant return pumps.

17.3.8 Reagents

The design for onsite reagent storage generally allows for three months of stock on hand at design usages to ensure continuity of supply throughout the wet and dry seasons. The reagents will be typically supplied to site in sea containers packed within palleted bulk bags or intermediate bulk containers (IBCs) for liquids. The exceptions will be quicklime, diesel and cyanide which will be delivered in bulk (quicklime, diesel) and specialty isotanks for sparging (cyanide). The sea containers, when emptied of bulk bagged or IBC materials will be returned to Abidjan port with the next delivery.

The reagents to be used at the Doropo processing plant include:

- Sodium cyanide for the dissolution of gold in leaching and elution;
- Quicklime for control of pH in leaching and adsorption;
- Carbon for the adsorption of leached precious metals in solution;
- Caustic for pH control in elution, electrowinning and cyanide destruction;
- Hydrochloric acid for acid washing of loaded carbon;
- Flocculant for solid-liquid separation in the ILR and pre-leach thickener;
- Coagulant for solid-liquid separation in the pre-leach thickener (temporary only);
- Sodium metabisulphite (SMBS) for cyanide destruction;
- Copper sulphate for cyanide destruction;
- Antiscalant to prevent scale formation in water services and elution;
- Grinding media for particle size reduction in the SAG and ball mills;
- Sulphamic acid for heat exchanger descaling;
- Diesel for fuel source for the elution heater, regeneration kiln and smelting furnace; and
- Borax, silica flour, soda ash and nitre for smelting fluxes.

There will be two sheds constructed to store the process reagents. The sheds will separate materials based on their material properties and storage compatibility. Grinding media and cyanide isotanks, whether full or emptied will be stored adjacent to, but outside of, the reagents sheds. This allows delivery truck access and heavy vehicle interaction to be confined to the west of the plant, whilst providing loading areas and wide radius turning capability. The access to the lime silo is separate from the reagent storage and make down facilities. The lime silo will be accessed via a dedicated road and turning circle to accommodate bulk tanker deliveries.

Quicklime

Quicklime will be trucked to site in bulk tankers and transferred into the silo. The lime quality will contain a minimum contain 90% available CaO (quicklime). The lime silo will reside over the SAG mill feed conveyor adjacent to the COS stockpile.

The quicklime addition will be metered by the lime silo rotary valve and fall directly onto the SAG mill feed conveyor. The quicklime will be dosed to maintain the pH of the leaching and adsorption circuit.

Cyanide

Sodium cyanide will be delivered to site in isotanks, with each isotank containing 20 tonnes of solid sodium cyanide pellets. Doropo will engage and use the Orica sparge system for cyanide transport and make-down onsite. The isotanks and sparge facility will be supplied by Orica, and a monthly rental fee paid.

The isotanks will be re-stocked, maintained and return to site from the Orica facility in Tarkwa Ghana. A minimum of 9 isotanks will be stored onsite, equating to one month of supply. The isotanks are intended to be delivered in convoys, with the empty isotanks returned at the next delivery.

The cyanide sparge system will produce a 30% sodium cyanide solution. The isotank will be coupled to the cyanide mixing tank inlet and outlet. A raw water and caustic solution will be batched from the cyanide mixing tank and pumped into the isotank via the cyanide mixing pump. The raw water dissolves the sodium cyanide pellets, and the caustic stabilises the dissolution pH, the process recirculates in this manner until all 20 tonnes of solids pellet is dissolved and the 30% solution produced. Thereafter the batched cyanide solution will be transferred into the cyanide storage tank using the cyanide mixing pump and diverted via valving. A transfer between mixing and storage vessels may only occur when there is sufficient free volume available within the cyanide storage tank.

From the cyanide storage tank, the cyanide solution will be circulated in a ring main, using separate control valves and flowmeters to regulate the addition of cyanide to the following locations:

1. Leach feed distribution box and CIL Tanks 1 and 3;
2. Elution column;
3. Intensive leach reactor.

The cyanide mixing area will be serviced by the cyanide area sump pump which may divert spillage to either the leach feed distribution box, tailings hopper or back into the cyanide storage tank. The cyanide mixing vessel will also have the ability to receive and mix cyanide from 1-tonne bulk bags, if required.

Hydrochloric Acid

Hydrochloric acid of 32% (w/w) strength will be delivered to site in sea containers contained within IBCs. The IBCs will be periodically unloaded into the hydrochloric acid storage tank, using the hydrochloric acid unloading pump. The hydrochloric acid tank will hold 5 m^3 equating to 3 days of process capacity.

Hydrochloric acid will be dosed to the acid wash column via the HCl dosing pump. The hydrochloric acid will be diluted and mixed in-stream with raw water to a concentration of 3% (w/w) for acid washing of the loaded carbon.

The hydrochloric acid area will be serviced by the acid area sump pump which diverts spillage to the tailings hopper for disposal.

Caustic (Sodium Hydroxide)

Caustic will be delivered to site as a 98% NaOH solid pearls in sea containers, packed in palleted bulk bags. The bulk bags will be unloaded from the container and stored in the relevant reagent shed. A forklift will transport the caustic bulk bags from the reagent store to the caustic mixing and storage facility.

Caustic onsite will be mixed in the caustic mixing tank, to produce a 20% (w/v) caustic solution. Raw water will partially fill the caustic mixing tank and agitator will start prior to the solid product being added using the caustic bag breaker and caustic hoist. The remaining raw water volume will then be added and solid dissolved and mixed. Thereafter, the batched caustic solution will be transferred into the caustic storage tank using the caustic transfer pump.

Caustic will circulate from the caustic storage tank into the caustic ring main via the caustic circulating pumps. The caustic dosing to the process will be controlled by separate control valves and flowmeters. The addition of caustic will be at the following locations:

1. Elution column;
2. Intensive leach reactor;
3. Cyanide destruction;
4. Cyanide dissolution.

The caustic mixing and storage area will be serviced by the caustic area sump pump which diverts spillage to the tailings hopper for disposal.

Activated Carbon

Activated carbon will be delivered to site in sea containers packed within palleted bulk bags. The bulk bags will be unloaded from the container and stored in the relevant reagent shed until needed.

Fresh carbon bags will be added to the quench hopper and transferred to the barren carbon screen for fines removal. The retained oversize carbon then reports to CIL tank 6 for gold adsorption. The barren carbon undersize containing the carbon fines reports to the discharge launder of CIL tank 6 and onto the cyanide destruction and tailings circuit for disposal to the TSF.

Flocculant

Flocculant will be delivered as a dry powder contained in sea containers packed within palleted bulk bags. The bulk bags will be unloaded from the container and stored in the relevant storage shed until made-down.

Onsite, Flocculant will be mixed in batches to produce a 0.25% product. The dry flocculant powder will be hoisted and emptied into the flocculant powder hopper. The powder hopper screw feeder and transfer blower transfer the flocculant powder to the wetting head where the powder will initially combine with raw water. A measured quantity of raw water will be added to the flocculant mixing tank and the agitator started. The flocculant/water mix gravitates into the mixing tank and when the powder quantity is completed raw water continues to fill the mixing tank via the wetting head. The mixing tank agitator will then run for a fixed time interval to continue mixing the flocculant. When the mix time has transpired the flocculant batch will be transferred into the storage tank via the flocculant transfer pump.

Flocculant will be dosed from the flocculant storage tank via individual transfer pipelines. The flocculant dosing pumps 1 and 2 will dose flocculants to the pre-leach thickener via a standby/duty pump arrangement. Flocculant dosing pump 3 pumps flocculant to the ILR as a dedicated line.

The flocculant mixing and storage tank area will be serviced by the flocculant area sump pump which diverts spillage and clean up to the tailings hopper.

Coagulant

Coagulant will be supplied to site in sea containers contained within IBCs. The dosing of coagulant is anticipated to be a temporary requirement for the initial months of operation, when processing oxide feed alone.

Coagulant, if required, will be dosed into the trash screen feed box via a skid mounted dosing pump.

Sodium Metabisulphite (SMBS)

SMBS will be delivered to site as a solid powder in sea containers, packed within palleted bulk bags. The bulk bags will be unloaded from the container and stored in the relevant reagent shed until made-down. A forklift will transport the SMBS bulk bags from the reagent store to the SMBS mixing and storage facility.

SMBS will be mixed onsite in the SMBS mixing tank to produce a 20% (w/v) SMBS solution. Raw water will partially fill the SMBS mixing tank and the mixing tank agitator will start. The bulk bags of SMBS powder will then be added to the mix tank via the SMBS hoist and SMBS bag breaker. The remaining raw water volume will be added and the SMSB powder dissolved and mixed. Thereafter, the batched SMBS solution will be transferred into the SMBS storage tank using the SMBS transfer pump.

SMSB will be dosed to the cyanide destruction circuit via the duty/standby SMBS dosing pumps. The SMBS dosing to the destruction process will be controlled by the speed of the dosing pumps.

The SMBS mixing and storage tanks will be vented, and sulphur dioxide fumes exhausted via the SMBS exhaust fan. A fixed sulphur dioxide gas detector will be installed within the SMBS make-down area for continuous gas monitoring.

The SMBS mixing and storage area will be serviced by the SMBS area sump pump the sump diverts spillage to the tailings hopper for disposal.

Copper Sulphate

Copper sulphate will be delivered to site as solid copper sulphate pentahydrate powder in sea containers packed within palleted bulk bags. The bulk bags will be stored in the relevant reagent shed until made-down. A forklift will be used to transport the copper sulphate bulk bags from the reagent store to the copper sulphate mixing and storage facility.

The copper sulphate will be mixed in the copper sulphate mixing tank to produce a 20% (w/v) solution. Raw water will partially fill the copper sulphate mixing tank and the mixing tank agitator will start. The bulk bags of copper sulphate powder are then added to the mixing tank via the copper sulphate hoist and bag breaker. The remaining raw water volume will be added, and the copper sulphate dissolved. Thereafter, the copper sulphate batch of solution will be transferred into the copper sulphate storage tank using the copper sulphate transfer pump.

Copper Sulphate will be dosed into the cyanide destruction circuit via the duty/standby copper sulphate dosing pumps. The copper sulphate addition to cyanide destruction will be controlled by the speed of the dosing pumps.

The copper sulphate mixing, and storage area will be serviced by the copper sulphate area sump pump which diverts spillage to the tailings hopper for disposal.

Antiscalant

Antiscalant will be delivered to site in sea containers contained within IBCs. The IBCs will be unloaded from the sea container and stored in the relevant reagent shed until dosed neat into the process. Antiscalant is dosed into the process water and elution system for scale prevention and control. The antiscalant IBCs will be replaced when empty at either dose location.

Antiscalant will be dosed to the process via a skid-mounted dosing pump attached to the IBC.

Grinding Media

Grinding media will be supplied to site in sea containers contained within bulk bags. The SAG mill top-up grinding media will be 125 mm balls, while the ball mill top-up charge will be 60 mm balls. A hard stand area adjacent to the reagent sheds will serve as storage for the grinding media. A forklift will periodically transport bagged media to each of the respective grinding media bunkers (SAG/Ball), where the media will be emptied into the bunker for use in the process.

SAG media stored in the SAG media bunker will then be front-end loaded into the SAG media charging hopper. The ball mill media in the ball mill media bunker will be loaded into a purpose-built kibble and added into the feed chute of the ball mill.

Goldroom Fluxes

The goldroom fluxes will be supplied to site as dry powder contained within palleted 25 kg bags. The fluxes will be stored in the goldroom and combined with the gold sludge in the flux mixing drum before being smelted. The goldroom fluxes include the use of sodium borate (borax), silica flour, sodium nitrate (nitre), and sodium carbonate (soda ash).

Sulphamic Acid

Sulphamic acid will be supplied to site as dry powder contained within palleted 25 kg bags. A 25 kg bag will be mixed with raw water in the sulphamic acid drum and then used to acid wash the elution heat exchangers. At the end of the heat exchanger acid washing, the spent acid mix will be drained into the HCl area sump pump and pumped to the tailings hopper for disposal.

17.3.9 Air Services

Two screw compressors will provide compressed air to the plant and crusher air receivers ad well as dried and filtered compressed air to the instrument air receiver. The compressors will be run as lead-lag. The high-pressure air from the crushing and plant air receivers will be reticulated throughput the process at 750 kPa. Plant air service points will be provided around the plant.

Pressure Swing Absorption Plants (PSA)

Oxygen will be generated onsite by two 15 t/d Pressure Swing Adsorption (PSA) plants. The generated oxygen will be of 90% purity and dosed to the leaching and cyanide destruction circuits.

The addition of oxygen via agitator downshaft sparging will be to the follow locations:

1. Leach Tank 1;
2. Leach Tank 2;
3. CIL Tank 1;
4. CIL Tank 2;
5. CIL Tank 3;
6. CIL Tank 4;
7. Cyanide Destruction Tank 1;
8. Cyanide Destruction Tank 2.

17.3.10 Water Services

Water services include:

- Raw water supplied by surface water harvesting and dams;
- Process water recycled within the process plant and recovered from the TSF;
- Potable water generated from Reverse Osmosis (RO) onsite; and
- Fire water which draws from a dedicated reserve in the raw water tank.

Raw Water

The raw water for the processing plant will be stored in a dedicated Water Storage Dam (WSD). The WSD is seasonally replenished from the Water Harvest Dam (WHD), which collects rainfall and runoff. The WSD may also receive groundwater collected from mine pit dewatering. The raw water is pumped from the WHD to the WSD via the duty/standby water transfer pumps.

Raw water for the processing plant will be pumped from the WSD via the duty/standby raw water supply pumps, which discharge into the raw water tank. The level of the raw water tank will control the transfer of water from the WSD, and the tank will nominally be kept full during normal operation. The raw water tank will provide a dedicated fire water reserve for the processing plant.

The following pumps will draw from the raw water tank:

- Duty/standby raw water pumps;
- Duty/standby RO feed pumps;
- Electric, diesel and jacking fire water pumps;
- Duty/standby gland water pumps.

The process plant raw water consumption has been estimated at 412 m^3/h when processing fresh ore and 451 m^3/h when processing oxide/transition ores. The raw water pumps (duty and standby) will draw water from the raw water tank to supply the following plant services:

- Crushing and reclaim area dust suppression sprays;
- Gravity concentrator fluidising water;
- ILR batching;
- Gland water addition;
- Flocculant mixing;
- Cyanide mixing;
- SMBS mixing;
- Copper sulphate mixing;
- Caustic mixing;
- Sulphamic acid mixing;
- Acid washing, elution and carbon transfer.

Raw water is also piped from the process plant to the camp raw water tank, where it is used as the fire reserve water for the camp. The camp fire water reserve is drawn from the camp raw water tank via the camp electric, diesel, and camp jacking fire water pumps.

Process Water

The processing plant process water tank receives process water from the pre-leach thickener overflow, tailings storage facility as decant water, event pond captured water and, if required, raw water from the WSD. The process water sources are collected within the 2,000 m^3 capacity process water tank. Process water pumps (duty and standby) will draw from the process water tank to supply the following services:

- Grinding area density control and mill discharge screening;
- Gravity, trash, tails, barren and loaded carbon screen spray water;
- Pre-leach thickener flocculant dilution;
- Carbon quench and transfer water;
- Cyanide destruction density control;
- Tailings hopper level control;
- Service points throughout the plant.

The process water consumption has been estimated at 789 m^3/h when processing fresh ore and 1,187 m^3/h when processing oxide/transition ores. The process water tank will nominally be kept full during normal operation, with the operation of the decant return water to be modulated to maintain the process water tank level. If insufficient water recovery is generated from thickening or TSF decant, then raw water will be used to supplement the process water supply.

Potable Water

Raw water from the raw water tank will be directed to a dedicated Reverse Osmosis (RO) plant. Water treatment will comprise RO and chlorine sterilisation, with the permeate flowing to the potable tank. Brine rejected from the RO plant will be pumped by the brine transfer pump to the tailings hopper.

Potable water pumps (duty and standby) will discharge into a ring main to supply the plant safety showers, site drinking water, ablution blocks, laboratory, crib room, store, and workshop. Pipe lagging, a water cooler and thermal relief valves will be used to avoid excessively high-water temperature at the safety showers.

Potable water will be piped from the process plant potable water distribution to fill the camp potable water tank. The potable water will then be distributed around the camp by the camp potable water pump (or diesel camp potable water pump in the event of power loss).

Fire Water

The intake for the raw water pumps will be set at an elevated level to ensure a fire water reserve will be always maintained. Fire water pumps (electric duty pump and diesel engine emergency standby) will draw water from the base of the raw water tank to supply the site fire water network, which will include hydrants and hose reels.

17.3.11 Electrical

The electrical power supply and distribution system for the plant is discussed in Part 18. The installed power and consumed power per annum are summarised in Table 17.1 below.

Plant Area	Installed Power (kW)	Consumed Power Fresh Feed (MWh/year)	Consumed Power Oxide/Transition Feed (MWh/year)
Crushing	650	2,839	2,839
Reclaim, Grinding & Classification	18,898	125,456	111,200
Carbon-In-Leach	1,976	9,928	9,928
Gold Recovery & Carbon Regeneration	187	868	868
Tailings Detoxification & Disposal	890	4,941	4,941
Reagents	740	4,425	4,425
Water & Air Services	1280	6,923	6,923
Laboratory, Workshop, Warehouse & Administration	1,017	1,578	1,578
Total	25,638	156,957	142,701

Table 17.1 Plant Power Requirements

17.4 Process Design

17.4.1 Process Design Criteria

The Doropo gold plant will process ore of variable fresh, transitional and oxide feed types from across nine different pits. The LOM feed is 57.6% fresh rock and 42.4% oxide/transitional saprolite or saprock. The largest ore sources are the Kilosegui and Souwa pits, at 35.4% and 29.2% of LOM ore source respectively. The details of the ore sources are shown below in Table 17.2.

PIT	Oxide/Transition (kt)	Fresh (kt)	Total (kt)
ENIODA	1,598	457	2,055
HAN	386	2,804	3,190
KEKEDA	1,517	900	2,417
KILOSEGUI	3,076	8,580	11,656
SOUWA	6,771	5,214	11,985
NOKPA	710	2,253	2,963
CHEGUE SOUTH	595	468	1,063
CHEGUE MAIN	2,208	688	2,896
ATTIRE	1,598	457	2,055
Total	16,861	21,364	38,225

Table 17.2 Ore Sources

The comminution circuit has been sized on the 85th percentile of ore competency and a SABC flowsheet selected for the size reduction of ores preceding leaching. The comminution parameters for the master composites and lithologies are summarised in Table 17.3. The fresh ore at 4.0 Mt/a with a grind size P_{80} of 75 µm is the DFS design point for the Doropo comminution circuit. The oxide/transition ores have more favourable comminution properties and allow a high throughput of 5.4 Mt/a with the same major equipment and P_{80} of 106 µm. The apron feeders and conveyors of the comminution circuit are sized for the oxide/transitional case based on the higher throughput and duty.

Master Composites (1-24) By Lithology		Ore SG	Bond Abrasion Ai	SMC Dwi kWh/m^3	SMC A x b	Bond Ball Work Index BWi (kWh/t)	Bond Ball Work Index RWi (kWh/t)
Overall	85th Percentile of Analyte	2.70	0.29	6.90	38.90	18.45	17.98
	Average	2.63	0.19	4.87	76.96	17.05	15.77
	Maximum	2.73	0.38	8.60	480.30	19.81	21.42
	Minimum	2.29	0.02	0.50	31.30	10.99	10.56

Master Composites (1-24) By Lithology		Ore SG	Bond Abrasion Ai	SMC Dwi kWh/m^3	SMC A x b	Bond Ball Work Index BWi (kWh/t)	Bond Ball Work Index RWi (kWh/t)
Fresh	85th Percentile of Analyte	2.70	0.28	7.27	37.04	18.42	18.21
	Average	2.67	0.20	5.81	48.68	17.42	16.76
	Maximum	2.73	0.38	8.60	85.20	19.59	21.42
	Minimum	2.57	0.02	3.10	31.30	15.86	12.65
Oxide/ Transitional	85th Percentile of Analyte	2.58	0.27	3.15	83.20	18.82	12.61
	Average	2.51	0.16	2.29	154.07	16.13	11.99
	Maximum	2.60	0.36	3.30	480.30	19.81	12.77
	Minimum	2.29	0.05	0.50	79.10	10.99	10.56

Table 17.3 Metallurgical Tested Comminution Properties

The Doropo ores contain a recoverable portion of gravity gold as well as varying silver. Therefore, a gravity circuit is included in the design plant flowsheet, with the gravity circuit to be fed via a portion of the cyclone underflow. The increased silver grades in the Enioda, Souwa, and Kekeda ores has been factored into and allowed for in the design of the adsorption, elution and electrowinning circuits.

The DFS design point for the leach circuit, pre-leach thickener and tails pumping are designed upon the oxide/transitional design case since they all are driven by the higher oxide/transitional ores throughput rate. Leaching will be conventional cyanidation with dissolved oxygen addition and activated carbon adsorption of the leached metal species.

The key design parameters for the processing facility are summarised in Table 17.4.

Description	Units	Oxide/Transitional Ore	Fresh Ore	Source
Annual Throughput	t/a	5,400,000	4,000,000	Centamin
Design Feed Grade (Gold)	Au g/t	2.00	2.00	Centamin
Nominal Feed Grade (Gold)	Au g/t	1.72	1.48	DFS test work
Design Feed Grade (Silver)	Ag g/t	1.56	0.86	DFS test work
Crushing Circuit				
Type		Single Stage Crush	Single Stage Crush	
Plant Utilisation	%	80	80	GRES
Required Crushing Rate	t/h	771	571	Calculated
Grinding Circuit				
Circuit Type		SABC	SABC	GRES
Plant Utilisation	%	91.3	91.3	Centamin
Design Treatment Rate	t/h	675	500	Centamin
Feed rate (F_{80})	mm	118	143	Calculated

Description	Units	Oxide/Transitional Ore	Fresh Ore	Source
Product Size (P_{80})	µm	106	75	Centamin
Gold Recovery				
Leach Tanks	No.	2	2	GRES
Adsorption Tanks	No.	6	6	GRES
Leach & Adsorption Residence Time	hrs	26.6	38.0	Test work
Elution Circuit Size	t	12	12	Test work
Elution Schedule	strips/week	6	6	GRES
Tailings				
Cyanide Destruct Method		Air/SO_2	Air/SO_2	GRES
Destruct Residence Time	hrs	1.5	2.1	Centamin
Tailings % Solids	% Sol.	47	48	Calculated

Table 17.4 Circuit Design Criteria

17.4.2 Reagent and Consumable Requirements

Plant raw water make-up and power requirements are discussed in Section 17.3.10 and 17.3.11 respectively. The consumption rates of key reagents and consumables are summarised in Table 17.5.

Reagent/Consumable	Fresh (t/year)	Oxide/Transition (t/year)
Grinding Media	5,080	5,130
Quicklime	1,200	7,182
Cyanide	1,469	2,095
Carbon	100	135
Flocculant	123	651
Hydrochloric Acid	211	211
Sodium Hydroxide	609	593
Diesel	248	335
SMBS	7,900	6,264
Copper Sulphate	280	594
Antiscalant	124	167

Notes:

- *Fresh ore accounts for 44.1% of the ore processed over the LOM (refer Table 17.2).*
- *Annual liner changes/consumption for crushers and mills are not shown here. Unit consumption rates are provided in Table 21.13 and Table 21.14.*
- *Annual consumption figures for minor reagents and consumables, including in the gold room and for the potable water treatment plant, are not shown here. Unit consumption rates are provided in Table 21.13 and Table 21.14.*

Table 17.5 Processing Plant Annual Consumption Rates for Key Reagents and Consumables

17.5 Plant Design

The key parameters and processes for the design and equipment selected are outlined below.

17.5.1 Crushing

The ROM bin is designed to accommodate dual tipping of ore from each side of the bin by CAT 785 trucks or CAT 993 FEL. The capacity of the crushing circuit is 571 t/h on fresh ores and 771 t/h on oxide/ transitional feeds. The feed top size to the ROM bin will be limited to 800 mm by square static grizzly screen to prevent oversize causing blockages in the crushing circuit.

The primary crusher will crush F_{100} circuit feed of 800 mm to a P_{80} product size of 118 mm for oxide/ transitional ores and P_{80} 143 mm for fresh ore. Pre-scalping of the -100 mm material enables the selection of a single toggle jaw crusher with a CSS of 140 mm for the primary crushing duty. The design point of 75 MPa, representing the 85th percentile of unconfined compressive strength (UCS) from metallurgical test work, was used for primary crusher sizing (refer Table 17.6).

17.5.2 Ore Storage and Reclaim

The stockpile feed conveyor transports the crushing circuit product to the COS. The COS has been designed with an assumed draw down angle of 60°. The design ore SG is 2.58 for oxide/transitional feeds and 2.70 for fresh ores, determined from comminution metallurgical testing across DFS ore sources. The COS is designed with a total capacity of 50,000 tonnes and have a live capacity of 12,000 tonnes to provide 18 hours of ore storage on oxide/transitional feeds and 24 hours on fresh ore.

Reclaim from the COS is via two apron feeders situated centrally beneath the COS. Each apron feeder is designed to draw up to 675 t/h, ensuring that a single feeder can feed the grinding circuit. The apron feeders discharge onto the SAG mill feed conveyor, which feeds into the SAG mill. The reclaim and SAG feed conveying system is designed to feed the SAG mill at up to 675 t/h of new feed. The SAG feed rate will be measured by the SAG mill feed conveyor weightometer.

17.5.3 Grinding and Classification

The grinding circuit is designed on an SABC configuration, using the 85th percentile of the PFS comminution test work for the fresh and oxide/transitional ores. The ore type comminution parameters were then modelled and mass balance to design the comminution circuit. The DFS metallurgical test work determined the optimal grind size for gold extraction as 75 µm for fresh ores and 106 µm for oxide/transitional ores. The comminution design and modelling are centred on achievement of a 75 µm grind for fresh ore types and 106 µm for oxide/transitional ores.

The comminution circuit modelling, design parameters, and equipment selection for each design case is summarised in Table 17.6 below.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Ore SG, 85th Percentile		2.58	2.70
UCS 85th Percentile	Mpa	75	75
CWi 85th Percentile	kWh/t	5.6	10.9
RWi, 85th Percentile	kWh/t	12.6	18.2
BWi, 85th Percentile	kWh/t	18.8	18.4
Abrasion Index		0.270	0.280
Annual Throughput	t/a	5,400,000	4,000,000
Crushing Rate	t/h	771	571
Crusher Utilisation	%	80	80
Crusher Feed F_{80}	mm	332	495
Crusher Product P_{80}	Mm	118	143
Grinding Rate	t/h	675	500
Plant Utilisation	%	91.3	91.3
Product Size P_{80}	Um	106	75

Table 17.6 Comminution Design Parameters

The primary grinding mill will be a SAG mill with an 8.70 m (IS) diameter and 5.0 m EGL. The SAG mill is designed with a Variable Voltage Variable Frequency (VVVF) variable speed drive and a motor power of 7,000 kW. The SAG mill speed is anticipated to range from 60-80% of critical speed to maintain mill filling, which is important given the range of material characteristics that will be processed.

The expected operating conditions for the primary grinding SAG mill are shown in Table 17.7.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Design Ball Load	%	13	13
Design Total Load	%	25	25
Mill Speed	%CS	71	76
Mill Motor Power Draw	kW/h	5,500	6,000
Ball size	mm	125	125
Mill discharge density	% solids	75	75

Table 17.7 SAG Mill Operating Conditions

The oxide material has a DWi of 2.6 kWh/m³ and a ratio of BRWi to BBWi is 0.7, and therefore is not expected to generate significant pebbles. However, the DWi for the fresh material is 7.4 kWh/m³ and, although the ratio of BRWi to BBWi is 1.0, there are areas of more component rock which has justified the inclusion of the pebble crusher for the fresh ore. Due to the differing ore properties, the design pebble rate is 15% of new feed for oxide/transitional ore and 20% for fresh ore, which at the different design rates equates to approximately 100 t/h of pebbles.

A ball mill will provide secondary grinding. The ball mill will be an overflow ball mill with a 6.8 m (IS) diameter and 10.2 m EGL. The ball mill is designed with a VVVF drive and motor power of 9,000 kW, which will be required to meet the differing power demands based on ore type processed. The designed recirculating load is 250% for oxide/transitional and 300% for fresh feeds. The design ball charge is 30%, with 60 mm grinding balls designed as the make-up charge.

The expected operating conditions for the secondary grinding ball mill are shown in Table 17.8.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Design Ball Load	%	30	30
Mill Speed	%CS	78	74
Mill Motor Power Draw	kW/h	8,750	8,200
Ball size	mm	60	60
Mill discharge density	% solids	70	70

Table 17.8 Ball Mill Operating Conditions

The mill discharge hopper is common to both the SAG and ball mills. The hopper is designed to contain 60 seconds of slurry flow at the maximum oxide/transitional flow, which equates to a mill discharge hopper of 39 m³.

The cyclone cluster will comprise of 15 installed cyclones, each of 400 mm diameter, and one blank outlet for sampling or expansion purposes. Of the 15 installed cyclones in the cluster, 14 are anticipated to operate for oxide/transitional feeds, while 11 operating with fresh ore feeds. The cyclones are expected to operate between 80-120 kPa and produce the targeted grind size. The cyclone spigots and vortex finders will be changed as needed to meet the target grind size based on ore type.

17.5.4 Gravity Gold Recovery

A gravity circuit, including a screen, centrifugal concentrator, and intensive leach reactor is, included in the design flowsheet. The Gravity Recoverable Gold (GRG) for each ore type was estimated by laboratory-scale concentration in the DFS test work, scaled GRG testing was not undertaken. The laboratory-scaled gravity recovery varies between deposits and lithologies, and a design point of 15% GRG was stipulated by Centamin. The 15% threshold represents half of the averaged gravity recovered by laboratory concentration within the 24 master composites summarised in Figure 17.2.

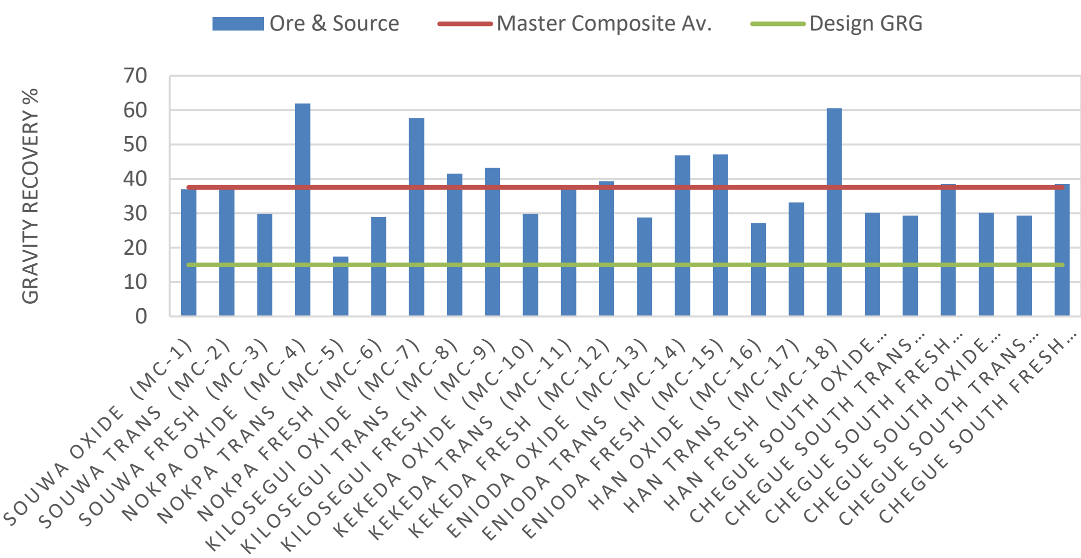


Figure 17.2 *Laboratory Scale Gravity Recovery by Master Composite*

The centrifugal gravity concentrator is anticipated to operate on a nominal 40-minute cycle time. This cycle time includes the discharge and flushing time, and each cycle produces approximately 60 kg of gravity concentrate. The concentrator under full operating conditions, is expected to produce 36 cycles per day, estimated to collect of 2,160 kg concentrate.

Gravity concentrate collected over a 24-hour period will be batch-treated in the intensive leach reactor. Laboratory test work has shown the intense cyanidation results in high gold extraction.

17.5.5 Leaching and Adsorption

The pre-leach thickener will be a hi-rate thickener of 35 m in diameter. The thickener is designed based on the oxide/transitional feed case with a specific settling rate of 0.75 t/m^{2}h. The fresh feeds test work indicates faster settling at 1.0 t/m^2/h. The thickener increases the slurry density feeding the Carbon-In-Leach (CIL) circuit from 39-41% solids to an underflow density of 48% solids for oxide/transitional feed and 50% solids for fresh ore feeds.

The leach circuit was designed with two leach tanks preceding adsorption in six CIL configured tanks to provide a minimum overall residence time of 30 hours for fresh ore. The metallurgical test work indicated:

- Gravity gold recovery prior to leaching improved overall recovery in both oxide/transitional and fresh ore types;
- The leach kinetics indicate that 80-90% of extracted gold is achieved in 24 hours of leaching, with several ores benefitting from additional leach time up to 30 hours;

- There are low levels of deleterious elements in the oxide/transitional and fresh ores, and these do not appear to impact leaching, and pre-robbing is unlikely;
- The use of oxygen addition over air sparging improved overall recovery in oxide/transitional and fresh ores, so oxygen sparging is incorporated in design;
- The addition of lead nitrate in pre-oxidation CIL provided no or marginal recovery benefit in the Chegue South and Main deposits, so lead nitrate dosing is not incorporated;
- The fresh ores of Souwa, Kekeda, Enioda, Han and Chegue Main contain sulphide sulphur. However, DFS tested peroxidation provided no or marginal improvement to recovery. Therefore, no allowance for pre-oxidation is included in the design;
- The use of a multiple shear reactor (MACH©) in direct cyanidation leaching provided no or marginal improvement to recovery, so the design excludes the use of a multiple stage shear reactor;
- An oxygen shear reactor recirculating slurry within Leach Tank 1 is marked for future expansion, should pre-oxidation be required. Oxygen addition will be sparged into the pre-leach thickener underflow, with oxygen addition by agitator downshaft sparging in Leach Tanks 1 & 2 and CIL Tanks 1 to CIL Tank 4;
- Soluble copper was noted in Kekeda and Enioda oxides and transitional ores. This copper is anticipated to leach and load onto carbon. Provision for cold cyanide washing prior to elution is noted to be included in the design, as a selectable addition to the elution sequence;
- The cyanide consumption for oxide/transitional ores averaged 0.38 kg/t and 0.35 kg/t for fresh ores, and lime consumptions averaged 1.33 kg/t in oxides/transitional ores and 0.30 kg/t for fresh ores;
- The viscosity design point for oxide was assumed to be 1,000 cP at 2 sec^{-1}. This assumption was confirmed and validated by the DFS testing. However, the Kilosegui oxide was double the design point at 50% solids, whereas at 45% solids, the Kilosegui oxide lies within the acceptable design viscosity at 936 cP at 4.2 sec^{-1}. A lower density or blend management will be required to treat this ore;
- Souwa, Kekeda, Enioda and Han head assays indicate the presence of appreciable silver. The anticipated silver recovery is accounted for in the adsorption, elution, and electrowinning design;
- The DFS test work for gold deportment by mineralogy or diagnostic leaching, oxygen uptake, adsorption constants, and carbon kinetics/equilibrium load results were not available at time of DFS plant design.

The anticipated gold extraction over time for the oxide/transitional and fresh ore design basis is shown in Figure 17.3. The extraction curves show the anticipated leach recovery over time, with coarse gold removed prior to leaching in the gravity circuit.

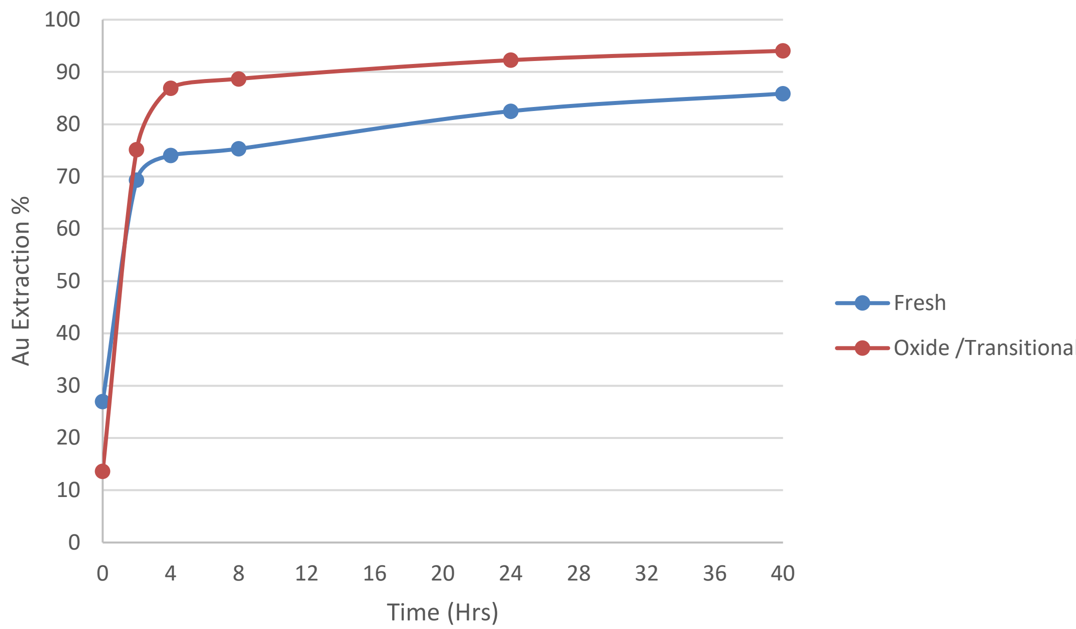


Figure 17.3 *Gold Extraction Curves for Oxide/Transition and Fresh Ores*

The leach circuit designed reagent scheme and parameters are summarised in Table 17.9.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Feed Pulp Density	% Solids	48	50
Leach Stages	number	2	2
Adsorption Stages	number	6	6
Intertank Screen Size	m^2	15	15
Tank Size	m^3	3,300	3,300
Total Residence Time	Hrs.	26.6	38.0
Leach pH		10	10
Leach Cyanide Concentration	ppm	300	250
Leach Oxygen Concentration	ppm	20	20

Table 17.9 Leach Circuit Design Parameters

The adsorption circuit is designed based on assumed efficiency and isotherm constants, with k of 1.00 h-1 and n of 0.8 and assumed efficiency of 98%. The DFS tested efficiency, isotherms and constants were not available at time of plant design. The assumed constants, tank volumes and DSF leach extraction curves have been modelled to generate the optimal carbon advance rate (t/day), carbon concentration (g/L) and dwell time (days), all while maintaining the gold solution losses. The key design parameters for the adsorption circuit are summarised in Table 17.10.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Carbon Concentration	g/L	8	8
Carbon in Circuit	t	158	158
Carbon Dwell Time	Days	14	14
Design Flemming Constants -k -n	h^{-1}	100 0.8	100 0.8
Carbon Advance Rate	t/d	11.3	11.3
Carbon Advance Period	H	6	6
Forwarding Rate	m^3/h	235	235

Table 17.10 Adsorption (CIL) Design Parameters

The carbon forwarding pump in CIL tank 1 pumps pulp and carbon to the loaded carbon screen. The loaded carbon screen is a horizontal vibrating screen, designed with a specific screening duty of 36 $m^3/h/m^2$ and slotted apertures of 0.83 mm. This is suitably sized for draining and washing of the slurry from the carbon prior to acid washing.

The tailings screen is a horizontal vibrating screen, designed with a specific screening duty of 44 $m^3/h/m^2$ for oxide and transition ores (31 $m^3/h/m^2$ for fresh) with slotted apertures of 1.0 mm.

In the unlikely event of catastrophic tank failure, slurry will flow to North via a spillway into the site event pond, where it will be captured and contained. The event pond is designed to hold 110% of the volume of one leach/CIL tank or contents of the leach feed thickener.

17.5.6 Elution and Gold Recovery

The elution and gold recovery circuit are designed on the daily movement of 12 tonnes of carbon. The 12-tonne capacity is a function of the CIL modelling, feed grade and gold extraction. The elution circuit will be a 12-tonne Zadra circuit processing six strips per week, as stipulated by Centamin, in preference to the AARL or split AARL method.

The elution column is designed to contain 12 tonnes of acid-washed carbon within a 304 stainless steel column. The anticipated loaded gold and silver carbon grades are 1,944 Au g/t, 1,689 Ag g/t for oxide/transitional feeds and 1,316 Au g/t, 738 Ag g/t for fresh ores. Based on these anticipated gold and silver loads, two electrowinning cells in parallel are required to recover the precious metals.

To maintain efficient stripping, every third Zadra strip is designed to be a complete water and reagent replenishment. Outside of these strips, a portion of the spent eluate (2 BV) will be reused, with the remainder bled to the leach feed distribution box as barren eluate. A top-up of raw water and reagents will occur before the elution of the next strip.

Parameter	Units	Oxide/Transition Ore	Fresh Ore
Elution Column Capacity	t m³	12 26.7	12 26.7
Operating Pressure	kPa	250	250
Elution Flowrate	BV/h	2	2
Fresh Eluate Make-up NaCN NaOH	# Batch % (w/w) % (w/w)	3 0.2 2.0	3 0.2 2.0
Make-up NaCN NaOH Raw Water	 m³ m³ m³	 0.4 0.2 52.7	 0.4 0.2 52.7
Top-up NaCN NaOH Raw Water	 m³ m³ m³	 0.2 0.1 26.4	 0.2 0.1 26.4
Strip Duration	BV Hr	4.5 16	4.5 16
Elution Heater Pre-Heat Temp Elution Temp	kW ⁰C ⁰C	2,500 95 135	2,500 95 135
CIL Electrowinning	# cells Configuration Cathode # Anode #	2 Parallel 15 16	2 Parallel 15 16
Rectifier Size	Amps (per cell)	2,200	2,200

Table 17.11 Elution Circuit Design Parameters

The CIL electrowinning cells are designed to operate as two units in parallel, each 1.0 m x 1.0 m in dimension and equipped with 2,200 A rectifiers. The CIL and gravity electrowinning cells will operate independently for metals accounting purposes. The CIL electrowinning cells have been designed for the anticipated and modelled gold and silver concentrations, with 15 cathodes and 16 anodes installed in each CIL cell. The cathodes are to be lined with stainless steel mesh for gold capture, with pressure cleaning in-situ for sludge recovery.

17.5.7 Carbon Regeneration

The carbon regeneration kiln is designed to treat 750 kg/h of carbon at an operating temperature of 700-750°C, allowing it to process 12 tonnes within 16 hours. The selected regeneration kiln is an indirect diesel-fired horizontal kiln. The regenerated carbon will discharge from the kiln into the carbon quench vessel, which is designed to hold 6 tonnes of carbon. The quench vessel will transfer the regenerated carbon to the barren carbon screen and CIL tank 6 in two batches. The carbon quench pump will transfer the quenched carbon to the barren carbon screen.

17.5.8 *Tailings and Cyanide Destruction*

The slurry discharging from the CIL circuit gravitates to the tailings screen. The horizontal vibrating tailings screen is designed based on the oxide/transitional specific duty of 44 $m^3/h/m^2$

The selected destruction method is the INCO process (Air/SO_2), which is to produce a plant tailings WAD cyanide below 50 ppm, for a tailings feed concentration of 100 ppm. A 1.62 molar ratio of SO_2 to CN was allowed in the design to account for side reactions and inefficiencies. The oxygen demand was determined from the SMBS addition rate with an adsorption efficiency of 20%.

The metallurgical test work shows insufficient natural copper in most ore types to provide sufficient copper in solution to catalyse the destruction reaction, exceptions being Enioda Oxide and Enioda Transitional ore. Therefore, the design includes the dosing of copper sulphate solution to the cyanide destruction circuit. The design also provides for caustic addition to the cyanide destruction feed box, to adjust or maintain pH if required.

The cyanide destruction test work shows that most of the cyanide present is as free cyanide, resulting in a rapid destruction process. An allowance of 1.5 hours residence time is appropriate to complete the reaction and achieve the target WAD CN concentration.

17.6 Plant General

17.6.1 *Metallurgical Accounting*

Metallurgical accounting will be facilitated by installed instrumentation and automatic samplers. The pre-leach thickener underflow and CIL tank 6 discharge streams will be fitted with two-stage automatic slurry samplers. The leach feed and tails samplers will composite a shift sample for gold and silver assay, providing leach feed grade, leached slurry residue grade (tails solids) and tails solution loss.

Solution samplers will be installed on the gravity pregnant solution transfer and gravity electrowinning barren solution pipelines. The solution assay, combined with the transferred pregnant solution flow meter, batch volume, provides in stream accounting for the recovered gravity gold. This will be cross checked against the doré poured from the gravity electrowinning cell.

The gravity gold recovered may then be combined with the leach feed grade to back calculate the head grade for the plant.

The overall milled tonnes per shift will be accounted by the weightometer installed on the SAG mill feed conveyor. The weightometer totaliser will record and historize tonnes per shift. The density and flow instruments installed on the pre-leach thickener underflow or tailings pipeline, combined to calculate mass flow, may be used as a secondary measure or cross check of milled tonnes.

In addition to the SAG mill feed conveyor, weightometers will be installed in the following locations:

- Primary crusher discharge conveyor for crushed tonnes per shift;
- Pebble crusher feed conveyor for mill pebble tonnes per shift.

The CIL gold recovered to elution will be measured by solutions samplers on the pregnant eluate and barren eluate pipelines. These, along with manual carbon samples for the loaded carbon, barren carbon and regenerated carbon, will be used to monitor strip efficiency. The CIL gold sludge collected and poured into doré from the leach/CIL circuit will be the final accounted measure.

Flowmeters on reagent dosage lines, including process, decant, and raw water lines, will be used to account for reagent and water consumptions. The recorded reagent consumptions will be reconciled with reagent deliveries and changes in site stock.

Gold-in-Circuit (GIC) surveys will be conducted periodically to reconcile the gold poured against the head grade, tonnage, and changes in circuit gold inventories.

17.6.2 Process Plant Control System

The process plant control system will be a Programmable Logic Control (PLC) based system operated from a centralised control room. The operator interfaces will utilise standard personal computers running proprietary process control software. The plant status will be monitored from the control room via PC-based Supervisory Control and Data Acquisition (SCADA) screens running the process control software. The Process Control System (PCS) controls the process interlocks, process controllers, and process alarms allowing operators to control and monitor the entire plant.

Plant equipment will be started and stopped remotely from the operator screens via start and stop sequences or by the operator manually starting or stopping individual items of equipment. The plant equipment will report back 'ready to run' and 'run' status and report fault states. The equipment will be designed with three modes of operation: remote group start/stop, remote, and local. The mode of operation will be selected by the plant control room operator. The normal mode of operation is anticipated to be remote operation, with individual circuits started/stopped by sequences initiated by the control room operator.

Field instrumentation will feed back into the PCS via remote I/O modules linked by ethernet networks. Digital and analogue instrument signals will provide real time monitoring of the process. Analogue I/O signals will provide measurement of process flow, pressure, density, pH, and temperature, and will control the position of modulating valves and actuators, and the speed of variable speed drives. Digital signals will provide the operational status and control of drives, valves, and actuators, and detect alarm conditions to annunciate warnings.

Supervisory controllers will be utilised by the PCS to maintain and control the process to operator setpoints of cascaded process values. In this method, the PCS self-regulates, minimising the need for operator intervention. There will be three modes of control: cascade, auto, and manual. In cascade control, the controller setpoint is adjusted by the PCS based on real time plant conditions. In automatic control, the controller adjusts to maintain a fixed setpoint, typically entered and adjusted by the control room operator. In manual mode, the control room operator manually adjusts the control variable from the control room.

Fixed Closed-Circuit Television (CCTV) cameras will be used to provide remote visual monitoring of specific sections of the circuit, with display screens located in the plant control room.

17.7 Equipment Selection

Equipment selection was undertaken through proposal inquiries to multiple vendors for all major packages. Equipment specifications and datasheets were prepared for each equipment package to facilitate quotation preparation by vendors. The quotations were technically and commercially evaluated to select suitable equipment for the Project.

17.7.1 Crushers

The Primary crusher will be a jaw crusher with pre-scalping of fine material performed via a vibrating grizzly. The jaw crusher will treat the grizzly oversize material. The pebble crusher will be a cone crusher to treat SAG mill critical-sized pebbles. Vendors provided their technically preferred solution based on the developed mass balance and respective crusher datasheets.

17.7.2 Grinding Mills

The primary mill and secondary mill will be a trunnion-supported mills with variable voltage, variable frequency (VVVF) drives for primary and secondary grinding. Vendors provided their technically preferred solutions based on the developed mass balance, comminution modelling and respective datasheets.

17.7.3 Gravity Circuit

The gravity circuit will treat a portion of cyclone underflow and will encompass a gravity scalping screen, gravity concentrator, and Intensive Leach Reactor (ILR). The scalping screen with be a horizontal vibrating screen feeding a single gravity concentrator, which will produce the gold concentrate for intensive cyanidation leaching to be performed in the ILR. Vendors provided their technically preferred solutions based on the developed mass balance, ore test work, and respective equipment datasheets.

17.7.4 Thickener

The pre-leach thickener will be carbon steel, high-rate, above-ground thickeners complete with a full truss bridge, multi-pinion system of planetary gearboxes, hydraulic rake drive system and control panel. Vendors provided their technically preferred solution based on the developed mass balance, dynamic thickening metallurgical test work, and thickener datasheet.

TABLE OF CONTENTS

18. PROJECT INFRASTRUCTURE 1
18.1 Introduction 1
18.2 Roads and Site Access 6
18.2.1 Site Access Road 6
18.2.2 Haul Roads 7
18.2.3 Airstrip Design 8
18.3 Logistics 10
18.3.1 Construction Logistics 11
18.3.2 Operational Logistics 12
18.3.3 Non-Mining Mobile Equipment 12
18.4 Geotechnical Considerations 13
18.4.1 Geotechnical Investigations 13
18.4.2 Investigation Findings 14
18.5 Tailings Storage Facility 26
18.5.1 TSF Design 26
18.5.2 TSF Dam Break Assessment 43
18.6 Surface Water Management 48
18.6.1 Site Water Balance 48
18.6.2 Water Harvest Dam 51
18.6.3 Water Storage Dam 55
18.6.4 Sediment Management 57
18.6.5 Monitoring 59
18.6.6 Raw Water and Potable Water 60
18.7 Power Supply and Distribution 61
18.7.1 Power Supply 61
18.7.2 Power Distribution 62
18.8 Bulk Earthworks 63
18.9 Mine Infrastructure 64
18.10 Administration and Processing Infrastructure 65
18.11 Accommodation Village 66
18.11.1 Security Camp 66
18.11.2 Main Camp 66
18.12 Sewage Treatment 67
18.13 Fuel and Lubricants 67
18.14 Communications 67

18.15 Security and Fencing 68
18.15.1 Process Plant Security 68
18.15.2 Accommodation 68
18.15.3 Mining Areas 68

LIST OF FIGURES

Figure 18.1 Site Layout (Source: Knight Piésold, 2024) 3
Figure 18.2 Mine Infrastructure Layout (Source: Knight Piésold, 2024) 4
Figure 18.3 Plant and infrastructure Layout (Source: GRES, 2024) 5
Figure 18.4 Transport Route 11
Figure 18.5 TSF General Arrangement - Stage 1 (Source: Knight Piésold, 2024) 27
Figure 18.6 TSF General Arrangement - Final Stage (Source: Knight Piésold, 2024) 28
Figure 18.7 Final Stage TSF Embankment - Typical Section (Source: Knight Piésold, 2024) 29
Figure 18.8 WHD Layout (Source: Knight Piésold, 2024) 53
Figure 18.9 WSD Layout (Source: Knight Piésold, 2024) 54

LIST OF TABLES

Table 18.1 Site Access Roads Design Parameters 6
Table 18.2 Site Haul Roads Design Parameters 7
Table 18.3 Site Airstrip Design Parameters 10
Table 18.4 Geotechnical Investigation Summary 14
Table 18.5 TSF Design Parameters 32
Table 18.6 Staged Embankment Construction 34
Table 18.7 Case 1 - Indicative Population at Risk (PAR) 45
Table 18.8 Case 2 - Indicative Population at Risk (PAR) 46
Table 18.9 Case 3 - Indicative Population at Risk (PAR) 46
Table 18.10 ANCOLD Consequence Category - Dam Failure 47
Table 18.11 ANCOLD Consequence Category - Environmental Spill 47
Table 18.12 ANCOLD Design Criteria 48
Table 18.13 Water Harvesting Dam Design Parameters 52
Table 18.14 Water Storage Dam Design Parameters 56
Table 18.15 Sediment Management System Design Parameters 57
Table 18.16 SCS Design Information 58
Table 18.17 Surface Water Management System Design Parameters 58

18. PROJECT INFRASTRUCTURE

18.1 Introduction

The Doropo Gold Project (the ‘Project’) is a greenfield development that will require the construction of various ancillary facilities and related infrastructure to support the mining, processing, and waste management operations. The plant and infrastructure locations have been selected to take advantage of local topography, to accommodate environmental considerations, and to reduce capital and operating costs.

Project facilities and infrastructure that support the operations will include:

- Access:
  - A site access road from the existing national road network to the mine site;
  - Haul roads from mine pits to the plant and waste storage areas;
  - Internal access roads in an around the plant and infrastructure areas; and
  - An airstrip;
- Water supply:
  - A Water Harvest Dam (WHD) for harvesting surface water;
  - A Water Storage Dam (WSD);
  - Surface water management and sediment control structures;
  - A potable water supply; and
  - Waste water treatment;
- Power supply and distribution:
  - Connection to the Côte d’Ivoire grid via a 90 kV transmission line (approx. 65 km) from a switchyard at Bouna;
  - A mine site substation, including a 90/11 kV transformer to provide an 11 kV feeder for the site;
  - 11 kV power distribution from the mine site substation to the plant and facilities; and
  - Emergency power via on-site diesel generation;
- On-site ancillary facilities:
  - Administration buildings and offices, including messing and ablutions facilities;
  - A medical clinic and emergency response facilities;
  - Security facilities and change house for plant operations and maintenance personnel;
  - Plant maintenance facilities, including a workshop, warehouse and stores;
  - Fuel storage and distribution facilities;
  - Sewage treatment;
  - A fire protection system; and
  - Communications infrastructure;

- Other supporting infrastructure:
  - Tailings Storage Facility (TSF);
  - Mining services area, comprising infrastructure to support the mining operation;
  - Waste rock storage areas; and
  - Explosives storage facilities;
- Accommodation facilities:
  - A 300-bed main camp; and
  - An 80-bed security camp.

The site layout and layout drawings for the plant and infrastructure are provided in Figure 18.1, Figure 18.2 and Figure 18.3.

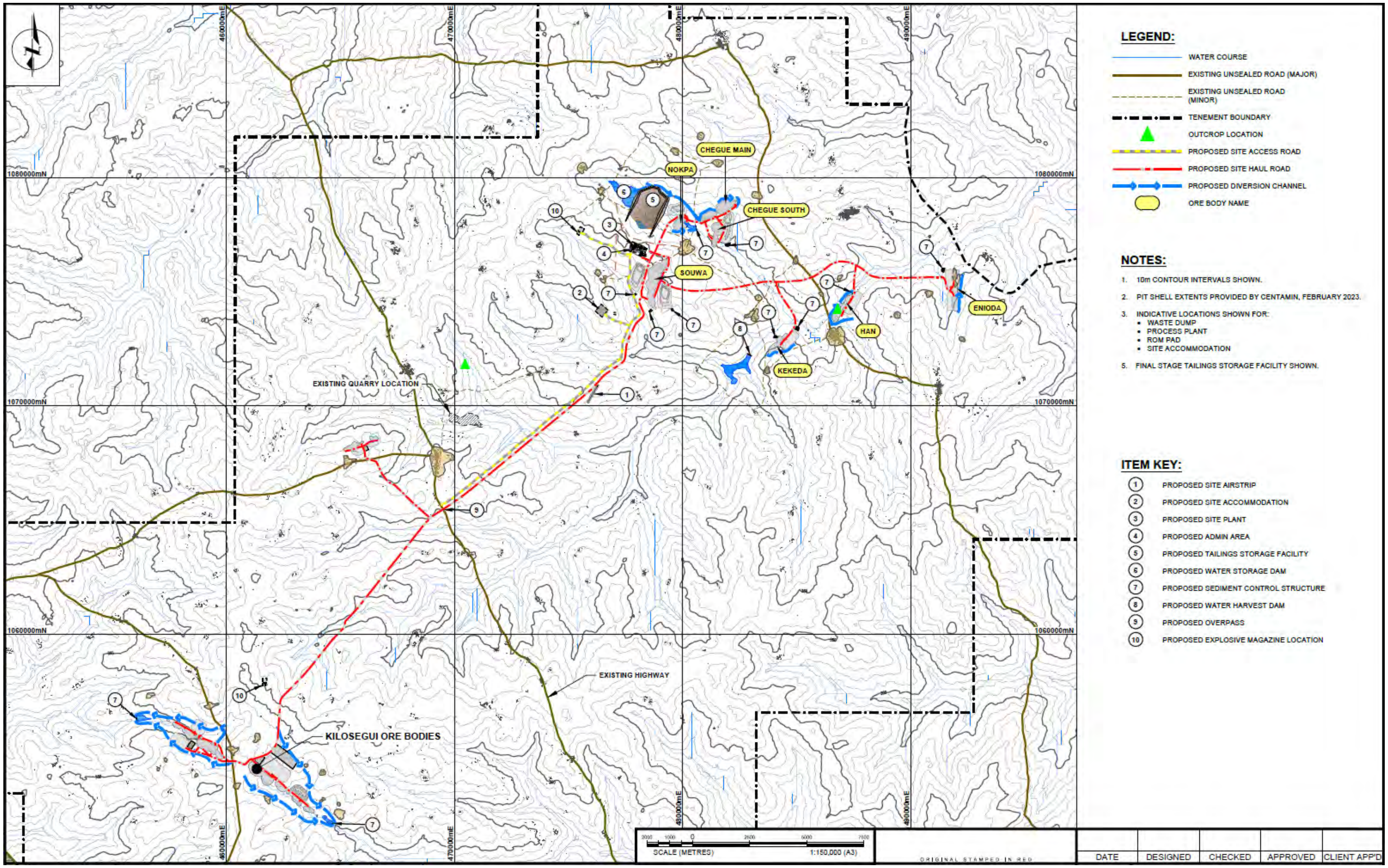


Figure 18.1 Site Layout (Source: Knight Piésold, 2024)

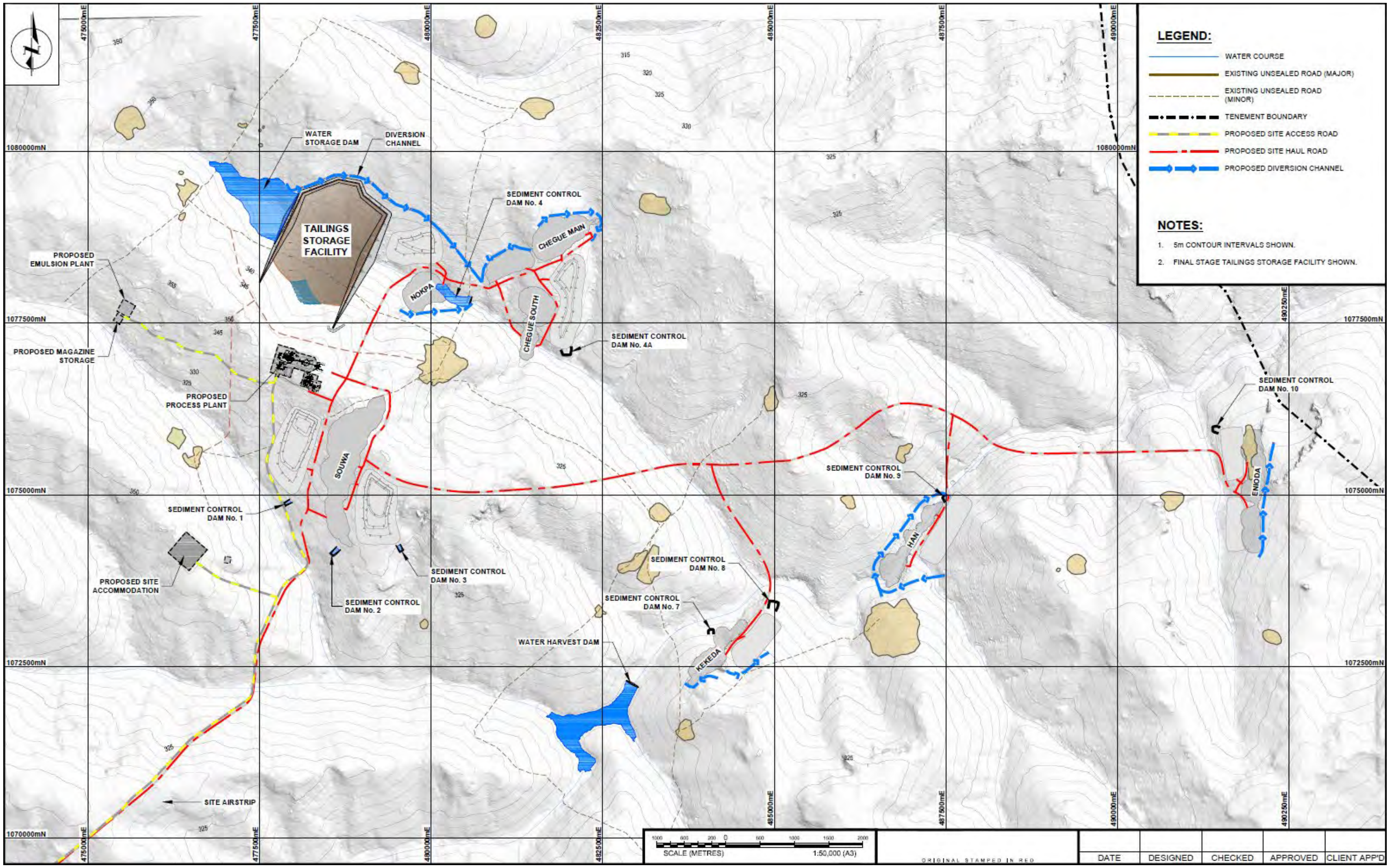


Figure 18.2 Mine Infrastructure Layout (Source: Knight Piésold, 2024)

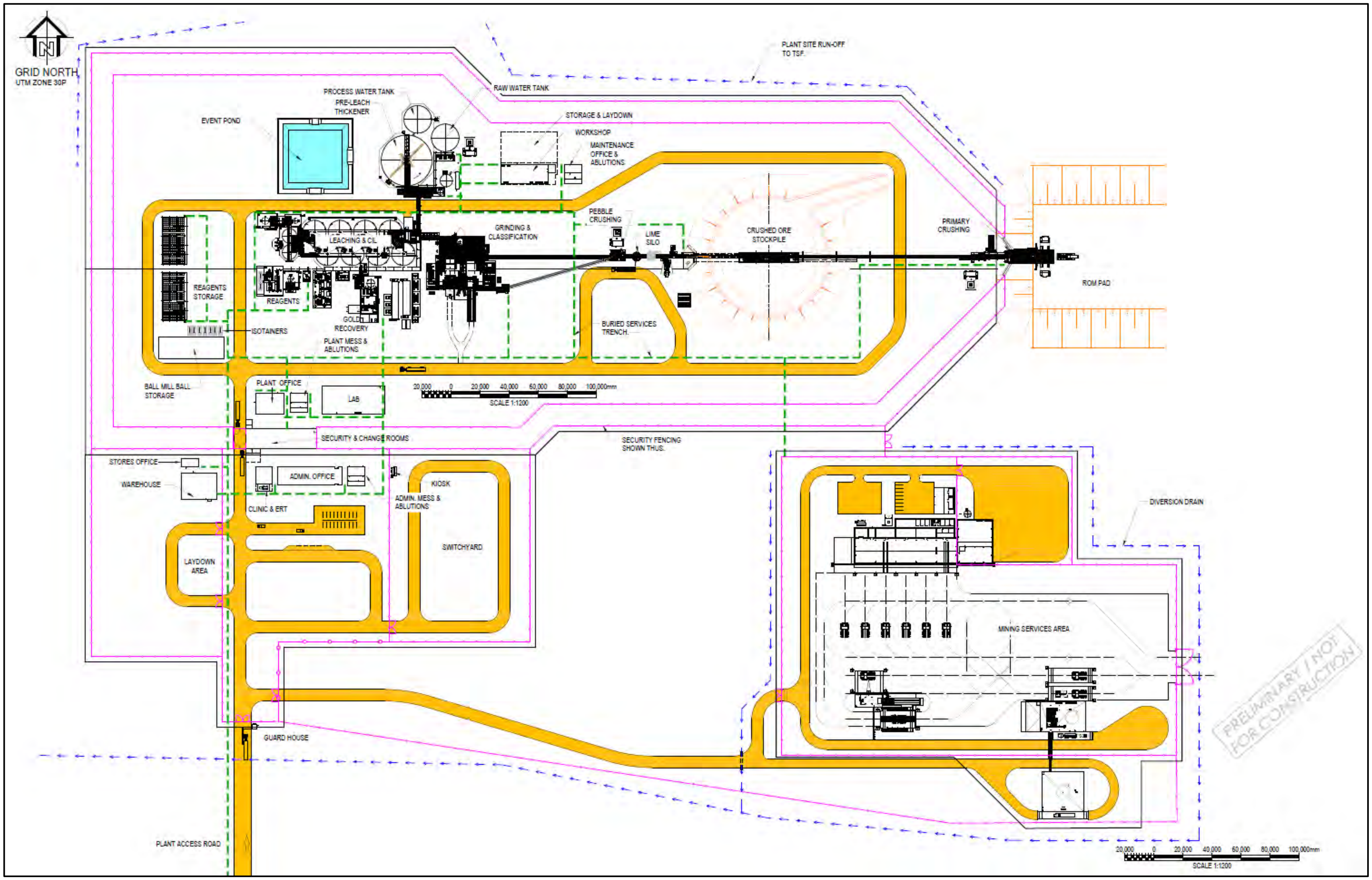


Figure 18.3 Plant and infrastructure Layout (Source: GRES, 2024)

18.2 Roads and Site Access

18.2.1 Site Access Road

Site Access roads were designed to facilitate the following movements during operation:

- Process Site to A1 National Highway;
- Process Site to Accommodation Village and Magazine; and
- Process Site to Airstrip.

The total length of the site access roads is 16 km. Design parameters for the site access roads are provided in Table 18.1. The Site Access Road from the A1 National Highway will be constructed pre-commissioning to allow access to the site.

Design Parameter	Value
Road Cross Section	Carriageway Width - 10 m Lane Width - 3.5 m Shoulder width - 1.5 m Safety Bunds - 0.5 m height where fill height > 2 m Crossfall - 2%
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria:	10 year ARI
Pavement	150 mm laterite gravel wearing course

Table 18.1 Site Access Roads Design Parameters

The site access road alignments are shown in Figure 18.1 and Figure 18.2.

The road vertical alignments were designed to balance cut to fill where practicable. Local balancing of cut and fill volumes will be completed during the next design phase. The vertical alignment design also includes allowances for fill build-up around stream crossings to ensure correct operation of crossings, and for excavations in areas of steeper terrain, to meet design parameters and safety requirements.

The site access roads will comprise two 3.5 m wide running lanes with a 1.5 m shoulder on each side resulting in a total formation width of 10 m. The road crossfall will vary along the alignment to accommodate drainage requirements. A 150 mm laterite wearing course will be placed over the subgrade/general fill.

Drainage ditches, turnouts and level spreaders will be incorporated into the site access road design during the next design stage. A drainage ditch will run along either side of the road formation as needed, conveying runoff to level spreaders and culvert structures. The drainage ditch will be grader cut, with the excavated material pushed into the road formation to be used as general fill.

Culvert crossings were designed at significant stream-crossing locations to convey all runoff resulting from a 10 year average recurrence interval (ARI) event. Culverts typically comprise either corrugated metal pipe structures or precast concrete box culverts, depending on the availability and cost considerations. The culvert inlets and outlets will comprise stone pitched headwalls. A minimum cover of 800 mm was adopted for the design, exceeding the standard the minimum cover requirements for culverts.

18.2.2 Haul Roads

Site haul roads were designed to facilitate the following haulage movements during operation:

- Pits to waste dump(s);
- Pits to ROM Pad;
- Pits to Mine Services Area; and
- Northern Pits to Kilosegui Ore Bodies.

The total length of the haul access roads is 30 km. Design parameters for the site access roads are provided in Table 18.2. Access for the mining fleet to the TSF embankment for structural fill (Zone C1) placement will be via a haul road constructed from the western waste dump during the early stages of operation.

Design Parameter	Value
Road Cross Section	Total road width - 16 m Double Lane Width - 12 m Safety Bunds - 1 m height Crossfall - 2%
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria:	10 year ARI
Pavement	200 mm laterite gravel wearing course (to be upgraded as required during the operation)

Table 18.2 Site Haul Roads Design Parameters

The haul access road alignments are shown in Figure 18.1 and Figure 18.2.

The road vertical alignments were designed to balance cut to fill where practicable. Local balancing of cut and fill volumes will be completed during the next design phase. The vertical alignment design included allowance for fill build-up around stream crossings to ensure correct operation of crossings, and for excavations in areas of steeper terrain, to meet design parameters and safety requirements.

The site haul roads will consist of two-way passing 12 m wide running lanes, with two 1.0 m high safety bunds, resulting in a total formation width of 17 m. A 200 mm laterite wearing course will be placed over the subgrade/general fill. Once competent waste rock is available from the operation, the site haul roads should be rock sheeted by the mining operation (this has not been included in the construction quantities).

Drainage ditches, turnouts and level spreaders will be incorporated into the haul access road design during the next design stage . Drainage ditches will run along either side of the road formation as required, to convey runoff to level spreaders and culvert structures. The drainage ditches will be grader cut, with the excavated material pushed into the road formation to be used as general fill.

Culvert crossings were designed at significant stream-crossing locations to convey all runoff resulting from a 10 year ARI. The culverts will typically consist of either corrugated metal pipe structures or precast concrete box culverts, depending on the availability and costs. The culvert inlets and outlets will comprise stone pitched headwalls. A minimum cover of 800 mm was adopted for the design, which exceeds the standard minimum cover requirements for culverts.

18.2.3 Airstrip Design

The airstrip design is summarised below:

- The airstrip was designed in accordance with the Australian Government Civil Aviation Safety Authority (CASA), RACI 6001 Aerodrome Design and Operation (ANAC) and International Civil Aviation Organization (ICAO) guidelines (Ref. 6, 7 and 8). The airstrip design parameters are provided in Table 18.3;
- The design aircraft for the Doropo airstrip is a 14-seat Cessna Caravan;
- The prevailing wind direction of the Doropo project site is from the south west (SW) to the south (S), based on data sourced from the Gaoua station (~67 km away from Doropo site). The airstrip will be orientated accordingly to ensure optimal operability;
- The runway surface is 850 m long and 20 m wide, with an 80 m wide surrounding runway strip inclusive of the runway. The runway cross-fall is less than 2% from the crown, and the vertical grade is less than 1% along the entire length to reduce earthwork volumes;
- The natural topography along the design alignment exceeds the maximum allowable longitudinal runway slope of 2.0% and/or the maximum allowable rate of change in slope. Therefore, cut and fill operations will be required to achieve compliance with ICAO guidelines;
- A clearway has been adopted, extending 30 m beyond the airstrip running length with a width of 80 m, to match the runway strip width, accounting for any locally raised topography;

- The apron is 100 m by 50 m, capable of comfortably parking two design aircraft. The taxiway, at its narrowest point, is 10.5 m wide with a horizontal turning radius of 24 m and a speed limit of 20 km/h when following the centreline. The taxiway has a 2.0% crossfall, and both the taxiway and apron have a longitudinal slope of 0.75%;
- The aircraft trafficked pavement design (runway, taxiway and apron) comprises two pavement layers constructed over a prepared subgrade. In order of foundation to surface, these can be summarised as follows:
  - Subgrade - scarified and moisture conditioned in-situ material to a 200 mm depth, compacted to achieve a California Bearing Ratio (CBR) of at least 15%. The subgrade material is expected to comprise in-situ material within the airstrip foundation, or material placed as part of the local cut/fill operations during airstrip construction;
  - Base course - 150 mm well-graded sandy gravel material compacted to achieve a CBR of at least 35%. Base course material is expected to comprise in-situ material in some areas along the airstrip, supplemented by material sourced from local borrow areas where necessary;
  - Wearing course - 150 mm fine grained gravelly material compacted to achieve a CBR of at least 35%. Wearing course material is expected to comprise primarily material sourced from local borrow areas, external to the airstrip location;
- Site investigation of the proposed airstrip was completed during the feasibility study. The investigation indicated topsoil thickness varying from 100 mm to 400 mm with an average thickness of 200 mm. Gravel, sandy, dense to very dense soils were observed beneath the topsoil for a majority of the test pits. Based on the investigation, the in situ material is generally expected to be suitable as bulk fill, subgrade and base course material for airstrip construction, subject to laboratory test results. The higher specification wearing course for the runway sheeting may need to be sourced from an external borrow source;
- External to the trafficked areas, the in-situ material will be graded to line and level, and proof rolled;
- The airstrip has been located to reduce external runoff across the airstrip alignment. The preliminary drainage concept includes diverting surface water on both sides of the airstrip and discharging it into natural drainage courses via v-drains along the runway strip and clearways. Runoff from the airstrip will be collected in the v-drains and discharged through silt traps into a natural water course;
- Based on review of topographical data and current site infrastructure design, no interactions were identified with the Obstacle Limitation Surfaces (OLS) for the airstrip, according to aerial survey data.

Design criteria and parameters adopted for the design are summarised in Table 18.3.

Design Parameter	Value
Design Aircraft	Cessna Caravan
Aircraft Type	1B
Max. Passengers	14
Runway Length	≥ 819 m
Runway Width	≥ 18 m
Runway Longitudinal Slope	≤ 2.0%
Longitudinal Slope Change Between Segments	≤ 2.0%
Longitudinal Slope Change	≤ 0.4% per 30 m
Longitudinal Slope Radius of Curvature	≥ 7,500 m
Runway Transverse Slope	≤ 2.0%
Runway Shoulders	Not required
Runway Strip Length	≥ 30 m
Runway Strip Width:	
- Graded Area	60 m
- Incl. Flyover Area	90 m
Runway Strip Longitudinal Slope	≤ 2.0% per 30 m
Runway Strip Transverse Slope	≤ 3.0%
Runway End Safety Area Length	Not required
Clearway	Not required
Wheel Distance to Runway Edge (taxiing)	≥ 1.5 m
Taxiway Width	≥ 7.5 m
Taxiway Strip Width:	
- Graded Area	≥ 20.5 m
- Cleared Area	≥ 40.0 m
Taxiway Longitudinal Slope	≤ 3.0%
Taxiway Transverse Slope Change	≤ 1.0% per 25 m (min. radius of curvature 2,500 m)
Taxiway Transverse Slope	≤ 2.0%
Turning Radius (turnaround)	≥ 4.59 m
Taxiway Turning Radius (≤ 20 km/h)	≥ 24.0 m
Runway Pavement Type	Unsealed

Table 18.3 Site Airstrip Design Parameters

18.3 Logistics

A route survey from Abidjan to the Project site was undertaken in March 2024.

The transport route will be 663 km in total, consisting of 607 km via the existing A1 National Highway from the Port of Abidjan - through Abengourou, Bondoukou and Bouna - to the village of Latourgo, and then via the proposed access road to the Project site.

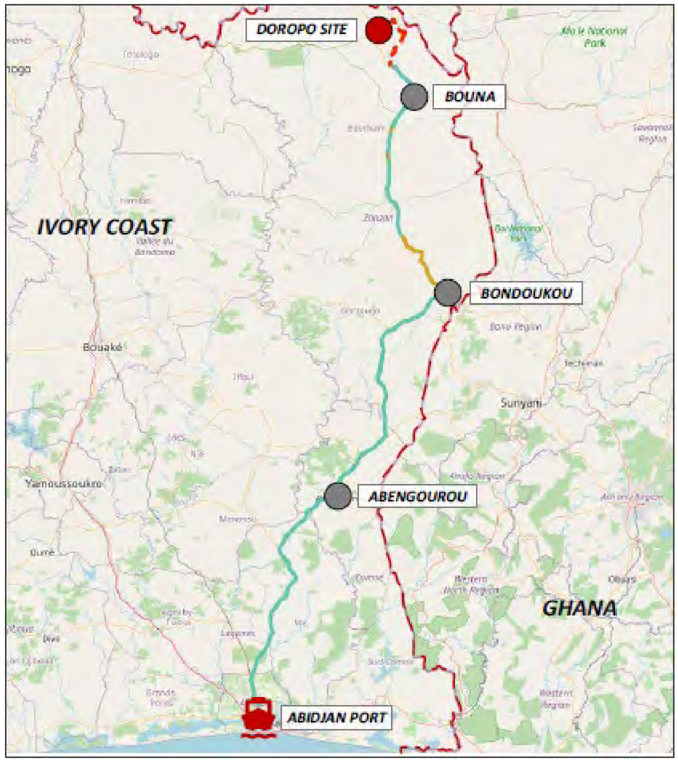


Figure 18.4 Transport Route

The survey identified a maximum transport envelope of 7 m wide by 6.1 m high (from ground), from the Port of Abidjan to the Project site.

The envelope height does not take into account electrical or telephone cables, which will need to be either lifted or disconnected - with the approval of local authorities - during the transport of cargo.

18.3.1 Construction Logistics

Materials and equipment will be transported to site using trucks via the route described above.

Local construction personnel will be transported between nearby villages and site, using buses and light vehicles. Expatriate and non-local construction personnel will initially be transported between Abidjan and site using light vehicles. Once the airstrip is established, these personnel will be transported via charter aircraft.

18.3.2 Operational Logistics

Diesel, process reagents, operating consumables and maintenance supplies will be transported to the site using trucks. Plant reagents will be shipped from the Port of Abidjan as containerised freight containing palleted, bulk bagged or liquid IBC reagents. There will be two notable exceptions, with the lime and cyanide to be delivered to site in tankers.

The lime will be trucked in bulk tankers that pneumatically transfer lime to the bulk lime silo, while the cyanide will be delivered in specialty ISOtankers from the Orica facility in Tarkwa, Ghana. After the cyanide is sparged at the stie, the ISOtanks will be returned, refilled, and resent in a cyclical loop.

Local operations and maintenance personnel will be transported between nearby villages and the site using buses and light vehicles. Expatriate and non-local operations and maintenance personnel will be transported between Abidjan and the site using charter aircraft.

Gold doré product will be transported from site via charter aircraft.

18.3.3 Non-Mining Mobile Equipment

A series of buses will transport personnel to/from the mine site, accommodation and security camps and to/from local villages. A fleet of 50 light vehicles will supplement the bus transportation and allow Centamin management and call-out personnel to travel between locations. The light vehicle fleet includes an ambulance for patient and first responder transport.

In addition to the light vehicle fleet a mobile equipment fleet services the roads, process plant and warehouse. The mobile fleet provides access or lifting for maintenance and/or operational needs and facilitates the management of spares. This mobile fleet supports the project's needs, outside of those provided by the mining fleet. The proposed mobile fleet includes:

- A grader;
- A wheel loader;
- Two backhoe loaders;
- Two skid-steer loaders;
- A telehandler;
- A 4t telehandler;
- A 45 t crane;
- A 100 t rough terrain crane;
- A rough terrain forklift;
- Two water trucks;
- A fire truck;
- A scissor lift;

- A bobcat;
- A forklift;
- A 3 t tip truck;
- A 8 t 4WD hiab truck;
- A container forklift, and
- A mobile diesel pump.

The ROM pad Front End Loader (FEL) and ROM rock breaker will be supplied by the Mining Contractor as part of the mining fleet.

18.4 Geotechnical Considerations

18.4.1 Geotechnical Investigations

A detailed geotechnical investigation including cored drill holes, in situ testing, test pitting and laboratory testing of undisturbed and disturbed representative samples was conducted during the feasibility design phase, with the following objectives:

- Assess the suitability of each proposed infrastructure location;
- Determining ground conditions at each infrastructure location to the required depth to achieve objectives, including field logging and laboratory testing;
- Assess the excavatability in and around infrastructure locations;
- Develop an understanding of variability in ground conditions in each area;
- Estimate the basin permeability within the WSD and WHD;
- Assess ground aggressivity towards concrete in critical areas by chemical testing (pH, Cl, SO_4);
- Assess the suitability of in situ materials for earthworks construction and allow mapping of potential borrow areas;
- Provide design parameters for foundation and earthworks design, including settlement analysis of settlement-critical process plant structures;
- Locate suitable sources of low permeability fill material, rip rap material, decant rock material and drainage media for Stage 1 construction;
- Measure groundwater elevations and fluctuations (via installation of piezometers in boreholes) in infrastructure areas;
- Allow estimation of topsoil depth and volumes to be stripped during construction.

Site investigation commenced on 5 December 2023 under the supervision of a geotechnical engineer from Knight Piesold (Knight Piésold) Perth. The site investigation and geotechnical logging were carried out in accordance with the Australian Standard for Geotechnical Site Investigations (Ref. 9), where practicable. A total of 36 boreholes and 166 test pits were sampled and inspected. A summary of the geotechnical investigation is presented in Table 18.4.

Infrastructure	Boreholes	Test Pits
Tailings Storage Facility	8	40
Plant Site	15	15
Water Storage Dam	2	20
Water Harvest Dam	1	12
WSD Diversion Channel	2	6
Han Diversion Channel	1	6
Airstrip	-	15
Accommodation Village	-	6
Site Access and Haul Access Roads	1	41
Access Road Overpass	4	4
Construction Materials	2	1
Total	36	166

Table 18.4 Geotechnical Investigation Summary

Representative samples were collected from the test pits and boreholes (disturbed and undisturbed samples) for laboratory testing. The laboratory testing was undertaken to classify and characterise the in-situ materials to assess their behaviour characteristics under embankment and foundation loading, and suitability for use in earthworks.

18.4.2 Investigation Findings

The results of the geotechnical investigations in each infrastructure area are summarised below.

Tailings Storage Facility

Sub-surface Conditions

The typical subsurface ground profile along the embankment footprint is summarised below:

- Topsoil with roots and other organic matter observed to depths of 300 to 500 mm within the TSF embankment footprint. Topsoil was generally recovered as silty sand, very loose to loose, with medium to coarse sand particles;
- Colluvium; gravel, clayey, medium dense to dense, brown to red-brown, slightly moist, medium plasticity fines, cemented laterite observed to depths of 1.35 to 3.0 m from the surface;
- Residual soil; clay, sandy, with gravel, medium plasticity, very stiff to hard, observed to depths of 2.4 to 7.5 m;
- Granodiorite, medium to coarse-grained, igneous with strength increasing and weathering decreasing with depth until the end of the borehole.

Standard Penetration Test refusals were encountered at shallow depths in the majority of boreholes.

The following is a summary of logged ground conditions within the TSF basin:

- The majority of the topsoil with biologically active layer was observed for the first 300 to 500 mm across the site. Topsoil was generally recovered as sand, clayey and silty, pale to dark grey very loose to loose, with medium to coarse sand particles;
- Alluvium/colluvium recovered as sand and gravel, clayey and sandy, medium dense, cemented observed to depths of 0.7 to 3.0 m from the surface;
- Residual soil recovered as clay, sandy, medium plasticity, red to orange-brown, stiff to hard, and sand, clayey, medium plasticity fines, medium dense to dense observed to depths of 1.4 to 4.3 m.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction:

- The TSF basin areas comprise extensive areas of sand and gravel directly underlying the topsoil horizon. It is likely that these materials have relatively high permeability and therefore a compacted soil liner will be required as a minimum to reduce the rate of seepage through the basin floor;
- Cut-off trenches for the TSF should typically terminate in a low permeability horizon. At the embankment, the cut-off trench will need to extend through the near surface sand/gravel horizon and is likely to vary with location with an average of 2 m depth into low permeability clay layers along the embankment alignment;
- Numerous artisanal pits were observed within the south and east TSF embankment and basin. A comprehensive survey is required to locate all potential artisanal pits, as they must be properly backfilled before construction commences;
- In-situ falling head tests indicated the presence of relatively high permeable soil layers, likely a function of the shallow permeable laterite. To limit seepage, it is recommended to use a soil liner along with an HDPE liner;
- A pedestrian footpath for the community is proposed between the plant site and the TSF. The footpath shall be fenced, to ensure personnel and livestock are kept away from the TSF and related structures at all times.

The construction sequencing for the TSF includes construction of the WSD and Stage 1 TSF embankment adjacent to each other. The space between these two embankments will be backfilled during the Stage 2 TSF raise, with the WSD downstream of the TSF western embankment. It is expected that the space between these two embankments will collect water during the wet season during Stage 1. Adequate dewatering measures must be put in place to keep the toes of the embankments reasonably dry, ensuring embankment stability and facilitating the construction of Stage 2.

Water Storage Dam

Sub-surface Conditions

The general stratigraphy within the WSD embankment is as follows:

- Topsoil; sand, dark brown, dry, with medium grained sand particles medium grained and sub angular to a depth of 0.3 m;
- Alluvium; clay, sandy, medium plasticity, very stiff to depths of 3.5 m followed by sand, with clay, low plasticity fines, very dense to depths of 6.0 m;
- Granodiorite, medium to coarse grained, porphyritic, massive, pale grey with strength increasing with depth and weathering decreasing with depth.

The general soil stratigraphy within the basin is as follows:

- Topsoil; sand, silty, trace gravel, non-plastic, dark brown, very loose observed across the WSD embankment and basin footprint for depths ranging from 0.2 to 0.4 m;
- Alluvium; sand, with gravel, trace clay, low plasticity fines, medium dense to dense; cemented laterite, recovered as gravel, sandy, trace clay, low plasticity fines, dense and clay, sandy, with gravel, medium plasticity, very stiff to hard between 0.2 to 2.9 m depths;
- Sand, clayey, medium to high plasticity, dense to very dense observed between 0.8 to 4 m across the WSD basin;
- Granodiorite, medium to coarse-grained, porphyritic, massive, pale grey with strength increasing with depth and weathering decreasing with depth.

It is noted that the sand and gravel layer underlies the topsoil across much of the WSD basin area. It is likely that the permeability of near surface basin soil will be relatively high.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction.

- The WSD basin areas comprise extensive areas of clayey sand and gravel directly underlying the topsoil horizon. It is likely that these soils have relatively high permeability;
- Cut-off trenches for the WSD should typically terminate in a low permeability horizon. At the embankment, the cut-off trench will need to extend through the near surface sand clayey/clay sandy horizon and is likely to vary by location, with an average depth of more than 2 m along the embankment alignment;
- Topsoil is present to depths between 200 and 400 mm across the footprints of the WSD embankment, with some roots penetrating deeper;

- No artisanal mining shafts were observed within the embankment footprint and the WSD basin. Prior to the start of construction, a reconnaissance will need to be conducted to locate shafts within the footprint and immediate vicinity;
- Dewatering and water diversion measures may be required prior to the start of construction of the WSD embankment depending on timing. The preferred time to start construction in the WSD area is at the end of the wet season.

Processing Plant Site

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Topsoil with roots and other organic matter to varying depths from the existing ground level, generally recovered as gravel and sand, silty, pale to dark grey and brown, non-plastic, dry, loose to medium dense, ranging from 0.1 to 0.5 m, with an average topsoil depth of 0.25 m;
- Laterite soil, comprising predominantly clay, gravelly, with sand, medium to high plasticity, very stiff to hard and gravel, clayey, with sand, low to medium plastic fines, very dense, cemented and gravel, clayey, medium plasticity fines, dense to very dense observed to depths from 0.6 to 4.1 m from the ground surface to 3.0 to 20.0 m, with an average laterite depth of ~7 m;
- Saprolite, comprising predominantly clay, sandy, trace gravel, stiff to very hard, medium plasticity, at depths from 3.0 to 28.7 m;
- Extremely weathered granodiorite, recovered generally as sand, clayey, trace gravel, observed from depths varying between 14 to 42.5 m;
- Highly to moderately weathered granodiorite, medium to coarse grained, porphyritic, massive, pale grey, low strength to very high strength at depths from 18.4 to 46.9 m;
- Fresh granodiorite, medium to coarse-grained, porphyritic, massive, pale grey, high to very high strength from 25.0 to 48.5 m.

Settlement analyses were undertaken in order to calculate settlements of the major process plant foundations. The key outcomes of the settlement analyses are summarised below:

- An iterative design approach encompassing adjustments to the foundation dimensions, foundation level, loading conditions, settlement requirements in collaboration with Centamin, GRES and Knight Piésold, will be required during the next stage of design;
- The estimated settlements will be adjusted based on the actual dead load, and the analyses will be reviewed during the next stage of design;

- The performance of the crusher foundation depends on the ground conditions, loading conditions from the primary crusher, and the influence of the ROM pad. The settlement estimates for the primary crusher do not account for the impact of the ROM pad, assuming that the primary crusher vault and equipment will be installed after completing ROM pad construction. The load distribution is complex and detailed 3D numerical finite element modelling (FEM) may be required during the next design stage;
- The construction sequences of the ROM pad, retaining wall, and crusher vault will significantly impact on the settlement of the crusher. If the surge bin and related steel works are installed after completion of the ROM pad, the post construction settlement of the bin will be smaller than estimated since the settlements due to the concrete dead load and pressure from the ROM pad will occur before the installation. Provision should be made to adjust the connection pins of the surge bin during installation to account for assumptions and uncertainties during design;
- Relatively large settlements are expected for structures (rock breaker etc.) founded on top of the ROM pad unless the fill used for constructing the ROM pad is structural fill compacted to 95% Modified Maximum Dry Density (MMDD);
- As the ore stockpile, tunnel and trestle foundation are not in plane strain conditions, the deformation and load distribution on the foundations will be highly complex. A detailed 3D FEM analysis needs to be undertaken to assess the stability of the trestle foundation and tunnel deformation . To ensure tunnel stability, the hoop stress condition should be maintained at all times, including during construction and operation. In addition to settlement and bearing capacity, the trestle foundation should also be analysed to ensure sufficient passive resistance against tilt from the ore stockpile, which will be addressed during the detailed design stage;
- The settlement analyses for CIL tanks were based on simplified elastic methods. Once the loading details and construction sequences are finalised, detailed finite element analysis may be conducted to estimate the likely total and differential settlements of the tanks if deemed critical;
- Pre-leach thickener is designed to be founded on pedestal footings (to be confirmed in detailed design). Settlement will increase if the foundation width expands. Given the presence of cemented ferricrete to shallow depths across the plant site, it is recommended to limit the foundation size to less than 3 m to reduce the foundation zone of influence extending beyond the ferricrete layer.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction:

- The terrain is generally flat with cemented laterite observed at shallow depths from the surface. Excavation difficulties may be encountered if low strength machinery (30 t and below) are used during construction. Previous experience suggests that lower-strength ferricrete should be excavatable using a D10 dozer or equivalent equipped with a single-tyne ripper attachment;

- To reduce foundation settlements, it is advisable to maintain the foundation at or below the natural ground level and restrict the use of fill material above the natural ground, unless compacted select fill material is employed. Additionally, cemented ferricrete is observed to shallow depths from the surface, with compressible soil observed to deeper depths. Foundation settlement will increase if large foundations are adopted with the zone of influence extending beyond the ferricrete layer;
- Proper drains should be installed to direct water away from the foundation. Some shrinkage and swelling of the finer grained near surface soils are possible with seasonal changes in moisture content. To limit seasonal shrinkage and swelling, settlement-sensitive foundations should be founded at least 0.5 to 1 m below the final formation level. A layer of low permeability clay/silt should be placed around footings to prevent moisture change. Comprehensive surface water management/drainage should also be provided to direct water away from the foundations.

Water Harvest Dam

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Thick topsoil with roots and other organic matter observed to an average depth of ~0.5 m, with thickness ranging from 0.2 to 0.7 m. A greater thickness of topsoil is possible due to the increased alluvium deposition and organic growth directly under the river channel basin;
- The soil horizon underlying the topsoil comprises clay with sand, medium to high plasticity, yellow and brown, very stiff to hard for depths between 0.2 to 3.7 m, and sand with silt and gravel, non-plastic to low plasticity fines, dense for depths between 0.4 to 3.4 m;
- Granodiorite, recovered as sand and gravel, dense to very dense, weathering decreasing with depth and strength increasing with depth, was observed.

Potential Geotechnical Risks and Construction Considerations

A catchment of 43,000 ha reports to the proposed WHD. The embankment is designed to serve as an overflow spillway. Due to the large catchment area and rainfall, overflow is likely to occur annually. Maintenance will be required every year after the wet season.

WSD Diversion Channel

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Topsoil; sand, silty, non-plastic fines, pale grey, very loose, with depths ranging from 0.1 to 0.6 m, with an average depth of 0.3 m;

- Cemented laterite; gravel, clayey, with sand, high plasticity fines, brown to orange-brown, dense to very dense, cemented, with depths of 1.2 to 2.9 m from the surface;
- Saprolite; clay, sandy, trace gravel, high plasticity, very stiff to hard, at depths of 2.8 to 4.5 m;
- Residual soil; clay, sandy, medium plasticity, cream and orange-brown, stiff to hard, at depths of 13.5 to 15.0 m;
- Extremely weathered granodiorite, recovered as clay, sandy, medium plasticity, slightly moist, very stiff to hard, at depths of 16.5 to 18.0 m;
- Granodiorite, massive, igneous, yellow-brown with strength increasing and weathering decreasing with depth, continuing until the end of the borehole.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction.

- Prior to construction, a reconnaissance should be undertaken to identify any artisanal shafts present to ensure the integrity of the channel and the adjacent TSF embankment;
- Investigations identified cemented laterite to a depth of ~3 m from the surface, containing small voids typical of cemented laterite. The material was recovered as gravel, clayey with sand, with the possibility of relatively high permeability;
- Firm to stiff clay was identified at depths ranging from 4.5 to 14 m. A detailed slope stability assessment is recommended during the next design stage to ensure the excavation has the required factor of safety against failure;
- Depending on abstraction from the WHD and the water level within the WSD, the western diversion channel is expected to flow during the peak wet season. To limit excavation, a gentle gradient was adopted for a significant portion of the channel length. A HDPE liner is proposed to reduce erosion, as well as maintain a laminar flow;
- The diversion channel is critical for ensuring that the WSD flood level complies with the Centamin ESIA guidelines and is outside the inundation extent for settlements/villages adjacent to the WSD. Maintenance of the diversion channel should be undertaken prior to start of the wet season every year.

Han Diversion Channel

The Han pit is along the basin of the river channel. The Han diversion channels are proposed to divert water from the southwest and southeast catchments to the proposed Han pit (refer Figure 18.1 and Figure 18.2).

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Topsoil; sand, silty, non-plastic fines, pale grey, very loose, ranging from 0.1 to 0.4 m, with an average depth of 0.3 m;
- Cemented laterite; gravel, clayey, with sand, high plasticity fines, brown to orange-brown, dense to very dense, cemented, and sand with trace silt, non-plastic, red-brown, slightly moist, dense at depths of 1.2 to 2.9 m from the surface;
- Saprolite; clay, sandy, trace gravel, high plasticity, very stiff to hard, at depths ranging from 0.8 to 3.7 m in some of the test pits;
- Granodiorite, massive, igneous, yellow-brown with increasing strength and decreasing weathering with depth until the end of the borehole.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction:

- Han pit is proposed along the basin of the river channel. Due to the large catchment area and the proposed pit along the existing channel, the design of DC11 is challenging. It is envisaged that DC11 will be integrated into the pit excavation surface, necessitating close collaboration with pit designers during the detailed design phase. Moreover, the presence of existing Han Bridge crossing and the nearby community impose significant constraints on the development of the diversion channel;
- Depending on the feasibility, it will be worth looking into the option of mining Han pit during the dry season alone. This will help reduce the earthworks volume required for the construction of the diversion channel;
- Management plans should be in place to ensure that personnel are evacuated from the pits during significant storms. This will require continuous monitoring of the forecasted rainfall to minimise the risk of personnel working in the pit during flooding events.

Airstrip

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Thick topsoil with roots and other organic matter to depths of ~150 to 400 mm with an average depth of 200 mm. Topsoil; gravelly, sandy, with silt, non-plastic fines, dry medium dense and silt, sandy, with trace gravel, non-plastic, soft to firm;

- Beneath the topsoil, a soil horizon comprising gravelly, sandy, with clay, high plasticity fines, red-brown, dense to very dense, cemented laterite was observed at depths of 0.4 to 4.2 m from the existing ground surface.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction:

- To meet the design requirements of the runway based on CASA/ICAO guidelines, cut and fill operations will be required. Permanent soil slopes must be adequately drained and rainfall run-off controlled to mitigate erosion and degradation of the slopes. The sides of excavations shall be benched and battered as necessary to ensure safety against sliding, cave-ins or danger to persons or structures;
- V-drains or culverts shall be constructed on the side of the runway strip and clearway to divert and discharge the surface runoff;
- Conventional earthmoving equipment, such as large dozers and excavators should be capable of excavating the near-surface materials across the site;
- Care must be taken to reduce the washing of fines from the airstrip running surface due to rain or water flow post-construction. Regular maintenance will be required. The maintenance frequency should be adjusted according to in situ conditions and the performance of the runway and airstrip. The maintenance frequency is expected to be higher during the first few years of use and to decrease over time. Routine maintenance will primarily consist of grading and watering. Occasional re-sheeting of the surface layers may be necessary during the pavement life.

Accommodation Village

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Thick topsoil with roots and other organic matter was observed at depths of ~0.2 to 0.5 m from the existing ground surface, with an average depth of 0.3 m across the footprint;
- Beneath the topsoil, a soil horizon comprising gravel, sandy, with clay, low plasticity fines, medium dense to very dense was observed at depths of 0.6 to 4.4 m. Of the 6 No. test pits, 5 No. test pits refused at shallow depth due to the presence of cemented laterite.

Potential Geotechnical Risks and Construction Considerations

Given the generally flat terrain with elevations ranging from RL 342 to 348 m, cut and fill operations are expected to be relatively small. Conventional earthmoving equipment, such as large dozers and excavators, should be sufficient for excavating the near-surface materials across the site. However, excavation difficulties may arise as the depth increases.

Access Roads

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Topsoil with roots and other organic matter was observed to an average depth of 300 mm from the surface. Depending on the location, the soil was silt, sandy, non-plastic, firm; gravel, sandy, with silt, non-plastic fines, loose to medium dense and sand, silty with trace gravel, non-plastic fines, very loose to loose;
- Beneath the topsoil, the soil horizon comprises residual soil recovered as clay, sandy and gravelly, medium to high plasticity, slightly moist to dry, stiff to hard; cemented laterite recovered as gravel, clayey, with sand, medium plasticity fines, medium dense to very dense and sand, gravelly, trace silt and clay, low to medium plasticity, medium dense to very dense;
- Beneath the layer, granodiorite recovered as gravel, sandy, trace silt, low plasticity fines, yellow-brown, very dense, extremely weathered rock. With depth, weathering decreases and strength increases.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction.

- The site access road runs across difficult terrain consisting of valleys, ridges and river crossings. As the terrain is rugged, cut and fill operations are expected during construction. Conventional earthmoving equipment, such as large dozers and excavators should be capable of excavating the near-surface materials across the site. However, there is a potential for excavation difficulties within the cemented ferricrete layer as well as when the excavation depth increases;
- Roadside drainage is required to divert surface run-off away from the road and embankment. The side drains should comprise a V-ditch configuration with 1V:2H side slopes;
- Where the subgrade is recovered as clay, sandy and gravelly, a proper base course comprising predominantly gravel with a CBR > 40% is recommended. Cemented ferricrete recovered as gravel could serve as basecourse subject to laboratory testing;
- A laterite gravel wearing course could be placed over the subgrade/general fill during the initial stage. Once the site quarry or mining operations have commenced, the wearing course should be reinforced with material from the quarry or mining operations. Routine maintenance and resurfacing of the access road will need to be carried out during operation;

- Culvert crossings will be required at significant stream crossings to convey runoff resulting from a 10 year ARI. Culverts typically comprise either corrugated metal pipe structures or precast concrete box culverts, depending on availability and cost. In the event of flood exceeding the 10 year ARI, the culverts would be unable to convey the flow, leading to overflow which could potentially cause significant damage to the roads. Damage to the roads could result in access to certain areas of the project being lost until repairs can be undertaken;
- It is expected that the existing local roads may intersect the proposed haul access roads at certain locations. Enhanced security and safety measures may be needed to protect personnel and livestock.

Access Road Overpass

Sub-surface Conditions

The typical subsurface ground profile is summarised below:

- Topsoil with roots and other organic matter observed at 300 to 400 mm from the existing ground surface. Topsoil; Sand, silty, trace clay, non-plastic, medium dense;
- Residual soil; Sand, clayey, with gravel, low plasticity, loose to medium dense observed at depths of up to 6 m;
- Granodiorite, medium to coarse-grained, granular, massive, grey-brown, very low to low strength, highly weathered to moderately weathered rock until the end of the borehole.

Potential Geotechnical Risks and Construction Considerations

The following are to be considered during construction.

- To limit settlements and increase foundation bearing capacity, ground improvement comprising excavation of loose sand and replace with compacted structural fill is expected. The structural fill shall be crushed mine waste with fines content less than 20% and a maximum particle size less than 75 mm. The structural fill shall be compacted in layers less than 300 mm thick and achieve a 95% modified maximum dry density. Depending on the particle size of the mine waste, or borrow sources if any nearby, crushing and screening may be required to limit the particle size within the required range;
- During construction, a temporary diversion road will need to be constructed alongside the main A1 highway to redirect traffic. This will require careful traffic management plans to ensure compliance with regulatory requirements, given that the A1 is a national highway. Existing underground utilities such as water, gas, and sewer lines may conflict with the tunnel alignment, requiring careful relocation or protection;

- The integrity of the tunnel relies on the principle of maintaining a compressive hoop stress condition around the circumference of the tunnel lining at all times. This should be maintained throughout construction. If the hoop stress condition is not maintained, the tunnel lining may experience excessive deformation and overstressing, or even collapse. Proper sequencing of excavation, backfilling, support installation, and lining placement is crucial to avoid excessive deformation that could disrupt the hoop stress condition;
- The design of the overpass is based on haul truck Volvo FMX460 8X4 tipper nominated by Centamin. If the nominated truck changes or other higher loaded vehicles are anticipated, the analysis should be reviewed.

Construction Materials

The site investigation suggests that in potential areas of cut the excavated material will comprise colluvial and alluvial soils, cemented laterite and weathered rock recovered as gravels. Depending on the depth of excavation these horizons are likely to excavate as a combination of gravel, silt, clay, and clayey silt, and will provide a suitable common fill for constructing the lower levels of bulk earthworks fill platforms and embankments. Of the soils likely to be encountered during bulk excavation, most of the coarse-grained materials will be suitable for embankment Zone C construction, whilst finer-grained soils with selective excavation will be suitable for embankment Zone A (refer to Figure 18.7).

If excavations extend into the moderately weathered rock horizon the resulting materials are likely to be coarse, but the end product will depend upon the design and implementation of the blasting programme (if any), as well as the pre-existing rock strength, weathering and joint spacing.

It is expected that substantial quantities of borrow material can be sourced from the open pit pre-strip, subject to laboratory testing. The zones sourced from pre-stripping depend on the material type (oxide, transition, fresh). Oxide material closer to the surface could be designated as Zone A subject to lab testing. Zones C and D are typically sourced during pre-stripping. If the size of the material from pre-stripping for Zone C is too large, increasing the powder factor during blasting often helps without much additional cost. Depending on the properties, oxide material may also be used as Zone C. Zones E and G will consist of selected materials from waste rock, while Zone F (drainage medium) shall be sourced from basins of WSD, WHD, Han and select locations of the site access road. Site investigation has identified potential borrow sources for Zone A, E, F, G, road basecourse and construction aggregates, subject to laboratory test findings.

18.5 Tailings Storage Facility

18.5.1 TSF Design

The TSF will comprise a side hill storage facility formed by multi-zoned earth fill embankments (refer Figure 18.7), comprising a total footprint area (including the basin area) of approximately 165 ha for the Stage 1 TSF, increasing to 287 ha for the final TSF. The TSF is designed to accommodate a total of 41.1 Mt of tailings (a slightly higher volume than the minimum LOM requirements of 38.2 Mt). Further expansion of the TSF is possible however, it should be evaluated if required. A general arrangement for the TSF Stage 1 is shown in Figure 18.5, and the final stage is shown in Figure 18.6.

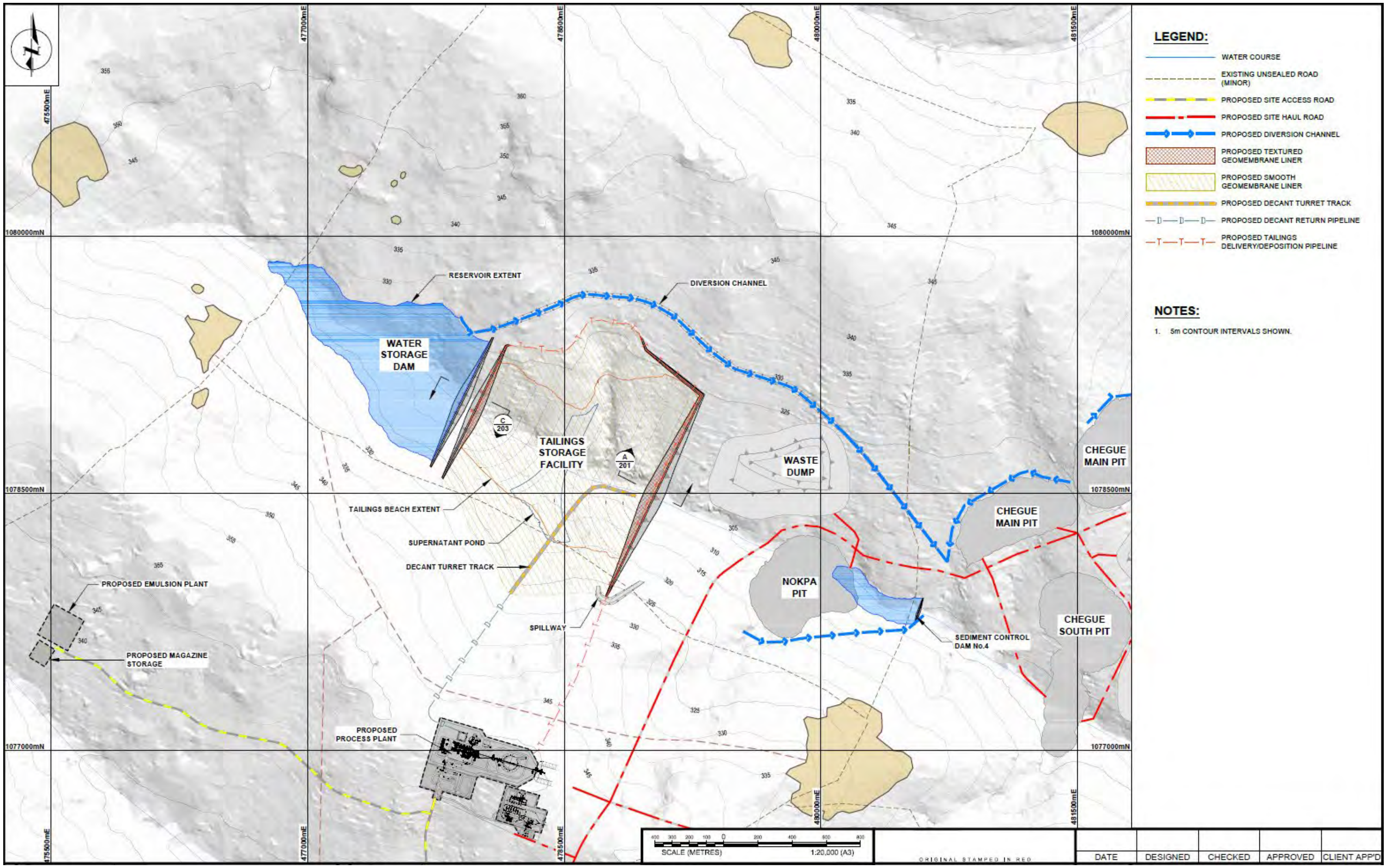


Figure 18.5 TSF General Arrangement - Stage 1 (Source: Knight Piésold, 2024)

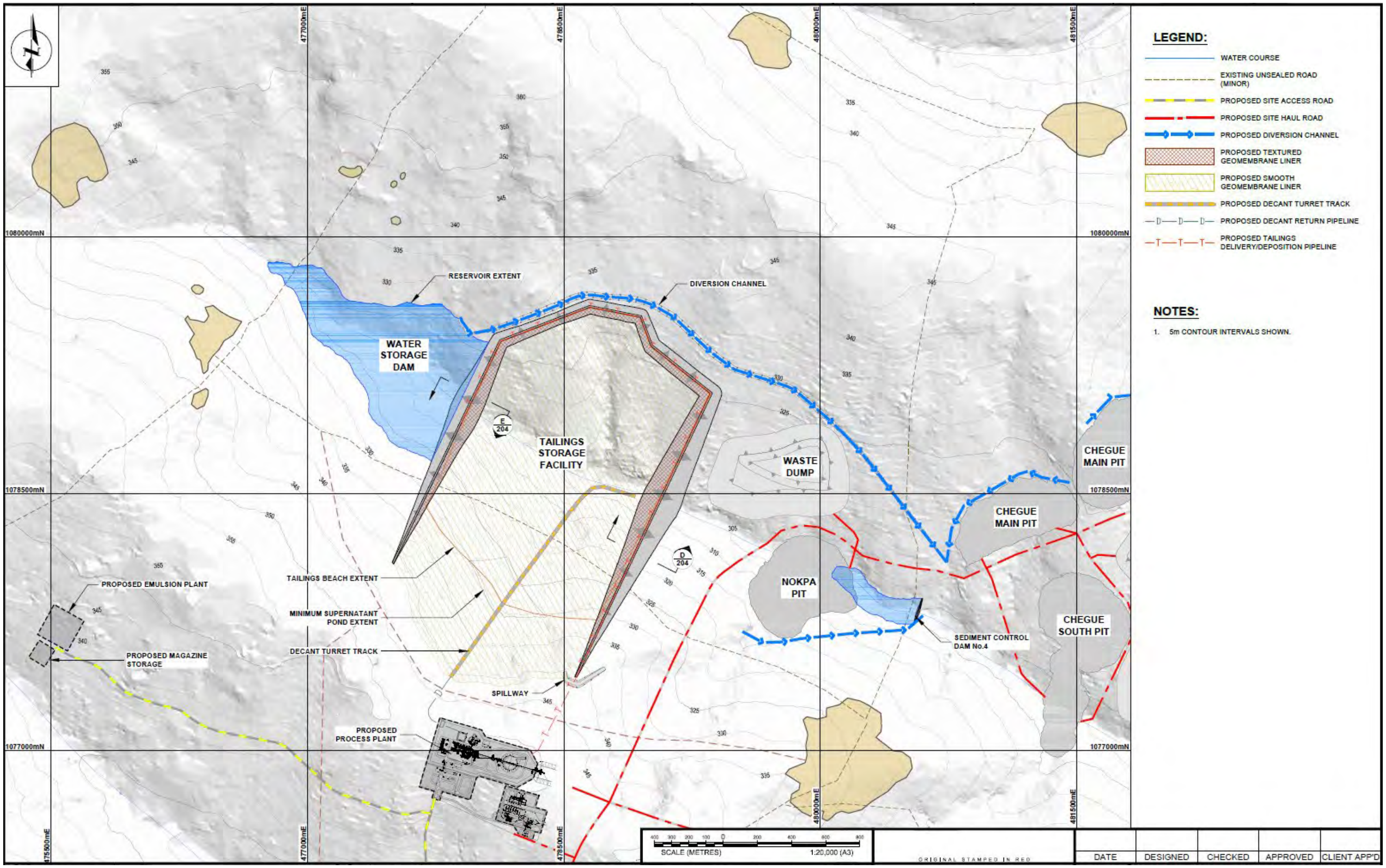


Figure 18.6 TSF General Arrangement - Final Stage (Source: Knight Piésold, 2024)

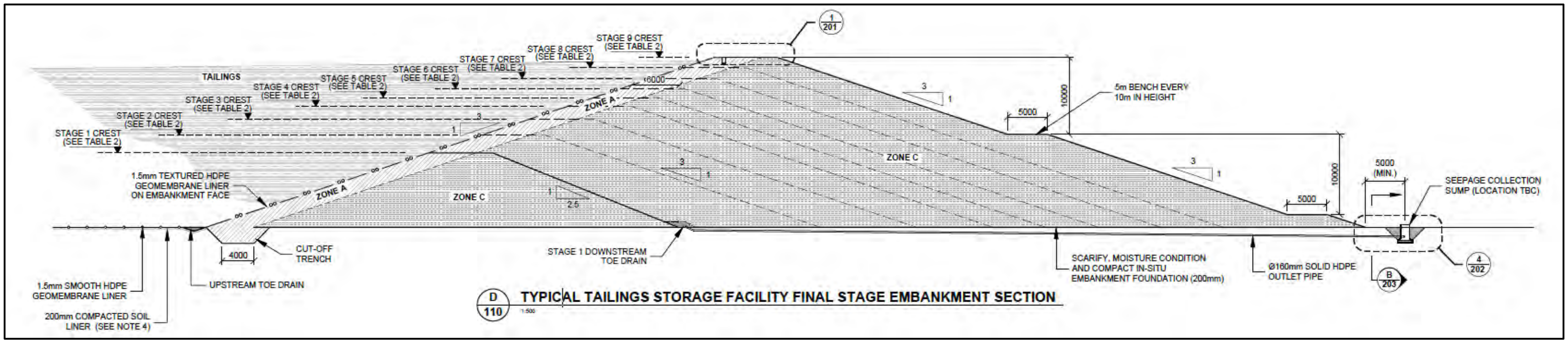


Figure 18.7 Final Stage TSF Embankment - Typical Section (Source: Knight Piésold, 2024)

The design criteria and parameters adopted for the TSF design are summarised in Table 18.5.

Design Parameter	Value
Design Standards	
TSF Consequence Category	High B
Dam Spill Consequence Category	Significant
Design Storage Allowance Parameters - Design Storage Allowance - Extreme Storm Storage	 - 1:10 AEP notional wet season runoff - 1:100 AEP 72 hour storm
TSF Emergency Spillway: - Spillway Capacity - Wave Freeboard Allowance	 1:100,000 AEP 1:10 AEP wind
TSF Closure Spillway: - Spillway capacity - Erosion protection	 1:100,000 AEP Storm Event + 1:10 AEP wind wave runup 1:100 AEP
Contingency Freeboard - Wave Run-up - Additional Freeboard	 1:10 AEP wind 0.3 m
Earthquake Loading - Operating - Final	 Operating Basis Earthquake (OBE) - 1,000 year ARI earthquake Safety Evaluation Earthquake (SEE) - 5,000 year ARI earthquake
Embankment Stability Criteria	
Stability Factors of Safety - Long-term undrained - Short-term undrained: - potential loss of containment - no potential loss of containment - Post-seismic	 1.5 1.5 1.3 1.0 - 1.2
Operations	
Capacity - Final - Starter	41 Mt of dry tails 5.8 Mt of dry tails - 18 months initial capacity
Production Rate	3.2 - 6.0 Mtpa.
Slurry Characteristics -Target% Solids -Beach Slope*1 -Density*1 -Stage 1 - Final	 50% solids by weight 100H:1V 0.73 - 1.25 t/m^3 (assumed, to be confirmed after density modelling) 1.40 t/m^3 (assumed, to be confirmed after density modelling)

Design Parameter	Value
Fluid Management	Full basin underdrainage, gravity system into collection sumps. Return to supernatant pond via submersible pumps. Leakage collection and recovery system (LCRS) installed beneath the composite basin liner in the main drainage course within the TSF basin, discharging to a collection sump, and pumped to the supernatant pond. Floating decant turret system for removal of supernatant solution. Return to the plant via a trailer-mounted pump. Seepage collection system within the TSF embankment to capture and return seepage to the TSF pond.
Embankment	
Embankment design: - Crest Width - Upstream Slope - Downstream Slope (interim) Downstream Slope (final, overall)	 10 m 3H:1V 3H:1V 3.5H:1V
General	Side hill storage facility with supernatant pond situated against the southern hillside of the valley. Deposition from the TSF embankments will push the pond to the southern hillside of the valley within the basin area. Minimum total freeboard of 0.8 m plus design pond elevation below spillway level. Minimum tailings freeboard of 0.5 m to embankment crest.
Construction Description - Cut-off Trench - Embankment - Embankment Raises - Decant System	 Upstream toe cut-off through residual/transported material. Multi-zoned earth fill embankment, with upstream low permeability zone. Upstream face HDPE geomembrane lined. Downstream raise construction methods for all raises. Floating decant turret with trailer-mounted pump and structural fill access causeway.
Basin	
Basin Liner	Compacted soil liner comprising primarily of in situ soils, scarified and recompacted throughout basin area to form a 200 mm liner. Where in-situ materials are unsuitable for soil liner, low permeability material (Zone A) will be imported.
Leakage Collection and Recovery Systems (LCRS)	1.5 mm smooth HDPE geomembrane liner above compacted soil liner in the TSF basin.
Tailings Underdrainage System	Leakage collection and recovery system (LCRS) installed beneath composite basin liner. Drains excavated in alluvial sands with slotted pipe, backfilled with drainage medium (Zone F2) and capped with low permeability material (Zone A) below basin soil liner.

Design Parameter	Value
Materials	
Material Supply	Low permeability fill (Zone A) sourced by civil contractor from local borrow or selected mine waste. Structural fill (Zone C) sourced by civil contractor from local borrow or selected mine waste. Structural fill (Zone C1) sourced from mining operation, either pre-strip (Stage 1) or run of mine waste (Stage 2+). Random fill (Zone D) sourced by civil contractor from local excavations, borrow or selected mine waste. Rip rap for erosion protection (Zone E) sourced from mining operation, local borrow or off-site quarry. Drainage material (Zone F1 and Zone F2) sourced from local river beds, local borrow or off-site quarry. Coarse rockfill for decant surround (Zone G) sourced from mining operation, local borrow or off-site quarry. Wearing course material sourced by civil contractor from local borrow or selected mine waste.
Rehabilitation	
Final Embankment Slopes	3.5H:1V (overall), with 5 m horizontal benches at 10 m height increments
Cover Profile	Generally shaped to achieve dry closure with no ponding (water-shedding)
Capping	Mine waste capillary break (0.5 m), low permeability mine waste (nominal 0.3 m thickness), covered with topsoil (0.20 m), re-vegetated.

Notes: (1) - Tailings properties have been assumed based on experience on similar regional projects.
(2) AEP - Annual Exceedance Probability, ARI - Average Recurrence Interval

Table 18.5 TSF Design Parameters

The Stage 1 TSF is designed for 18 months of storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements however, this may be adjusted to biennial raises to suit mine scheduling during the operation. Downstream raise construction methods will be utilised for all TSF embankment raises. A downstream seepage collection system will be installed within and downstream of the TSF embankment to capture and return seepage from the TSF.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 200 mm thick compacted soil liner will be constructed in the TSF basin area, comprising either reworked in-situ material or imported low permeability material. The area within the TSF basin will be lined with a 1.5 mm HDPE geomembrane liner, overlying the compacted soil liner.

The TSF design incorporates an underdrainage system to reduce the pressure head acting on the basin liner system, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of collector and finger drains. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin. In addition, a leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner to reduce water pressure build-up on the basin liner. Solution recovered from the underdrainage system and LCRS will be pumped to the top of the tailings mass via a submersible pump, reporting to the supernatant pond.

Supernatant water will be removed from the TSF via a decant turret system. The supernatant pond will be maintained in the southern valley within the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

An operational emergency spillway will be available at all times during operation to protect the integrity of the constructed embankments in the event of an emergency overflow. The emergency spillway will initially be constructed to the south of the west embankment.

The closure spillway will be excavated south of the facility, at the low point on the final tailings beach, where it will safely discharge into the existing water course after operation ceases, thus ensuring the TSF becomes a fully water-shedding structure on closure.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals around the TSF embankment. Deposition extents are shown in Figure 18.6. During the early stages of operation, when the risk of water shortages is highest, the deposition plan will be managed to optimise water recovery.

A pipeline containment trench will be constructed during Stage 1 to contain both the tailings delivery pipeline and decant return pipeline between the TSF and Process Plant, mitigating environmental impact if the pipeline bursts.

Embankment Construction

The TSF embankment will be constructed in stages to suit storage requirements. It is envisaged that the embankment(s) will be raised annually by an earthworks contractor, with the downstream structural fill zone constructed by the mining fleet on an ongoing basis during operation. Staged embankment crest elevations are presented in Table 18.6.

Stage	Tailings Storage (Cumulative) (Mt)	Tailings Storage (Cumulative) (Mt)	Maximum TSF Embankment Height (m)	TSF Deposition Elevation*2 (m RL)
1*2	5.8	327.0	19.8	323.4
2	10.3	328.9	21.7	326.8
3	14.6	331.2	24.0	329.4
4	19.6	333.3	26.1	331.8
5	25.3	335.1	27.9	334.2
6	30.4	337.0	29.8	336.1
7	34.5	338.7	31.5	337.4
8	38.4	340.3	33.1	338.6
9	41.4	341.3	34.1	339.6

Notes: *(1) Includes freeboard requirements documented in Table 18.5.*
(2) Stage 1 embankment designed for 18-month storage capacity.

Table 18.6 Staged Embankment Construction

Staged embankment crest elevations were determined based on the stage storage curve for the facility. The embankment will have an upstream slope of 3H:1V (to facilitate HDPE geomembrane liner installation), an operating downstream slope of 3H:1V and a minimum crest width of 10 m. The downstream slope of 2.5H:1V and a crest width of 8 m is adopted for Stage 1. The final downstream embankment profile will consist of an overall slope of approximately 3.5H:1V, comprising a 3H:1V slope with 5 m benches at 10 m vertical intervals. The embankment upstream face will be lined with textured HDPE geomembrane liner to allow safe egress from the TSF (refer Figure 18.7).

The Stage 1 TSF is designed to provide 18 months of storage capacity, allowing flexibility for Stage 2 construction to be carried out during the subsequent dry season when conditions are more suitable for construction. Downstream raise construction methods will be utilised for all stages of embankment construction.

The TSF embankment comprises an upstream low permeability zone (Zone A), and a downstream structural fill zone (Zone C1).

Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow sources to form the low permeability zone of the embankments. The material will be moisture conditioned, spread and compacted in 300 mm (uncompacted) layers. The target permeability of the zone shall be less than 1 x 10^{-8} m/s after conditioning and compaction;

- Zone C1 material shall be delivered to the embankment by the mining operation, levelled with a dozer and traffic compacted by loaded haul trucks on an ongoing basis during the operation;
- The layer thickness will vary depending on the material types, with an indicative guide as follows:
  - Oxide waste material - 0.5 m to 1 m layers (uncompacted), paddock dumped;
  - Transitional waste material - 1 m to 2 m layers (uncompacted), paddock dumped;
  - Fresh waste material (coarse rockfill) - 2 m to 5 m layers (uncompacted), dumped from a tip head;
- When material from the Open Pit is not available, Zone C material shall be sourced from borrow. The material will be moisture conditioned, spread and compacted in 500 mm (uncompacted) layers;
- Zone F1 shall be clean sand/gravel drainage material supplied to a stockpile adjacent to the works. The Zone F1 material will be sourced from local borrow (if available) or off-site.

A cut off trench will be located beneath the upstream low permeability zone (Zone A) of the embankment. The cut-off trench will be excavated to extend through to competent low permeability foundation material. The cut-off trench will be constructed continuously along the embankment and backfilled with low permeability fill (Zone A). The excavated material will be placed in the embankment (as Zone A or Zone C), if suitable, or else hauled to waste.

Seepage Control

To reduce seepage losses through the TSF, increase water return to the process plant, and increase the settled densities of deposited tailings, a number of seepage control and underdrainage collection features were integrated into the design. The seepage control and underdrainage collection systems will consist of the components described below.

Cut Off Trench

Primary seepage control from the TSF will consist of a cut-off trench excavated into foundation soils to competent foundation material, backfilled with low permeability (Zone A) fill to reduce seepage losses through the embankment foundations. The cut-off trench will be extended to reach competent low permeability foundation material. It will be constructed directly below the upstream low permeability zone (Zone A) of the embankment, and excavated for the entire embankment length, to limit potential seepage at any level. If the cut material is suitable as fill, it may be replaced in the excavation in compacted layers; alternatively, suitable low permeability material will be imported, conditioned, placed and compacted in the trench, using the same compaction specifications as Zone A material.

Compacted Soil Liner

A compacted soil liner (200 mm thick) will be constructed over the TSF basin area, comprising either reworked in-situ material or imported Zone A material. The construction methodology for the compacted soil liner will be as follows:

- The TSF basin area will be cleared, grubbed, and topsoil stripped;
- Borrow areas within the TSF basin will be shaped to facilitate full drainage into the TSF and the final surface graded and compacted to maximise potential runoff;
- Where in-situ material is suitable (classified as Zone A material), scarify to a depth of 200 mm, moisture condition and compact in-situ material to achieve target density;
- Where in-situ material is unsuitable (too coarse for classification as Zone A material or low plasticity to non-plastic), suitable material will be won from borrow or sourced from borrow, placed, spread, moisture conditioned and compacted (thickness 200 mm), subsequent to scarifying (to a depth of 200 mm), moisture conditioning and compaction of in-situ material. This will primarily occur within natural drainage courses in the TSF basin.

The compacted soil liner surface shall drain positively and be finished with a smooth drum vibratory roller to promote runoff.

HDPE Geomembrane Liner

A geomembrane basin liner, comprising 1.5 mm smooth HDPE, will be installed over the compacted soil liner. The upstream slope of the TSF embankment and decant tower areas will be lined with a 1.5 mm textured HDPE liner. The HDPE will be anchored within a rectangular trench on the embankment crest and around the perimeter of the TSF basin.

Underdrainage System

The underdrainage system will be installed throughout the TSF basin area (over the HDPE geomembrane liner) and is designed to reduce the phreatic surface within the tailings mass, and subsequently the TSF embankments. The underdrainage system has several benefits, as follows:

- Reduces seepage through the basin and under/through the embankment. This is beneficial to the environment and promotes increased embankment stability;
- Drains the tailings mass, thus increasing the density of the tailings and providing a more efficient facility in terms of constructed storage capacity;
- Increases the strength of the tailings mass immediately adjacent to the embankment.

The design of the underdrainage system leverages the natural topography , requiring less basin re-shaping. The underdrainage system comprises two interconnected drainage networks, namely the collector drains and the finger drains.

Finger drains will be installed at approximately 100 m centres over the TSF basin area. The finger drains will consist of a 63 mm draincoil pipe covered with a 300 mm mound of drainage medium (Zone F1), wrapped in geotextile. A flap of HDPE liner will be installed over the upstream side of the finger drains to provide erosion protection from sheet flow prior to inundation with tailings. The finger drains will connect into the collector drains.

The collector drains will consist of a 100 mm draincoil pipe contained within a 400 mm deep trench, backfilled with drainage medium (Zone F1), double-wrapped in geotextile. The collector drains will be located on either side of a shaped drainage trench, approximately 8 m wide. The collector drains will be constructed so that the top surface of the drains is flush with the surrounding ground surface to reduce erosion losses and damage to the drains from tailings supernatant and rainfall runoff. The collector drains will feed directly into the underdrainage collection sump.

Underdrainage Collection Sump

An underdrainage collection sump will be excavated at the low point within the TSF basin, adjacent to the embankment upstream toe. The underdrainage sump will collect solution from the underdrainage network, and collected solution will be pumped back onto the tailings beach by a submersible pump situated at the base of a riser pipe running up the embankment upstream face, with flows reporting to the supernatant pond for recycling back to the process plant. The underdrainage sump will consist of the following components:

- An excavation nominally 2 m deep (minimum), 6 m by 3 m at the base. The sump will be backfilled with clean gravel material (Zone F2), overlain with drainage sand/gravel material (Zone F1) and wrapped in geotextile;
- Two 630 mm diameter HDPE riser pipes, running from the base of the sump, along the embankment upstream face to the crest elevation. The bottom 3 m of each riser pipe will be slotted. The riser pipes will be located on the embankment face and ballasted with pipes filled with concrete and fixed with a steel brace;
- A submersible pump will be situated at the base of one of the riser pipes (slotted section), within the collection sump. The pump will operate with a level control. A pump sled will be installed to facilitate removal of the underdrainage pump. The second riser pipe will provide redundancy in the event of pipe failure.

Leakage Collection and Recovery System

Seepage from the existing drainage courses within the TSF basin, beneath the basin composite liner, will be collected by the Leakage Collection and Recovery System (LCRS). The LCRS is independent of all other seepage control components. The LCRS drains will consist of 100 mm diameter draincoil pipes contained within a 900 mm deep trench beneath the collector drains, backfilled with drainage medium (Zone F2) to 700 mm depth (wrapped in geotextile), and overlain by a 200 mm low permeability (Zone A) material cap. The LCRS drains will feed directly into the LCRS collection sump.

In addition to providing a secondary drainage system beneath the TSF basin liner during operation, the LCRS has the following benefits:

- Improves accessibility of basin work areas during construction by helping to dewater the drainage course alluvial material;
- Reduces hydrostatic pressure from the alluvial material after the construction of the basin composite liner (in particular beneath the HDPE geomembrane liner).

The LCRS will report to a collection sump at the lowest point in the TSF basin, adjacent to the underdrainage collection sump. The LCRS sump will lie beneath the basin composite liner and consist of the following components:

- An excavation nominally 2 m deep, 6 m by 3 m at the base, filled with selected clean gravel material encapsulated in geotextile. The sump will be covered with a compacted soil liner and HDPE geomembrane liner (so that the sump sits beneath the basin liner);
- Two 630 mm diameter HDPE riser pipes, running from the base of the sump, on the embankment upstream face to the crest elevation. The bottom 3 m of each riser pipe will be slotted. Each riser pipe will be situated within a trench on the embankment face and covered with cement-stabilised backfill material (to fill the trench flush with the embankment face);
- A submersible pump will be situated at the base of the riser pipe (slotted section), within the collection sump. The pump will operate with a level control. A pump sled will be installed to facilitate the removal of the LCRS pump. The second riser pipe will provide redundancy in the event of pipe failure.

Downstream Seepage Collection System

A seepage collection system will be constructed within and downstream of the TSF Stage 1 embankment.

The downstream embankment toe drain will comprise a 160 mm draincoil pipe laid at the embankment toe, covered by a 1 m wide, 500 mm deep trapezoidal profile of drainage sand (Zone F1). This will be overlain by a 200 mm thickness of erosion protection material (Zone E). The downstream toe drain will capture and direct flow to the low point(s) of the downstream toe.

At the low point(s) along the downstream toe drain, flow will be directed into an HDPE outfall pipe, finally reporting to a seepage collection sump outside of the final TSF embankment footprint.

The downstream seepage outfall pipe is a buried, 160 mm diameter solid HDPE pipe with a minimum fall of 0.5% and a minimum cover depth of 300 mm. A bedding layer of 50 mm should be placed beneath the outfall pipe.

Each seepage collection sump consists of a buried concrete tower structure, comprising a 25 MPa concrete base and a 1.2 m diameter solid precast concrete pipe section.

Decant System

Supernatant water will be removed from the TSF via a floating turret system located in an excavated trench along the primary natural drainage course within the TSF basin. The turret(s) will be connected to a trailer-mounted pump (or other means of easily moving the pump) located on an access causeway running alongside the decant trench. The pump will be moved along the causeway as required during operation as the supernatant pond moves away from the TSF embankment to allow for seasonal fluctuations.

The decant system pumps will operate automatically, reclaiming water from the TSF and pumping it via a pipeline to the process plant.

Regular movement of the decant turret will be required to ensure it is located in sufficient water depth to reduce the risk of tailings solids being pumped back to the process plant and to ensure that good quality recycle water is returned to the process plant. The decant turret trench will be excavated to a depth of approximately 2.0 m to ensure sufficient water depth for the turret at all times, and to allow for some flexibility when moving the pump. The turret requires that the maximum vertical height between the turret (in the trench) and the pump (on the causeway) is less than 6 m at all times.

Emergency Spillway

The TSF has sufficient capacity to completely contain all design criteria storm events and rainfall sequences, as listed in Table 18.5.

Under normal operating conditions, with the TSF managed in accordance with standard operating procedures, the available stormwater storage capacity will be in excess of the design storm event volumes and no discharge from the TSF is expected.

In the event that a storm event greater than the TSF design criteria occurs, exceeding the available storage capacity during operation, rainfall and supernatant water which cannot be attenuated and stored within the supernatant pond will discharge from the TSF in a controlled manner via an engineered spillway. Discharge under these conditions is required in order to protect the integrity of the embankments from overtopping failure.

The operational emergency spillway will be constructed as part of each embankment raise. The spillway will be relocated in subsequent raises to higher ground as the embankments are raised. In all cases, the spillway channel is designed to discharge downstream of the TSF into the existing natural drainage course east of the TSF. For all stages, the emergency spillway inlet will be lined with smooth HDPE geomembrane liner.

The emergency spillway system will be designed on the basis that attenuation of the storm event occurs within the supernatant pond. The attenuation in the TSF and spillway design will allow single storm events with an average recurrence interval of up to the Probable Maximum Precipitation (PMP) to be discharged from the TSF while protecting the embankment from overtopping.

The closure spillway will be constructed by excavating a channel from the lowest point of the final tailings beach south of the west embankment to the adjacent water course. The TSF will be a fully water-shedding structure on closure. The closure spillway will allow conveyance of PMP storm events without attenuation in the TSF.

Tailings Deposition System

The deposition of tailings into the TSF will be sub-aerial from the TSF embankment, to locate the supernatant pond against the southern hillside within the TSF basin. The tailings distribution line will run adjacent to the deposition areas.

The deposition will occur from multiple spigots inserted along the tailings distribution line. The deposition location(s) will be moved progressively along the distribution line as required to control the location of the supernatant pond. After initial establishment of the tailings beaches, a suitable cycle time will be determined to evenly deposit the tailings around the TSF, thereby maintaining the supernatant pond at a suitable location and maintaining the formation of the tailings beach.

TSF Operation

<u>*Deposition Objectives*</u>

The tailings deposition strategy was designed and will be managed throughout the operation to meet the following objectives:

- Deposition of a basin lining within the inundation area;

- Maintenance of freeboard against the upstream embankment face;
- Deposition to improve sub-aerial drying and consolidation of tailings;
- Deposition to effectively utilise the net available storage capacity;
- Effective management of the size and location of the supernatant pond;
- Reduce the volume of water stored on the TSF at any time;
- Lower the operating costs of the tailings distribution system;
- Reduce down time by providing operational flexibility;
- Facilitate the implementation of the storage closure strategy;
- Reduce potential for dust generation.

Deposition Techniques

The deposition of tailings into the TSF will be sub-aerial from the TSF embankment, to locate the supernatant pond against the southern hillside within the TSF basin.

The tailings delivery pipeline will run from the plant site to the TSF embankment in an HDPE-lined trench or along the TSF perimeter and then adjacent to the deposition areas.

The deposition will occur from multiple spigots inserted along the tailings distribution line. The deposition location(s) will be moved progressively along the distribution line as required to control the location of the supernatant pond. After the initial establishment of the tailings beaches, a suitable cycle time will be determined to evenly deposit the tailings around the TSF, thereby maintaining the supernatant pond at a suitable location and maintaining the formation of the tailings beach.

The proposed tailings deposition method is the sub-aerial technique, which allows for the maximum amount of water removal from the facility by the formation of a large beach for drying and draining.

In conjunction with a small supernatant pond size, sub aerial deposition should increase the settled density of the tailings, and hence improve the storage potential and efficiency of the facility. During the early stages of operation, the deposition plan will be modified to improve the return water efficiency (limiting exposed tailings beach areas to reduce evaporation losses).

Tailings deposition will commence at the lowest point within the TSF basin area (adjacent to the TSF embankment). The deposition will then be moved on either side of this initial point to line the basin area whilst controlling the location of the supernatant pond. As such, the supernatant pond will be pushed into the southwestern valley within the TSF basin.

The tailings will initially be deposited in the TSF from the deposition areas in a manner that encourages the formation of beaches, over which the slurry will flow in a laminar non-turbulent manner and allow the supernatant pond to migrate up the valley. The solids will settle as deposition continues and water will be released to form a thin film on the surface of the tailings. A degree of segregation of the tailings will occur against the embankment, promoting de-watering of the tailings through the toe drain and thus enhancing stability, consolidation and reducing basin drainage.

Deposition of tailings will be carried out on a cyclic basis with the tailings being deposited over one area of the storage until the required layer thickness has been built up. The deposition will then be moved to an adjacent part of the storage to allow the deposition layer to dry and consolidate. This will facilitate optimal storage to be achieved over the whole area.

After deposition on a particular area of beach ceases and settling of the tailings has been completed, further de-watering will take place due partly to drainage into the underdrainage system but mainly due to evaporation. As water evaporates and the moisture content drops, the volume of tailings will reduce to maintain a condition of full saturation within the tailings. This process will continue until the interaction between the tailings particles hinders volume reduction.

Seepage Assessment

Seepage analyses were undertaken for the TSF embankment and basin to estimate the position of the phreatic surface within the embankment for both Stage 1 and final configurations.

The seepage model was developed using the analysis program SEEP/W (Ref. 12).

The phreatic surfaces obtained from the seepage modelling were incorporated into the stability assessment.

Stability Assessment

The stability of the proposed Stage 1 and final TSF embankment sections was assessed under static and seismic loading conditions using limit equilibrium methods. 'SLOPE/W' (Ref. 13), is a limit equilibrium program that will be used for the analysis.

The stability of the TSF was assessed in order to confirm the factors of safety against slope failure considering long-term drained conditions, short-term undrained conditions and post seismic conditions. The effect of pore water pressures on embankment stability was modelled during the stability assessment. Critical failure surfaces for the TSF were identified for the stability modelling. A critical failure surface is the potential failure surface that gives the lowest factor of safety and represents a failure that would likely cause significant damage if failure were to occur. Shallow failure surfaces that give low factors of safety, but do not represent a critical breaching of the embankment have not been included.

The minimum stability criteria based on Australian National Committee on Large Dams (ANCOLD) guidelines (Ref. 3) are provided in Table 18.5.

Based on the slope stability assessment, the Doropo TSF will have satisfactory factors of safety, meeting the recommended minimum factors of safety by ANCOLD (Ref. 3), and therefore should be stable as designed.

The stability analyses indicate that during operation the embankment profile satisfies all requirements for operational minimum factors of safety. All stability models were completed with the minimum embankment profiling. During operation, embankments will be further buttressed by ongoing mine waste placement.

18.5.2 TSF Dam Break Assessment

A dam breach assessment based on ANCOLD guidelines (Ref. 3) was carried out for the Doropo TSF to estimate the Population at Risk (PAR), business risk, and environmental impact in the event of a dam failure. The likelihood of failure is not considered in the ANCOLD Consequence Category Assessment.

Dam Breach Consequence Assessment

The key design parameters related to the dam break assessment are as follows:

- The final TSF configuration (41 Mt stored tailings) was assessed for each option;
- An emergency spillway exists as part of the TSF design;
- It was assumed that failure would occur at the ultimate facility height and at a location where the potential Population at Risk (PAR) will be the highest. This corresponds to a final embankment crest elevation of RL 341.3 m for a maximum height of approximately 34.1 m.

Dam breach methods were obtained from ANCOLD (Refs. 3 and 4) and Rico (Ref. 11). As these methods are structured for water dams in defined creeks or rivers, the methods were modified to account for tailings solids in relatively flat topography. The modelled inundation assists in determining the impact areas and PAR due to a dam breach. A significant failure of the embankment would result in a release of tailings and water depending on the location of the breach, its size and the cause.

In addition, a liquor breach for each scenario was considered to assess the impact in the event of pond mobilisation. If the supernatant pond (or a portion of the pond) mobilises during embankment failure, significant erosion of the tailings beach will occur during outflow. This would lead to tailings being carried downstream as suspended solids within the liquid outflow. The liquor breach is expected to be conveyed downstream via existing natural drainage courses and topographical features, where the flow will pass by several populated settlements, agricultural fields and local roads. It is possible that liquor/tailings flow could enter the Black Volta River system (~60 km from the TSF). In the event that liquor/tailings flow enters the Black Volta River system, it would continue approximately 145 km downstream passing through Burkina

Faso, Ghana, and Cote D'Ivoire, until encountering the Bui Dam located in Ghana. The area around the Bui Dam forms part of the Bui National Park.

The critical dam breach scenario is a failure of the TSF embankment at its highest point, due to the potential to mobilise the maximum volume of tailings downstream.

A dam breach would be expected to result in the following outcomes:

- It is possible that liquor/tailings flow could cross the Cote D'Ivoire/Burkina Faso Border (~15 km from the TSF);
- The low-lying areas of several villages which are primarily of agricultural areas adjacent to the Pouene river are potentially at risk of being within the tailings inundation path;
- The village of Pouenodoura (~30 km from the TSF) which lies on both sides of the river in Burkina Faso is potentially at risk of being within the tailings inundation path. The other towns tend to have a larger set back from the river;
- It is possible that liquor/tailings flow could enter the Black Volta River system (~60 km from the TSF);
- In the event that liquor/tailings flow enters the Black Volta River system, it would continue approximately 145 km downstream passing through Burkina Faso, Ghana, and Cote D'Ivoire, until encountering the Bui Dam located in Ghana. The area around the Bui Dam forms part of the Bui National Park;
- The entirety of the Nokpa, Chegue South, Kekeda, Han, and parts of the Chegue Main pits are at risk of inundation if present;
- The plant site is located on high ground and is not expected to be inundated with tailings, access from several pits would be cut off;
- The settlements of Hierewedouo, Loukoura, Lagbo, and Lagbobouro are located on higher ground and are not expected to be impacted, however, access to these locations may be cut off from certain directions.

Population At Risk Assessment

As part of the estimation of the hazard/consequence category for the TSF, an assessment of the indicative Population at Risk (PAR) was undertaken. For the analysis, the end condition was assessed as tailings discharge resulting from an embankment breach (modelled at the final height) for a range of conditions between dry and wet from the TSF embankment. Three cases were assessed:

- Case 1: Failure through the eastern side of the embankment where all pits located downstream of the TSF have been mined to design profiles, with no backfilling having occurred;
- Case 2: Failure through the eastern side of the embankment where only the Nokpa pit immediately downstream of the TSF has been mined;

- Case 3: Failure through the western side of the embankment into the WSD where all pits located downstream of the TSF have been mined to design profiles, with no backfilling having occurred.

The intention of modelling Case 2 is to determine the likely inundation extents should limited attenuation be available from mine pits, either as a result of them not having been mined, or having limited capacity at the time of failure. Nokpa pit was identified as the first major pit downstream of the TSF to be mined and was therefore chosen as the sole pit to be incorporated in the model. It is not anticipated that all people will be downstream of the TSF at the same time; hence a percentage estimate of each role was approximated. A total PAR was then calculated, as shown in Table 18.7, Table 18.8 and Table 18.9.

Scenario	Number	Affected (%)	PAR
TSF Operation Personnel*1	2	10	0.2
TSF Construction Personnel - Embankment*1,2	30	50	15
Plant Site*3	146	0	0
Site Camp*3	300	0	0
Nokpa Pit*4	0	100	0
Chegue South Pit*5	25	100	25
Chegue Main Pit*4	0	50	0
Kekeda Pit*4	0	100	0
Han Pit*5	25	100	25
Loukoura*5	426	1	4.3
Lagbo*6	1,013	1	10.1
Lagbobouro*5	315	1	3.2
Hierewedouo*5	796	1	8
Total	3,078	-	90.8

Notes: (1) The number of personnel has been estimated based on experience of similar projects within the region.
(2) TSF Construction Personnel only operational during TSF construction raises.
(3) Population to be confirmed, but not anticipated to be within inundation extents.
(4) Pit assumed inactive. It was assumed that a maximum of two open pits would be operational at any given time.
(5) Population estimate provided by Centamin.
(6) Population provided by Centamin.

Table 18.7 Case 1 - Indicative Population at Risk (PAR)

Scenario	Number	Affected (%)	PAR
TSF Operation Personnel*1	2	10	0.2
TSF Construction Personnel - Embankment*1,2	30	50	15
Plant Site*3	146	0	0
Site Camp*3	300	0	0
Nokpa Pit*4	10	100	10
Loukoura*4	426	1	4.3
Lagbo*5	1,013	1	10.1
Lagbobouro*4	315	1	3.2
Hierewedouo*5	796	1	8
Total	2,242	-	50.8

Notes: (1) The number of personnel has been estimated based on experience of similar projects within the region.
(2) TSF Construction Personnel only operational during TSF construction raises.
(3) Population to be confirmed, but not anticipated to be within inundation extents.
(4) Population estimate provided by Centamin.
(5) Population provided by Centamin.

Table 18.8 Case 2 - Indicative Population at Risk (PAR)

Scenario	Number	Affected (%)	PAR
TSF Operation Personnel*1	2	10	0.2
TSF Construction Personnel - Embankment*1,2	30	25	7.5
Plant Site*3	146	0	0
Site Camp*3	300	0	0
Nokpa Pit*4	0	0	0
Chegue South Pit*4	0	0	0
Chegue Main Pit*5	25	0	0
Kekeda Pit*5	25	0	0
Han Pit*5	0	0	0
Loukoura*5	426	0	0
Lagbo*6	1,013	0	0
Lagbobouro*5	315	0	0
Hierewedouo*5	796	0	0
Penedouo*5	200	0	0
Lassouri*6	242	0	0
Total	3,078	-	7.9

Notes: (1) The number of personnel has been estimated based on experience of similar projects within the region.
(2) TSF Construction Personnel only operational during TSF construction raises.
(3) Population to be confirmed, but not anticipated to be within inundation extents.
(4) Pit assumed inactive. It was assumed that a maximum of two open pits would be operational at any given time.
(5) Population estimate provided by Centamin.
(6) Population provided by Centamin.

Table 18.9 Case 3 - Indicative Population at Risk (PAR)

Based on the above estimate, a PAR of '>10 to 100' is recommended for the TSF.

The PAR due to an environmental spill would be limited, and on this basis a PAR of '<1' was adopted for the Environmental Spill Consequence Category.

Dam Failure Severity Level

An assessment to determine the severity level of impacts from a large-scale failure of the facility (Dam Failure Severity Level) and a spillway flow or other water release (Environmental Spill Severity Level) was conducted.

Based on significant damage to infrastructure, business and public health the Severity Level of a dam failure would be 'Major', primarily due to the anticipated business and public health impacts if tailings were to impact local communities downstream.

The severity level of an environmental spill would be considered 'Major' due to the potential public health risks for settlements downstream of the TSF.

Consequence Category

Based on the PAR calculation and Dam Failure Severity Level review summarised above, the ANCOLD (Ref. 3) consequence categories for the Doropo TSF are provided in Table 18.10 and Table 18.11, for dam failure and environmental spill, respectively.

Stage	PAR	Severity of Damage or Loss	ANCOLD Consequence Category
Final	≥10 to <100	Major	High B

Table 18.10 ANCOLD Consequence Category - Dam Failure

Stage	PAR	Severity of Damage or Loss	ANCOLD Consequence Category
Final	<1	Major	Significant

Table 18.11 ANCOLD Consequence Category - Environmental Spill

Design Requirements

Upon assessment of the minimum design parameters recommended by both ANCOLD and the GISTM, a set of design parameters was adopted on the basis of achieving compliance with both standards. Consequently, for each parameter, the more stringent of the two standards was adopted for the Doropo TSF design. The resulting design parameters adopted are summarised in Table 18.12.

Design Parameter	Value
Dam Failure Consequence Category	High B
Dam Spill Consequence Category	Significant
Design Storage Allowance Parameters	
Design Storage allowance Extreme Storm Storage Contingency Freeboard (Wave Run-up) Contingency Freeboard (Additional Freeboard)	1:10 AEP2 notional wet season runoff1 1:100 AEP2 72 hour storm 1:10 AEP2 wind 0.3
Emergency Spillway Design Parameters	
Design Flood Wave Freeboard Allowance	1:100,000 AEP2 1:10 AEP2 wind
Design Earthquake Loadings	
Operating Basis Earthquake (OBE) [Operations and closure (active care)] Safety Evaluation Earthquake (SEE) [Passive-closure (passive care)]	0.02% AEP2 (1 in 5,000) 0.01% AEP2 (1 in 10,000)
Factors of Safety	
Long-term drained Short-term undrained (potential loss of containment) Short-term undrained (no potential loss of containment) Post-seismic	1.5 (Effective strength) 1.5 (Consolidated Undrained Strength) 1.3 (Consolidated Undrained Strength) 1.0-1.2 (Post seismic Shear Strength)

Notes: (1) Assuming no evaporation and 100% runoff coefficient.
(2) Annual Exceedance Probability (AEP).

Table 18.12 ANCOLD Design Criteria

Including the parameters provided in Table 18.12, the Doropo TSF design is characterised by downstream embankment construction throughout operation, a composite basin liner system, and an underdrainage network in low-lying areas of the basin. These features are considered to be at the higher end of TSF design features/specifications for the region as a whole. It is noted that a lower consequence category for the Doropo TSF would not result in significant cost reductions.

18.6 Surface Water Management

18.6.1 Site Water Balance

Water Balance Modelling

The management of water is a critical aspect of the design for Doropo Project. To understand and control the flow of water around the site, a water balance model was developed.

The primary objectives of the water balance modelling are summarised below:

- Establish the filling rate for tailings solids within the TSF;
- Estimate the in-situ tailings density within the TSF, taking into consideration tailings properties from laboratory testing and the TSF basin storage parameters;
- Determine supernatant pond volumes within the TSF under average climatic conditions throughout operation;
- Determine supernatant pond volumes within the TSF for design wet rainfall sequences and storm events, check TSF stormwater storage capacity and confirm the suitability of the current TSF design philosophy;
- Determine staged embankment crest elevations, to ensure containment of tailings and design supernatant pond volumes;
- Determine the likelihood of recycle water shortfalls during average conditions and design dry rainfall sequences;
- Determine the required WSD capacity to store make-up water for these shortfalls;
- Determine the required WHD capacity and abstraction rate (from WHD to WSD) to provide supplemental make-up water for shortfalls;
- Assess risk factors for water balance modelling.

The water balance modelling included the TSF, WHD, WSD, and process plant, with a view to determining site water storage requirements. Design wet conditions were modelled to ensure that the TSF is designed with sufficient storage capacities to comply with ANCOLD guidelines (Ref. 3).

For the water management modelling, the following water requirements and priorities were assumed to supply the required process plant water demand:

- The water present in the ore was included;
- Water collected from Plant Site runoff was not included as the reliability of surface water management is unknown;
- The minimum raw water requirement (for use in the process plant) is sourced from the WSD. The water reservoir in the WSD is supplemented by water abstracted from the WHD and pit dewatering (not considered in the water balance);
- The additional raw water requirement for dust suppression and wash down (external to the process) is sourced from the WSD. Seasonal variation for dust suppression was considered;
- After allowing for the bulleted items above, the model attempts to supply the remaining process water requirement from the TSF supernatant pond (recycle);
- After including the process water return from the TSF, any further water requirements (make-up water) are provided by the WSD;
- If the process water requirement is not met, the model then determines the shortfall volume.

Various design rainfall conditions were modelled for selected operational years. The following rainfall sequences were modelled:

- Average conditions;
- 100 year ARI, 1 year dry rainfall sequence;
- 100 year ARI, 1 year wet rainfall sequence;
- 100 year ARI, 72 hour duration storm event superimposed over average rainfall sequence;
- 10 year ARI wet season (120 days' duration), with 100% runoff and no evaporation.

For all design rainfall conditions, it is assumed that average climatic conditions occur prior to and subsequent to the design events or rainfall sequences.

Water Balance Modelling Results

Tailings Storage Facility

- The TSF is designed to hold the tailings generated, plus the design rainfall conditions and has sufficient stormwater storage capacity for all design storm events and rainfall sequences;
- The critical design rainfall event in terms of pond elevation occurs during the design storm event in the last month of each stage of operation when the TSF stormwater capacity is at a minimum. There is sufficient stormwater storage capacity in the TSF for this situation during all stages of operation;
- The supernatant pond volume peaks in October each year (at the end of the wet season);
- The water balance remains positive after decommissioning (increasing pond volume); therefore, the supernatant pond should be removed (and treated if necessary and released) as soon as practicable after decommissioning.

Plant Site and Water Supply

- A process water shortfall is expected to occur under average and design dry climatic conditions;
- All make-up water requirements can be provided by the WSD reservoir, supplemented by the WHD for design dry conditions.

Water Storage Dam

- It is necessary that the WSD is completed early to allow a full wet season for filling prior to commissioning;
- For a basin permeability of 1×10^{-7} m/s, a WSD storage capacity of 2,000,000 m^3 is required to provide sufficient make-up water, supplemented by an abstraction rate of 125 L/s (unfactored) from the WHD. This includes allowances for storage losses due to evaporation and seepage in both the WSD and WHD reservoirs.

<u>Water Harvesting Dam</u>

- A WHD capacity of 500,000 m^3 is required to prevent shortfalls under design dry conditions;
- The required abstraction rate from the WHD to the WSD is 125 L/s (unfactored).

18.6.2 Water Harvest Dam

The WHD is the primary water collection structure and can store up to 500,000 m^3 of water at the maximum operating level. The design intent of the WHD is that the reservoir will be frequently pumped to the WSD during each wet season, with a view to filling the WSD reservoir to its maximum storage level prior to each dry season. The WHD general arrangement is shown in Figure 18.8.

The embankment crest elevation for the WHD was determined based on the stage storage curve for the reservoir. Design parameters for the WHD are provided in Table 18.13.

Design Parameter	Value
Storage Capacity	0.5 Mm^3
Spillway Capacity	100 year ARI/critical duration
Earthquake Loading	As per Table 18.5.
Factors of Safety	As per Table 18.5.
Operation	
Fluid Management	Abstraction tower system for removal of water. Abstraction to the WSD via submersible pump.*1
Embankment	
Embankment design: - Crest Width - Upstream Slope - Downstream Slope	 6 m 3H:1V 6H:1V
Construction Description - Cut-off Trench - Embankment	 Central cut-off through residual/transported material. Multi-zoned earth fill embankment, with central low permeability core.
Materials	
Material Supply	Low permeability fill (Zone A) sourced by civil contractor from local borrow. Structural fill (Zone C) sourced by civil contractor from local borrow. Random fill (Zone D) sourced by civil contractor from local excavations or borrow. Rip rap for erosion protection (Zone E) sourced from off-site quarry or local borrow. Drainage material (Zone F1) sourced from local river beds or off-site quarry. Coarse rockfill (Zone G) sourced from local borrow or off-site quarry. Wearing course material sourced by civil contractor from local borrow or selected mine waste.

Design Parameter	Value
Rehabilitation	
Rehabilitation	Breach or remove WHD, replace topsoil, rip along contour and re-vegetate.

Notes: (1) Design by others.

Table 18.13 Water Harvesting Dam Design Parameters

The WHD has a catchment area of 43,000 ha, and when the pond volume is at maximum level, the reservoir surface area will be 55 ha.

The water collected in the WHD will be pumped to the WSD. Water will be removed from the WHD by submersible pump(s) situated at the base of an abstraction tower. The abstraction tower will comprise a 25 MPa concrete base and 1.8 m square slotted precast concrete sections. The abstraction tower will be surrounded by free-draining coarse rockfill (Zone G), and an access causeway will be constructed to the tower.

The WHD embankment comprises a central low permeability core (Zone A), with upstream and downstream free-draining coarse rockfill (Zone G). Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability core of the embankment. The material will be moisture conditioned, spread and compacted in 300 mm (uncompacted) layers. The target permeability of the zone shall be less than 1 x 10^{-8} m/s after conditioning and compaction;
- Zone G material shall be won from borrow (if available) or sourced off-site;
- Zone F1 shall be clean sand/gravel drainage material supplied to a stockpile adjacent to the works. The Zone F1 material will be sourced from local borrow (if available) or off-site.

A cut-off trench will be located beneath the low permeability core (Zone A) of the embankment. The cut off trench will be cut to extend through to competent low permeability foundation material. The cut-off trench will be constructed continuously along the embankment and backfilled with low permeability fill (Zone A).

Discharge from the WHD will occur in a controlled manner via the outlet structure. As the WHD is expected to fill during each year of operation, it is anticipated that the overflow will occur frequently each wet season and discharged water will report to the existing stream bed downstream of the WHD.

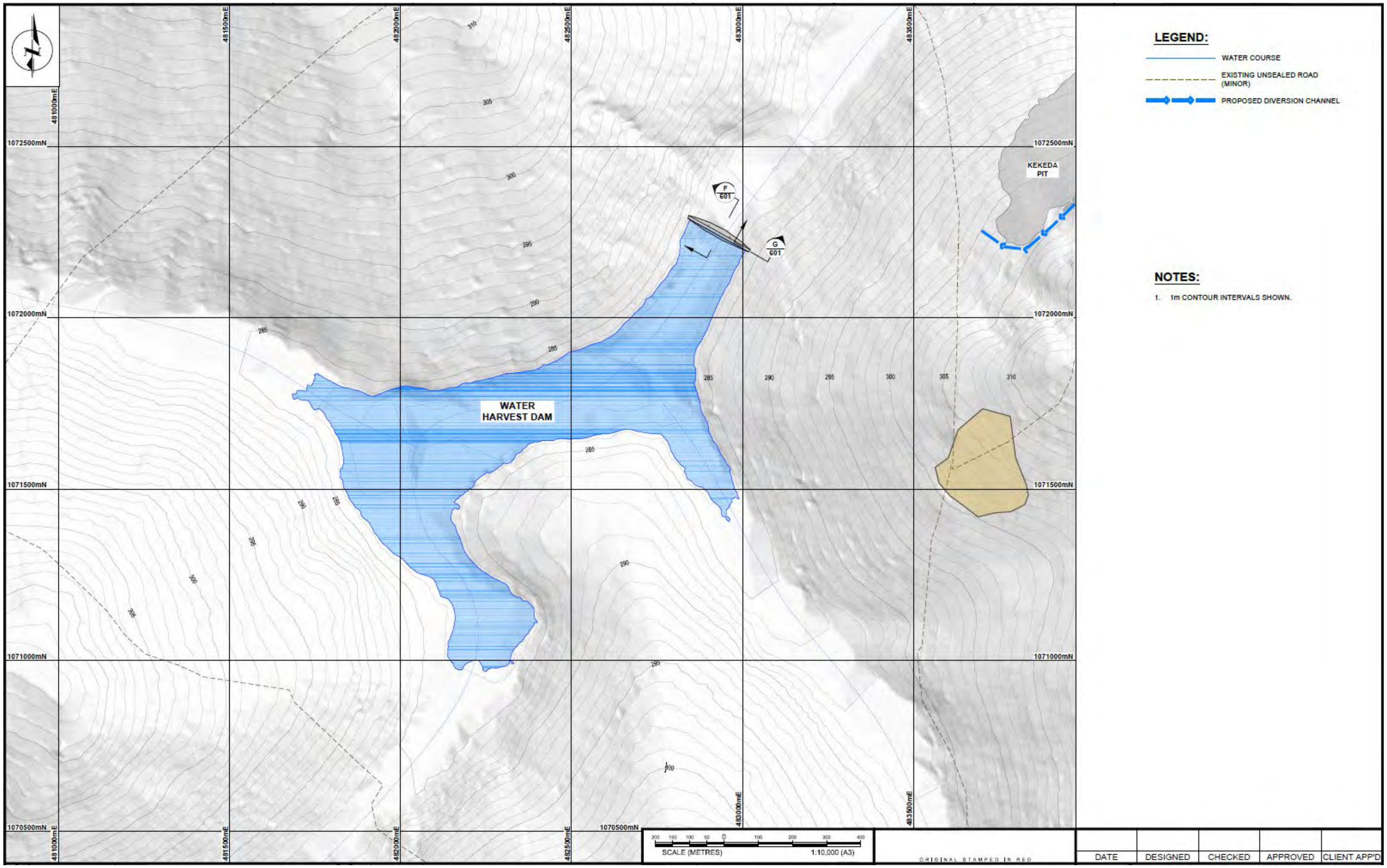


Figure 18.8 WHD Layout (Source: Knight Piésold, 2024)

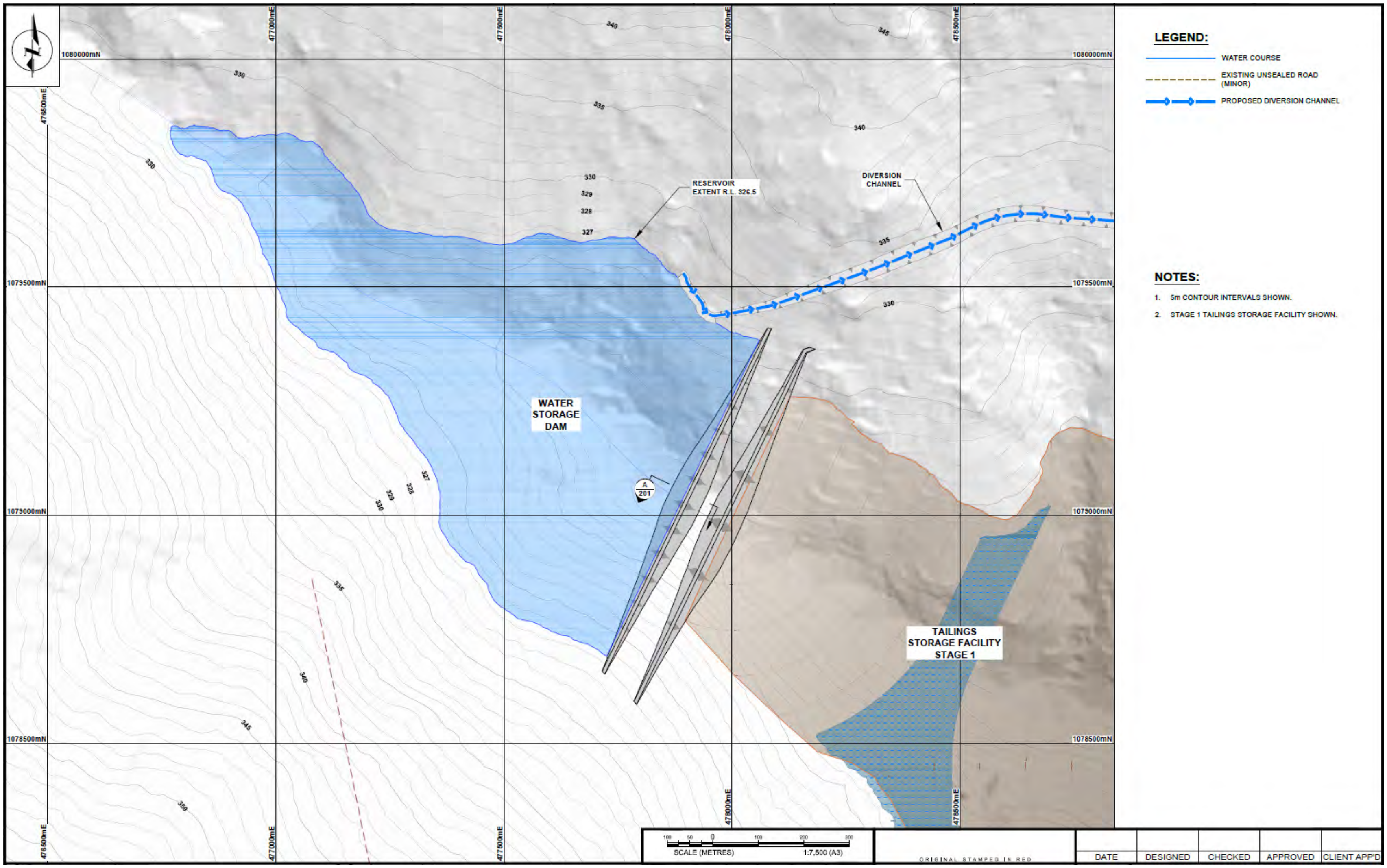


Figure 18.9 WSD Layout (Source: Knight Piésold, 2024)

18.6.3 Water Storage Dam

The WSD is the primary storage pond for clean process water on site and can store up to 2,000,000 m^3 of water at its maximum operating level. The WSD general arrangement is shown in Figure 18.9.

The embankment crest elevation for the WSD was determined based on the stage storage curve for the reservoir. Design parameters for the WSD are provided in Table 18.14.

Design Parameter	Value
Storage Capacity	2.0 Mm^3
Spillway Capacity	100 year ARI/critical duration
Earthquake Loading	As per Table 18.5.
Factors of Safety	As per Table 18.5.
Operation	
Fluid Management	Abstraction from the WSD (to the process plant) via floating pump.*1
Embankment	
Embankment design: - Crest Width - Upstream Slope - Downstream Slope	 8 m 3H:1V 2.5H:1V
Construction Description - Cut-off Trench - Embankment	 Central cut-off through residual/transported material. Multi-zoned earth fill embankment, with central low permeability core.
Materials	
Material Supply	Low permeability fill (Zone A) sourced by civil contractor from local borrow or selected mine waste. Structural fill (Zone C) sourced by civil contractor from local borrow or selected mine waste. Structural fill (Zone C1) sourced from mining operation, either pre-strip (Stage 1) or run of mine waste (Stage 2+). Random fill (Zone D) sourced by civil contractor from local excavations, borrow or selected mine waste. Rip rap for erosion protection (Zone E) sourced from mining operation, local borrow or off-site quarry. Drainage material (Zone F1 and Zone F2) sourced from local river beds, local borrow or off-site quarry. Coarse rockfill for decant surround (Zone G) sourced from mining operation, local borrow or off-site quarry. Wearing course material sourced by civil contractor from local borrow or selected mine waste.

Design Parameter	Value
Rehabilitation	
Rehabilitation	Handover to the community

Notes: (1) Design by others.

Table 18.14 Water Storage Dam Design Parameters

The WSD has a catchment area of 1,100 ha, with the maximum flood extent limited to RL 328 m. The WSD is intended to be recharged by water abstracted from the WHD and rainfall runoff from its upstream catchment. Pit dewatering will also be pumped to the WSD. It is assumed that dust suppression and wash down water will be sourced from the WSD.

The water collected in the WSD will be pumped back to the plant to supply plant raw water requirements and process make-up water requirements. Water will be recovered from the WSD by a floating pump.

The WSD embankment comprises a central low permeability core (Zone A), with outer structural zones (Zone C). Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability core of the embankment. The material will be moisture conditioned, spread, and compacted in 300 mm (uncompacted) layers. The target permeability of the zone shall be less than 1 x 10^{-8} m/s after conditioning and compaction;
- Zone C material shall be won from borrow to form the outer structural zones. The material will be moisture conditioned, spread, and compacted in 500 mm (uncompacted) layers.

A cut-off trench will be located beneath the low permeability core (Zone A) of the embankment. The cut-off trench will be cut to extend through to competent low permeability foundation material. The cut-off trench will be constructed continuously along the embankment and backfilled with low permeability fill (Zone A).

The WSD basin area will be cleared, grubbed, and topsoil stripped to ensure that the process water supply remains free of organic material.

Discharge from the WSD will occur in a controlled manner via an engineered diversion channel, in order to protect the integrity of the embankments from overtopping failure and limit the WSD flood level to RL 328 m. The WSD diversion channel will be excavated to the north of the TSF, and runs along the northern TSF embankment, before discharging into the existing water stream. The diversion channel will be lined with 1.5 mm textured HDPE geomembrane liners where the gradient is 0.1%. Beyond the waste dump near Nokpa pit, the gradient changes to more than 1%, and erosion protection material (Zone E) is adopted. Depending on the abstraction from WHD, WSD is expected to fill during each year of operation. It is anticipated that the diversion channel will flow each wet season, and discharged water will report to the existing stream bed downstream of the TSF, west of Nokpa.

18.6.4 Sediment Management

The main component of the site sediment control system will be Sediment Control Structures (SCSs). SCSs are sediment dams that will be constructed in the downstream reaches of catchments impacted by site infrastructure, to attenuate sediment-laden runoff and facilitate settling of sediments before discharge. Water quality in the SCS reservoirs should be monitored, and the water may be pumped back to the plant for treatment (if required) or used for process make-up water requirements and dust suppression (if suitable). Significant upstream vegetation clearing and grubbing should not occur prior to the construction of the SCS embankments. Design parameters for the SCSs are summarised in Table 18.15.

Design Parameter	Value
Design Maximum Water Depth	1.75 m for 2 year ARI
Construction Description	Homogeneous low permeability (Zone A) material earth fill embankment.
Spillway - Configuration - Capacity	 Overflow via gravity discharge system (open channel). 100 year ARI storm event, occurring when pond is at spillway inlet level.
Rehabilitation	
Rehabilitation	Breach or remove SCS, replace topsoil, rip along contour and re-vegetate.

Table 18.15 Sediment Management System Design Parameters

SCSs reduce flow velocities, facilitating sediment settling. For minor events and, depending on storage within the structure prior to a rainfall event, they may completely contain runoff.

A total of ten SCSs have been included based on the site layout. This requirement should be reviewed during the next design phase if the Open Pit were to expand or infrastructure is relocated.

It is noted that the WHD will also act as a sediment basin downstream of some site infrastructure.

The SCSs were designed to limit maximum water depth to 2 m for safety reasons (drowning risk). As such, the maximum embankment height for the SCSs is approximately 3 m. The SCS embankment will be a homogeneous earth fill embankment comprising low permeability fill (Zone A), won from local borrow areas within the SCS basin if possible. The downstream face will be revegetated.

SCS embankment construction shall commence at the lowest point in the valley. This will capture runoff generated during the SCS construction period and reduce the risk of significant sediment release during SCS construction. Significant upstream vegetation clearing and grubbing should not occur prior to construction of the SCS embankments.

Details of the SCSs are provided in Table 18.16.

Structure	Embankment Height (m)	Height to Spillway (m)	Storage Capacity (m3)	Catchment Area (Ha)	Reservoir Area (Ha)
SCS 01	2.7	1.8	2,700	98	0.3
SCS 02	2.9	1.8	4,830	150	0.5
SCS 03	2.5	1.5	2,290	37	0.3
SCS 04	1.5	0.4	2,490	371	1.0

Table 18.16 SCS Design Information

SCS 05 to 10 are expected to be operational from year-3 onward. Consequently, their design was not included in the feasibility study. It is more prudent to design these systems once a deeper understanding of the operation and infrastructure is available.

Discharge from the SCSs will be to the environment downstream of the project site via an engineered spillway. The spillway will be lined with erosion protection material (Zone E). As the SCS is expected to fill frequently, it is anticipated that the spillway will flow frequently during each wet season. Discharged water will report to the existing stream bed downstream of the SCS.

Diversion channels will be constructed upstream of SCSs to divert clean runoff around the sediment basin, improving attenuation of sediment-laden runoff. Design parameters for the diversion channels are summarised in Table 18.17.

Design Parameter	Value
Flow capacity - Diversion Channels	2 year ARI
Base width (minimum)	1 m
Side slopes - Channel	2H:1V to 2.5H: 1V
Water freeboard	0.3 m (minimum)

Table 18.17 Surface Water Management System Design Parameters

Additional local sediment control measures may be incorporated into the system as required, with examples listed below:

- Brushwood Barriers;
- Geotextile Silt Fences;
- Drop Inlet Structures;
- Chute Structures;
- Rock Check Dams;

- Gabion Sediment Weirs;
- Rock Filter Dams;
- Level Spreaders.

18.6.5 Monitoring

A monitoring programme for the TSF will be developed to monitor any potential problems which may arise during operations. The monitoring will include:

- Monitoring bores and surface water sampling stations downstream of the TSF;
- Standpipe piezometers in the TSF, WSD and WHD embankments to monitor the phreatic surface;
- Survey pins to check embankment movement.

If the monitoring programme indicates that potential problems are developing, an increase in monitoring frequency will be implemented and a response plan developed.

Seepage Monitoring

The TSF design incorporates a number of measures to reduce the amount of seepage that will occur from the TSF, in order to mitigate the extent of any effects on the downstream environment.

Two groundwater monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during the operating life and following decommissioning. In some instances, a fill platform/causeway may be required to ensure year-round access to the monitoring station.

The monitoring bore station consists of one shallow bore, extending to a depth of 10 m in the deep surface horizon, and one deep bore terminating in fresh rock (depth to be assessed by Centamin based on hydrogeology assessment). The shallow bore is intended to detect any seepage from the TSF flowing within the surface sediment, whilst the deep bore will monitor the chemical composition of the groundwater. Each borehole will be cased and screened over an interval set in the field during installation and sealed to surface with low permeability grout. The PVC tube for the monitoring bores will be 100 mm diameter. It is recommended that the boreholes are constructed before commissioning of the TSF in order to accumulate baseline data specific to the TSF location.

Stability Monitoring

Pore water pressures will be monitored at several locations within the TSF and WSD embankments to allow the stability of the structures to be assessed. Standpipe piezometers will be installed on the TSF, WSD and WHD embankments.

Each standpipe will consist of a 50 mm diameter PVC tube slotted at the base or supplied with a filter tip. The slotted section will be surrounded by sand, and bentonite pellets will be placed above the sand to provide a seal. The remainder of the hole will be sealed with a bentonite/cement grout. The top of the piezometer will be provided with a lockable cap to prevent tampering or vandalism. The base of each piezometer will be located within the embankment fill to ensure that the phreatic surface within the embankment, as opposed to the natural groundwater level, is being measured.

Additional piezometers will be installed as the TSF embankments are raised, to monitor the development of the phreatic surface in the embankments. During each TSF embankment raise, existing piezometers will be sealed with cement/bentonite grout mix via tremie pipe and grout pump and new piezometers installed on the raised embankment crest. Alternatively, once the base of the piezometers is beneath the embankment phreatic surface (i.e. returning water level readings), vibrating wire piezometers may be installed in the standpipes and cables extended with each raise. This would preclude the requirement to decommission and install standpipes during each raise.

Survey Pins

Survey pins will be installed at regular intervals along the TSF, WSD and WHD embankment crests to monitor embankment movements and assess the effects of any such movement on the embankment. Note that the WHD crest survey pins should be monitored during the initial WHD filling only, as they will likely be disturbed by the WHD overflow. The details of each pin will be recorded on installation, and regular survey checks will be completed during operation with a view to detecting displacement.

Operational Audits

The TSF will undergo annual audits by the Engineer of Record (the Designer) to ensure that the facilities are operating in a safe and efficient manner in accordance with the design intent.

For the assumed TSF consequence category of ‘High B’, an additional third-party audit should be conducted by a Dams Specialist on a biennial basis (starting in Year 1), as per ANCOLD guidelines (Ref. 3).

18.6.6 Raw Water and Potable Water

The raw water for the processing plant and ancillary facilities will be sourced from the WSD with seasonal replenishment from the WHD and supplemented by pit dewatering. The raw water will be stored within the 2,000 m^3 raw water tank located within the processing plant. The raw water usage is calculated to be 451 m^3/h when processing oxide ore and 412 m^3/h on fresh ores.

Initially, raw water will supply both the process and raw water requirements for the processing plant. This will continue until the process water begins recycling from the leach feed thickener and TSF decant. At steady state production the raw water nominal make up is estimated to be 243 m^3/h on oxide/transitional ores and 230 m^3/h on fresh ores. The raw water tank and distribution system will provide raw water to the processing plant whilst maintaining the firewater reserve for the processing plant.

A dedicated raw water tank at the camp will be fed from the plant raw water distribution system. The camp raw water tank holds 350 m^3 and provides the firewater reserve for the accommodation facilities.

The potable water for the project will be supplied by reverse osmosis (RO) treatment of the raw water with chlorine sterilisation. The RO plant will be located at the processing plant and will receive raw water feed-stock from the raw water distribution system. The permeate produced reports to the 200 m^3 potable water tank where the potable water pumps discharge into a ring main to supply the plant safety showers, site drinking water, ablution blocks, laboratory, crib room, stores, and workshop. The potable water demand has been sized for the 300bed accommodation camp and 80-bed security camp and facilitates drinking, ablution and general needs for 380 people, as well as general potable water use in the processing plant.

Potable water will be piped from the process plant potable water distribution system to fill the 350 m^3 camp potable water tank. The potable water will then be distributed around the camp by the camp potable water pump or camp diesel potable water pump, in the event of power loss.

18.7 Power Supply and Distribution

18.7.1 Power Supply

ECG Engineering Pty Ltd (ECG) were engaged by Centamin to invest, select and design the optimal high voltage power supply solution to the proposed Doropo Gold Plant.

The planned source of primary electric power for the project is via connection to the Côte d'Ivoire electricity grid. ECG determined that the most appropriate grid connection is to upgrade the Bouna Substation by extending the existing 90 kV bus, adding a 90 kV transmission line feeder, construction of 65 km of 90 kV single circuit lattice tower transmission line, and constructing a substation at Doropo site.

The company La Société des Energies de Côte d'Ivoire (CI-ENERGIES) own the National Interconnected Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

The Doropo Plant Substation will be owned and operated by CIE and Doropo mine would take a 90 kV tariff metered feeder, installing a 90/11 kV transformer in their substation and taking an 11 kV feeder to the Plant Main 11 kV Main Switchboard.

The 90 kV supply at Bouna should be of good quality and reliable. The recently constructed 90 kV transmission line from Bondoukou to Bogofa and Bouna is lightly loaded. The supply at Bondoukou is from the 225 kV network and is part of a ring-main system which is very reliable and based on an ECG risk assessment, is not expected that a full back-up power station is required as part of the grid connection works.

The design of the proposed power supply solution has been based on an estimated installed load for the site of 27 MW and a maximum demand (operating) load of 21 MW.

Emergency power will be provided by on-site diesel generators (2 x 1,000 kVA).

18.7.2 Power Distribution

The electrical system has been sized to take into account all anticipated Project loads, including the processing plant and supporting infrastructure such as the accommodation camps, mining contractors' area and ancillary loads such as the workshop, warehouse, laboratory and administration buildings.

The site power distribution will be at 11 kV with the working voltage being 415 V. The main 11 kV switchboard will be located in the high voltage (HV) switchroom which will be located adjacent to the grinding area, in close proximity to the largest drives in the plant including the SAG mill motor (7 MW), ball mill motors (2 x 4.5 MW) and cyclone feed pumps (750 kW).

This board will be supplied via a feeder from the main 90 kV/11 kV transformer located in the Doropo Plant Substation and will distribute power to the following transformers within the plant area:

- Primary crushing area switch room;
- Coarse ore storage and reclaim area switch room;
- Grinding, leaching, gold recovery and reagents area switch room;
- Leach feed thickening and water services area switch room.

Kiosks will be installed to supply the administration and mining contractors' areas.

Power to the remote facilities, including the accommodation camps, WHD transfer pumps, WSD raw water supply pumps and the TSF decant return pumps, will be provided by underground cable and 11 kV overhead power lines running from the main 11 kV switchboard to the remote loads.

Prefabricated transportable switch rooms constructed of non-combustible materials will be installed and equipped with VESDA smoke detection systems and hand-held fire extinguishers. Switchrooms will be elevated above the ground on steel columns to allow for the installation of cables from below the switchboards. These rooms will house the 415 V motor control centres (MCC), variable speed drives, instrument marshalling and programmable logic controller (PLC) cubicles required for the equipment being supplied and controlled.

The 415 V MCCs will be of a Form 4 design, either a single-sided or back-to-back configuration and arranged for connection from below the boards. The PLC equipment associated with the motor control drive modules will be built into one or more of the MCC tiers or installed in a dedicated PLC cabinet. PLC inputs and outputs (I/O) will be hard-wired between drive modules and the PLC racks.

Communications between the MCC and control system human machine interface (HMI) will be via Ethernet and by fibre or copper as appropriate. Communications in the remote facilities (accommodation camps, WHD transfer pumps, etc.) will utilise radio telemetry.

Low voltage variable speed drives will be the VVVF six-pulse type and will be either wall or floor mounted depending upon their size and weight.

All drives will have local control stations with start and stop buttons adjacent to the drive to provide local control for maintenance. Selected drives will also be remotely operable from the central control room via the operator interface terminal. The operating status of all drives will be displayed on the operator interface mimic pages. Any drive fault will be reported by the control system and an alarm will be initiated and logged.

Control voltage for all drives and local control stations will be 24 VDC.

Cable ladders to be installed throughout the plant will be NEMA 20B type, hot dipped galvanised. Cable ladders located in areas exposed to acid or cyanide solutions will be constructed from 316 stainless steel. Where necessary cable ladder bends, risers, tees and reducers of the same specification will be installed. Peaked cable ladder covers will be fitted where cables in the ladder are subjected to direct sunlight or where there is the potential for mechanical damage.

Low voltage (LV) power cables in processing areas will generally be steel wire armoured with XLPE or PVC insulation. Screened cables will be used for all variable speed drive applications.

LED light fittings will be used for the general plant lighting. Battery back-up lighting will be installed in all switch-rooms and access ways to ensure safe evacuation in the event of a blackout. All LV power circuits and sub circuits will be protected by instantaneous residual current devices (RCDs).

The plant electrical, control, and instrumentation systems will all be designed and installed in accordance with the relevant Australian Standards, IEC Standards, and Ivorian statutory requirements.

18.8 Bulk Earthworks

The plant and infrastructure bulk earthworks activities include:

- Clearing and grubbing;
- Topsoil removal, stockpiling, and management;

- Bulk cut and fill to establish the process plant levels;
- Import suitable fill materials from borrow locations to construct process plant pads;
- Removal of surplus materials and waste to stockpiles; and
- Construction of the plant event pond.

Topsoil will be removed across the site and stored on-site for future reuse during mine closure. Subsoil will generally be left in situ, except where excavated for cut to fill purposes and where surplus excavated material is hauled to stockpile and for concrete foundations.

Following stripping activities, the earthworks pads for key areas within the site will be sloped to reduce cut to fill earthworks volumes. Based on the elevations from cut to fill (with the balance hauled to stockpile) the bulk earthworks will level the site at the key areas and result in suitable drainage gradients towards the north and the south of the main axis of the processing plant (refer Figure 18.3).

Imported fill is assumed to be excavated from borrow pits located within a 5 km one-way haul distance from the centre of the plant site earthworks pads.

18.9 Mine Infrastructure

Mining is based on all site establishment, preparation and production related activities being undertaken by a suitably qualified and experienced mining contractor. These activities include, but are not limited to:

- Construction of all mining infrastructure required for undertaking mining operations such as workshops, warehousing, fuel and lubricants area, washdown facilities, administration building, crib rooms and ablutions;
- Construction of all mine haul roads and ore haulage roads from the satellite pits to the processing plant;
- Site preparation for pits, WRD's, roads etc. (i.e., clearing and grubbing of vegetation, removal and stockpiling of topsoil material);
- Pit-dewatering via in pit sump pumps;
- Establishment and maintenance of suitable surface water management infrastructure;
- Site rehabilitation and closure works such as re-profiling WRDs to final landform, rehandling and spreading stockpiled topsoil material etc.

Centamin will provide oversight and management of the mining contractors and undertake all technical requirements for the mining mobilisation, establishment and operation such as grade control, mine planning, mine surveying etc.

18.10 Administration and Processing Infrastructure

Administration, process plant buildings and other facilities have all been allowed for in the design and cost estimate, comprising:

- Gatehouse;
- Main administration office;
- Clinic and emergency response building;
- Administration mess and ablutions;
- Warehouse and stores;
- Plant security and change house;
- Plant office;
- Plant mess and ablutions;
- Sample preparation and metallurgical laboratory;
- Reagents storage sheds;
- Plant control room;
- Plant workshop; and
- Maintenance office and stores.

There are two main types of buildings being proposed for the project; steel portal frame type and prefabricated, flatpack modular construction type.

Offices and amenity area structures will generally be of prefabricated, flatpack construction, consisting of steel framed modules with sandwich panel walls, steel framed insulated roofs, and vinyl covered waterproof plywood floors. This building construction type was chosen for its high level of thermal insulation, cost-effectiveness and reduced site assembly requirement.

The workshop, warehouse, reagent stores and the sample preparation laboratory will be of steel frame construction, consisting of steel members assembled in portal frames with insulated or plain steel panels serving as both side cladding and roof cladding. The steel frames will be erected on reinforced concrete foundations and concrete floor slabs will be provided. The thickness of the floor slabs will vary to accommodate the loads associated with the building use. Equipment footings and structures constructed within the plant buildings will be independent of the building structure. Portal frame construction was selected to enable installation of gantry cranes where required for use during operations and to achieve the large column-free spaces required for the plant operations, storage and workshop buildings.

An area will be provided for the mining contractor to establish their offices, workshops and other facilities. Power, potable water and connection to the site sewerage facilities will be provided.

18.11 Accommodation Village

The accommodation village will be located approximately 3 km southwest of the process plant site. The village will consist of:

- Security camp, for 80 personnel; and
- Main camp, for 300 personnel.

The camps will be serviced by common utilities and services.

The security camp will be constructed initially, followed by the main camp.

18.11.1 Security Camp

The security camp will accommodate up to 80 personnel and include:

- 20 two-person accommodation units, with ensuite ablutions;
- 10 four-person accommodation units;
- 2 shower and ablution blocks;
- Kitchen, with cool room and freezer room;
- Dining room;
- Recreation room;
- Gym; and
- Security gatehouse.

18.11.2 Main Camp

The main camp will accommodate up to 300 personnel, and include:

- 20 one-person accommodation units, with ensuite ablutions;
- 48 two-person accommodation units, with ensuite ablutions;
- 46 four-person accommodation units;
- 7 shower and ablution blocks;
- Kitchen, with cool room and freezer room
- Dining room;
- Wet mess, with beer garden and ablution block;
- Recreation room;
- Gym;
- Laundry;
- Linen store and cleaning storeroom;
- Camp administration building, with first aid station and shop;

- Maintenance shed and compound;
- Security gatehouse;
- 2 multi-purpose sports courts;
- Football (soccer) field; and
- Swimming pool.

18.12 Sewage Treatment

Sewage and wastewater from the process plant, mine services area, administration complex and accommodation village will gravitate into collection pits where pumps will macerate the contents into a pulp and transfer to a modular wastewater treatment plant located at the accommodation village.

18.13 Fuel and Lubricants

The fuel storage and pumping system will be a vendor supplied package and be included within the fuel supply contract. The fuel facility will be located adjacent to the mine services area.

18.14 Communications

Centamin plan to engage with a suitable mobile service provider for their main communication connection with the Project, which will comprise a fibre backbone via a microwave link.

The main elements of the site communications system will include:

- External telephone/data/Internet services via a fibre optic network.
- A local mobile service provider will be contracted to upgrade existing facilities on site and provide a link into the local, national and international telecommunication network.
- A radio network will be established with dedicated operational, security and emergency channels.
- Internal voice and data communications will be established at the site data centre by a hardwired fibre and/or wireless connection.
- Communications to the remote facilities, including the TSF and WHD transfer pumps,) will utilise radio telemetry.
- Network connections and Wi-Fi coverage for administration, accommodation and infrastructure buildings.
- Closed-circuit television (CCTV) systems will be provided as part of the plant security system.
- Access control systems will be provided for infrastructure locations.
- A main datacentre and storage facility for the IT server infrastructure. This will include network switches and a firewall allowing for a Virtual Private Network (VPN) link.

All mining communications will be provided by the mining contractor. The exception is that Centamin will provide mobile and handheld radios for their own light vehicles and staff accessing the mining areas.

18.15 Security and Fencing

18.15.1 Process Plant Security

Two security levels will apply to the processing plant and supporting infrastructure areas.

The supporting infrastructure and facilities will be surrounded by a perimeter fence with a security gatehouse controlling the access to these areas.

Additional security will be provided for the processing plant area, which will be surrounded by a double fence separated by 15 m with a central 'no man's land' between the fences. The no man's land area will be cleared and grubbed, with lighting and closed circuit television (CCTV) cameras provided at regular intervals along the fence line for monitoring purposes. Access to and from the processing plant area will be controlled by a separate security and change house, with turnstile and search facilities. Vehicle access to the plant will be controlled via twin gates adjacent to the security and change house.

Fixed wire mesh panels with locked personnel gates will be installed around the gravity concentrators and the intensive leach reactor. CCTV cameras will be located to monitor for these areas.

The gold room will have access restricted to authorised personnel, with security guards providing 24 hour coverage. Access to the goldroom by authorised personnel or vehicle traffic will be via secured air locks. The gold room security system will include controlled access security and CCTV cameras.

18.15.2 Accommodation

The security and main camps will be each be perimeter fenced and include a security gatehouse, attended by security guards on a 24-hour basis.

18.15.3 Mining Areas

Active mining areas will be fenced off where required to ensure that communities and livestock are barricaded from accessing or entering into the pits. Waste rock will also be used as a method of barricading certain areas of the pit, particularly close to the pit exit and entry ramp areas. Blast exclusion zones will be managed during blast events.

TABLE OF CONTENTS

19. MARKET STUDIES AND CONTRACTS 1
19.1 Introduction 1
19.2 Contracts 1
19.3 Market Outlook 2
19.4 Gold Demand 2
19.5 Available Smelting and Refining Options and Costs 2
19.6 Payment Terms 2
19.7 Shipping 2

LIST OF TABLES

Table 19.1 Historical Annual Average LBMA Gold Prices (S&P Capital IQ) 1
Table 19.2 Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023) 2

19. MARKET STUDIES AND CONTRACTS

19.1 Introduction

Gold is a readily traded market that operates internationally. Major trading centres are located across all time zones with most global gold trading volumes passing through the London Over-The-Counter (OTC) market, US futures market and the Shanghai Gold Exchange. In 2023, the average London Bullion Market Association ("LBMA") gold price increased by 13% over the year closing at US$2,065/oz, having started the year at US$1,823/oz. Similarly, with annualised average prices, 2023 prices averaged US$1,943/oz versus an average price of US$1,802/oz in 2022. Prices rose steadily throughout the year with a low of US$1,810/oz and a high of US$2,079/oz. During the first half of 2024, gold prices have risen by 13% starting the year at US$2,065/oz and closing the half at US$2,326/oz.

Period	Annual Average US$/oz
2017	1,258
2018	1,269
2019	1,393
2020	1,771
2021	1,799
2022	1,802
2023	1,943
2024 H1	2,205

Table 19.1 Historical Annual Average LBMA Gold Prices (S&P Capital IQ)

The Study assumes that the Project would generate income from the sale of gold doré. The amount of gold produced from the LOM is 1,667 koz with further upside from ongoing drilling. Metallurgical testing to date has indicated recoveries ranging from the low 80's of high 90's, depending on material type and grade, with 89% used in the pit optimizations.

The gold price used for the reserve optimisation parameters was US$1,450/oz which is well below current market prices and price forecasts, it is also well aligned with industry norms for reserve pricing.

19.2 Contracts

For the Study, the strategy is to utilise the services of a mining contractor(s) for all mining activities on site. Contract terms are as per industry norms particular to the West African region where the Doropo deposit is located.

For the doré produced from the proposed Doropo treatment plant, in the absence of letters of interest or letters of intent from potential smelters or buyers of gold, general smelter terms for similar projects have been applied.

19.3 Market Outlook

Gold prices are expected to be supported by expectations of interest rate cuts by the major global economies and ongoing global geopolitical uncertainty and conflict, these are expected to sustain demand for safe haven assets such as gold from investors and central banks.

Period	Consensus Forecast US$/oz
2024	2,235
2025	2,297
2026	2,226
2027	2,148
2028	2,154

Table 19.2 Consensus Forecast Gold Prices (S&P Capital IQ as of 30/07/2023)

19.4 Gold Demand

Demand will ultimately be determined by the drivers mentioned in the Market Outlook section, with additional influence by fabrication for jewellery which is linked to economic strength and pricing.

19.5 Available Smelting and Refining Options and Costs

A typical gold refinery rate of US$4.00/oz freight plus insurance charge has been used based on recent comparable studies.

19.6 Payment Terms

A typical refinery gold payable rate of 99.95% has been used based on recent comparable studies.

19.7 Shipping

Bullion transportation is included within the refining rate.

TABLE OF CONTENTS

20. ENVIRONMENTAL AND SOCIAL ASSESSMENT 1
20.1 Introduction 1
20.2 Legislative Requirements, Guidelines and Corporate Commitments 6
20.2.1 National Authorities and Agencies 6
20.2.2 Ivoirian Environmental and Social Legislation 6
20.2.3 Ivoirian Permits and Approvals 7
20.2.4 Corporate ESG Policies and Standards 8
20.2.5 Project Discharge Standards 9
20.3 Baseline Setting 10
20.3.1 Physical 10
20.3.2 Socio-economic 14
20.3.3 Ecology 19
20.4 Management Systems/Approach 23
20.5 Physical Environmental Impacts, Management and Mitigation 28
20.5.1 Open Pits, Mining and Waste Rock 28
20.5.2 Mineral Processing and Tailings 28
20.5.3 General and Hazardous Waste 28
20.5.4 Water Supply and Quality 28
20.5.5 Erosion and Sediment 29
20.5.6 Air Quality 29
20.5.7 Climate and Greenhouse Gas Emissions 30
20.5.8 Noise, Vibration and Flyrock 30
20.5.9 Visual Amenity 31
20.6 Socio-economic Impacts, Management and Mitigation 31
20.6.1 Socioeconomic 31
20.6.2 Land Acquisition, Livelihood Restoration and Resettlement 33
20.6.3 Transport and Traffic Safety 36
20.6.4 Community Health and Safety 37
20.6.5 Artisanal Gold Mining 37
20.6.6 In-migration 38
20.6.7 Community Development Fund 38
20.7 Biological and Ecological Impacts, Management and Mitigation 39
20.8 Archaeology and Cultural Heritage 41
20.9 Cumulative Impacts 41
20.10 Stakeholder Engagement 42

20.11 Occupational Health and Safety 42
20.12 Emergency Preparedness and Response 43
20.13 ESMMP Framework 43
20.14 Rehabilitation and Mine Closure 44
20.15 Resettlement and Livelihood Restoration 45
20.16 Conclusions 45

LIST OF FIGURES

Figure 20.1 Main Project Development Area (MPDA) 4
Figure 20.2 All proposed Project Development Areas, Including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs 5
Figure 20.3 Topography over the Doropo Gold Project Area 11
Figure 20.4 Doropo Gold Project Location and Administrative Boundaries 14
Figure 20.5 Hand Dug Well Lagbo 17
Figure 20.6 Hand Pump Dupindouo 17
Figure 20.7 Lagbo Health Centre (H&B, May 2023) 17
Figure 20.8 Holidouo Community School (H&B, May 2023) 17
Figure 20.9 Shrub Savannah (Wet Season) 19
Figure 20.10 West African Bowal (Dry Season) 19
Figure 20.11 Gallery Forest (Wet Season) 19
Figure 20.12 Swampy Thicket (Wet Season) 19
Figure 20.13 Patas Monkey (Erythrocebus patas) 21
Figure 20.14 Bushbuck (Tragelaphus scriptus) 21
Figure 20.15 Dwarf Crocodile (Osteolaemus tetraspis) 21
Figure 20.16 C.rouxae 21

LIST OF TABLES

Table 20.1 Key Studies Conducted to Support the Project Feasibility Studies and Statutory ESIA 2
Table 20.2 Key Air Quality, Noise and Water Standards, Guidelines and Legal Requirements 10
Table 20.3 Project Design Criteria 27
Table 20.4 Land Cover within the Doropo PDAs 35

20. ENVIRONMENTAL AND SOCIAL ASSESSMENT

20.1 Introduction

The development of the Doropo Gold Project (Project) requires an Environmental Permit and Mining (Exploitation) Permit in line with Ivoirian legislation. Earth Systems and H&B Consulting were commissioned by Ampella to review the environmental and social aspects of the Project and prepare an Environmental and Social Impact Assessment (ESIA) in compliance with key Ivoirian regulatory requirements, and in accordance with international best practice.

An environmental and social baseline has been established for the Project with extensive field studies undertaken by the ESIA consultants since February 2022 to support Project Prefeasibility and Feasibility design studies as well as the statutory ESIA. These studies have included those related to socio-economic conditions, land and water use, surface and groundwater resources, terrestrial and aquatic ecology and biodiversity, air quality, noise and vibration, traffic and transportation, as well as archaeology and cultural heritage (0).

Specialist Study	Study Lead
Initial Baseline Monitoring and Modelling Studies	
Socio-Economic, Land and Water Use Baseline Study comprising an initial scoping visit in October 2021; two data collection missions undertaken in February and May of 2022	H&B Consulting (Dr ADOU Djané -sociology specialist) and Earth Systems
Terrestrial Ecology Baseline Study comprising a dry season study in February and wet season study in May 2022	H&B Consulting (Pr KASSI N'Dja Justin - botany specialist and Dr AKPATOU Bertin - fauna specialist) and Earth Systems
Aquatic Ecology Baseline Study comprising an initial site scoping visit in February 2022 and a wet season study in October 2022	H&B Consulting (Pr EDIA Oi Edia - aquatic ecology specialist) and Earth Systems
Groundwater and Surface Water Quality Baseline Monitoring and Modelling initiated in August 2022 and ongoing	Earth Systems and H&B Consulting (Pr YAO Koffi Blaise - hydrology and hydrogeology specialist)
Archaeology and Cultural Heritage Baseline Study comprising two field missions in February and May 2022	H&B Consulting (Dr YAO Narcisse - archaeology specialist)
Meteorological and Climate Review in 2022	Earth Systems
Geochemical Analysis Program for the characterisation of waste rock, including static and kinetic testing in 2022	Piteau Associates

Specialist Study	Study Lead
ESIA Baseline and Modelling Studies	
Socio-Economic, Land and Water Use Baseline Studies carried out in May, July and October 2023	H&B Consulting (Dr ADOU Djané - sociology specialist) and Earth Systems
Terrestrial Ecology Baseline Studies comprising a wet and dry season field mission in 2023 (i.e. during the end of the dry season in May 2023 and middle of the wet season in August 2023)	H&B Consulting (Pr KASSI N'Dja Justin - botany specialist and Dr AKPATOU Bertin - fauna specialist) and Earth Systems
Aquatic Ecology Baseline Studies comprising a wet season study in September 2023.	H&B Consulting (Pr EDIA Oi Edia - aquatic ecology specialist) and Earth Systems
Camera trap protocol carried out during the months of September, October and November 2023 in order to confirm the presence/likely absence of IUCN Red listed threatened or potentially critical habitat qualifying species	H&B Consulting (Dr AKPATOU Bertin - fauna specialist)
Soils and Land Capability Study carried out in September 2023	H&B Consulting (Dr Yéboua Firmin KOUASSI - pedologist) and Earth Systems
Meteorological and Climate Review	Earth Systems
Air Quality, Noise and Vibration Baseline Monitoring and Modelling during the wet and dry season of 2023 (i.e. June 2023 and October 2023)	Earth Systems
Traffic, Transport and Accessibility Baseline Study conducted in May 2023	H&B Consulting (M. TOA Bi Gohouré Charles - traffic and transportation specialist)
Archaeology and Cultural Heritage Baseline Study carried out in May 2023	H&B Consulting (H&B Consulting (Dr YAO Narcisse - archaeology specialist)

Table 20.1 Key Studies Conducted to Support the Project Feasibility Studies and Statutory ESIA

The Project development will consist of eight open mine pits, one processing plant, one Tailings Storage Facility, waste rock dumps adjacent to each pit, accommodation and office facilities. The development of the Project will be phased, commencing with a 'Main' Project Development Area (MPDA) with extension to the Kekeda, Han, Enioda, and Kilosegui Satellite Pits in subsequent years. All deposits are located within a 7 km radius of the MPDA, with the exception of the Kilosegui Pit located approximately 30 km to the south-west, and will be connected by ore haul roads.

The ESIA consultants have worked with Ampella in examining the environmental and social impacts associated with various project design and layout alternatives for the Project as part of the ESIA process. Significant changes have been made to the Project layout in the design process which has allowed potential negative impacts on environmental and social values to be minimised and positive impacts to be strengthened.

Project Development Areas (PDAs) have been defined to encompass key elements of Project infrastructure, including appropriate safety and security buffer zones of approximately 500 m from pit shell extents. The five proposed PDAs are shown in Figure 20.1 and Figure 20.2. PDAs have been carefully designed to minimise potential environmental and social impacts. Proposed PDAs comprise of:

- One, larger, 'Main Project Development Area' (MPDA), containing three resource deposits (Souwa, Nokpa, Chegue) and the primary mine infrastructure (i.e. TSF, WSD, process plant, mine services area and accommodation camp);
- Four separate PDAs for each of the satellite deposits (Kekeda, Han, Enioda, Kilosegui), their respective WRDs and buffer zones. Each satellite PDA will be connected to the Main PDA via ore haul roads.

Some mining areas will be mined for less than a 2-year period prior to rehabilitation and closure. Ampella will require full control over land access within the PDAs, to maintain Project security and community safety. Ampella will acquire rights of use to the land within PDA boundaries from host communities.

A number of mine components and supporting facilities will be established outside PDA boundaries. These include the Water Harvest Dam (WHD), a sediment pond downstream of the MPDA, the accommodation camps, the explosives magazine and emulsion plant, and the airstrip.

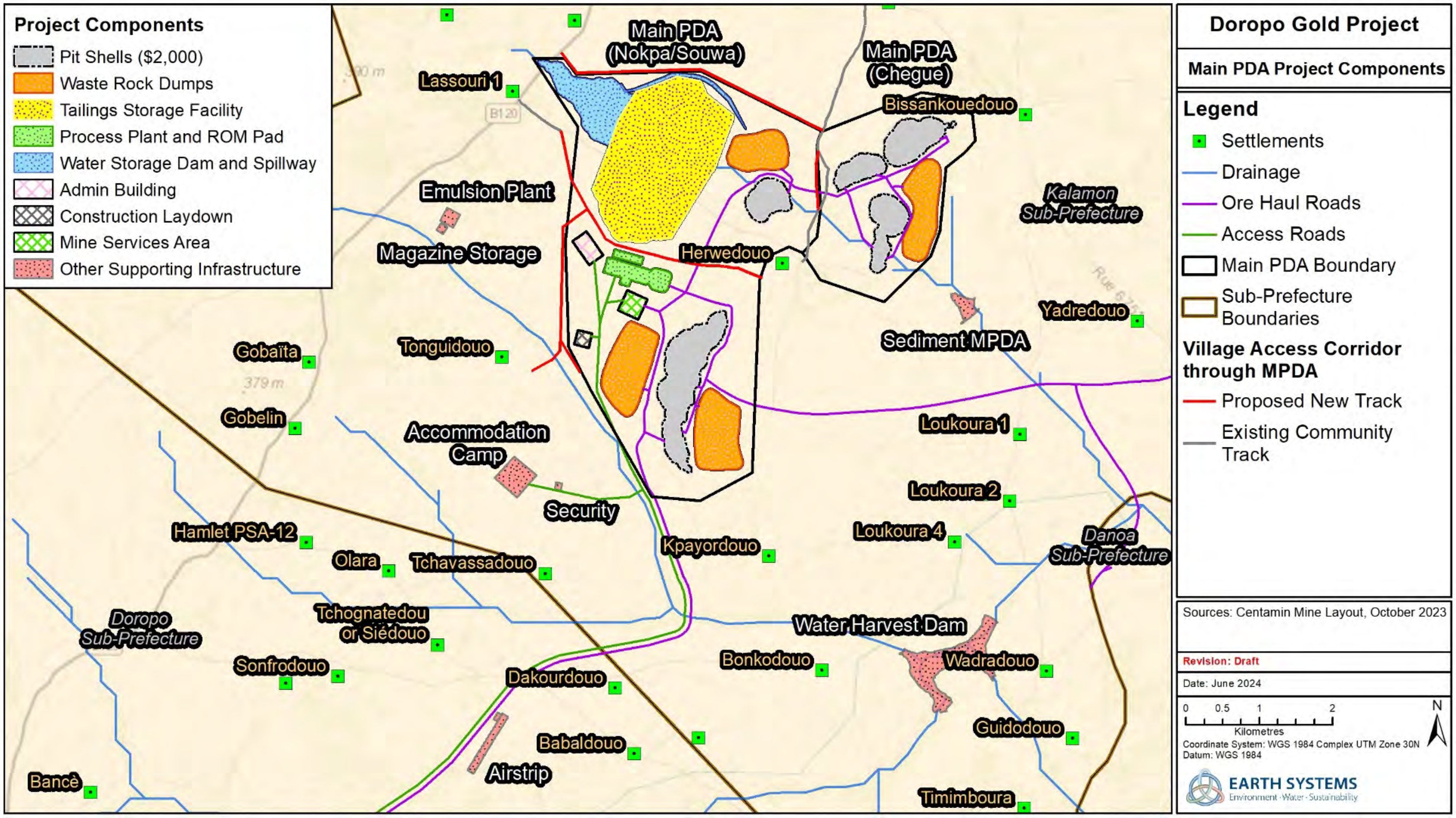


Figure 20.1 Main Project Development Area (MPDA)

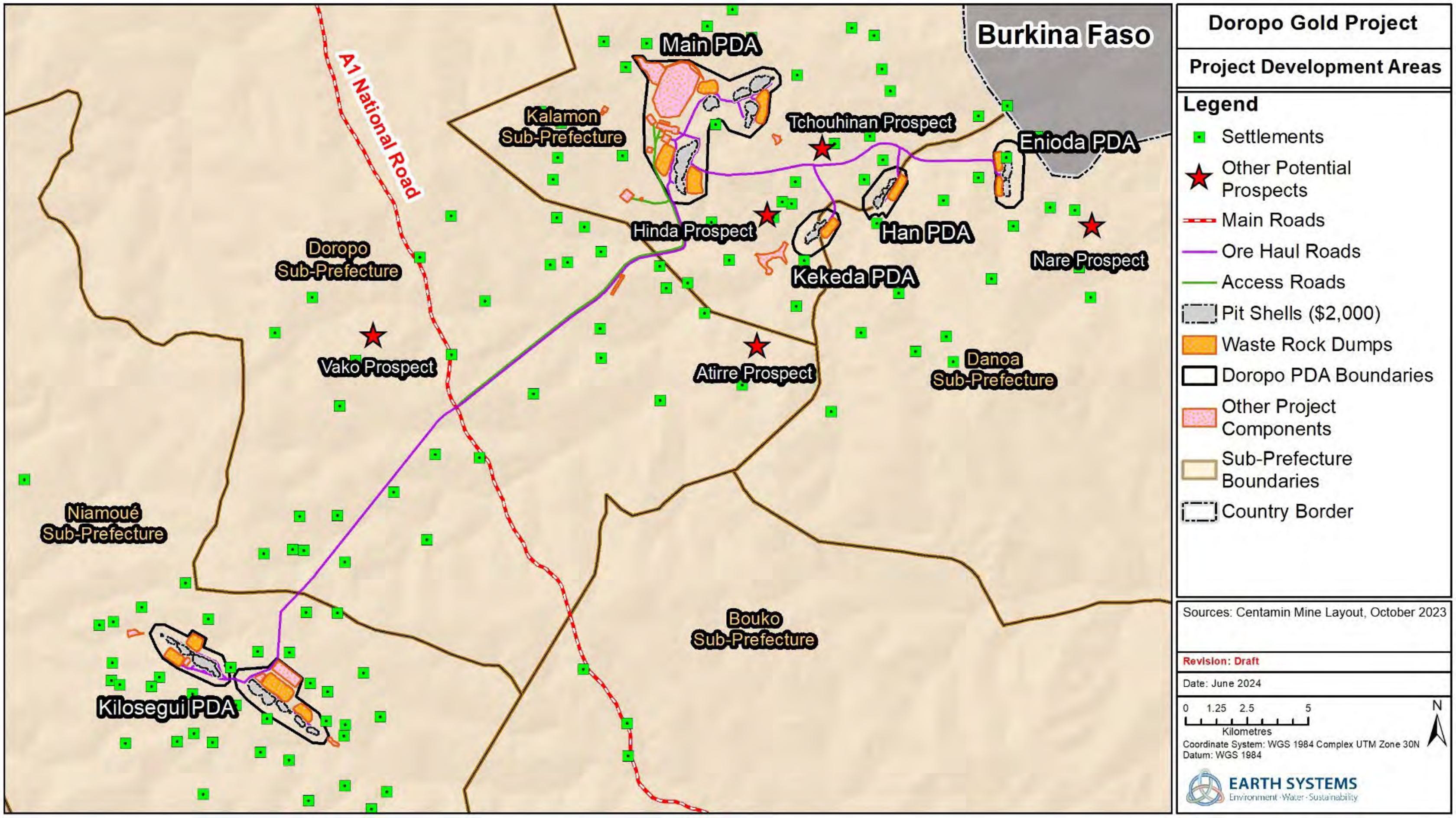


Figure 20.2 All proposed Project Development Areas, Including the MPDA and Han, Kekeda, Enioda and Kilosegui Satellite Pit PDAs

20.2 Legislative Requirements, Guidelines and Corporate Commitments

20.2.1 National Authorities and Agencies

The main government agency involved in the environmental permitting process is the ANDE (l'Agence Nationale de l'Environnement), which falls under the Ministry of Environment, Sustainable Development and Ecological Transition. ANDE is the central authority responsible for overseeing national ESIAs from initial application/selection, scoping, assessment, review and decision making on permit approval. The final decision on permit approval rests with the Minister of the Environment (Decree 96-894, Article 19). Within the ANDE, there are four sub-directorates, including one for EIA and another for project planning, monitoring and review. At the regional level, Ministry of Environment, Sustainable Development and Ecological Transition services are provided by the offices of the regional directorates and departmental directorates.

Another government agency responsible for environmental management and the registration of hazardous facilities is the Centre Ivoirian Anti-pollution (CIAPOL). CIAPOL is responsible for monitoring pollution levels in water, soil and air.

20.2.2 Ivoirian Environmental and Social Legislation

Key Ivoirian legislation relevant to the environmental and social permitting of the Project include:

- Mining Code (2014);
- Environment Code (1996);
- Water Code (1998);
- Forestry Code (2019);
- Labour Code (2015); and
- Social Security Code (1999).

The Mining Code (2014) is the primary legislation regulating the mining industry in Cote d'Ivoire that was promulgated in 2014 (Law No. 2014-138 of 24 March 2014) and covers both exploration and mining activities.

The Environment Code is the principal environment law in Côte d'Ivoire and supplemented by several decrees including the Decree No. 96-894 (08 November 1996) specifying rules and procedures for studies related to the environmental and social impact.

20.2.3 Ivoirian Permits and Approvals

Environmental Permit

The adoption of Law No. 96-766 of 3 October 1996 on the Environmental Code and the promulgation of Decree No. 96-894 of 8 November 1996 on the rules and procedures applicable to environmental impact assessments in the Republic of Côte d'Ivoire, require that mining projects are subjected to an ESIA. The requirement for an ESIA to be undertaken is also stipulated under Article 141 of the Mining Code.

The ESIA Report was submitted to the ANDE in early 2024 for approval by the Mining Administration, Environment Administration (ANDE) and other competent authorities provided for by the mining legislation. The ESIA Report was prepared in accordance with the Terms of Reference (TOR) no.101-0423/wp provided by the ANDE to Ampella in April 2023.

The ESIA review by government and the public has been completed and a final ESIA Report incorporating the results of public consultation and comments from the Inter-ministerial Technical Committee, was subsequently accepted by the ANDE on the 10th of April 2024.

Ampella has been granted an Environmental Permit from the Ministry of Environment, Sustainable Development and Ecological Transition with receipt of Decree No.000322/MINEDDTE/ANDE of June 13, 2024 approving the Environmental and Social Impact Assessment (ESIA) of the Project and stipulating the following terms:

- Article 2: Ampella is required to fulfil commitments as detailed in the Environmental and Social Management Plan (ESMP) attached to the ESIA;
- Article 3: The Decree is limited in scope to the environmental requirements while noting the Project is subject to the approval of the Ministry of Mines;
- Article 4: The National Agency of the Environment (ANDE) is responsible for ensuring compliance with the environmental regulations and shall have access to the Project facilities for the purpose of monitoring;
- Article 5: Applicable processes in the event of regulatory non-compliance;
- Article 6: Any deviation from Project design that may arise during Project implementation shall be brought to the attention of ANDE for prior approval;
- Article 7: Ampella is responsible for any damaged caused to the environment including restoration measures in accordance with regulatory provisions;
- Article 8: The Decree becomes null and void if the Project is not implemented within a period of 3 years from date of signature;
- Article 9: Ampella is subject to a regulatory environmental audit 3 years from the date of signature;

- Article 10: Ampella shall notify ANDE of the effective start date for the Project i.e. construction phase. Ampella is required to prepare semi-annual reports on the implementation of the ESMP.

Mining Permit

The Ministry of Mines, Petroleum and Energy will be responsible for granting the Doropo Mining Permit. Application for the Mining Permit requires the provision of a Feasibility Study the content of which is defined in Article 28 of the Mining Code as follows and which includes:

- A study of the socio-economic impact of the Project;
- A study of the environment impact (land, water, air, fauna, flora and human settlements) impact of the project with appropriate recommendations in accordance with the code of the environment and its subordinated documents; and
- A Community Development Plan.

Ampella understands that there is no legal requirement for a Resettlement Action Plan (RAP) or Livelihood Restoration Plan within the Ivoirian Mining Code but has chosen as part of the ESIA to develop a Resettlement and Livelihood Restoration Framework (RLRF) which documents the land and displacement impacts expected for this Project.

Other Permits and Authorisations

A number of additional permits and authorisations will be acquired for the Project prior to Construction and/or Operation to meet Ivoirian regulatory requirements:

- Establishment of a hazardous (classified) facility;
- Establishment of a Hydrocarbon storage facility;
- Authorisation for the use of explosive substances;
- Authorisation for the import and use of hazardous chemicals.

20.2.4 Corporate ESG Policies and Standards

Ampella is committed to meeting international standards of good practice in the areas of environmental protection, social development, and health, safety and security. Ampella has adopted a set of Policies that outline its key Health, Safety and Environment (HSE) and Social Responsibility objectives. Key policies relevant to Project include:

- Code of Conduct (2022);
- Social Responsibility Policy (2021);
- Environment Policy (2021);
- Energy and Climate Change Policy (2023);

- Tailings Management Policy (2024);
- Human Rights (2021);
- Safety, Health and Wellbeing Policy (2021); and
- People Policy (2021).

Ampella's policies align with their values and international good practice, in particular the Responsible Gold Mining Principles (RGMPs) of the World Gold Council. Centamin sets key performance indicators (KPI) each year and assesses performance against these benchmarks on a regular basis. Ampella is also committed to the Global Industry Standard on Tailings Management (GISTM) with the objective to cause no harm to people or the environment through tailings facility design, operation and closure.

20.2.5 Project Discharge Standards

The Project will need to consider and comply with:

- Discharge and emissions guidelines for potential off-site releases of water, waste and airborne contaminants; and
- Ambient guidelines for the protection of environmental values (e.g. protection of aquatic fauna and fisheries, drinking water, etc.).

A list of key Ivoirian standards for the Project, as well as international standards, is presented in Table 20.2. Where standards or limits do not exist in Ivoirian Law, guidelines, standards or limits used by other countries (e.g. EU, USEPA, UK etc.) or organisations (e.g. IFC, WHO etc.) will be adopted in lieu.

Source	Relevant Guidelines, Standards and Legal Requirements	Year
Waste/Wastewater Discharge and Monitoring		
Côte d'Ivoire	Order no. 01164/MINEEF/CIAPOL/SDIIC of 4 November 2008, on the Regulation of Discharges and Emissions from Installations Classified for Environmental Protection	2008
World Bank	Environmental Health and Safety Guidelines - Mining	2007
	Environmental Health and Safety Guidelines - General - Environmental	2007
	General EHS Guidelines: Wastewater and Ambient Water Quality	2007
ICMI	International Cyanide Management Code	2021
Air Quality		
Côte d'Ivoire	Decree no. 2017-125 of 22 February 2017, on air quality	2017
World Bank	Environmental Health and Safety Guidelines for Mining	2007
	General EHS Guidelines: Air Emissions and Ambient Air Quality	2007
WHO	Air Quality Guidelines - Global Update	2021
Soil Quality		
UK	Soil Guideline Value	2009

Source	Relevant Guidelines, Standards and Legal Requirements	Year
Aquatic Fauna/Fresh Waters		
Côte d'Ivoire	N/A	N/A
United States	National recommended water quality criteria; republication. United States Environmental Protection Agency (USEPA)	2009
European Union	Directive 2008/105/EC of the European Parliament and of the Council of 16 December 2008 on environmental quality standards in the field of water policy, amending and subsequently repealing Council Directives 82/176/EEC, 83/513/EEC, 84/156/EEC, 84/491/EEC, 86/280/EEC and amending Directive 2000/60/EC of the European Parliament and of the Council	2008
European Union	Directive 2006/44/EC of the European Parliament and of the Council of 6 September 2006 on the quality of fresh waters needing protection or improvement in order to support fish life	2006
Drinking Water		
Côte d'Ivoire	N/A	N/A
WHO	Guidelines for Drinking Water Quality, fourth edition	2011
European Union	Council directive 9883/EC of November 1998 on the quality of water intended for human consumption	1998
Noise and Vibration		
Côte d'Ivoire	Decree 2016-791 and Noise Standards No. 01164, 4 November 2008	2008
World Bank	Environmental Health and Safety Guidelines for Mining	2007

Table 20.2 Key Air Quality, Noise and Water Standards, Guidelines and Legal Requirements

20.3 Baseline Setting

20.3.1 Physical

The Project Area lies in the Sudanian bioclimatic zone and is characterised by a typically sub-tropical climate, with relatively high, uniform temperatures and a distinct wet season (April to October) and dry season (November to April) resulting from the annual movement of the Inter Tropical Convergence Zone (ITCZ). Long-term mean annual rainfall data 2000-2021 was recorded at 911 mm with a standard deviation of ±223.1 mm at the nearby Gaoua station, located approximately 63 km north of the Project. The wind direction experienced throughout the wet season is in the south-westerly direction in the wet season and north-east in the dry seasons, when the 'Harmattan' winds are experienced.

Although temperatures can vary on a day-to-day basis, mean monthly temperatures generally range between 25°C and 34°C. Maximum mean monthly temperatures range between 29°C and 38°C, and minimum mean monthly temperatures range between 20°C and 24°C.

The natural landscape surrounding the Project is characterised by relatively subdued relief, with typically sandy surface soils and a rare presence of rocky outcrops. The eastern and western sides of the area are bound by greenstone belt hill chains, while large peneplains bound the Project to the north and south.

Elevations range from about 250 m to 420 m above sea level, with the highest points forming a drainage divide between the Comoe basin to the west and the Black Volta basin to the east. The topography of the area is gently undulating, with average slopes of the various pit catchments ranging between 0.4% and 4.0%. The main water course, the Pouene River, flows from southwest to northeast through the Project footprint.

The regional hydrogeology is characterised by a crystalline basement environment comprising the weathered residual overburden (the regolith), the transition zone between the bedrock and the regolith, and the fractured bedrock. Unweathered and non-fractured basement rocks (typically granodiorites and granites at Doropo) are generally considered to contain negligible quantities of groundwater, while fractured areas of bedrock may host some localised groundwater flow. The basal section of regolith and the deeply weathered bedrock are generally considered to have the highest yield of groundwater.

The Project Area groundwater levels are generally deeper than the regional average but are shallow when compared to the weathering depth, indicating that a significant proportion of saprolite and saprock are saturated. The shallowest groundwater levels occur near the Nokpa deposit (between 2 and 4 m), while the deepest levels are in the Souwa (10 - 22 m) and Enioda (12 - 17 m) pit areas.

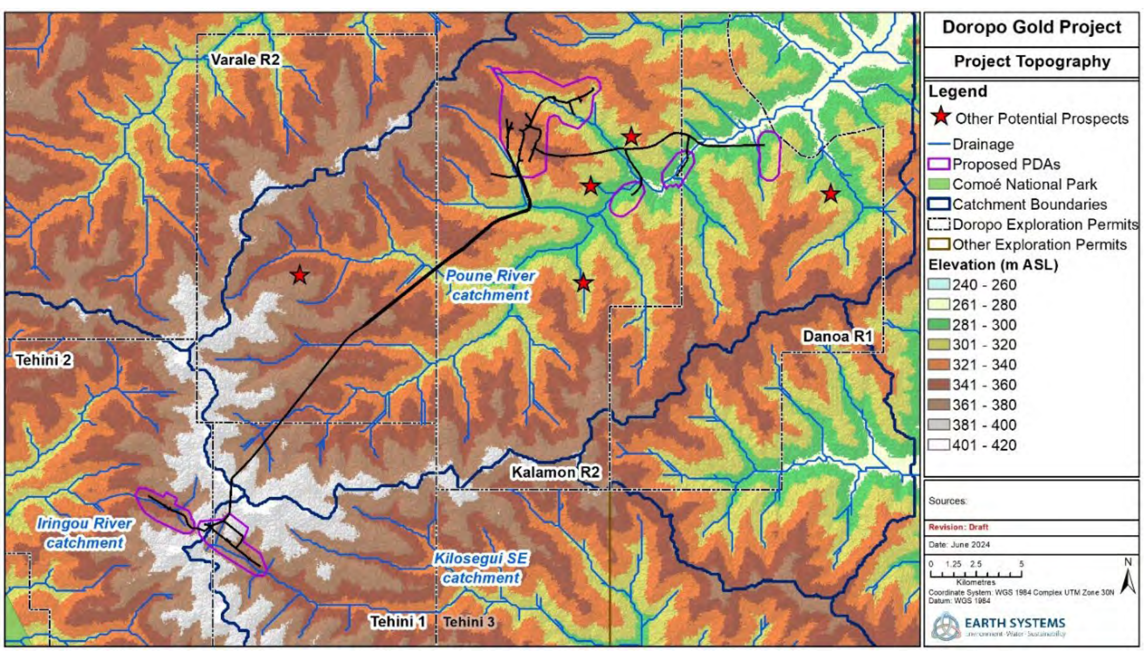


Figure 20.3 Topography over the Doropo Gold Project Area

The majority of the Project footprint is situated within the Black Volta subbasin which forms part of the wider Volta Basin. The Black Volta is a perennial river that discharges into the Gulf of Guinea in the Atlantic Ocean. The Project footprint is situated within three separate surface water catchments. These include:

- The Pouene River catchment in which the entire Main PDA and most of the satellite pit areas will be located. The Pouene River drains to the northeast into the Black Volta River;
- The drainage from the southeast section of the Kilosegui deposit drains to the Black Volta River;
- The drainage from the northwest section of the Kilosegui deposit drains to the Iringou River catchment, a tributary of the Comoé River that flows through the Comoé National Park.

The Pouene River is non-perennial with flow occurring during the wet season as late as December. Surface water drainage within the other Project catchments is ephemeral with water flow occurring only in response to storm events during the wet season.

The physical-chemical analysis of surface water indicates that whilst contaminated with e coli it is generally otherwise of good quality, with near neutral pH, low to elevated concentrations of suspended solids and elevated organic matter, iron and phosphorus concentrations; likely due to geological factors and human land use.

Baseline groundwater quality was assessed based on monitoring data collected from sites representing community boreholes and Project exploration boreholes. Groundwater in the Project area was found to generally comply with WHO standards, but elevated concentrations of coliform bacteria was observed at some community water points. The pH of groundwater ranges between 6.44 and 8.28 and electrical conductivity varies between 123 - 809 across sampled locations.

The Bounkani Region experiences challenges related to vegetation and erosion, with natural ground cover observed to be generally sparse, and cultivated plots demonstrated reduced ground vegetation density.

Variations in soil composition occur between different topographical areas, with mid and lower slope soils offering more favourable conditions for cultivating larger crops, owing to their deeper soil profile. Transported soils and clays are present in the lower valleys and drainage systems possessing a greater Cation Exchange Capacity and higher water-soluble ion and plant available nutrient concentrations. The Project area is predominantly suited for the cultivation of food crops, particularly rice, due to a much higher clay content and limited drainage capacity.

By contrast, sandy lateritic soils are dominant in the highlands and are highly leached due to the continuous alternating dry and wet seasons. The soils are acidic and crumbly in texture, with most soluble silica and soluble minerals removed, leaving only insoluble ions (iron and aluminium), giving the soil a rusty red hue.

Lack of soluble ions (nitrogen, potassium, calcium) make soils in higher areas poor for growth, and the sandy texture means there is poor water retention.

The main sources of existing air emissions in the Project Area include particulates from dust, smoke, pollen and spores distributed by seasonal climatic conditions; local vehicular traffic exhaust and dust emissions; open cooking using firewood or fossil fuels; and biomass burning. The ambient particulate readings from air quality monitoring in the Project Area indicate that baseline concentrations of PM10 during the dry season consistently exceed the WHO guidelines (2021) and Côte d'Ivoire Standards. Although the PM10 concentrations are lower during the wet season, there is still a significant number of recordings above the WHO guidelines for 24-hour mean and Côte d'Ivoire Standards.

Generally, monitored gases are below the WHO (2021) guidelines. SO_2 and NO_2 concentrations were higher during the monitoring period in October 2023 than June 2023, which may be due to the lightning and woodsmoke in the area. The most common pollutant gases present across the region in varying concentrations are:

- Sulfur dioxide (SO_2): derived from the combustion of fossil fuels and some industrial processes;
- Nitrogen oxides (NO_x): from bushfires, or poorly maintained vehicles or generators;
- Carbon dioxide (CO_2): produced by natural processes (i.e. cellular respiration) and the burning of firewood and fossil fuels;
- Ozone (O_3): associated with NO_x, biomass burning, and is naturally produced by lightning.

Ambient noise levels recorded in June and October 2023 were generally below WHO and Côte d'Ivoire guidelines for daytime LAeq noise of 55 db(A) and 60 dB(A) respectively and nighttime LAeq noise of 45 dB(A) at all monitoring locations. The main sources of existing noise emissions in the vicinity of the Project Area include:

- Vehicle and motorcycle use;
- Water pumps and motorised machines;
- Generators used for electricity generation;
- Domestic animals, birds, wildlife and insect activity; and
- Thunder and rain.

All baseline vibration monitoring sites detected standard background levels of surface and near surface seismic waves. In addition to background tectonic micro seisms, potential sources of vibration in the data may include human activity nearby or wind and other natural atmospheric phenomena. The vibration levels are in general so low that individual sources cannot be determined, due to the lack of development in the Project area. Existing baseline vibration in the vicinity of the Project includes:

- Vehicle and motorcycle use, generators used for electricity generation, water pumps, and construction activity;
- Artisanal mining - including rock crushers, vehicle use; and
- Natural vibration - thunder, micro seismic activity.

20.3.2 Socio-economic

Administration and Governance

The Project is located in the North-Eastern part of Côte d'Ivoire, in the Bounkani Region, approximately 480 km North-East of the capital city of Abidjan. The Project lies within the administrative boundary of the Prefectures of Doropo and Bouna; and the Sub-Prefectures of Doropo, Kalamon, Danoa and Niamoué.

The Region is characterized by a customary system of land governance held by the Koulango authorities, according to whom land resources are the exclusive property of the King of Bounkani. Within the Project area, legal land title has yet to be implemented and the customary system of land tenure is the only established land management mechanism. Villages are administered by a chief who is responsible for day-to-day affairs and is assisted by a council of public figures or 'notables'. Other relevant figures in the village governance structure are traditional leaders of cultural heritage practices and representatives of women and youth.

The eastern extent of the sub-prefectures of Kalamon and Danoa define the international border between Côte d'Ivoire and Burkina Faso. The Main PDA is located approximately 10 km southwest of the border, while the Enioda deposit is located less than 1 km away at its northern extent.

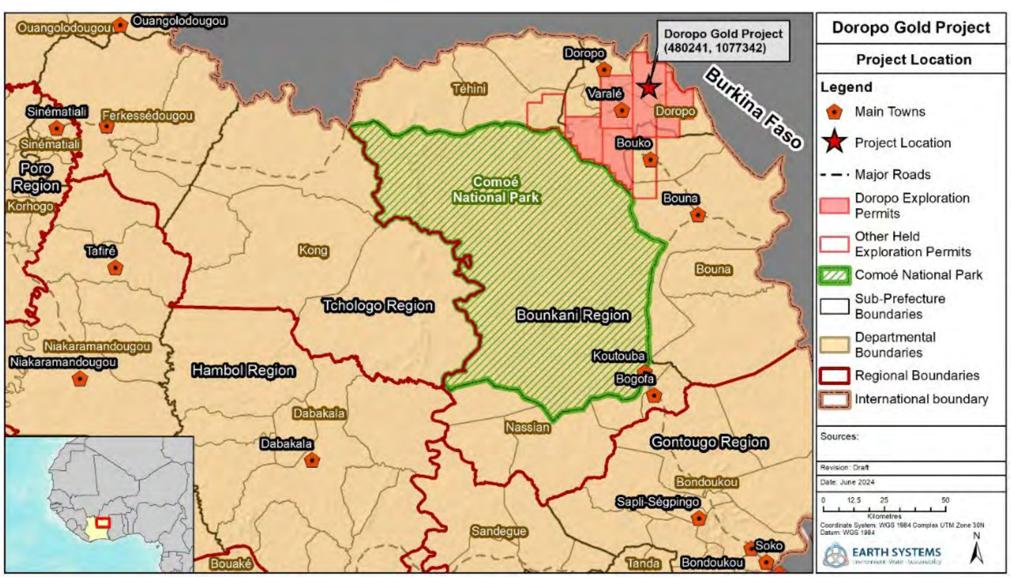


Figure 20.4 Doropo Gold Project Location and Administrative Boundaries

Population and Demographics

The population within the Project area is largely of Lobi ethnicity who have migrated to the area from Burkina Faso primarily over the last fifty years. Other ethnicities present include Koulango, Mossi, Lohron, Mauritanian, and Malinké.

A total of 26 villages were identified in the Project's main area of influence. Of these, a total of 18 settlements were identified to own land within PDA boundaries including: Herwedouo, Bissankouedouo, Lassouri, Tonguidouo, Gola, Holidouo, Fafoudouo, Lagbo, Béhindjinadouo, Wadaradouo, Dupindouo, Gangata, Gbelta, Fangadouo, Legitedouo, Mabridouo, Norfadouo, and Simatedouo.

Surveys completed in May 2023 identified the village of Holidouo, comprising 33 households (314 persons), physically located within the boundaries of a Project PDA. Holidouo village will require full resettlement due to development of the Enioda PDA. No other PDAs are expected to require relocation or resettlement of local villages, although further work will be required to ensure further resettlement is not required due to economic displacement.

Livelihoods and Income

The primary livelihood for most households consists of commercial agriculture and subsistence farming, with an average income of a typical household reported to be $100-200 per month. The main commercial crop grown for income is cashew and to a lesser extent yam. Other crops and vegetables are also grown on a smaller scale in market gardens, such as aubergines, tomatoes, chillies and okra. Agriculture is almost exclusively rain fed, and changes to weather patterns and soil fertility are of considerable impact and concern.

Illegal artisanal and small-scale mining (ASM) is an additional source of income undertaken by many residents within the Project area. Villages where ASM is recorded as a significant income source include Lagbo, Herwedouo, Wadaradouo, Bissankouedouo and Holidouo.

The two main animal husbandry systems practiced within Côte d'Ivoire are nomadic/semi-nomadic pastoralism and sedentary herding. Livestock in the Project area include cattle, sheep, poultry, goats, pigs, ducks, and turkeys. Cattle herders reported increasing challenges associated with the expansion of crops that reduce land availability for livestock grazing.

Small businesses exist in larger villages and areas where ASM is practiced, such as Lagbo and Herwedouo. Businesses include shops, services such as mechanics and hairdressers, and hospitality such as restaurants and bars.

The collection of NTFPs is a traditional para-agricultural activity that involves various plant products such as *Parkia biglobosa* (néré), *Vitellaria paradoxa* (shea) and *Adansonia digitata* (baobab) fruits. The processing of shea nuts is a supplementary source of income for women in most villages in the Project area. Harvested shea nuts are manually ground to extract the oils and further processed to produce various products for sale.

Community Assets and Infrastructure

There are 14 modern health centres in the Department of Doropo, however, they are constrained by poor facilities and low service capacity. The Doropo Urban Health Centre is the only facility in the district that provides advanced treatments and surgeries. Villages have reported travelling into Burkina Faso for medical treatment. The main challenges reported by village-level health centres include basic maternity facilities; no means of transport to Doropo for emergency medical complications; no funding to pay for electricity, forcing staff to pay out of pocket; and a lack of resources for testing and treatment of malaria.

Access to water varies considerably, with no municipal or piped water supply systems present in any of the PDA villages. Water sources for most villages comprise of a combination of groundwater and surface water for human, livestock and agricultural use. Reliance on surface water is particularly prevalent within villages surrounding the Kilosegui PDA. Many of the water resources are reported to be unreliable, poor quality, or insufficient to meet the needs of the villages.

In the communities surrounding the MDPA, Han, Enioda, and Kekeda PDAs there are six primary schools and four under construction. One school is near to the Kilosegui PDA. Secondary education is available in the towns Doropo, Saye, Danoa and Kalamon, and higher education is available in the city of Bouna.

The need for additional schools or higher quality education was listed as a top development priority for most villages during focus group discussions. Access to education is constrained by: long travel distances; poor quality of infrastructure and lack of equipment; availability of teachers; cost of school fees and materials; poor health; and the requirement for children to support the family with livelihood activities. Girls are disproportionately disadvantaged due to requirements to support with family activities, menstrual stigmatisation and de-prioritisation of education.

Figure 20.5 Hand Dug Well Lagbo

Figure 20.6 Hand Pump Dupindouo

Figure 20.7 Lagbo Health Centre (H&B, May 2023)

Figure 20.8 Holidouo Community School (H&B, May 2023)

Community Health, Safety and Nutrition

The main causes of mortality in Côte d'Ivoire were communicable, maternal, neonatal, and nutritional diseases. A malnutrition rate of 46% among children under the age of five was recorded by the Doropo Health District and high infant mortality. Testing and medication for malaria are free from local government health centres, however, supply is not always reliable.

HIV/AIDS was not indicated as a particular health concern by local health authorities, however Côte d'Ivoire has the highest prevalence of HIV/AIDS in West Africa, with an estimated 1.8% of people between the ages of 15 - 49 living with HIV. The lack of knowledge and social stigma of HIV in rural areas are constraints to patients being diagnosed.

ASM is often practiced by the youth and is associated with many acute and chronic health risks including physical injury from poorly maintained machinery; toxin poisoning such as mercury; and silica dust exposure leading to acute respiratory problems.

Traffic and Transport

The Project is currently accessed via the recently paved National A1 Road running from Bouna to Doropo, The secondary roads connecting the towns of Kodo - Kalamon, Kalamon - Danoa and Varale - Danagnara are in fair condition relative to other local tracks which have surfaces in a poor state of repair.

Archaeology and Cultural Heritage

Archaeological and Cultural Heritage reconnaissance carried out in the Project area during 2022 and 2023 identified a total of three (3) archaeological sites within and in the vicinity of the Project footprint. This includes an ancient settlement site within the footprint of the Water Storage Dam and one metallurgical site located approximately 1 km east of Wadaradouo within the Kekeda PDA. The third archaeological site was outside of the Project footprint and comprises a less significant metallurgical site located south of Gbelta 2 and west of Gbelta 1 where scattered remains of ancient iron reduction activity were identified over a radius of roughly 8 m.

The sacred sites of the Lobi and Koulangou people are numerous and often comprise natural features in the landscape including rivers, forests, trees, hills, holes, and rocks. These sites serve to connect the souls of the living with those of dead ancestors, and to worship their animist deities. Some sites are home to powerful spirits and others are ritual or ceremonial sites where vows and wishes are made in return for a sacrifice, usually an animal or crops. Most Lobi villages bury the dead within the village, adjacent to their houses where they are maintained daily. In recent years, some villages have established dedicated cemeteries on the outskirts of the habitation area.

A total of 21 sites of cultural heritage were identified within PDA boundaries. This consists of one forest, two drainage lines, one hill, one cemetery, one former settlement site, and 14 other sacred sites. Of these, 11 sites (one forest, one hill, nine other sites) are within the physical boundary of Project components and will be permanently impacted. Cultural and sacred sites within the area reflects human occupation over a significant timespan with sites identified important to villages of both Lobi and Koulango ethnicity. These sites will require a better understanding prior to disturbance.

Historically, the settlements in the Project area would have exhibited a substantial array of intangible cultural heritage, notably tales, legends, proverbs, dances, and traditional practices. Throughout the region however, there is now a tendency to abandon and neglect these practices. These circumstances can in part be explained by the shift towards a more modern way of life (especially among younger generations).

20.3.3 Ecology

Terrestrial Habitats

The majority of the Project area consists of modified habitats which predominantly comprise cultivated areas (primarily cashew plantations and cashew intercropping). Field surveys undertaken in 2022 and 2023 identified five principal natural vegetation types within the Project area, namely wooded savannah, shrub savannah, gallery forest and swampy thicket with small areas of wooded tree savannah and West African bowal habitat. All natural habitat types identified in the Project area have moderate to high disturbance levels, with some habitats severely fragmented due to human settlement, cultivation, grazing, timber harvesting, uncontrolled burning, and a combination of artisanal mining and Project exploration activities. Gallery forest and swampy thicket are of conservation value and should be considered priority habitats due to their potential to host a higher diversity of species and/or rare or threatened species.

Figure 20.9 Shrub Savannah (Wet Season)

Figure 20.10 West African Bowal (Dry Season)

Figure 20.11 Gallery Forest (Wet Season)

Figure 20.12 Swampy Thicket (Wet Season)

Aquatic Habitats

The hydrology of the Project area consists of a network of ephemeral streams which are largely situated within the Pouene River catchment, which drains north-east into the Black Volta River. The Kilosegui Pit straddles two surface water catchments; the Iringou River catchment which drains west to the Comoé River, the only permanent river in the region (Kahlheber, 2003), and the Kilosegui SE catchment which drains east towards a tributary of the Black Volta River.

The streams and drainage lines in the Project area are predominantly ephemeral and are often bordered by gallery forest and swampy thicket. The ephemeral savanna waters provide fish habitat throughout the wet seasons, including feeding, growing or spawning sites for numerous fish species (Kahlheber, 2003) and in the dry season can serve as a refuge for aestivation of adult beetles (Reintjes, 2004). The riparian habitat is used by semi-aquatic fauna, including the Vulnerable dwarf crocodile.

Critical Habitat

An IFC PS6 (IFC, 2012) aligned Critical Habitat Screening was carried out to determine the potential presence of Critical Habitat. The screening identified six terrestrial species and 31 aquatic species that likely or possibly qualify for Critical Habitat (TBC, 2021a). Field surveys were undertaken to identify the presence of Critical Habitat qualifying species, and other priority species in the Project area.

<u>Terrestrial species:</u> Six species of terrestrial fauna were assessed as likely or possibly qualifying for Critical Habitat based on PS6 criteria (TBC, 2021a). *Cynisca rouxae* was recorded during the 2023 wet season terrestrial biodiversity field surveys, and veldkamp's fruit bat and was reported to occur in the Project area by local survey respondents but not confirmed by field survey. Potentially Critical Habitat Qualifying fauna that were not recorded during the baseline surveys include: white-thighed colobus (*Colobus vellerosus*); white-naped mangabey (*Cercocebus lunulatus*); Western chimpanzee (*Pan troglodytes ssp. Verus*); and African forest elephant (*Loxodonta cyclotis*).

<u>Aquatic species:</u> Twenty-eight fish species were found to potentially meet the thresholds for Critical Habitat based on PS6 criteria (TBC, 2021a). Four of these species were identified in field surveys (*Hepsetus odoe, Marcusenius senegalensis, Marcusenius ussheri and Schilbe mandibularis*). Three macrophyte species identified as potentially Critical Habitat qualifying were not recorded in the baseline studies.

Other Priority Species

- Flora: Six globally threatened species were recorded in the Project area comprising one species classified as Endangered (*Pterocarpus erinaceus)* and five species as Vulnerable (*Khaya senegalensis, Afzelia africana, Pavetta lasioclada, Antrocaryon micraster and Vitellaria paradoxa*) on the IUCN Red List of Threatened Species (IUCN, 2023). Of these species, *Afzelia africana*, is fully protected under Ivoirian legislation (Décret nº 66-122);

- **Avifauna**: The Vulnerable tawny eagle (*Aquila rapax*) and Endangered bateleur (*Terathopius ecaudatus*) were recorded during 2022 and 2023 baseline field surveys;
- **Herpetofauna**: The dwarf crocodile (*Osteolaemus tetraspis*, Figure 20.15) was identified in the 2022 and 2023 field surveys. The semi-aquatic dwarf crocodile is listed as Vulnerable on the IUCN Red List of Threatened Species and is listed under Appendix I of CITES, due to a suspected decline in population size caused by habitat loss and exploitation. The African dwarf crocodile inhabits permanent pools in swamps and areas of slow-moving freshwater in rain forest areas.

Figure 20.13 Patas Monkey (Erythrocebus patas)

Figure 20.14 Bushbuck (Tragelaphus scriptus)

Figure 20.15 Dwarf Crocodile (Osteolaemus tetraspis)

Figure 20.16 C.rouxae

Ecosystem Services

The ecosystem services within the Project area are derived from both modified and natural habitats including fields and fallow areas, shrub savannah, wooded savannah and gallery forest and seasonal wetlands.

Priority Ecosystem Services

In line with IFC's Performance Standards, an analysis of ecosystem services was carried out in order to identify priority ecosystem services relevant to the Project. Priority ecosystem services are two-fold and can be defined under either or both of the following categories:

- Type I - Those services on which the Project is most likely to have an impact, and, therefore, which result in adverse impacts to affected communities; and
- Type II - Those services on which the Project is directly dependent for its operations.

Agricultural production, fishing, timber forest product (TFP) collection, non-timber forest products (NTFP) collection (for food and traditional medicine), and water supply are considered priority provisioning ecosystem services (Type I) for the Project. Water supply is also considered a Type II Ecosystem Service. Hydrological services are considered a Type I and Type II Priority Ecosystem Service, and Cultural Services are also considered a Type I Priority Ecosystem Service for the Project. Priority Ecosystems services are described below:

- Agricultural production - Agriculture is the main source of food and income for the people of the region. The main crops grown are cashew nuts, yams, rice, millet, sorghum, peanuts, cassava, beans, chilli peppers, and corn;
- Fishing - There are no permanent water courses conducive for year-round fishing and it was reported that some families have sacred links with the watercourses and aquatic resources which precludes fishing and consumption of local aquatic resources. Drainages and seasonal wetlands are used as areas for small-scale fishing during the rainy season;
- Timber Forest Products (TFPs) - Villages in the Project area reported that timber and firewood products are generally collected in the dry season and are collected near villages, in the fields or in fallow land where forest has started to regenerate slowly. A range of species are collected for the use of firewood (38 species were reported during the 2023 local knowledge surveys), and which are generally collected by women. Commonly collected species include *Daniellia Olivera, Pterocarpus erinaceus, Khaya senegalensis, Terminalia macroptera, Vitellaria paradoxa.* Firewood is the main source of energy for cooking for all surveyed villages in the Project area with the exception of Lagbo where charcoal, butane gas and firewood are all used;
- Non-Timber Forest Products (NTFPs) - NTFPs encompass all biological materials other than timber which are extracted from natural forests for human use. These include foods, medicines, spices, essential oils, resins, gums, latexes, tannin, dyes, ornamental plants, wildlife (products and live animals), and raw materials used for building materials. NTFPs are sourced from forested, agricultural and fallow areas. Shea tree, a multipurpose tree, is an important NTFP for villages in the Project area providing nutrient rich fruit pulp and kernels as well as a range of other derived products with edible and medicinal applications. Shea is an important source of income for women who reportedly use the money from the sale of shea to buy condiments;

- **Hydrological Services and Water Cycling** - In the Project area, the Pouene River and its tributaries are the primary source of hydrological services for domestic water use (e.g. consumption, washing, bathing), provision of habitat for aquatic species and other freshwater products, attenuation of flood events, water and nutrient cycling, and spiritual and aesthetic values such as cultural importance and sense of place;
- **Cultural Services** - The ecosystems, landscapes and biodiversity in the Project area also have important cultural values for the local communities. There is a strong attachment to sacred forests, which are a guarantee of security and well-being for the Lobi community. There is also a strong attachment to the spirits of dead ancestors. The Lobi are an animist people who believe in the presence of deities and in the presence of dead people who can act on the lives of living people. Several animals are considered important by local people including Vultures, Seba's Python (cultural totem for some Lobi people), and Patas and red monkeys (cultural totem).

Protected Areas

The Project sits outside of any protected area or proposed protected area. The nearest protected area in Côte d'Ivoire is Comoé National Park and Koulbi National Forest. The Comoé National Park is a Biosphere Reserve, UNESCO World Heritage Site (WHS), IUCN Category II Protected Area. It is located approximately 7 km south-west of the Kilosegui deposit and 40 km from the edges of the central deposits. The Park accounts for over 15% of Côte d'Ivoire's terrestrial protected area and is the third largest protected area in West Africa.

The Koulbi National Forest classified forest is located approximately 20 km to the east of the Project and falls entirely within Burkina Faso. The area covers 400 km^2 and was designated as a classified forest in 1955. It falls under the management of the Direction Générale des Eaux et Forêts in Burkina Faso.

20.4 Management Systems/Approach

Ampella is committed to safeguarding the environment from impacts to water, land, climate, air quality and biodiversity, and engaging stakeholders on effective solutions. This includes a commitment to avoid, minimise, mitigate and/or remediate our impacts on the environment, and to maintain overall ecosystem health and resilience in the areas in which we operate.

Ampella's approach to responsible environmental stewardship is formally set out in Centamin's Environmental Policy, which outlines commitments to:

- Comply with all applicable legal and regulatory requirements of the countries in which we operate, and where such legislation or requirements are lacking or absent, apply an internationally recognised standard;

- Ensure environmental risks and opportunities are captured in Ampella's risk management framework;
- Assess environmental and social impacts to inform planning decisions, processes, and implement risk-based management plans;
- Set measurable performance targets to drive accountability and improve environmental performance;
- Safe management of tailings storage facilities and hazardous materials and alignment to good industry practice;
- Support global efforts under the Paris Agreement to reduce our contribution to climate change and measure and report our GHG emissions in accordance with the Greenhouse Gas Protocol
- Monitor impacts and conduct periodic reviews of environmental performance;
- Be transparent in communicating environmental performance to stakeholders.

Centamin's Environment Policy is supported at operational level by an HSES Management Systems Standard and a tailored environmental management plan that considers the regulatory context of the country and unique environmental risks specific to each site.

Environmental and social design criteria were developed during the feasibility process to provide guidance for the design and development of the Project. Project design criteria and relevant international standards aimed at protecting priority environmental and social values in the Project area and are detailed below.

Key Issue/Risk	Project Design Criteria/Considerations	Relevant Standards/Guidelines
Resettlement (Physical Displacement)	Minimise resettlement requirements through careful project design. Project design to promote local employment and skill development. Contracts to specify 'local content' clauses relating to prioritising local acquisition of goods and services. Competitive tenders for all aspects of project construction and operation to include clauses requiring bidders to set out their approach to local employment and capacity building.	IFC Performance Standard (PS) 5 The Decree of 25 November 1930 on the expropriation for public interest in Côte d'Ivoire
Resettlement (Economic Displacement)	Compliance with national and international norms/guidelines for livelihood restoration/resettlement. Mine design to establish a designated Project Development Area (PDA) within which facilities will be located.	IFC PS5 Land Acquisition and Involuntary Resettlement (2012)

Key Issue/Risk	Project Design Criteria/Considerations	Relevant Standards/Guidelines
Dam/TSF Safety	Conformance with the GISTM Compliance with international norms/guidelines for facility design, construction and operation Independent review of dam/TSF safety to be undertaken during Feasibility Study Dam safety/design criteria based on assumption that agricultural land will continue to be used directly down gradient from the dam wall.	GISTM international standard for the safer management of tailings storage facilities. The GISTM was released in August 2020. World Bank (WB) Operational Policy OP 4.37 Safety of Dams (2001) ANCOLD International Commission on Large Dams (ICOLD)
Water Quality (excluding cyanide)	Design to prioritise objective of achieving zero discharge during operations. Design to prioritise diversion of uncontaminated surface water runoff around Project facilities. Downstream surface water and groundwater quality must achieve applicable water quality standards/guidelines (discharge, ambient, drinking water).	Law n° 98-755 of 23 December 1998 on water Code defining the mechanism for the sustainable management of water resources CIAPOL Standards for water quality IFC General EHS Guidelines (Discharge) (2007) IFC EHS Guidelines for Mining (Discharge) (2007) World Health Organisation (WHO) Guidelines for Drinking Water Quality (2011) EU Ambient Water Quality Guidelines
Cyanide Management	In accordance with the International Cyanide Management Code requirements.	International Cyanide Management Institute (ICMI), International Cyanide Management Code (2012)
Air quality, noise and vibration	Compliance with national and international norms/guidelines. Where possible maximise distance between operations and settlements. Pit design and operation to minimise possibility of wind-blown dust affecting settlements. Design to minimise use of fossil fuels and generation of greenhouse gases.	IFC EHS Guidelines for Mining (2007) Decree n°2017-125 of 22 February 2017 on Air Quality and Order n°01164/MINEF/CIAPOL/SDIIC of 04 November 2008 discharges regulating ICPE and discharges. WHO Guidelines
Flyrock	Include an exclusion zone around the mine pits and other blast areas during blasting.	IFC EHS Guidelines for Mining (2007)
Comoé National Park (CNP)	4km Buffer zone distance set up by Ampella from CNP boundary for all activities with potential to adversely affect habitat value. Where possible, maximise the distance between Project infrastructure and the CNP. Design to avoid impacts on the CNP.	IFC PS6 Biodiversity Conservation and Sustainable Management of Living Natural Resources (2012) UNESCO Operational Guidelines for the Implementation of the World Heritage Convention (2013)

Key Issue/Risk	Project Design Criteria/Considerations	Relevant Standards/Guidelines
Priority Terrestrial Habitat and Species	Project footprint to avoid priority and critical habitat types where possible and minimise footprint area where this is not possible. Mine design to establish a designated Project Development Area (PDA) within which facilities will be located. Project design to avoid adverse impacts on priority and endangered species where possible.	IFC PS6 Biodiversity Conservation and Sustainable Management of Living Natural Resources (2012)
Archaeological/Cultural Sites	Project footprint to avoid archaeological/cultural/sacred sites where possible.	IFC PS8 Cultural Heritage
Community Safety	Define controls for community access around Project facilities including fencing.	IFC PS4 Community Health, Safety and Security (2012)
Transport and Transport Infrastructure	Size (width) of transport corridors and associated vegetation clearance to be minimised Avoid Project roads passing directly through village settlements. Avoid loss of community access where possible. Compliance with international guidelines for transport of hazardous materials (including cyanide).	IFC General EHS Guidelines (2007)
Local Economic Development	Project design to promote local employment and skill development. Contracts to specify 'local content' clauses relating to prioritising local acquisition of goods and services. Competitive tenders for all aspects of project construction and operation to include clauses requiring bidders to set out their approach to local employment and capacity building.	IFC PS5 Land Acquisition and Involuntary Resettlement (2012) IFC A Guide to Getting Started in Local Procurement (2011) IFC Strategic Community Investment (2010)
Human Rights and Security	All security personnel, contractors and line managers with a 'duty of care' to be provided VPSHR training.	The VPSHR promotes a set of voluntary principles that guide companies on providing security for their operations while respecting human rights.
Transparency	Report annually on all payments to government including annual profit share, royalties and tax in line with EITI guidelines, UK listing rules and the Extractive Sector Transparency Measures Act ("ESTMA")	EITI - global standard to promote the open and accountable management of extractive resources.

Key Issue/Risk	Project Design Criteria/Considerations	Relevant Standards/Guidelines
Management and Management Systems	Aspire to align its management systems to ISO 45001 Occupational Health and Safety and ISO 14001 Environmental Management.	ISO standards that provide a framework for implementing effective management controls and a systematic approach to setting and achieving improvement targets (e.g. 14001).
Climate Change	Support global efforts to achieve the climate change goals to reduce GHG emissions outlined in international guidance, while also building operational resilience in the face of global warming. Aspire to Net Zero.	United Nations Framework Convention on Climate Change ("UNFCCC") and the Paris Agreement.
Mine Closure	Design for geotechnical and geochemical stability of mine facilities post-closure. Closure ensures the safety and security of local communities and no long-term environmental risk. Design to maximise the post closure benefit to local communities of Project land and infrastructure i.e. roads, water supply, productive land. Avoid adverse social and economic impacts arising from community dependence on the Project through the development of sustainable livelihoods and communities. Leave a tangible positive legacy.	IFC General EHS Guidelines (2007) IFC PS5 Land Acquisition and Involuntary Resettlement (2012) IFC Strategic Community Investment (2010)

Table 20.3 Project Design Criteria

20.5 Physical Environmental Impacts, Management and Mitigation

20.5.1 Open Pits, Mining and Waste Rock

The largest waste product by volume is expected to be waste rock, most of which will be placed in designated Waste Rock Dumps (WRDs) within the PDA boundaries. Approximately 187.5 million tonnes of waste rock are expected to be mined to enable access to 38.2 million tonnes (Mt) of ore for processing into gold. A key aim for waste rock management is in most circumstances to minimise infiltration of oxygen and water into the WRD and retard the potential acid and metal leachate generation process.

Waste rock analysis indicates that the geochemical risk was found to be very low due to low S contents and significant acid neutralisation capacity. As such, waste rock is not expected to generate acid and metalliferous drainage (AMD) but may result in salinity and potentially neutral metalliferous drainage impacts. Additional test work is required on waste rock samples specific to each of the deposits to further constrain the potential for saline/neutral metalliferous drainage.

20.5.2 Mineral Processing and Tailings

In the MPDA, ore will be processed at the processing plant and the waste from this processing will be deposited in the Tailings Storage Facility. The TSF will be lined and is designed to accommodate in excess of 38.2 Mt of tailings. The TSF has been designed to ANCOLD guidelines and Ampella is committed to be in conformance with the Global Industry Standard on Tailings Management (GISTM) and the International Cyanide Management Code (ICMC).

20.5.3 General and Hazardous Waste

A number of other waste streams will be generated during Construction and Operations, including biomass from land clearance, non-mineralised waste rock, tailings, tyres, waste oil, various packaging materials for consumables and scrap materials.

The Project is expected to have a low residual impact on local communities in relation to the generation of hazardous and non-hazardous solid and liquid waste. As with all mine sites risks will need to be carefully managed to ensure that there are no significant accidental spills or release of contaminants.

20.5.4 Water Supply and Quality

No municipal or piped water system exists in the Project area, and all community water sources consist of surface and/or groundwater sources. Many villages within the Project area have stated that current water resources are insufficient to meet the needs of the population, either due to quantity, reliability of supply or quality.

Reducing the impact of the Project on water resources is a key environmental design criteria for the Project, and Ampella has already provided significant support for the installation of groundwater bores for village communities. However, there are likely to be significant Project-related impacts on the amenity of downstream water resources in some drainages. These impacts are summarised below.

20.5.5 Erosion and Sediment

Site preparation activities, such as clearing, grubbing, and grading and construction of Project components have potential to have a high impact on soils. Erosion is a key cause of degradation, and therefore every effort to minimise erosion through good industry practice including progressive revegetation, should be implemented.

There is a risk of soil contamination from hydrocarbon spills throughout the lifecycle of the Project. Spills can continue to persist in the environment, causing potential health risks or reductions in soil productivity long after the Project has closed. Spill prevention and risk mitigation must therefore be a high priority for the duration of the Project.

Soils will be permanently removed from the mine pits. Stockpiled topsoil will be used to restore soil capability on impacted lands wherever possible, including the progressive rehabilitation of the WRDs, TSF and mine footprint.

20.5.6 Air Quality

Dust and particulates are anticipated to be generated from Construction, Operation, and Closure activities around the Project area, particularly in the vicinity of mining pit areas. Blasting operations and haulage are identified as the largest potential source of particulates. Consideration should be given to meteorological conditions during blasting activities to avoid the low wind speed and blustery wind conditions. Haul roads should be constructed from competent materials, supplemented by road watering and traffic controls to minimise fugitive dust emissions.

Air emissions will be most pronounced during construction and operations. Dust will be the main emission, and this will be mostly experienced during the dry season. Mitigation measures such as the use of water trucks to suppress dust near sensitive receptors will be important during the dry season.

Village consultations should be undertaken regularly to qualitatively assess the impacts of dust generation on sensitive receptors. This information can be utilised to improve dust suppression techniques.

20.5.7 Climate and Greenhouse Gas Emissions

The Project is expected to generate approximately 447,279 t CO_2e during construction and 1,425,231 t CO_2e during the operation phase with opportunity for further carbon abatement to be investigated during detailed design. Over the Project life, after rehabilitation and revegetation, the total GHG emissions are estimated to be 1,449,356 t CO_2e. Based on an estimated gold production of 1.591 Moz during the life of the mine, the estimated operational GHG intensity equates to approximately 0.91 t CO_2e per ounce of gold produced. The expected annual GHG emissions for the Project during the operations phase equate to approximately 1.2% of Côte d'Ivoire's current annual GHG emissions[1]. Ampella is investigating opportunities for reducing carbon emissions through the detailed design phase.

20.5.8 Noise, Vibration and Flyrock

The Project has been designed to mitigate noise and vibration to the extent practicable, however Project development will introduce new sources of noise and will elevate noise emissions in the area during the construction, operation, and closure phases.

The significance of noise impacts to the local acoustic environment will depend on a range of factors including topography, timing of activities conducted, duration of noise emissions, and weather conditions. Noise emissions from mine pits will also vary depending on the orientation, geometry and amount of absorption provided by the pit walls. Depending on atmospheric conditions and topography, some low levels of operational noise is expected to reach nearby sensitive receptors including the villages of Herwedouo, Wadradouo, Dupindouo, Mabridouo, Gbelta, Lagbo, and Holidouo. No discernible increases in background noise levels are expected to occur in other nearby sensitive receptor villages during Construction and Operations. When mining and ore processing stops, noise emissions should return to pre-mining levels.

The most significant vibrations from mining operations are typically associated with blasting activities. Air vibrations are not likely to exceed the recommended limit of 120 dBL (decibels linear) in Project affected villages, following the resettlement of Holidouo village from the Enioda Pit. Ground vibrations as a result of blasting are not likely to represent any significant impacts to surrounding sensitive receptors as vibration in these areas is predicted to be lower than the recommended limit of 12.7 mm/s for the Project.

Noise and vibration monitoring will be required over the mine life to confirm the residual impact predictions and allow management measures to be adapted accordingly.

[1] Based on approximate annual Cote d'Ivoire emissions (for fossil fuels and industry during 2021) of 11.7 million tCO2/year (sourced from https://ourworldindata.org/co2-and-greenhouse-gas-emissions), and Earth Systems' estimate of Doropo Project Operations emissions of 142,523 tCO2/year.

Blasting of ore and rock during the mining process can generate flyrock, rock fragments that may be propelled through the areas surrounding the blast area, comprising a safety hazard for people in proximity to blasting. Flyrock risks will be most present during operations but may need to be considered during construction. Though implementation of effective management measures, health and safety risks of flyrock are expected to be significantly diminished. The potential impacts of flyrock are expected to be minor given the limited flyrock distances and no settlements located within the blast exclusion zones of the pits. There should be no blasting requirements during decommissioning and closure and therefore no flyrock impacts or risks.

20.5.9 Visual Amenity

Potential impacts on visual amenity from the Project will include alterations to the local topography and landscape due to vegetation clearance/land disturbance, Project lighting, and the presence of Project infrastructure. Visual impacts will be minimised to the extent practical as progressive rehabilitation and revegetation is undertaken. Post mine closure, local topography will be permanently modified by the presence of waste rock dumps and pit voids which will impact on visual amenity. Overall visual amenity impacts for key sensitive receptors are expected to be low.

20.6 Socio-economic Impacts, Management and Mitigation

20.6.1 Socioeconomic

National Economy

The key benefits of the proposed Project for Côte d'Ivoire will be financial, economic and social benefits at the national, regional and local levels. On a local level, the Project can provide a significant injection of income and employment opportunities to the region, providing opportunities for training and skill development. It has the capacity to catalyse economic and social development in the area, improve access to infrastructure, goods and services, and leave a lasting positive legacy. Royalties and taxes from the Project would also provide needed income for the National Government of Cote d'Ivoire.

The generation of royalties, taxes and dividends from gold mining to the Côte d'Ivoire National Government contributes significantly to the national goals of the country. The three main taxes and fees imposed on companies operating in the mining sector are royalties, salary withholding tax and industrial and commercial profits tax. The Côte d'Ivoire Government also has the right of a 10% free carried ownership interest in the development.

Based on the current gold price which is in excess of US$2,000 per oz, government royalties would be 6% of sales revenue or approximately US$20M per year.

Overall the Project would be expected to contribute over US$2B (based on the current gold price) to the Ivoirian economy in the form of taxes, royalties, dividends, salaries, payments to local suppliers and infrastructure development. During operation Ampella can be expected to be one of the largest corporate taxpayers in Côte d'Ivoire.

Mines operating in Cote d'Ivoire are required to pay 0.5% of their turnover to a Local Mining Development Fund, which aims to finance community development projects in mining regions. This Fund is expected to generate approximately US$17M over a 10-year life of the Project with approximately US$2M generated each year for the first five years of operation.

Through effective management, the injection of income and economic opportunity into the local community as a result of the Project will be a major benefit. Through employment, training, procurement and livelihood improvement programs, the Project can be a catalyst for sustained economic growth in the Doropo Region. The Company's existing social investment program will also continue to complement Project-related activities and contribute to achieving local development objectives.

Employment and Local Economy

The Project is expected to be a driver for economic growth in the region through employment expenditure, infrastructure development, and social investment. A significant proportion of the capital expenditure and operating costs for the Project will be spent in the Prefecture of Doropo, resulting in flow-on benefits to the local community.

Through its mineral exploration and feasibility study activities, the Project has already generated job opportunities and economic activity in the region. This will be magnified significantly if the Project moves forward to development. During the Construction Phase, a period of approximately two years, the Project will create more than 1,000 new direct employment opportunities plus additional indirect employment through the local supply chain. Employment will result in an injection of income and socio-economic uplift in nearby villages and the surrounding area.

Ampella will uphold international requirements concerning modern slavery risks across the supply chain, and the approach to diversity, equal opportunity and pay, discrimination, employee relations and labour conditions.

Employment opportunities will continue during the Operations Phase with an overall workforce expected to stabilise at approximately 1,000 direct employees and contractors. Operations will continue for 10-years, although most large gold mining projects typically extend their life through further mineral exploration.

Based on the economic opportunities created during mining operation and the more stable employment of the Operations Phase it could be expected that a significant town would develop proximal to the mine. This would be expected to develop near to the MPDA. There are many examples of successful and thriving towns throughout the world that commenced their life as a mining town.

Local procurement of goods and services for the Doropo Mine will be promoted during the Operations Phase and continue to enhance socio-economic uplift for local villages and towns.

Training and Skills Development

Ampella is committed to support the development of a responsible mining sector in Côte d'Ivoire. Ampella will bring professional, safety and quality management systems to its operations which will positively impact and influence employees, contractors, suppliers and business partners.

Mining requires diverse skill sets including engineering, geology, human resource management, accounting, logistics, procurement, planning, etc. which are all transferrable across companies and industry sectors.

Ampella is committed to training and skills development of its workforce through the Construction and Operations Phases of the Project. Operations employment will be longer term than typical construction jobs and will provide opportunity for entry into well paid mining employment.

20.6.2 Land Acquisition, Livelihood Restoration and Resettlement

Within the Project area, legal land title has yet to be implemented and the customary system of land tenure is the only established land management mechanism. The Project area has a complex land ownership history that will be addressed sensitively through stakeholder engagement during development.

Land within the Project area is primarily occupied and used by people of the Lobi ethnic group who settled in the area from Burkina Faso over the last 100-years. Historically the Koulango had a kingdom centred on Bouna and were the traditional custodians of the land. Low numbers of Peuhl or Fulani livestock herders also use the land and occupy a few small hamlets in the Project area.

The key social issues and risks that will need to be managed very carefully by the Project are:

- High levels of economic and social vulnerability in Project affected villages which could be exacerbated by Project-related land acquisition;
- Negotiating agreement for the transfer of traditional land rights from the community to the Project noting the overlapping land rights and interests of different ethnic groups, particularly the Koulango and Lobi;
- Ensuring a fair and equitable system of compensation for the loss of land, assets and livelihoods;

- Implementing strong livelihood restoration measures in addition to appropriate cash compensation for land acquisition; and
- Land and socio-economic fragmentation resulting from the extended Project footprint.

Land and Livelihood

The loss of access to land will have direct and indirect impacts to the communities living in these areas. A total of 23 villages own land within the footprint of Project infrastructure and the PDAs. Village land impacts are summarised as follows:

- 7 villages with major impacts (>25% land impacted): Herwedouo, Bissankouedouo, Holidouo, Wadaradouo, Dupindouo, Gbelta, Lehitedouo;
- 9 villages with medium impacts (5-25% land impacted): Tonguidouo, Lassouri, Lagbo, Fafoudouo, Behindjinadouo, Mabridouo, Fangadouo, Gangata, Norfadouo/Yolkoubiel;
- 7 villages with minor impacts (<5% impacted): Gola, Simatedouo, Bonkodouo, Douhoundo, Loukoura, Sonfrodouo, Talo.

To minimise the extent and duration of loss to livelihoods, Ampella will permit community access and controlled use of land within the PDA where this does not present a security risk to the Project or safety risk to the community. This includes lands that are scheduled for development in the later years of mine life, namely the satellite PDAs. Ampella will also implement progressive rehabilitation which will permit the timely return of little disturbed land to the community post-mining, including the blast exclusion zones associated with the satellite PDAs. All villages reported using drainage channels to which access will be impacted by the Project, hence alternative sustainable water resources should be carefully considered.

PDAs cover a combined total area of 3,783.4 ha as detailed below.

Land Cover Category	MPDA	Enioda PDA	Han PDA	Kekeda PDA	Kilosegui PDA	Grand Total
Grazing Land/Forest Resource Use Areas	728.4	48.9	16.8	35.1	95.6	924.8
Cashew Plantation (Mature)	391.7	71.8	20.4	44.9	297.5	826.3
Forest and Natural Habitat Fragments	363.2	38.8	62.3	96.1	219.4	779.8
Other Crops (e.g. Maize, Yam, Millet)	250.1	54.4	24.1	32.7	292.6	653.9
Cashew Plantation (Immature) and Other Crops	72.2	30.4	2.9	0.8	112.8	219.1
Recently Fallow	46.9	6.0	74.4	11.8	37.7	176.8

Land Cover Category	MPDA	Enioda PDA	Han PDA	Kekeda PDA	Kilosegui PDA	Grand Total
Artisanal Mining (Active and Inactive)	53.2	10.4	26.2	17.4	3.0	110.2
Intensive Agriculture (e.g. Rice)	27.8	3.3	0.0	0.0	7.6	38.7
Roads	9.7	4.1	3.6	4.5	4.9	26.8
Settlement Area	1.7	15.2	0.7	0.7	0.6	18.9
West African Bowal	6.4	0.0	0.0	0.0	0.0	6.4
Sacred Forest	2.2	0.0	0.0	0.0	0.0	2.2
Grand Total	1953.6	283.3	231.4	243.5	1071.7	3783.4

Table 20.4 Land Cover within the Doropo PDAs

All losses of household and community land, crops or economic trees, assets and infrastructure will require replacement and/or compensation. The compensation and livelihood restoration process for identified losses of land and assets will be undertaken in line with the Ivoirian legislation supported by international industry good practices including the International Finance Corporation (IFC) Performance Standards. This will be managed through rigorous stakeholder engagement, consultations with the Ivoirian government and legal consultants, to form an appropriate land acquisition and compensation plan.

A stand-alone Resettlement and Livelihood Restoration Framework (RLRF) has been prepared as part of the ESIA and provides the necessary strategic framework for the social planning of the Project, and encompasses resettlement, livelihood restoration and compensation strategies.

Physical Displacement of Project Affected Persons

Ampella has evaluated various Project design and layout alternatives as part of the ESIA process. Significant changes have been made to the Project layout in the design process to reduce the extent and duration of loss to community land, assets and livelihoods.

Involuntary resettlement of communities has been avoided for all villages located within or proximal to PDAs except for the village of Holidouo. A total of 314 people from 33 households associated with the village Holidouo reside within the Enioda PDA and will require resettlement and livelihood restoration as a result the development. A resettlement site with replacement housing and necessary infrastructure will need to be provided at least 12 months prior to the development of the Enioda Pit which is expected to commence at the start of Operations Year-4. Several structures on Gbelta land have been identified within the edge of the Kilosegui East PDA. Any permanent residents residing in these structures may require temporary relocation or resettlement.

There are two small Fulani hamlets located on Herwedouo land that lie within the MPDA boundary. Each hamlet contains one household consisting of three people. In addition, one household was identified north of the village within the MPDA and is understood to have 3 residents. Resettlement of any residents permanently residing in these locations will be required.

Community Assets and Infrastructure

Community assets associated with the village of Holidouo will be permanently lost due to the development of the Enioda PDA. This includes one community school with two classrooms, one nursery school, two churches, one football pitch, one water source and two water pumps. The Main PDA is expected to impact on one borehole associated with the village Hérwedouo. The Kilosegui PDA is expected to impact on three open water sources. No other physical impacts on community assets and infrastructure are expected.

20.6.3 Transport and Traffic Safety

In general, motorised traffic in the Project area is low, with the most common methods of transport used by the local population being walking, motorcycles and motor-tricycles. Unsealed and rudimentary tracks and roads provide access between settlements, artisanal mine sites, agricultural areas and to natural resource areas.

Project development will result in the upgrade and maintenance of road infrastructure and is expected to increase the availability of transport services. Key measures to mitigate the impacts to traffic, accessibility and transport include:

- Diversion of community access tracks around PDAs to maintain socio-economic connectivity between settlements and important land use areas;
- Establishment of controlled safety crossings where community roads intersect Project infrastructure, including haul roads;
- Strict control of Project vehicles operating on community roads including GPS tracking, enforcement of speed limits and avoidance of vehicle movements at night to the extent possible;
- Community road safety awareness programs;
- Regular maintenance of community roads where they are subject to increased use and deterioration.

At Project Decommissioning and Closure Phases, community access will be restored within the various PDAs, subject to any post-mining safety restrictions. Many of the roads constructed and upgraded for the Project may also remain for community use depending on final land use.

20.6.4 *Community Health and Safety*

The presence of the Project is generally expected to improve access to local health infrastructure and services in local villages in the area, including access to potable water and sanitation facilities, through the Project's mitigation and community investment programs.

The Project, however, has the potential to result in an increase in the prevalence of vector-related, respiratory, soil and water-borne diseases. Project water facilities could create breeding habitats for disease bearing mosquitos; and Project-related in-migration could increase the transmission of sexually transmitted infections (STIs).

Ampella is committed to work closely with local health services to monitor and address changes in levels of community health, safety and nutrition through regular community consultations as per the Stakeholder Engagement Plan and biennial socio-economic surveying for villages in the vicinity of the Project. Management measures will be adapted accordingly. Ampella aims to minimise incidences of vector-borne and water-related diseases through water quality and waste management. Monitoring of in-migration and strengthening of community infrastructure through an appropriate community development plan (accommodation, WASH services, healthcare, health education) are essential to mitigate and reduce impacts.

20.6.5 *Artisanal Gold Mining*

Artisanal gold mining is active within the Project area, including on some of Ampella's prospects at varying levels of intensity. This activity is not licensed, is recognised as illegal within Ampella's permit areas and operates as a clandestine activity.

In situations where the nature and extent of ASM presents an unacceptable risk to the safety and security of the Project, Ampella works with the artisanal miners themselves, local communities and the authorities to control the activity. As a consequence, Ampella has effectively reduced the incidence of illegal artisanal mining on its prospects. The Ivoirian Government supports Ampella in restricting illegal artisanal mining activity on its licences through the mobilisation of public security patrols.

Artisanal mining can provide an important source of cash income for some communities through control of rights to mineral resources, sale of mineralised rock, the provision of labour or other services (accommodation, food, etc.). Artisanal mining can also attract migrant miners, most notably from Burkina Faso. Some villages (e.g. Lagbo, Herwedouo) have grown as a result of this activity.

Local communities are aware that illegal artisanal miners will be displaced from the PDAs as a consequence of Project development and will not receive direct compensation for the loss of income from this activity. In consultation with the Ministry of Mines, Ampella is committed to investigate opportunities to formalise artisanal mining at sites outside of its Mining Permit. More broadly, Ampella shall support affected communities with the development of alternate livelihoods.

20.6.6 In-migration

Project development has potential to exacerbate levels of in-migration to the Project area, arising from employment, business opportunities, land compensation and artisanal mining. In-migration can have both positive and negative socio-economic impacts on host communities. It can increase the availability of goods and services, increase costs, increase levels of inequality, intensify land use and degrade natural habitat.

Various towns and villages in the Project area are expected to expand as a consequence of Project development, particularly those that offer improved levels of urban infrastructure such as mains water, electricity, access to markets and communications.

To the extent possible, Ampella aims to mitigate the adverse impacts of in-migration while also enhancing the positive impacts. These measures include: local employment procedures accompanied by workforce training and skills development; local procurement procedures accompanied by technical and organisational capacity building programs; livelihood restoration and improvement program for Project affected persons; and contractor management plans.

Some out-migration can be expected post closure of the mine arising from wind-down of Project-related employment and procurement activities.

In addition, Ampella will work in partnership with local authorities to plan for in-migration and seek to optimise allocation of the Community Development Fund to mitigate adverse impacts.

20.6.7 Community Development Fund

In accordance with the Ivoirian Mining Code (2014) Article 125, a Community Development Fund (CDF) managed by a Local Mining Development Committee (CDLM) will be established, to finance projects that promote sustainable community development for villages directly or indirectly affected by mining explorations. This includes investment in projects that support economic empowerment and social infrastructure. It also aims to increase the involvement of affected communities in community development decision making to better cater towards their specific needs. Under the Code, mining companies must contribute 0.5% of total revenue to the CDF.

20.7 Biological and Ecological Impacts, Management and Mitigation

During every stage of the Project, Ampella will implement the mitigation hierarchy to avoid, minimise and restore/rehabilitate impacts on biodiversity. The key management and mitigation measures will be incorporated into a Biodiversity Management Plan for the Project, prior to the commencement of Construction.

The Project area is predominantly comprised of modified habitat, with a large proportion of the original vegetation being converted to cultivated land. Remnant areas of natural habitats are significantly impacted by anthropogenic activities including artisanal mining, agriculture, fires, livestock grazing and illegal logging.

Effort has been made through design to reduce the Project footprint and the requirement for vegetation clearance to the maximum extent possible. The most significant impact to habitat and flora from the Project will be the direct loss of remnant natural habitat, notably gallery forest and swampy thicket. Edge/fragmentation effects due to vegetation clearance will also add to the total area impacted.

Ampella has in place a strict procedure to control vegetation clearance and identify the potential presence of threatened species. This procedure will apply during Construction and Operation where feasible. Progressive restoration and rehabilitation activities will partially address habitat loss and fragmentation, including the propagation of threatened species.

Subject to ongoing investigation, remnant areas of gallery forest and swampy thicket, and waterways within the Project area may qualify as Critical Habitat due to the presence of:

- Endemic worm lizard *C.rouxae.* Little ecological information exists for this species, however specimens have been collected in the CNP in open savannah and from leaf litter in tree savannah and gallery forest. The species was found in gallery forest in the Project area, as well as in agricultural land, suggesting its tolerance to anthropogenic disturbances.
- Four migratory fish species *Hepsetus odoe, Marcusenius senegalensis, Marcusenius ussheri and Schilbe mandibularis* which rely on the aquatic waterways in the Project area. *H.odoe* occurs in coastal rivers, lakes and swamps, preferring deeper water such as channels and lagoons. *Marcusenius ussheri* is demersal, and considered a strictly freshwater species, but has been observed in brackish waters. *Marcusenius senegalensis* is found in across West-Africa in running waters, including the Comoé and Volta Basins, and in freshwater lakes, but is absent in brackish environments. *Schilbe mandibularis* is predominantly found in freshwater habitats but has been recorded in brackish lagoons.

Further assessment of populations within the Project area will be undertaken to definitively determine Critical Habitat status of these species.

Impacts to natural and critical habitat will be managed through the implementation of a Biodiversity Management Plan, and residual impacts will be offset by the Project. This could involve the development of agreed land use plans, with the involvement of local authorities and communities, to mitigate the impact of anthropogenic land use on species and their habitat in the periphery zone to the Comoe National Park.

Project in-migration may have a further impact on habitats and fauna in the Project area, including increased exploitation of timber and non-timber products. Forest stands (e.g. gallery forest) are most likely to be targeted for hunting and timber collection; and aquatic resources for fish and other aquatic fauna such as frogs, crocodiles and turtles.

Any changes to water quality and hydrology associated with Project development will negatively impact habitats and their floristic assemblages, and consequently aquatic fauna. Changes in water quality may also impact terrestrial fauna which rely on these water resources. The primary impact to surface water quality will likely be suspended sediments generated from increased erosion and sediment transport during construction activities. Vegetation clearance and earthwork is likely to cause localised suspended sediment impacts to drainages in the Project area. To mitigate this, sediment control structures will be put in place as part of the early-works program for Project Construction. There is a risk to aquatic habitats and species associated with spills and discharges of hazardous materials, which will be most significant during the Operation phase of the Project.

Downstream flow on the Pouene river will be reduced to a minor extent during the Construction and Operation Phase from the filling and storage of water within the WSD and WHD. Changes to hydrology associated with Project development may negatively impact habitats and their floristic assemblages, and consequently aquatic fauna. Gallery forest and ephemeral water bodies would be the most sensitive to hydrological change as these habitats are highly reliant on seasonal water flows. Reductions in stream flow volume, particularly during the dry season will also impact water availability for wildlife. Ampella will monitor changes in flow and water availability and will implement adaptive measures as necessary.

Key impacts to ecosystem services associated with the Project will result from the loss of land used to access provisioning ecosystems services, namely land used for agriculture, and forest resource use (TFP/NTFP collection). The Project is also expected to impact on the amenity of waterways for potable water use downstream of mine infrastructure. Project-related in-migration may further compound impacts to ecosystem services. Effective implementation of the RMCP and livelihood restoration measures in the RLRF will assist in minimising long-term impacts on agriculture and natural resource use within Project areas.

20.8 Archaeology and Cultural Heritage

A total of 21 sites of cultural heritage and two archaeological sites were identified within PDA boundaries. Of these, 11 cultural heritage sites and two archaeological sites, namely the Gblogblo archaeological site and Wadaradouo mettalurgical site, will be permanently lost. Pre-clearance surveys on the two archaeological sites and their surrounding areas will be carried out by a specialist archaeologist to recover, catalogue, and preserve any artifacts of cultural or historical significance. Thorough preconstruction surveys will be undertaken within Project footprint areas by qualified personnel to ensure sites of archaeological or cultural heritage value are fully investigated and protected from disturbance or otherwise appropriately managed. A Chance Find Procedure will be implemented during Construction and Operations. In the event of an accidental discovery, Ampella will ensure to inform the Ministry of Culture, which is required to undertake necessary steps to collect identified artifacts in accordance with Law no. 87-806 of 28 July 1987.

Careful engagement with communities will be required for the appropriate management of cultural sites. In the case of sacred sites, this is likely to involve rituals to appease the spirits in accordance with local customs.

Fencing and signage around sites within PDA boundaries will be implemented where practicable to provide protection and indicate areas of archaeological or cultural importance that should be avoided.

Upon mine closure, full access to archaeological and cultural heritage sites that were not lost to mine operations will be reinstated to the local community where feasible.

20.9 Cumulative Impacts

The Project area is in an agricultural and artisanal mining landscape, with few industrial-scale projects in the region. The granitic domain that characterises the region had previously been considered as non-prospective for gold deposits, and as such no significant gold mine is operational in the area. Given the lack of nearby major developments/activities, the Project is not expected to significantly hinder the development of any other existing or planned projects and is not expected to result in any significant environmental and social cumulative impacts.

Existing anthropogenic activities have resulted in the loss and fragmentation of natural habitats, creating increased pressure on remnant natural habitats and biodiversity. Development of the Project will exacerbate this loss through the removal of over 1,000 ha of habitat within the PDAs. Ampella will support ecology/biodiversity initiatives in the region, to achieve an overall no net loss to Critical Habitat as a result of Project development.

The biggest contribution from the Project is likely to be associated with further enhancement of the socio-economic development of the region and development of a stronger mining skills base in the Doropo region. A significant proportion of the capital expenditure and operating costs for the Project will be spent in the Prefecture of Doropo, resulting in significant flow-on benefits to the local community.

Ampella has incorporated environmental and social considerations into numerous aspects of the Project design, and a detailed management program has been developed, which will assist in minimising adverse cumulative impacts associated with the Project. Local, regional and national level government and stakeholder consultation will need to be undertaken throughout the life of the Project to understand and manage any potential cumulative environmental and social risks.

20.10 Stakeholder Engagement

Ampella initiated active community engagement with key stakeholders during exploration activities from 2013. Stakeholder engagement and community consultation increased in support of environmental and social baseline studies, impact assessment and participatory planning processes from 2022. These consultation activities have helped build mutually beneficial relationships in the Project affected villages.

Key stakeholder groups include:

- Project affected villages and host communities;
- Women, youth, ethnic minority groups and other vulnerable groups;
- Local administrative authorities, technical agencies and government regulators;
- Local businesses and suppliers with a commercial interest; and
- Civil society, rural development organisations and aid agencies.

The ESIA included a formal public consultation period during which Project affected villages confirmed their support for Project development.

Ongoing stakeholder consultation will be critical to establishing a robust license to operate. Ampella has developed a detailed *Stakeholder Engagement Plan* as part of the ESIA that identifies requirements for stakeholder consultation, participation, and disclosure activities and requirements.

20.11 Occupational Health and Safety

Occupational health and safety management practices will comply with applicable Ivoirian law and align with international standards and guidelines to avoid or minimise potential risks to Project personnel and communities.

Prior to construction, Ampella will develop a site wide OHS Management Plan aligned to ISO 45001.

20.12 Emergency Preparedness and Response

An Emergency Preparedness and Response Plan (EPRP) will be developed in accordance with national regulatory requirements and good industry practice; and prior to Project Construction.

The objective of the EPRP will be to ensure that the site has adequately identified, planned for, and are able to respond effectively to credible emergency, crisis and disaster situations that could adversely affect employees, the surrounding environment and/or local communities. The key objectives for Emergency Preparedness and Response for the Project are to:

- Prevent injury or loss of life;
- Minimise property loss, damage to equipment and the environment;
- Ensure organisational structures are in place to facilitate a prompt and co-ordinated approach to emergency management; and
- Ensure all necessary equipment, personnel and other resources are available for effective control of an emergency situation.

20.13 ESMMP Framework

The ESIA identified the potential impacts and risks of the Project and a professional management and mitigation program has been developed in accordance with Cote D'Ivoire legislation and standards (i.e. ISO 14001, OHSAS 18001 etc.). The effective implementation and regular updating of the Environmental and Social Management and Monitoring Plan (ESMMP) and other management plans in response to changing needs will ensure that environmental and social impacts attributable to the Project are minimised and potential environmental and social benefits are maximised.

The ESMMP includes a range of Standard Operating Procedures that provide general environmental and social management measures to manage Project impacts. The ESMMP also outlines the framework for the overall Environment, Social, Health, Safety Management System established for the Project including responsibilities and targets for management and monitoring activities. In particular, the ESMMP describes the suite of management measures and monitoring programs to be implemented during the construction, operations and closure phases of the Project.

The ESMMP will be used in conjunction with the following standalone management plans, which have been developed as part of the ESIA:

- Stakeholder Engagement Plan (SEP);
- Rehabilitation and Conceptual Mine Closure Plan (RCMCP); and
- Resettlement and Livelihood Restoration Framework (RLRF).

The Plans will be supported by numerous procedures, forms, registers, etc. that will be developed and implemented as required.

To facilitate the implementation of the environmental and social management program for the Project, Ampella will continue to maintain a Health, Safety, Environment and Social Department. The responsibility of the department will include the implementation of the ESMMP and other environmental and social management plans developed for the Project.

Implementation of the detailed monitoring strategy prepared as part of the ESMMP will be important to allow assessment of the effectiveness of the existing management measures, and to identify the need for improved or additional measures. Ampella will also support the Government of Cote D'Ivoire, via the Regional Environmental Monitoring Committee, to monitor the activities of Ampella. In addition, regular internal and independent external audits of the environmental management system will be commissioned by Ampella.

With careful implementation of the proposed management and livelihoods restoration measures, the Company commits to developing the Project in a way which provides a net socio-economic benefit to local communities and to Cote d'Ivoire without compromising the integrity of the broader environment.

20.14 Rehabilitation and Mine Closure

A Rehabilitation and Conceptual Mine Closure Plan has been prepared as part of the ESIA. Ampella's overall closure vision for the Project is to close and rehabilitate the Project to a physically, chemically and ecologically stable landscape that is safe and legally compliant; and to provide a positive legacy for host communities. During the Life of Mine (LOM), land will be progressively returned to the community, where it is deemed safe and secure for community use.

At closure, mine pits will be allowed to form pit lakes, while the Tailings Storage Facility (TSF) and Waste Rock Dumps (WRD) will be rehabilitated to ensure long term stability of these landforms. The WRDs and TSF will be revegetated to self-sustaining savannah ecosystems that mimic local biodiversity and natural habitats. Water storage facilities will likely be repurposed for community use. Further buildings or infrastructure will be repurposed based on the outcomes of ongoing stakeholder consultation or rehabilitated to provide self-sustaining ecosystems comprising local native species.

During the operation of the Project further consultation with Government and community will be required to refine this Plan to ultimately achieve mine completion in accordance with legislative requirements and host community expectations.

Consistent with industry leading practice, mine closure planning has been an integral part of Project development, construction, and operations planning.

Ampella will establish an appropriate financial instrument to ensure that sufficient funds are available to cover the cost of mine closure. Côte D'Ivoire legislation requires that at the start of operations, the mining operator opens and credits an escrow account within a Côte D'Ivoire financial institution.

20.15 Resettlement and Livelihood Restoration

A stand-alone Resettlement and Livelihood Restoration Framework (RLRF) has been prepared and provides the necessary strategic framework for the social planning of the Project and encompasses livelihood restoration and improvement strategies. It sets out the objectives, eligibility criteria for Project Affected Persons, entitlements, legal and institutional framework, modes of compensation, participation and consultation procedures, and grievance redress mechanisms which will be used to compensate and restore the livelihoods and living standards of Project Affected Persons.

The RLRF has been developed in accordance with the Ivoirian legal and institutional framework, and international standard requirements under the International Finance Corporation (IFC) Performance Standard 5 (IFC, 2012). This framework considers World Bank's environmental and social safeguard policies, in particular OP 4.12 Involuntary Resettlement (The World Bank Inspection Panel, 2016).

The RLRF applies to Project Affected Persons, households and communities that are expected to be physically and/or economically displaced. In total 23 villages were identified as owning land within the PDA boundaries. Census data collected by the Ivoirian Government in 2021 states these villages house a combined total of 8,397 residents across 1,055 households (RGPH, 2021). Household surveys undertaken for the Project in May 2023 identified a total of approximately 3,500 residents across 442 households residing in villages proximal to Project infrastructure (Herwedouo, Lassouri, Bissankouedouo, Holidouo, Lagbo, Wadaradouo, Dupindouo, Gbelta, and Gangata).

Involuntary resettlement has been avoided for all villages located within or proximal to PDAs except for the village of Holidouo. A total of 314 people from 33 households associated with the village Holidouo reside within the Enioda PDA and will require resettlement and livelihood restoration as a result the development.

20.16 Conclusions

An environmental and social baseline has been established for the Project with extensive field studies undertaken by Ampella since February 2022 to support Project pre-feasibility and feasibility design studies as well as the statutory ESIA.

Ampella has been granted an Environmental Permit from the Ministry of Environment, Sustainable Development and Ecological Transition with receipt of Decree No.000322/MINEDDTE/ANDE of June 13, 2024 approving the Environmental and Social Impact Assessment (ESIA) of the Project.

The ESIA has identified the potential impacts and risks of the Project based on available information and a professional management and mitigation program has been developed in accordance with Cote d'Ivoire legislation and industry good practice. Key impacts, risks and opportunities that are expected from the development of the Doropo Gold Project include:

- Economic benefits and income for the Government of Cote D'Ivoire;
- Work opportunities and skill development for the people of Cote D'Ivoire and in particular for communities proximal to the Project;
- Direct impacts on land within the Project Development Areas, the resettlement of one village and the impact on livelihoods of local communities near the proposed mining development;
- Impacts on biological resources in the vicinity of the Project; and
- Impacts on the drainage lines and tributaries within the mine development, and risk associated with the use of hazardous materials.

Ampella is committed to establishing an integrated Environmental and Social Management System (ESMS) for the Doropo Gold Project which is consistent with Government of Cote D'Ivoire legislation and aligned with international standards. This management system will provide the Company with a procedural framework for implementing, achieving, reviewing and maintaining the Company's environmental and community policies and all environmental and social management targets.

The effective implementation and regular updating of the Environmental and Social Management Plan (ESMP) and other management plans in response to changing needs will ensure that environmental and social impacts attributable to the Project are minimised and potential environmental and social benefits are maximised.

A Resettlement and Livelihood Restoration Framework has been prepared and provides the necessary strategic framework for the social planning of the Project and encompasses livelihood restoration and improvement strategies. It sets out the eligibility criteria for Project Affected Persons, entitlements, modes of compensation, and participation and consultation procedures that will be used to compensate and restore the livelihoods and living standards of Project Affected Persons.

Successful environmental and social management for the Project will require ongoing consultation with the Government of Cote d'Ivoire, local communities and other stakeholders to ensure stakeholder interests continue to be taken into account in the planning and development of the Project.

The Project has been designed to enable it to be closed in a safe and stable manner once mining and mineral processing has been completed. Mine closure will include the restoration of disturbed drainages, stabilisation of affected areas and removal of infrastructure that will not be transferred to the Government. Importantly, the Project should be designed to leave a strong Project legacy post closure. Notable opportunity for community development includes improved access to water and energy infrastructure and the intensification of agricultural activities.

With careful implementation of the proposed management and livelihoods restoration measures, the Project is expected to be able to be developed in a way which provides a net socio-economic benefit to local communities and to Cote d'Ivoire without compromising the integrity of the broader environment.

TABLE OF CONTENTS

21. CAPITAL AND OPERATING COSTS 1

21.1 Introduction 1

21.1.1 Scope of Capital and Operating Cost Estimates 1

21.1.2 Key Contributors 1

21.1.3 Estimate Currency and Base Date 2

21.2 Capital Cost Estimate Summary 2

21.2.1 Project Costs 3

21.2.2 Mining Costs 3

21.2.3 Processing Plant and Infrastructure Costs 4

21.2.4 Owner's Costs 6

21.3 Basis of Capital Cost Estimate Development 6

21.3.1 Methodology 6

21.3.2 Mining 7

21.3.3 Earthworks 7

21.3.4 Concrete 7

21.3.5 Structural Steel 8

21.3.6 Equipment 8

21.3.7 Platework 9

21.3.8 Piping 9

21.3.9 Electrical and Instrumentation 9

21.3.10 Installation Labour 10

21.3.11 Owner's Mobile Plant 10

21.3.12 Project Spares 10

21.3.13 Initial Fills 11

21.3.14 Plant Services and Infrastructure 11

21.3.15 Other Infrastructure 11

21.3.16 Accommodation 12

21.3.17 Transport 12

21.3.18 Indirect Onsite Costs 12

21.3.19 Engineering, Procurement and Construction Management 13

21.3.20 Growth 13

21.3.21 Project Contingency 13

21.3.22 Owner's Costs 13

21.3.23 Qualifications and Assumptions 14

21.3.24 Exclusions 15

21.3.25 Sustaining Capital 15

21.4 Operating Cost Estimate 16
21.4.1 Project Operating Costs 16
21.5 Mining Operating Costs 16
21.6 Processing Plant Operating Costs 17
21.6.1 General 17
21.6.2 Estimating Method and Accuracy 17
21.6.3 Summary 18
21.6.4 Operating Cost Centres 21
21.6.5 Operating Cost Areas 27
21.7 Site General and Administration 28
21.8 Exclusions 29

LIST OF FIGURES

Figure 21.1 Processing Operating Cost Estimate Summary by Cost Centre (Fresh) 19
Figure 21.2 Processing Operating Cost Estimate Summary by Cost Centre (Oxide/Transition) 19
Figure 21.3 Processing Operating Cost Estimate Summary by Area (Fresh) 20
Figure 21.4 Processing Operating Cost Estimate Summary by Area (Oxide/Transition) 21

LIST OF TABLES

Table 21.1 Foreign Exchange Rates 2
Table 21.2 Project Capital Cost Estimate Summary 3
Table 21.3 Processing and infrastructure Capital Cost Estimate Summary 5
Table 21.4 Owner's Cost Estimate Summary 6
Table 21.5 Mechanical Equipment Pricing 9
Table 21.6 Sustaining Capital Estimate Summary 16
Table 21.7 Project Operating Cost Estimate Summaries 16
Table 21.8 Mining Operating Cost Estimate Summaries 17
Table 21.9 Processing Operating Cost Estimate Summaries by Cost Centre 18
Table 21.10 Processing Operating Cost Estimate Summaries by Area 20
Table 21.11 Power Summary - Fresh Feed 22
Table 21.12 Power Summary - Oxide-Transition Feed 22
Table 21.13 Fresh Ore Reagent Cost and Consumptions 24
Table 21.14 Oxide-Transitional Ore Reagent Cost and Consumptions 24
Table 21.15 Processing Plant Labour Cost Summary 26

21. CAPITAL AND OPERATING COSTS

21.1 Introduction

21.1.1 Scope of Capital and Operating Cost Estimates

Capital and operating cost estimates have been developed for a mining operation capable of treating 4,000,000 t/annum of fresh Doropo ore (5,400,000 t/annum of oxide-transitional ore) and comprising the following:

- A conventional open pit mine using a truck and shovel operating methodology;
- A processing plant producing gold doré, with gold recovered utilising a flow sheet as described in Part 17 of this Technical Report;
- Utilities and infrastructure required for the operation, maintenance, and administration of mining and processing activities;
- Tailings storage facility and waste rock dumps complete with water management and drainage infrastructure;
- Airstrip, access roads, and haul roads;
- Dams for the harvesting and storage of surface water;
- Mine pit dewatering infrastructure; and
- Surface water management and sediment control structures.

21.1.2 Key Contributors

The capital cost estimate for the Doropo Gold Project (the Project) has been compiled by GRES based on inputs received from:

- Orelogy - provided costs associated with mine establishment and associated facilities;
- GRES - provided the costs for the processing plant and associated infrastructure;
- Knight Piésold - provided quantities for the tailings storage facility, water harvest dam, water storage dam, site access roads, haul roads, airstrip and sediment control structures. Pricing, incorporating unit rates and costs were developed and applied by GRES;
- ECG - provided the estimate for the power supply connection infrastructure; and
- Centamin - provided Owner's costs and compiled the life of mine (LOM) sustaining capital cost estimate, supported by Knight Piésold.

Operating cost estimates for the Doropo Gold Project has been compiled by GRES based on inputs developed by:

- Orelogy - for mining contractor and mine management costs;
- GRES - for the processing costs;

- ECG - for the cost of power;
- Centamin - for the Site General and Administration (G&A) costs, as well as labour organisation charts, project manning, labour rates and operational manning build-up.

21.1.3 *Estimate Currency and Base Date*

The capital and operating cost estimates are expressed in United States dollars (US$), unless noted otherwise, with a base date of the second quarter 2024 (2Q24).

The foreign exchange rates adopted for the capital and operating estimates are shown in Table 21.1.

Currency	Exchange Rate	Exchange Rate
Australian Dollar	USD: AUD	1.58
Euro	USD: EUR	0.94
Great British Pound	USD: GBP	0.82
Egyptian Pound	USD: EGP	49.44
Canadian Dollar	USD: CAD	1.38
Swiss Franc	USD: CHF	0.96
South African Rand	USD: ZAR	16.9
West African Franc	USD: XOF	605

Table 21.1 Foreign Exchange Rates

21.2 Capital Cost Estimate Summary

The Project Capital cost estimate (mining, processing and infrastructure) developed for the Feasibility Study (FS) is based upon an Engineering, Procurement, and Construction Management (EPCM) approach for the process plant and infrastructure and contract mining for mine development.

The estimate includes all costs associated with project management, process engineering, design engineering, drafting, procurement, construction management, and commissioning services required to construct and commission the processing facility and its associated support infrastructure. Additionally, the estimate also includes costs related to the establishment of the mining services facilities, spare parts, and the provision of first fills and consumables required for the commencement of operations.

The estimate has been based upon preliminary engineering designs, material quantity estimates derived from these designs, multiple budget price quotations for major process equipment, and budget rates for the supply of bulk commodities. Unit rates for site installation works were based on market inquiries specific to the Project from contractors experienced in the construction of minerals processing projects in Côte d'Ivoire and throughout West Africa.

Pricing for the estimate was obtained predominantly during the first quarter of 2024 (1Q24). Pricing was provided in a variety of currencies, which was then converted to US$ using the foreign exchange rates shown in Table 21.1. The estimate accuracy is considered to be -10%/+15% based on the following:

- Developed engineering quantities from preliminary calculations and layout drawings;
- Budget quotations obtained for major items and site-based contract works;
- The capital cost estimate was broken down using a conventional Work Breakdown Structure (WBS) based on plant areas (i.e. crushing, coarse ore storage and handling, gravity recovery and separation, grinding and classification, etc.) and sub-categories of commodity groups (i.e., earthworks, concrete, structural steel, platework, piping, etc.).

21.2.1 Project Costs

The overall Project capital cost estimate has been summarised in Table 21.2.

Area	Total US$M
Mining, including pre-strip	22.4
Processing, including infrastructure	271.3
Owner's Costs	50.6
Project Contingency	29.0
Estimate Total (-10%/+15%)	373.2

Table 21.2 Project Capital Cost Estimate Summary

21.2.2 Mining Costs

The scope of work items included in the mine establishment capital cost estimate are summarised below:

- Pre-strip costs incurred up to the stage of first process plant production;
- Mobile mining equipment and ancillary equipment;
- Mine office complex, crib room and ablutions;
- Explosives magazine;
- Heavy vehicle and light vehicle workshops, including lube facility and stores;
- Wash down facility, including oily water separator;
- Tyre changing facility.

The capital cost estimate for mine establishment, including pre-stripping is US$22.4M. Further details of the mine establishment costs have been included in Part 16, Mining Methods.

The mining contractor's operating costs are included in Section 21.5.

21.2.3 *Processing Plant and Infrastructure Costs*

The Doropo Gold Project processing facility has been designed to process 4,000,000 tonnes per annum of fresh ore.

The process plant capital cost estimate includes the following facilities:

- ROM stockpile area, single stage crushing circuit and conveyors;
- Crushed ore stockpile and gravity reclaim system;
- Grinding, classification and gravity gold recovery;
- Leach feed thickening and Carbon in Leach circuit;
- Elution and gold recovery circuit;
- Cyanide destruction and tailings discharge (disposal) system;
- Reagents storages, mixing and distribution systems.

The infrastructure items included in the process plant capital cost estimate are summarised below:

- Access road to the mine, roads internal to the process plant and roads accessing infrastructure items such as the magazine and accommodation village;
- Accommodation village;
- Communications systems;
- Site offices, maintenance workshops, warehousing, stores and laboratory facilities;
- Utilities (water, air and power) and waste stream treatment requirements to support the Project (to the agreed battery limits and environmental guidelines).

The capital cost estimate includes all processing and infrastructure costs required prior to the commencement of production. The estimated processing plant and infrastructure capital costs are summarised in Table 21.3.

Area	Total US$M
Earthworks	6.8
Airstrip	1.5
Water Harvest Dam (WHD)	1.6
Site Access Roads	1.4
Haul Road	0.0
Sediment Control Structures	1.2
Crushing & Screening	5.3
Coarse Ore Storage & Handling	7.4
Grinding & Classification	32.2

Area	Total US$M
Pebble Crushing & Conveying	1.7
Gravity Recovery & Separation	1.2
Leaching & Adsorption	15.9
Gold Recovery	3.1
Reagent Mixing & Distribution	2.0
Power Reticulation - Plant	20.0
Power Connection	23.6
Water Storage & Reticulation	2.1
Raw Water Supply	0.5
Tails (Thickening &) Disposal	2.2
Tailings Dam	0.3
Tailings Storage Facility (TSF)	28.8
Tailings Return Water	0.3
Fuel Storage & Distribution	0.1
Air Services Supply & Reticulation	2.3
Administration Buildings & Offices	2.9
Plant Workshop/Stores	1.5
Laboratory	1.9
Village	9.6
Plant Piping	13.0
Project Management	7.2
Engineering & Drafting	9.0
Site Supervision & Management	11.6
Site Construction Cranes & Equipment	2.4
Site Construction Facilities	0.0
Commissioning	1.8
Mobile Equipment	3.2
Initial Fills	2.7
Spare Parts	4.1
Opening Stocks	6.3
Mobilisation & Demobilisation	26.7
Contractor Indirect Costs	6.1
Estimate Total (-10%/+15%)	271.3

Table 21.3 Processing and infrastructure Capital Cost Estimate Summary

21.2.4 *Owner's Costs*

A breakdown of the Owner's capital cost estimate is provided in Table 21.4, which includes all costs associated with Centamin's Owner's team, early works activities (including site access tracks, site facilities), mobile and fixed plant, insurances, HSE, and pre-production operating costs.

Area	Total US$M
Site Facilities (pioneer camp, temp. power, fuel, potable water, etc.)	7.3
Earthworks (road upgrades, access tracks, etc.)	2.7
Mobile & fixed plant	0.3
Owner's project team costs	12.0
Project meals and lodging	3.6
Office costs (Abidjan and Perth)	0.5
Computers, software & communications	0.6
Insurances & permitting	2.7
HSE	15.1
Owner Transport and Logistics	1.5
Operations pre-production	4.3
Estimate Total (-10%/+15%)	50.6

Table 21.4 Owner's Cost Estimate Summary

21.3 Basis of Capital Cost Estimate Development

21.3.1 *Methodology*

The estimate has been compiled from first principles to provide costs for each area of the work and sub-costs for each commodity utilised to construct each area.

The process design criteria and flowsheets were developed for the Project, from which preliminary plant equipment selections were made and plant general arrangement layout drawings were developed.

Sufficient engineering design was undertaken to ensure the constructability and operability of the layouts were considered, and adequate detail in the equipment specifications was established. This level of design detail was developed to enable material quantities to be estimated to the accuracy level required for the FS.

Competitive market pricing was sought for vendor-supplied equipment items, site labour, and bulk material supply rates, all of which were incorporated into the estimate. The site-based installation rates adopted include all applicable charges and indirect costs necessary to develop the total Project cost.

21.3.2 *Mining*

Budget pricing was sought from a range of suitably qualified mining contractors in West Africa, based on mine plans developed utilising the outcomes from the previous pre-feasibility study (PFS) for the Project. The mining cost estimate was then developed utilising the various physicals derived from the LOM production schedule and the contractor provided rates. This did not include Centamin's mining team and related costs which were developed by Centamin.

Further details of the mining estimating methodology are provided in Part 16 of this Technical Report.

21.3.3 *Earthworks*

Quantities

Plant and site infrastructure earthworks quantities were developed by GRES using topographical survey data obtained by Centamin and a 3D modelling software package to produce a bill of quantities.

Design and quantity development for the TSF, WHD, WSD, airstrip, access road, surface water management (SWM) and sediment control structures (SCS) were completed by Knight Piésold Pty Ltd (Knight Piésold) on behalf of Centamin. Included in the capital cost estimate are the construction costs forecast to be expended prior to first ore production, which comprises Stage 1 of the TSF and full construction of the WHD, WSD, airstrip, access road, SWM and SCS.

All costs related to future lifts for the TSF are included in the sustaining capital cost estimate, as well as the costs for construction of the haul road between the future Kilosegui pit and the plant ROM pad.

Earthworks Rates

Pricing was sought from reputable subcontractors to obtain representative rates for the various elements of the work.

It has been assumed that a single subcontractor will perform all the nominated earthworks for the Project. The rates adopted for the estimate were inclusive of equipment hire, operator and maintenance labour, fuel, consumables, materials, indirect costs, including accommodation and transport of the contractor's personnel, as well as margin.

The sustaining capital cost estimate utilises Knight Piésold's estimated costs.

21.3.4 *Concrete*

Concrete quantities were calculated for each area based on the general arrangement drawings, layout drawings, and preliminary design loadings developed for the FS.

Market rates for materials and labour were sought from reputable subcontractors to obtain representative rates for the various elements of the work. The rates obtained for site concrete supply and installation were inclusive of equipment, labour, fuel, consumables, materials, indirect costs, including accommodation and transport of the contractor's personnel, as well as margin for each category of work.

21.3.5 Structural Steel

Structural steel quantities were estimated for each area based on the general arrangement drawings, layout drawings, and preliminary structural designs developed for the Project.

Market rates from reputable steel fabrication subcontractors were applied to provide a basis for the estimated supply costs. The rates are ex-works and inclusive of shop detailing, materials, fabrication labour, painting materials and labour, consumables, indirect costs and margin for each type of structural steelwork (three mass classifications), conveyor gantries, grid mesh flooring, hand railing, and stair treads.

21.3.6 Equipment

The process design criteria were used to develop the mechanical equipment list, which defines the requirements and sizes of all the mechanical equipment. Specifications and/or data sheets were developed for all major equipment.

Written budget quotations from enquiries, accompanied by the engineering specifications and data sheets, were requested from recognised suppliers for all of the following major equipment in the plant:

- Primary crusher;
- Apron feeders;
- Vibrating feeders;
- Wet screens;
- Pebble crusher;
- Lime silo;
- Dust collectors;
- Mills;
- Cyclones;
- Gravity concentrator and intensive leach reactor;
- Thickener;
- Agitators;
- Interstage screens;
- Carbon regeneration kiln;
- Acid and elution columns;
- Elution heater;
- Drying oven;
- Smelting furnace;
- Electrowinning cells;
- Oxygen plant;
- Slurry Pumps;
- Process Pumps;
- Flocculant plant;
- Water treatment;
- Cranes and hoists.

Minor equipment items were priced using either sole-source email enquiries or from the GRES database.

The percentage by value of the mechanical equipment pricing that has been obtained by budget pricing offers received from multiple suppliers is provided in Table 21.5.

Description	Quantity
Budget Pricing	94.2%
Historical/Database Pricing	5.5%
Allowances	0.3%
Total	100%

Table 21.5 Mechanical Equipment Pricing

21.3.7 Platework

The process design criteria and mechanical equipment list developed for the Project were used to size the required fabricated steel plate tanks, bins, chutes, and launders. Material quantities were calculated by GRES on an item-by-item basis from the specified requirements, dimensions depicted in the layout drawings and/or developed, from plate work used for similar applications on previous projects.

Market rates for fabrication of plate work items were obtained from multiple subcontractors deemed capable of meeting the Project quality standards, schedule, and scope. These rates include materials supply, fabrication, surface preparation, and final painting in the shop, overheads, and margin.

The rates for the site-constructed tanks (including the leach, CIL, and cyanide destruction tanks) were obtained from reputable companies specialising in site tank construction.

21.3.8 Piping

A piping line list and valve list indicating size and specification was developed from the P&ID's and layout drawings. Process plant and general piping quantities, and pipes supports, were calculated using the layout drawings.

Overland piping quantities were calculated based on process requirements and geographical locations of the village, WHD, WSD and TSF.

Material supply rates were provided by piping suppliers for the various specifications of pipes, valves, and fittings. These rates were inclusive of materials, packing, overheads, and margin.

21.3.9 Electrical and Instrumentation

The electrical, instrumentation, and control quantities have been compiled using the Project scope, single-line diagrams, P&ID's, layout drawings, equipment list, and load list. The instrument list was also developed from the P&IDs.

Cable and material quantities were estimated based on layout, switch room locations, specific equipment requirements, and drive requirements.

Budget pricing was obtained for major electrical components (i.e. transformers, switch rooms, motor control centres (MCCs), variable speed drives (VSDs), medium voltage switchgear). Rates are inclusive of materials, overheads, and margin.

Overhead power lines were benchmarked on a supply and install per kilometre rate basis against recently completed installations. The rates adopted were inclusive of materials, installation, testing, freight, overheads, and margin.

21.3.10 Installation Labour

Construction and installation labour costs were based on estimated man-hours associated with the equipment and fabricated items to be installed in each area of the plant. The estimated hours for installation reflect the labour force productivity for Côte d'Ivoire construction sites and the application of industry-standard labour rates for each type of work.

Budget enquiries were issued to the market for each major discipline, including bulk earthworks, concrete works, SMPP installation and, EI&C installation.

GRES completed reviews of the tender submissions received before selecting the preferred contractor for each task. The rates provided by the preferred contractor were adopted for the estimate.

21.3.11 Owner's Mobile Plant

A list of Owner's mobile equipment was developed for the Project. These items will be purchased, with estimated costs for the mobile plant and light vehicles provided by Centamin.

21.3.12 Project Spares

Commissioning spares were included in the estimate, based on supplier submissions received for the mills, gravity concentrator, intensive leach reactor, slurry pumps, cyclones, interstage screens, electrowinning cells, smelting furnace, and oven.

An allowance of 0.25% of the mechanical supply cost was made for commissioning spares for the remainder of the process plant equipment. An allowance of 0.5% of the electrical supply cost was made for the electrical commissioning spares.

Capital/insurance spares were included in the estimate, based on supplier submissions, for the mills, vibrating screens, gravity concentrator, intensive leach reactor, slurry pumps, elution heater, electrowinning cells, smelting furnace, and oven.

An allowance of 1% of the mechanical supply cost was made for insurance spares for the remainder of the process plant equipment. An allowance of 2% of the electrical supply cost was made for the electrical insurance spares.

An additional allowance of 5% of the mechanical and electrical supply cost was also made as a provision for 3 months of operating spares for opening stocks.

21.3.13 Initial Fills

Allowances for first fills include the supply of consumables, grinding media, reagents and lubricants. The first fill quantities were derived from the process design criteria.

21.3.14 Plant Services and Infrastructure

The capital cost estimates compiled for the plant services and infrastructure components of the Project were based on requirements dictated by the current process plant design/capacity and plant layout.

The main items included in the FS scope of work, and therefore the capital cost estimate, have been summarised below:

- Administration and plant buildings including offices, toilet blocks and crib rooms;
- Gatehouse;
- Plant control room;
- Clinic and emergency response room;
- Plant security and change house building;
- Plant site workshop and stores buildings complete with portable offices;
- Reagent stores; and
- Laboratory, complete with equipment.

Budget enquiries were issued to reputable vendors for the supply of all buildings.

21.3.15 Other Infrastructure

The following work scopes have been designed and estimated by third parties:

- The works associated with the power supply connection for the Site have been designed and estimated by ECG Engineering (Ref 23). The scope includes upgrading the existing Bouna Substation (by extending the existing 90 kV bus), adding a 90 kV transmission line feeder, constructing 65 km of 90 kV single circuit lattice tower transmission line, and constructing a substation at the Doropo site.

21.3.16 *Accommodation*

The capital estimate includes provision for a 300-bed main camp to provide accommodation for salaried staff and an 80-bed security camp to accommodate security staff, including all amenities, communications, entertainment, and recreational facilities.

Enquiries were issued to relevant contractors for the supply and installation of these accommodation facilities. The proposed scope was inclusive of furnishing, electrical, plumbing, kitchen and laundry requirements, as well as entertainment and recreational facilities.

The accommodation facilities will be used during the construction phase to accommodate the owner's team and EPCM personnel while on site. Costs associated with accommodation of subcontractor personnel are included in the subcontractor's rates.

21.3.17 *Transport*

Major equipment suppliers were requested to provide estimates of the costs to transport their equipment to the site. Where suppliers did not include transport costs, an international logistics provider has supplied transport rates from supplier's works to the site. Sea freight and packing charges have been included in the transport rates.

Transport costs for fabricated steel items have been allowed based on quantities expected to be loaded per sea container or truck and freight costs per sea container or break-bulk freight costs between selected suppliers and the site. Where possible, fabricated steel components will be suitably packed onto skids and containerised to minimise handling costs.

21.3.18 *Indirect Onsite Costs*

Where applicable, mobilisation and demobilisation costs were requested in the budget inquiries issued to reputable subcontractors for each major discipline, including bulk earthworks, concrete works, SMPP installation and EI&C installation.

The responses were then used to determine the fixed and recurring preliminaries costs, including mobilisation and site establishment, off-site overhead costs, on-site overheads (facilities, equipment, project management, etc.), demobilisation, and site disestablishment.

The contractor's pricing included provisions for all costs associated with travel, accommodation, meals, and local transportation for the contractor's workforce.

21.3.19 *Engineering, Procurement and Construction Management*

The estimate for engineering, procurement, construction management, and commissioning services was based on the hours required for a multi-disciplined project delivery team, consisting of suitably qualified and experienced personnel, to successfully complete the project scope within the scheduled timeframe. The involvement of each design discipline was estimated based on the complexity of the individual tasks and benchmarked against actual costs incurred on recent similar projects.

Rates used for each resource category to generate the cost estimate were based on commercial charge-out rates levied in 2014. The rates are consistent throughout the estimated implementation programme, and no allowance has been made for any escalation of these rates.

21.3.20 *Growth*

Growth, commensurate with the level of design and estimating confidence, has been included in the direct costs for the capital cost estimate. These allowances have been based on the Project scope described in this FS Technical Report and do not include provision for any changes being made to the process flowsheet, major equipment selections or process plant layout and design during detailed design.

Growth allowances in the estimate vary for different types of costs according to the level of engineering development associated with equipment/materials pricing, estimates of material quantities, estimates of equipment and labour requirements, and site costs.

21.3.21 *Project Contingency*

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency allowance adopted for the estimate was based on the project scope as defined in this FS Technical Report and does not cover scope changes, design growth, or the listed qualifications and exclusions.

A contingency analysis has been applied to the estimate, considering scope definition, materials/ equipment pricing, and installation costs. Contingency applicable to various owner's cost inputs have been specified by Centamin.

The allowance calculated for the capital estimate was US$29.0M, or 8.4% of estimated project costs.

21.3.22 *Owner's Costs*

The capital cost estimate includes allowances for a project owner's team, early operations staff recruitment, and the associated equipment and overheads. The owner's cost estimate has been compiled by Centamin and includes the following:

- Owner's Project Management team;
- Operations team mobilisation;
- Owners Consultants;
- Insurances and permitting;
- Mining Contractor mobilisation and site establishment;
- Mine pre-strip;
- Pioneer camp;
- Site access roads and tracks;
- Potable water supply;
- Communications;
- Mobile and fixed plant;
- Site facilities, including temporary power, and secure laydown areas;
- Site accommodation and catering costs during the construction period;
- Security services during the construction period;
- Capitalised pre-production operating costs;
- HSES management and mitigation.

21.3.23 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- The capital cost estimate has been compiled to represent an EPCM project execution by a competent and proven engineering and construction organisation capable of delivering within the schedule and required budget accuracy in conjunction with the client's owner's team;
- Mining will be performed by an experienced and competent mining contractor;
- Prices of materials and equipment with an imported content have been converted to US$ at the exchange rates stated previously in this report. All pricing received has been entered into the estimate using native currency;
- Subcontractor rates and distributable costs include mobilisation/demobilisation, recurring costs, direct and indirect labour, construction equipment, construction cranes up to 100 t, materials, materials handling and offloading, temporary storage, construction facilities, off site costs, insurances, flights, construction fuel, tools, consumables, meals and PPE;
- An allowance for heavy lift cranes has been included in the estimate;
- The bulk commodities for earthworks, including imported material, assume that suitable construction/fill materials will generally be available from borrow pits within 5 km of the work fronts;

- Concrete supply at the site will be provided by a mobilised batching plant, with cement, aggregates, and sand sourced from local suppliers. Construction water is to be provided by the owner for use by the subcontractors and is assumed to be of good quality. Any costs associated with improving the construction water source or sourcing water for construction from off-site have not been included;
- Engineering quantities for the TSF, WHD, WSD, airstrip, access road, surface water management, and SCS have been provided by Knight Piésold, with subcontractor rates applied to complete the capital costs for these items. Knight Piésold subconsultant costs for design and construction supervision are included;
- The estimate includes an allowance for mill installation supervision by the vendor.

21.3.24 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Project sunk costs incurred by the owner prior to project implementation;
- Foreign currency exchange rate fluctuation from the nominated rates;
- VAT or other taxes;
- Interest charges or capital financing costs;
- Working capital; and
- Escalation of supply and contractor prices beyond the estimate base date.

Where relevant, these items have been addressed in the financial model.

21.3.25 Sustaining Capital

The development of the sustaining capital included inputs from:

- Knight Piesold - for the estimate phases of the:
  - TSF;
  - WSD;
  - Sediment control structures;
  - Haul road development from satellite pits;
- Piteau Associates - for the pit dewatering requirements;
- Centamin - for the ESG requirements including resettlement and loss compensation.
- Centamin - closure and rehabilitation costs.

Area	Total US$M
Infrastructure Phased Development	49.5
Pit Dewatering	4.5
ESG	6.0
Closure and Rehabilitation	36.0
Estimate Total (-10%/+15%)	96.0

Table 21.6 Sustaining Capital Estimate Summary

21.4 Operating Cost Estimate

21.4.1 Project Operating Costs

The Project Operating cost estimate (mining, processing, and infrastructure) developed for the FS is based on a mining services contractor model for the open pit mining. The ore will be treated by crushing, grinding, gravity gold recovery, and cyanide leaching in the CIL plant to produce gold doré.

The LOM operating costs for the Project and unit cost per tonne of ore treated are summarised in Table 21.7.

Project Area	LOM	
	Cost US$M	Unit Cost US$/t ore treated
Mining	869	22.7
Processing	462	12.10
General and Administration[1]	156	4.1
Total	1,486	38.9

Note: 1) General and Administration (G&A) costs presented above include Site G&A costs of US$147M over the LOM and mine lease / permitting costs of US$9M over the LOM.

Table 21.7 Project Operating Cost Estimate Summaries

21.5 Mining Operating Costs

A summary of calculated unit operating costs related to mining are provided in Table 21.8 on a US$/t mined and a US$/t ore treated basis.

Area	Total (LOM) US$	Total (LOM average) US$/t mined	Total (LOM average) US$/t ore treated
Mining Contractor Costs			
Early Works	15.58	0.07	0.41
Fixed Costs	124.75	0.55	3.26
Site Preparation	26.67	0.12	0.70
Drill and Balst	173.68	0.77	4.54
Load and Haul	363.57	1.61	9.51
Rehabilitation	3.27	0.01	0.09
Dayworks	20.76	0.09	0.54
Satellite Pit Ore Haulage	87.69	0.39	2.29
Rehandle	29.50	0.13	0.77
Owner's Mining Supervision Costs	23.5	0.10	0.61
Estimate Total	869	3.85	22.73

Table 21.8 Mining Operating Cost Estimate Summaries

Further details of the mining estimating methodology are provided in Section 16.10 of this Technical Report.

21.6 Processing Plant Operating Costs

21.6.1 General

The processing operating cost estimates are based on the provision of all new equipment in the plant and take into account costs associated with the existing site conditions and project location. The operating costs for the processing operation include reagents, consumables, labour, power, maintenance, and processing administration costs.

An estimate for the Site G&A costs is discussed in Section 21.7.

The processing facilities will operate 365 days per year and 24 hours per day. The following expected plant utilisations have been adopted:

- 80% for the crushing circuit;
- 91.3% for the milling and all downstream processing areas;

21.6.2 Estimating Method and Accuracy

Operating cost estimates were developed for the processing operation treating 4,000,000 t/annum of fresh Doropo ore or 5,400,000 t/annum of oxide-transition Doropo ore.

The processing operating cost basis reflects the nominated plant throughput, the process design criteria, the associated steady-state mass and water balance, and the reagent consumption rates determined via test work completed by Centamin. Reagent consumption rates have been calculated on a per dry tonne of mill feed basis, using the existing test work and calculated operating data for the process. The operating costs for the Doropo processing plant have been estimated to an accuracy of -10%/+15%.

The operating costs are presented in United States dollars (US$) and reflects an estimate base date of second quarter 2024 (2Q24).

The costs cover the processing of ore from the ROM pad battery limit. This includes the areas covering crushing, ore storage and reclaim, grinding, pre-leach thickening, leaching, adsorption, elution and gold recovery, cyanide detoxification and tails pumping, along with site services (power, air, water).

21.6.3 Summary

The operating cost estimate for the processing plant at full production is US$53,312,438 per year or US$13.33/t of fresh ore treated and US$57,311,027 per year or US$10.54/t of oxide-transition ore treated.

Summaries of the operating costs by cost centre are provided in Table 21.9, Figure 21.1 (fresh) and Figure 21.2 (oxide-transition).

Cost Centre	FRESH		OXIDE - TRANSITION	
	Cost (US$/year)	Unit Cost (US$/t)	Cost (US$/year)	Unit Cost (US$/t)
Power	19,687,868	4.92	17,899,757	3.31
Maintenance Spares & Consumables	2,937,056	0.73	2,937,056	0.54
Operating Consumables	19,143,700	4.79	24,517,299	4.54
Labour	10,721,474	2.68	10,721,474	1.99
Laboratory	415,340	0.10	415,340	0.08
Other	407,000	0.10	407,000	0.08
Total	53,312,438	13.33	57,311,027	10.54

Table 21.9 Processing Operating Cost Estimate Summaries by Cost Centre

In the 'Other' category, the dominant cost areas are contract labour (mainly for mill relines), metallurgical test work, lubricants and general processing plant consumables.

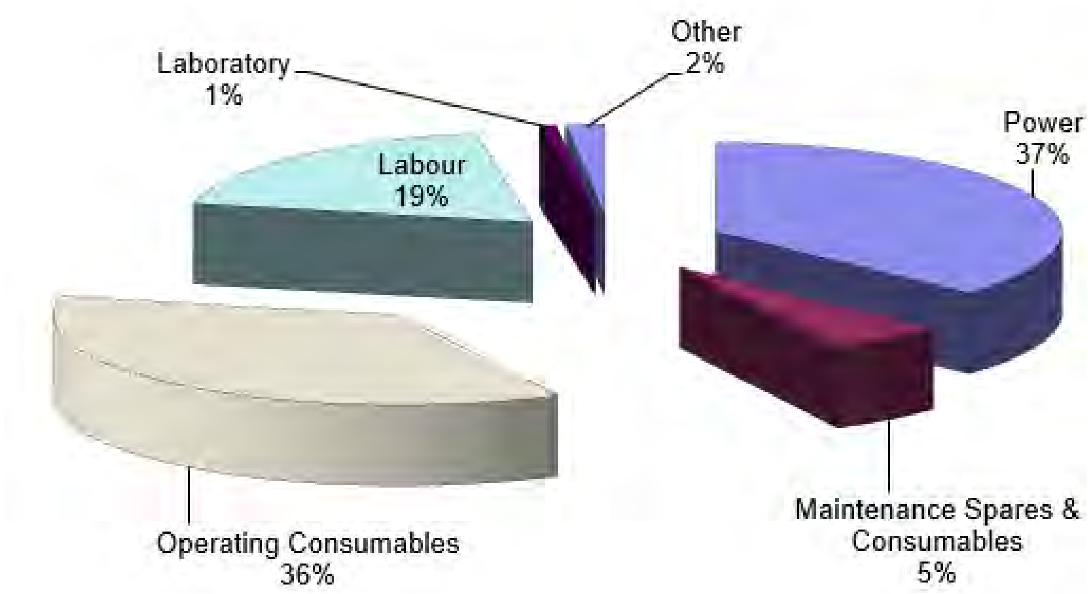


Figure 21.1 Processing Operating Cost Estimate Summary by Cost Centre (Fresh)

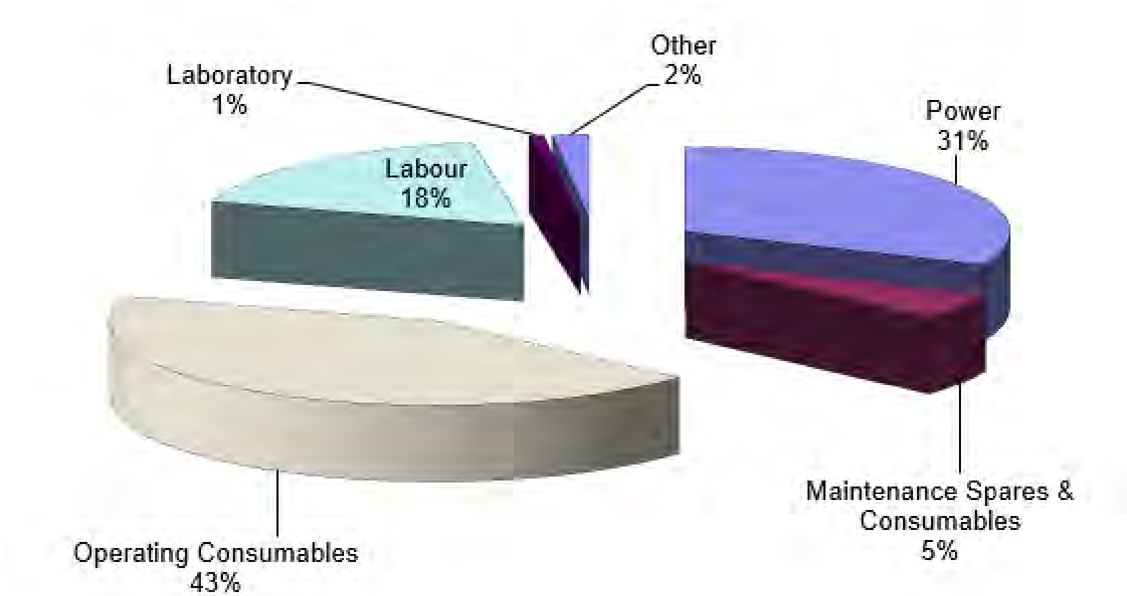


Figure 21.2 Processing Operating Cost Estimate Summary by Cost Centre (Oxide/Transition)

Summaries by plant functional area are provided in Table 21.10, Figure 21.3 (fresh) and Figure 21.4 (oxide-transition).

Plant Area	FRESH		OXIDE - TRANSITION	
	Cost (US$/year)	Unit Cost (US$/t)	Cost (US$/year)	Unit Cost (US$/t)
Crushing	725,512	0.18	725,512	0.13
Reclaim, Grinding & Classification	26,443,277	6.61	26,712,219	4.95
Carbon-In-Leach	6,031,425	1.51	9,003,075	1.67
Gold Recovery & Carbon Regeneration	1,424,413	0.36	1,573,967	0.29
Tailings Detoxification & Disposal	5,626,531	1.41	5,697,753	1.06
Reagents	1,173,153	0.29	1,173,153	0.22
Water & Air Services	1,500,040	0.38	1,624,162	0.30
Warehouse	50,658	0.01	50,658	0.01
Laboratory & Workshop	7,629,502	1.91	7,629,502	1.41
Administration	2,707,927	0.68	2,707,927	0.50
Total	53,312,438	13.33	57,311,027	10.54

Table 21.10 Processing Operating Cost Estimate Summaries by Area

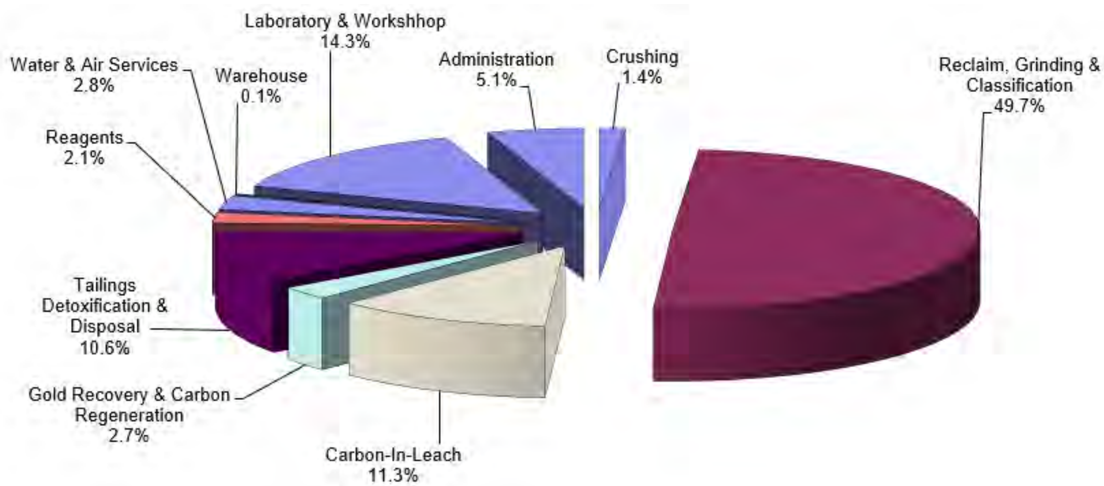


Figure 21.3 Processing Operating Cost Estimate Summary by Area (Fresh)

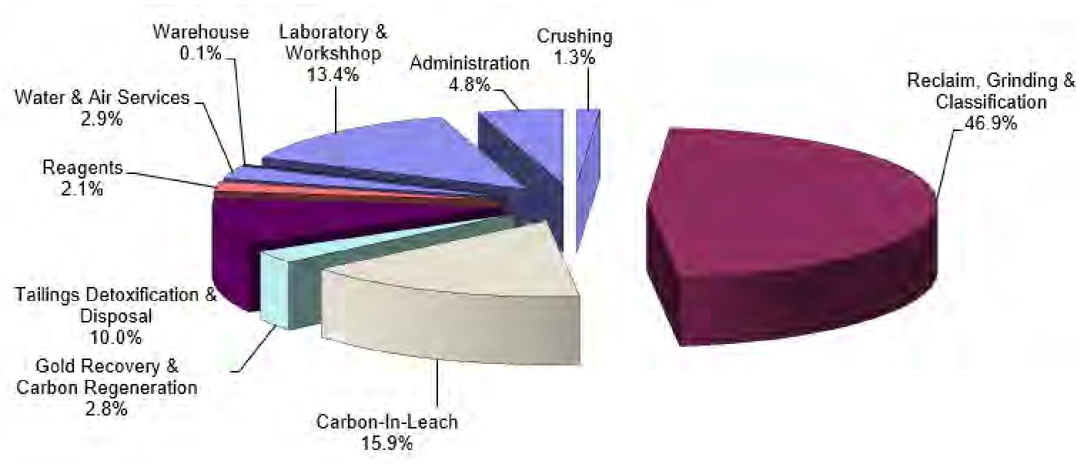


Figure 21.4 Processing Operating Cost Estimate Summary by Area (Oxide/Transition)

21.6.4 Operating Cost Centres

Power

The power consumption for the crushers, SAG, and ball mills has been calculated from comminution modelling using ore properties determined by metallurgical test work. The power consumption for the remainder of the processing plant has been calculated from the installed power of equipment drives with a utilisation factor for the plant against operational hours. The power consumption for the non-processing infrastructure has been estimated.

The average annual power draw and power cost by plant area are summarised in Table 21.11 for fresh feed and Table 21.12 for oxide-transition feed. The power requirements and costs for the non-processing infrastructure are also included.

The power cost estimate has been based on grid power at an average unit cost of US$0.125/kWh, as developed by ECG based on the estimated annual power consumption at the site.

Cost Centre	Power Consumed (kWh/year)	Power Cost (US$/year)	Power Unit Cost (US$/t)
Crushing	2,838,973	356,107	0.09
Reclaim, Grinding & Classification	125,455,662	15,736,539	3.93
Carbon-In-Leach	9,928,218	1,245,347	0.31
Gold Recovery & Carbon Regeneration	867,780	108,850	0.03
Tailings Detoxification & Disposal	4,940,824	619,753	0.15
Reagents	4,424,674	555,009	0.14
Water & Air Services	6,922,734	868,354	0.22

Cost Centre	Power Consumed (kWh/year)	Power Cost (US$/year)	Power Unit Cost (US$/t)
Warehouse	164,688	20,658	0.01
Laboratory & Workshop	1,278,194	160,330	0.04
Administration	134,904	16,922	0.00
Total	156,956,651	19,687,868	4.92

Table 21.11 Power Summary - Fresh Feed

Cost Centre	Power Consumed (kWh/year)	Power Cost (US$/year)	Power Unit Cost (US$/t)
Crushing	2,838,973	356,107	0.07
Reclaim, Grinding & Classification	111,200,393	13,948,429	2.58
Carbon-In-Leach	9,928,218	1,245,347	0.23
Gold Recovery & Carbon Regeneration	867,780	108,850	0.02
Tailings Detoxification & Disposal	4,940,824	619,753	0.11
Reagents	4,424,674	555,009	0.10
Water & Air Services	6,922,734	868,354	0.16
Warehouse	164,688	20,658	0.00
Laboratory & Workshop	1,278,194	160,330	0.03
Administration	134,904	16,922	0.00
Total	142,701,382	17,899,757	3.31

Table 21.12 Power Summary - Oxide-Transition Feed

Maintenance Spare Parts and Materials

Maintenance materials costs for the plant have been factored from the capital cost estimate. The allowance covers mechanical spares and wear parts but excludes crushing and grinding wear components and grinding media, which are included in the consumables cost.

The maintenance cost also excludes site maintenance labour, which is included in the labour cost, and the fuel consumption and maintenance of plant mobile equipment, which is included in the site General and Administration (G&A) costs.

Operating Consumables

Operating consumables include the costs for reagents and wear components such as crusher and mill liners, screening media, and gold recovery consumables.

- Grinding media of 125 mm for the SAG mill and 60 mm balls for the ball mill;
- Cyanide and lime used in the leaching process;
- Hydrochloric acid and sodium hydroxide used in elution;
- SMBS and copper sulphate used in cyanide destruction;
- Flocculant used in the pre-leach thickener;
- Diesel used as the fuel for the elution heater, regeneration kiln and smelting furnace;
- Wear liners for the crushers and grinding mills.

The diesel price of US$1.038/L, delivered to site, is based on a contractor submission obtained by Centamin. The remaining costs are based upon received vendor reagent pricing, combined with the consumptions per ore type determined by metallurgical test work. The costs for transport to site have been added.

Table 21.13 and Table 21.14 provides a breakdown of the reagents and grinding media cost estimate for the fresh and oxide-transitional ores.

Reagent/Consumable	Usage (kg/t)	Unit Cost (US$/kg)	Annual Cost (US$/year)	Unit Cost (US$/t)
Grinding Media (125 mm Balls)	0.430	1.06	1,815,434	0.45
Grinding Media (60 mm Balls)	0.840	1.02	3,436,695	0.86
Quicklime	0.300	0.45	542,553	0.14
Cyanide	0.367	2.70	3,965,475	0.99
Carbon	0.025	2.55	255,000	0.06
Flocculant	0.031	2.13	256,850	0.07
Hydrochloric Acid	0.053	0.22	46,400	0.01
Sodium Hydroxide	0.152	0.74	450,956	0.11
RO Water Treatment Reagents	0.020	0.63	50,633	0.01
Diesel	0.062	1.04	257,350	0.06
Sulphamic Acid	0.0002	9.17	5,503	0.00
SMBS	1.975	0.45	3,555,000	0.89
Copper Sulphate	0.070	2.61	731,080	0.18
Antiscalant	0.031	3.80	471,200	0.12
Primary Crusher Linings			146,821	0.04
SAG Mill Linings			2,145,000	0.54
Ball Mill Linings			310,772	0.08
Pebble Crusher Linings			44,970	0.01
Goldroom Fluxes and Consumables			41,746	0.005
Screen Media			33,630	0.01
PSA Plant consumables			22,000	0.01
Laboratory Consumables			117,089	0.03
Cyanide Sparge/Isotank rental			441,542	0.11
Total			19,143,700	4.79

Table 21.13 Fresh Ore Reagent Cost and Consumptions

Reagent/Consumable	Usage (kg/t)	Unit Cost (US$/kg)	Annual Cost (US$/year)	Unit Cost (US$/t)
Grinding Media (125mm Balls)	0.29	1.06	1,652,890	0.31
Grinding Media (60mm Balls)	0.66	1.02	3,645,352	0.68
Quicklime	1.33	0.45	3,247,181	0.6
Cyanide	0.388	2.7	5,657,040	1.05
Carbon	0.025	2.55	344,250	0.06
Flocculant	0.121	2.13	1,379,498	0.26
Hydrochloric Acid	0.039	0.22	46,401	0.01
Sodium Hydroxide	0.11	0.74	438,561	0.08
RO Water Treatment Reagents	0.02	0.63	68,354	0.01
Diesel	0.062	1.04	347,422	0.06
Sulphamic Acid	0.0002	9.17	7,429	0
SMBS	1.16	0.45	2,818,800	0.52
Copper Sulphate	0.11	2.61	1,550,934	0.29
Antiscalant	0.031	3.8	636,120	0.12
Primary Crusher Linings			146,821	0.03
SAG Mill Linings			1,501,500	0.28
Ball Mill Linings			310,772	0.06
Pebble Crusher Linings			56,213	0.01
Goldroom Fluxes & consumables			47,502	0
Screen Media			33,630	0.01
PSA Plant consumables			22,000	0
Laboratory Consumables			117,089	0.02
Cyanide Sparge / Isotank rental			441,542	0.08
Total [1]			24,517,299	4.54

Note: 1) Operating costs presented for oxide-transitional ores exclude coagulant. Thickening test work indicates coagulant would only be required for oxide-transitional ore from the Nokpa pit.

Table 21.14 Oxide-Transitional Ore Reagent Cost and Consumptions

The consumption rates of reagents and other processing consumables have been calculated from metallurgical test work and comminution circuit modelling. Where consumption rates could not be sourced from test work, rates have been estimated based on operational experience. No additional allowance for process upset conditions and wastage of reagents has been made.

Reagent costs have been sourced from budget quotations obtained by GRES or Centamin operations and in-house data.

The diesel fuel usage for the elution heater, carbon regeneration, and goldroom furnace has been calculated from first principles and are included in the reagent costs. The diesel consumption for mobile plant is included in the site G&A costs.

Allowances have been made for water treatment reagents and specialty processing supplies. General consumables and PPE are included elsewhere. Lubricants are included in the maintenance cost centre and are excluded from consumable costs.

Labour Costs

The operating labour cost for the processing plant is estimated to be US$10,721,474 per year, which equates to US$2.68/t for Fresh ores and US$1.99/t for oxide-transitional feeds. Table 21.15 provides a breakdown of the manning levels and salaries that contributed to the labour cost estimate.

The processing plant labour costs include all plant operation and maintenance personnel as well as laboratory personnel. The estimate includes the FIFO travel costs for regional and expatriate personnel.

Accommodation, messing, and bus transportation costs are excluded (captured in the site G&A costs) and owners mining personnel and management costs are also excluded (captured within the mining costs).

Position	Personnel	Roster	Total Cost US$/annum
Process and Administration			
Processing Plant Manager	1	6/2	486,369
Process Clerk	1	2/1	15,500
Senior Metallurgist	1	6/2	361,869
Plant Metallurgist	3	6/2	571,369
General Foreman	1	6/2	283,869
Mill Trainer	1	6/2	240,417
Shift Supervisor	4	2/1	296,286
Process Operator - Leading Hand	3	2/1	69,750
Process Operator	33	2/1	528,000
Day Crew Supervisor	1	2/1	57,071
Plant Day Crew	9	2/1	144,000
Goldroom Supervisor	2	6/2	479,333
Goldroom Operator	3	2/1	48,000
Metallurgical Technician	3	2/1	46,500
Laboratory Manager	1	6/2	286,417
Laboratory Supervisor	3	6/2	544,369
Laboratory Operator	15	2/1	360,000
Sub-Total	85		4,819,119

Position	Personnel	Roster	Total Cost US$/annum
Maintenance			
Maintenance Superintendent	1	6/2	343,869
Maintenance Planner	2	2/1	176,143
Maintenance Clerk	2	2/1	38,750
Mechanical Senior Supervisor	4	6/2	1,141,667
Mechanical Supervisor	3	2/1	921,750
Fitter Leading Hand	1	2/1	71,431
Fitter Senior	6	2/1	342,429
Fitter	6	2/1	96,000
Boilermaker Leading Hand	2	6/2	492,833
Boilermaker	6	2/1	96,000
Instrument Tech Leading Hand	2	6/2	492,833
Instrument Tech	3	2/1	48,000
Electrical Leading Hand	3	6/2	739,250
Electrician Senior	3	6/2	719,000
Electrician	6	2/1	96,000
Trade Assistant	9	2/4	86,400
Sub-Total	59		5,902,355
Total	144		10,721,474

Table 21.15 Processing Plant Labour Cost Summary

The manning levels and rosters used to determine the labour operating cost were based on similar operations. The estimate of the labour contingent has been based on a three-shift operation with allowances for leave and absenteeism coverage. Provision has been made in the manning numbers to accommodate annual and sick leave requirements.

The roster is based on expatriate personnel working six weeks on site and two weeks offsite, with regional and local staff working two weeks onsite with one week offsite.

The role positions, salaries, travel and on-costs have been provided by Centamin.

Laboratory Costs

The on-site laboratory will provide sample preparation and assay services for the mine grade control, exploration, plant and environmental samples. A total of approximately 129,000 samples per year have been allowed, with the laboratory set out to accommodate gold analysis via fire assay.

Routine water quality analysis and mineralogy are intended to be conducted off-site. External sample transport and consumables allowances are included in the estimate.

21.6.5 Operating Cost Areas

Crushing

The crushing and ore storage area cost is estimated to be US$725,512 per year, equating to US$0.18/t of fresh ore or US$0.13/t of oxide-transitional ore treated. The major costs are for power, maintenance spare parts, and primary crusher liners. The crusher will be fed by direct tipping and/or front-end-loader (FEL) load-out from ROM stockpiles. The operation and maintenance of the FEL are part of the mining contractor's scope.

Reclaim, Grinding and Classification

The reclaim, grinding, and classification area cost is estimated to be US$26,712,219 per year for oxide ore and US$26,443,277 per year for fresh ore, equating to US$4.95/t and US$6.61/t of ore treated respectively. The major costs include power, grinding media, SAG mill linings, maintenance spare parts, and quicklime.

Carbon in Leach

The CIL area cost is estimated to be US$9,003,075 per year for oxide-transitional ores and US$6,031,425 per year for fresh ore, equating to US$1.74/t or US$1.51/t of ore treated respectively. The major costs include sodium cyanide, power, flocculant, carbon, and maintenance spares.

Gold Recovery and Carbon Regeneration

The gold recovery area (including gravity recovery, carbon acid washing, elution, and carbon regeneration) is estimated to cost US$1,573,967 per year for oxide-transitional ore and US$1,424,413 per year for fresh ores equating to US$0.29/t and US$0.36/t of ore treated respectively. The major costs include labour, diesel, antiscalant, and maintenance spares.

Tailings Detoxification and Disposal

The tailings disposal area cost is estimated to be US$5,697,753 per year for oxide-transitional ore and US$5,626,531 per year for fresh ores, equating to US$1.06/t or US$1.41/t of ore treated respectively. The major costs include cyanide destruction reagents such as copper sulphate, SMBS, and sodium hydroxide, along with power and labour.

Reagent Storage and Distribution

The reagent storage and distribution area cost is estimated to be US$1,173,153 per year, equating to US$0.22/t for oxide-transitional ore or US$0.29/t of fresh ore treated. The major costs include power, as well as cyanide sparge and isotank rental.

Water and Air Services

The water and air services areas are estimated to cost US$1,624,162 for oxide-transitional ores and US$1,500,040 for fresh ores per year, equating to US$0.30/t and US$0.38/t of ore treated respectively. The major costs include power and antiscalant.

Warehouse, Laboratory and Workshop

The warehouse, laboratory, and workshop cost is estimated to cost US$7,680,160, equating to US$1.42/t for oxide-transitional ores and US$1.92/t for fresh ores treated. The major costs are salaries, expatriate and regional staff travel, and maintenance spares.

The leasing and running costs for the mobile equipment is covered under Centamin G&A.

Processing Administration

The processing administration costs relate to the management and operations labour costs not directly allocated to a plant area, as well as the power and maintenance of site offices and administration buildings.

The estimated annual cost is US$2,707,927, equivalent to US$0.68/t when processing fresh ores and US$0.50/t for oxide-transitional ores.

21.7 Site General and Administration

Site G&A costs include administration costs that relate to the overall site operations rather than specifically to mining or processing. The site G&A costs were provided by Centamin and include:

- Communications;
- Specialist service providers (e.g. site security, camp catering and cleaning, medical services and staff transport);
- External consultants (e.g. OH&S and environmental);
- Insurances;
- Financial costs (audit fees, legal fees and banking charges);
- General freight costs;
- Government charges (e.g. tenement fees and environmental licence);
- General office expenses.

The total estimated site G&A cost is US$17,316,490 per annum, equivalent to US$4.33/t when treating fresh ore and US$3.21/t when treating oxide-transition ore.

21.8 Exclusions

The following items are excluded from the operating cost estimate:

- All corporate head office costs;
- Withholding and other taxes;
- Any impact of foreign exchange rate fluctuations;
- Any escalation from the base date of the estimate;
- Any contingency allowance;
- Any land or crop compensation costs;
- Any rehabilitation or closure costs;
- Any royalties or community levies;
- Tailings storage facility maintenance and lifts, as well as rehabilitation and closure costs;
- Transport of gold from site, including vaulting, refining and payability charges.

Where relevant, these items have been addressed in the financial model.

TABLE OF CONTENTS

22. ECONOMIC ANALYSIS 1
22.1 Introduction 1
22.2 Summary 1
22.2.1 Sensitivity Tables 2
22.3 Principal Assumptions and Inputs 3
22.3.1 Depreciation 4
22.3.2 Company Tax 4
22.3.3 Refining Costs 4
22.3.4 Silver Credits 4
22.3.5 Royalties 4
22.3.6 Working Capital 4
22.3.7 Closure Costs 4
22.3.8 Other 4

LIST OF TABLES

Table 22.1 Economic Summary 2
Table 22.2 Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz 2
Table 22.3 Operating Expenditure % Change vs Gold Price US$/oz, $NPV_{8\%}$ in US$M 3
Table 22.4 Up-front Capital Cost % Change vs Gold Price US$/oz, $NPV_{8\%}$ in US$M 3

22. ECONOMIC ANALYSIS

22.1 Introduction

An economic analysis has been undertaken by Centamin and incorporates the Feasibility Study outputs including, milled tonnages and grades for the ore and the associated recoveries, gold price (revenue), operating costs, bullion transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The purpose is to provide an estimate of project cashflows and overall economics for evaluation.

The evaluation method assumes that the Project has been assessed on a 100% ownership basis, with no debt financing. The outputs are a Project Cashflow, Net Present Value ("NPV") at a 8% discount rate ("NPV8%") and the Internal Rate of Return ("IRR"). The basis of the estimate utilises inputs from Orelogy (Project physicals, namely mining and processing, and mining operating costs), GR Engineering Services (Processing plant and supporting infrastructure capital costs and processing operating and G&A costs), Knight Piésold (Project Infrastructure), ECG Engineering (HV Power Supply), and Centamin (owners G&A, relocation costs, social programmes and economic analysis).

Centamin will continue to review their tax position regarding, corporate income tax, withholding taxes and VAT to ensure that the DFS accurately reflects the most suitable and relevant taxation for the asset.

22.2 Summary

The results of the economic model show potential within the asset. The model applies a long-term gold price of US$1,900/oz, below consensus forecasts, on a flat line basis from commencement of production. The Project is estimated to produce 167 koz per annum over its 10-year mine life, at an average cash cost of US$892/oz gold produced and an average all-in sustaining cost (ASIC) of US$1,047/oz gold sold. The initial capital is expected to be US$373M.

The Project economics indicate a Pre-tax NPV8% of US$568M and an IRR of 40% and a Post-tax NPV8% of US$426M and an IRR of 34%. Table 22.1 provides a summary of the Projects physical, financial and economics metrics.

Economic Summary @ US$1,900/oz Au	Units	Value
Mine life	Years	10
LOM ore processed	kt	38,226
LOM strip ratio	w:o	4.9:1
LOM feed grade processed	Au g/t	1.53
LOM gold recovery	%	89%
LOM gold production	koz	1,667
Upfront capital cost	US$M	373

Economic Summary @ US$1,900/oz Au	Units	Value
Life of Mine Average:		
Gold, average annual production	oz	167
Cash costs per ounce	US$/oz	892
AISC per ounce	US$/oz	1,047
Project years 1 to 5		
Gold, average annual production	oz	207
Cash costs per ounce	US$/oz	817
AISC per ounce	US$/oz	971
Pre-Tax Economics		
Net present value - 8%	US$M	568
Internal Rate of Return	%	40%
Post-Tax Economics		
Net present value - 8%	US$M	426
Internal rate of return	%	34%
Payback period	Years	2.1

Table 22.1 Economic Summary

22.2.1 Sensitivity Tables

The after tax NPV sensitivity (in US$M) comparing varying discount rate percents and gold price in US$/oz is presented in Table 22.2. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

Discount Rate	1,500	1,600	1,700	1,800	1,900	2,000
5%	199	275	364	454	543	613
6%	175	247	332	416	501	568
7%	153	221	301	382	462	526
8%	132	197	273	350	426	487
9%	113	174	247	320	393	450
10%	95	154	223	293	362	417

Table 22.2 Sensitivity in US$M of Discount Rate vs Gold Price in US$/oz

The after tax NPV sensitivity (in US$M) comparing operating expenditure fluctuations with varying gold prices in US$/oz is presented in Table 22.3. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

% Change	1,500	1,600	1,700	1,800	1,900	2,000
-20%	316	380	457	533	610	670
-10%	224	288	365	442	518	578
0%	132	197	273	350	426	487
10%	40	105	182	258	335	395
20%	-51	13	90	167	243	303

Table 22.3 Operating Expenditure % Change vs Gold Price US$/oz, $NPV_{8\%}$ in US$M

The after tax NPV sensitivity (in US$M) comparing up-front capital expenditure fluctuations with varying gold prices in US$/oz is presented in Table 22.4. The reported result for the Doropo Gold Projects after tax economic performance is highlighted in bold.

% Change	1,500	1,600	1,700	1,800	1,900	2,000
-20%	201	265	342	419	495	555
-10%	166	231	308	384	461	521
0%	132	197	273	350	426	487
10%	98	162	239	316	392	452
20%	64	128	205	281	358	418

Table 22.4 Up-front Capital Cost % Change vs Gold Price US$/oz, $NPV_{8\%}$ in US$M

22.3 Principal Assumptions and Inputs

The economic evaluation for the project was based upon:

- Capital cost estimates prepared by GR Engineering, ECG, Knight Piésold and Centamin;
- Mine schedule and mining operating cost estimates based on contract mining prepared by Orelogy with inputs from Centamin;
- Mining Technical Services for labour and grade control were estimated by Centamin;
- Process operating cost estimates prepared by GR Engineering;
- General and administration (G&A) cost estimates prepared by Centamin;
- Metallurgical performance characterised by test work conducted on representative composite samples from the Doropo deposits;
- Typical sustaining capital cost estimates for the infrastructure, relocation and closure as estimated by Knight Piésold and Centamin;
- Côte d'Ivoire government royalties. The cash flow analysis excludes any effects due to inflation;
- A gold price of US$1,900/oz;
- Construction completed over a 24-month period shown in the 'Pre-production' period;
- The financial assessment has been undertaken in United States Dollars (US$).

22.3.1 Depreciation

Provision has been made for depreciation using a unit of production method for all capital.

22.3.2 Company Tax

Corporate tax rate of 25% of taxable profit has been used.

22.3.3 Refining Costs

A typical refinery gold payable rate of 99.95% and an additional US$4.00/oz freight plus insurance charge has been used based on recent comparable studies.

22.3.4 Silver Credits

No silver production or revenue has been factored into the model.

22.3.5 Royalties

The Côte d'Ivoire royalty rate for gold is as defined below:

- 3.0% if gold price is ≤ US$1,000/oz;
- 3.5% if gold price is >US$1,000 and ≤US$1,300/oz;
- 4.0% if gold price is > US$1,300 and ≤ US$1,600/oz;
- 5.0% if gold price is > US$1,600 and ≤ US$2,000)/oz;
- 6.0% if gold price is US$ (>2,000)/oz.

In addition to this there is an additional royalty of 0.5% of revenue paid to a community development fund.

22.3.6 Working Capital

Allowance for working capital has been included in the Owners cost estimate.

22.3.7 Closure Costs

In the economic model, rehabilitation is ongoing during the operation, with an additional final allowance for demolition and closure activities to occur after mining and processing have ceased. The total allowance for rehabilitation and closure in the model is US$36M.

22.3.8 Other

- The cash flow model is based on 100% project ownership, i.e. no allowance for minority shareholders and government free carry;
- No provision has been made for interest or cost of capital;
- No provision has been made for escalation or inflation;

- A US$8M provision has been made for salvage value of the remaining assets upon cessation of mining and processing activities.

Flat US$1,900/oz Au	Unit	LOM	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11-Y14
MINING SCHEDULE															
Total material moved	kt	225,788	-	-	23,312	27,775	27,519	28,000	28,000	26,287	22,910	22,000	16,364	3,621	-
Total waste moved	kt	187,563	-	-	19,198	22,919	23,180	23,616	23,287	21,717	18,485	18,114	13,929	3,119	-
Total ore mined	kt	38,226	-	-	4,115	4,856	4,339	4,384	4,713	4,570	4,425	3,886	2,434	503	-
Stripping ratio	w:o	4.9x	-	-	4.7x	4.7x	5.3x	5.4x	4.9x	4.8x	4.2x	4.7x	5.7x	6.2x	-
Au grade - ore mined	g/t	1.53	-	-	1.66	1.52	1.89	1.64	1.44	1.40	1.30	1.27	1.61	1.82	-
Contained Au - ore mined	koz	1,876	-	-	220	238	264	232	218	206	185	159	126	29	-
PROCESSING SCHEDULE															
Total ore processed	kt	38,220	-	-	3,867	4,735	4,151	4,426	4,483	4,484	4,443	4,477	2,646	507	-
Au grade - processed	g/t	1.53	-	-	1.71	1.55	1.92	1.64	1.46	1.40	1.30	1.23	1.56	1.81	-
Contained gold - processed	oz	1,876	-	-	212	235	256	234	211	202	186	177	133	30	-
Au recovery	%	89%	-	-	94%	91%	88%	89%	88%	87%	88%	87%	87%	87%	-
Recovered gold	koz	1,667	-	-	200	213	225	208	187	176	163	154	115	26	-
CASHFLOW SUMMARY															
Gross revenue	US$m	3,166	-	-	379	405	427	395	355	335	310	293	218	49	-
Less: Royalties	US$m	(174)	-	-	(21)	(22)	(23)	(22)	(19)	(18)	(17)	(16)	(12)	(3)	-
Less: RC & transport	US$m	(7)	-	-	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(0)	(0)	-
Net revenue	US$m	2,986	-	-	358	382	403	372	334	316	292	276	206	46	-
Operating Costs															
Mining	US$m	(869)	-	-	(66)	(98)	(97)	(117)	(119)	(114)	(97)	(86)	(61)	(13)	-
Processing	US$m	(462)	-	-	(41)	(55)	(54)	(54)	(55)	(55)	(54)	(55)	(33)	(7)	-
Site G&A	US$m	(156)	-	-	(13)	(18)	(18)	(18)	(18)	(18)	(18)	(18)	(12)	(2)	-
Total operating costs	US$m	(1,486)	-	-	(121)	(171)	(169)	(190)	(192)	(187)	(170)	(159)	(106)	(22)	-
Operating Margin	US$m	1,499	-	-	237	211	234	182	142	129	123	117	100	24	-
Construction capital	US$m	(373)	(165)	(188)	(21)	-	-	-	-	-	-	-	-	-	-
Sustaining capital	US$m	(60)	-	-	(5)	(9)	(11)	(10)	(9)	(7)	(5)	(4)	(1)	-	-
Closure & Rehab (incl. bond payments)	US$m	(36)	-	-	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(2)	(1)	(10)	(17)
Change in working capital	US$m	-	-	-	(25)	(7)	(0)	(2)	1	1	3	2	8	15	4

Flat US$1,900/oz Au	Unit	LOM	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11-Y14
Net cashflow pre-tax	US$m	1,038	(165)	(188)	186	195	222	169	133	123	120	113	106	29	(5)
Income tax	US$m	(228)	-	-	-	(44)	(36)	(40)	(28)	(19)	(16)	(16)	(14)	(13)	(1)
Net post-tax cashflow	US$m	810	(165)	(188)	186	151	186	128	105	103	103	98	92	16	(5)
Cumulative post-tax cashflow	US$m		(165)	(353)	(166)	(16)	170	299	404	507	610	708	800	816	810
Cash operating cost	$/oz	892	-	-	605	802	752	915	1,029	1,058	1,038	1,030	922	858	-
All-in sustaining cost	$/oz	1,047	-	-	741	955	911	1,079	1,190	1,211	1,183	1,176	1,048	1,346	-

Table 22.5 Forecast Project Cash Flows

TABLE OF CONTENTS

23. ADJACENT PROPERTIES..1

23. ADJACENT PROPERTIES

No operating mines or active exploration is occurring adjacent to the Doropo Gold Project.

TABLE OF CONTENTS

24. OTHER RELEVANT DATA AND INFORMATION 1
24.1 Project Implementation 1
24.1.1 Overview 1
24.2 Development Methodology 1
24.2.1 Mine Development 1
24.2.2 Processing Plant and Infrastructure 2
24.2.3 Horizontal Packages 2
24.3 Project Management 3
24.3.1 Owner's Team 3
24.3.2 Principal EPCM Contractor 3
24.3.3 Project Execution Plan 4
24.3.4 Project Controls, Planning and Scheduling 6
24.4 Implementation 6
24.4.1 Implementation Schedule 6
24.4.2 Basis of Schedule 8
24.5 Engineering 10
24.6 Procurement and Logistics 10
24.7 Construction 11
24.7.1 General 11
24.7.2 Construction Works 12
24.7.3 Health and Safety 12
24.7.4 Environmental Management 12
24.7.5 Employee and Industrial Relations 13
24.7.6 Construction Quality Control 13
24.8 Completions and Commissioning Process 13
24.9 Operations Readiness 14

LIST OF FIGURES

Figure 24.1 Summary of Project Implementation Schedule 7

LIST OF TABLES

Table 24.1 Key Milestone Dates for Project Implementation 8

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

24.1.1 Overview

The Project implementation approach proposed for the Doropo Gold Project is for Centamin (the 'Owner') to engage a principal Engineering, Procurement and Construction Management (EPCM) Contractor to provide design, procurement, and construction management services for the delivery of the processing plant and selected infrastructure facilities, which will be handed over to the Owner's operating team on completion.

The development of the mine and the design and construction of the tailings storage facility (TSF), water harvest dam (WHD), water storage dam (WSD), airstrip, incoming 90 kV transmission line and 90 kV/11 kV switchyard, security camp and main camp will be undertaken by specialist consultants/contractors directly engaged by the Owner.

The capital cost estimate discussed in Part 21 and preliminary implementation schedule discussed below have been developed based on an EPCM implementation strategy, which aligns with this approach.

Centamin will establish an Owner's representative team to manage the requirements for the delivery of the Doropo Gold Project.

The principal EPCM Contractor will prepare a comprehensive Project Execution Plan (PEP) for the Project in conjunction with the Owner. This plan will define in detail the implementation methodology and strategies for delivering the design, engineering, procurement, construction and commissioning the Project successfully, in accordance with the requirements of the Owner and the agreed project objectives.

The plan will ensure that a consistent approach will be adopted by all parties delivering the Project, and will be a mandatory control document for all aspects of the Project.

24.2 Development Methodology

24.2.1 Mine Development

All mine development activities will be undertaken by a contractor selected and engaged by the Owner. The contractor will provide the physical infrastructure required to perform the services under contract.

The Mining Services Area (MSA) will be located south of the processing plant, with suitable access to the starting mine pit, waste dumps and ROM pad from mine haul roads. The MSA will consist of mine support facilities and infrastructure such as offices and mining workforce facilities, maintenance workshop, wash bay, re-fuelling and oily-water treatment facilities.

24.2.2 *Processing Plant and Infrastructure*

The implementation strategy for the process plant and selected infrastructure is an EPCM approach, where an EPCM Contractor will be responsible for managing all aspects of the design and procurement, field engineering, quality assurance and control, safety and commissioning under the supervision of the Owner's team.

The EPCM Contractor will also provide key supervisory roles for construction activities under the direction of an EPCM Construction Manager, who will report to the EPCM Project Manager.

24.2.3 *Horizontal Packages*

Several work packages will be split out from the principal EPCM Contractor's scope and managed directly by the Owner.

The development of the access road and WHD will be early activities, with the design and construction managed directly by the Owner's team, along with construction of the security camp, which will be utilised during the early stages of the construction phase.

The design and construction management of the incoming 90 kV transmission line and 90 kV/11 kV switchyard will be undertaken by a specialist consultant, managed by the Owner's team.

The design and construction of the main camp will be carried out by a specialist contractor, also managed by the Owner's team.

The design of the TSF, WSD and airstrip will be handled by a third-party technical consultant under the management of the Owner's team. The construction management of these facilities will also be undertaken by the Owner's team.

The timing of early works construction activities will be critical to ensure they coincide with the dry seasons, maximising performance and productivity, particularly for earthworks and initial/major civil works. These activities are to be completed by the Owners team ahead of the Final Investment Decision (FID) to facilitate overall project schedule compression as indicated.

Owner activities, including the construction of access roads, accommodation, water supply for construction, and bulk earthworks, have been carefully planned to ensure subsequent activities can proceed under the most favourable environmental conditions. This is especially important for constructing the WHD before the first wet season to enable early water harvesting.

24.3 Project Management

24.3.1 Owner's Team

Centamin will establish a Project Owner's team to monitor all aspects of the Project's development and implementation.

The Owner's team will:

- Provide project management services to oversee the in-country and offshore activities of the principal EPCM Contractor;
- Co-ordinate the specialist consultants and contractors engaged directly by the Owner for the provision of the horizontal packages and other services not included in the principal EPCM Contractor's scope;
- Review and approve engineering designs, providing specialist technical input;
- Provide site construction expertise, supervision and management of the site works for horizontal package scopes;
- Formulate and manage mine development with the selected Contractor;
- Oversee the operation of the accommodation facilities developed for the Project;
- Liaise with community and authorities for project reporting and approvals; and
- Ensure that the required operations readiness activities are undertaken within the construction period, including implementation of the asset management system, execution of supply contracts, recruitment and training of the workforce and ensure all approvals required to operate are in place.

24.3.2 Principal EPCM Contractor

The principal EPCM Contractor will provide a range of project management, engineering, drafting, international and in-country procurement, contract management, fabrication management, logistics co-ordination, construction management and commissioning services necessary to provide a complete, safe, quality and technically compliant processing plant and infrastructure. This will also include working with the Owner's team to assist with procurement and contract management of free issued equipment and services.

The Project management tasks will include:

- Preparing project-specific management plans and implementing the Project in compliance with these plans;
- Establishing project controls to provide the planning, monitoring and reporting through which the Project will be managed to meet its scope, cost, time and reporting requirements;
- Preparing equipment specifications, reviewing tenders, procuring equipment, and expediting equipment deliveries;

- Managing detail design works, including resource allocation and progress review to ensure that design deliverables are completed within a suitable time frame to enable the works to proceed unhindered;
- Organising site resources, both internal and external, and ensuring that appropriate resources are available, at the right time, to complete the site works in accordance with the Project schedule;
- Managing pre-commissioning activities.

Detailed organisation charts will be developed in the Project Execution Plan further defining key offsite and onsite EPCM roles. Some of the key offsite EPCM roles are listed below:

- Project Sponsor;
- Project Manager;
- Procurement and Contracts Manager;
- Safety Manager;
- Quality Control Manager;
- Project Cost Controller;
- Senior Project Engineer;
- Process Manager;
- Engineering Manager;
- Senior Process Engineer;
- Senior Mechanical Engineer;
- Senior Civil Engineer;
- Senior Structural Engineer;
- Senior Electrical Engineer;
- Lead Drafter.

Overall responsibility for health and safety, scope, schedule, budget, and quality within the boundaries of the EPCM Contract will reside with the EPCM Project Manager. The EPCM Project Manager will be supported at a corporate level by the EPCM peer review team, who will act as coordinators and advisors regarding the EPCM Contractor's corporate quality requirements.

24.3.3 Project Execution Plan

The EPCM Contractor will prepare a detailed Project Execution Plan (PEP) for the Project in conjunction with the Owner's team. This plan will define the implementation methodology and strategy for delivering the design, engineering, procurement, construction and commissioning of the Project, successfully and in accordance with the requirements of the agreed Project objectives.

The plan will ensure that a consistent approach is adopted by all parties managing the Project, and it will serve as a mandatory control document for all aspects of the Project.

The PEP will cover:

- The scope of work for the Project and clear definition of the various scope battery limits;
- The Project deliverables;
- How the EPCM Contractor's project team will manage the execution of the Project schedule through the various phases of the Project from contract award through to completion and commissioning;
- How the EPCM Contractor's project team will manage resources and subcontractors;
- Guidance on the interface and reporting between the EPCM Contractor's project team and the Owner's team during design, procurement, construction and commissioning;
- Details of the Project control tools the EPCM Contractor's project team will use to monitor and control progress of the works and relevant reporting formats.

The PEP will be the primary document of a suite of documents as listed below which will apply specifically to the Project and will be progressively developed by various team members and entities during the initial phase of the Project:

- Engineering Management Plan;
- Procurement and Contracting Plan;
- Logistics and Materials Management Plan;
- Construction Management Plan;
- Construction Safety Management Plan;
- Construction Environmental Management Plan;
- Human Resources and Industrial Relations Management Plan;
- Risk Management Plan;
- Quality Management Plan;
- Community and Cultural Affairs Management Plan;
- Commissioning Plan.

The Engineering Management Plan developed for the Project will define the principles and guidelines to be adopted by the EPCM's design team during the design phase of the Project. It will also describe the various engineering deliverables to be handed over during the various phases of the Project, from procurement through to plant handover and project close-out.

The Safety Management and Environmental Management Plans will be issued to all contractors tendering for site work as part of the enquiry document. Each Contractor will be required to demonstrate a satisfactory prior commitment to safety and environmental management and present a site-specific plan for their proposed involvement in the Project.

24.3.4 Project Controls, Planning and Scheduling

A Project Controls Plan will be implemented with two primary objectives:

- To monitor progress against expectations and provide guidance for when corrective action is required;
- To control and manage change to the core objectives.

Upon commencement of the Project, the EPCM Contractor will develop a detailed schedule for the works. The Project scope, project control estimate, and schedule shall form the baseline against which progress will be measured. Progress will be monitored and earned value management principles used to generate an S-curve based upon calculated actual cost, planned value and earned value.

Progress will be measured against the following key deliverables, with the method of measurement being dependent upon the nature of the deliverable:

- The engineering portion of the work will be monitored against the creation of design documentation, including development models and the issue of drawings and design documents marked 'Approved for Construction';
- The procurement portion will be measured against key milestones for each package such as the issue of tenders, package award and receipt of goods;
- The construction portion will be measured against physical quantities of material installed and estimates of installation progress and construction man-hours.

Changes to the Project baseline will only occur as approved variations under the EPCM Contractor's variation management procedure. Regardless of which party instigates the proposed change, the effects to scope, cost and schedule will be fully quantified with the variation agreed by both parties before proceeding.

A construction risk assessment workshop will be held prior to the commencement of site works to identify and mitigate potential issues.

24.4 Implementation

24.4.1 Implementation Schedule

A preliminary implementation schedule has been developed for the Project based on an EPCM execution methodology. The schedule outlines the execution of the Project scope, with the exception of mining and government approvals, which will be commenced in advance of the implementation program.

The schedule covers the major EPCM activities required to deliver the Project including:

- Detailed design engineering;
- Procurement and delivery;
- Construction; and
- Commissioning.

A high-level summary of the preliminary implementation schedule for the Project is provided in Figure 24.1.

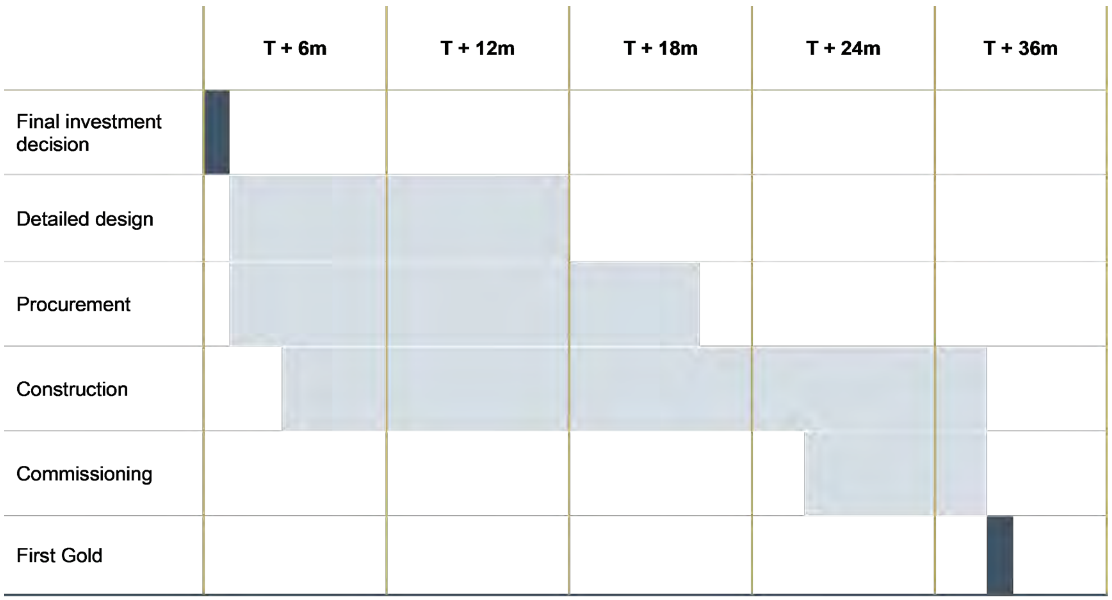


Figure 24.1 Summary of Project Implementation Schedule

The overall duration for the delivery of the processing plant and infrastructure component of the Project has been estimated to be 117 weeks from award of the EPCM Contract to Practical Completion. First gold is scheduled to be poured in week 118.

The significant schedule milestone dates are provided in Table 24.1.

Milestone	Date
Award of EPCM	3-Feb-25
Commence Procurement of Long Lead Equipment	4-Feb-25
Commence Process Plant Earthworks	30-Jun-25
Commence Process Plant Concrete Works	13-Oct-25
Detailed Design and Engineering Completed	9-Jan-26
Commence Dry Commissioning	6-Mar-26
Practical Completion	1-May-27

Milestone	Date
First Ore to Plant	3-May-27
First Gold Poured	10-May-27
Commissioning Complete and Handover	5-Jun-27

Table 24.1 Key Milestone Dates for Project Implementation

The schedule has allowed for a period of early engineering prior to the final investment decision, aiming to achieve the shortest possible timeline for the overall project development. This period will concentrate on finalising the layout/process design criteria and the development of engineering deliverables relating to the tendering and award of early contracts, including bulk earthworks, camp facilities, and related infrastructure.

24.4.2 Basis of Schedule

Schedule Class/Level

The implementation schedule has been developed to a level 3 detail.

Work Breakdown Structure

The work breakdown structure for the schedule is aligned with the structure utilised in Part 21 and adopted for the Capital Cost Estimate.

Project Calendars

The following calendars have been created and assigned to the relevant tasks in the Project schedule:

- Engineering design and project management - 8 hours per day, 5 days per week;
- Procurement - 8 hours per day, 5 days per week;
- Manufacturing and delivery - vendor durations;
- Mobilisation and site establishment - 8 hours per day, 5 days per week;
- Construction, construction in-directs and commissioning - 10 hours per day, 6 days per week (60 hours per week).

The calendars incorporate relevant public holidays and a Christmas break period of two weeks.

The durations for activities undertaken during the wet season have been estimated giving appropriate consideration to a reduced level of productivity expected during those periods.

Basis of Activity Durations

The schedule uses “Fixed Duration and Units” for all tasks in the schedule.

Schedule durations have been manually and independently set for each task to match the effort and resource levels required. These estimates have been based on data provided by construction contractors and experience with similar projects completed by GRES.

Resourcing

Construction resources covering all disciplines have been created in the schedule, with all direct and indirect man-hours derived from the construction contractors tender submissions and the capital cost estimate for construction activities.

Engineering and Design

Detailed design and drafting has been planned to a combine area and discipline level for the EPCM scope.

Procurement and Contracts

Procurement activities have generally been based on the following durations:

- Equipment and materials - 6 weeks from preparation of tender documentation through to evaluation and award;
- Construction contracts - 10 weeks from preparation of tender documentation through to evaluation and award.

Vendor data is assumed to be available 4 weeks after award, to allow engineering design to progress.

Fabrication and Delivery Lead Times

Lead times (delivered to site) for major equipment items have been based on budget pricing and delivery submissions received from reputable vendors.

The supply and fabrication of materials, including structural steel and platework, have been based on budget pricing and delivery submissions received from reputable fabricators.

Critical Path

The schedule critical path is driven by the following activities:

- Mobilisation for civil works;
- Leaching area civil works;
- Leaching area tank construction;
- Leaching area steelwork installation;
- Completion of leaching area mechanical installation (agitators);
- Completion of leaching area cable terminations;

- Completion of pre-commissioning;
- Dry and wet commissioning; and
- Ore commissioning and ramp up.

Near critical activities include:

- Confirmation of plant layout and mill tender specifications;
- Plant bulk earthworks design;
- Procurement of grinding mills;
- Tender and award of plant earthworks;
- Plant earthworks;
- Grinding mills installation.

24.5 Engineering

The EPCM Contractor will undertake design development in accordance Engineering Management Plan. The design work will include the necessary design development to finalise the documentation for the works including project specific basis of design, flowsheets, process design criteria and mass balances, process materials, equipment selection and sizing, design parameters and the site and plant layouts necessary to meet the intended use.

In collaboration with the Owner's team, the EPCM Contractor will determine the required process design criteria and controls functionality necessary for the processing plant to achieve the intended performance and purpose and to have made appropriate selections of process, plant, equipment and all other necessary selections.

The layout, equipment selection and accessibility within the facilities will also take into consideration the safety, ergonomics, engagement and general ease-of-use for onsite operators and maintainers.

Design in each area will be performed by the respective area engineering and design teams with overall coordination performed by the Project Engineering Manager who will direct the technical and resourcing aspects of this portion of the Work in conjunction with the Project Manager.

24.6 Procurement and Logistics

The EPCM Contractor will undertake procurement and contracts management for the Project. Evaluation and recommendation for award of packages will be performed jointly, with the Owner's team undertaking commercial evaluations and the EPCM Contractor managing technical evaluations. The Owner will place purchase orders for all major trade packages including process equipment packages, shop fabricated structural steel, plate work, piping and site works packages.

The EPCM Contractor will manage all contract works.

Equipment and material packages will be competitively tendered, except where the Owner specifies a particular supplier or make of equipment or where sole sourced equipment is justified and agreed with the Owner. Equipment and material packages will be sourced from local, in continent and international suppliers.

The key considerations in developing the procurement strategy will be the mitigation of risk associated with the Project, exposure to cost overruns and exposure to delivery time overruns.

Equipment suppliers will be selected on the basis of technical compliance, previous performance and availability to supply relevant equipment within the Project timeline. The selection criteria for various fabrication and construction contracting organisations will include a combination of track record and reputation, quality, safety, location, capacity, commitment and performance to the schedule and price.

A logistics services provider will be engaged to consolidate all Project freight, provide sea passage to Abidjan, arrange port and customs clearance, and arrange road transport to site.

A road survey analysis has been undertaken between the port of Abidjan and the Doropo Project site (AGL and Antrak, 2024) to provide the basis for in-country logistics for the FS. This will be updated during the Front-End Engineering Design (FEED) phase of the Project.

24.7 Construction

24.7.1 General

The EPCM Contractor will manage the construction of the processing plant and selected infrastructure facilities in accordance with the Construction Management Plan. The EPCM Contractor will supply construction management and supervision, with the site works contractors supplying the labour and additional supervision for the scopes of work.

The construction management team will ensure that a safe working environment is available to all construction personnel. Safe work procedures and risk assessments will be developed prior to commencement of significant tasks to allow time to plan and implement measures to reduce the residual risk to an acceptable level. The safety objective being the elimination of risk by use of the hierarchy of controls defined in the Construction Risk Assessment Workshop (CRAW) and Safety Management Plan.

24.7.2 Construction Works

The Owner's team will undertake an early works package, including engaging a bulk earthworks contractor to establish access to the site for construction equipment and materials. Access and earthworks pads will also be provided to the initial temporary facilities, which will include the EPCM's construction offices and project laydown areas.

Bulk earthworks for the process plant will then be undertaken by the earthworks contractor. Concrete works will commence in areas identified to provide earliest access for installing site constructed tanks and major structural steel and equipment foundations.

Structural, mechanical, platework and piping (SMPP) and electrical and instrumentation (E&I) installation packages will be structured not to cause excessive interface issues between the separate contractors. Specialist subcontractors will be engaged to perform portions of the work requiring specialist equipment and experience, including large tank installation works and mill installations.

24.7.3 Health and Safety

Construction activities shall be performed in accordance with the Construction Safety Management Plan (CSMP). The CSMP will set out the framework for health and safety and the minimum standard to be deployed in order to achieve the objectives for construction health and safety performance whilst ensuring compliance with the relevant applicable laws and regulations in Cote d'Ivoire.

The primary health and safety objective for the construction work is that everyone involved actively works towards achieving zero harm to people and assets. To achieve this objective, key principles will be reinforced by management during the Project's development.

The EPCM Contractor will appoint a Safety Manager and mobilise safety advisors to the site for the duration of the construction and commissioning activities being performed on site. All work undertaken for this Project will be carried out with the utmost regard and attention to safety issues.

24.7.4 Environmental Management

The EPCM Contractor will ensure all construction activities comply with the Environmental Management Plan (EMP) developed for the Project.

The EPCM Contractor will implement a system to facilitate the adoption and maintenance of sound environmental management and operating practices, ensuring the continual improvement of environmental performance in accordance with Project requirements and the environmental procedures of the Owner.

24.7.5 Employee and Industrial Relations

The Human Resources/Industrial Relations Management Plan will set out the processes and controls to be implemented during the construction of the Project.

Project personnel, including the construction team, will be recruited, appointed, and managed in accordance with the Owner's and the EPCM Contractor's policies. Local contractors and suppliers will be encouraged to tender for all project works and contracts for which they are qualified to undertake, and they will be assessed based on their ability to meet the required conditions. Direct negotiations are planned with smaller local business groups with specific contract packages to encourage, where possible, local sourcing of some project requirements.

24.7.6 Construction Quality Control

The EPCM Contractor will perform all construction verification in the presence of the Owner's representative or their delegate. The construction verification will generate a punch list detailing any items that require rectification. Each item on the punch list will be jointly categorised according to the impact it may have on safety, commissioning activities, and the operation of the facilities.

24.8 Completions and Commissioning Process

The processing plant will be commissioned by a combined team consisting of the EPCM Contractor's personnel, the Owner's operations personnel, and vendor representatives.

The objectives of the commissioning processes are primarily as follows:

- To program the completion of construction, construction verification, pre-commissioning, and start-up of the processing plant to enable design performance criteria of the plant to be achieved safely, in the minimum possible time, and in the most cost-effective manner;
- To validate that the Project facilities have been designed, manufactured, and installed to the required specifications and to ensure that they are ready for commercial use;
- To ensure that the plant is satisfactorily completed so that it can be operated safely, with commissioning modifications fully documented, as-built drawings updated, and the new project changes communicated to site personnel;
- To test and adjust, as necessary, the operation of the various components of the process;
- To enable sufficient specialist plant knowledge to be passed on to the Owner's operations personnel in a hands-on manner, allowing them to continue plant operation on a full-time commercial basis.

To facilitate methodical commissioning of the works, the EPCM Contractor will implement a systems-based approach. This will include the grouping the works into commissioning systems and commissioning subsystems.

Commissioning systems will consist of a group of integrated commissioning subsystems from a common area of the works that can be operated simultaneously for the purposes of ore commissioning. Commissioning subsystems are made up of a group of integrated items of plant and equipment that can be operated as a standalone unit.

The EPCM Contractor will establish a commissioning progress tracking system to record commissioning test pack progress and completion from construction verification to ore commissioning. The commissioning tracking system will be populated with relevant project information and available for use prior to the start of construction verification activities commencing on site.

The completed (signed off) test packs and commissioning tracking system up to and including wet commissioning will form part of the Manufacturer's Data Report (MDR).

24.9 Operations Readiness

In conjunction with the Owner's procurement and operations managers, the Owner's Project Manager will ensure that the operations readiness activities are undertaken within the construction period. This includes the implementation of the asset management system, execution of supply contracts, stocking of the warehouse with adequate inventory, recruitment and training of the workforce, and securing the necessary government approvals to operate.

The EPCM Contractor will provide the information and deliverables required for the Owner's team to undertake Operational Readiness, including but not limited to provision of:

- Engineering documentation and deliverables;
- Quality and Construction verification documentation, commissioning records;
- Supplier/vendor data, including spares lists and operating manuals;
- Certified plant details for incorporation into the Owner's classified plant register.

TABLE OF CONTENTS

25. INTERPRETATION AND CONCLUSIONS 1
25.1 Metallurgy 1
25.2 Mineral Resources 1
25.3 Mining Methods 1
25.3.1 Risks 2
25.3.2 Opportunities 3
25.4 Recovery Methods 3
25.4.1 Processing Risks 3
25.4.2 Opportunities 4
25.5 Project Infrastructure Risks and Opportunities 4
25.5.1 Beach Slope 4
25.5.2 Achieved Densities 5
25.5.3 Life of Mine Planning 5
25.5.4 Engineered Soil Cover 6
25.5.5 Operating TSF Embankment Downstream Profile 6
25.5.6 Availability of Mine Waste 6
25.5.7 Wet Season Construction 6
25.5.8 Sediment Generated By Artisanal Mining Works 7
25.5.9 Runoff Coefficients 7
25.5.10 Survey 7
25.5.11 Construction Materials 7
25.5.12 Power Supply 8
25.6 Environmental, Permitting, Social and Community Impact 8
25.7 Project Implementation Schedule 9

25. INTERPRETATION AND CONCLUSIONS

25.1 Metallurgy

The metallurgical work carried out to date indicates that gold can be satisfactorily recovered from Doropo ores using conventional Carbon in Leach (CIL) cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

25.2 Mineral Resources

Cube has produced a Mineral Resource estimation update for the Doropo Gold Project in line with the scope of its engagement by Centamin. The Mineral Resource has been reported in accordance with CIM Definition Standards - For Mineral Resources and Mineral Reserves (CIM Council, 2014), with an effective date of 30 October 2023.

The Mineral Resource is reported within pit shells using a metal price assumption of US$2,000/oz Au, and is reported above 0.3 g/t Au in-pit. The qualified person believes this is a reasonable approach, considering the potential mine life and considerations for reporting Mineral Resources in accordance with the CIM Definition Standards (CIM Council, 2014).

The Mineral Resource is considered to have Reasonable Prospects for Eventual Economic Extraction (RPEEE) on the following basis:

- The Project is located in a mining jurisdiction with multiple operating gold mines, with no known impediments to land access or tenure status;
- The volume, orientation and grade of the Mineral Resource is amenable to mining extraction via traditional open pit methods;
- Current metallurgical recovery based on available definitive feasibility level metallurgical test work was used in a pit optimisation to generate the resource pit shells.

The Mineral Resource Qualified Person is of the opinion that the data used in the preparation of the Mineral Resource estimates were collected in a manner consistent with industry good practice and are therefore fit-for-purpose. The Mineral Resource estimate was undertaken using a range of appropriate statistical, geostatistical and 3D visual analysis tools and methods, using reliable and proven software. The Doropo mineralisation is still open, especially at depth; the potential therefore exists to further extend it.

25.3 Mining Methods

Based on the comprehensive mining study undertaken for the Doropo Feasibility Study, Orelogy's Qualified Person has determined that the Doropo Gold Project is both technically and economically viable.

25.3.1 Risks

Operational

- Mining selectivity and ore definition will be important to the success of the mining operation to ensure ore loss and dilution are minimised;
- The geotechnical wall slope criteria has been developed to a high level of detail based on the modelled weathering surfaces. These surfaces show considerable variability in depth and thickness. It is likely that during operations these surfaces will change and therefore the resulting wall slopes will vary, particularly through the weathered zones;
- Due to the multiple pit mining operation, careful selection of a reliable mining contractor will need to be made to ensure effective management of the multiple working faces and fleets. The Owners mining team will similarly need to be adequately staffed to manage the complexity of the mining operation;
- The multiple pit scheduling requires some ore to be mined in advance and stockpiled to ensure the ore is available for processing. If this ore is stockpiled at the pits, it may present a potential ore security risk. Therefore, the ore should be transported and stockpiled at the central stockpiling area adjacent to the processing plant. This will mitigate the security risk of the ore at the cost of incurring the ore transport costs in advance of the ore being required for feed.

Schedule

- Bench turnover rates in Yrs. 1 & 2 are aggressive and will need to be achieved to provide the scheduled stockpile balance;
- Scheduled ramp up in production requirements in Yr. -1 & Yr. 1 is aggressive (zero to 3 dig fleets in three months);
- Pit dewatering assumption (5 l/s for each pit) is most likely understated. The mining contractors indicated this amount was significantly below their experience and will impact the ability to achieve scheduled bench turnover rates in some areas;
- The interaction of both mining extraction activities and the ore haulage activities from the satellite pits will need to be carefully and systematically managed to minimise any negative impacts on regional communities;
- The large number of satellite pits that require ore to be transported to the processing facility introduces a potential risk to ore continuity.

Cost

- No specific cost and schedule allowances have been included for controlled vibration blasting on the project. This is primarily applicable to Han, northern Souwa and Nokpa due to their proximity to habitation;
- The potential requirement to increase levels of pit dewatering will have a cost implication.

25.3.2 Opportunities

- Presplit quantities are much higher than previous designs, due to the use of 75-degree walls in the pit designs. There is opportunity to reduce presplit requirements by using a flatter face angle (say 60 - 65 degrees) in some areas without affecting the pit design;
- Continuing with 4 digging fleets for a longer period after Year 6 will reduce the mining contractor fixed costs;
- Waste L&H mining rates assume creation of ex-pit dumps. However, there is opportunity to undertake backfilling in a number of areas, though this may impact gold production;
- No allowance has been made for free-dig of the completely weathered material. This may be possible which will reduce early mining costs in each mining area.

25.4 Recovery Methods

25.4.1 Processing Risks

The key processing risks and planned mitigation measures include:

- The coarse ore stockpile has been assumed to have a 60 degree draw down angle in design. This assumption has not been validated with test work and represents a risk to the live stockpile capacity available;
- The design currently excludes the installation of a fixed rock breaker to service the primary crusher. However, provision has been made in the structural design to facilitate the future installation of a fixed rock breaker, should this be required to reduce unplanned downtime;
- The Kilosegui oxide material, when thickened to 50% solids, had a test viscosity above the design point for agitation. The Kilosegui oxide material should be fed to the plant blended with fresh ores so that the high viscosity is mitigated and mixing, settling and leaching residence times can be maintained;
- The Souwa, Enioda, Kilosegui, Han and Chegue Main ores showed the presence of sulphide sulphur. The FS master composite mineralogy suggests this is a function of the occurrence of pyrite and/or pyrrhotite. However, the FS mineralogy did not distinguish between pyrite and pyrrhotite. Since pyrrhotite is a known reactive sulphide, and its presence may consume dissolved oxygen and cyanide within the leaching circuit to negatively impact leach recovery, further mineralogy work should be undertaken to differentiate between these minerals;
- The FS oxide master composites are shown to contain a high proportion of clay minerals; predominately smectite-type minerals with variable iron compositions. Smectites are swelling clays which may adversely impact slurry rheology. Blend management should successfully negate any adverse rheology impact;

- The dynamic thickening test work during the FS indicated that the flocculant additions required for the settling of oxide ores was >80 g/t, which may result in trace reagent carryover and carbon fouling. The flocculant dosing requires further optimisation and, should dosages remain >80 g/t, steam injection into the regeneration kiln should be considered to alleviate the potential for build-up and carbon fouling;
- The FS test work and metallurgical work to date have been completed with Perth tap water in lieu of site collected water. Although the harvested surface water is expected to be of good quality, the process impacts from the use of site collected water, if any, are unknown. Test work at FS optimised conditions with site water should be performed to confirm the results.

25.4.2 Opportunities

Processing opportunities for the Doropo gold project include:

- Potential operating cost reduction through further optimisations in the concentrations of SMBS, oxygen and Cooper sulphate (cyanide destruction reagents) to destroy WAD cyanide and reduce reagents levels;
- The assumptions used in the FS for the number of cyanide isotanks required are considered conservative. Confirmation of the routes and loop timing for the transport of cyanide isotanks between site and Orica facility in Tarkwa (Ghana) may support a reduction in the total number of tanks required. This could reduce the isotank rental costs per month and hence reduce processing operating costs;
- Since the metallurgically test work results for oxygen demand were not available for the FS design, the oxygen requirements for the leaching and cyanide destruction processes are based on benchmarked assumptions. The design assumptions are considered conservative and there may be opportunity to rationalise the size of the onsite PSA plants;
- Further metallurgical test work on flocculation and coagulation requirements for thickening, particularly in the first year of operation, should be undertaken to optimise the reagent costs and leach density;
- Further optimisation of the blending and blended grind size may lead to potentially higher throughput and/or higher gold recovery for softer ores. Blending may also mitigate the high viscosities anticipated when feeding the Kilosegui oxide material.

25.5 Project Infrastructure Risks and Opportunities

25.5.1 Beach Slope

The design is based on an average tailings beach slope of 1% (100H:1V). However, the beach slope is heavily dependent on the grind size and the ore blend. Thus, small changes in plant performance or design, ore type, or the ore blend have the potential to change the tailings beach slope.

There are a number of approaches which can be used in response to measured beach slopes that are consistently different to the beach slope used for design. One advantage of staging construction on an annual basis is the ability to modify the design each year based on measured data obtained from the TSF. In these cases, the timing and height of the subsequent embankment raises can be modified to bring the schedule back into line with the design, and the subsequent lifts will be on an annual basis essentially as per the design raised heights.

Steeper Beach Slope

If the measured beach slope is steeper than the design slope, the tailings rate of rise against the TSF embankment will be faster than expected, and the Stage 1 TSF will reach its tailings storage capacity earlier than the design. If this were to become an issue, the response would be to move Stage 2 construction of the TSF forward. Commencing the construction one or two months earlier would not have a significant impact as the construction would still be predominantly in the dry season. It should be noted that Stage 1 capacity is 18 months, allowing for a controllable adjustment of the construction sequence. In addition, the deposition line could be extended further southwest to provide additional tailings storage capacity without impacting the operation significantly.

It should be noted that for steeper beach slopes the potential tailings storage would be reduced, but the storm water storage capacity would be increased accordingly.

Flatter Beach Slope

If the measured tailings beach slope is flatter than the design slope, the capacity of the Stage 1 TSF to store tailings would be increased. The overall TSF stormwater storage capacity will not be affected, unless Stage 2 construction is deferred beyond the original construction schedule.

25.5.2 Achieved Densities

The staged TSF embankment crest elevations are based on the ore blend and throughput used for the water balance modelling. Changes in these characteristics and/or throughput will result in changes in the achieved densities in the TSF. Similar to the variations in tailings beach slope, this may result in an adjusted construction schedule for the first raise, either earlier or later than the design timing. It is recommended that monitoring of throughput, ore blend, rate of rise and achieved densities be undertaken so that suitable planning and staging of the future embankment construction can occur.

25.5.3 Life of Mine Planning

Any changes to the life of mine plan or throughput will impact upon the tailings management requirements for the site. Any significant increases in throughput may result in lower tailing densities being achieved within the TSF, thus increasing construction costs. Any decrease to the total tonnage may require reconsideration of the proposed closure plan, as the closure spillway may become prohibitively deep.

In addition to the impacts on the TSF design, any changes to the operating throughput and percent solids of the tailings may impact water demands.

25.5.4 Engineered Soil Cover

The current design for closure and decommissioning of the TSF includes an engineered fill cover constructed over the tailings beach as the most suitable long-term solution. The configuration of this cover has been assumed for this study. On-going tailings geochemistry characterisation testing during operation may indicate that it will be possible to grow plants on the final (drained) tailings surface subsequent to compaction. If this is the case, the costs for rehabilitation of the tailings surface could be significantly reduced.

25.5.5 Operating TSF Embankment Downstream Profile

If the TSF embankment Zone C material comprises coarse, clean rockfill of high strength sourced from the open pit, the stability of the TSF embankment downstream face using a steeper slope (2.5H:1V) could be considered subject to confirmation by a stability assessment once the rockfill properties are known. The final profile of 3.5H:1V (overall) is required for rehabilitation purposes.

25.5.6 Availability of Mine Waste

Design of the TSF is based on structural fill material being sourced from the open pit mining operations. Based on initial review of the mining schedule, this should be suitable for Stage 2 and the construction of future raises. If waste from pre-stripping is not readily available during the Stage 1 construction, additional borrows will be required in the proximity of the TSF. Although this is possible, the capital cost will increase. Utilising a civil earthworks fleet to win materials from the Open Pit footprints of Souwa and Nokpa pits are to be assessed in terms of economic viability considering haulage distances.

Likewise, suitable low permeability fill material may be stockpiled by the mining operation at locations in close proximity to the TSF embankment, for use by civil contractors in future stages. This may reduce earthworks rates during future raise construction.

25.5.7 Wet Season Construction

The ability to construct earthworks during wet seasons can be limited, so construction over the life of the project needs to be carefully planned and monitored so that approval, budgeting and logistics are in place to allow works to be completed promptly and prior to the onset of wet conditions.

25.5.8 Sediment Generated By Artisanal Mining Works

Significant artisanal mining works have been noted within the stream bed upstream of the Han diversion and WHD. Stream flows in these areas may collect a significant amount of sediment, which may impact the clarity available in the WHD reservoir (for abstraction to the WSD). This should be assessed by the environmental consultant to determine the requirement for additional source control upstream of the Han diversion and WHD reservoir to reduce this sediment load. This may comprise a series of rockfill check dams within the main stream bed.

25.5.9 Runoff Coefficients

The WHD is the primary supply of makeup water at Doropo, therefore the accurate estimation of the undisturbed runoff coefficient is critical to ensure that sufficient makeup water is available to avoid project shortfalls. The estimation of undisturbed runoff coefficients was limited to a desktop study and the actual values may vary. If runoff is lower than estimated then the WHD capacity may need to be increased. Conversely, if the runoff is higher than estimated additional erosion protection measures and/or regular maintenance may be required. It is recommended that the initial filling of the WHD is monitored so that the water balance model can be calibrated and site-specific runoff coefficients can be determined.

25.5.10 Survey

Inaccurate base survey is a common cause of variations between expected and actual quantities, particularly in reference to bulk fill earthworks volumes. Topographical contours at times can be generated with small amounts of survey pickup, and as such there is significant interpolation by computer programs.

Accurate basin pickup is required as this will have a significant impact on both bulk fill volumes and the storage capacity of the facility.

If there is less storage capacity than currently designed, an earlier start to Stage 2 construction may be required in order to continue to provide the required stormwater storage capacity. It is recommended that a stripped ground survey be completed during construction of the TSF, WSD and WHD to reconcile the required embankment heights prior to embankment construction. This may result in some cost reductions (or increases).

25.5.11 Construction Materials

The geotechnical investigation, as described in Part 18, has identified potential sources of construction materials required for the project infrastructure within the project site area, including general fill, rock armour, aggregates and drainage medium. However laboratory testing will be required to confirm the suitability of these materials for use.

25.5.12 Power Supply

ECG believe that the most appropriate grid connection is to upgrade the Bouna Substation by extending the existing 90 kV bus, adding a 90 kV transmission line feeder, construction of a ~65 km of 90 kV single circuit lattice tower transmission line, and constructing a substation at Doropo site. The Doropo Plant Substation would be owned and operated by Cote d'Ivoire Energies (CIE) and Doropo mine would take a 90 kV tariff metered feeder, installing a 90/11 kV transformer in their substation and taking an 11 kV feeder to the Plant Main 11 kV Main Switchboard.

The 90 kV supply at Bouna should be of good quality and reliable since the recently constructed 90 kV transmission line from Bondoukou to Bogofa and Bouna is lightly loaded. The supply at Bondoukou is from the 225 kV network and is part of a ring-main system which is very reliable and based on ECG's risk assessment, it is not expected that a full back-up power station is required as part of the grid connection works.

Integration of a solar Photo Voltaic (PV) supply is an option to supplement the grid supply and provide up to 30% of the plant energy requirements. A solar Independent Power Producer commercial strategy could reduce the high voltage (HV) power supply dependency from the grid by up to 30% and reduce operating costs by approximately 7.5%, providing net savings of approximately US$4.4M over the life of the project. If Centamin elect to develop and own the solar PV plant outright, the estimated capital costs would be approximately US$19M.

Approximately 39% of the generation on the CIE network is made up of Hydro, with the rest made up of natural gas and dual fuel generators.

25.6 Environmental, Permitting, Social and Community Impact

The key benefits of the proposed Project for Côte d'Ivoire will be financial, economic and social benefits at the national, regional and local levels. On a local level, the Project can provide a significant injection of income and employment opportunities to the region, providing opportunities for training and skill development. It has the capacity to catalyse economic and social development in the area, improve access to infrastructure, goods and services, and leave a lasting positive legacy. Royalties and taxes from the Project would also provide needed income for the National Government of Cote d'Ivoire.

The generation of royalties, taxes and dividends from gold mining to the Côte d'Ivoire National Government contributes significantly to the national goals of the country. The three main taxes and fees imposed on companies operating in the mining sector are royalties, salary withholding tax and industrial and commercial profits tax. The Côte d'Ivoire Government also has the right of a 10% free carried ownership interest in the development.

Based on the current gold price, which is in excess of US$2,000 per oz, government royalties would be 6% of sales revenue or approximately US$20 million per year.

Overall the Doropo Project would be expected to contribute over US$2 billion (based on the current gold price) to the Ivorian economy in the form of taxes, royalties, dividends, salaries, payments to local suppliers and infrastructure development. During operation Ampella can be expected to be one of the largest corporate taxpayers in Côte d'Ivoire.

Mines operating in local communities in Cote d'Ivoire are required to pay 0.5% of their turnover to a local mining development fund, which aims to finance community development projects in mining regions. This Fund is expected to generate approximately US$17 million over a 10-year life of the Project with approximately US$2 million generated each year for the first five years of operation.

Through effective management, the injection of income and economic opportunity into the local community as a result of the Project will be a major benefit. Through employment, training, procurement and livelihood improvement programmes, the Project can be a catalyst for sustained economic growth in the Doropo Region. The Company's existing social investment programme will also continue to complement Project-related activities and contribute to achieving local development objectives.

25.7 Project Implementation Schedule

The implementation schedule developed for the Project has been based on the project specific scope using logic driven activity links identifying the critical and near critical tasks. The schedule has been reviewed and benchmarked against similar projects, and is considered realistic and appropriate for a FS.

In order to achieve the implementation schedule, It is critical that Centamin performs and executes their early works programme to prepare the site for the construction phase.

TABLE OF CONTENTS

26. RECOMMENDATIONS 1
26.1 Mineral Resource Recommendations 1
26.2 Mining Methods Recommendations 1
26.3 Metallurgical Test Work Recommendations 1
26.4 Environmental and Social Recommendations 2
26.5 Project Infrastructure Recommendations 3
26.6 Project Implementation Recommendations 3

26. RECOMMENDATIONS

26.1 Mineral Resource Recommendations

Cube recommends the following with respect to the Doropo Gold Project:

- Centamin should give consideration to constructing a geo-metallurgical model to support future studies;
- Consideration should be given to pre-production work, such as such as the conversion of the first two years of Mineral Resources in the production profile into Measured Resources to de-risk the production profile. This would include drilling the starter pits for Souwa, Nokpa and Kilosegui. Consideration could then be given to how to best set up a fit-for-purpose mine geology system.

26.2 Mining Methods Recommendations

Orelogy recommends the following with respect to the Doropo Gold Project.

- Mining selectivity and ore definition will need be mitigated through the use of industry standard ore control techniques such as:
  - In-advance and detailed RC grade control drilling and modelling;
  - Digital ore mark-out combined with ore spotting where required;
  - The use of smaller excavators and blast balls are recommended for highly selective zones;
  - A robust mining contractor tender process is required in the next phase to ensure the eventual contractor selection has the capacity, capability and experience to undertake the work to the required complexity and quality.

26.3 Metallurgical Test Work Recommendations

IMO notes that the detailed metallurgical test work programme was continuing for the Doropo Gold Project under the direction of Centamin to support the design as part of the continuous improvement programme. Ongoing test work is listed below:

- Bulk Carbon-in-Leach (CIL) cyanidation testing of Souwa, Nokpa, Kilosegui, Enioda and Chegue Main Master composites under optimised conditions;
- Dynamic thickener and detoxification test work;
- Pre-Leach & Carbon-in-Leach test work;
- Carbon-in-Leach (CIL) cyanidation testing of all Variability composites under optimised conditions;
- Equilibrium carbon loading tests on all Master composites;

- Rheology test work has recently been reported and currently under review. Initial results indicate that 55% and 60% solids are a viable option;
- Oxygen demand tests; and
- Diagnostic leach tests on CIL leach tails.

GRES recommends the completed FS metallurgical test work be reviewed and used to supplement existing data in the next design phase.

Additionally, GRES recommends the testing discussed below be undertaken, to supplement the plant design data set:

- Collection and analysis of site water;
- Leach cyanidation on the FS master composites, at optimised CIL conditions using site water;
- Settling tests on leached CIL residues focused on flocculant screening, flocculant dosing and if necessary, coagulant dose optimisation;
- Further mineralogy testing to distinguish between pyrite and pyrrhotite;
- Diagnostic leaching on CIL leach residues, and
- Detoxification testwork to further define reagent consumption.

Grade Recovery Curves

The current grade-recovery curves were prepared from limited variability results from the FS leach test campaign. The regression analysis confidence can be improved by addition of the full FS master and variability composite leach results.

Reduce Cyanide Consumption and Improve Leach Kinetics

The results from the shear reactor tests conducted in the FS programme were inconclusive. Further tests at optimum cyanide concentrations may identify reductions in cyanide consumption. The tests conducted at optimised conditions may also report higher overall gold recovery. Lower cyanide requirements will also translate to lower cyanide detoxification reagent costs.

26.4 Environmental and Social Recommendations

The effective implementation and regular updating of the Environmental and Social Management Plan (ESMP) and other management plans in response to changing needs will ensure that environmental and social impacts attributable to the Project are minimised and potential environmental and social benefits are maximised.

It is recommended that Centamin maintain a robust licence to operate during all phases of the mine life cycle, based on trust and mutual respect with all stakeholders. The strength of these relationships is to be underpinned by:

- Engagement - proactively engage stakeholders based on inclusion, transparency and integrity;
- Risk and impact management - integrate stakeholder considerations into managing risks to develop long term, positive cumulative impacts;
- Mutual value creation - collaborate to catalyse socio-economic development so communities can prosper during construction, operations and after mining activities cease.

Strong lines of communication are to be maintained with host communities to understand their needs and identify opportunities for long-term mutually beneficial outcomes.

26.5 Project Infrastructure Recommendations

It is recommended that a stripped ground survey be completed during the construction of the TSF, WSD and WHD to reconcile the required embankment heights before embankment construction. This may result in some cost reductions or increases.

It is also recommended that the initial filling of the WHD be monitored to calibrate the water balance model and determine site-specific run-off coefficients. The requirement for borehole water supplementation of raw water supply should be assessed in further detail.

Adequate fencing should be installed in areas where communities and their livestock are vulnerable to earthwork features such as drainage channels, water sources, traffic and mining activities.

Supplementing the HV supply with PV solar energy should be considered as part of the Project to minimise the overall operational carbon footprint.

26.6 Project Implementation Recommendations

The ability to construct earthworks during wet seasons can be limited, therefore construction throughout the life of the Project must be carefully planned and monitored to ensure that approval, budgeting and logistics are in place, allowing works to be completed promptly before the onset of wet conditions.

TABLE OF CONTENTS

27. REFERENCES.. 1

27. REFERENCES

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22. Piteau Associates Ltd, “*Definitive Feasibility Study of Dewatering Requirements for the Doropo Project Open Pits*”, Doropo Gold Project, Côte d’Ivoire, 24 July 2024.

23. ECG Engineering Ltd, “*Grid Power Supply*” Definitive Feasibility Report, Doropo Gold Project, Côte d’Ivoire, 8 March 2024.

24. Independent Metallurgical Operations Pty Ltd, “*Doropo DFS Metallurgical Test Work Report*”, Report # 6599, August 2024.

Knight Piésold Pty Limited
A.B.N. 67 001 040 419

Level 1, 184 Adelaide Terrace
EAST PERTH WA 6004, Australia
(PO Box 6837, EAST PERTH WA 6892)

Telephone +61 (0)8 9223 6300
Facsimile +61 (0)8 9223 6399
Email perth@knightpiesold.com

CERTIFICATE OF QUALIFIED PERSON

I, David John Toomey Morgan, of Knight Piésold, do hereby certify that:

a) I am a Director with Knight Piésold with a business address at Level 1, 184 Adelaide Terrace, East Perth, WA 6004.

b) This certificate applies to the technical report titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of 18 July 2024 (the "Technical Report") prepared for Centamin plc (the "Issuer").

c) I am a graduate of University of Manchester, (BSc, Civil Engineering, 1980), and University of Southampton (MSc, Irrigation Engineering, 1981). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and a chartered Professional Engineer and member of the Institution of Engineers Australia (Australia 974219). My relevant experience includes Project Director – Geita Gold Mine, Project Director – Ahafo Gold Project, Project Director – Akyem Gold Project.

d) By reason of my education, affiliation with a professional association and past relevant work experience, I am a "qualified person" for the purposes of National Instrument 43-101-Standards of Disclosure for Mineral Projects ("NI 43-101").

e) I personally inspected the Doropo Project on 29th Jan to 31st Jan 2022 and Professional Engineers under my control have conducted site investigations on my behalf.

f) I am responsible for Sections 1.16.2, 18.2, 18.4, 18.5, 18.6, 25.5, and contributed to Sections 1.13 and 26.5 of the Technical-101 Report.

g) I am independent of Centamin PLC as described in Section 1.5 of NI 43-101.

h) I was a co-author and involved in the preparation of the technical report titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of 27 June 2023.

i) I have read NI 43-101 and the Technical Report, and the parts of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101 and Form 43-101F1.

j) At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 3, September, 2024 at Perth, Western Australia, Australia.

David J T Morgan

ABN 33 084 725 557
88 Thomas Street West Perth
Western Australia 6005
Ph +61 8 9254 6900
Fax +61 8 9322 1808
imo@indmetops.com.au
www.indmetops.com.au

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report prepared for Centamin plc (the "Issuer")'s Doropo Gold Project in Northeastern Côte d'Ivoire entitled: "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of 18 July 2024 (the "Technical Report").

I, Grant Harding, FAusIMM, do hereby certify that:

1. I am a Consulting Metallurgist with Independent Metallurgical Operations Pty Ltd and am employed at 88 Thomas St, West Perth Western Australia 6005.
2. I graduated with a B. App. Science (Mineral Science) from Murdoch University in 1994 and have worked continuously in the Mining Industry as a Metallurgist for 30 years.
3. My operations experience includes senior roles as General Manager, Processing Manager and Senior Metallurgist within process plants employing gravity concentration, sulphide conventional flotation, free-milling and refractory sulphide treatment as well as conventional oxide CIL/CIP leaching, and Merrill-Crowe processing of high-grade gold and silver ores. In addition to direct operational experience, I have undertaken a range of consulting roles across a wide range of gold oxide and sulphide projects, including projects located in Africa, SE Asia, South America, Australia and the Pacific Rim.
4. I have been a Fellow of the Australasian Institute Mining and Metallurgy since 2008, #106854.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6. I was responsible for preparation of Section 1.8, 1.19.2, 1.20.1, 13.0, 25.1 and 26.3 of the Technical Report relating to the Doropo Gold deposits, Northeastern Côte d'Ivoire.
7. I have not visited the Doropo Gold Project, Côte d'Ivoire site.
8. I am independent of the issuer as independence is described in Section 1.5 of NI 43-101. I have read NI 43-101,Form 43-101F1 and the Technical Report,and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated September 3, 2024

Grant Harding, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, Ph.D., SME RM
Director of Metallurgy
Forte Dynamics, Inc.
12600 W Colfax Ave, Ste A-540 | Lakewood, CO 80215
Email: dmalhotra@fortedynamics.com

This certificate applies to the report entitled: "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with effective date of 18 July 2024 (the "Technical Report").

I, **Deepak Malhotra, Ph.D.**, do hereby certify that:

1) I am the Director of Metallurgy for Forte Dynamics, Inc., with a business address of 12600 W Colfax Ave, Ste A-540, Lakewood, Colorado 80215 USA.
2) I graduated with a degree in Metallurgical Engineering, Master of Science in 1973 from the Colorado School of Mines in Golden, Colorado. In addition, I graduated with a degree in Mineral Economics, Ph.D. in 1978 from the Colorado School of Mines in Golden, Colorado. My relevant experience includes working as a metallurgist and mineral economist for a total of 50+ years since my graduation with specific expertise in mineral processing, metallurgical testing, and recovery methods. I am a member of the Society of Mining Engineers.
3) I have not visited and inspected the property which is the subject of the Technical Report.
4) I am responsible for Sections 1.12, 1.16.1, 1.16.3, 1.20.1, 17, 18.1, 18.7.2, 18.8, 18.10, 18.11, 18.12, 18.15, 21.1, 21.2.1, 21.2.3, 21.3, 21.4, 21.6, 21.8, 24.4, 24.5, 24.6, 24.8, 25.4, 25.7 and 26.3 of the Technical Report.
5) I am independent of the issuer.
6) I have had no prior involvement with the property that is the subject of the Technical Report.
7) As of the publication date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8) I consent to the filing of the Technical Report.

Dated this 3rd day of September 2024.

Signed, Sealed "Deepak Malhotra, Ph.D., SME RM
Deepak Malhotra, Ph.D., SME-RM

Deepak Malhotra, Ph.D., SME RM
Print Name of Qualified Person

3 September 2024 Ref: 0995-COR-0002

Centamin plc.
Company registration number: 109180
2 Mulcaster Street, St Helier,
JERSEY JE2 3NJ.

RE: The report prepared for Centamin plc (Centamin) titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of 18 July 2024 (the Technical Report)

CERTIFICATE OF QUALIFIED PERSON – ROSS MALCOLM CHEYNE

I, Ross Cheyne, Bachelor of Engineering Hons (Mining) do hereby certify that:

1. I am a Principal Mining Consultant with:

    Orelogy Consulting Pty Ltd,
    Level 2
    101 St Georges Terrace,
    Perth,
    WA 6000,
    AUSTRALIA.

2. I graduated from the University of Auckland, New Zealand, with a Bachelor of Engineering Degree (Hons) in Mining.

3. I am a Fellow of the Australasian Institute of Mining and Metallurgy (Member No. 109345).

4. I have worked as a Mining Engineer for a total of 34 years since 1990. I have worked on gold mining operations for a period of 3 ½ years. I have worked in West Africa for 1 year. I have been a consultant for 24 years and have worked on multiple PEA's, PFS's and FS's for gold projects in West and Sub-Saharan Africa. I have acted as QP signing off on Mineral Reserves and Ore Reserves under National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and JORC reporting guidelines.

5. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-

101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. For this Technical Report, I am wholly responsible for the preparation of sections 1.10, 1.11, 1.19.4, 1.20.3, 15, 16.1, 16.2, 16.4, 16.5, 16.6, 16.7, 16.8, 16.9, 16.10, 18.9, 21.2.2, 21.3.2, 21.5, 25.3 and 26.2. I am partially responsible for report section 1.11

7. I have not visited the Site.

8. I was involved in the preparation of the technical report titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of 27 June 2023.

9. I am independent of Centamin as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101, Form 43-101F1 and the items of the Technical Report for which I am responsible and such items have been prepared in compliance with that instrument and form.

11. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Items of the Technical Report that I am responsible for contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 3 September 2024

Ross Cheyne (FAusIMM)
Principal Consultant
Orelogy Consulting Pty Ltd

Certificate of Qualified Person – Craig Barker

I, Craig Lawrence Barker, BSc., Post Grad. Dip. (Geology), FAIG do hereby certify that:

1. I am the Group Mineral Resource Manager for Centamin plc with a business address at 2 Mulcaster Street, ST HELIER JERSEY JE2 3NJ, JERSEY.

2. This certificate applies to the NI43-101 Technical Report entitled 'National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire' with an effective date of July 18, 2024 (the "Technical Report").

3. I graduated with a Bachelor of Science in Geology from the University of Adelaide (1994) and a Post Graduate Diploma in Geology from the University of Western Australia (1998). I am a Fellow of the Australian Institute of Geoscientists. My membership number is 3141.

4. I have worked as a Geologist for more than 25 years since graduating from university with mining and exploration companies in Australia, Africa, Laos, Bulgaria, Armenia, Cambodia and Mongolia in exploration, project development, production and management roles.

5. I have read the definition of 'qualified person' as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Project ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.

6. I personally inspected the Doropo Project on August 29 and 30, 2021.

7. I am responsible for Items 1.1 to 1.6, 1.14, 1.15, 1.16, 1.17, 1.18, 1.19.1, 2 to 9, 18.3, 18.13, 18.14, 18.7.1, 19, 20, 21.2.4, 21.3.22, 21.7, 22, 23, 24.1, 24.2, 24.3, 24.7, 24.9, 25.5.12, 25.6, 25.7, 26.4, 26.5, 26.6 and 27 of the Technical Report.

8. I am not independent of the issuer, applying the test set out in Section 1.5 of NI 43-101 since I am a full-time employee at Centamin plc.

9. I was a co-author and involved in the preparation of the technical report titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of June 27, 2023.

10. I have read NI 43-101, Form 43-101F1 and the items of the Technical Report for which I am responsible and such items have been prepared in compliance with that instrument and form.

11. As of the "effective date" stated in the Technical Report, to the best of my knowledge, information and belief, the Items of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: September 3, 2024

Craig Barker
FAIG,
Group Mineral Resource Manager – Centamin plc

Certificate of Qualified Person – Michael Millad

As a Qualified Person and a co-author of the technical report titled: “National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire” for Centamin plc (the “Issuer”), with Effective Date of July 18, 2024 (the “Technical Report”), I, Michael Millad do hereby certify that:

1) I am a Director and Principal Geologist/Geostatistician with Cube Consulting Pty Ltd at its office located at Level 4, 1111 Hay Street, West Perth, Western Australia, Australia.
2) I am a professional geologist having graduated with a BSc (Hons) in Geology from Rhodes University (1993), MSc (Economic Geology) from Rhodes University (2004) and CFSG from École des Mines de Paris (2007).
3) I am a Member of the Australian Institute of Geoscientists (member #5799).
4) I have practised my profession as a geologist for the past 29 years in the mineral resources sector and engaged in the mining geology and resource estimation of mineral projects both within Australia and internationally.
5) I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6) I have authored and take responsibility for Items 1.7, 1.9, 1.19.3, 1.20.2, 10 to 12, 14, 25.2 and 26.1 of the Technical Report.
7) I have conducted a site inspection between August 29 and 30, 2021.
8) I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9) I was a co-author and involved in the preparation of the technical report titled “National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d’Ivoire” with an effective date of June 27, 2023.
10) I have read NI 43-101, Form 43-101F1 and the Technical Report, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument and Form.
11) As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated September 3, 2024 at West Perth, Western Australia, Australia

Mike Millad, BSc (Hons), MSc, CFSG, MAIG

Director and Principal Geologist/Geostatistician - Cube Consulting Pty Ltd

1394-9257-6526.2

Certificate of Qualified Person – Flavie Isatelle

As a Qualified Person and a co-author of the technical report titled: “National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire” for Centamin plc (the “Issuer”), with Effective Date of July 18, 2024 (the “Technical Report”), I, Flavie Isatelle, do hereby certify that:

1) I am a Principal Geologist/Geostatistician with Cube Consulting Pty Ltd at its office located at Level 4, 1111 Hay Street, West Perth, Western Australia, Australia.
2) I am a professional geologist having graduated with a MSc in Geology from Ecole Nationale Supérieure de Géologie (National School of Geology), Nancy, France (2008) and MSc (Geostatistics) from School of Mines in Fontainebleau, France (2018).
3) I am a Member of the Australian Institute of Geoscientists (member #8539).
4) I have practised my profession as a geologist for the past 12 years in the mineral resources sector and engaged in the mining geology and resource estimation of mineral projects both within Australia and internationally.
5) I have read the definition of “Qualified Person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6) I have authored and take responsibility for Items 1.7, 1.9, 1.19.3, 1.20.2, 10 to 12, 14, 25.2 and 26.1 of the Technical Report.
7) I have not visited the Doropo Project.
8) I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9) I was a co-author and involved in the preparation of the technical report titled “National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d’Ivoire” with an effective date of 18 July 2024.
10) I have read NI 43-101, Form 43-101F1 and the Technical Report, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument and Form.
11) As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated September 3, 2024 at West Perth, Western Australia, Australia

Flavie Isatelle, MSc (Geology), MSc (Geostatistics), MAIG

Principal Geologist/Geostatistician

Cube Consulting Pty Ltd

srk

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www.srk.com

Subject Certificate of Qualified Person

This certificate applies to the technical report prepared for Centamin plc (the "Issuer") for the Doropo Project in Côte d'Ivoire titled "National Instrument 43-101 Technical Report for the Doropo Gold Project, Northeastern Cote d'Ivoire" with an effective date of July 18, 2024 (the "Technical Report"). I, Samson Tims, do hereby certify that:

1. I am a Senior Consultant with SRK Consulting (Canada) Inc. My contributions were written while employed and working at SRK Consulting (UK) Ltd., headquartered at 5th Floor, Churchill House, 17 Churchill Way, Cardiff CF10 2HH, United Kingdom.
2. I am a professional geologist having graduated with a BSci in Geology from the University of Alberta (2012).
3. I am a member of Engineers and Geoscientists British Columbia (EGBC), member number 50186 and The Association for Professional Engineers and Geoscientists of Alberta (APEGA), member number 91536.
4. I have practiced my profession as a geologist in both consultant and operations roles for 12 years in the mineral resources sector engaged in mining rock mechanics within Canada and internationally.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I have authored and take responsibility for Chapter 1.11.2 and 16.3 of the Technical Report.
7. I completed a site visit of 2 days on Friday October 13 and Saturday October 14, 2023, and visited the core facilities and accessible proposed pit locations. Han and Enioda areas were not visited due to security concerns.
8. I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9. My only previous involvement with the project was with geotechnical characterization of the open pits for the Pre-Feasibility Study completed in 2023.
10. I have read National Instrument 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated September 3, 2024.

Samson Tims

PROFESSIONAL
PROVINCE OF
S. J. TIMS
50186
BRITISH COLUMBIA
GEOSCIENTIST