Preliminary Economic Assessment
NI 43-101 Technical Report
Oko West Gold Project

Prepared for:

MINING VENTURES

5025, Lapinière Blvd., Suite 1050
Brossard, Québec
Canada J4Z 0N5

Christian Beaulieu, P.Geo., Minéralis Services-Conseils Inc.
Neil Lincoln, P.Eng., Lincoln Metallurgical Inc.
Alexandre Burelle, P.Eng., Evomine Consulting Inc.
Derek Chubb, P.Eng., Environmental Resources Management
Paul Murphy, P.Eng., Consultant for G Mining Services Inc.

Prepared by:

G MINING SERVICES INC.
5025, Lapinière Blvd., Suite 1010
Brossard, Québec
Canada J4Z 0N5

Effective Date: September 4th, 2024
Issue Date: October 11th, 2024

IMPORTANT NOTE

General Conditions and limitations

Use of the report and its contents

This report has been prepared for the exclusive use of the Client or its agents. The factual information, descriptions, interpretations, comments, recommendations and electronic files contained herein are specific to the projects described in this report and do not apply to any other project or site. Under no circumstances may this information be used for any other purposes than those specified in the scope of work unless explicitly stipulated in the text of this report or formally interpreted when taken individually or out-of-context. As well, the final version of this report and its content supersedes any other text, opinion or preliminary version produced by G Mining Services Inc.

Preliminary Economic Assessment NI 43-101 Technical Report – Oko West Gold Project

Revision #

Cuyuni-Mazaruni, Guyana

G Mining Ventures

5025, Lapinière Blvd.
Suite 1050, Brossard, Québec
Canada J4Z 0N5
Tel: 450-923-9176
E-mail: info@gmin.gold
Web Address: https://gmin.gold/en-US/

G Mining Services Inc.

5025, Lapinière Blvd.
Suite 1010, Brossard, Québec
Canada J4Z 0N5
Tel: (450) 465-1950 • Fax: (450) 465-6344
E-mail: m.gignac@gmining.com
Web Address: www.gmining.com

October 11th, 2024

Qualified Persons

Prepared by:

(signed and sealed) “Christian Beaulieu” Date: October 11, 2024

Christian Beaulieu, P.Geo.,
Vice President
Minéralis Services-Conseils Inc.

(signed and sealed) “Neil Lincoln” Date: October 11, 2024

Neil Lincoln, P.Eng.
President and Consulting Metallurgist
Lincoln Metallurgical Inc.

(signed and sealed) “Alexandre Burelle” Date: October 11, 2024

Alexandre Burelle, P.Eng.
Mine Planning & Financial Analysis Consult.
Evomine Consulting Inc.

(signed and sealed) “Derek Chubb” Date: October 11, 2024

Derek Chubb, P.Eng.
Senior Partner
Environmental Resources Management

(signed and sealed) “Paul Murphy” Date: October 11, 2024

Paul Murphy, P.Eng.
Project Manager
Independent Consultant

Table of Contents

1 SUMMARY 1-1
1.1 Introduction 1-1
1.2 Terms of Reference 1-2
1.3 Reliance on Other Experts 1-2
1.4 Property Description and Location 1-3
1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography 1-3
1.6 History 1-4
1.7 Geological Setting and Mineralization 1-4
1.8 Deposit Types 1-6
1.9 Exploration 1-6
1.10 Drilling 1-7
1.11 Sample Preparation, Analyses and Security 1-7
1.12 Data Verification 1-8
1.13 Metallurgical Testing and Mineral Processing 1-9
1.14 Mineral Resources Estimate 1-11
1.15 Mineral Reserve Estimate 1-13
1.16 Mining Methods 1-13
1.17 Recovery Methods 1-14
1.18 Project Infrastructure 1-15
1.19 Market Study and Contracts 1-18
1.20 Environmental Studies, Permitting and Social or Community Impact 1-18
1.21 Capital and Operating Costs 1-19
1.22 Economic Analysis 1-22
1.23 Adjacent Properties & Other Relevant Data and Information 1-25
1.24 Other Relevant Data and Information 1-26
1.25 Interpretation and Conclusions 1-26
1.26 Recommendations 1-26
2 INTRODUCTION 2-1
2.1 Scope of Work 2-2
2.2 Sources of Information and Data 2-2
2.3 Site Visit 2-3

2.4 Effective Date 2-4
2.5 Sources of Information 2-4
2.5.1 Previous Technical Reports 2-4
2.5.2 Agreements, Mineral Tenure, Surface Rights and Royalties 2-4
2.6 Units of Measure, Abbreviations and Nomenclature 2-4
3 RELIANCE ON OTHER EXPERTS 3-1
4 PROPERTY DESCRIPTION AND LOCATION 4-1
4.1 Location 4-1
4.2 Property Description and Title 4-2
4.3 Legal Surveys 4-2
4.4 Oko West Mineral Tenure and Requirements 4-5
4.5 Oko West Project Ownership and Agreements 4-6
4.6 Surface Rights 4-7
4.7 Royalties and Other Encumbrances 4-7
4.8 Environmental Liabilities 4-7
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 5-1
5.1 Accessibility and Roads 5-1
5.2 Climate 5-3
5.3 Local Resources 5-4
5.4 Infrastructure 5-4
5.4.1 Services Buildings and Ancillary Facilities 5-4
5.4.2 Power Supply and Distribution 5-6
5.4.3 55Water 5-7
5.5 Physiography 5-7
5.5.1 Vegetation 5-7
5.5.2 Topography 5-7
6 HISTORY 6-1
6.1 Prior and Current Ownership 6-1
6.2 Exploration History 6-1
6.2.1 Historical Exploration 6-1
6.2.2 Reunion Gold Exploration 6-2
6.3 Historical Drilling 6-2
7 GEOLOGICAL HISTORY AND MINERALIZATION 7-1
7.1 Regional Geology 7-1
7.2 Property Geology 7-3

7.2.1 Regolith 7-4
7.2.2 Granitoids 7-8
7.2.3 Volcano-Sedimentary Sequence 7-10
7.2.3.1 Volcanoclastics 7-14
7.2.3.2 Siliciclastic Sediments 7-15
7.2.3.3 Carbonaceous Sediments 7-16
7.3 Structural Geology and Metamorphism 7-17
7.4 Tectonic Events and Gold Mineralization 7-20
7.5 Gold Mineralization 7-22
8 DEPOSIT TYPES 8-1
9 EXPLORATION 9-1
9.1 Geophysics 9-1
9.1.1 Airborne Geophysics 9-1
9.1.2 Ground Magnetics 9-2
9.1.3 Induced Polarization and PDP 9-3
9.2 Geology Reconnaissance and Stream-Sediment Geochemistry 9-4
9.3 Soil Geochemistry 9-5
9.4 Trenching 9-6
10 DRILLING 10-1
10.1 Drilling Statistics by Year 10-1
10.2 Ongoing Exploration Drilling 10-2
1.1 General Drilling Procedures 10-3
10.2.1 Safety 10-3
10.2.2 Hole Numbering 10-3
10.2.3 Drill Rig Supervision 10-3
10.2.4 Drill Site Preparation 10-4
10.2.5 Drill Rig Setup 10-4
10.2.6 Drillhole Surveys 10-5
10.2.7 Environmental Management 10-5
10.2.8 Recovery and Re-drilling a Hole 10-6
10.3 Diamond Drilling 10-6
10.3.1 Wedging 10-10
10.3.2 Down Hole Motor (DHM) 10-11
10.4 Reverse Circulation Drilling 10-11
11 SAMPLE PREPARATION, ANALYSIS AND SECURITY 11-1
11.1 Core Handling and Sampling 11-1
11.2 RC Sample Handling and Sampling 11-3
11.3 Sample Transit, Security and Chain of Custody 11-4
11.4 Sample Analysis Methods 11-7

11.5 Data Management 11-8
11.6 Density Measurements 11-9
11.7 Quality Assurance and Quality Control (QA/QC) Procedures 11-10
11.7.1 Blanks 11-11
11.7.2 Certified Reference Materials 11-13
11.7.3 Duplicates 11-19
11.7.4 Umpire Check Assays 11-26
11.8 External Audit – Qualitica Consulting Inc, September 2022 11-29
11.9 QP Conclusions and Recommendations 11-29
12 DATA VERIFICATION 12-1
12.1 Site Visits 12-1
12.2 QP Duplicate Samples 12-7
12.3 Drill Core Inspection 12-11
12.4 Drillhole Database Verification 12-13
12.5 Database Validation and Verification by the Qualified Person 12-14
12.6 QP Commentary and Conclusions 12-15
13 MINERAL PROCESSING AND METALLURGICAL TESTING 13-1
13.1 Introduction 13-1
13.2 Sample Selection 13-1
13.3 Intensive Cyanidation Test Work 13-3
13.4 Comminution Test Work 13-5
13.5 Chemical Analysis 13-6
13.6 Mineral Analysis 13-7
13.7 Gravity Test Work 13-9
13.8 Whole-of-Ore Leach Tests 13-10
13.9 Gravity-Leach Tests 13-10
13.10 Carbon-in-Leach Tests 13-10
13.11 Gravity-CIL Tests 13-11
13.12 Cyanide Destruction Tests 13-12
13.13 Acid Base Accounting Tests 13-13
13.14 Gold Recoveries 13-14
13.15 Recommendations 13-15
14 MINERAL RESOURCE ESTIMATES 14-1
14.1 Introduction 14-1
14.2 Estimation Methodology 14-3

14.3 Resource Database 14-4
14.4 Geological Models 14-7
14.4.1 Lithology and Weathering Models 14-7
14.4.2 Mineralization Model 14-9
14.5 Assays, Capping and Compositing 14-13
14.5.1 Raw Assays 14-13
14.5.2 Capping 14-13
14.5.3 Compositing 14-18
14.6 Density Measurements 14-22
14.7 Variography 14-24
14.8 Block Modelling 14-26
14.9 Block Model Interpolation 14-27
14.10 Grade Estimation Validation 14-29
14.10.1 Visual Validation 14-29
14.10.2 Global Statistical Validation 14-30
14.10.3 Local Statistical Validation – Swath Plot 14-30
14.11 Mineral Resources 14-31
14.11.1 Mineral Resources Classification 14-31
14.11.2 Reasonable Prospects of Eventual Economic Extraction (RPEEE) 14-35
14.12 Mineral Resource Statement 14-39
14.12.1 Cut-Off Grade Sensitivities 14-42
15 MINERAL RESERVE ESTIMATES 15-1
16 MINING METHODS 16-1
16.1 Summary 16-1
16.2 Geotechnical Considerations 16-2
16.2.1 Rock Mass Characterization 16-2
16.2.2 Open Pit 16-3
16.2.3 Slope Stability 16-3
16.2.4 Underground 16-4
16.2.5 Stope Sizing 16-5
16.2.6 Dilution 16-7
16.2.7 Ground Support 16-7
16.3 Hydrogeology 16-8
16.4 Open Pit Mining 16-8
16.4.1 Pit Optimization 16-8
16.4.2 Slope Recommendations 16-9
16.4.3 Pit Optimization Parameters and Cut-Off-Grade 16-11
16.4.4 Mine Phases 16-16
16.4.5 Ramp Designs 16-19
16.4.6 Waste Rock Storage Facility 16-24

16.4.7 Mineralized Material Stockpile 16-25
16.4.8 Mine Haulage Roads 16-26
16.4.9 Open Pit Production Schedule 16-28
16.4.10 Mine Operations and Equipment Selection 16-35
16.4.10.1 Drilling and Blasting 16-35
16.4.10.2 Loading 16-37
16.4.10.3 Hauling 16-39
16.4.10.4 Support Operations 16-41
16.4.10.5 Mine Dewatering 16-41
16.4.11 Mining Fleet Requirements 16-42
16.4.12 Mobile Crushing Plant 16-46
16.4.13 Mine Maintenance 16-46
16.4.14 Mine Management & Technical Services 16-46
16.5 Underground Mining 16-46
16.5.1 Underground Mining Method 16-46
16.5.2 Cut-Off Grade 16-49
16.5.3 Potentially Economical Portion of the Mineral Resource Estimate 16-51
16.5.4 Underground Mine Design 16-53
16.5.4.1 Development Design 16-53
16.5.4.2 Stope Design 16-58
16.5.4.3 Physicals Summary 16-61
16.5.4.4 Development and Production Rates 16-62
16.5.5 Development and Production Sequencing 16-63
16.5.6 Underground Mine Equipment 16-66
16.5.7 Underground Mine Ventilation and Cooling 16-67
16.5.8 Underground Mine Services 16-72
16.5.8.1 Dewatering 16-72
16.5.8.2 Cemented Rockfill Plant 16-74
16.5.8.3 Compressed Air 16-75
16.5.8.4 Communications 16-75
16.5.8.5 Fuel Storage and Distribution 16-75
16.5.8.6 Explosives Storage and Handling 16-75
16.5.8.7 Personnel and Underground Material Transportation 16-75
16.5.8.8 Equipment Maintenance 16-76
16.5.9 Underground Mine Safety Measures 16-76
16.5.9.1 Emergency Exits 16-76
16.5.9.2 Refuge Stations 16-76
16.5.9.3 Mine Rescue 16-77
16.5.9.4 Emergency Stench System 16-77
16.6 Mine Manpower 16-77
16.6.1 Open Pit Mine Manpower Requirements 16-78
16.6.2 Underground Mine Manpower Requirements 16-84
16.7 Combined Production 16-90
17 RECOVERY METHODS 17-1
17.1 Introduction 17-1

17.2 Process Design Criteria 17-3
17.3 Process Plant Description 17-5
17.3.1 Primary Crushing 17-5
17.3.2 Material Stockpiles 17-5
17.3.3 Grinding 17-6
17.3.4 Gravity Gold Recovery 17-6
17.3.5 Pre-Leach Thickening and CIL 17-7
17.3.6 Cyanide Detoxification 17-7
17.3.7 Acid Wash and Elution 17-8
17.3.8 Carbon Regeneration 17-9
17.3.9 Electrowinning and Gold Room 17-9
17.3.10 Tailings Storage Facility 17-10
17.4 Reagents 17-10
17.4.1 Sodium Cyanide 17-10
17.4.2 Hydrated Lime 17-11
17.4.3 Copper Sulphate 17-11
17.4.4 Sodium Metabisulphite 17-11
17.4.5 Sodium Hydroxide 17-11
17.4.6 Hydrochloric Acid 17-11
17.4.7 Flocculant 17-11
17.5 Plant Services 17-12
17.5.1 Plant & Instrumentation Air 17-12
17.5.2 Oxygen Generation 17-12
17.5.3 Fresh and Fire Water 17-12
17.5.4 Potable Water 17-13
17.5.5 Gland Seal Water 17-13
17.5.6 Process Water 17-13
17.6 Metallurgical Accounting 17-13
17.7 Plant Control System 17-14
17.8 Plant Consumption 17-17
17.8.1 Energy 17-17
17.8.2 Reagents & Consumables 17-17
17.9 Process Plant Personnel 17-18
17.10 Recommendations 17-19
18 PROJECT INFRASTRUCTURE 18-1
18.1 Site Layout 18-1
18.1.1 Roads and Drainage 18-1
18.2 Site Infrastructure 18-3
18.3 Camp Accommodations 18-3
18.3.1 Dormitory 18-3
18.3.2 Kitchen & Lunchroom 18-5

18.3.3 Camp Office Welcome Centre and Laundry 18-6
18.3.4 Recreational Centre 18-7
18.3.5 Greenhouse and Nursery 18-8
18.4 Mine Infrastructure 18-8
18.4.1 Mine Maintenance Facility & Warehouse area. 18-8
18.4.2 Main Administration Building 18-10
18.4.3 Mine Dry 18-10
18.4.4 Explosive Storage 18-11
18.5 Process Infrastructure 18-12
18.5.1 Mill Offices 18-12
18.5.2 Assay Laboratory 18-12
18.5.3 Reagent Storage 18-13
18.6 Waste Storage and Tailings Facilities 18-14
18.6.1 Waste Storage Facility 18-14
18.6.2 Tailings Storage Facility 18-19
18.7 Water Management 18-25
18.7.1 Industrial / Fire Water 18-25
18.7.2 Potable Water 18-25
18.7.3 Sewage Treatment and Oil-Water Separation 18-25
18.8 Fuel Storage and Distribution 18-25
18.9 Power Supply and Distribution 18-26
18.10 Communications 18-27
18.11 Offsite Infrastructure 18-28
19 MARKET STUDIES AND CONTRACTS 19-1
19.1 Gold Market 19-1
19.2 Metal Price 19-1
19.3 Contracts 19-2
20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 20-1
20.1 Environmental and Social Conditions 20-1
20.1.1 Baseline Studies 20-2
20.1.1.1 Physical Baseline 20-3
20.1.1.2 Biological Baseline 20-10
20.1.1.3 Social Baseline 20-14
20.1.2 Future Environmental and Social Baseline Studies 20-19
20.2 Environmental Permitting 20-22
20.2.1 Guyana Environmental Protection Act 20-22
20.2.1.1 EPA's Role in EIAs 20-22
20.2.2 Regulatory Timelines 20-29
20.3 Mine Closure 20-30

21 CAPITAL AND OPERATING COSTS 21-1
21.1 Initial Capital Expenditures 21-2
21.1.1 Basis of Estimate 21-2
21.1.2 Initial CAPEX Summary 21-3
21.1.3 Infrastructures 21-5
21.1.4 Power Supply and Electrical 21-6
21.1.5 Water Management 21-7
21.1.6 Surface Equipment 21-8
21.1.7 Mining 21-9
21.1.8 Process Plant 21-10
21.1.9 Construction Indirects 21-11
21.1.10 General Services 21-12
21.1.11 Pre-production and Commissioning Expenditures 21-13
21.1.12 Contingency 21-15
21.2 Sustaining Capital 21-15
21.3 Closure Costs 21-17
21.4 Operating Costs 21-20
21.4.1 Mining Costs 21-24
21.4.2 Processing Costs 21-27
21.4.3 Power Costs 21-29
21.4.4 General and Administration Costs 21-30
22 ECONOMIC ANALYSIS 22-1
22.1 Overview 22-1
22.2 Cautionary Statements 22-1
22.3 Key Assumption 22-2
22.3.1 Gold Price 22-2
22.3.2 Fuel Price and Energy 22-2
22.3.3 Other Assumptions 22-3
22.4 Metal Production and Revenues 22-3
22.5 Royalties 22-6
22.6 Capital Expenditures 22-6
22.7 Initial Capital 22-6
22.7.1 Sustaining Capital 22-7
22.8 Working Capital 22-7
22.8.1 Reclamation & Closure Cost 22-8
22.8.2 Salvage Value 22-8
22.9 Operating Cost Summary 22-8
22.10 Taxation 22-11
22.11 Economic Results 22-11
22.12 Sensitivity Analysis 22-14

23 ADJACENT PROPERTIES 23-1
23.1 G2 Goldfields Inc. 23-2
23.1.1 Mineral Rights 23-2
23.1.2 Exploration Work 23-4
23.1.3 Mineral Resources 23-4
24 OTHER RELEVANT DATA AND INFORMATION 24-1
24.1 Project Execution Plan 24-1
24.2 Project Schedule 24-2
25 INTERPRETATION AND CONCLUSIONS 25-1
25.1 Summary 25-1
25.2 Geology and Mineral Resources 25-3
25.3 Mining 25-4
25.4 Metallurgical Testing and Mineral Processing 25-5
25.5 Recovery Methods 25-6
25.6 Environmental, Social and Permitting Considerations 25-7
25.7 Capital and Operating Costs 25-9
25.8 Economic Analysis 25-10
25.9 Risks and Opportunities 25-11
25.9.1 Risks 25-11
25.9.2 Opportunities 25-11
26 RECOMMENDATIONS 26-1
26.1 Geology and Mineral Resources 26-1
26.2 Mining 26-2
26.2.1 Geotechnical Studies for Pit Slopes 26-2
26.2.2 Geotechnical Studies for Underground Workings 26-2
26.2.3 Alternatives Optimization 26-2
26.3 Metallurgical Testing and Mineral Processing 26-2
26.4 Recovery Methods 26-3
26.5 Project Infrastructure and Plant Design 26-3
26.6 Environmental, Permitting and Social Considerations 26-4
27 REFERENCES 27-1

List of Figures

Figure 1.1: Site Plan View 1-16
Figure 4.1: Oko West Project Location and Access 4-1
Figure 4.2: Oko West Prospecting Licence 4-2
Figure 4.3: Outline of the Oko West Property 4-3
Figure 5.1: Location Map of Major Access Roads and Topographic Features 5-2
Figure 5.2: Location Map of Local Roads in the Project Area 5-3
Figure 5.3: Overhead View of the Oko West Camp Showing Available Facilities 5-5
Figure 5.4: Drone View of the Oko West Camp, Looking South 5-6
Figure 7.1: Simplified Geological Map of the Guiana Shield 7-1
Figure 7.2: Schematic Geology Map of North-Central Guyana 7-3
Figure 7.3: Oko West Permit Simplified Geology and Geomorphology Map 7-4
Figure 7.4: Outcrop of Carbonaceous and Volcanoclastic Sediments Saprolite (Exploration Block 5) 7-6
Figure 7.5: Oko West Kairuni Zone Geology and Mineralization Map with Exploration Blocks 7-7
Figure 7.6: Granitoid Rock Samples 7-9
Figure 7.7: Geological Cross-Section 701800 N, Looking North 7-10
Figure 7.8: Longitudinal Inclined Section Along Mineralized Zone (Blocks 1 to 6) 7-11
Figure 7.9: Geological Cross-Section 701800 N, Looking North 7-12
Figure 7.10: Geological Cross-Section 701560 N, Looking North 7-13
Figure 7.11: Volcanoclastic Core Samples 7-14
Figure 7.12: Siltstone and Sandstone Core Samples 7-15
Figure 7.13: Carbonaceous Sediment Core Samples 7-17
Figure 7.14: Mineralized Structures in Core Samples 7-19
Figure 7.15: Mineralized Structures in Core Samples (EV and SV) 7-20
Figure 7.16: Tectonic Events and Gold Mineralization at Oko West (Lacroix and Hainque, 2024) 7-22
Figure 7.17: Gold Mineralization in Carbonaceous Sediment Core Samples 7-23
Figure 7.18: Gold Mineralization in Volcanoclastic and Siltstone / Sandstone Core Samples 7-24
Figure 9.1: RTP 1VD Map of Airborne Magnetic Data 9-2
Figure 9.2 : Map of RTE Ground Mag Coverage over Terrain 9-3
Figure 9.3: Map of IP Chargeability Coverage with Mineralized Domains at 0 m RL Over Terrain 9-4
Figure 9.4: Stream Geochemistry Survey and Reconnaissance Mapping Points Plotted on Geology 9-5
Figure 9.5: Map of Soil Geochemical Program with Anomalies. Lithology Background Superimposed Over Terrain 9-6
Figure 9.6: Map of Trenching Results in Kairuni Zone Plotted on Geology and Soil Geochemistry 9-8
Figure 10.1: Sandvik 710 Diamond Drill Rig at the Oko West Project 10-7
Figure 10.2 : RC OKWR22-128 and Diamond Twin Hole OKWD22-127 10-8

Figure 10.3: RC OKWR21-019 and Diamond Twin Hole OKWD22-125. 10-9
Figure 10.4: Reverse Circulation Rig (compressor and booster not shown) 10-12
Figure 10.5 : Map of Scout RC Program with Anomalies on Lithology 10-13
Figure 10.6: Rotary Splitter Attached to Metzke Cyclone 10-14
Figure 11.1: Drill Core Cutting – Oko West Gold Project 11-3
Figure 11.2: Chip Logging Geologist 11-4
Figure 11.3: Bagging 11-5
Figure 11.4: Pulp Rejects Storage 11-6
Figure 11.5: Sample Analysis Procedures Schema as Implemented at Oko West Gold Project 11-8
Figure 11.6: Density Measurements Balance Setup 11-9
Figure 11.7: Identification of CRMs Before Submission to Laboratory 11-11
Figure 11.8: Blank Used by Company as Quality Control Sample 11-12
Figure 11.9: Blanks Control Chart for Gold – 2021-2024 Drilling Program 11-13
Figure 11.10: Sample Control Chart of CRM OREAS 240 11-15
Figure 11.11: Sample Control Chart of CRM OREAS 236 11-16
Figure 11.12: Sample Control Chart of CRM OREAS 211 11-17
Figure 11.13: Sample Control Chart of CRM OREAS 250b 11-18
Figure 11.14: Sample Control Chart of CRM OREAS 254b 11-19
Figure 11.15: RC Field Duplicates – 2020-2024 Drilling Programs 11-21
Figure 11.16: Trench Field Duplicates – 2020-2024 Drilling Programs 11-22
Figure 11.17: Diamond Drill Hole (½) Field Duplicates – 2020-2024 Drilling Programs 11-23
Figure 11.18: Coarse Reject Duplicates – 2020-2024 Drilling Program 11-25
Figure 11.19: Pulp Duplicates Check – 2020-2024 Drilling Programs 11-26
Figure 11.20: Actlabs vs MSA Labs Check Assays 11-28
Figure 12.1: Verification of Drill Collar – OKWD23-220 12-3
Figure 12.2: Sample Reject Storage in Georgetown (top) & Core Storage Facilities Onsite (bottom) 12-4
Figure 12.3: Outcrop Inspection (top) and Closeup of Outcrop OKWT20-001. 12-5
Figure 12.4 : Core Cutting Facility (top) and Core Sampling Facility (bottom) 12-6
Figure 12.5: QP Sampling (Top) and QP Sample with Security Tag (Bottom). 12-10
Figure 12.6: Scatter Plot Showing Original Assays (X-axis) vs. QP Duplicate Assays (Y-axis) 12-11
Figure 12.7: Typical Examples of Gold Mineralized Intervals Displaying Various Alteration Assemblages and Veining 12-12
Figure 12.8: Contact Between Overburden Material and Saprolite (top, water bottle as scale), and Primary Textures in Saprolite (bottom) 12-13
Figure 13.1: Metallurgical Sample Location 13-2
Figure 13.2: Mineral Content 13-8
Figure 13.3: Mineral Analysis – Sulphur Distribution 13-9

Figure 14.1 : Drillhole Database, Coloured by Drillhole Type 14-6
Figure 14.2: Geological Model Plan View and Vertical Section 14-8
Figure 14.3: Weathering Model Isometric View 14-9
Figure 14.4: Oko West Mineralization Model 14-11
Figure 14.5: Oko West Mineralization Model - Zoom In 14-12
Figure 14.6: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the LDZ Mineralized Domain 14-15
Figure 14.7: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2 Mineralized Domain 14-16
Figure 14.8: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2HG Mineralized Domain 14-17
Figure 14.9: Oko West Database Histogram of Sampled Interval Lengths 14-20
Figure 14.10: Composited and Uncomposited Assays Comparative Bar Charts 14-21
Figure 14.11: Density Model Coloured by Density Value 14-23
Figure 14.12: Oko West Experimental Variograms for Mineralized Domain AU_2 and AU_2_HG Combined 14-24
Figure 14.13: Mineralized Domain Block Model 14-27
Figure 14.14: Oko West Block Model and Composites Visual Validation 14-29
Figure 14.15: Swath Plots for X (along strike), Y (along cross-strike) and Z for Domain AU_2 and AU_2HG Combined 14-31
Figure 14.16: Mineral Resource Classification with Pit Outline Optimized Using Whittle 14-33
Figure 14.17: Underground Mineral Resource Classification Constrained Within Stopes Optimized from DSO 14-34
Figure 14.18: Open-Pit Optimization with Block Model Coloured by Gold Grades (g/t) 14-38
Figure 14.19: Underground Stope Optimization with Block Model Coloured by Gold Grades (g/t) 14-39
Figure 14.20: Indicated and Inferred Grade-Tonnage Curves for In-pit Resource 14-43
Figure 14.21: Indicated and Inferred Grade-Tonnage Curves Within Underground Stopes Modelled at Different Cut-off Grades Using DSO 14-44
Figure 16.1: Slope Configuration 16-9
Figure 16.2: Pit by Pit Graph @ USD 1,750/oz Gold Price 16-15
Figure 16.3: End of LOM Pit Layout 16-17
Figure 16.4: Phase Limits 16-18
Figure 16.5: Phase 0 16-20
Figure 16.6: Phase 1 16-21
Figure 16.7: Phase 2 16-22
Figure 16.8: Phase 3 16-23
Figure 16.9: Phase 4 16-24

Figure 16.10: Waste Storage Facility 16-25
Figure 16.11: Double Lane Ramp Design Criteria 16-27
Figure 16.12: Single Lane Ramp Design Criteria 16-27
Figure 16.13: Open Pit Mine Production by Material Type (without stockpile reclaim) 16-29
Figure 16.14: Open Pit Mine Production by Phase 16-29
Figure 16.15: Mineralized Material Type Production Mined 16-29
Figure 16.16: Mine Development Y-1 16-31
Figure 16.17: Mine Development Y4 16-32
Figure 16.18: Mine Development Y9 16-33
Figure 16.19: Mine Development – Y13 (End of LOM) 16-34
Figure 16.20: Average Cycle Times by Material Type 16-40
Figure 16.21: Truck Requirements 16-40
Figure 16.22: Dewatering Volumes and Quantity of Pumps Over Time 16-42
Figure 16.23: Typical Longitudinal Stoping Sequence 16-48
Figure 16.24: Typical Transverse Stoping Sequence 16-49
Figure 16.25: Underground Mine Transverse Stope Types 16-52
Figure 16.26: Typical Larger Level Plan View 16-54
Figure 16.27: Typical Smaller Level Plan View 16-54
Figure 16.28: Mine Development Longitudinal View - Looking West 16-55
Figure 16.29: Underground Mine Development Longitudinal View - Looking North 16-55
Figure 16.30: Underground Mine Longitudinal View by Mining Blocks - Looking West 16-58
Figure 16.31: Underground Mine Longitudinal View by Mining Blocks - Looking North 16-59
Figure 16.32: Underground Mine Longitudinal View by Mining Method - Looking West 16-59
Figure 16.33: Underground Mine Longitudinal View by Mining Method - Looking North 16-60
Figure 16.34: Underground Mine Plan View with the Open Pit 16-60
Figure 16.35: Underground Mine Mineralized Material Production by Zone 16-64
Figure 16.36: Underground Mine Ventilation Network – Looking West 16-70
Figure 16.37: Underground Mine Ventilation Network – Isometric View 16-71
Figure 16.38: Underground Mine Dewatering Network 16-73
Figure 16.39: OP and UG Mine Workforce 16-78
Figure 16.40: Gold Processed 16-92
Figure 16.41: Gold Production per Mine 16-93
Figure 17.1: Overall Flowsheet 17-2
Figure 18.1: General Site Plan 18-2
Figure 18.2: Typical Camp Dorm – Type A 18-4
Figure 18.3: Typical Camp Dorm – Type B 18-5
Figure 18.4: Typical Camp Dorm – Type C 18-5

Figure 18.5: Typical Kitchen & Lunchroom – 3D VIEW 18-6
Figure 18.6: Typical Kitchen & Lunchroom – PLAN VIEW 18-6
Figure 18.7: 3D View of Camp Office and Laundry Area 18-7
Figure 18.8: Camp Office / Welcome Centre - Plan View 18-7
Figure 18.9: Maintenance Facility and Warehouse Area 18-9
Figure 18.10: Wash Bay 18-9
Figure 18.11: Main Administration Building 18-10
Figure 18.12: Mine Dry - Plan View 18-11
Figure 18.13: Mill Office - 3D VIEW 18-12
Figure 18.14: Assay Laboratory 18-13
Figure 18.15: Reagent Storage Floor Plan 18-14
Figure 18.16: WSF Configuration 18-15
Figure 18.17: Final Configuration of WSF 18-17
Figure 18.18: Sumps Planned Locations of the WSF 18-18
Figure 18.19: Typical Dam Sections 18-19
Figure 18.20: TSF Embankments and Anticipated Foundation Treatment Areas 18-20
Figure 18.21: Fuel Storage 18-26
Figure 18.22: Offsite Infrastructure Arrangement 18-29
Figure 19.1: Monthly Average Gold Price 19-2
Figure 20.1: EIA Process for Guyana 20-30
Figure 22.1: Gold Production Over LOM 22-5
Figure 22.2: Gold Production Over LOM by Mining Method 22-5
Figure 22.3: Initial CAPEX by Month 22-7
Figure 22.4: After-Tax Total Cash Flow Sensitivity 22-16
Figure 22.5: After-Tax NPV (5%) Sensitivity 22-16
Figure 22.6: After-Tax Internal Rate of Return Sensitivity 22-17
Figure 22.7: After-Tax Payback Period Sensitivity 22-17
Figure 23.1: Oko West Prospecting Licence (PL) and Adjacent Mineral Permits 23-2
Figure 23.2: Mineral Title Held by G2 Goldfields (Ilieva et al., 2022) 23-3
Figure 24.1: Oko West Project Schedule – Level 1 24-3
Figure 25.1: Processing by Rock Type 25-5

List of Tables

Table 1.1: In-pit and Underground Mineral Resources Estimate at Oko West 1-11
Table 1.2: Capital Expenditures Summary 1-20
Table 1.3: Opex Costs 1-21
Table 1.4: Project Economic Results Summary 1-23
Table 1.5: Gold Price Sensitivity 1-24
Table 1.6: Opex Sensitivity 1-25
Table 1.7: Initial Capex Sensitivity 1-25
Table 1.8: Cost Estimate Associated with Recommendations 1-27
Table 2.1: Summary of Qualified Persons 2-2
Table 2.2: Site Visit Dates of Qualified Person 2-3
Table 2.3: Table of Abbreviations 2-5
Table 4.1: Coordinates Defining the Oko West Prospecting Licence 4-4
Table 9.1: Trenching Statistics 9-9
Table 10.1: Drilling (DD-RC) and Surface Trenches Conducted on the Project by Year, Up to February 7, 2024. 10-1
Table 10.2: Drilling (DD-RC) and Surface Trenches Conducted on Project in 2024, Up to June 25, 2024. 10-2
Table 10.3: Drilling Recovery by Weathering Domain 10-10
Table 11.1: Summary of CRM Performance Results as Implemented on Oko West Drilling Resource Area (2020-2024) – CRM Samples Assayed by Actlabs 11-14
Table 11.2: Quality Control (QC) Duplicates Submitted by the Company 11-20
Table 12.1: Validation of Drill Collar Coordinates (PSAD 1956 UTM Zone 21N) 12-2
Table 12.2: Independent Core Duplicate Results 12-8
Table 12.3: Pulp Re-assay Results 12-9
Table 13.1: Intensive Leach Results 13-4
Table 13.2: Average Intensive Leach Gold Recoveries 13-5
Table 13.3: Comminution Summary 13-5
Table 13.4: Comminution Design Parameters (Fresh Rock) 13-6
Table 13.5: Chemical Analysis 13-7
Table 13.6: Overall Gold Extraction Results 13-11
Table 13.7: Cyanide Destruction Summary 13-12
Table 13.8: Acid-Base Accounting Summary 13-14
Table 13.9: Gold Recoveries 13-14
Table 14.1: In-pit and Underground Updated Mineral Resource Estimate at Oko West. 14-2
Table 14.2: Summary of Drillholes and Assays Used in the Oko West Resource Estimate 14-5

Table 14.3: Oko West Gold Assays Statistics (length weighted) 14-13
Table 14.4 : Capping Applied to Oko West Mineralized Domains 14-14
Table 14.5: Statistics of Uncapped and Capped Assays of Oko West, per Domain (length weighted) 14-18
Table 14.6: Uncomposited and Composited Statistic by Mineralized Domain 14-19
Table 14.7: Host Rocks Density Statistics by Weathering Profile 14-22
Table 14.8: Variogram Parameters Used for the Mineral Resource Estimation per Domain 14-25
Table 14.9: Oko West Block Model Parameters 14-26
Table 14.10: Search Ellipsoids by Mineralized Domains and Search Passes 14-28
Table 14.12: Sample Search Criteria by Passes 14-29
Table 14.13: Mean Grade Comparison Between Composites and Blocks, per Domains (volume weighted) 14-30
Table 14.14: Parameters Used for Open Pit Whittle Optimization and Open-pit Cut-off Grade Assumptions 14-35
Table 14.15: Parameters Used for Stope Optimization and Underground Cut-off Grade Assumptions 14-36
Table 14.16: Oko West Deposit In-pit Mineral Resource Estimate – Effective Date February 7, 2024. 14-40
Table 14.17: Oko West Deposit Underground Mineral Resource Estimate – Effective Date February 7, 2024. 14-41
Table 14.18: Oko West In-Pit Cut-off Grade Sensitivity 14-45
Table 14.19: Oko West Underground Cut-off Grade Sensitivity 14-45
Table 16.1: Q' Rock Mass Classifications 16-5
Table 16.2: Open-Pit Slopes Angles per Rock Type and Face Orientation 16-9
Table 16.3: Geotechnical Berms 16-10
Table 16.4: Overall Slope Angle for Open-Pit 16-10
Table 16.5: Economics Optimization Parameters 16-12
Table 16.6: Whittle Shell Results 16-14
Table 16.7: Pit Shell Selection 16-15
Table 16.8: Mining Resources by Phase 16-16
Table 16.9: Waste Storage Facility Capacity and Design Parameters 16-25
Table 16.10: Open Pit Mining Schedule Summary 16-30
Table 16.11: Drill and Blast Parameters 16-35
Table 16.12: Loading Fleet Productivity Assumptions 16-38
Table 16.13: Equipment Usage Assumption 16-43
Table 16.14: Major Equipment Purchase Schedule 16-44
Table 16.15: Support Equipment Purchase Schedule 16-45
Table 16.16: Underground Mine Cut-offs Calculation Parameters 16-50
Table 16.17: Underground Mine Detailed LHOS Mining Cost 16-51
Table 16.18: Underground Mine Backfill Dilution Parameters 16-52

Table 16.19: Underground Mine Stope Optimizer Parameters....16-53
Table 16.20: Underground Mine Development Parameters....16-56
Table 16.21: Underground Mine Development Factors....16-56
Table 16.22: Underground Mine Development Type and Dimensions....16-57
Table 16.23: Underground Mine Design Summary....16-61
Table 16.24: Underground Mine Physicals Summary....16-62
Table 16.25: Underground Mine Scheduler Rates....16-63
Table 16.26: Underground Mine Production Plan....16-65
Table 16.27: Underground Mine Mobile Equipment Fleet....16-66
Table 16.28: Underground Mine Fresh Air Requirements per Equipment....16-68
Table 16.29: Underground Mine Fresh Air Requirements....16-69
Table 16.30: Underground Mine Ventilation System Details....16-72
Table 16.31: Underground Mine Dewatering Assumption....16-72
Table 16.32: Underground Mine Pumping Requirements Details....16-73
Table 16.33: Open Pit Mine Operations Workforce....16-79
Table 16.34: Open Pit Maintenance Workforce....16-81
Table 16.35: Open Pit Mine Engineering and Geology Workforce....16-82
Table 16.36: Open Pit Mine Total Workforce....16-83
Table 16.37: UG Engineering Workforce....16-85
Table 16.38: UG Geology Workforce....16-86
Table 16.39: UG Mine Operations Workforce....16-86
Table 16.40: UG Mine Maintenance Workforce....16-88
Table 16.41: Total UG Workforce....16-89
Table 16.42: Total Mining Workforce....16-89
Table 16.43: Yearly LOM Production Details....16-91
Table 17.1: Key Process Design Criteria....17-4
Table 17.3: Reagents Consumption....17-17
Table 17.4: Consumables Consumption....17-18
Table 17.5: Process Plant Personnel....17-18
Table 18.1: Camp Capacity....18-4
Table 18.2: Characteristics of WSF....18-16
Table 18.3: Dam Crest Elevation Versus Accumulated Fill Volume for the TSF....18-21
Table 18.4 :Reservoir Elevation vs Storage Volume....18-21
Table 18.5: Yearly Embankment and Compaction Volumes During LOM....18-24
Table 20.1: Legal Framework for National Environmental Management in relation to Mining....20-24
Table 20.2: Required Permit Overview....20-28
Table 21.1: Initial and Sustaining Capital Expenditures Summary....21-1

Table 21.2: Initial Capital Expenditures Summary 21-4
Table 21.3: Infrastructures Capital Expenditures 21-5
Table 21.4: Power Supply and Communications Capital Expenditures 21-7
Table 21.5: Water Management Capital Expenditures 21-8
Table 21.6: Surface Equipment Capital Expenditures 21-9
Table 21.7: Mining Capital Expenditures 21-9
Table 21.8: Processing Capital Expenditures 21-10
Table 21.9: Construction Indirect Capitals 21-11
Table 21.10: List of Departments Included in General Services 21-12
Table 21.11: General Services Expenditures 21-13
Table 21.12: Pre-Production & Commissioning CAPEX 21-14
Table 21.13: Sustaining Cost Summary 21-16
Table 21.14: Closure Cost Summary 21-17
Table 21.15: Closure Schedule 21-19
Table 21.16: Operating Costs Summary 21-20
Table 21.17: Total Operating Costs Summary by Year 21-22
Table 21.18: Diesel Fuel Price 21-24
Table 21.19: OP Mining Cost Summary 21-25
Table 21.20: UG Mining Cost Summary 21-26
Table 21.21: Processing OPEX 21-27
Table 21.22: Grinding Media and Reagent Consumption 21-28
Table 21.23: Consumables Operating Costs 21-29
Table 21.24: Power Operating Costs 21-30
Table 22.1: Total Power Cost in USD/kWh 22-3
Table 22.2: Milling Production Schedule Summary 22-4
Table 22.3: Operating Cost Summary (USD) 22-9
Table 22.4: Operating Cost Summary per Tonne 22-10
Table 22.5: Project Economic Results Summary 22-12
Table 22.6: Project Cash Flow Summary 22-13
Table 22.7: Gold Price Sensitivity 22-14
Table 22.8: OPEX Sensitivity 22-15
Table 22.9: Initial CAPEX Sensitivity 22-15
Table 25.1: Oko West Preliminary Economic Assessment Highlights 25-2
Table 25.2: Initial and Sustaining Capital Expenditures Summary 25-10
Table 26.1: Cost Estimate Associated with Recommendations 26-1

1 SUMMARY

1.1 Introduction

G Mining Ventures Corp. ("GMIN" or the "Company") mandated G Mining Services Inc. ("GMS") as lead consultant along with other engineering consultants to prepare a Preliminary Economic Assessment ("PEA") under the supervision of the QPs for the Oko West Project ("GYOW" or "Project"), located in Cuyuni-Mazaruni Mining Districts, Guyana.

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this Report and the PEA is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an open pit and underground mine thereat, including processing facilities and related infrastructures. This Report provides operating and capital costs estimations and an economic analysis of the Project.

This Report declares the same Mineral Resource Estimate ("MRE") statement published on February 26th, 2024. It is based, for the most part, on a drilling database dated from February 7th, 2024, and supporting geological information. The Mineral Resource statement reported herein was prepared in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The Oko West Project does not contain Mineral Reserves.

The qualified persons ("QP") of this Technical Report are the following:

- Mr. Christian Beaulieu, P.Geo., Vice President for Minéralis Services-Conseils Inc.
- Mr. Derek Chubb, P.Eng., Senior Partner at Environmental Resources Management (ERM).
- Mr. Neil Lincoln, P.Eng., President and Consulting Metallurgist for Lincoln Metallurgical Inc.
- Mr. Alexandre Burelle, P.Eng., Mine Planning and Financial Analysis Consultant for Evomine Consulting Inc.
- Mr. Paul Murphy, P.Eng., Project Manager as an independent consultant.

Mr. Christian Beaulieu visited the Project site between April 18 and April 19, 2023.

1.2 Terms of Reference

Unless otherwise stated, all the information and data contained in the Report or used in its preparation has been provided by Reunion up to February 7th, 2024. The units of measure presented in this Technical Report, unless noted otherwise are in the metric system. Currency is expressed in United States dollars ("USD"), unless stated otherwise.

1.3 Reliance on Other Experts

This Technical Report, prepared by GMS under the supervision of Qualified Persons (QPs) for GMIN, is based on the following:

- Information available to GMS at the time of preparation.
- Assumptions, conditions, and qualifications outlined in the report.
- Data, reports, and opinions provided by RGD, GMIN, and other third-party sources.

The QPs believe that the assumptions and interpretations in this report are factual and reasonable. They have relied on the data provided and believe no material facts have been withheld. The QPs have taken appropriate steps to ensure the reliability of the work and do not disclaim responsibility for the report.

The QPs have also relied on experts for environmental, legal, political, and tax matters. The following consultants were involved in preparing various reports for the project:

- ERM: Conducted environmental and social baseline and regulatory support. ERM's work is detailed in Section 20.
- TEC3: Conducted surface geotechnical investigations and provided guidelines for Sections 18 and 20.
- Alius Mine Consulting: Offered underground design guidelines, summarized in Section 16.
- NewFields Mining Design & Technical Services**: Provided open pit slope design guidelines, also summarized in Section 16.

The results and opinions in this report depend on the accuracy and completeness of the experts' information as of the report's effective date. The QPs are only responsible for the sections of the report identified in their "Certificates of Qualified Persons" submitted to Canadian Securities Administrators. Any third-party use of this report beyond provincial securities laws is at the user's own risk.

1.4 Property Description and Location

The Oko West Project straddles the Cuyuni-Mazaruni Mining Districts (administrative Region 7) in north central Guyana, South America. The Project is located approximately 100 km southwest of Georgetown, the capital city of Guyana and about 70 km from Bartica, the capital city of Region 7. The Project is accessible by the Puruni and Aremu laterite roads from the town of Itabali at the confluence of the Cuyuni and Mazaruni rivers.

The Oko West Project comprises one (1) Prospecting Licence (PL 004/2022) issued to Reunion Gold Inc., the Company's 100%-owned Guyanese subsidiary, on September 23, 2022. The PL is valid for three (3) years and is renewable for up to two (2) years. The PL has a surface area of approximately 10,890 acres (4,407 hectares).

The Guyana Government holds the surface rights to the Prospecting Licence area. The PL allows Reunion to occupy the area.

1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The Project can be accessed via numerous methods: by helicopter directly from the Eugene F. Correia International Airport near Georgetown to the site, fixed wing plane from the Eugene F. Correia International Airport to the Bartica airstrip, or by car then by boat. From Itabali to the Project site, one can use the Puruni or the Aremu laterite roads, requiring four-wheel drive vehicles.

The climate is equatorial and humid, with two dry seasons, one from approximately March to mid-April and the other from August to November. The dry season's onset and duration vary from year to year. The heaviest precipitation is expected in May and June.

The Oko West Project has operated throughout the year without any interruptions related to the weather. Laterite road conditions deteriorate significantly during the rainy seasons and might cause transportation delays.

The region's infrastructure is underdeveloped, lacking power, roads, communications, and general services. The city of Bartica (population about 10,000), at the Essequibo, Mazaruni and Cuyuni rivers' confluence, is the primary hub for artisanal mining activity in northwest Guyana. The town of Itabali, at the left margin of the Mazaruni river, is the gateway for the road transportation of goods and services to all the artisanal mining operations not reachable by a river, including the Project.

There is no available grid electrical power in the region. The entire Guyana power system currently runs on heavy fuel or diesel thermal plants installed along the coast and at Linden and Bartica. There are no power lines or substations in the project vicinity.

1.6 History

The discovery of gold in the region dates to the end of the 19th century by artisanal miners or "porknockers". Between 1966 and 1979, the British Geological Survey conducted regional field mapping and undertook geophysical surveys in the vicinity of the Project.

After a long hiatus, the Guyana Geology and Mines Commission (GGMC) conducted the Lower Puruni Regional Geochemistry Programme in 2002, which covered the Project area, identifying gold and molybdenum anomalies from stream sediment samples. Between 2010 and 2015, extensive alluvial and elluvial mining was done in the region. Local artisanal miners mined several gold-rich quartz veins at Crusher Hill, north of the Oko West project area.

The first modern exploration campaigns were undertaken in 2016, where Sandy Lake Gold Inc. (later to be renamed G2 Goldfields Inc.) collected grab samples at Crusher Hill, a primary prospect north of Oko West, and reported high gold grades in shaft stockpiles associated with quartz and quartz-carbonate veins.

Reunion personnel first visited the Oko West area on October 4, 2018, to inspect outcrops and collect rock chip samples.

On April 22, 2024, G Mining Ventures Corp. and Reunion Gold Corporation announced the combination of both companies, setting the stage for the creation of a leading intermediate gold producer. Through this transaction, GMIN acquired RGD's Oko West Project. On July 9, 2024, G Mining Ventures, as well as Reunion Gold, announced the shareholders approval of the transaction.

No historical drilling, prior to December 2020, is known to have been completed on the Project.

1.7 Geological Setting and Mineralization

The Project is located within the Guiana Shield, which corresponds to the northeastern portion of the Amazonian Craton. With a total area of 900,000 km^2, it covers eastern Venezuela, Guyana, Suriname, French Guiana, the northern end of Brazil and easternmost Colombia.

This Trans-Amazonian Province is composed of large Rhyacian (2.20-2.05 Ga) granite-greenstone belts, including volcano-sedimentary rocks, metamorphosed to greenschist facies, intrusive granitoids and TTG (tonalite-trondhjemite-granodiorite) gneisses. In Guyana, the greenstone belts are described from deepest to shallowest as basalt ± ultramafic rocks, intermediate to felsic volcanic rocks, and finally, tuffs and turbiditic sedimentary rocks. They host multiple gold deposits; however, little is known about the relationship between gold mineralization, magmatism, and deformation.

Two (2) major tectonic events have affected the Trans-Amazonian Province: a D_1 event involving into a N-S convergence of the Archean African and Amazonian cratons (2.18 to 2.13 billion years ago), followed by the closure of the volcanic arc basins, defined as the D_2 event (2.11 to 2.06 billion years ago), and marked by granitic magmatism, minor mafic intrusions, and regional greenschist metamorphism, as well as folding of the volcano-sedimentary formations. At Oko West, the Oko, Aremu and Puruni plutons are most likely the result of the D_1 tectonic event. These plutons caused deformation of the Barama-Mazaruni Supergroup volcano-sedimentary rocks (2.12 billion years old), leading to the formation of gold occurrences within local structures.

Gold mineralization at Oko West straddles the north-south striking contact between Barama-Mazaruni Supergroup greenstone belt rocks to the west and a granitoid pluton to the east (the Oko pluton). Locally, the Barama-Mazaruni Supergroup sequence comprises mafic volcanics, volcaniclastics, and siliciclastic and carbonaceous sediments and is the main host to mineralization at Oko West.

Lacroix and Hainque (2024) propose a geological evolution for the Oko West mineralization, based on field observations and structural analyses:

- **D0**: Deposition of volcano-sedimentary sediments, mafic volcanic rocks, and granitoid intrusions, leading to early potassic veins and metasomatism.
- **D1a**: A compressional event caused folding (F1) and the formation of bedding-parallel veins that may have contained minor gold.
- **D1b**: Tight folding continued, creating a penetrative foliation (S1) and transposing early veins (EV1) into S1, resulting in locally dismembered and sheared veins.
- **D2**: N-S folding developed S2 crenulation and late-stage quartz-Sulphide veins (EV2). Most of the gold was deposited during this phase along fractures related to F2 and S1.
- **D3**: Formation of two (2) sets of conjugated fractures or faults, potentially with significant displacement.

Long chemical weathering typical of humid equatorial paleoclimate produced a lateritic profile up to 100 m thick from the surface. This profile is typically composed of a veneer of pisolitic colluvium or latosol overlaying a massive clay zone, which pass into a mottled zone and then saprolite / saprock before reaching unweathered rocks at depth.

1.8 Deposit Types

The Oko West gold mineralization can be classified as a structurally controlled, orogenic gold mineralization. Nearby in French Guiana, orogenic-type gold deposits are mainly related to D_2 tectono-metamorphic deformation (between 2.1 and 2.0 Ga). The mineralization occurs along shear zones in greenstone belts and is associated with granitic magmatism. Recent data from the Karouni orogenic gold deposit in Guyana support this timing, as gold mineralization has been dated to 2.084 Ga ± 14 Ma. In Suriname, mineralized shear zones develop along contacts between units of varying rheologies but also, to a lesser degree, parallel to axial plane cleavages in fold noses at the Rosebel gold mine.

1.9 Exploration

Modern exploration of the Oko West Project comprises geophysics, reconnaissance stream-sediment geochemistry, soil geochemistry, trenching and drilling. All modern exploration of the Project has been conducted by Reunion.

Gold mineralization at Oko West was first identified to the north of the Project, and after some initial reconnaissance, a stream sediment survey was conducted using Bulk Leach Extractable Gold (BLEG) techniques for gold analysis. Although this survey did not cover the current known extents of gold mineralization, a soil geochemical survey was completed over the east of the Project and defined a gold anomaly straddling the contact between the Oko pluton to the east and the volcano-sedimentary sequence to the west, with a strike length of approximately 6 km. Trenching was subsequently undertaken over the anomaly and intersected 5.98 g/t Au over 69.0 m (trench 44) until the program was interrupted due to the Covid-19 pandemic. The trenching program successfully validated the soil geochemical anomalies and confirmed the presence of significant in-situ gold mineralization in a sequence of sediments striking north-south and at the contact with the Oko pluton granitoid.

In August 2019, the Canadian company Terraquest covered the project area with an airborne geophysical survey of about 690-line km at a 200 m line spacing. At the time of writing, ground geophysical surveys (magnetic and IP surveys) are ongoing.

1.10 Drilling

Drilling commenced at Oko West in December 2020, with three (3) reconnaissance holes targeting primary mineralization beneath the previously reported trench results. After the initial discovery, drilling from 2021 to 2024 has been mainly focused on delineating gold mineralization and satisfying the drill spacing required to calculate an in-pit and underground Mineral Resource Estimate ("MRE").

Drilling methods at Oko West comprise of diamond drilling ("DDH") and reverse circulation drilling ("RC"). As of the effective date of this report, 193,041.1 m of drilling and trenching has been conducted on the Project, of which 131,379.8 m is DDH, 52,926.0 m is RC, and the remaining 8,753.3 m is trenching. Beginning late 2023, a delineation program using wedges and directional drilling was started by Reunion Gold to convert underground resource to the Indicated category. A total of 6,542.1 m was drilled using this methodology.

Drill core recovery is considered excellent, averaging 98.2% in fresh rock. The lateritic profile is drilled with HQ-diameter drill rods, and NQ-diameter drill rods are used once hard ground conditions are encountered.

RC drilling is used for reconnaissance scout drilling to test regional soil anomalies and to test for strike extensions of known mineralization. RC drill samples are sourced from an onboard splitting system on the drill rig to ensure sample quality and representativity. RC drillholes are ended when water is encountered on three (3) consecutive metres.

1.11 Sample Preparation, Analyses and Security

Diamond drill core samples are collected on average at every 1.3 m from drill core, but very between 0.1 and 2.85 metres. Sample intervals are marked by geologists. Samples are selected in potential mineralized zones based on logged geological features, such as rock type, mineralization, alteration, and veining. Reverse circulation (RC) chip samples are collected at every metre. The splitter on the rig will produce 2 kg samples for the primary lab, the field duplicate sample and the bulk sample for storage and future reference.

Blanks, certified standards, and duplicates are inserted at the same time as the sampling process is performed. Certified reference material (CRMs or standards) and blanks include one (1) control sample every ten (10) samples interchanging between standard and blank, or the equivalent of one (1) blank and one (1) standard at every 20 samples (5%). The position of blank and standard samples is adjusted to control mineralized intervals and test lab contamination. The Company's procedures of quality control (QC) samples are designed to insert one (1) standard, one (1) blank, and one (1) field duplicate at every

20 samples generated by drilling. The primary laboratory (Actlabs) sends pulp duplicates directly to the secondary laboratory (MSA Labs) for umpire check assays.

GMIN uses a sample tag system containing the sample information, including date, target, hole or trench, interval from-to in metres, sampler name and analytical code. Access to samples is only possible by cutting the tag. The samples are sent via boat and truck to the primary laboratory in Georgetown (Guyana) accompanied by one (1) company employee for the entire trip to witness that all samples reach the laboratory safely.

Bulk density measurements are taken in-house on all representative core from the lithological intervals, including mineralized and non-mineralized units, with varying degrees of hydrothermal alteration and weathering.

Sample batches are prepared following the Actlabs Code RX1 procedure. Samples are weighed and dried; crushed (<5 kg) to a fineness of 80% passing 2 mm. A riffle split of 250 g is taken from the crushed material and pulverized (mild steel) to 95% passing 105 µm (140 mesh). At Actlabs, gold analysis code FAAA-1A2 is performed using a 50 g fire assay (FA) with atomic absorption spectrometry (AAS) finish. For gold values above the upper detection limit (> 3,000 ppb), samples are assayed by fire assay with gravimetric finish (FAGRA-13A). If visible gold is observed by the geologist during the logging and sampling, the analytical method 1A4 Au fire assay metallic screen is prioritized, and the sample before and after the visible gold is also analyzed using the metallic screen method.

The assay reports by both the primary and secondary labs are distributed by e-mail directly to recipients listed in the work order, including gDat Solutions, a third-party, independent database manager.

The QP concludes that the sample preparation, analysis, and security procedures applied by Reunion are acceptable. Documentation of sampling procedures used to support the diamond and reverse circulation drilling programs is considered by the QP as consistent with best industry practice. In addition, the QP believes that sample preparation, analysis, and security procedures implemented by Reunion are comparable with the best industry standards, and robust controls are in place to ensure the integrity of the assay database.

1.12 Data Verification

Pascal Delisle, Director of the Geology and Resource department at G Mining Services (GMS), along with Émile Boily-Auclair, Mineral Resources Estimation Engineer at GMS, conducted a site visit of the Oko West Project from January 30 to February 2, 2024. They verified drill collar locations, toured core processing

facilities, reviewed sampling protocols, inspected outcropping mineralization and trenches, and collected independent verification samples.

Mr. Christian Beaulieu P.Geo., consulting geologist for GMS and qualified person (QP) under NI 43-101 visited the project on April 18 and April 19, 2023. During this site visit, the QP inspected mineralized intervals, alteration assemblages and QAQC protocols and conducted field checks of trenches and to validate drill collars. Some trenches were cleaned, and sampling protocols were assessed directly in the field with site geologists; drill core was reviewed with Reunion personnel. Drill core review permitted to observe clear relationships between gold grades (or presence of mineralization) and rock alteration / strain within mineralized domains LDZ, AU_2, AU_3, AU_3A. It is in the opinion of the QP that GMIN work practices at Oko West are in line with the CIM Best Practice Guidelines (2019).

M. Delisle and M. Boily-Auclair visited both the preferred independent laboratory, Actlabs, and the umpire laboratory, MSALABS, in Georgetown, Guyana. They meticulously inspected the sample preparation facilities and chain-of-custody protocols, ensuring transparency and robustness in the handling of samples throughout the analysis process.

The validity of the drilling database including assay certificates, collar locations, downhole surveys, and twin drill holes was reviewed. RC drilling was approved for resource estimation, except for one (1) excluded hole due to potential gold grade smearing. Overall, the QP expressed confidence in the accuracy and integrity of the drilling data and procedures at Oko West.

In addition to reviewing the sampling procedures, GMS conducted a comparative analysis of duplicate samples sent to both Actlabs and MSALABS. The results indicated a good correlation between the original assays and the duplicate assays, with slight variations attributable to factors such as sample size and laboratory processes. Despite these variations, no bias was identified, affirming the accuracy and consistency of the sampling process.

1.13 Metallurgical Testing and Mineral Processing

A metallurgical test work program conducted from May to September 2023 at Basemet Laboratories (BML) aimed to assess the metallurgical response of material domains within the Oko West deposit, determine initial metallurgical recoveries, and develop an initial flowsheet. The scope included chemical analysis, mineralogy, comminution, gravity, leach, cyanide detoxification, and acid-base tests.

Samples were selected from three weathering zones (saprolite, transition, and fresh rock) and main geological units (volcanics, metasediments, and carbonaceous sediments), resulting in 18 master

composites. Gold content in the samples varied between 0.50 and 2.48 g/tonne (as expected) and silver ranged between 0.1 and 1.8 g/tonne. Sulphur in the samples measured between 0.01 and 0.77 percent, indicating a relatively small sulphide mineral component. Base metal content in the samples were low, i.e. 67 ppm Cu, 88 ppm Zn, 7 ppm Pb (average). Arsenic and mercury levels were low at 8.5 ppm As and < 1 ppm Hg. The Preg-robbing value (PRV) of select samples was measured and indicated near negligible values of preg-robbing index (PRI).

Bulk Mineral Analysis (BMA) using QEMSCAN, was conducted on the samples. Pyrite accounted for the main sulphide mineral in almost all the samples. Chalcopyrite, sphalerite and other sulphides were also detected in lower concentrations in the majority of the other samples. The non-sulphide suite of minerals varied, consisting mainly of quartz, feldspars, muscovite/illite and chlorite and clays.

From a material hardness point of view, fresh rock is more competent, hard and abrasive in comparison to the saprolite and transition material. The fresh rock exhibits competent material (Axb 15th percentile of 32.4), hard grindability (85th percentile BBWi of 14.8 kWh/tonne) and is mildly abrasive (Ai of 0.133).

Gravity recoverable gold tests were performed on all samples using a Knelson concentrator. Gravity gold recovery for the Fresh Rock samples ranged between 36% and 63%. Gravity gold recovery for the saprolite samples ranged between 27% and 46%.

Acid base accounting (ABA) tests were completed on blended composites (following cyanide destruction testing) and on waste rock samples. All samples, except for one of transition samples had net neutralizing potential (NNP) greater than zero indicating those materials are potentially acid neutralizing. All samples except for the transition samples had neutralizing potential ratio (NPR) greater than 4.1, indicating no potential for ARD.

Whole-of-ore leach tests showed high overall gold extraction rates, with finer primary grind sizes resulting in higher extraction but also higher cyanide consumption. Subsequent gravity-leach tests and carbon-in-leach (CIL) tests showed consistently high gold extraction rates.

Overall, gold recoveries from gravity-leach tests yielded the best results, with average Au recovery of 96.0% for saprolite, 95.0% for transition and 92.5% for fresh material types.

It is recommended to conduct variability metallurgical test work to confirm the metallurgical response across different material zones which includes chemical analysis, quantitative mineralogy, comminution, gravity, pre-robbing, gravity tails leach and CIL, cyanide destruction, sequential triple contact carbon loading, oxygen uptake, settling, and acid-base accounting tests.

1.14 Mineral Resources Estimate

The MRE was prepared by Pascal Delisle, P.Geo., Director of the Geology and Resource department at G Mining Services (GMS), and Émile Boily-Auclair, engineer in mineral resources estimation at GMS. This Mineral Resource has been revised and approved by Mr. Christian Beaulieu, consulting geologist for G Mining Services and independent qualified person (QP) as defined in the National Instrument 43-101.

The MRE methodology is summarized below:

- Drillhole database validations.
- 3D modelling of host units (lithological model).
- 3D modelling of gold-bearing domains.
- Geostatistical analysis for data conditioning: mineralization domain validation, density assignment, capping assumptions, compositing and variography.
- Block modelling and grade estimation.
- Resource classification and grade interpolation validations.
- Grade and tonnage sensitivities to different cut-off grade scenarios.

The MRE considers 397 diamond drill holes, 292 reverse circulation holes and 59 trenches, completed between December 2020 and January 2024 by Reunion.

The effective date of the mineral resource estimation is February 7th, 2024, and the MRE statement is listed in Table 1.1.

Table 1.1: In-pit and Underground Mineral Resources Estimate at Oko West

Category	Updated MRE Tonnage (kt)	Updated MRE Au grade (g/t)	Updated MRE Contained Gold (Koz)
Pit Constrained Resource			
Indicated	64,115	2.06	4,237
Inferred	8,107	1.87	488
Underground Constrained Resource			
Indicated	491	1.85	29
Inferred	11,510	3.01	1,116

Category	Updated MRE Tonnage (kt)	Updated MRE Au grade (g/t)	Updated MRE Contained Gold (Koz)
Total Open Pit and Underground			
Indicated	64,606	2.05	4,266
Inferred	19,617	2.54	1,603

**Notes on Mineral Resources:*
The Mineral Resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).

1. *The Qualified Person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., consulting geologist of G Mining Services Inc.*
2. *The effective date of the Mineral Resource Estimate is February 7, 2024.*
3. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g/t Au in saprolite and alluvium / colluvium, 0.313 g/t Au in transition and 0.37 g/t Au in fresh rock.*
4. *Underground Mineral Resources are reported inside potentially mineable volume (i.e., must take material) and include below cut-off material (COG: 1.38 g/t Au).*
    a. *A change in the reporting method for the underground part of the deposit explains the differences in tonnage and average grade between this PEA and the MRE published in February 2024. Tonnage of potentially mineable material stated below cut-off (i.e., must take material) is declared for this constrained underground Mineral Resource Estimate, regardless of prior classification. Blocks have been reclassified inside each stope based on deposit knowledge and continuity and reflect the existing classification. No change in total ounces is observed.*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 0.5m x 2.5 m, and has been reported inside an optimized pit shell and optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Inverse Distance for domains AU_2A, AU_2B and AU_5, and Ordinary Kriging for all other domains. Capping was applied on eight domains, ranging from 5 g/t Au to 80 g/t Au.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$14.51/t for saprolite and alluvium / colluvium, with a 96% processing recovery US$17.16/t for transition with a 95% processing recovery and US$19.80/t for fresh rock based on 92.5% processing recovery.*
    c. *Inter-ramp angles of 30° in saprolite and alluvium/colluvium, 40° in transition and 50° in fresh rock.*
    d. *Royalty rate of 8%.*
10. *UG optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$73.26/t for fresh rock.*
    c. *Stope height of 30 m, strike length of 20 m, maximum width of 25 m and minimum width of 2 m.*
    d. *The Deswik.SO (DSO) was used to constrain the Resources.*
    e. *Royalty rate of 8%.*
11. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*
12. *These Mineral Resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*
13. *These Mineral Resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

The QP has determined that there are no known factors or issues that could significantly impact the Mineral Resource Estimate (MRE), other than the typical risks associated with mining projects, such as environmental, permitting, taxation, socio-economic, marketing, and political factors, as well as additional risk factors related to indicated and inferred mineral resources.

It was determined that the database used for estimation is reliable, and that the current drilling information is of sufficient quality for interpreting the boundaries of gold mineralization with confidence. Additionally, the assay data used for the mineral resource estimation and block modelling is considered reliable by the QP. The mineral resource estimation methodology and key assumptions considered for the MRE are described in the following sections.

1.15 Mineral Reserve Estimate

This Preliminary Economic Assessment (PEA) of the Oko West gold deposit is based on Indicated and Inferred Mineral Resources. Because of the inclusion of Inferred Resources, it is not applicable to determine Mineral Reserves at this stage of the project. Economic zones will be classified as mineralized material only.

1.16 Mining Methods

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) and mechanized long hole open stoping for the underground (UG) mine. The initial milling rate is set at 6 Mtpa for processing hard rock, increasing to 7 Mtpa when incorporating saprolite, following a 5-month ramp-up during the open pit phase. The milling process is designed to operate for 13 years, with stockpiles peaking at 4.4 Mt by Year 2 to maintain consistent mill feed. A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and to be used in an economic analysis except as allowed in PEA studies. There is no guarantee that Inferred Mineral Resources can be converted to Indicated or Measured Mineral Resources and, as such, there is no guarantee the Project economics described herein will be achieved.

The OP will utilize a fleet of diesel-powered equipment, including drills, haul trucks, and hydraulic shovels. The Project consists of a main pit that is deeper and centered on Block 4, with two (2) smaller sub-pits positioned on the southern extension to the main one. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LOM) of 13 years. A total of 60.7 Mt of mineralized material will be mined at an average diluted gold grade of 1.72 g/t Au. A total of 364.6 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.0 tonnes of waste per tonne of mineralized material. The primary production equipment includes 22 m^3 diesel-hydraulic shovels paired with 136-t off-highway mining trucks for the mineralized material and waste. The mining operation is planned to be fully owner-operated, with pre-production mining scheduled over approximately 24 months

to secure construction material and to remove overburden to allow access to the mineralized material. A total of 28.4 Mt of waste and overburden as well as 3.5 Mt of mineralized material will be mined in the pre-production and ramp-up period.

The UG operation consists of one (1) mine separated in three (3) zones: the main zone and two (2) satellites zones, all accessible from a surface mine portal through the main decline ramp. The selected mining method is long hole open stoping (LHOS), including transverse stoping and longitudinal stoping variations.

The LOM for the UG mine is expected to be 13 years including construction, development, pre-production and the full production period. Over this LOM, the UG mine is expected to be in production for 11 years, including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed and sustain full production. Initially, a contract mining period is anticipated for the construction and development of the mine followed by a transition to full owner-operated mining activities.

The UG mine is expected to achieve an average production rate of 4,250 tpd of mineralized material, with 4,000 tpd derived from stope production and 250 tpd from lateral development. Development of the UG mine includes approximately 47.0 km of lateral and 3.2 km of vertical development to be excavated. A total of 14.5 Mt of mineralized material is expected to be mined at an average diluted gold grade of 3.19 g/t Au. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

1.17 Recovery Methods

The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing material to produce doré. The flowsheet is based on metallurgical test work, industry standards and conventional unit operations.

The process plant is designed to nominally treat 6 Mtpa of fresh rock and will consist of comminution, gravity concentration, cyanide leach and adsorption via carbon-in-leach (CIL), carbon elution and gold recovery circuits. CIL tailings will be treated in a cyanide destruction circuit and pumped to a tailings storage facility.

The key project design criteria for the process plant are listed below:

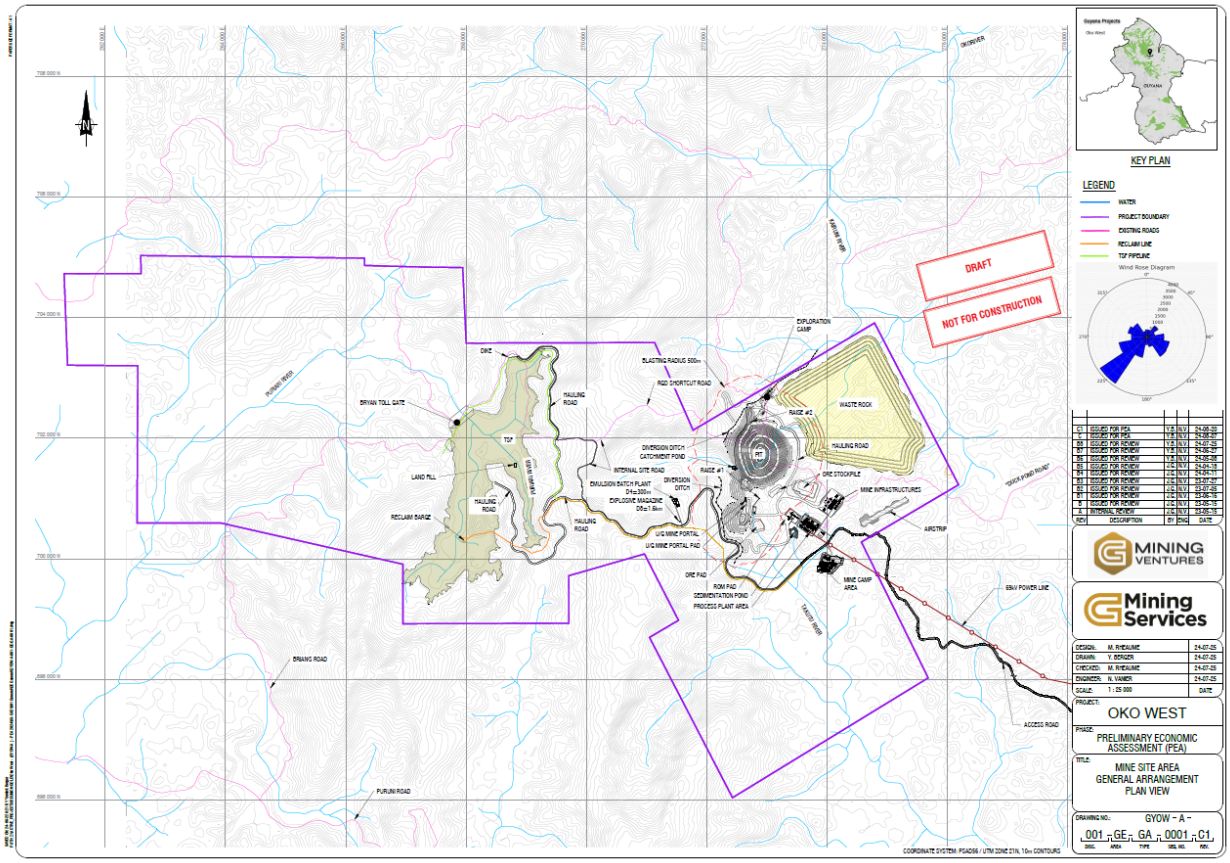
- Nominal throughput of 6 Mtpa of fresh rock and up to 7 Mtpa when treating softer material.
- Primary crushing of material.

- In-line coarse material stockpile and reclaim.
- Grinding consisting of semi-autogenous (SAG) mill and ball mill with hydrocyclones producing a final product P80 of 75 µm.
- Gravity concentration to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening.
- Cyanide leaching, and carbon adsorption via a Carbon-in-Leach (CIL) circuit. CIL residence time of 48 hours to achieve optimal gold extraction.
- Carbon elution via 10-t Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIL tailings using SO_2 / air process to produce weak acid dissociable (WAD) cyanide levels of less than 10 ppm.
- Tailings pumping to a tailings storage facility.
- Air and oxygen circuits.
- Water systems (potable water, raw water, gland seal water and process water).
- Sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control if and when required.
- Equipment selection based on suitability for the required duty, reliability, and ease of maintenance.
- Plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

1.18 Project Infrastructure

The project infrastructures are identified in the general site plan in Figure 1.1.

Figure 1.1: Site Plan View



The Project requires several infrastructures to support mining and processing operations as summarized below.

- Roads and Access:
  - Road access will be via a new the existing Puruni road with the addition of a planned 13 km laterite surfaced road. Replacement of two (2) existing bridges on the Puruni road are planned to improve traffic reliability.
  - Service roads of 20 km will connect the various infrastructures located on the property, notably the airstrip, explosives storage facility, tailings storage facilities, operations site and camp site.
  - Air access will be via a new, category 2, 850 m long airstrip that will be used during construction. This airstrip will be used for personnel, supplies, medical emergencies and exporting gold.
- Support infrastructure:
  - A gate and guardhouse will be located at the property entrance along the access road.

  - Permanent camp facility will have a 1,500-person capacity during construction with three types of camp modules. The camp will easily accommodate the operation' requirements. The permanent camp facility includes kitchen and dining area, recreation facilities, camp office and laundry facilities with associated water and sewage services.
  - A greenhouse and nursery.
- Mine infrastructures:
  - Facilities near the mine to allow easy access for heavy equipment while a safe traffic is ensured by segregation of light and heavy traffic.
  - Permanent mine maintenance facility will have ten (10) heavy duty bays with an additional five (5) light duty bays and two (2) maintenance / welding bays. The maintenance facility will include warehousing capacity, office space for the maintenance staff, tool crib and lube storage.
  - Wash bay for heavy duty vehicles will allow equipment to be washed prior to maintenance activities. The wash bay will be equipped with an oil-water separator.
  - Fuel storage will have the capacity to support site during 7 days of operation.
  - Explosive storage facility is designed for a capacity of 160 t of emulsion using 40 t skid mounted tanks, 18 t of explosives products in a magazine with another magazine for accessories. Storage capacity is sufficient for 30 days at peak consumption.
- Process Infrastructure:
  - Assay laboratory configured to process up to 350 samples per day for mine grade control, exploration and metallurgical samples.
  - Mill office with an area of 705 m^2.
  - Reagent Storage of 1,300 m^2.
- Waste Storage and Tailings Facility:
  - Covering an area of 370 ha, the waste storage facility has a maximum storage capacity of approximately 258 Mm^3.
  - Enclosed within a large valley inside the prospecting licence, the tailings storage facility will store conventional tailings. Existing water courses will require building a main and saddle dam, with the addition of the North dyke in the 4^{th} year of operation. The total embankment volume is estimated at 1.9 Mm^3 over the life of mine.

1.19 Market Study and Contracts

The Oko West Project will produce gold in doré bars. Oko West is authorized to export the gold doré bars from Guyana to an internationally recognized refiner.

The gold price was determined based on historical prices and the consensus of long-term estimates from banking analysts. The long-term gold price assumption used in the PEA is USD 1,950/oz Au, in line with analyst consensus commodity price forecasts published by CIBC Global Mining Group in August 2024 (CIBC, 2024).

G Mining Ventures has entered into a Master Service Agreement with G Mining Services, a Canadian engineering and mine development firm specializing in the mining sector, for the engineering and construction management of the Oko West Project. Transportation and refining contracts for gold doré bars will be negotiated and finalized during the construction phase of the project.

1.20 Environmental Studies, Permitting and Social or Community Impact

- A Project specific baseline study program was initiated in 2022, and information collected to date has helped build an understanding of the local and regional environmental and social conditions that have informed the PEA. Additional environmental and social data collection will need to be collected, and additional studies will be required to contribute to the next stages of Project design, the identification and mitigation of potential impacts on its receiving environment and the submission of an environmental impact assessment (EIA) for regulatory purposes. These studies will need to further incorporate ancillary project components such as power supply and site access roads.
- Data collected to date and reviewed by ERM has not identified any material issues of concern that would, at this stage, deem a project not viable in this location.
- The Project area has not been identified as a priority area of conservation interest by the Government of Guyana, nor does it fall in or near a Guyana Protected Area, a World Heritage Site, an International Union for Conservation of Nature Key Biodiversity Area, or an Alliance for Zero Extinction site. Despite the absence of designated protected areas nearby, the Project Area's ecological value highlights the importance of integrating conservation considerations into project planning and implementation.
- Reunion Gold has established strong relationships with some key stakeholders in the Project area, including formal titleholders, organizations, businesses, and/or individuals. Engagement to date has included long-term and informal relationship-building, public scoping consultation meetings in December 2023 as required for regulatory purposes, and increasingly focused efforts that align with

mine planning and development. Feedback and perspectives on the Project collected as part of stakeholder engagement thus far center on the following themes:

- Livelihood diversification and income-generating opportunities (especially for Indigenous women) with the mine and/or supported by the Oko West Project.
- Education, including programs to support ongoing school attendance and limit secondary school drop-out rates by adolescents who leave school to support their families by finding work in mining.
- Safety, health, and wellbeing, including concerns about impacts on Indigenous communities as a result of any potential increases in traffic and non-community visitors with mine development (including ancillary facilities).

1.21 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total USD 1.510 billion consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 6 Mtpa of fresh rock. Initial capital costs total USD 936.2M (including $100.3 million for contingency and net $28.8 million in pre-production revenue), which will be expended over a 32-month design, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations and also the mining underground development. Sustaining capital costs are estimated to be $537.5 million including indirect and do not include contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs a total of $36.6 million and includes a 20% contingency.

The capital cost estimate is according to AACEI standard Class 4 and is accurate to a -10% / +20% range. The base date of the CAPEX estimate is Q2-2024. The initial capital expenditure (“CAPEX”) duration is planned over a period of 32 months, assumed from May 2025 to end of December 2027. The initial CAPEX estimate is aligned with an owner-managed project delivery model. Expenditures are presented in US dollars.

The initial CAPEX is estimated at USD 936.2M. This amount includes net pre-production revenues of approximately USD 28.8M for 16 koz of gold recovered during commissioning.

The CAPEX includes a contingency of USD 100.3M, which is 12% of the total before contingency or 18% of the direct costs. The total hours for construction for the initial CAPEX phase are 8.5M hours.

Sustaining capital is required for various reasons. For the open pit mine, it is necessary to purchase new equipment and major components for the equipment. For the underground mine, sustaining capital includes all development work and the required equipment. Sustaining capital is required during operations for additional equipment purchases for the mine. Additional work is required for raising the main embankment of the tailings storage facility (TSF). The continued raising of the TSF will be completed by the mine operations team with fill material from the open pit mine. An increase in power capacity is also planned by adding generators. The sustaining capital is estimated at USD 537.5M. Table 1.2 shows the capital expenditures.

Table 1.2: Capital Expenditures Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
100 – Infrastructure	70,763	5,091	**75,854**
200 – Power and Electrical	118,243	25,598	**143,841**
300 – Water Management	16,318	11,267	**27,585**
400 – Surface Operations	45,952	-	**45,952**
500 – Mining	128,910	447,518	**576,428**
600 – Process Plant	190,010	22,000	**212,010**
700 – Construction Indirect	107,496	-	**107,496**
800 – General Services / Owner's Cost	111,432	-	**111,432**
900 – Pre-production, Start-up, Comm.	46,746	26,020	**72,766**
990 – Contingency	100,304	-	**100,304**
Total	**936,174**	**537,494**	**1,473,668**

No residual value was estimated at this stage of the study.

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at USD 36.6M OPEX and is summarized in Table 1.3. The OPEX includes mining, processing, general services and administration (G&A), transportation, refining and royalties. Power costs are separated from processing costs. The average OPEX is USD 853/oz Au or USD 51.15/t milled over the LOM. The

all-in-sustaining costs (AISC) which includes closure, reclamation and sustaining capital costs averages USD 986/oz Au or USD 59.13/t milled.

Table 1.3: Opex Costs

Operating Costs per Tonnes Milled		
Open Pit Mining Cost	t/milled	13.13
Underground Mining Cost	t/milled	10.76
Material Rehandling	t/milled	0.15
Processing Cost	t/milled	9.04
Power Cost	t/milled	5.93
General & Administration Cost	t/milled	4.14
Refining Cost	t/milled	0.48
Total Site Cost	**t/milled**	**43.62**
Royalty Cost	t/milled	7.53
Total OPEX Cost	**t/milled**	**51.15**
Sustaining Capital	t/milled	7.19
Closure Cost	t/milled	0.49
Land Payments	t/milled	0.30
All-In Sustaining Costs (AISC)	**t/milled**	**59.13**

Operating Costs per Payable Ounce		
Open Pit Mining Cost	t/oz	219
Underground Mining Cost	t/oz	179
Material Rehandling	t/oz	2
Processing Cost	t/oz	151
Power Cost	t/oz	99
General & Administration Cost	t/oz	69
Refining Cost	t/oz	8
Total Site Cost	**t/oz**	**728**
Royalty Cost	t/oz	126
Total OPEX Cost	**t/oz**	**853**
Sustaining Capital	t/oz	120
Closure Cost	t/oz	8
Land Payments	t/oz	5
All-In Sustaining Costs (AISC)	**t/oz**	**986**

1.22 Economic Analysis

The PEA is preliminary in nature and includes Inferred Mineral Resources, which are considered too geologically speculative to be categorized as Mineral Reserves with economic considerations. Therefore, there is no certainty that the PEA will be realized.

The economic analysis was conducted using a 5% discount rate. Cash flows were discounted from the start of construction, all costs before this period were considered as sunk costs.

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. A 5% discount rate is commonly used as the base case for gold projects.

A summary of the Project economic results is presented in Table 1.4. The total after-tax cash flow over the Project life is USD 2,583M and NPV 5% is USD 1,837M before tax and USD 1,367M after tax. The after-tax Project cash flow results in a 3.8-year payback period from the commencement of commercial operations with an IRR of 23.7% before tax and 20.8% after tax.

Table 1.4: Project Economic Results Summary

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	USD/oz	1,950
Mine Life (operation years)	Mt	12.7
OP Mill Feed Tonnage	Mt	61
UG Mill Feed Tonnage	Mt	15
Total Mineralized Material Mined	***Mt***	***75***
Total Waste Mined (OP and UG)		***367***
Total Tonnage Mined (OP and UG)		***443***
Strip Ratio	***waste: mineralized material***	***6.0***
Average Milling Throughput	***Mtpa***	***6***
Average Milling Throughput	***tpd***	***16,110***
Gold Head Grade	g/t	2.00
OP Head Grade	g/t	1.72
UG Head Grade	g/t	3.19
Contained Gold	koz	4,848
Average Gold Recovery (%)	%	92.8%
Total Gold Production	koz	4,500
Average Annual Gold Production	koz	353
Operating Costs (LOM Average)		
Open Pit Mining Cost	USD/t mined	$2.49
Underground Mining Cost	USD/t milled	$55.45
Processing Cost	USD/t milled	$14.97
G&A Cost	USD/t milled	$4.14
Total Site Costs	USD/t milled	$51.15
Total Site Costs	USD/oz	$728
Government Royalties	USD/oz	$126
Total Operating Cost	USD/oz	$853
AISC	**USD/oz**	**$986**
Capital Costs		
Capital Costs	*USD MM*	*$836*
Contingency	*USD MM*	*$100*
Total Capital Cost	USD MM	$936
Initial UG Capital Costs (Sustaining Capital)	USD MM	$124
OP and UG Sustaining Capital	USD MM	$413
Life of Mine Sustaining Capital	USD MM	$537
Closure Costs	USD MM	$37
Total Capital Costs	USD MM	$1,510
Financial Evaluation		
After-Tax NPV 5%	USD MM	$1,367
After-Tax IRR	%	21%
Payback	years	3.8

A sensitivity analysis was conducted on the base case pre-tax and after-tax Cash Flow, NPV (5%), IRR, and payback of the project using the following variables:

- Metal prices
- Operating cost
- Initial capital cost

Table 1.5 to Table 1.7 summarize pre-tax and after-tax sensitivity analysis.

Table 1.5: Gold Price Sensitivity

	Pre-Tax				After-Tax			
Gold Price (USD/oz)	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**
1,300	650	57	5.7%	10.4	650	57.3	5.7%	10.4
1,400	1,071	331	9.0%	8.3	905	245.9	8.1%	8.3
1,500	1,492	605	12.0%	6.9	1,158	427.1	10.5%	6.9
1,600	1,912	879	14.8%	5.9	1,475	638.5	13.0%	5.9
1,700	2,333	1,153	17.5%	5.1	1,792	848.5	15.4%	5.2
1,800	2,754	1,426	20.0%	4.4	2,108	1,057.0	17.6%	4.5
1,900	3,175	1,700	22.5%	3.8	2,425	1,264.0	19.8%	4.0
1,950	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
2,000	3,596	1,974	24.8%	3.4	2,742	1,470.6	21.8%	3.6
2,100	4,016	2,248	27.1%	3.0	3,059	1,677.1	23.7%	3.3
2,200	4,437	2,522	29.3%	2.0	3,375	1,883.5	25.6%	3.0
2,300	4,858	2,796	31.4%	2.0	3,692	2,090.0	27.4%	2.0
2,400	5,279	3,070	33.5%	2.0	4,009	2,296.4	29.2%	2.0
2,500	5,700	3,343	35.5%	2.0	4,326	2,502.8	30.9%	2.0
2,600	6,120	3,617	37.4%	2.0	4,642	2,708.9	32.5%	2.0

Table 1.6: Opex Sensitivity

	Pre-Tax				After-Tax			
OPEX	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**
80%	4,030	2,260	27.1%	3.1	3,067	1,685.0	23.7%	3.3
90%	3,708	2,049	25.4%	3.3	2,825	1,526.2	22.3%	3.6
Base Case	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
110%	3,063	1,626	21.8%	4.0	2,342	1,208.3	19.2%	4.2
120%	2,740	1,414	19.9%	4.4	2,100	1,049.1	17.6%	4.6

Table 1.7: Initial Capex Sensitivity

	Pre-Tax				After-Tax			
Capex	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**
80%	3,578	2,019	28.8%	2.0	2,728	1,511.0	25.2%	3.1
90%	3,482	1,928	26.0%	3.2	2,656	1,439.2	22.8%	3.5
Base Case	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
110%	3,289	1,746	21.7%	4.0	2,511	1,295.2	19.0%	4.2
120%	3,192	1,655	19.9%	4.4	2,439	1,222.8	17.5%	4.5

1.23 Adjacent Properties & Other Relevant Data and Information

According to the Guyana Geology and Mines Commission, the Oko West Prospecting Licence is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders, and one group of medium-scale mining and prospecting permits controlled by G2 Goldfields, namely the Oko Gold Project.

A mineral resource estimate was produced by Micon Ltd. And published on March 27, 2024, for the Oko Gold Project. The MRE considers the "Oko Main" deposit, as well as a newly defined area south of the Oko Main called the "Ghanie" Zone. Open pit and Underground Indicated Mineral Resources are estimated at 2,364 kt, grading 9.03 g/t Au for 686 koz Au for the Oko Main Zone and 3,344 kt, grading 2.20 g/t Au for 236 koz Au for the Ghanie Zone. Open pit and Underground Inferred Mineral Resources are estimated at

2,413 kt, grading 6.38 g/t Au for 495 koz Au for the Oko Main Zone and 12,216 kt, grading 1.54 g/t Au for 236 koz Au for the Ghanie Zone.

1.24 Other Relevant Data and Information

This Preliminary Economic Assessment (PEA) of the Oko West gold deposit is based on Indicated and Inferred Mineral Resources. Because of the inclusion of Inferred Resources, it is not applicable to determine Mineral Reserves at this stage of the project. Economic zones will be classified as mineralized material only.

1.25 Interpretation and Conclusions

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this PEA Report is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an open pit and underground mine thereat, including processing facilities and related infrastructures. This NI 43-101 Technical Report confirms the potential technical feasibility and potential economic viability based on an open pit mining and underground operation with average gold production at 353 koz per year over a 12.7-year life-of-mine ("LOM"). It is recommended to advance the Project to the Feasibility Study phase.

1.26 Recommendations

Following the results of the financial analysis of this Preliminary Economic Assessment (PEA), which demonstrates a positive project economics, GMS recommends that additional work be undertaken to initiate a Feasibility Study for the project.

Table 1.8 summarizes the proposed budget to advance the project to the Feasibility Study stage, considering the recommendations discussed in this section. The proposed Feasibility Study budget totals $31.7M.

Table 1.8: Cost Estimate Associated with Recommendations

Description	Amount (USD 000)
Infill Drilling and extensions	15,000
Resources & Mining Engineering	622
Metallurgical Testing Program	610
Geotechnical Studies for Mining	1,065
Geotechnical Drilling & Testing	1,500
Environment – Baseline Survey and ESIA	5,000
Project Engineering	5,000
Contingency at 10%	2,900
Total	**31,697**

2 INTRODUCTION

G Mining Ventures Corp. ("GMIN" or the "Company") mandated G Mining Services Inc. ("GMS") as the lead consultant along with other engineering consultants to prepare a Preliminary Economic Assessment ("PEA") under the supervision of the Qualified Persons "QPs" for the Oko West Project ("GYOW" or "Project") located in the Cuyuni-Mazaruni Mining Districts, Guyana.

GMIN is a gold mining company focusing on acquisition, exploration, and development of precious metal projects. Its flagship property is Tocantinzinho Gold Project located in the State of Pará, Brazil. The Company's common shares trade on the Toronto Stock Exchange (TSX: GMIN) and the OTC markets (OTCQX: GMINF).

On April 22, 2024, G Mining TZ Corp. (formerly G Mining Ventures Corp.) ("GMIN"), Reunion Gold Corporation ("Reunion Gold") and Greenheart Gold Inc. (formerly 15963982 Canada Inc.) ("Greenheart Gold", and collectively with GMIN and Reunion Gold, the "Parties"), entered into an arrangement agreement, which was subsequently amended effective June 7, 2024, setting forth the terms and conditions on which the Parties agreed to complete a plan of arrangement under Section 192 of the Canada Business Corporations Act (the "Arrangement"). Pursuant to the Arrangement, the successor issuer, New GMIN, has acquired (i) all of the issued and outstanding common shares in the capital of GMIN, and (ii) all of the issued and outstanding common shares in the capital of Reunion Gold, thereby combining the business of these two entities.

This Technical Report was prepared by GMS for GMIN. It was prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this report and the PEA is the evaluation of the technical feasibility and economic viability of the Project, notably the development of an open pit and underground mine, including processing facilities and related infrastructures. It is based, for the most part, on a drilling database dated 7th of February 2024, and supporting geological information. The report contains all technical information pertaining to the previous exploration of the Project, drilling methods, sampling and QA/QC protocols, data verification undertaken by the Qualified Person for this Technical Report, preliminary metallurgical test work program results and the Mineral Resource Estimate ("MRE") for the Project published on February 26, 2024 (RGD, 2024) and subsequent NI 43-101 technical report on April 11, 2024 (RGD, 2024). This Report provides operating and capital cost estimations and an economic analysis of the Project. The Oko West Project does not contain Mineral Reserves at this stage.

The intention of this Technical Report is to provide sufficient, clear and unambiguous technical and scientific information relating to the Project available at the effective date of the report. The qualified persons ("QP")

understand that a copy of this Report will be filed with the Canadian securities commissions and be publicly available.

2.1 Scope of Work

All sections of this Technical Report, excluding Section 20 – Environmental Studies, have been prepared by G Mining Services Inc., a mining consulting firm based out of Brossard, Québec, Canada. The QPs are entirely independent of the issuer (G Mining Ventures Corp.) as described in Section 1.5 of the NI 43-101 standard of disclosure for mineral projects. The QPs involved in the mandate do not hold an interest in the issuer or its related entities. The relationship between GMIN and GMS is solely professional, and GMS is being compensated based on a commercial-fee basis that is not contingent on the results presented in this Technical Report.

The QPs responsible for each section of the Technical Report are mentioned in Table 2.1.

Table 2.1: Summary of Qualified Persons

Qualified Person	Company	Title	Report Sections
Christian Beaulieu, P.Geo. (OGQ 01072)	Minéralis Services-Conseils Inc.	Vice President	1.4-1.12, 1.14, 1.23, 1.24, 1.25, 1.26, 2, 3, 4-12, 14, 23, 25.2, 26.1, 27.
Neil Lincoln, P.Eng. (PEO 100039153)	Lincoln Metallurgical Inc.	President and Consulting Metallurgist	1.13, 1.17, 13, 17, 21.1.8, 21.4.2, 25.4, 26.3, 26.4.
Alexandre Burelle, P.Eng. (OIQ 5019855)	Evomine Consulting Inc.	Mine Planning and Financial Analysis Consultant	1.15, 1.16, 1.21, 1.23, 1.24, 1.25, 1.26, 2, 3, 15, 16, 19, 21.1.7, 21.4.1, 22, 25.3, 26.2, 27.
Derek Chubb, P.Eng. (PEO 90328121)	Environmental Resources Management (ERM)	Senior Partner	1.20, 1.24, 1.25, 1.26, 2, 3, 20, 21.3, 25.6, 26.6, 27.
Paul Murphy, P.Eng. (OIQ 43320)	Independent Consultant	Project Manager	1.1-1.3, 1.18, 1.19, 1.23, 1.24, 1.25, 1.26, 2, 3, 18, 21.1, 21.2, 21.4.3, 21.4.4, 24, 25, 26.5, 27.

2.2 Sources of Information and Data

Unless otherwise stated, all the information and data pertaining to the Mineral Resource Estimate contained in the Report or used in its preparation has been provided by GMIN up to February 7, 2024. The abovenamed QPs have no reason to doubt the reliability of the information provided.

Sources of information include:

- Discussions with GMS and GMIN personnel.
- Inspection of the Oko West Project area, including drill collars, trenches, drill core, and ground conditions.
- Inspection of the principal and secondary laboratories.
- Drilling database received from independent, third-party database manager (gDat Solutions).
- Geological Interpretations, provided by GMIN.
- Exploration data, compiled and provided by GMIN.
- Preliminary metallurgical test works results.
- Technical and scientific reports by external consultants.
- All figures and tables cited using references in Section 27.

All currencies in this Report are expressed in United States dollars (USD) unless otherwise stated.

2.3 Site Visit

In accordance with NI 43-101 regulations, a current personal inspection was completed by the below mentioned QP to the Oko West Project as part of the data validation process.

Table 2.2: Site Visit Dates of Qualified Person

Qualified Person	Site Visit Scope	Dates
Christian Beaulieu, P.Geo.	Geology and Resources	April 18 to 19, 2023
Neil Lincoln, P.Eng.	Did not visit the site	N/A
Alexandre Burelle, P.Eng.	Did not visit the site	N/A
Derek Chubb, P.Eng.	Environmental and Social Impact	October 7 to 10, 2023
Paul Murphy, P.Eng.	Did not visit the site	N/A

The site visits covered the following aspects:

- Drill core inspection and visual comparison with assay values.
- Identification of drilling locations and validation of drill collar coordinates.
- Inspection and location of trenches.

- Audit of logging, sampling, and QA/QC protocols.

2.4 Effective Date

The PEA is derived using the Company's mineral resources estimate effective as at February 7, 2024 (the "MRE"). The effective date of the PEA is September 4, 2024.

The issue date of the Technical Report is October 11, 2024.

2.5 Sources of Information

2.5.1 Previous Technical Reports

NI 43-101 Technical Report Oko West Gold Project Cuyuni-Mazaruni Mining Districts, Guyana, G Mining Services, Effective date February 26, 2024, Issue Date: April 11, 2024.

NI 43-101 Technical Report Oko West Gold Project Cuyuni-Mazaruni Mining Districts, Guyana Property, G Mining Services, Effective Date: June 1, 2023, Issue Date: July 14, 2023.

NI 43-101 Technical Report Oko West Gold Project Cuyuni Mining District, Guyana, G Mining Services, Effective Date: December 31, 2022, Issue Date: March 20, 2023.

2.5.2 Agreements, Mineral Tenure, Surface Rights and Royalties

The issuer provided details regarding mining titles, royalty agreements, environmental liabilities, mineral agreement and permits. The QPs are not qualified to offer any legal opinion on property titles, ownership, or potential litigation.

2.6 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. All dollar figures quoted in this report refer to United States dollars (USD, US$ or $), unless otherwise noted. A list of the main abbreviations and terms used throughout this Technical Report is presented in Table 2.3. Unless otherwise specified, source for tables and figures is GMS, 2024.

Table 2.3: Table of Abbreviations

Abbreviations	Full Description
AA	Atomic-Absorption
Ag	Silver
As	Arsenic
Au	Gold
C	Carbon
CAD	Canadian Dollar
CIL	Carbon-in-leach
CoG	Cut-off Grade
CRM	Certified Reference Material
Cu	Copper
DD	Diamond Drilling
DGPS	Differential Global Positioning System
FA	Fire Assay
Fe	Iron
FS	Feasibility Study
g	Gram
gpt or g/t	Grams per tonne
g/L	Gram per litre
GMIN	G Mining Ventures Corp.
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)
GPS	Global Positioning System
ha	Hectares
h	Hour
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
ISO	International Organization for Standardization
kg	Kilograms
kg/t	Kilograms per tonne

Abbreviations	Full Description
km	Kilometre
km/h	Kilometre per hour
L	Litre
M	Mega or Millions (000,000s)
m	Metre
m^2	Square metre
m^3	Cubic metre
mg	Milligram
mm	Millimetre
ml	Millilitre
min	Minute
MRE	Mineral Resource Estimate
Mt	Million tonnes
Mtpa	Million tonnes per annum
NI 43-101	National Instruments 43-101- Canadian Standards of Disclosure for Mineral Projects
NQ	Drill Core Diameter (47.6 mm)
oz	Troy Ounce (31.10348 grams)
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Pb	Lead
ppb	Parts per Billion
ppm	Parts per Million
QP	Qualified Person
RC	Reverse Circulation
S	Sulphur
SD	Standard Deviation
Sec	Second (time)
t	Tonnes (1,000 kg) (metric ton)
t/y or tpy	Tonnes per year
t/d or tpd	Tonnes per day

Abbreviations	Full Description
t/h or tph	Tonnes per hour
t/m^3	Tonnes per cubic metre
USD	United States Dollar
wk	Week
XRF	X-ray Fluorescence
y	Year

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by GMS, under the supervision of the QPs, for GMIN. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of the preparation of this Report.
- Assumptions, conditions, and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by RGD, GMIN and other third-party sources.

The QPs of this Technical Report believe that the basic assumptions contained in the information indicated above are factual and accurate and that the interpretations are reasonable. The QPs of this Technical Report have, to the extent applicable, relied on this data and have no reason to believe that any material facts have been withheld. The QPs of this Technical Report have taken all appropriate steps, in their professional judgement, to ensure that the work, information, or advice from the above indicated information is sound and the QPs do not disclaim any responsibility for this Technical Report.

In preparing this Report, the QPs have fully relied upon certain work, opinions and statements of experts concerning environmental, legal, political or tax matters. The authors consider the reliance on other experts, as described in this section, as being reasonable based on their knowledge, experience and qualifications. The following companies and consultants have been retained by GMIN, to prepare various reports for the Project and have been relied upon in preparation of this Technical Report.

The companies and their involvements are listed below:

- GMS: overall Report and PEA coordination, property description and location, accessibility, history, geological setting, deposit type, exploration, drilling, sample preparation, data verification, mineral resource estimation, mining methods, project infrastructures, permitting and adjacent properties, operating costs, capital costs, economic analysis and project execution plan.
- ERM: a sustainability consultancy, conducted environmental and social baseline data collection and interpretations. ERM is relied upon for the relevant portions of Section 20 (Environmental Studies, Permitting and Social or Community Impact). They are also working on various baseline studies for the Project in the period of 2022 to 2024 and are currently preparing the ESIA for the Project.
- TEC3: an engineering and geotechnical consultancy, conducted surface geotechnical investigation campaigns for the Project and provided surface geotechnical guidelines and is relied upon for the relevant portions of Sections 18 (Project Infrastructures including Tailings Storage Facility and Waste Stockpile slope design).

- Alius Mine Consulting: an engineering and geotechnical consultancy, provided underground design guidelines. Their guidelines and recommendations are summarized in Section 16.
- NewFields Mining Design & Technical Services: an engineering and geotechnical consultancy, provided open pit slope design guidelines. Their guidelines and recommendations are summarized in Section 16.

The results and opinions expressed in this Technical Report are conditional upon the information provided by the Experts listed as being current, accurate and complete as of the effective date of the Technical Report. The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their “Certificates of Qualified Persons” submitted with this Technical Report to the Canadian Securities Administrators. Except for the purposes contemplated under provincial securities laws, any other use of this Technical Report by any third party is at the party’s sole risk.

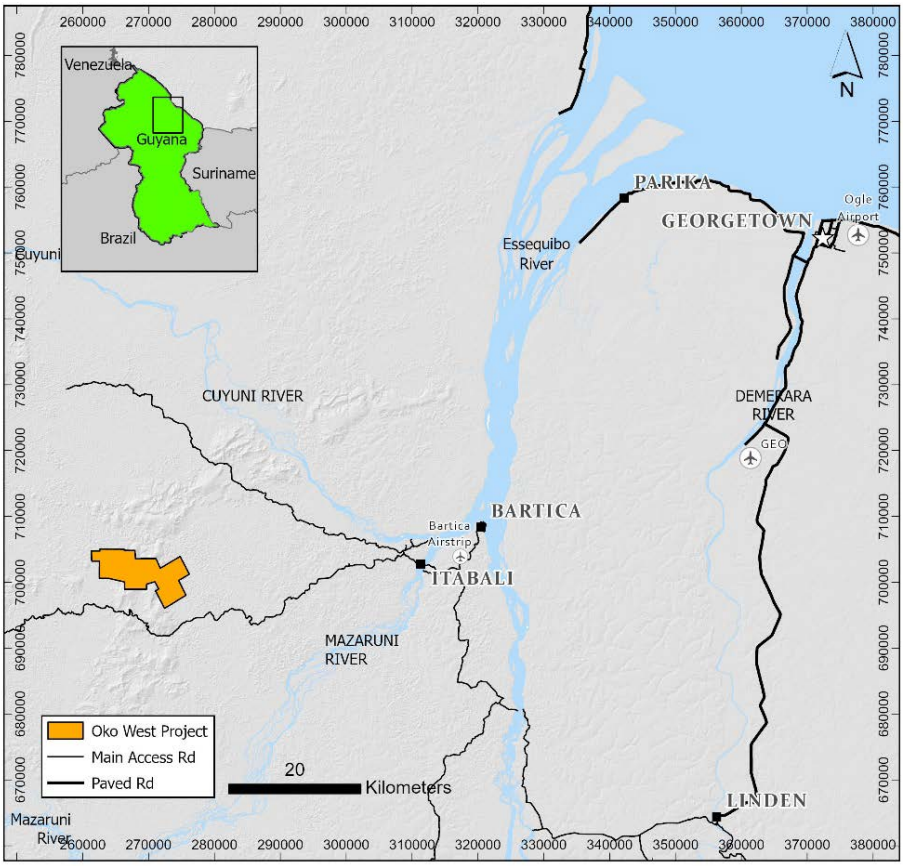
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Oko West Project straddles the Cuyuni-Mazaruni Mining Districts (administrative Region 7) in north-central Guyana, South America. The Project is located approximately 100 km southwest of Georgetown, the capital city of Guyana and approximately 70 km from Bartica, the capital city of Region 7 (Figure 4.1). Bartica is accessible by a 20-minute direct flight from the Ogle airport in Georgetown or by road and boat from Parika on the Essequibo River. There are regular boat services between Bartica and Parika.

The Project is accessible by the Puruni and Aremu laterite roads from the town of Itabali at the confluence of the Cuyuni and Mazaruni rivers (Figure 4.1). Several trails reach the Project area from these two (2) roads. The Project is also accessible by helicopter; the helicopter pad at the campsite is located at the coordinates 6°20´54.5” N and 59°03´13.8” W.

Figure 4.1: Oko West Project Location and Access

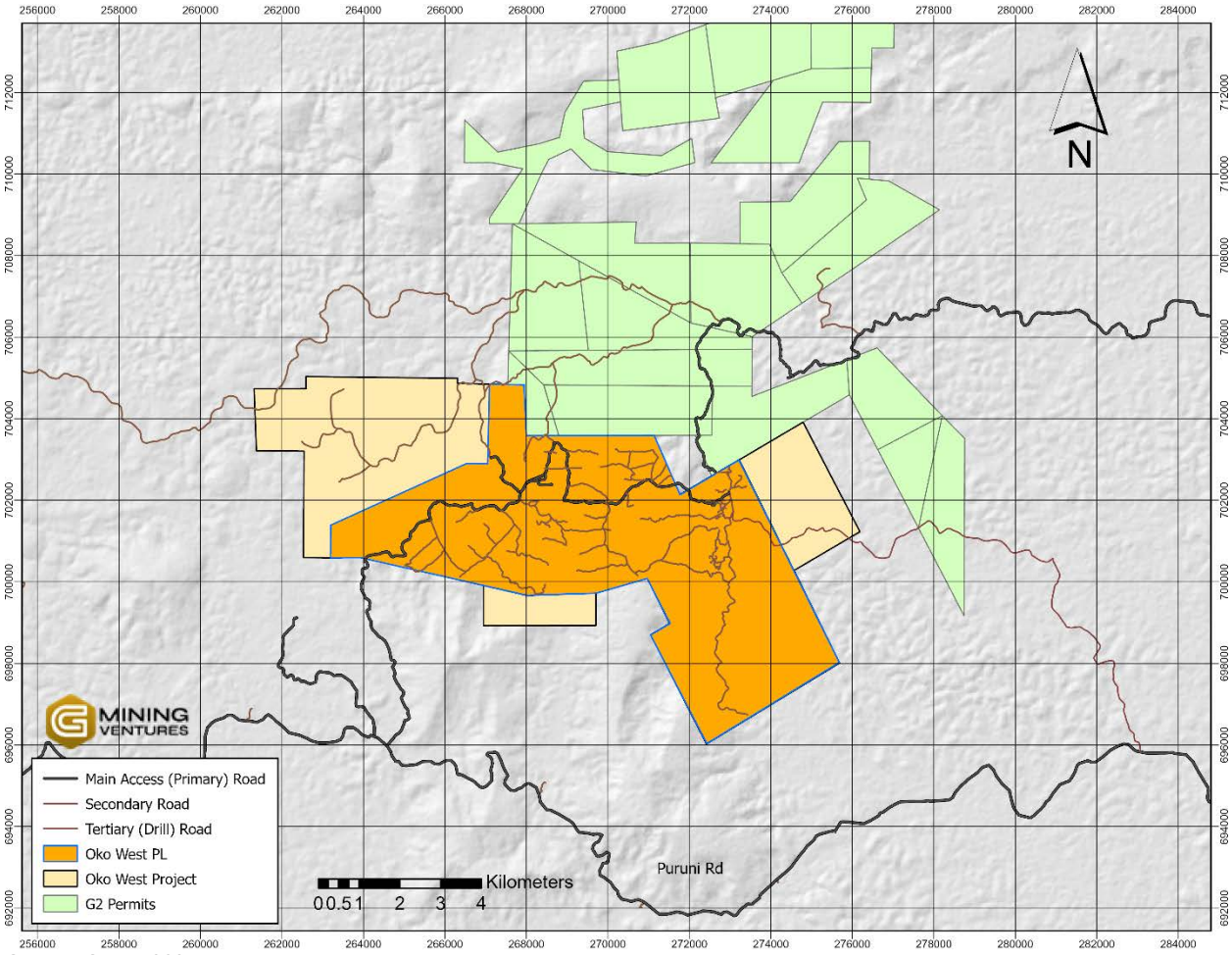


Source: GMIN, 2024

4.2 Property Description and Title

The Oko West Project comprises one Prospecting Licence (PL 004/2022) issued to Reunion Gold Inc., GMIN's 100%-owned Guyanese subsidiary, on September 23, 2022. The PL is valid for three (3) years and is renewable twice for a period of one (1) year each time. The PL has a surface area of approximately 10,890 acres (4,407 hectares) (Figure 4.2). In 2024, additional land has been added to the property as medium scale permits that will be converted to a Mining Licence.

Figure 4.2: Oko West Prospecting Licence



Source: GMIN, 2024

4.3 Legal Surveys

The Oko West Prospecting Licence boundaries and corners are described in PL 004/2022 by geographic coordinates (latitude and longitude). These boundaries and corners have not been surveyed in the field. Table 4.1 lists the coordinates for each PL corner and Figure 4.3 shows the location of each corner.

Figure 4.3: Outline of the Oko West Property

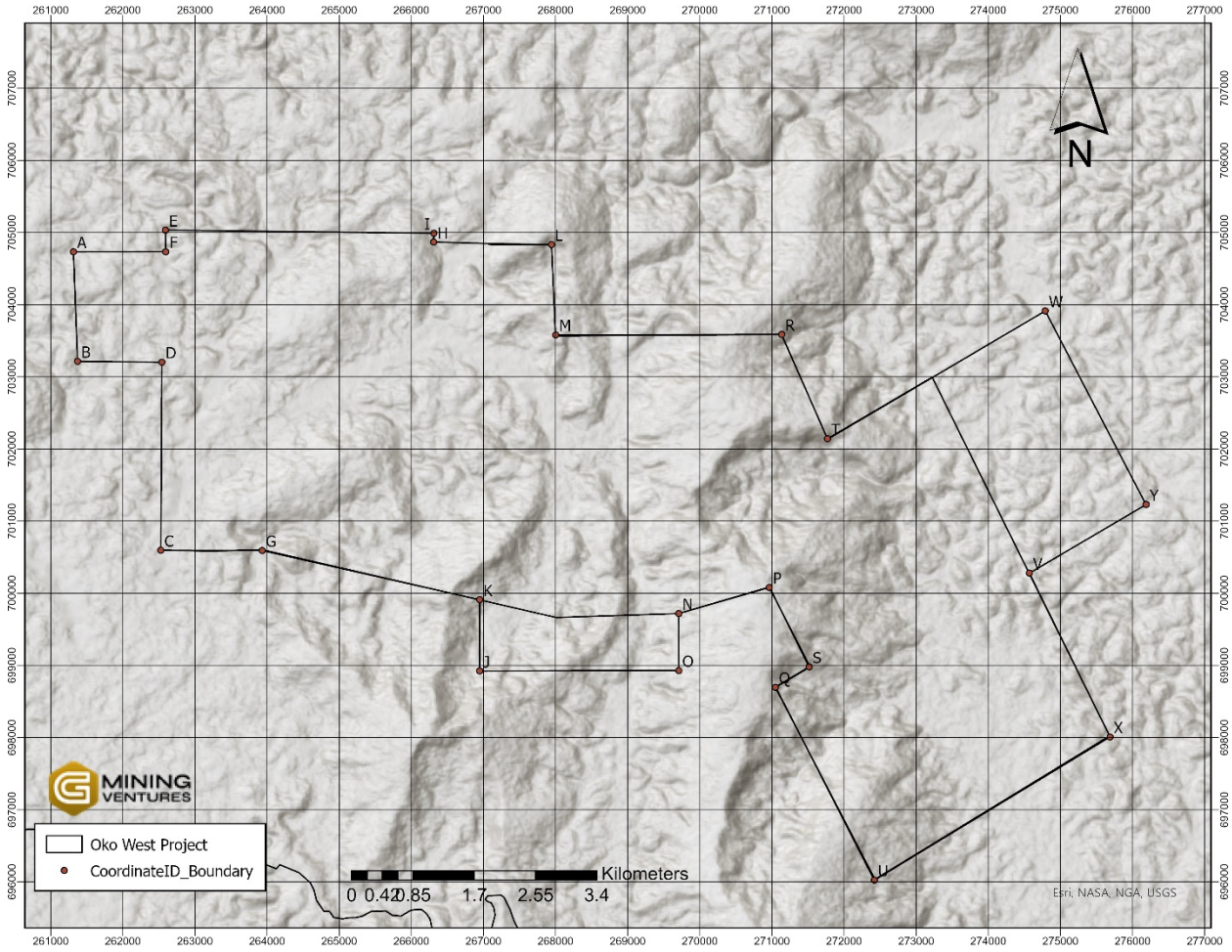


Table 4.1: Coordinates Defining the Oko West Prospecting Licence

Coordinate ID	PSAD 56 UTM Zone 21N	
	Easting	Northing
A	261316.37	704732.06
B	261373.11	703213.18
C	262528.15	700595.01
D	262542.06	703203.78
E	262590.27	705032.62
F	262596.22	704733.47
G	263932.81	700592.21
H	266309.54	704869.26
I	266312.47	704989.60
J	266948.18	698921.70
K	266948.64	699909.86
L	267943.57	704832.72
M	268000.69	703580.72
N	269707.80	699719.32
O	269709.19	698927.61
P	270964.23	700080.34
Q	271047.93	698696.07
R	271135.07	703587.56
S	271515.41	698979.27
T	271770.27	702142.03
U	272420.23	696022.49
V	274570.76	700279.01
W	274790.35	703913.59
X	275688.45	698010.06
Y	276187.11	701228.69

4.4 Oko West Mineral Tenure and Requirements

Mining in Guyana is managed by the Guyana Geology and Mines Commission (GGMC) under the Mining Act of 1989. The Act establishes that the State is the owner of all subsurface mineral rights in Guyana and authorizes the GGMC to manage these resources.

The Mining Act allows for three (3) scales of operation: small, medium, and large-scale permits or licences. Small-scale claims, medium-scale prospecting permits (PPMS) and medium-scale mining permits (MPMS) can only be issued to Guyanese citizens and partnerships, cooperatives or companies beneficially owned by Guyanese citizens. Foreign companies can enter into joint venture arrangements with local titleholders.

Large-scale prospecting licences ("PL") and medium-scale mining permits ("MP") can be issued to Guyanese citizens as well as Guyanese and foreign companies. PLs cover areas between 500 and 12,800 acres. The PL grants an exclusive right of occupation and exploration in the PL area. PLs are valid for a period of three (3) years with two (2) rights of renewal of one (1) year each.

GMIN's local subsidiary, as holder of PL 004/2022, has the exclusive right of occupation over the PL area (10,890 acres) for the purpose of exploring for gold. Rental rates are US$0.50 per acre for the first year; US$0.60 per acre for the second year, and US$1.00 for the third year. Pursuant to PL 004/2022, the local subsidiary must submit to the GGMC semi-annual and annual progress reports as well as annual audited financial statements. The work commitment during the first year of the PL is US$240,000, and a performance bond of 10% of that amount has been provided. Three (3) months prior to each anniversary date of the PL, a work program and budget for the following year must be presented for approval for the work to be undertaken during the following year.

In October 2022, GMIN's local subsidiary applied with the Environmental Protection Agency of Guyana ("EPA") to obtain an Environmental Authorization ("EA") to conduct exploration activities on the PL. EPA representatives visited the project and issued a letter confirming that they have no objection to Reunion continuing its exploration activities. This is discussed further in Section 20.

At any time during the term of the PL, the company may apply for a mining licence for any part or all of the PL area. In connection with the application, the following will need to be submitted: a technical and economic feasibility study, a mine plan, an Environmental Impact Statement, and an Environmental Management Plan. Mining licences are usually granted for twenty years or the life of the mine, whichever is shorter; renewals are possible. GMS and the QP has seen documentation relating to the grant of the PL and proof of payment of rental fees and the performance bond and is satisfied with its authenticity.

4.5 Oko West Project Ownership and Agreements

GMIN's local subsidiary, Reunion Gold Inc., is the 100% registered and beneficial owner of the Oko West Project and PL.

Pursuant to an agreement with a Guyanese citizen (Optionor) entered into in August 2018, Reunion Gold Inc. became entitled to conduct exploration activities over the area comprising nine (9) MPMS held by the Optionor. This area covers approximately 9,425 acres representing 86.5% of the total Oko West Project area. Under the agreement, the company had an option to acquire a 100% interest in the mineral rights over the area for an initial period of five (5) years. On February 23, 2023, Reunion Gold Inc. exercised the option.

Should the Oko West Project go into commercial production, the Optionor will be entitled to receive a contingent consideration of US$5.00 per ounce of gold produced from the 9,425 acres area. A first payment will be due within 30 days after commencement of commercial production on all ounces included in the feasibility study mining plan. An additional US$5.00 per ounce produced above the amount in the feasibility study will be payable quarterly thereafter.

In March 2024, Reunion Gold Inc. entered into an option agreement with the same Optionor for the Northwest extension mining permits, consisting of three (3) MPMS, adjacent to the PL, which is valid for a period of five (5) years with a possible 2-year extension. An initial cash payment has been made to exercise this option and is subject to the same payment terms consisting of a first payment that will be due within 30 days after commencement of commercial production on all ounces included in a feasibility study mining plan. An additional US$5.00 per ounce produced above the amount in the feasibility study will be payable quarterly thereafter. The NW extension provides for additional prospective exploration ground.

Reunion Gold Inc. also entered into an agreement to conduct exploration activities and acquire mineral rights over two (2) other MPMS held by another titleholder. The two (2) MPMS cover an area of approximately 1,465 acres. This option has been exercised and no additional consideration is due.

Reunion Gold Inc. entered into an agreement in August 2024 to purchase additional MPMS from a private group of individuals for the Eastern and Southern extensions to the Project PL at a cost of $4.3M yet to be paid at closing of the transaction and transfer of title. These two property extensions are to allow for a more optimal project layout facilitating the positioning of the waste rock dump and tailings storage facilities.

The combined surface area of the PL, medium scale mining permit agreements and NW option agreement result in a total surface area of 67 km^2.

4.6 Surface Rights

The Guyana Government holds the surface rights to the Prospecting Licence area. The PL allows the company to occupy the area.

4.7 Royalties and Other Encumbrances

The Government of Guyana imposes a royalty of 8% on gold produced from large-scale mines within the country. The Mineral Agreement signed between the Government of Guyana, the Guyana Geology and Mines Commission and Reunion Gold Inc. (GMIN's local subsidiary) establishes an 8% royalty applicable to open pit gold production and 3% from underground gold production. Also, refer to the contingent consideration described in Section 4.4 above. There are no other known royalties, back-in rights, payments, or encumbrances on the Project.

4.8 Environmental Liabilities

There are no known environmental liabilities associated with the Oko West Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

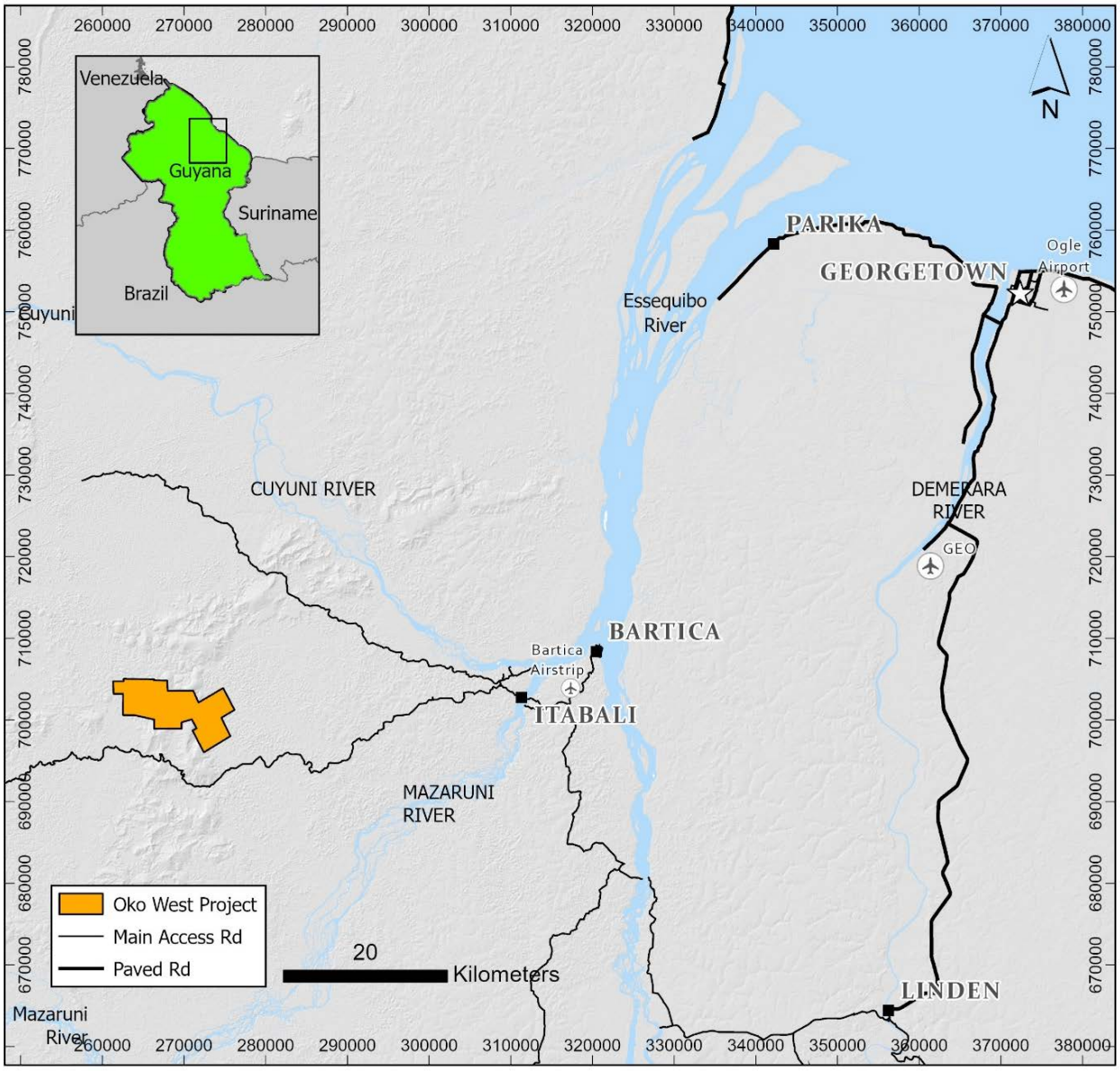
5.1 Accessibility and Roads

From the city of Georgetown, the project area can be accessed as follows (Figure 5.1 and Figure 5.2):

- By helicopter from the Ogle airport to the helicopter pad at the campsite – a flight of about 45 minutes.
- By fixed-wing plane from the Ogle airport to the Bartica airstrip (about a 20-minute flight) and from there by car, crossing the Mazaruni river by ferry to Itabali. An airstrip will be constructed at the Project (35 min).
- By car from Georgetown to the town of Parika at the mouth of the Essequibo River, (1-hour drive), and from there by speedboat to Itabali (1 hour and 20-minute ride), finishing with a car ride (1 hour and a half).
- By four-wheel drive vehicle from Georgetown to Linden on a paved road, from Linden to Itabali on laterite roads and two (2) ferry crossings. The trip from Georgetown to the project site using this route takes about eight (8) hours, depending on road conditions.

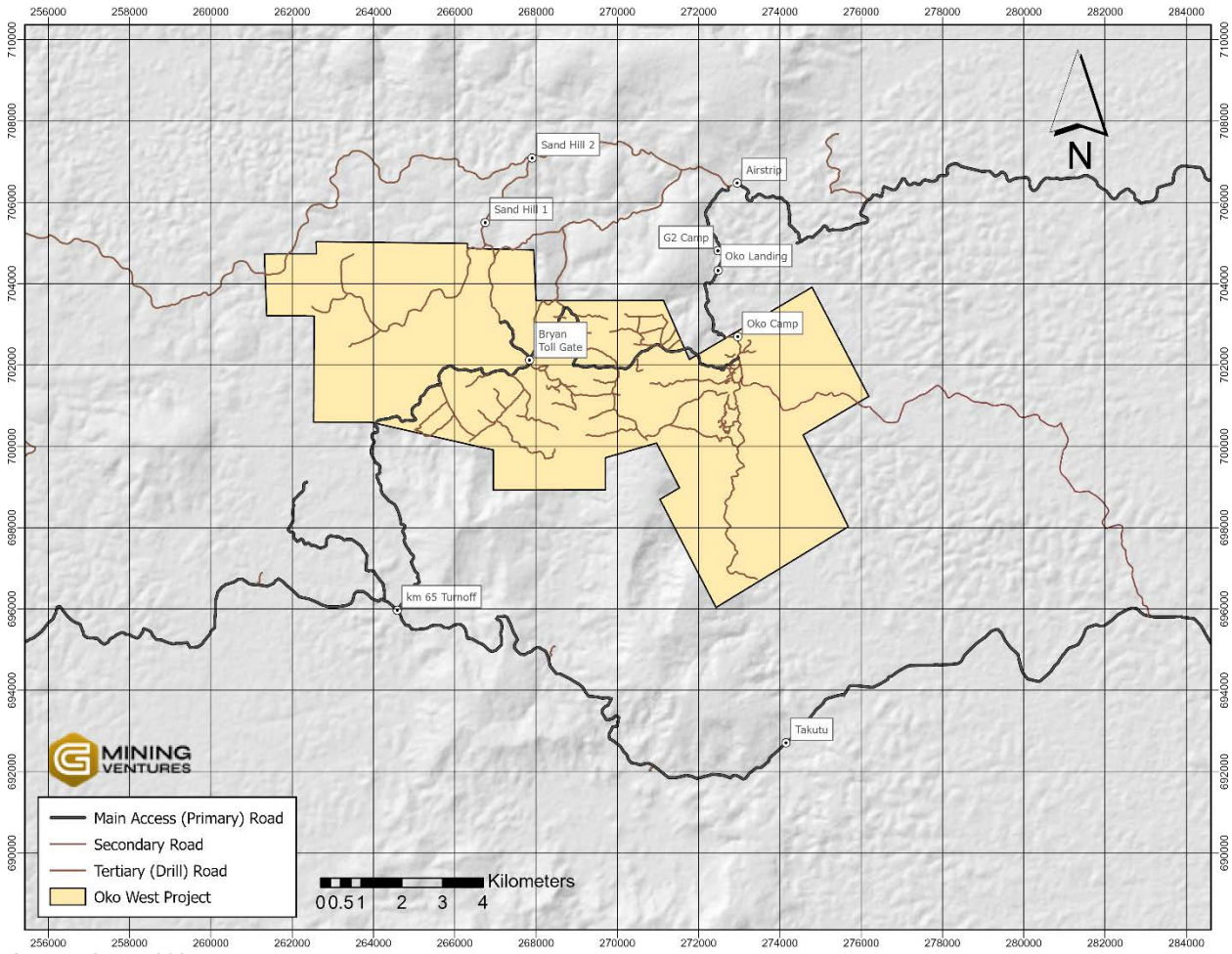
From Itabali to the project site, one can use the Puruni or the Aremu laterite roads, requiring four-wheel drive vehicles. The Project has been using the Puruni road due to its better condition and proximity to the site. The Puruni road has been recently maintained to service the Toroparu Project, designed for mine development and copper concentrate export. A light four-wheel drive vehicle can reach the Oko West Project campsite in about three (3) hours in good weather conditions. The vehicle will follow the Puruni road until Kilometre 65 at Lion Mountain and past the mining community of Takutu (with Digicel cell tower). From this turnoff, a private road leads to Bryan's camp, with a toll controlling access to passage (6 km) to the Oko West campsite (Figure 5.2). The Oko West campsite can also be reached by another trail running north, passing through the communities of Sand Hill and Oko Landing. Alternatively, the campsite can also be reached from the Puruni road via the Duckpond trail, an old logging road, although the road condition is poor and often unpassable.

Figure 5.1: Location Map of Major Access Roads and Topographic Features



Source: GMIN, 2024

Figure 5.2: Location Map of Local Roads in the Project Area



Source: GMIN, 2024

5.2 Climate

Guyana falls under the Köppen climate classification as "Af". The climate is equatorial and humid, with two (2) dry seasons, one (1) from approximately March to mid-April and the other from August to November. The dry seasons' onset and duration vary from year to year. The heaviest precipitation is expected in May and June. The average yearly temperature is about 26.5°C; in the interior regions of Guyana, one can expect typical daily highs around 34°C to 36°C and typical nighttime lows of 16°C to 18°C.

Reunion installed a weather station at the Oko West campsite, and this data will be available for the environmental baseline study of climate. The Guyana Hydrometeorological Service operates several regional meteorological stations with extensive historical climate data available.

The Oko West Project has operated throughout the year without any interruptions related to weather. Laterite road conditions deteriorate significantly during the rainy seasons and might cause transportation delays.

5.3 Local Resources

There are limited local resources available for the project:

- Aggregate made from the Bartica gneiss and with good specifications for concrete, currently produced at the Bartica quarry, 60 km to the east and on the other side of the Mazaruni river.
- High-quality hardwood is available for construction and foundation pylons.
- Duricrust is available for laterite road surfacing or capping.

5.4 Infrastructure

The region's infrastructure is underdeveloped, lacking power, roads, communications, and general services.

The city of Bartica (population of about 10,000), at the Essequibo, Mazaruni and Cuyuni rivers' confluence, is the primary hub for artisanal mining activity in northwest Guyana, often called the "gateway to the interior". Its main industrial activity is a quarry providing aggregate and boulders (for sea wall construction) for the entire western Guyana, shipped by barges to Parika. It houses the government administrative offices for Region 7 and has basic commercial, transportation and manufacturing facilities. It is linked to Parika by regular river "buses" transporting people and cargo. It has a hospital, an elementary-level school, and cell phone communication.

The town of Itabali, at the left margin of the Mazaruni river, is the gateway for the road transportation of goods and services to all the artisanal mining operations not reachable by a river. It is also where wood logs harvested in the region get loaded into barges for transportation to the sawmills in Parika. Itabali would be the logical place to install docking facilities to service an eventual mining operation at Oko West, including fuel depots.

5.4.1 Services Buildings and Ancillary Facilities

The Oko West Project is still in the exploration stage, and the campsite facilities currently service a team doing essentially mineral resources and exploratory drilling, as well as an environmental study on the property. Figure 5.3 illustrates the facilities on-site as of November 2023. The campsite has been upgraded

to accommodate more people and provide better facilities. Figure 5.4 shows a recent aerial view of the camp.

Figure 5.3: Overhead View of the Oko West Camp Showing Available Facilities



Oko-West Camp Layout

Q2 / 24

1-a, 1-b	Security Check point / Planned South Gate	13-a	Ablution on platform A	25	Power generating station (3 gensets)	38	Planned admin and engineering office
2	Security house / dormitory	14-a,14-b	Recreational center and laundry	26	Swingfog and pesticide shed	39	Ablution and septic toilets (C-North)
3-a, 3-b	Drilling contractor product storage and workshop	15-b	Female staff and visitors' house (septic toilet)	27	LV Parking	40	Planned UTV garage
4	Water well: 70m deep, pump at 46m, 14 m³/hour	15-a	Male visitors' house (septic toilet)	28	Old gym	41	Site clinic annex (D&A testing)
5	Water storage and treatment: 20,000 L	16-a,16-b	Canteen and kitchen	29	Latrines (2 outhouses)		
6	Diesel farm: 2 x 22,000 L	17-b	RC sample duplicates storage	30	Surface water well		
7	Helipad	18-a, 18-b	Core stores A and B	31-a, 31-b	Core store C and core laydown area		
8-a, b, c	RGI mech. workshop, carpentry, LV service bay	19-a, b, c	Core logging shacks	33	Recycled PET storage		
9	Dormitories C1–C5, B1–B7, A1-A2	20	Core cutting facility (3 saws)	34	Drilling products storage		
10	Medical / First Aid center	21, 22	Warehouse, temporary admin office	35	Senior staff and guest house (septic)		
11	Gym expansion	23	New gasoline dispensing station	36-a, 36-b	Ablution and septic toilets		
12	Exploration office and IT / satcom	24-a, 24-b	ATV garage	37-a, 37-b	Recycling area; Garbage pits		

Source: GMIN, 2024; North to the right of the image.

Figure 5.4: Drone View of the Oko West Camp, Looking South



Source: GMIN, 2023.

5.4.2 Power Supply and Distribution

There is no available grid electrical power in the region. The entire Guyana power system currently runs on heavy fuel or diesel thermal plants installed along the coast and at Linden and Bartica. There are no power lines or substations in the project vicinity.

The nearest power-generating project being considered is Amaila falls on the Kuribrong river in central Guyana. This hydropower project envisages generating 165 MW and sending the electricity via a new power line to Linden, where it would be distributed. The Guyana Government has been seeking the engagement of a foreign company to build the project on the basis of a “build-own-operate-transfer” contract. If this hydropower plant goes ahead, the Oko West Project could procure power at Linden and build a transmission line to the mine site (about 135 km along existing roads).

ExxonMobil is on track to deliver natural gas from its offshore Guyana operations to the mainland by the end of 2024. This Gas-to-Energy (GtE) Project aims to construct an Integrated Natural Gas Liquid (NGL) plant and a 300-megawatt (MW) combined cycle power plant at Wales, West Bank Demerara (WBD). The project will utilize natural gas from the Stabroek Block’s Liza Phase 1 and 2 Projects. The pipeline, which is 12 inches in diameter, will cover approximately 200 km from offshore to onshore facilities. It is expected to transport about 50 million standard cubic feet per day (mscfpd) of dry gas, with a maximum capacity of

120 mscfpd. There are plans to produce Liquefied Natural Gas (LNG) for export which could be a source of fuel for the Oko West power plant.

5.4.3 55Water

The project area lies in the water divide between the Cuyuni river to the north and the Mazaruni river to the south. Alternatively, the project could harvest water from local creeks and store it in a reservoir. The tailings storage facility will be constructed and will serve to capture water and build up an initial starter reservoir to feed the process plant and will recycle water back during operations. The ongoing environmental baseline information will provide essential information on the local hydrology.

5.5 Physiography

5.5.1 Vegetation

The entire project region is covered by what Granville (1988) called an “upland moist forest”. This equatorial evergreen forest type is the most common and floristically richest forest formation of the Guianas, found extensively on undulating terrain and well-drained ferralic and sandy soils. Their general characteristics are the presence of a high and dense canopy at 20-45 m and emergent trees up to 50-60 m. They have enormous species diversity, between 100 and 300 tree species per hectare, showing a correlation between higher rainfall and increasing diversity. The forests on higher elevations and over duricrust appear less species-rich and have a lower canopy, many lianas, and a scrubby undergrowth. The ongoing baseline environmental study will provide essential information on the area's flora.

5.5.2 Topography

The Oko West permit area is underlain by a belt of greenstone rocks composed mostly of mafic volcanics. These rocks' weathering released iron, forming pisolitic concretions near the paleosurfaces. Once the water table stopped reaching the upper levels of the lateritic profile, these pisolitic concretions coalesced, forming concrete-hard duricrusts. This weathering phenomenon explains the local topography, where duricrusts hold higher elevations. The highest point in the project area, at 340 m, corresponds to a duricrust surface on the access road to the campsite (Figure 5.2). The eastern flank of this north-south ridge hosts the Oko West gold mineralization, with an average elevation of 130 m. The terrain drops to the east, reaching the lowest point along Kairuni creek, at 66 m. The campsite has an average elevation of about 80 m. Further east, the topography is characterized by rolling hills over the Oko granitic pluton with an average elevation of about 100 m.

6 HISTORY

6.1 Prior and Current Ownership

Reunion Gold Inc. is a Guyanese subsidiary wholly owned by G Mining Ventures Corp (GMIN) following the merger with Reunion Gold Corporation (RGD).

G Mining Ventures Corp. (“GMIN”) (TSX: GMIN) (OTCQX: GMINF) and Reunion Gold Corporation (“RGD”) (TSXV:RGD) (OTCQX:RGDFF) announced the entering into a definitive agreement to combine the two companies on April 22, 2024. The merger via a plan of arrangement was overwhelmingly approved by GMIN shareholders as well as Reunion Gold securityholders at their respective annual general and special meetings held on July 9, 2024, and was subsequently approved by the Ontario Superior Court of Justice (Commercial List) on July 11, 2024.

Reunion Gold Inc. currently holds a 100% interest in the Oko West Project and Prospecting Licence (PL). This PL was issued on September 23, 2022, by the Guyana Geology and Mines Commission (GGMC), following the relinquishment of 11 medium-scale mining permits held by two (2) Guyanese entrepreneurs with whom Reunion had signed option agreements (see Section 4 for more details). These entrepreneurs had acquired some of these permits from other local parties and registered them to their names. The previous permit holders produced an unknown quantity of gold in an artisanal basis, primarily by alluvial mining.

The Oko West Project was part of the initial Strategic Alliance between the Company and Barrick Gold Corporation from February 2019 until January 2020, when Barrick Gold communicated to Reunion its decision to exclude Oko West Project from the Alliance.

6.2 Exploration History

6.2.1 Historical Exploration

The discovery of gold in the region dates back to the end of the 19th century by artisanal miners. In 1935, Dr. Grantham, from the British geological survey, described mining activity in the region. British Guiana Goldfields Ltd. recognized the area's potential and extended the road from Kartabu Puruni road to Bird Page on the Oko River, establishing a trading outpost in 1938. In 1939, it drilled some 400 Banka holes in the Oko River area and 110 holes in the Kairuni river area, a tributary of the Oko River. Despite encouraging results, they abandoned the Oko area and focused on the more profitable veins at the Aremu mine area to the northwest (Narain, 1985).

In the mid-60s, Aero Mineral Surveys Ltd., a Canadian company funded by the United Nations, flew a regional magnetic and radiometric survey over northern Guyana. From 1966 to 1979, McDonald, Schielly and Garson, working for the British Geological Survey, conducted regional field mapping and produced compilation maps of the region. The geophysical data was a major impetus for further work, which included an investigation by Schielly in 1969 of an area with magnetic anomalies on the northern border of the project area. He produced two (2) separate maps, one showing geology and structure, and one interpretative map assessing the regional stress fields (Narain, 1985).

In 2002, the Guyana Geology and Mines Commission (GGMC) conducted the Lower Puruni Regional Geochemistry Programme, which covered the project area, identifying gold and molybdenum anomalies from stream sediment samples. Between 2010 and 2015, extensive alluvial and eluvial mining was done in the region, where local miners mined several gold-rich quartz veins at Crusher Hill, north of the Oko West Project area. There is still artisanal gold mining activity in the region.

In 2016, Sandy Lake Gold Inc. collected grab samples at Crusher Hill, a primary prospect north of Oko West, reporting high gold grades in shaft stockpiles associated with quartz and quartz-carbonate veins in narrow mineralized zones (Ilieva, 2018). This reconnaissance became the basis for Sandy Lake Gold to enter into an option agreement with Michael Vieira in 2018 for this area. Sandy Lake Gold was renamed to G2 Goldfields Inc. in 2019 and continues to explore its Oko Main zone.

There is no record of the amount of gold production from artisanal mining within the Oko West Project area.

6.2.2 Reunion Gold Exploration

In 2018, the local permit holder, Mr. Bryan Stephens, introduced Reunion Inc. to a number of claims under his control that resulting in a site visit to a mining pit to inspect outcrops and alluvial waste piles. Grab samples were collected on this initial visit and included samples BS-02, BS-03 and BS-04, which assayed 2.40, 6.00 and 45.84 g/t Au, respectively. This visit and the interpretation of a favourable local geological context prompted the option of two (2) groups of medium-scale mining permits held by Mr. Bryan Stephens, Oko West and Oko East. Reunion later relinquished the Oko East block from the agreement and proceeded to explore the Oko West claim block as described elsewhere in this report.

6.3 Historical Drilling

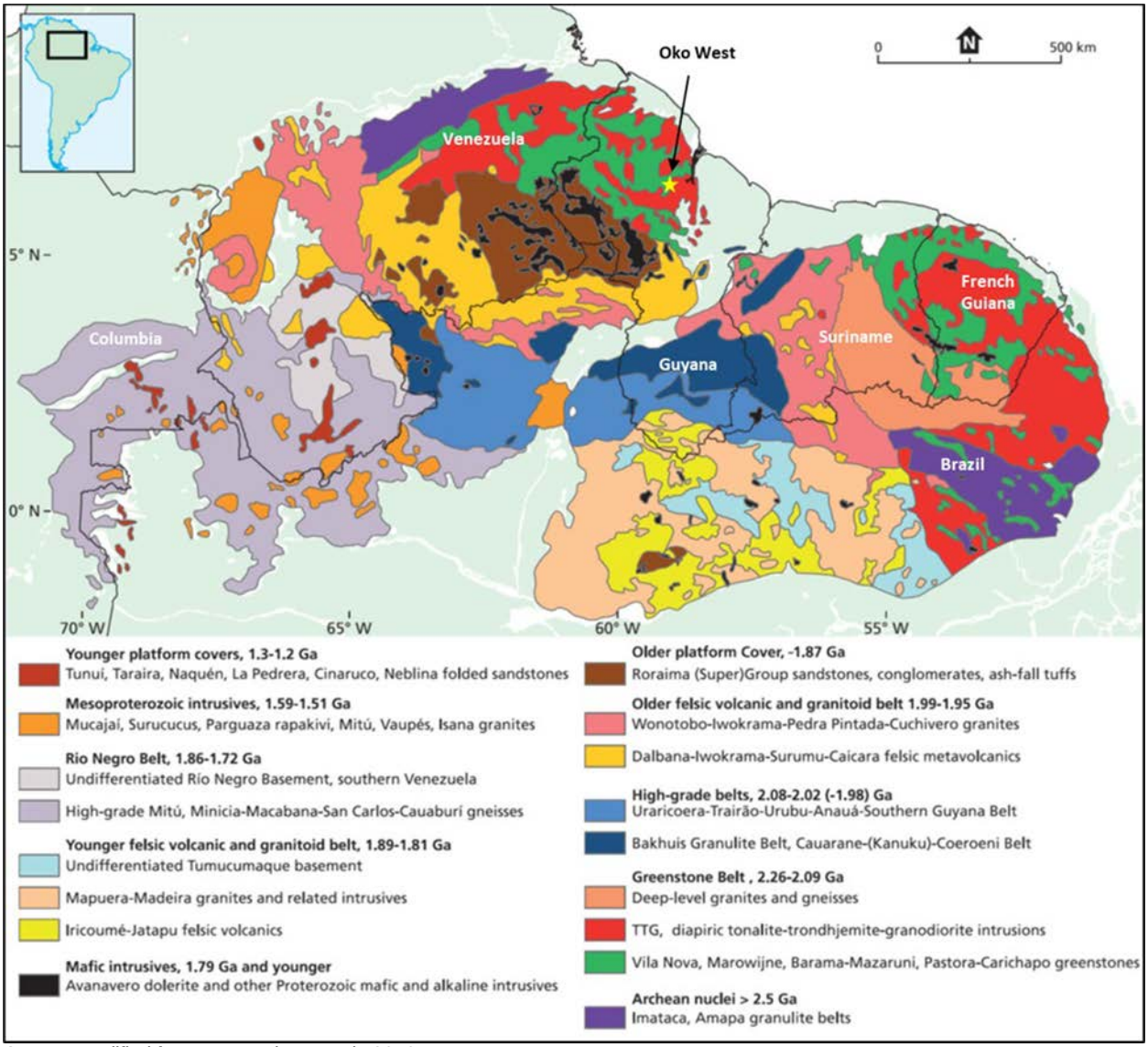
There is no known historical drilling completed within the project boundaries.

7 GEOLOGICAL HISTORY AND MINERALIZATION

7.1 Regional Geology

The Oko West Project is located approximately 110 km west-southwest of Georgetown, Guyana, in the Trans-Amazonian province of the Guiana Shield (Figure 7.1).

Figure 7.1: Simplified Geological Map of the Guiana Shield



Source: modified from Kroonenberg et al., 2016

The Guiana Shield corresponds to the northeastern portion of the Amazonian Craton. With a total area of 900,000 km^2, it covers eastern Venezuela, Guyana, Suriname, French Guiana, the northern end of Brazil, and easternmost Colombia (Daoust 2016; Tedeschi et al. 2020). The Guiana Shield is mainly composed of

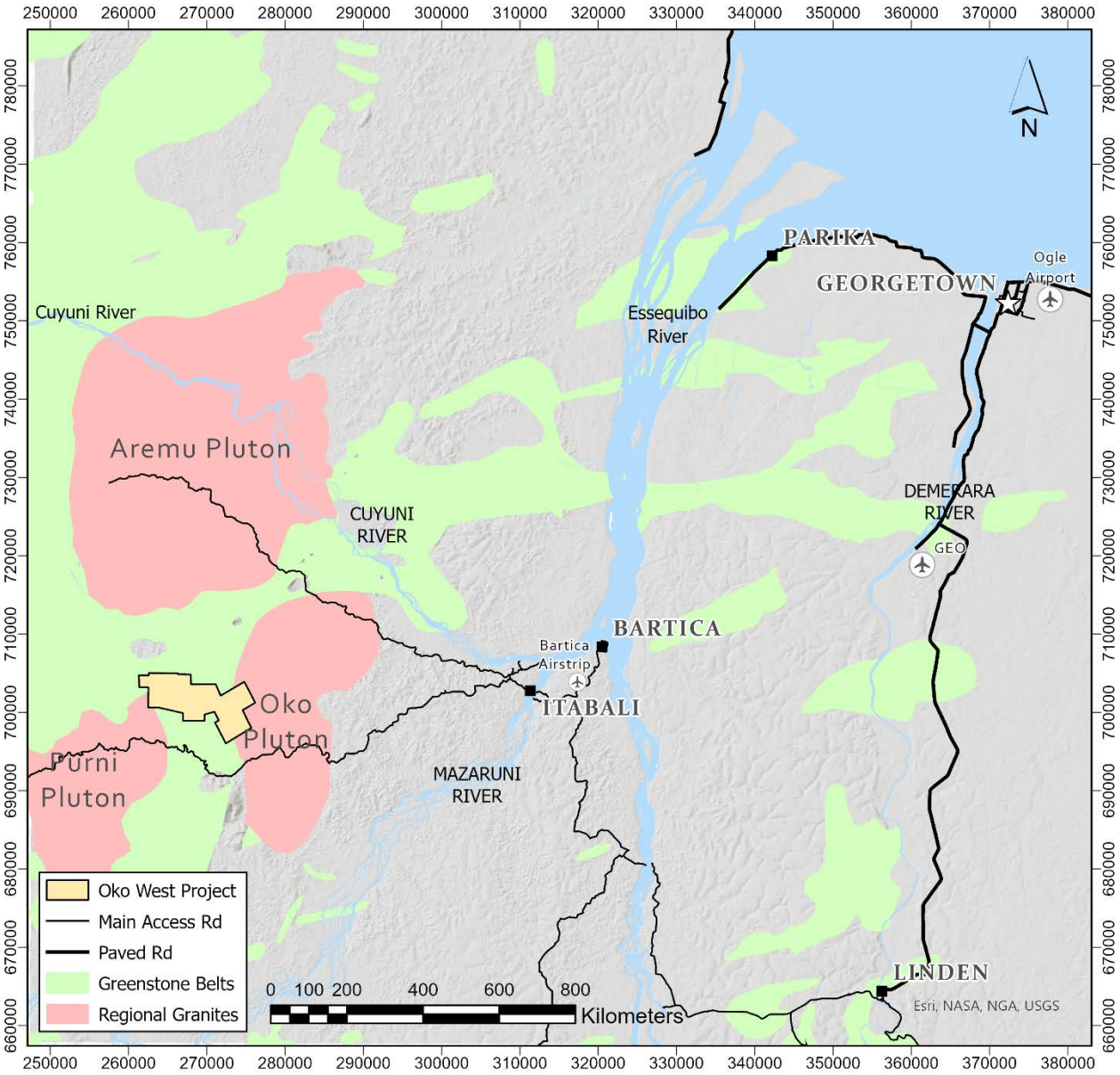
Paleoproterozoic rocks accreted during the Trans-amazonian orogeny (2.2 - 2.0 Ga) and affected by tectonic, metamorphic, and intrusive events (Vanderhaeghe et al. 1998; Milési et al. 2003). Small Archean relics are preserved in eastern Venezuela (Imataca Complex) and northern Brazil (Amapá) (Tassinari et al., 2004; Tedeschi et al., 2020).

The Trans-Amazonian Province is composed of large Rhyacian (2.20 - 2.05 Ga) granite-greenstone belts, including volcano-sedimentary rocks, metamorphosed to greenschist facies, intrusive granitoids, and TTG (tonalite-trondhjemite-granodiorite) gneisses (Vanderhaeghe et al., 1998; Santos et al., 2000; Tedeschi et al. 2018, Tedeschi et al., 2020). In Guyana, the greenstone belts are described from deepest to shallowest as basalt ± ultramafic rocks, intermediate to felsic volcanic rocks, and finally, tuffs and turbiditic sedimentary rocks (Gibbs, 1980; Tedeschi et al., 2020). They host multiple gold deposits; however, little is known about the relationship between gold mineralization, magmatism, and deformation (Tedeschi et al., 2020).

Two (2) major tectonic events, D1 and D2, which took place during the Trans-Amazonian orogeny, have been documented (Ledru et al., 1991; Gibbs and Barron, 1993; Vanderhaeghe et al., 1998; Delor et al., 2003a). The major Trans-Amazonian orogeny begins between 2.26 and 2.20 Ga, forming a juvenile oceanic crust from tholeiitic magmatism. It continues between 2.18 and 2.13 Ga with the first major tectonic event D1, associated with the N-S convergence of the Archean African and Amazonian cratons. Between 2.11 and 2.08 Ga, the convergence of the Archean African and Amazonian cratons evolves towards an oblique NE-SW convergence, with the closure of the volcanic arc basins (Delor et al., 2003b). This sinistral strike-slip regime, defined as the D2a event, is marked by granitic magmatism, minor mafic intrusions, and regional greenschist metamorphism. It also led to the folding of the volcano-sedimentary formations of the greenstone belts and the development of thrust faulting and EW to NW-SE regional sinistral strike-slip shear zones (Vanderhaeghe et al., 1998; Delor et al., 2003a; Tedeschi et al., 2020). The D2b event occurred between 2.07 and 2.06 Ga (Delor et al., 2003a). It is marked by dextral reactivation of WNW ESE strike-slip shear zones and significant crustal thinning, leading to mantle rise and regional high-temperature metamorphism (Delor et al., 2003a; Tedeschi et al., 2020) and peraluminous intrusions.

The D1 intrusive event in the project region is likely represented by the intrusion of three (3) plutons: Oko, adjacent to Oko West, Aremu to the north, and Puruni to the southwest (Figure 7.2). The Barama-Mazaruni Supergroup volcano-sedimentary rocks (2.12 Ga) caught between these plutons were thoroughly deformed, and local structures host several gold occurrences.

Figure 7.2: Schematic Geology Map of North-Central Guyana

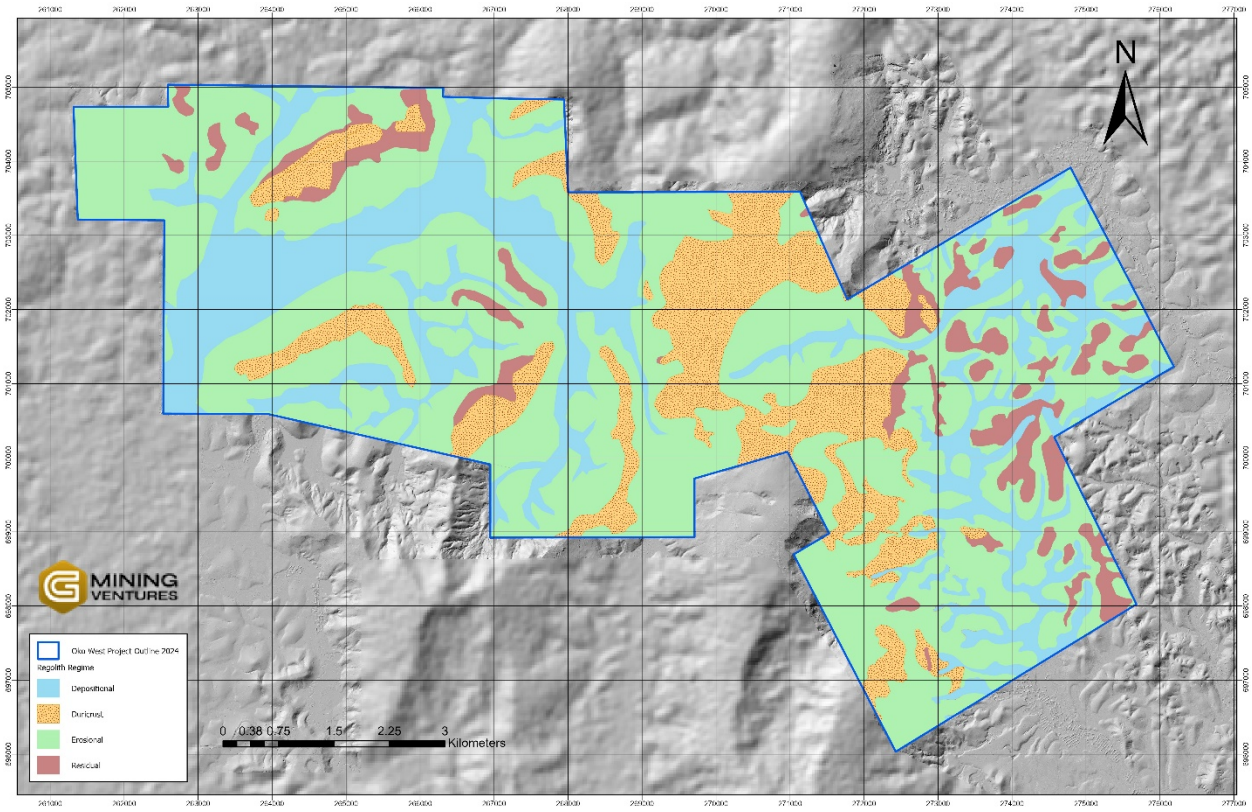


Source: GMIN 2024

7.2 Property Geology

The Oko West Project straddles the north-south striking contact between rocks of the Barama-Mazaruni Supergroup greenstone belt to the west and a granitoid pluton to the east (the Oko pluton). The Barama--Mazaruni Supergroup sequence comprises mafic volcanic flows, volcaniclastics, and siliciclastic and carbonaceous sediments (Figure 7.3). The following sections describe each of these geological units and their role in gold mineralization.

Figure 7.3: Oko West Permit Simplified Geology and Geomorphology Map



Source: GMIN, 2024

7.2.1 Regolith

Long chemical weathering typical of humid equatorial paleoclimate produced a thick lateritic profile down to a depth below 100 m from the surface. This profile is typically composed of a veneer of pisolitic colluvium and latosols overlaying a massive clay zone, which pass into a mottled zone and then saprolite / saprock before reaching unweathered rocks at depth.

At the topographic highs in the centre of the project area (Figure 7.3), the profile is different; characterized by a superficial duricrust, either in situ or broken, passing into a massive clay zone and then the mottled zone. This duricrust, remains of a peneplain, can be several metres thick and form a protective cap, slowing the erosion of the underlying rock. To the south and at the Puruni road, this ridge is called Lion Mountain. Not coincidentally, the duricrust developed only above areas underlain by mafic volcanic rocks, rich in iron, a key ingredient to form duricrust through water table movement and bedrock leaching.

Gold mineralization at Oko West rocks outcrop on the eastern flank of a north-south trending ridge, covered by a veneer of pisolitic colluvium up to two (2) metres thick and blocks of duricrust that survived erosion. The clay layer and mottled zone below vary in thickness up to three (3) metres and do not allow the

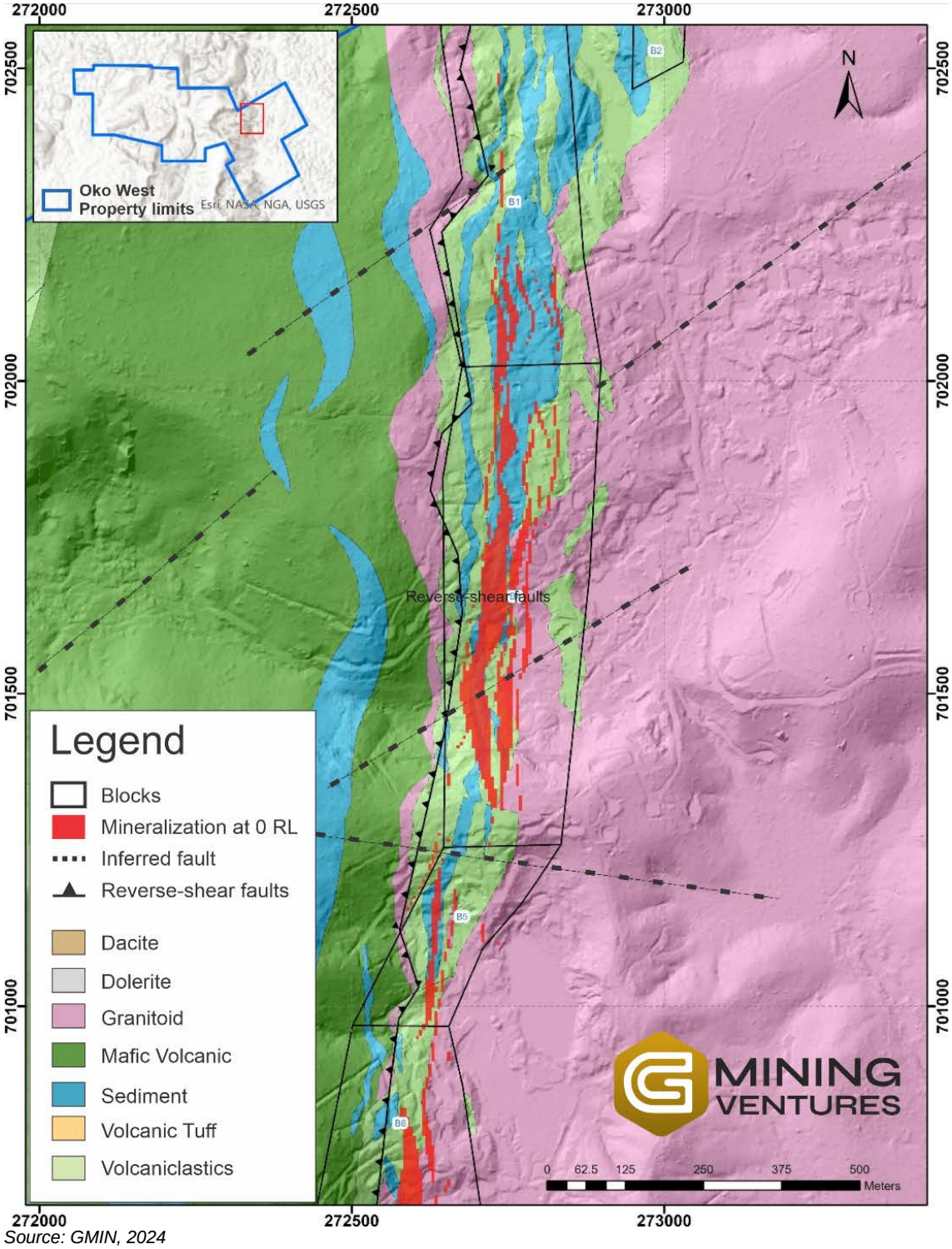
recognition of lithologies, although they are considered to be in situ. The saprolite allows the recognition of lithologies and geological structures. The transition zone or "saprock" to unweathered rocks exhibits partially preserved mineralogy and structural features typical of the protolith. It is noticeable that weathering is deeper overlying gold mineralization, facilitated by the abundance of easily dissolved carbonate minerals and faults and shear zones that allowed weathering to reach greater depths. Overall, the lateritic or weathered profile ranges from 16 m to 118 m over the area (Exploration Blocks 1, 4 and 5), and averages 65 m in thickness. The thickness of the saprolite zone averages 43 m, and the transition zone averages 22 m. The mineralized saprolite and transition zones are entirely preserved and untouched by artisanal miners over the strike length of gold mineralization. Figure 7.4 presents a typical section through outcropping gold mineralization in the saprolite. A geological map outlining the various zones at Kairuni is presented in Figure 7.5.

Figure 7.4: Outcrop of Carbonaceous and Volcanoclastic Sediments Saprolite (Exploration Block 5)



Source: Reunion, 2022. Photo by C. Bertoni

Figure 7.5: Oko West Kairuni Zone Geology and Mineralization Map with Exploration Blocks



Source: GMIN, 2024

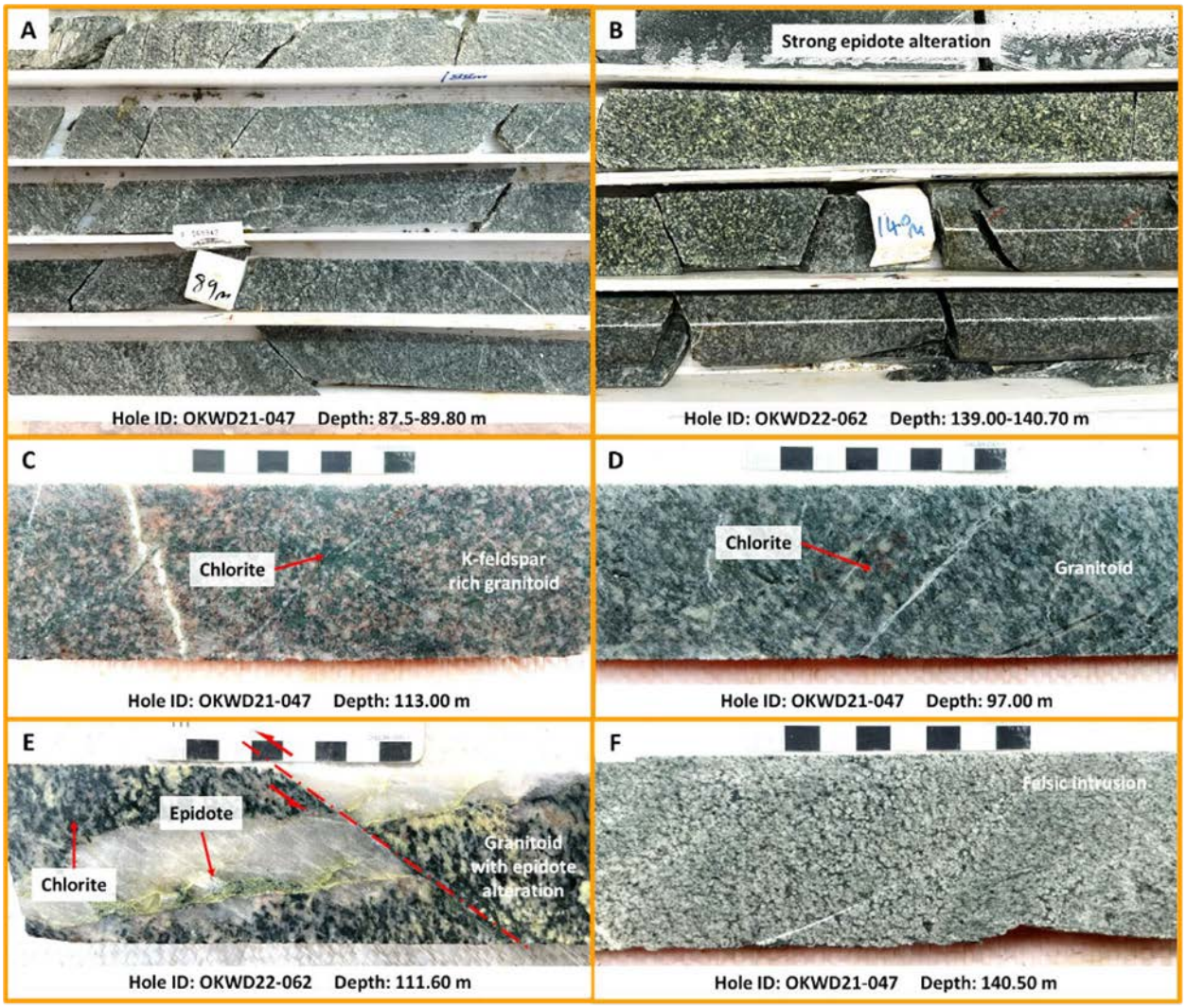
7.2.2 Granitoids

The pluton flanking gold mineralization to the east and acting as its hanging wall does not appear to belong to the Bartica Gneiss complex (Figure 7.5). It is interpreted as a later intrusion of unknown dimensions, being less foliated and lacking evidence of partial melting, hereby called the "Oko pluton" Tedeschi (2022) dated a sample from the "Bryan pit" area at 2,107 ± 6 Ma. The rock is primarily coarse-grained and slightly foliated, described petrographically by Thompson (2022) as a metamorphosed quartz-monzodiorite and granodiorite (Figure 7.6). At the deposit scale, its contact with the sedimentary sequence is sharp and locally sheared. There is no evidence of pre- or syn-orogenic contact metamorphism. The locally fractured and sheared zones near its contact can be mineralized, but do not represent significant thicknesses.

The volcano-sedimentary sequence shows a strongly sheared contact with a footwall granitoid (see cross-section in Figure 7.7) intersected by drilling in exploration Blocks 1 to 5. The footwall granitoid could either be a faulted sliver of the hanging wall granitoid, or a sill. The footwall granitoid is truncated in Block 6 but appears locally as thin lozenges or fault slices into Block 7 and 8. The footwall granitoid likely played an important role in the development of deposit, providing a strong rheological contrast for the folding of volcano-sedimentary beds, resulting in dilation zones during D2 folding. The shear zone at the contact can be up to 20 m wide, showing even mylonitic textures hosted mainly by the sediments. The footwall granitoid has a sill-like geometry up to 120 m wide, and its composition overlaps the classification boundaries between granodiorite and quartz monzonite. Bent plagioclase, polygonised quartz grains, and multiple chlorite-white mica-rutile-magnetite shear seams are evidence of deformation during lower greenschist regional metamorphism (Thompson, 2022). The western contact of this sill is a mafic volcanic rock extending westwards to the center of the project area (Figure 7.5 and Figure 7.7). A whole rock analysis by Tedeschi (2022, pers. comm.) reveals that the mafic volcanics have a tholeiitic basalt composition.

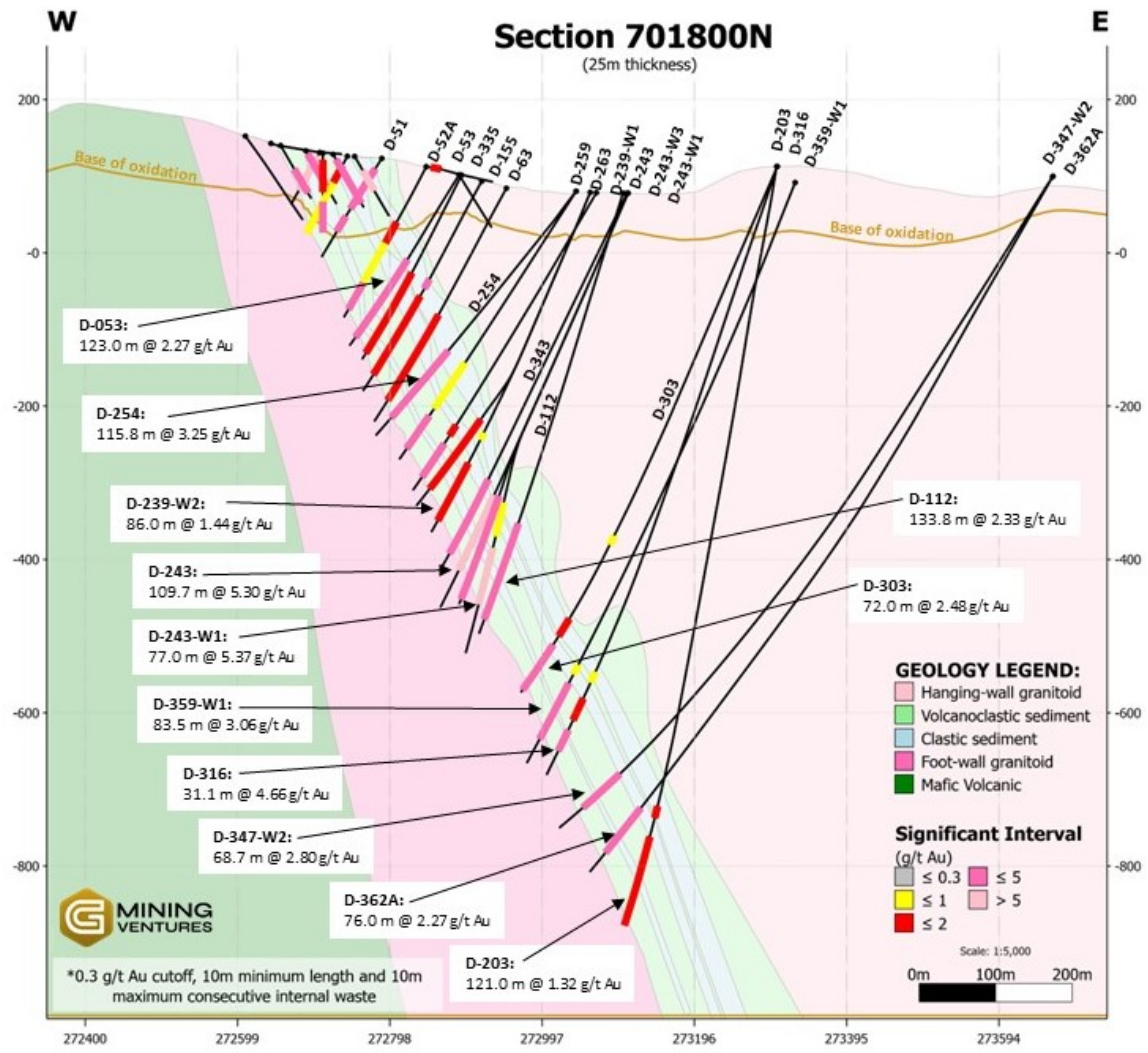
Smaller intrusive bodies (decimetric) with the same granitoid composition and texture can often be observed within the volcano-sedimentary units. Mostly parallel to the bedding, they highlight the preferential path created by slippage during folding. Larger intrusive bodies (decimetric to metric) can also be locally observed at the contact between units with different rheological characteristics, such as massive volcanic rocks and sediments. These larger intrusions are often associated with strong alteration (K-feldspath or epidote assemblages) and metasomatism affecting surrounding volcano-sedimentary rocks.

Figure 7.6: Granitoid Rock Samples



Source: Hainque et al., 2022
**Notes: A: Granitoid in hole D21-047. B: Granitoid showing strong epidote alteration in hole D22-062. C: K-felspar-rich granitoid with strong chloritization in hole D21-047. D: Granitoid with strong chloritization in hole D21-047. E: Granitoid with strong epidotization and evidence of reverse faulting in hole D22-062. F: Felsic intrusive body at the contact between units with different rheological characteristics (carbonaceous sediments and volcaniclastics) in hole D21-047.*

Figure 7.7: Geological Cross-Section 701800 N, Looking North

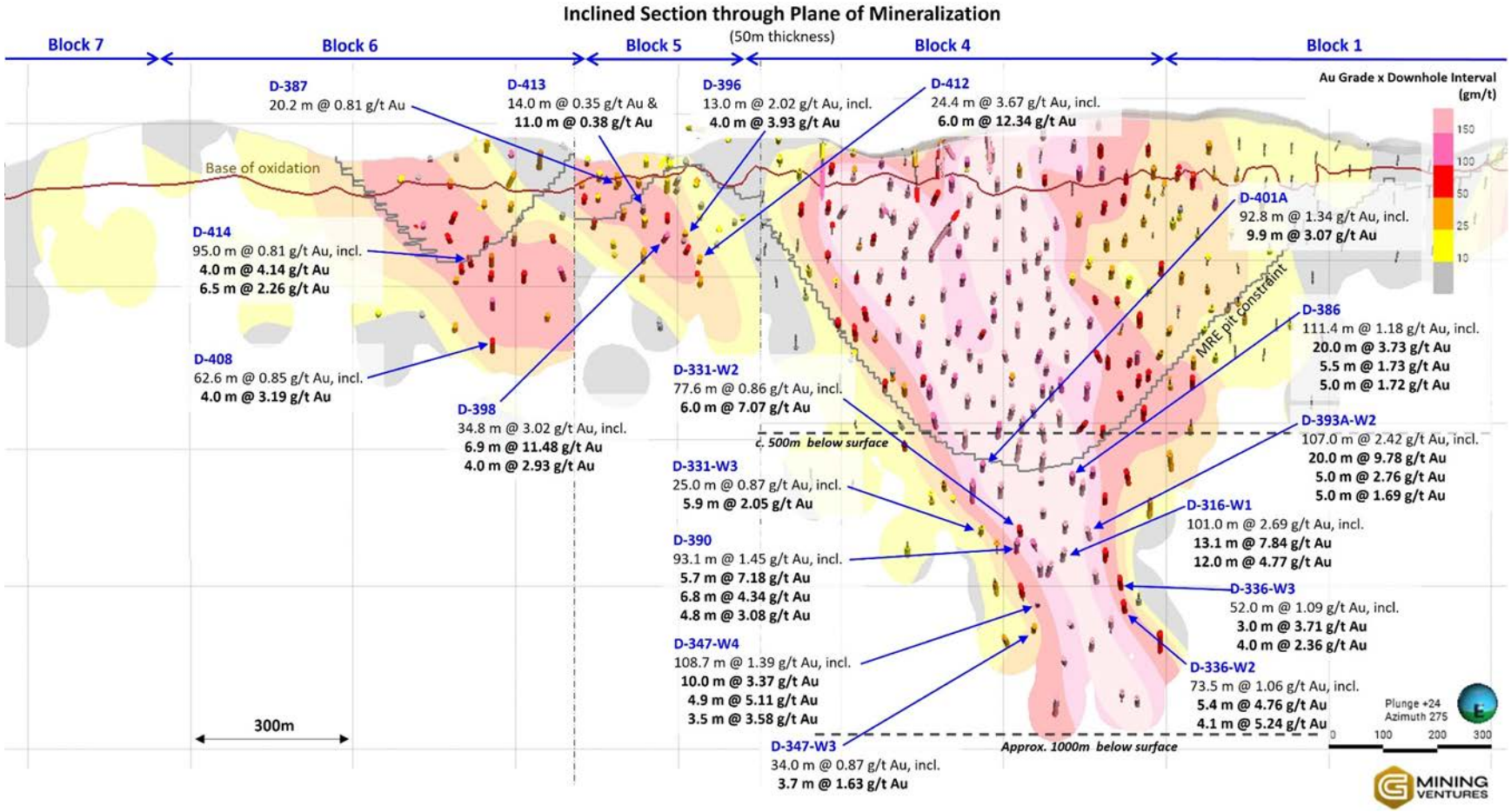


Source: GMIN, 2024

7.2.3 Volcano-Sedimentary Sequence

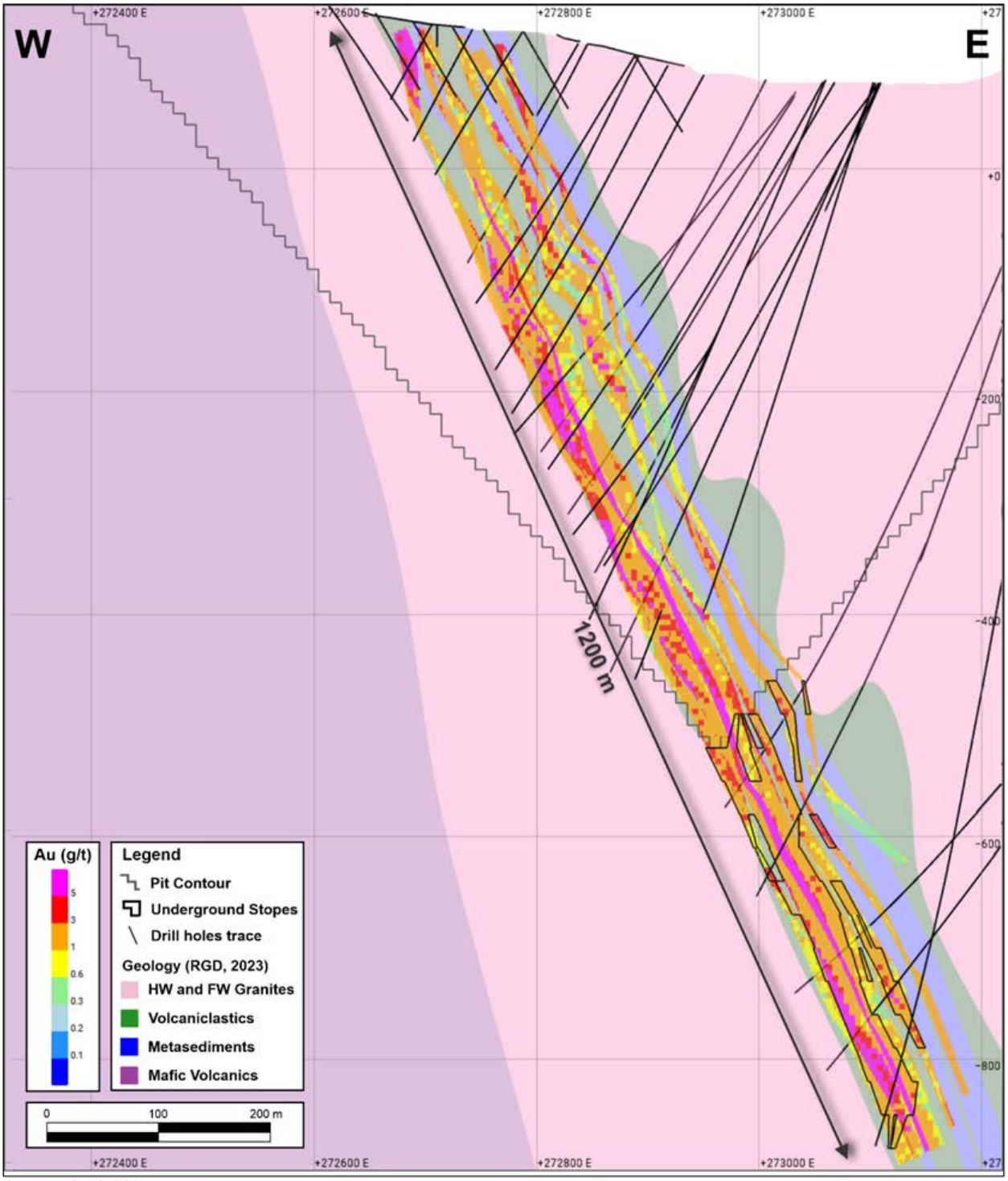
The sequence hosting the bulk of the Oko West gold mineralization is composed of clastic rocks: siliclastic, volcanoclastic, and carbonaceous, and is better understood in the Kairuni zone, the 2.5 km northernmost extent of known gold mineralization. Some of the volcaniclastics may be coherent volcanics, such as flows and associated hyaloclastites, but this has yet to be established. The sequence is 100 to 200 m wide and has an overall tabular geometry, dipping steeply to the east and "sandwiched" between the Oko pluton to the east and the footwall granitoid to the west (Figure 7.5). The units are intercalated and strongly deformed. The mineralized sequence has been intercepted by drilling to a depth of 600 m from the surface (drillhole D22-112) but is known to continue much further down-dip (see longitudinal section in Figure 7.8). This sequence straddles the contact with the Oko pluton for the entire length of the project area within the Kairuni and Takutu zones.

Figure 7.8: Longitudinal Inclined Section Along Mineralized Zone (Blocks 1 to 6)



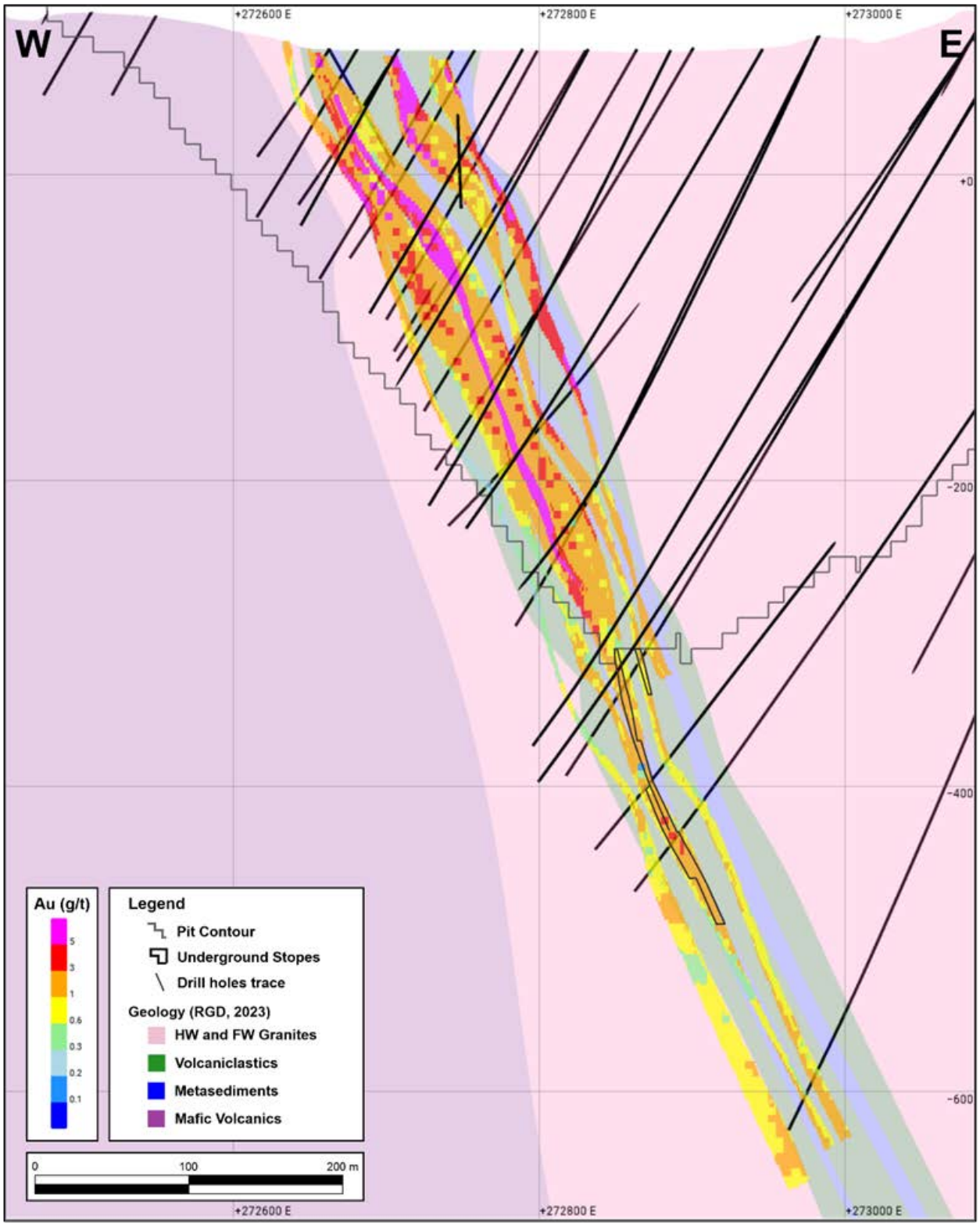
Source: GMIN, 2024

Figure 7.9: Geological Cross-Section 701800 N, Looking North



Source: GMS 2024

Figure 7.10: Geological Cross-Section 701560 N, Looking North

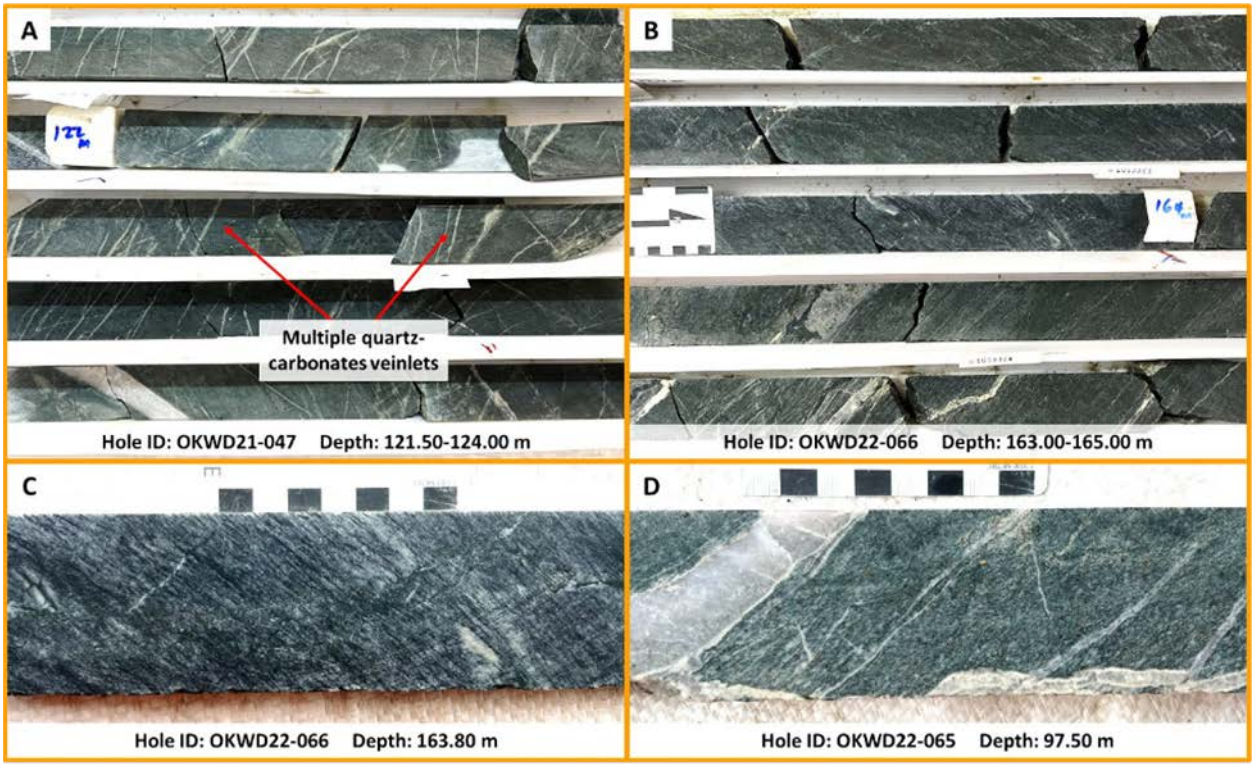


Source: GMS, 2024

7.2.3.1 Volcanoclastics

Volcaniclastics are herein defined as clastic rock containing predominantly volcanic particles of any shape or size and does not imply any specific clast-forming, transport or depositional process (McPhie et al.,1993). They show local evidence of bedding and crossbedding as well as more massive facies with possible perlitic textures suggesting both sedimentary and effusive origins respectively. They present numerous imbricated quartz-carbonate veinlets transposed to the foliation. Volcanoclastic rocks are highly chloritized and usually have a dark green colour in unweathered rock and a purple colour in weathered rock (Figure 7.11). They are mainly composed of silt to sand sized grains, but locally contain polymict and monomict fragments up to several cm in size. They can also show some evidence of siliciclastic elements, such as quartz and sericite, as contacts between the volcaniclastics and siliciclastics appear gradational and thus, conformable. Moreover, the frequent numerous transposed quartz-carbonate veinlets can give them a strong bedding-like characteristic. However, these volcanoclastics usually show stronger chloritization and are darker than the siltstones and sandstones.

Figure 7.11: Volcanoclastic Core Samples

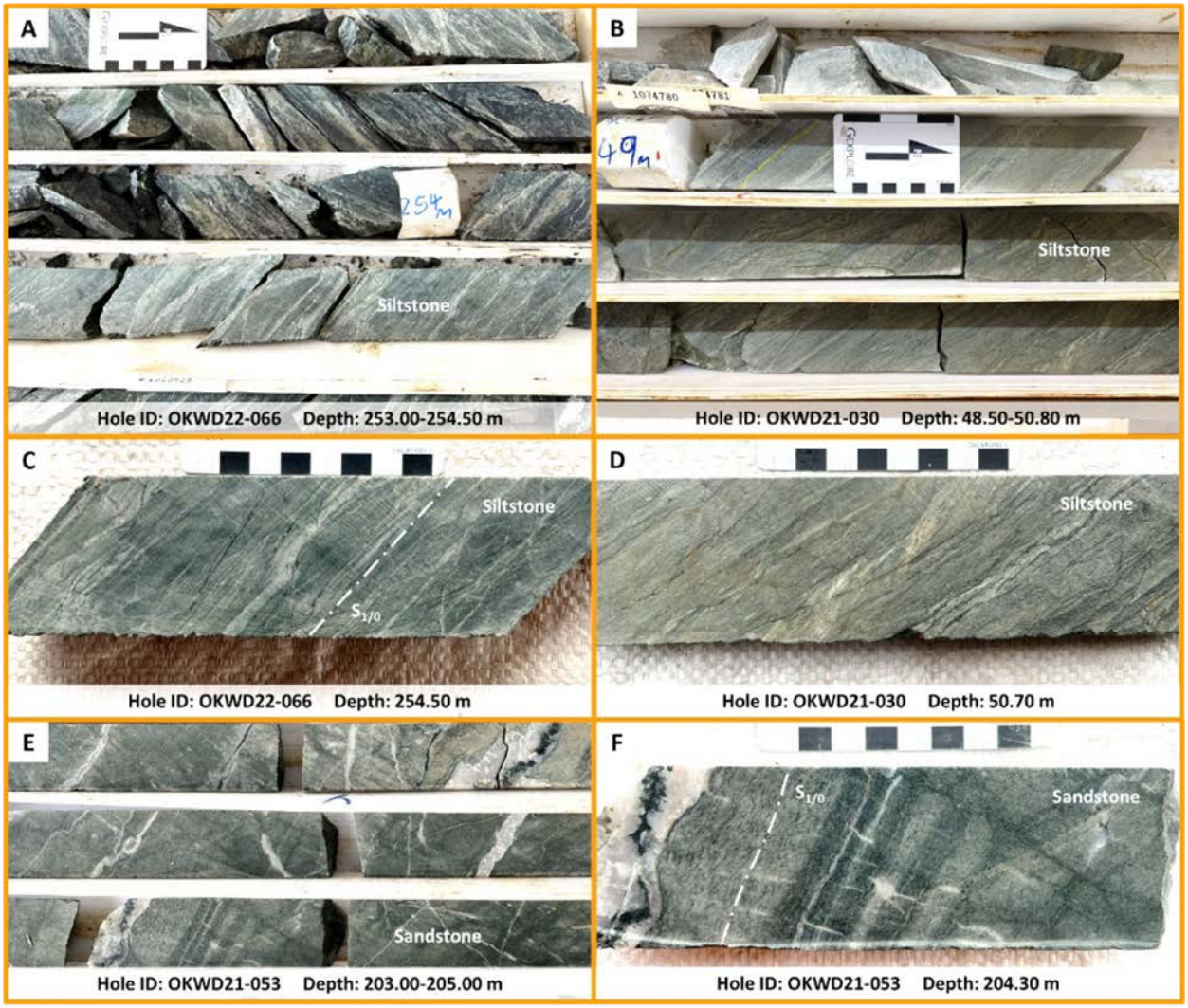


Source: Hainque et al., 2022
**Notes: A: Fine volcanoclastics with imbricated quartz-carbonate veinlets transposed to the main foliation in hole D21-047. B: Coarser and bedded volcanoclastics showing evidence of bedding in hole D22-066. C and D: Bedded volcanoclastics in holes D22 066 and D22-065.*

7.2.3.2 Siliciclastic Sediments

The siliciclastic sediments correspond to interlayered sandstones and siltstones. They present a pale grey / beige colour and are usually highly carbonatized and moderately sericitized, with quartz and probably fully carbonitized and/or sericitized feldspar (Figure 7.12). Locally they can be pale green with weak chloritization. They show clear bedding, preserved even in zones of intense S2 fabric development, and local evidence of crossbedding. The weathered product of these sedimentary rocks is generally yellow / orange. Sandstone-dominated rocks occasionally appear near the contact with volcaniclastics. Siltstones may grade into carbonaceous sediments.

Figure 7.12: Siltstone and Sandstone Core Samples

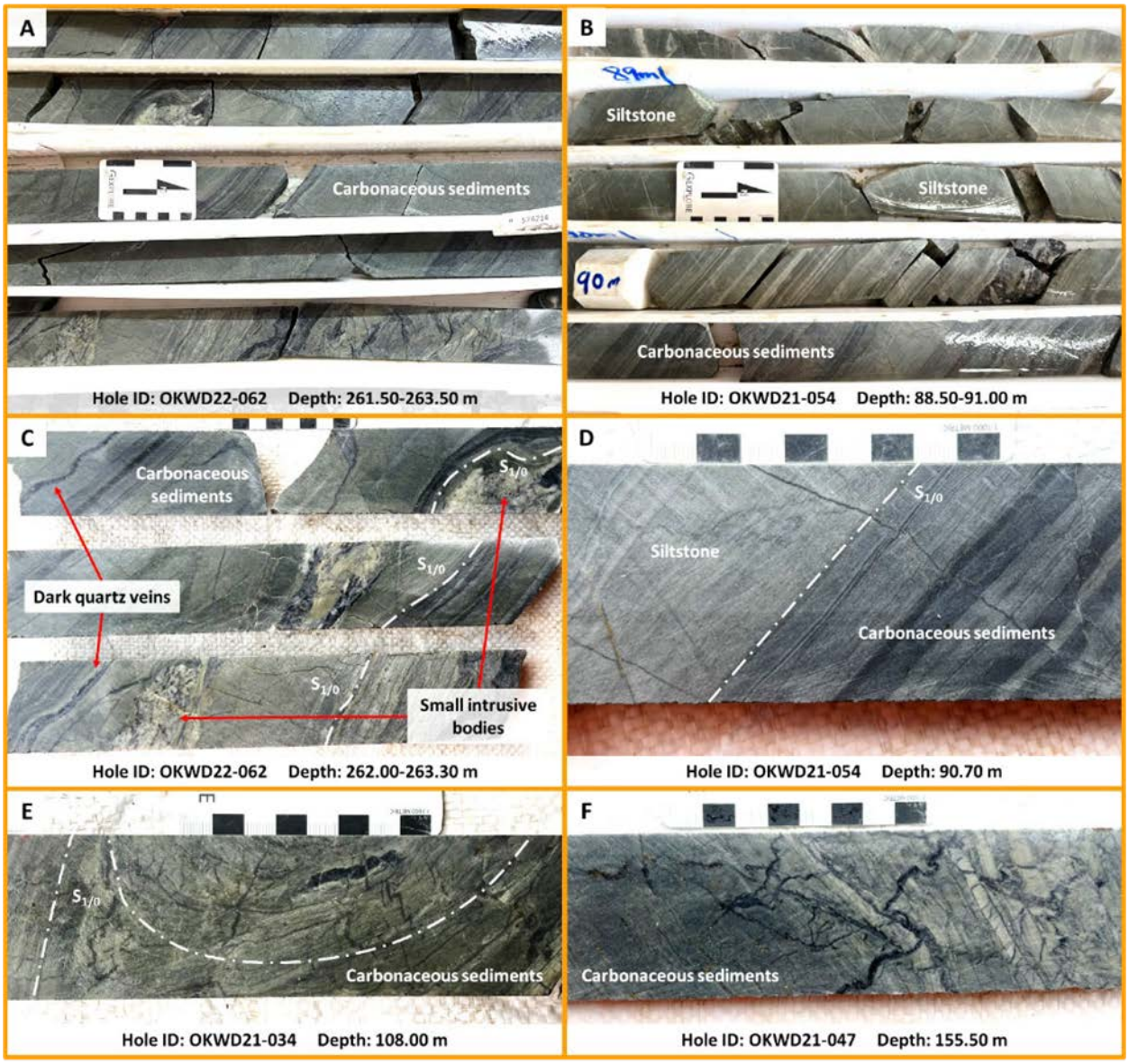


Source: Hainque et al., 2022
**Notes: A and B: Siltstone in holes D22-066 and D21-030. C and D: Bedded siltstones in holes D22-066 and D21-030. E: Sandstone dominated rock in hole D21-053, crosscut by several quartz veins. F: Well- sorted sandstones in hole D21-053.*

7.2.3.3 Carbonaceous Sediments

Carbonaceous sediments correspond to siliciclastic turbidite-like facies, alternating fine carbonaceous bearing sandstone / siltstone, and fine carbonaceous-rich shale layers (graphitic schists), as shown in Figure 7.13. While weathered, this lithology develops a grey colour and can show the remaining shale layers. Due to the weak nature of the carbonaceous material, this lithology also composes a preferential "decollement" layer accommodating bedding-parallel slippage during deformation. This bedding-parallel slippage likely allowed responsible for small intrusive bodies (intercalated sediments and granitoid) and the emplacement of dark grey quartz extension and shear veins. The dark grey colour of these veins might be related to a chemical reaction between the carbonaceous material and the hydrothermal fluid.

Figure 7.13: Carbonaceous Sediment Core Samples



Source: Hainque et al., 2022
**Notes: A: Carbonaceous sediment in hole D22-062, with clear bedding and several dark quartz veins. B: Bedded siltstone and carbonaceous sediment in hole D21-054. C: Carbonaceous sediment in hole D22- 062, with several dark quartz veins, evidence of folding, and small intrusive bodies. D: Contact between siltstone and carbonaceous sediment in hole D21-054. E: Folded carbonaceous sediment in hole D21-034. F: Carbonaceous sediment crosscut by multiple small dark quartz veins in hole D021-047.*

7.3 Structural Geology and Metamorphism

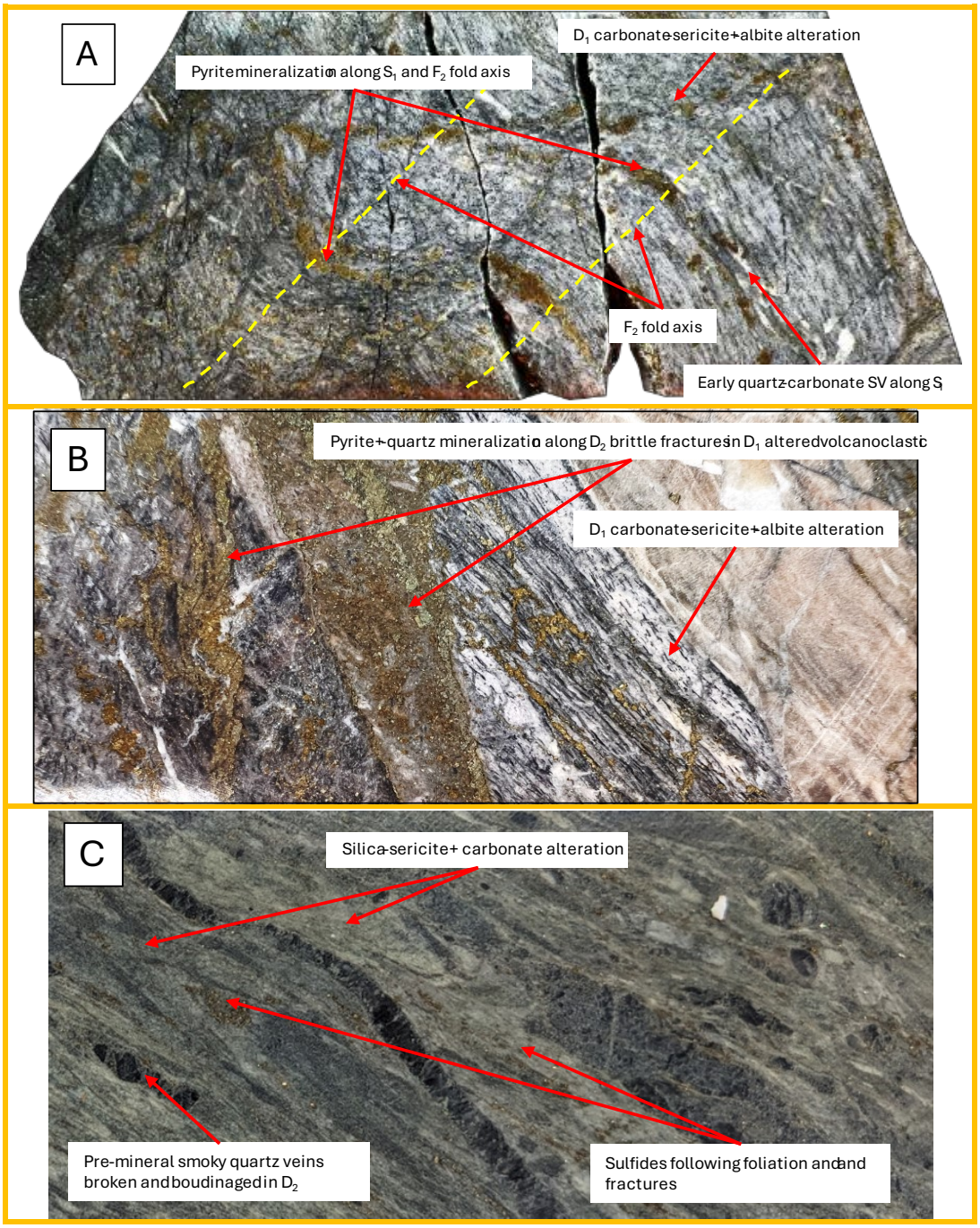
Rhyacian regional greenschist metamorphism and associated deformation modified clastic and plutonic rocks. The metamorphism of clay size grains in sedimentary and volcanoclastic rocks was preceded by dehydration and decarbonization. Metamorphism of granitoids is limited to the recrystallization of protolith minerals unless metamorphic fluids gain access via grain boundaries and/or litho-structural conduits (fractures, shear zones). In this case, hydration and, locally, carbonatization transformed magmatic

minerals into metamorphic mineral assemblages compatible with the greenschist grade of metamorphism. The intensity or degree of change varies with fluid access, typically increasing toward pluton contacts and internal litho-structural conduits. The intrusion of granodiorite / quartz monzodiorite occurred before (syn volcanism) or in the early stage of Rhyacian orogenesis. The petrography work by Thompson (2022) shows that the metaclastites, matrix carbonate, chlorite, plagioclase, and white mica are recrystallized protolith minerals and/or products of metamorphism. Stress-related mobilization of matrix minerals in adjacent and/or nearby rocks is the preferred explanation for the formation of carbonate and carbonate-plagioclase-quartz veins. That is, overall rock compositions remain essentially unchanged.

Oko West outcrop and core observations by Lacroix (2022), Hainque et al. (2022), and Oko West Project geologists demonstrate that the area is marked by polyphase deformations, with a first N-S tight folding from an E-W shortening event, followed by a second E-W fold overprint from a N-S shortening event. Gold mineralization occurs predominantly within volcanoclastic, siliciclastic, and carbonaceous sediments, characterized by sulphide ± silica (pyrite, chalcopyrite, sphalerite) overprinting earlier silica, carbonate, and sericite alteration (Figure 7.14 and Figure 7.15). The mineralized intervals are generally associated with higher intensities of pre-mineral alteration and veining including quartz / quartz carbonate shear veins (SV) and multiple generations of extension veins (EV). The early stage EVs were transposed to the foliation during fold tightening by bedding-parallel slip. Later EVs are crosscutting S1/0 and the first generations of EVs, marking a deformation continuum.

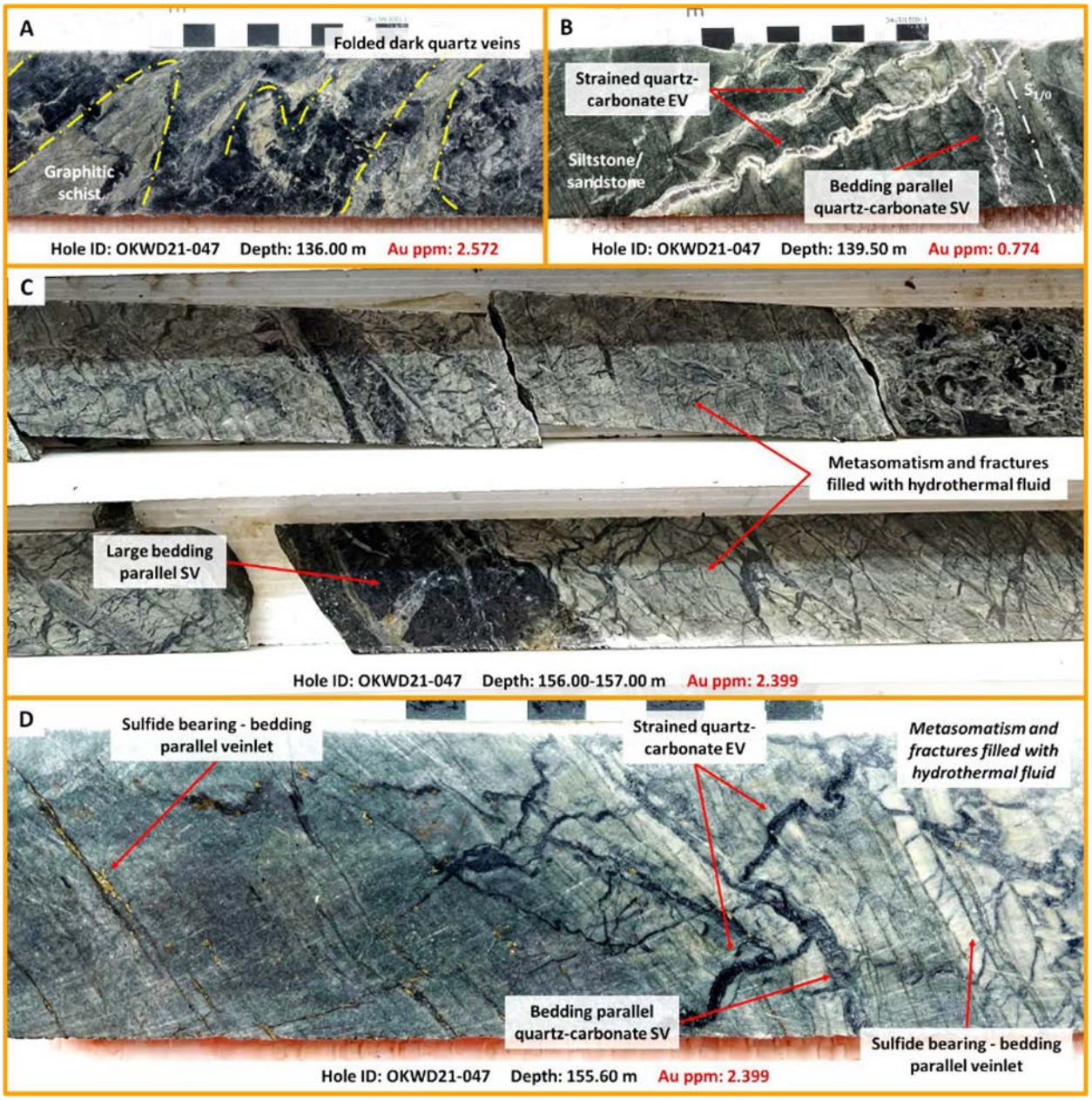
One major D1 event was probably responsible for the continuous development of the main quartz / quartz carbonate EV-SV system and related quartz, carbonate and sericite alteration (D1a and D1b). This event was characterized by tight folding, with the main volcano-sedimentary package sandwiched between the footwall and hanging wall granitoids, and responsible for the emplacement of most of the veining and alteration and possibly minor amounts of gold mineralization. The D2 event is associated with the development of S2 crenulation and mineralization. Most of the gold was emplaced during this phase in F2 fold axis and along brittle fractures brecciating the earlier D1 vein systems and associated alteration. The tight folding D1 and the broad open D2 fold events produced a subdued type-2-fold pattern, affecting the stratigraphy and mineralization. In summary, the mineralized sediments record a history of multiphase deformation that overlapped in time and space with lower greenschist regional metamorphism.

Figure 7.14: Mineralized Structures in Core Samples



Source: Hainque et al., 2022.
**Notes: Images A and B show quartz-feldspar crystal rich volcanoclastics with strong overprinting carbonate-sericite-albite pervasive alteration, subsequently folded and brecciated during D2. Image C shows the same relationships including pre-mineral smoky quartz EV-SV, in a quartz-sericite-carbonate+-albite altered siltstone. Sulphide mineralization is concentrated along F2 fold axis and D2 fractures along with pre-existing D1 fabrics, such as S1.*

Figure 7.15: Mineralized Structures in Core Samples (EV and SV)



Source: Hainque et al., 2022
**Notes: Bedding parallel SV, folded EV, and fractures due to metasomatism. A: Folded sulphide-bearing dark quartz-carbonate veins in hole D21-047. B: Quartz-carbonate SV and multiple folded EV in siliciclastic sediment in hole D21-047. C: Highly fractured interval (hardened by metasomatism) filled with sulphide-bearing quartz-carbonate veinlets (stockwork) and large SV in hole D21-047. D: Sulphide bearing fractures in a medium hardened by metasomatism in hole D21-047.*

7.4 Tectonic Events and Gold Mineralization

Combining observations from the Oko West Project team and structural analyses of the drill core and field visits, Lacroix and Hainque (2024) have proposed the following geological evolution for the mineralization at Oko West, reflecting on the regional tectonic regimes described in Section 7.3 (Figure 7.16):

D0: Deposition of the volcano-sedimentary sediments and mafic volcanic rocks and emplacement of the granitoid intrusions responsible for early potassic veins and metasomatism.

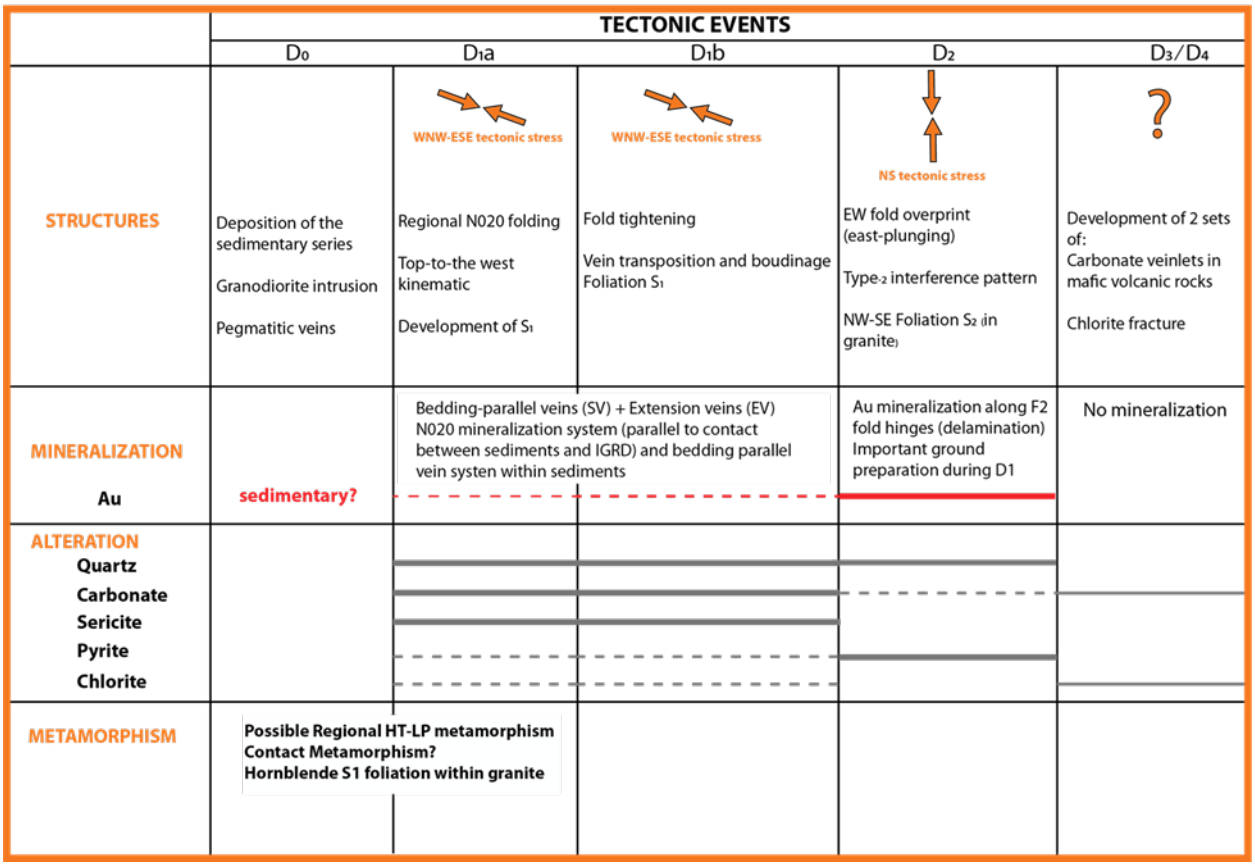
D1a: WNW-ESE to NW-SE compressional event related to F1 folding, associated with the development of S1. Bedding-parallel veins and "en-echelon" EV started developing during folding and may have contained minor gold.

D1b: Late stage of tight folding, marked by the formation of penetrative foliation, S1, and the transposition of early extension veins (EV1) to S1 and further development of the EV system. The transposed EV1 veins are locally dismembered and sheared, forming sigmoidal tectonic clasts imbricated along the foliation.

D2: N-S fold overprint, associated with the development of S2 crenulation and may be associated with the development of the late quartz-Sulphide EV2 vein system. The more discrete nature of S2 highlights less tight folding, suggesting that F2 might have a larger amplitude than F1. Most of the gold was deposited during this phase, focused along F2, S1 and D2 brittle fractures.

D3: Development of two (2) sets of conjugated fractures / faults with possible significant offset.

Figure 7.16: Tectonic Events and Gold Mineralization at Oko West (Lacroix and Hainque, 2024)



Source: Lacroix and Hainque, 2024

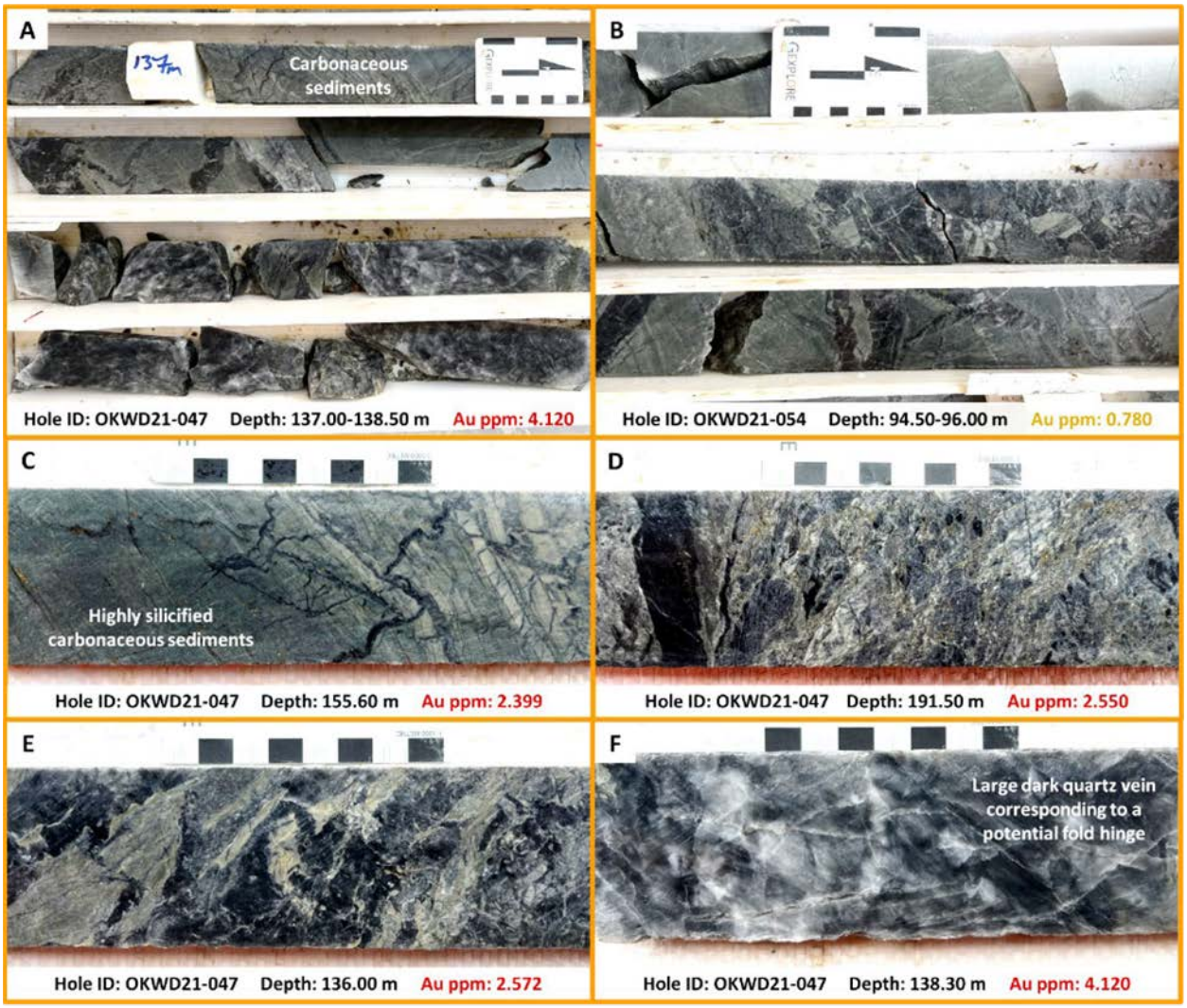
7.5 Gold Mineralization

Gold mineralization mainly occurs within volcanoclastic, siliciclastic, and carbonaceous sediments, which have an overall tabular geometry dipping to the east. Strong evidence of silica and carbonate alteration can be observed within the mineralized zone, with more intense sericitization, as well as with the presence of multiple sulphides (pyrite, chalcopyrite, sphalerite) disseminated within the altered rock, along bedding / laminations or small fractures / veinlets, or as envelopes around the quartz and quartz-carbonate veins. Despite the spatial association, most of the alteration is pre-mineral and served to harden the rock allowing for brittle deformation and dilation during the D2 mineralization stage.

Other hydrothermal alteration does not seem directly related to the mineralization, or their association with it is unclear. Chloritization moderately to highly affects all lithologies in mineralized and non-mineralized areas. Epidotization was only observed in non-mineralized areas, mainly in the upper and intermediate granitoids, in the upper volcano-sedimentary sequence between them, and locally in the lower main volcano-sedimentary package. Magnetization has been observed in mineralized and non-mineralized areas, with strong variability, even within the same unit (probably due to alteration to other oxides).

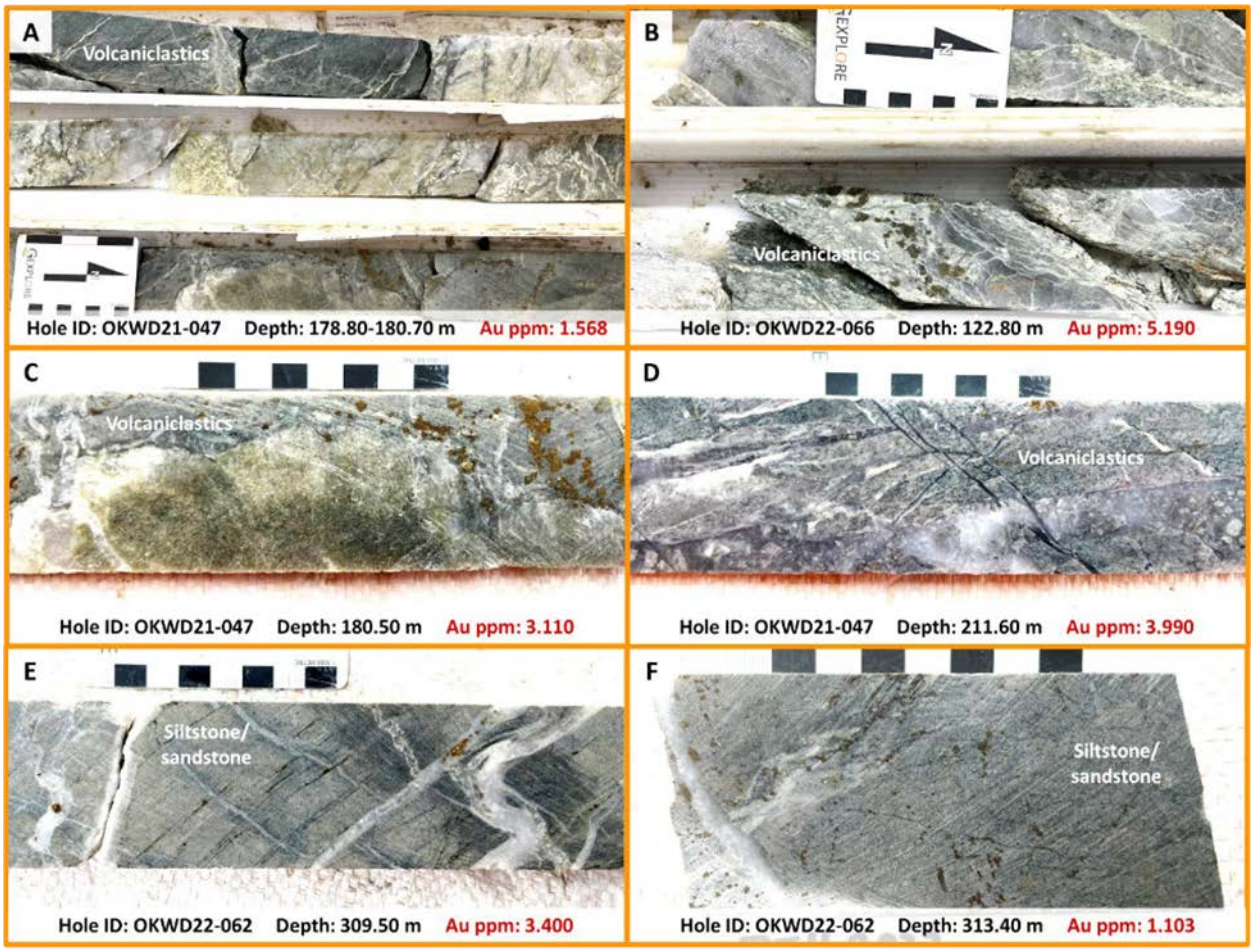
The hydrothermal fluids responsible for sulphide and gold emplacement produced a network of Sulphide bearing quartz and Sulphide-only stringers spatially associated with EV quartz-carbonate vein systems and their hydrothermal alteration that partially or totally overprint the parent rock. In the carbonaceous sediments, the quartz / quartz-carbonate vein system is characterized by a dark grey smoky colour (Figure 7.17), whereas in the volcanoclastic and siltstones / sandstone lithologies, it is characterized by a white-grey colour (Figure 7.18). The dark gray colour of the quartz / quartz-carbonate veins in the carbonaceous sediments could be related to a geochemical reaction between the carbonaceous material and the hydrothermal fluid. This carbonaceous material may have acted as a reducer for the later Sulphide and gold bearing fluids, leading to particularly high grades of gold.

Figure 7.17: Gold Mineralization in Carbonaceous Sediment Core Samples



Source: Hainque et al., 2022
**Notes: A and B: Mineralized carbonaceous sediment intervals in holes D21-047 and D21-054, with strong silicification and carbonatization. C: Mineralized stockwork in hole D21-047. D: Highly brecciated mineralized interval with strong silicification, carbonatization, and sulfidation in hole D21-047. E: Folded sulphide-bearing dark quartz-carbonate veins in hole D21-047. F: Large dark quartz-carbonate vein corresponding to a potential fold hinge in hole hinge in hole D21-047.*

Figure 7.18: Gold Mineralization in Volcanoclastic and Siltstone / Sandstone Core Samples



Source: Hainque et al., 2022

**Notes: A: Mineralized volcaniclastic interval in hole D21-047, with strong silicification and carbonatization, associated with metasomatism. B: Mineralized volcaniclastic interval in hole D22-066, with multiple large pyrites. C: Metasomatism from the hydrothermal fluid and multiple pyrites showing pressure shadows in hole D21-047. D: Highly strained and fractured mineralized volcaniclastic interval with multiple sulphide- bearing quartz-carbonate veins in hole D21-047. E and F: Mineralized siliciclastic interval with multiple sulphide-bearing quartz-carbonate veins and veinlets in hole D21-062.*

8 DEPOSIT TYPES

The relationship between gold mineralization, magmatism, and deformation in the Guiana Shield is poorly defined by the literature, despite the presence of several major deposits (Tedeschi et al., 2020).

In French Guiana, orogenic-type gold deposits are mainly related to the regional D2 tectonic-metamorphic deformation (between 2.1 and 2.0 Ga). The quartz veins-related mineralization occurs along shear zones in greenstone belts and is associated with granitic magmatism (Milési et al., 1995; Milési et al., 2003). Tedeschi et al. (2020) outline that recent data from the Karouni orogenic gold deposit in Guyana support this timing, as gold mineralization has been dated to 2.084 Ga ± 14 Ma. The Karouni orebodies are primarily related to shear-hosted quartz-carbonate-chlorite ± tourmaline-pyrite-gold veins in high MgO basalts and high TiO_2 dolerite sills and granodiorite. This mineralization type occurred within dilatational bends formed by the late dextral transcurrent movement of strike-slip shear zones. They are controlled by rheological contrast, as brittle deformation of dolerite sills and granodiorite resulted in better mineralized extensional veins than ductile deformation of basalts. Thus, gold mineralization is found at the interface between these lithologies and structures (Tedeschi et al., 2018). Similarly, at the Wenot Lake deposit at Omai, gold mineralization occurs along a shear zone straddling the contact of sedimentary and volcanic sequences (Bertoni et al., 1991). At the Rosebel gold mine in Suriname, mineralized shear zones developed along contacts between units of varying rheologies, but also, to a lesser degree, parallel to axial plane cleavages in fold noses. The bulk of mineralization at the Royal Hill deposit, for example, is hosted in bedding-parallel quartz-carbonate-tourmaline veins along lithological contacts (Wasel et al., 1997, and Daoust, 2016).

9 EXPLORATION

This section describes all non-drilling-related exploration work carried out in the Project area. The discovery of gold mineralization at Oko West is relatively recent and limited historical exploration work was carried-out in the Project area other than work by artisanal miners directly relating to historic alluvial mining.

9.1 Geophysics

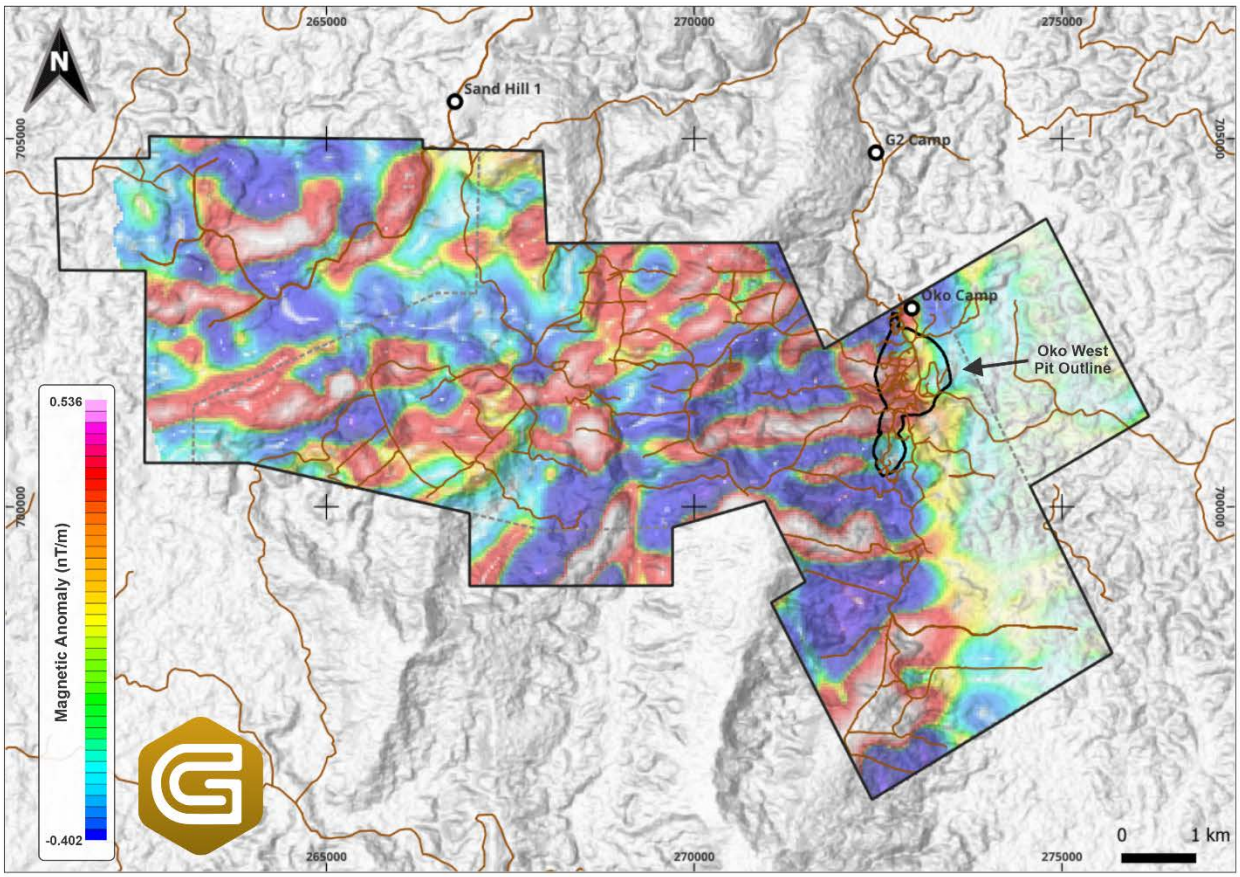
9.1.1 Airborne Geophysics

In August 2019, the Canadian company Terraquest covered the project area with an airborne geophysical survey of about 690-line km using the specifications below:

Parameters	Specification
Aircraft	King Air C90 Registration C-GCFZ
Primary Airborne Geophysical Sensors	High-Resolution Magnetics (Cesium Vapour), Gamma Ray Spectrometers
Base Station Sensors	High Resolution Diurnal Magnetics (Cesium Vapour); Base GPS L1/L2 12 channel, fully kinematic grade
Aircraft Magnetometer Sensitivity	+/- 0.005 nT
Magnetometer Noise Envelope (4th diff.)	+/- 0.5 nT counting at 0.1 Hz
Traverse Line Direction	344° / 164°
Control Line Direction	074° / 254°
Traverse Line Spacing	200 m
Traverse / Control Intersection Tolerance	± 15 m
Survey Clearance (AGL)	~60 m above the canopy
Magnetic Sample Interval	7-8 m (10 Hz sampling)
Average Local Ferry	~130 km
Aircraft Survey Velocity - Nominal	~205-240 km/h
Average Ferry Speed	~325 km/h

This geophysical survey assisted in the interpretation of the area's structural geology and lithological distribution (Figure 9.1), and clearly defined the contact between greenstone volcanics and the Oko pluton to the east. Although the line spacing of 200 m is relatively wide, the distinction between magnetic highs and lows can be readily identified.

Figure 9.1: RTP 1VD Map of Airborne Magnetic Data

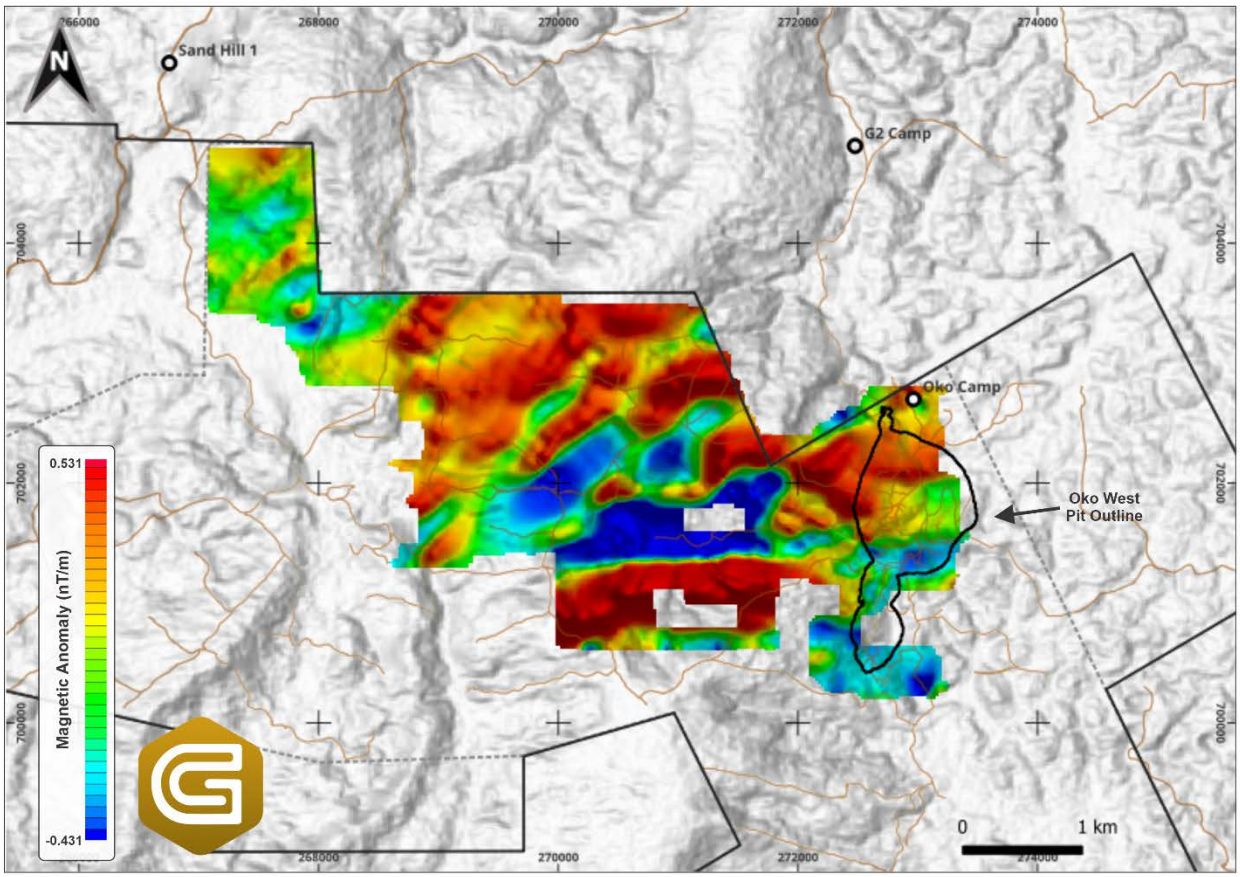


Source: GMIN, 2024

9.1.2 Ground Magnetics

Ground Magnetics survey has been ongoing on the Oko West property since early 2023 with GMIN own equipment and personnel. Sectors of surveys are chosen quarterly based on exploration interest (Figure 9.2). Since early 2024, multiple test lines were re-surveyed using the continuous walking mode to ensure that the best functional settings were being applied and former issues resolved. Infill lines were planned and surveyed in blocks 5, 6, 7 and 8. The data was still being processed at the time this report was written.

Figure 9.2 : Map of RTE Ground Mag Coverage over Terrain

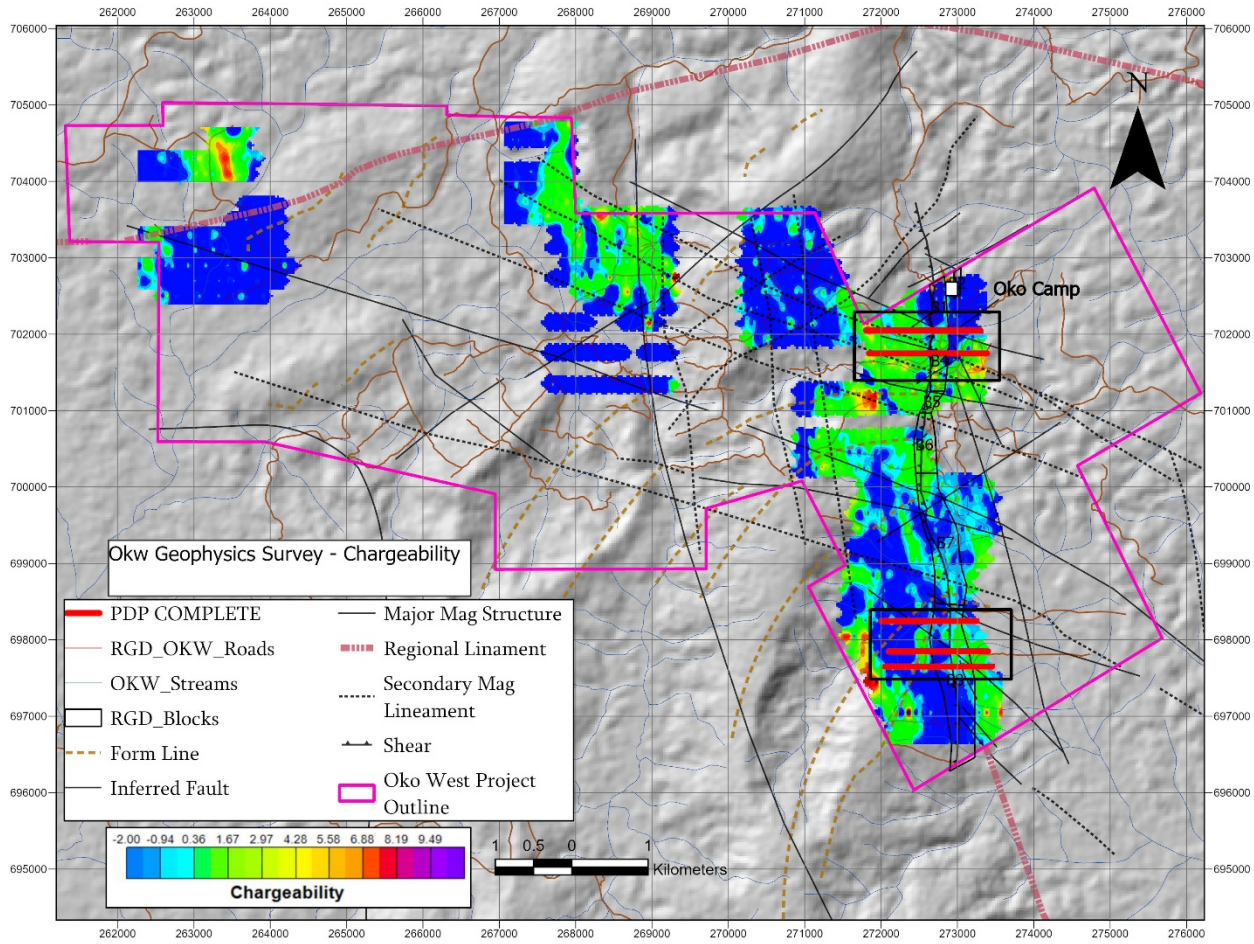


Source: GMIN, 2024

9.1.3 Induced Polarization and PDP

The induced polarization geophysics program was completed towards the end of March 2024. Five (5) areas of interest were surveyed and used for exploration targeting across the Oko West property using chargeability and resistivity data. Two (2) pole dipole (PDP) lines were done in Block 4and three (3) in the East Takatu area. These surveys were used to plan drillholes in Blocks 7 and 8 which target potential mineralization at depth (Figure 9.3).

Figure 9.3: Map of IP Chargeability Coverage with Mineralized Domains at 0 m RL Over Terrain



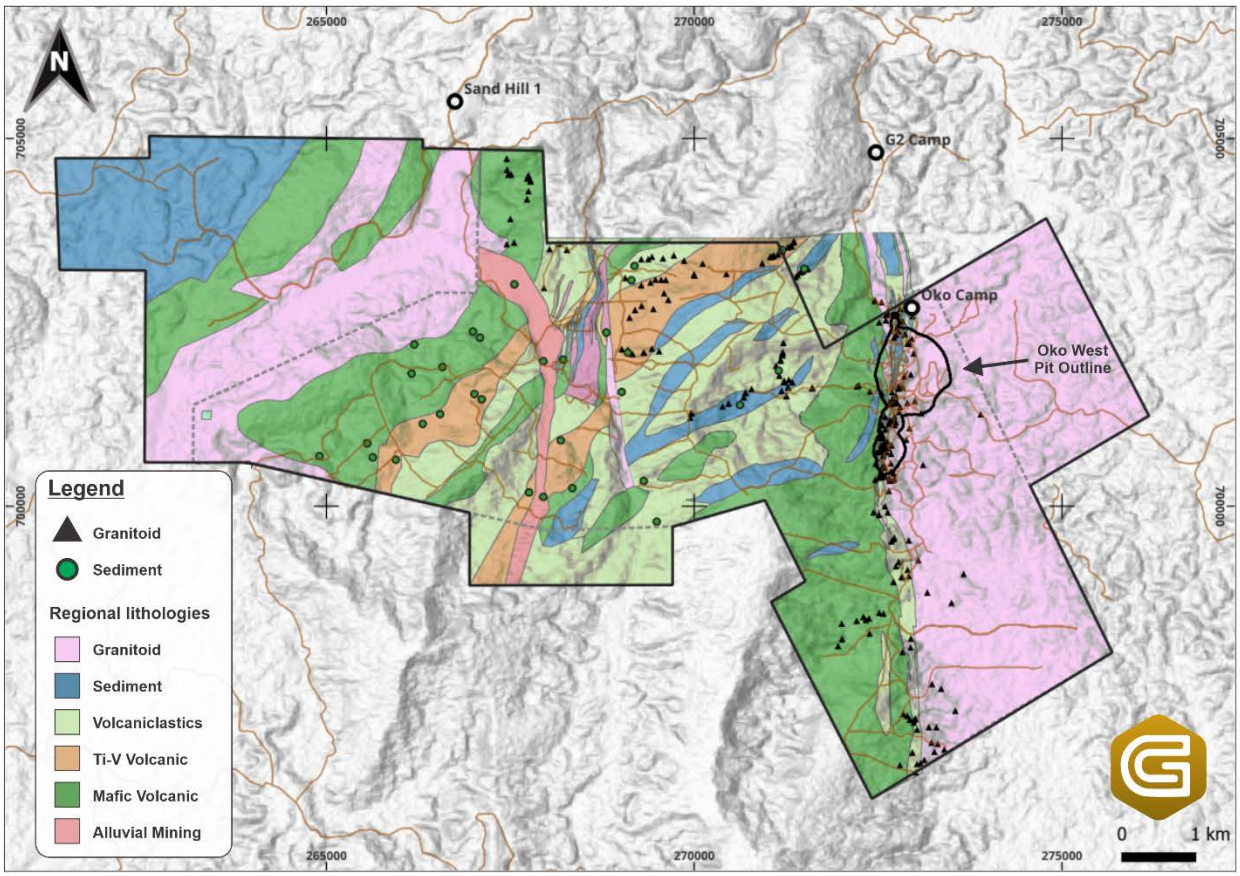
Source: GMIN, 2024

9.2 Geology Reconnaissance and Stream-Sediment Geochemistry

During September and October 2019, geologists Jorge Tachibana from RGD and Deuel Garner from GMIN completed a reconnaissance of the area's geology by mapping and sampling available outcrops (Tachibana and Garner, 2019). A total of 18 rock samples, 19 regolith samples, and eight (8) hand-auger samples were collected.

A stream-sediment sampling campaign collecting both BLEG samples (Bulk Leach Extractable Gold) and sediment samples for ICP-MS was carried out from 35 sites in the central-western part of the project area (Figure 9.4). This stream-sediment campaign did not cover creeks along the eastern edge of the permit area, where the gold mineralization at Oko West was later discovered.

Figure 9.4: Stream Geochemistry Survey and Reconnaissance Mapping Points Plotted on Geology



Source: GMIN, 2024

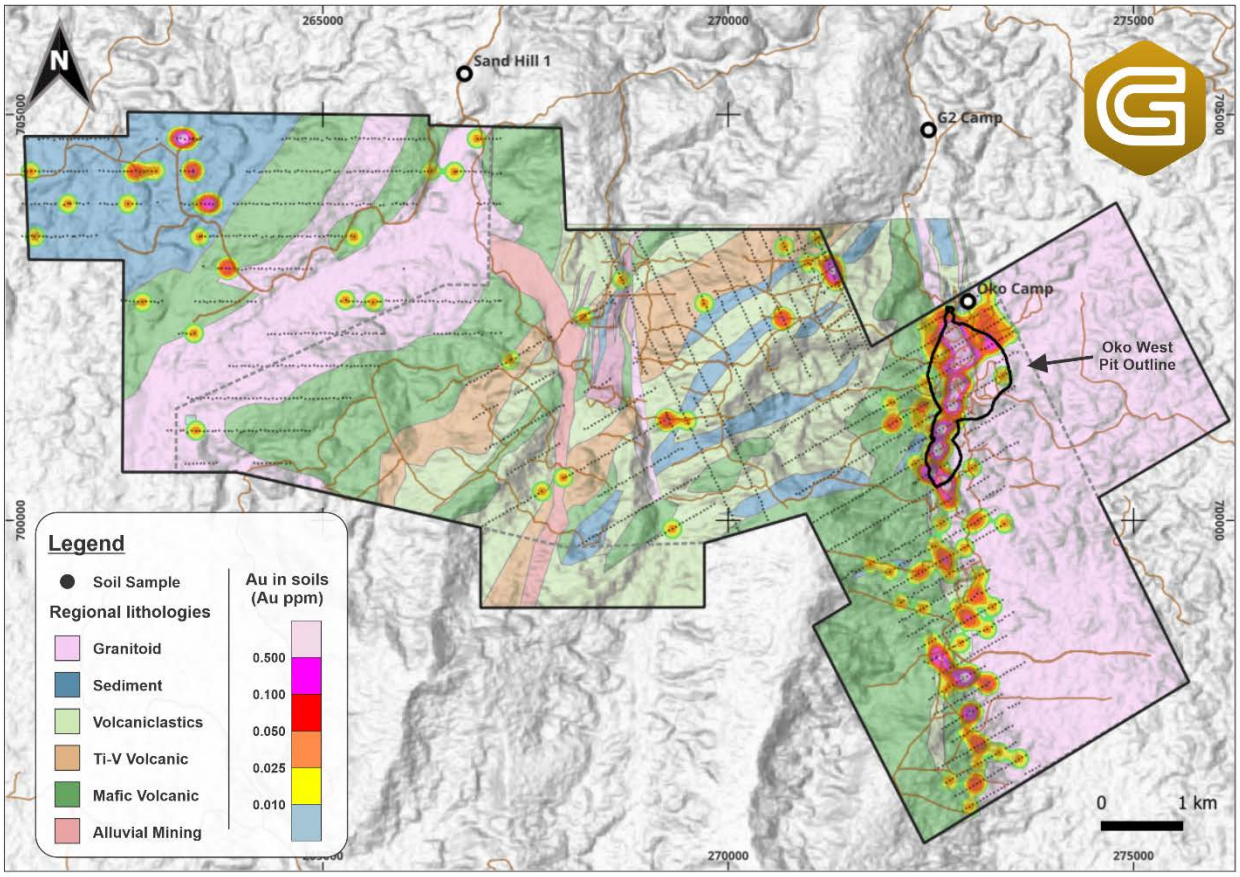
9.3 Soil Geochemistry

Early in 2020, Reunion completed a soil sampling survey covering the eastern-central portion of the Project area previously not covered by the stream-sediment survey. The eastern-central area drains many creeks with artisanal alluvial workings (Figure 9.5). The soil grid was oriented at 050° azimuth to cover both potential north-south and east-west trending mineralized structures. The project team collected soil samples at an average depth of 30 cm, every 50 m along lines and spaced 200 m apart. This survey defined gold anomalies straddling the contact between the Oko pluton to the east and the volcano-sedimentary sequence to the west, with a strike length of approximately 6 km. The anomalies are continuous at the northern end, at the headwaters of Kairuni creek, but become discontinuous at the southern half, at the headwaters of Takutu creek. The Company's exploration team observed that there were no primary historical workings up-terrain from the alluvial gold workings and that potentially intact bedrock mineralization could be present.

A second soil geochemistry program tested the permit's central area, exploring other potential source areas (Figure 9.5) relating to alluvial workings that straddle the western edge of the project. Results were comparatively weak and likely hindered by a thick duricrust layer blanketing the sector.

In total, the soil geochemistry program collected 1,691 samples within the project boundaries, mostly in 2019 and 2020. A total of 91 hand auger holes drilled within trenches generated an additional 351 samples within the saprolite profile.

Figure 9.5: Map of Soil Geochemical Program with Anomalies.
Lithology Background Superimposed Over Terrain



Source: GMIN, 2024

9.4 Trenching

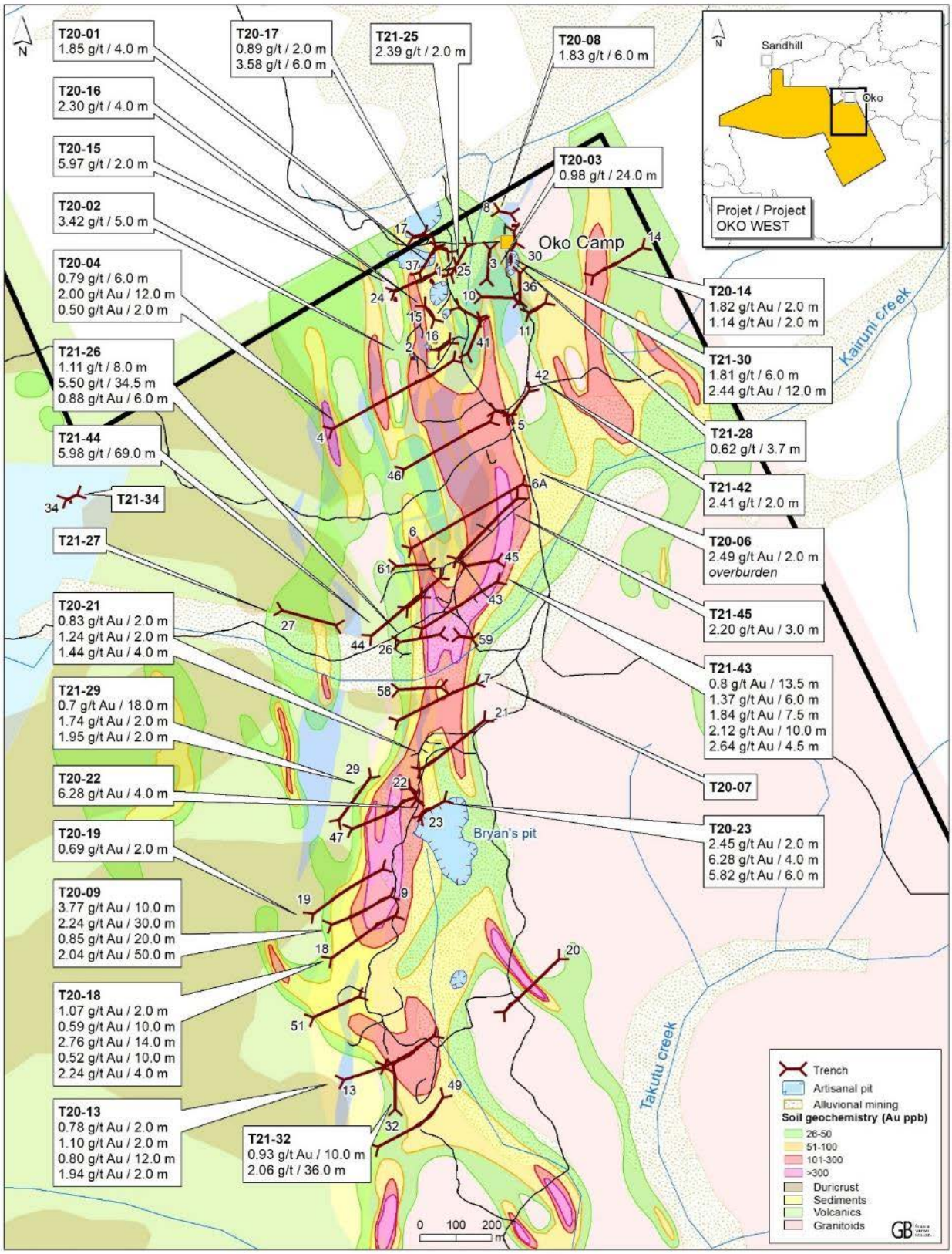
The soil geochemistry results identified significant gold anomalies at the headwaters of creeks with historical alluvial gold production needed to be explored further. The Company mobilized two (2) excavators and one (1) bulldozer from its Aremu project to launch a trenching program.

In the first quarter of 2020, the Company initiated the trenching program to test two (2) groups of anomalies, one south of the camp (exploration blocks 1 and 3) and another next to the so-called Bryan pit (exploration block 5 - Figure 9.6). The trenching program consisted of excavator-dug trenches along soil survey lines to a safe "shoulder" depth (approximately 1.2 m to 1.5 m) in an attempt to reach saprolitic material. Hand cut channels on the trench walls provided continuous samples. Only in-situ material was sampled, avoiding colluvium and alluvium, with an average sample length of 1.5 m (2-3 kg of material). Trenches were spaced at 200 m and attempted to traverse the entire width of the soil anomalies. Whenever saprolite was identified, the trench geology was mapped. Trenches are considered sub-horizontal drillholes for database purposes and were surveyed accordingly. As the trenches were dug along soil lines and at an angle to the north-south striking shear zones, some mineralized intervals do not represent true widths.

Trenching was interrupted during the second quarter of 2020 due to the Covid-19 pandemic and recommenced in September 2020, continuing uninterrupted until June 2021, when resource drilling commenced. During this period, trenching focused on exploration Block 4, intersecting several well-mineralized intervals and supporting geological interpretations.

The trenching program successfully validated the soil geochemical anomalies. They confirmed the presence of significant in-situ gold mineralization in a sequence of sediments striking north-south and at the contact with the Oko pluton granitoid. Figure 9.6 summarizes the trench sampling results in the Kairuni zone. Trenching is a powerful tool for exploring areas with soil anomalies and outcropping mineralization, providing quick access to bedrock geology and systematic sampling.

Figure 9.6: Map of Trenching Results in Kairuni Zone Plotted on Geology and Soil Geochemistry



Source: Reunion, 2022

Table 9.1 summarizes trenching statistics by year.

Table 9.1: Trenching Statistics

Year	Trenches Done	Length (Metre)	Average Trench Length (Metre)	Number Samples
2020	25	3,093	124	1,385
2021	41	4,466	109	2,116
2022	13	207	32	235
2023	6	758	126	312
2024	0	0	0	0
Total	**85**	**8,736**	**117**	**4,048**

10 DRILLING

This section includes details on drilling procedures employed at the Oko West Project and a compilation of drilling statistics by drilling type. Drilling and trenching can be summarized into the following types:

- Surface trenching (TR) to gather in-situ surface samples of saprolitic and alluvium / colluvium material.
- Shallow scout Reverse Circulation (RC) drilling focusing on the definition of geochemical targets beneath the duricrust layer.
- Diamond drilling (DD) targeting depth extensions to shallow trenching and RC defined gold anomalies.

Given the relatively recent nature of the discovery of gold mineralization at Oko West, no historical drilling data exists on the Project.

10.1 Drilling Statistics by Year

A summary of the drilling and trenches campaigns performed by the Company is presented in Table 10.1. GMS is not aware of any drillholes that exist on the project before 2020.

Table 10.1: Drilling (DD-RC) and Surface Trenches Conducted on the Project by Year, Up to February 7, 2024.

Period	Type of Hole / Trench	Number of Holes / Trench	Total Length (m)	Total Assayed Length (m)
2020	DD	3	462.0	462.0
	TR	25	3,092.6	2,739.0
2021	DD	54	6,644.0	5,443.0
	RC	109	8,686.0	8,594.0
	TR	41	4,465.7	4,029.7
2022	DD	164	42,658.2	33,867.7
	RC	199	16,651.5	16,401.0
	TR	13	419.0	411.0

Period	Type of Hole / Trench	Number of Holes / Trench	Total Length (m)	Total Assayed Length (m)
2023	DD	182	70,547.7	36,960.9
	RC	1,436	26,550.0	19,072.0
	TR	6	758.0	624.0
	Wedge (DD)	17	6,039.9	2,917.6
2024	DD	11	4,525.8	2,550.5
	RC	16	1,038.5	1,022.0
	Wedge (DD)	2	502.2	399.7
Total	***DD***	**414**	**124,837.7**	**79,284.1**
	RC	***1,760***	***52,926.0***	**45,089.0**
	TR	***85***	***8,735.3***	**7,803.7**
	Wedge (DD)	**19**	**6,542.1**	**3,317.3**
Total		**2,278**	**193,041.1**	**135,494.0**

10.2 Ongoing Exploration Drilling

Exploration drilling was ongoing at the time the Oko West Mineral Resource Estimate was produced. The database cut-off date was chosen to give enough time to the engineering departments to complete their studies for this PEA.

A summary of the drilling and trenches campaigns performed by the Company after the database closure date is presented in Table 10.2. The QP inspected the drilling completed between the database cut-off date and up to June 25, 2024, and does not believe that new information would materially impact the Project. Most of the drilling is aimed at resource conversion.

Table 10.2: Drilling (DD-RC) and Surface Trenches Conducted on Project in 2024, Up to June 25, 2024.

Period	Type of Hole / Trench	Number of Holes / Trench	Total Length (m)
2024	DD	81	24,774.4
	RC	15	315.0
	TR	10	370.8
	Wedge (DD)	19	24,386.5
Total		**125**	**49,846.7**

1.1 General Drilling Procedures

10.2.1 Safety

Pre-start inspections require the drilling operator to undertake the following steps before commencing work:

- Inspect 'critical' safety components of the drill rig.
- Report any faults to ensure prompt repair, distinguishing between those faults that will halt operation at a later time and those that require immediate repair or correction.
- Ensure all crew members are wearing PPEs.

10.2.2 Hole Numbering

Drillhole numbering follows a convention by which each collar must have a unique identifier consisting of the following:

- An alpha prefix denotes the project name (e.g., Oko W.) followed by one (1) letter indicating the drill type (e.g. "D" for diamond, "R" for reverse circulation) and year.
- A dash followed by a numeric suffix (e.g. OKWR22-066).
- The drillhole is sequential from year-to-year, i.e. does not revert to "1" each year.
- Twinned or re-drilled holes have the same Hole-ID as the original with a character suffix (e.g. OKWR22-005A).
- Wedge holes have the same Hole-ID as the parent with an additional suffix consisting of a dash “W” and a sequence number (e.g. OKWD23-239-W2 is the second wedge on parent hole OKWD23-239).

10.2.3 Drill Rig Supervision

The geologist has the authority and ability to perform the following duties:

- Ensure holes are drilled by following the approved drilling program.
- Enforce all safety procedures.
- Visit the drill rig at any time during each shift.
- Issue instructions to the drilling supervisor and resolve disputes.
- Issue instructions and supervise the crew carrying out sampling activities.

- Instruct the driller to cease operations if any of the following occur:
  - Accident or safety incident.
  - Dangerous drilling conditions that could result in loss of equipment or injury.
  - Environmental incident.
  - Excessive deviation detected from down-hole surveys.
  - Excessive and continuous water influx of the hole can result in high sample contamination.
  - Examine the core / RC chips at the planned end of the hole depth.

10.2.4 Drill Site Preparation

The layout of the drill site permits safe drilling operations following these guidelines:

- Locate the approximate position of the collar with backsight and foresight according to the planned azimuth. Allow enough space on the site for all the contractor's equipment, including core storage and sumps.
- Arrange for site preparation equipment (if needed) and ensure that the site preparation is completed as specified before the excavator leaves the drill site. The site must be level to prevent trip hazards.
- Ensure the drill site is not in a hazardous location to workers or equipment, including landslides and falling trees.
- Prepare a piece of wood to temporarily mark the location of the drill collar at the end of drilling and avoid the hole from being covered.

10.2.5 Drill Rig Setup

The drilling supervisor receives instructions detailing the following for each hole: hole number, location of the collar, hole dip and azimuth, expected hole length and methods or downhole survey.

The drill rig is aligned with a Reflex TN14 gyrocompass. Prior to 2023, azimuth alignment was done using surveyed foresight (collar) and backsight (sighter) stakes (pegs) and running a string or laser sight between the foresight and back site to aid in alignment.

The following steps must be followed:

- Collar the hole within one (1) metre of the surveyed collar location.
- The hole dip (inclination) was checked using a clinometer.

- Once the rig setup is complete, obtain another set of handheld GPS coordinates for the drill collar and record this location as the current one in the database.

10.2.6 Drillhole Surveys

Collar Survey: The drill collar monument is surveyed by differential GPS to obtain accurate coordinates and elevation.

Downhole Survey: The hole is surveyed using a Reflex surveying tool (Gyro Sprint-IQ-A4 or EZ Trac A4). Survey measurements are taken either continuously or at every ten metres (10 m) while the instrument is lowered and hoisted. After the survey tool has been hoisted and disconnected, the survey information is extracted by connecting a portable tablet and downloading the data. The test is considered successful after an examination of the results at the drill site.

10.2.7 Environmental Management

Site waste control procedures include the following:

- Washing and servicing equipment, cleaning hydrocarbon or other chemical spills, and waste disposal is part of the responsibilities of the drilling contractor.
- Waste hydrocarbons, empty hydrocarbon containers and chemical containers must be placed in leak-proof receptacles and disposed of at the nearest designated hydrocarbon containment area.
- If fuel or oil spillage occurs at the drill site, the driller immediately notifies the geologist in charge, who advises collecting and disposing of the contaminated rock or soil. Clean-up of spills is at the contractor's cost if not otherwise specified in the drilling contract.
- Waste generated by drilling operations is placed in suitable containers and disposed of at a designated trash disposal site.
- The drillers will contain water generated from drilling operations in sumps at the drill site.

Site-specific rehabilitation procedures include the following:

- Removal of waste and equipment.
- Plugging of drillhole collars.
- Backfilling of sumps.
- Removal or scarifying of sample rejects from splitters.

- Clean-up of all fuel and oil spills.
- Removal of all bulk bags from the site location.

10.2.8 Recovery and Re-drilling a Hole

A drillhole is re-drilled when:

- The sample order has been lost, and sample quality and recovery have been compromised.
- Excessive deviation from the planned drillhole trace.
- The hole did not reach its target depth because drilling equipment (rods, hammers, or bits) lost down-hole could not be recovered.
- The contractor informed the geologist of dangerous drilling or poor hole conditions that hindered further progress.

The decision to re-drill a hole is also based on discussions with the contractor's on-site representative.

10.3 Diamond Drilling

Drilling by Reunion first commenced at the Oko West Project on December 9, 2020, and at the time of writing, eight (8) diamond drill rigs are on-site.

The following diamond drill rigs have been used:

- One (1) Longyear 90.
- One (1) Sandvik 740.
- Six (6) Sandvik 710’s (Figure 10.1).

Figure 10.1: Sandvik 710 Diamond Drill Rig at the Oko West Project

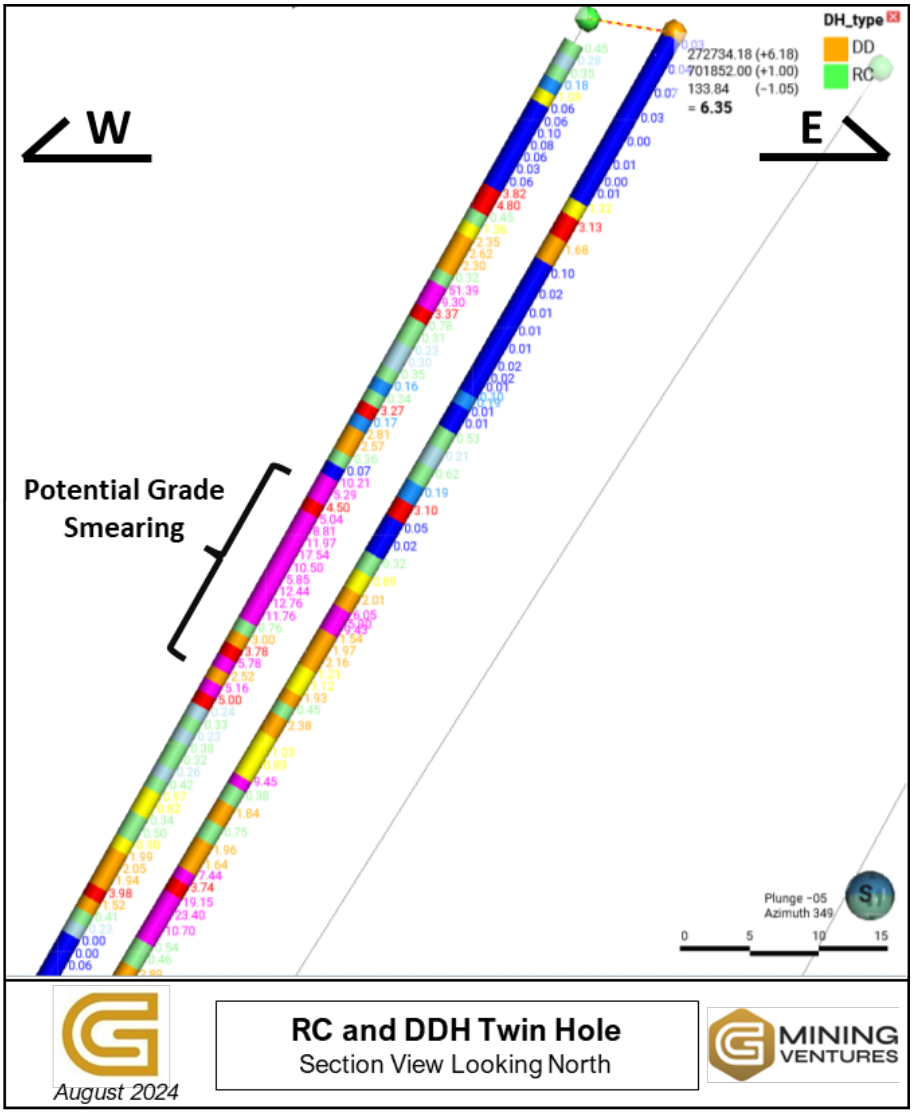


Source: GMS, 2024.

From the project outset, both diamond and reverse circulation drilling were undertaken simultaneously. Diamond drill core provides excellent geological information both in saprolite and unweathered rocks. RC drilling provides poorer-quality geological information but is a fast, cost-effective, and reliable sampling tool for the lateritic profile. At the initial stages of the program, RC drilling was used to scout for gold mineralization in the saprolite and delineate the footprint of mineralization near surface. The resulting gold anomaly was followed up with diamond drillholes that tested not only the mineralization's lateral extent, but also depth continuity. Diamond drill core demonstrated that several initial RC holes were drilled sub-parallel into the mineralized envelope at a sub-optimal angle. As the project evolved, RC holes were restricted to the shallow western edges of the gold anomaly and subsequently entirely replaced by diamond drilling as resource delineation drilling started.

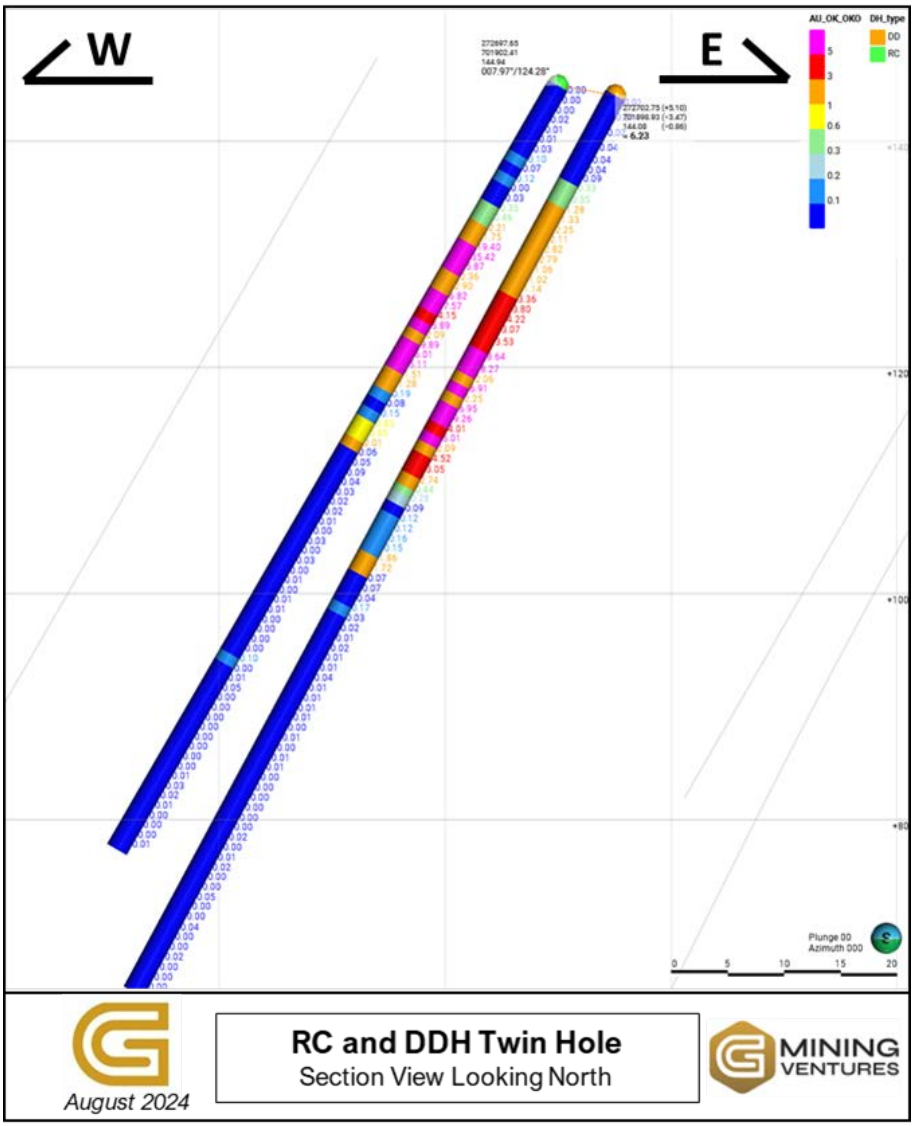
A total of 16 diamond twin holes were drilled to validate the early RC assays. Only one (1) discrepancy was acknowledged between RC OKWR22-128 and diamond twin hole OKWD22-127. The RC hole was flagged due to potential grade smearing and removed from the mineral estimation database. Figure 10.2 presents the mineralized intervals from the RC hole with the twin diamond hole. Figure 10.3 presents an example of RC and DD twin holes assayed intervals that correlates well together.

Figure 10.2 : RC OKWR22-128 and Diamond Twin Hole OKWD22-127



**Note: The potential grade smearing observed in the RC hole.*

Figure 10.3: RC OKWR21-019 and Diamond Twin Hole OKWD22-125.



**Note: The similar grade ranges observed in both RC and DD twin hole.*

Diamond drill core recovery is excellent in unweathered rock, often greater than 95% but poorer in RC drilling, with a sample recovery of an average of 68%. Table 10.3 presents the sample recovery statistics by type of drilling and weathering.

Table 10.3: Drilling Recovery by Weathering Domain

Type of Hole	Weathering Domain	Num. of Measurements	Avg. Sample Recovery %
DD	Alluvium / Colluvium	1,032	84.6
	Saprolite	17,421	91.3
	Transition	6,671	92.4
	Fresh Rock	91,812	98.2
RC*	Alluvium / Colluvium	1.681	48.2
	Saprolite	9,310	72.5
	Transition	3,822	71.2
	Fresh Rock	6,208	65.4

**Note: RC drilling recoveries are an approximation*

All diamond drilling, regardless of the drill rig, followed these procedures:

- Drilling azimuths have been consistently targeted at 270 degrees, except for a few holes designed to intersect interpreted cross-cutting structures and drilled at due north or south.
- Drilling dips of most holes have been designed to start at -60 degrees. A few in-fill holes had different initial angles, and deeper drilling uses higher angles (up to -80 degrees) due to surface constraints.
- The entire lateritic profile is drilled with HQ core size (63.5 mm core diameter), changing to NQ size (47.6 mm core diameter) once in unweathered rocks.
- The drill core is placed in core boxes with pertinent Hole-ID, drill run and depth information at the drill site and transported by the drilling company to the camp core shed for logging and sampling.
- Drillholes are always initiated and stopped in the presence of a project geologist.
- The drilling crew will perform drill core orientation in unweathered rocks as often as practically possible.
- The drill crew will perform down-hole surveys using the prescribed equipment at the end of each hole.

10.3.1 Wedging

In diamond drilling, wedging is a procedure used to change the direction of the borehole using permanent or retrievable wedge. At Oko West, wedging is used to ensure specific targets are reached when deviation is observed in drill holes. Wedging is also used to create secondary holes branching off from a single ‘Parent Hole’’, reducing the meterage needed for resource conversion during infill programs.

A summary of the wedging procedure performed by Major Drilling is described below:

- Pre-installation checks and preparation for wedge installation.
- Off-Bottom wedges and wedges placement.
- Reaming and core drilling.
- Downhole survey to verify wedges position.
- Proceed with normal drilling operations.

10.3.2 Down Hole Motor (DHM)

For hard-to-reach targets, Down Hole Motor (DHM), has been implemented in late 2023 at Oko West.

The DHM simplified procedure is as follows:

- Preparation and maintenance.
- Bypass wedge installation and hole cutting.
- DHM handling and testing.
- Drilling and testing, using Reflex tools for direction.
- Drilling verifications for direction.
- Reaming and resuming drilling.

10.4 Reverse Circulation Drilling

Reverse circulation drilling started on July 26, 2021, and continues using a Maxidrill 400 mounted on a Cat 315DL base (Figure 10.4) using four-inch drill pipes and a four-inch DTH hammer. It uses a Metzke cyclone with two (2) cone chutes and a rotary splitter attachable to the cyclone. A Sullair 1150 / 1350 CFM at 500/350 psi compressor mounted on a steel tracked carrier is occasionally supported by an Atlas Copco B4-41 booster. Major Drilling owns and operates this rig.

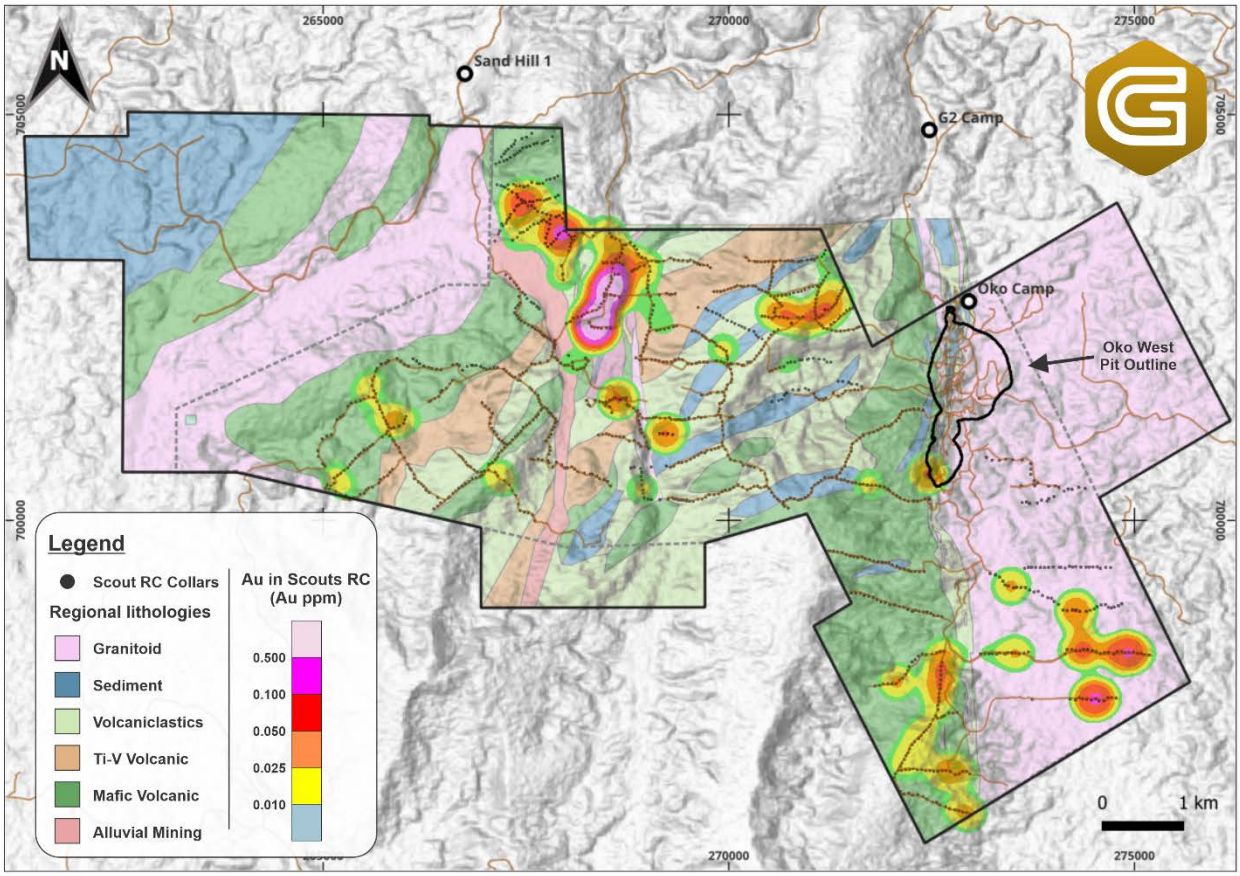
On November 7, 2022, Major Drilling mobilized a scout rig (ED-167) mounted on a Morooka MTS 850, capable of rotary air blast, air core and reverse circulation drilling to depths up to 100 m. This rig is being used for reconnaissance scout drilling of geochemical anomalies (Figure 10.5).

Figure 10.4: Reverse Circulation Rig (compressor and booster not shown)



Source: Reunion, 2022. Photo by C. Bertoni

Figure 10.5 : Map of Scout RC Program with Anomalies on Lithology



Source: GMIN, 2024

The drill rig has an onboard sampling system consisting of a rotary splitter attached to a Metzke cyclone (Figure 10.6). Where drilling has not intersected water and the sample is dry, each primary sample is:

- Material for an entire interval is weighed to estimate recovery. The recovery is calculated based on the theoretical mass drilled every metre by weathering status.
- Three (3) bags of samples are collected from the splitter. Samples from these bags are then remixed and shaken six (6) times in two (2) directions for homogeneity.
- The splitter on the rig separates the actual samples for lab processing, the field duplicate sample and the bulk sample for storage and future reference.
- All primary sample bags and bulk sample bags are marked with a unique sample number identifier. A pre-printed sample tag with the corresponding number is placed in the primary sample bag.
- The sample is weighed to ensure a mass of approximately 2 kg.
- Duplicate samples are placed in bags with corresponding hole numbers written on them, returned to camp and placed in an appropriate storage location.

Figure 10.6: Rotary Splitter Attached to Metzke Cyclone



Source: Reunion, 2022

For each primary sample, the corresponding reject sample is:

- Sub-sampled, sieved, and two (2) representative samples are taken for the chipboard and chip tray.
- Logged noting geologic features and photographed.

The following drilling and sampling practices are followed to minimize contamination between samples:

- Drilling stops (5 – 15 seconds) while air circulation is maintained to flush out all remaining material at the end of each sample interval. This time is occasionally reduced when drilling in a zone with a high influx of water, reducing contamination.
- The splitter is cleaned thoroughly after each rod change (6 m). In case of water influx, this cleaning frequency might be changed.
- The inside of the cyclone and the collection box are thoroughly cleaned after each drill rod change when drilling in fresh rock and, more often as required in weathered rock or when drilling under wet conditions.
- Sample bags are tied off immediately after filling to prevent contamination using a cable-tie.

The intersection of groundwater is expected but may be avoided by using an auxiliary compressor and booster. Where wet samples are unavoidable due to groundwater inflow, the following process is employed.

- Connect the booster at the first sign of moisture, ensuring that subsequent samples do not suffer contamination and recoveries improve.
- Collect a preliminary sample split (sub-sample) by spearing in six (6) different directions through the bag with a scoop or available spatula.
- The geologist should abandon the hole if water is encountered for three (3) consecutive metres before the planned hole depth is reached.

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

This section describes the sample preparation, analysis, and security procedures for the diamond drilling (DD), reverse circulation drilling (RC) and trenching programs performed on the Oko West Project. It also includes a quality assurance and quality control (QA/QC) program as part of the sample assaying process.

11.1 Core Handling and Sampling

A total of 414 DD, 1,760 RC, 19 wedges of DD holes and 85 trenches were completed between January 2020 and February 2024. The following section describes the procedures for core handling, logging, and sampling implemented by the Company on the Oko West Project.

Drill rigs are aligned with the planned drillhole information and inspected by the project geologists. The hole number, azimuth, and dip are marked at the collar of the drillhole.

Core boxes for each drillhole are labelled with a unique number consisting of an alpha prefix, usually denoting project name (OKW) followed by one (1) letter indicating the drill type (“D” for diamond, “R” for reverse circulation) and year (e.g. OKWD22-067). Twinned or redrilled holes have the same Hole-ID as the original with a letter suffix (e.g. OKWR22-005A).

Numbered boxes containing drill cores are transported to the Company logging core shack. Geotechnical measurements are then conducted; recovery, RQD, orientation marks, core diameter and hardness are recorded.

Geological logging and structural measurements are taken by the geologist in charge of the drillholes. Data is entered directly into the drillhole master database. For the core logging, the following data is retained and described: lithology, alteration, structure, mineralization, and sample intervals.

Diamond drill core samples are collected on average at every 1.3 m from drill core but vary between 0.1 m and 2.85 m. Sample intervals are marked by geologists. Samples are selected in potential mineralized zones based on logged geological features, such as rock type, mineralization, alteration, veining etc.

Blanks, certified reference materials (CRMs or standards), and duplicates are inserted at the same time as the sampling process is performed.

The core boxes are then photographed, with details of each core box clearly visible. Digital photographs are taken dry and wet and kept archived for review purposes.

The core cutting is supervised by the geologists and geotechnicians to ensure that the samples are cut in order, and the sample number vs the depth are not mixed up. Samples are marked along the cutline with a different colour from the core-orientation line. The core is cut from the bottom to the top to avoid combining with the sample below. The same side of the core is always put in the sample bag. The side with the orientation line must be kept in the core box, while the other side goes into the sample bag (Figure 11.1).

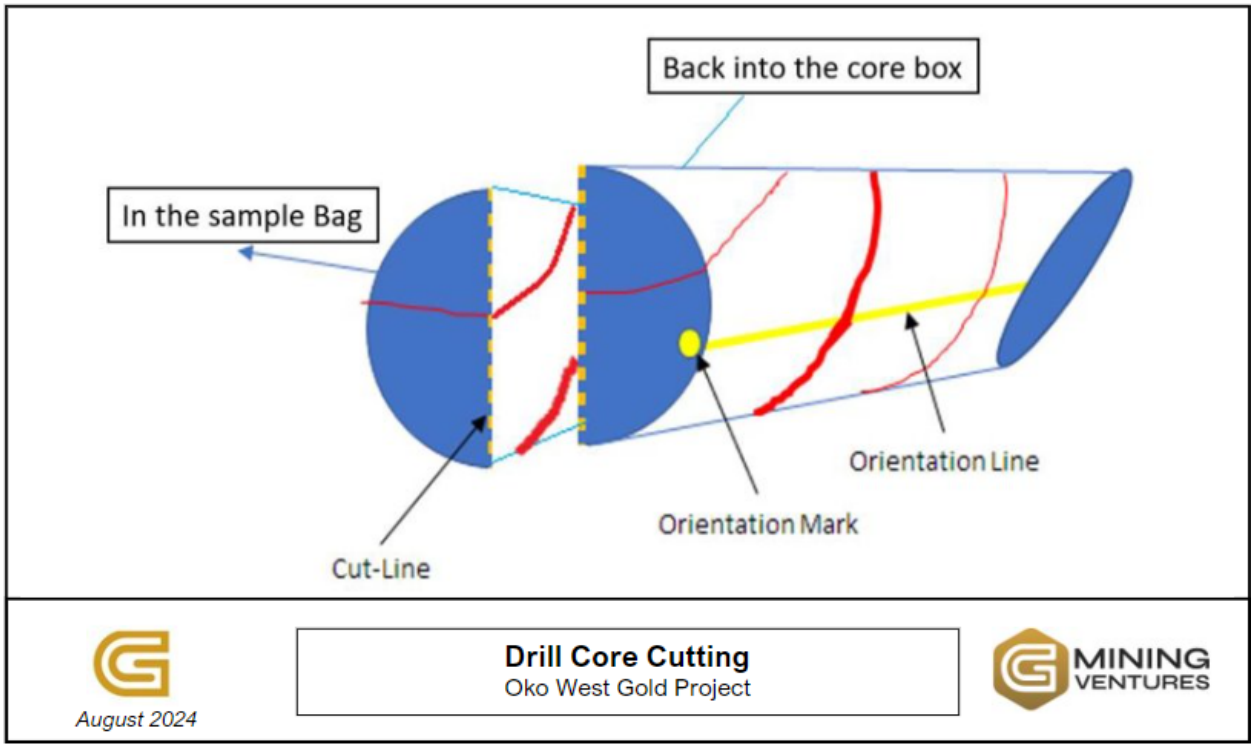
The second stub of the sample tag remains stapled in the box at the end of the sample interval to identify the sample length and location The third stub and a metal tag bearing the same sample ID is inserted into the corresponding sample bag with ½ the core.

In the case of field duplicates, the two sample stubs will be stapled side by side in the box. The primary sample is always ½ core. For a ½ core duplicates, the second half of the core goes into the second sample bag, leaving no core in the box. For ¼ core duplicates, the remaining half will be cut in half again. The ¼ goes into the secondary sample bag and ¼ is kept in the core box.

Bags are closed with zip-ties immediately after the sample is placed in the bag. Samples are placed in order in the core shed outside the cutting station. Samples from different holes are kept separate. Drill core samples, blanks, and certified standards are then bagged and ready to be submitted to the primary laboratory.

The laboratory is instructed to return the rejects with the sample stubs in the bags even if there is no sample in the bag (e.g. bags containing a standard).

Figure 11.1: Drill Core Cutting – Oko West Gold Project



Source: Reunion, 2022

11.2 RC Sample Handling and Sampling

Reverse circulation (RC) chip samples are collected at every meter. RC drilling protocols and contamination mitigation measures are described in Section 10.

The splitter on the rig will produce 2 kg samples for the primary lab, the field duplicate sample and the bulk sample for storage and future reference. All primary sample bags and bulk sample bags are marked with a unique sample number identifier. A pre-printed sample tag, corresponding number, is placed in the primary sample bag.

Duplicate samples are placed in bags with corresponding hole numbers written on them, taken back to the exploration camp, and placed in an appropriate storage location. Rejects are sub-sampled, sieved and two (2) representative samples are taken for chipboard and chip tray to be logged noting geologic features and photographed (Figure 11.2).

Figure 11.2: Chip Logging Geologist



Source: Reunion, 2022. Photo by C. Bertoni

For the RC samples, the second and third stubs are inserted in the sample bag, still attached to each other. The laboratory will detach these, placing one in the coarse reject bag and another in the pulp reject bag.

11.3 Sample Transit, Security and Chain of Custody

The sample tags are kept in a locked cabinet and issued to the core shed personnel on an as-needed basis. The sample tag stubs are completed, and the books used are kept in a sample book library for future reference. Sample tags contain the sample information, including date, target, hole or trench, interval from-to in metres, sampler name and analytical code.

Samples submitted display the sample number and are individually tied with plastic tags and packed in rice bags (six samples per rice bag), which lists the project initials (Oko West) and the batch number (photo below). The rice bags are also securely tied (Figure 11.3). Access to samples is only possible by cutting the tag. The laboratory is instructed to recycle the sample bags, return the rejects on the same bags, and

show the original batch numbers. In the case of damaged bags, the lab is to replace them, re-writing the same labels on a new bag.

Figure 11.3: Bagging



Source: Reunion, 2022. Photo by C. Bertoni

The proper storage of sample rejects received from the lab is essential for a proper QA/QC system; thus, assuring the integrity of samples and their easy retrieval. The laboratory must return rejects quoting their original batch or work order numbers, as these rejects will be stored by batch groups. Laboratories create their own job numbers, which are not used for storage records.

Pulps are stored in Kiva plasticized cardboard NQ core boxes by a batch of 40 samples (photo below) and stored safely in locked sea containers (Figure 11.4).

Coarse rejects are stored in raffia bags at the reject library on shelves by batch groups.

Figure 11.4: Pulp Rejects Storage



Source: GMS, 2024

At the beginning of the project, the samples are sent via boat and truck to the primary laboratory in Georgetown (Guyana), accompanied by one (1) company employee for the entire trip to witness that all samples reach the laboratory safely. Now, the samples are sealed on site, loaded on the sample truck, and sealed to the truck tarp. The Truck driver signs a manifest on site and the lab confirms receipt of the truck with tags intact upon delivery. Along with the requisition form, a receipt form is submitted to the lab, listing the samples included in the batch and containing space for the signature of a lab representative at the time of delivery. The lab acknowledges the receipt of a work order by e-mail to the company. The primary laboratory, Activation Laboratories Ltd. (Actlabs) sends pulp duplicates directly to the secondary laboratory (MSA Labs) for umpire check assays.

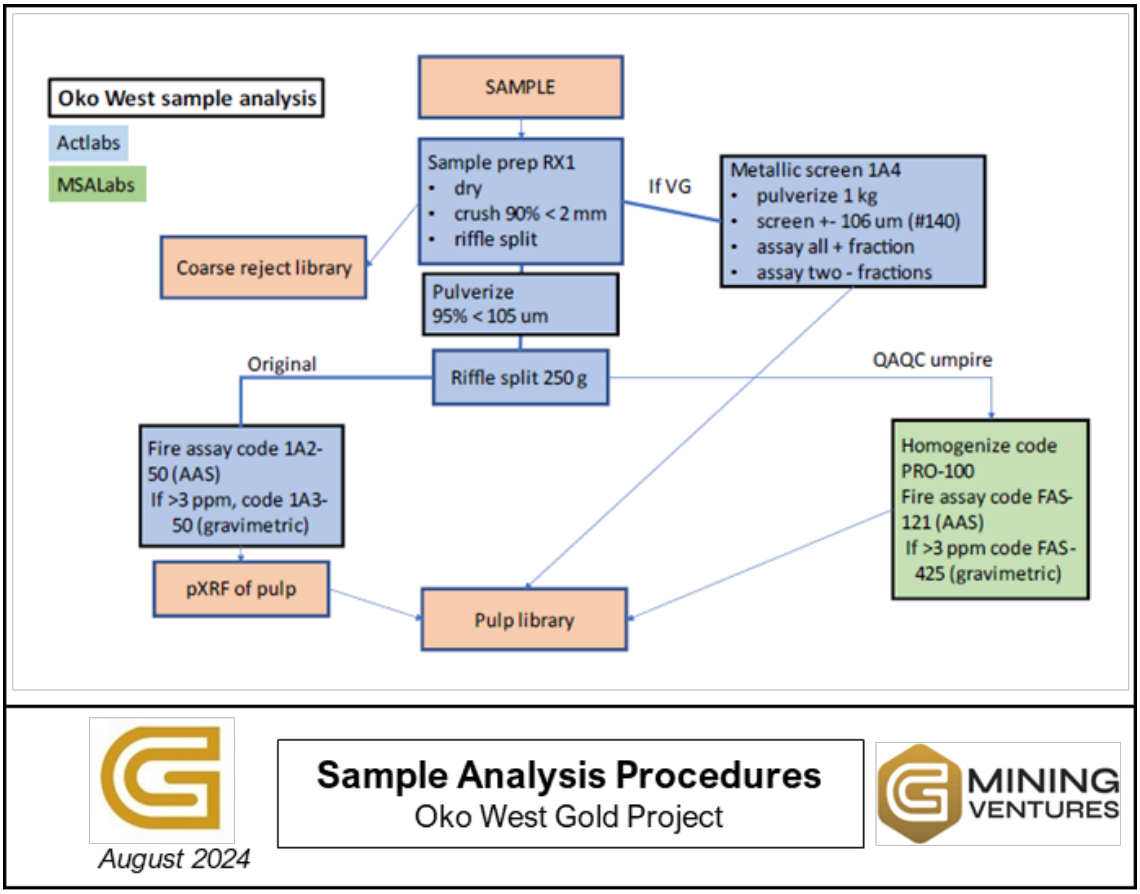
11.4 Sample Analysis Methods

Sample batches are prepared following the Actlabs Code RX1 procedure. Samples are weighed, dried, crushed (<5 kg) to a fineness of 80% passing 2 mm. A riffle split of 250 g is taken from the crushed material and pulverized (mild steel) to 95% passing 105 µm (140 mesh). The laboratory technician uses sand to clean in between each sample.

At Actlabs, gold analysis code FAAA-1A2 is performed using a 50 g fire assay (FA) with atomic absorption spectrometry (AAS) finish. For gold values above the upper detection limit (> 3,000 ppb), samples are assayed by fire assay with a gravimetric finish (FAGRA-13A). If visible gold is observed by the geologist during the logging and sampling, the analytical method 1A4 Au fire assay metallic screen is prioritized, and the sample before and after the visible gold are also analysed using the metallic screen method.

At MSA Labs, pulp rejects samples from the primary laboratory are homogenized by mat rolling 100 times (code PRO-100) and gold assaying is performed by fire assay with AAS finish of 50 g pulverized material (FAS-121). If the gold value is above the upper detection limit (> 3 ppm), then a gravimetric finish is used (FAS-425). Figure 11.5 illustrates the analytical process selected to assay the Project gold samples.

Figure 11.5: Sample Analysis Procedures Schema as Implemented at Oko West Gold Project



Source: Reunion, 2022

11.5 Data Management

The assay reports by both the primary and secondary labs are distributed by e-mail directly to recipients listed in the work order, including gDat Solutions (www.gdatsolutions.com), a third-party, independent database manager. The report includes a csv file and a pdf file, both containing the same information. The report includes the assay results for samples and CRM submitted and results for the lab CRM used. gDat imports the QA/QC results into acQuire software and runs a routine that produces graphs and statistics for each QA/QC sample, indicating if they "passed" or not the QA/QC criteria. A report is sent by gDat via e-mail to the Company senior geologists, who review the results. If results are approved, gDat is informed by e-mail and incorporates the results into the database. If not, Reunion will ask the lab to re-run the samples until the results are approved. Only when approved by the project geologists, the assay results will be incorporated into the database by gDat. All the e-mail correspondence is kept on record.

gDat Solutions use the acQuire Geoscientific Information Management (GIM) with complete independence. They are the only personnel authorized to introduce database changes and make copies available to

third-parties. Project geologists can consult the database and download its contents via open database connectivity (ODBC). No personnel have direct access to the database.

11.6 Density Measurements

Bulk density measurements are taken in-house on all representative cores from the lithological intervals, including both mineralized and non-mineralized units, with varying degrees of hydrothermal alteration and weathering. The frequency of density measurements is determined by the project geology team. To determine the dry bulk density of a sample, the water displacement method is used. The sample is assumed to displace a volume of water equal to its own bulk volume. The sample, a piece of dry and clean drill core, is weighed to determine the dry mass using an electronic scale which is connected to a suspended basket immersed in water. The entire piece of the core is submerged in the water and weighed again. Saprolitic core samples are tightly wrapped with a thin film of plastic and weighted. The sample is immersed in water within a suspended basket and weighed again (Figure 11.6).

Figure 11.6: Density Measurements Balance Setup



Source: Reunion, 2022

11.7 Quality Assurance and Quality Control (QA/QC) Procedures

The Company's exploration team has implemented a rigorous quality assurance and quality control program (QA/QC) to ensure the validity and integrity of its sampling procedures and data management. The QA/QC protocol applies to all drill holes (DH and RC) and trench samples.

The QA/QC protocols were designed to certify drill holes sample collection, analysis, and data management and were designed in accordance with CIM Mineral Reserve and Mineral Resource Best Practice Guidelines (CIM, 2019).

The QA/QC program adopted for all drill and trench sampling is sent to the primary lab in batches corresponding to an entire trench or drillhole. Samples from different trenches or drill holes are not mixed-up in the same submission order. The insertion of control samples is distributed among the original drillhole or trench samples in each batch. Certified reference materials (CRMs or standards) and blanks include one (1) control sample every ten (10) samples interchanging between standard and blank, or the equivalent of one (1) blank and one (1) standard at every 20 samples (5%). The position of blank and standard samples is adjusted to control mineralized intervals and test lab contamination.

The Project geologists request the primary lab to generate a duplicate pulp sample at a frequency of one (1) in 15 to 20 samples (about 5%) from expected mineralized and unmineralized intervals. The primary lab produces these samples and forwards them to the secondary lab for check or umpire assays, including additional standards provided by the Company.

The Company procedures of quality control (QC) samples are designed to insert one (1) standard, one (1) blank, and one (1) field duplicate at every 20 samples generated by drilling.

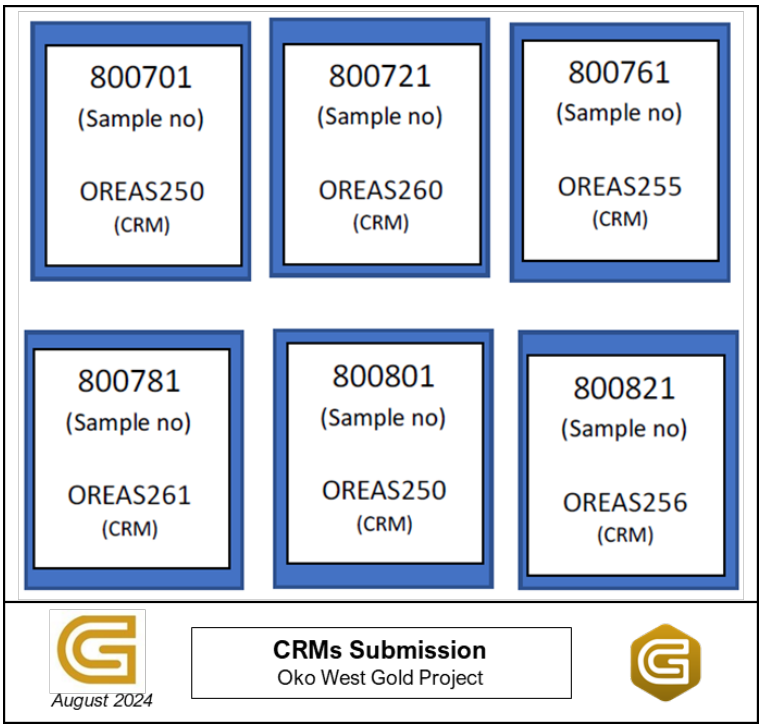
Field duplicates are taken from RC drilling. One (1) field duplicate is introduced at every 20 samples generated by drilling. This duplicate is collected directly at the rig's Metzke splitter.

For the trench channel sampling, one (1) standard, one (1) blank and one (1) field duplicate are inserted at every 20 samples generated by sub-horizontal channels near the base of the trench. The field duplicate is collected directly at the channel by enlarging it.

For diamond drill holes (DH), two (2) half core field duplicates are inserted into each mineralized zone, providing the zone is long enough. At the end of 2023, this procedure was updated to require ½ core duplicates in infill drilling (40 m spacing) and ¼ core duplicates in exploration drilling (80 m spacing or greater).

All CRMs being inserted are photographed with their respective sample numbers before the CRMs (Figure 11.7) identification is erased, thus documenting the insertion for eventual inspection or audit.

Figure 11.7: Identification of CRMs Before Submission to Laboratory



Source: Reunion, 2022

GMS reviewed the analytical quality control data produced by the Company for the 2020, 2021, 2022, 2023 and 2024 drilling programs to confirm that the analytical results were reliable.

Data relating to blanks, certified reference materials (CRMs), field, coarse and pulp duplicates and umpire check assays data were received from Reunion in comma-delimited format spreadsheets. From September 10, 2020, to February 07, 2024, a total of 5,570 blanks, 5,901 CRMs and 5,550 duplicates were submitted to Actlabs. The control samples represent approximately 14% of the total number of samples submitted for assaying. These totals include only final and passing batches. Analyses of data from CRMs and blank samples are normally illustrated in time-series plots to identify extreme values (outliers) or trends that may indicate issues with the overall data accuracy and precision.

11.7.1 Blanks

To monitor any contamination during the sample preparation and assaying at the laboratory, blank materials are inserted into the sampling stream every 20th sample.

The blank used in the drilling and trenching programs was sourced from a barren sample of crushed granite. The Company uses a coarse blank composed of a granitoid rock aggregate from the Toolse and Teperu quarry in Guyana (Figure 11.8). The bulk coarse blank material is kept in a covered plastic bucket at the site core shed, and the blank sample is prepared in advance by placing 1 kg of the material in a zip-lock bag. This material was recently re-tested for its gold content (work order 22-OKWG-198).

Figure 11.8: Blank Used by Company as Quality Control Sample

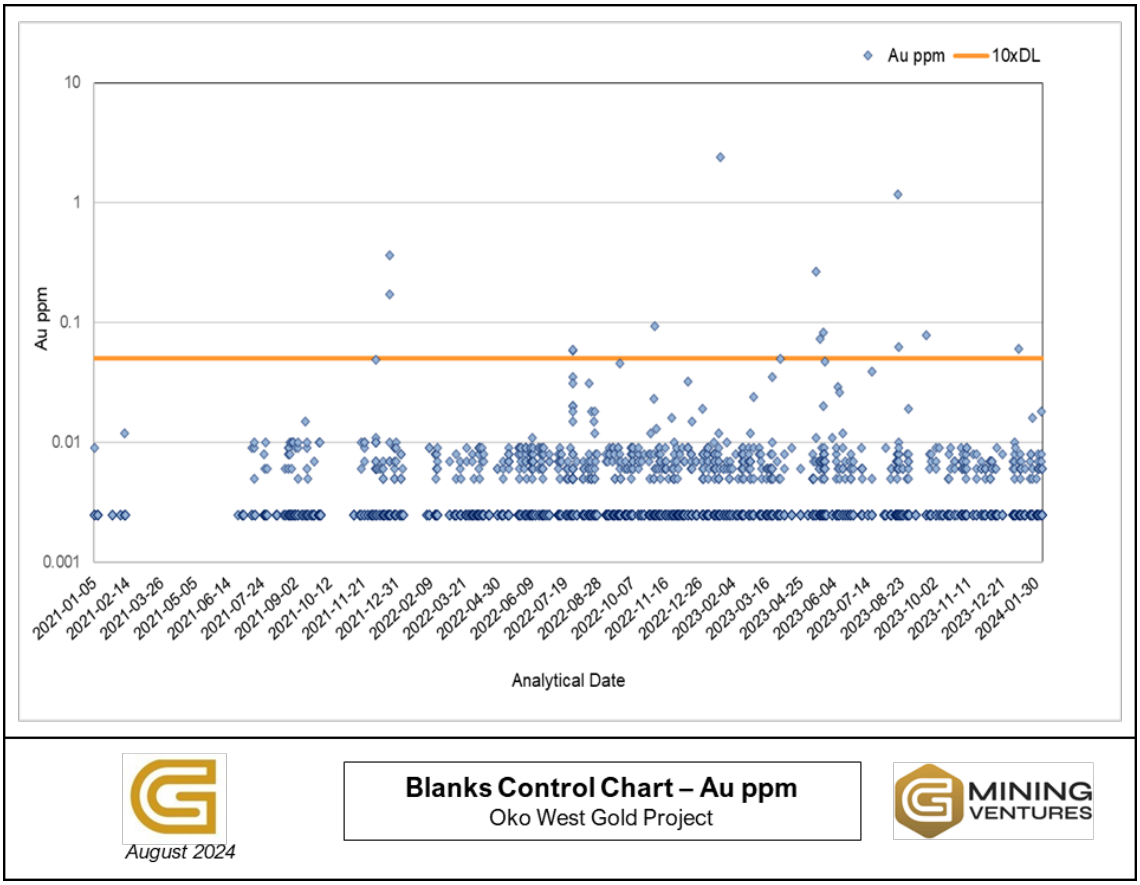


Source: Reunion, 2022

Blank materials are considered failed when the returned gold value exceeds 0.05 ppm, which is equivalent to ten (10) times the lower detection limit (DL) of 0.005 ppm for both FAAA-1A2 and FAGRA-1A3 analytical methods. A total of 5,570 blank samples were submitted to Actlabs. The results are considered good, with 99.8% of blanks falling within the accepted control limit, which demonstrates that the samples show no systematic contamination with contiguous mineralized intervals. Some of the failures seem to be related to mislabels and the insertion of CRMs instead of the barren material.

Figure 11.9 shows the performance of the blank samples and the control limits (10 times DL).

Figure 11.9: Blanks Control Chart for Gold – 2021-2024 Drilling Program



Source: GMS, 2024

11.7.2 Certified Reference Materials

Certified reference materials (CRMs or standards) were chosen within low and high gold grade ranges and from two (2) types of material, oxide, and sulphide, used to monitor the laboratory performance and to assess bias on the analytical results. The Company purchases commercial CRMs prepared by Ore Research and Exploration (Australia) (OREAS) and Canadian Resource Laboratories. The CRM packets are kept in clean plastic containers at the field geology office and brought to the core shed only when needed to be introduced to the sample batches.

Quality control (QC) samples are classified as failures for a reference material if the results are outside ± three (3) standard deviations (SD) of the certified gold value. Failures observed in the CRM results are investigated and replaced by the re-analyzed value when it is considered necessary. Overall, the results tabulated below indicate that the CRMs are good and minimal bias regarding the precision of the expected values vs. the assayed value is present through the assaying period.

OREAS 250b, 260 and CDN-GS-16 were removed from the quality control sampling program by the Company due to poor analytical performance at the primary laboratory, the values obtained often being between the control limits, systematically different from the certified value over several periods.

The results of the analytical quality control data produced by the CRM samples used between 2020 and 2024 drilling programs and submitted to Actlabs is summarized in Table 11.1.

Table 11.1: Summary of CRM Performance Results as Implemented on Oko West Drilling Resource Area (2020-2024) – CRM Samples Assayed by Actlabs

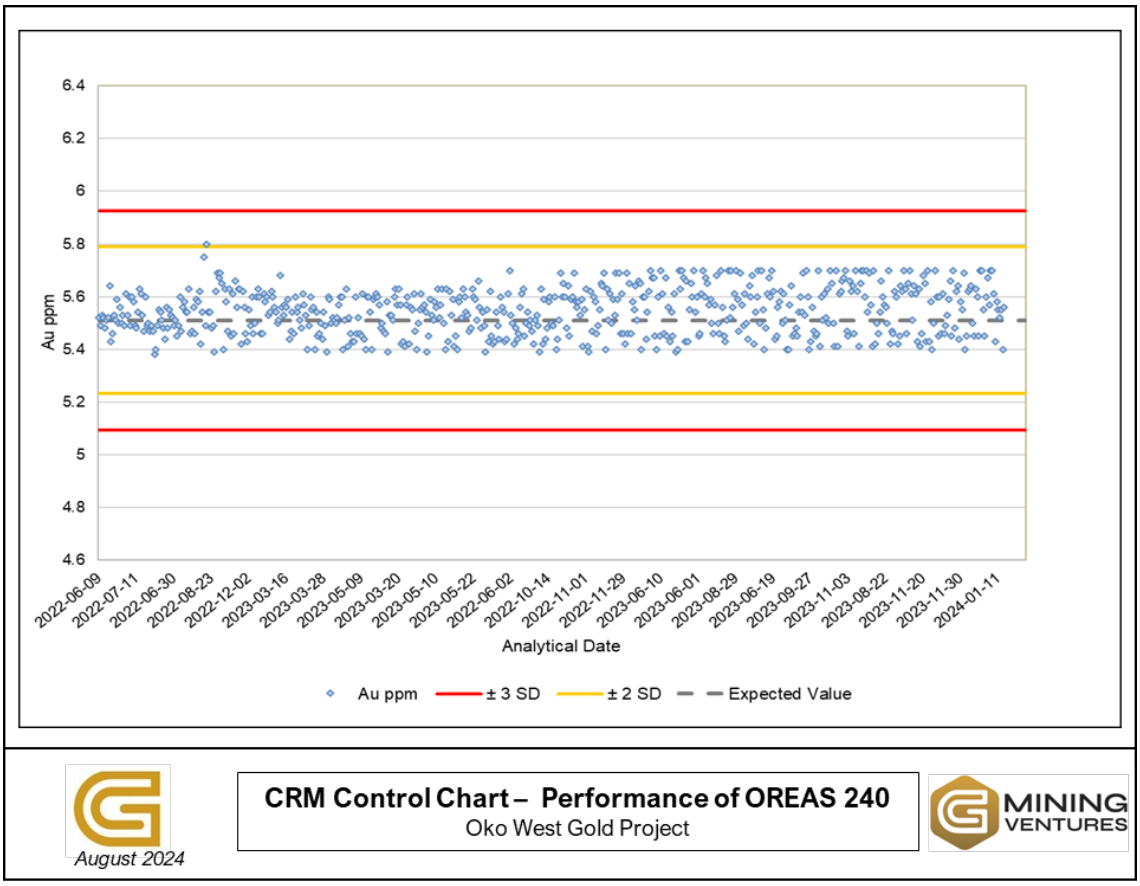
Material Type	Certified Standard (CRM)	Num. of Submitted CRM	Certified Au Value (ppm)	Standard Deviation (SD)	Num. of Failures	% Passing
Sulphide	CDN-GS-16*	11	16.48	0.315	0	100.0%
Sulphide	CDN-GS-1U*	102	0.968	0.043	0	100.0%
Sulphide	CDN-GS-P1A*	140	0.143	0.004	1	99.3%
Sulphide	OREAS211	937	0.768	0.027	0	100.0%
Sulphide	OREAS221*	152	1.06	0.036	1	99.3%
Sulphide	OREAS234	150	1.2	0.03	0	100.0%
Sulphide	OREAS236	905	1.85	0.059	0	100.0%
Sulphide	OREAS238*	83	3.03	0.08	0	100.0%
Sulphide	OREAS239*	84	3.55	0.086	2	97.6%
Sulphide	OREAS240	619	5.51	0.139	0	100.0%
Oxide	OREAS250*	126	0.309	0.013	0	100.0%
Oxide	OREAS250b	398	0.332	0.011	1	99.7%
Oxide	OREAS251b	333	0.505	0.017	1	99.7%
Oxide	OREAS252*	121	0.674	0.022	0	100.0%
Oxide	OREAS252b	238	0.837	0.028	0	100.0%
Oxide	OREAS254*	217	2.55	0.076	0	100.0%
Oxide	OREAS254b	404	2.53	0.061	0	100.0%
Oxide	OREAS255*	57	4.08	0.087	0	100.0%
Oxide	OREAS256*	64	7.66	0.238	0	100.0%
Oxide	OREAS256b	200	7.84	0.207	0	100.0%
Oxide	OREAS258*	43	11.15	0.259	0	100.0%
Oxide	OREAS260	157	0.016	0.0018	2	98.7%

Material Type	Certified Standard (CRM)	Num. of Submitted CRM	Certified Au Value (ppm)	Standard Deviation (SD)	Num. of Failures	% Passing
Oxide	OREAS261	354	0.0486	0.0023	0	100.0%
Oxide	OREASH1*	6	0.012	0.001	2	66.7%

**Note: Discontinued CRMs*

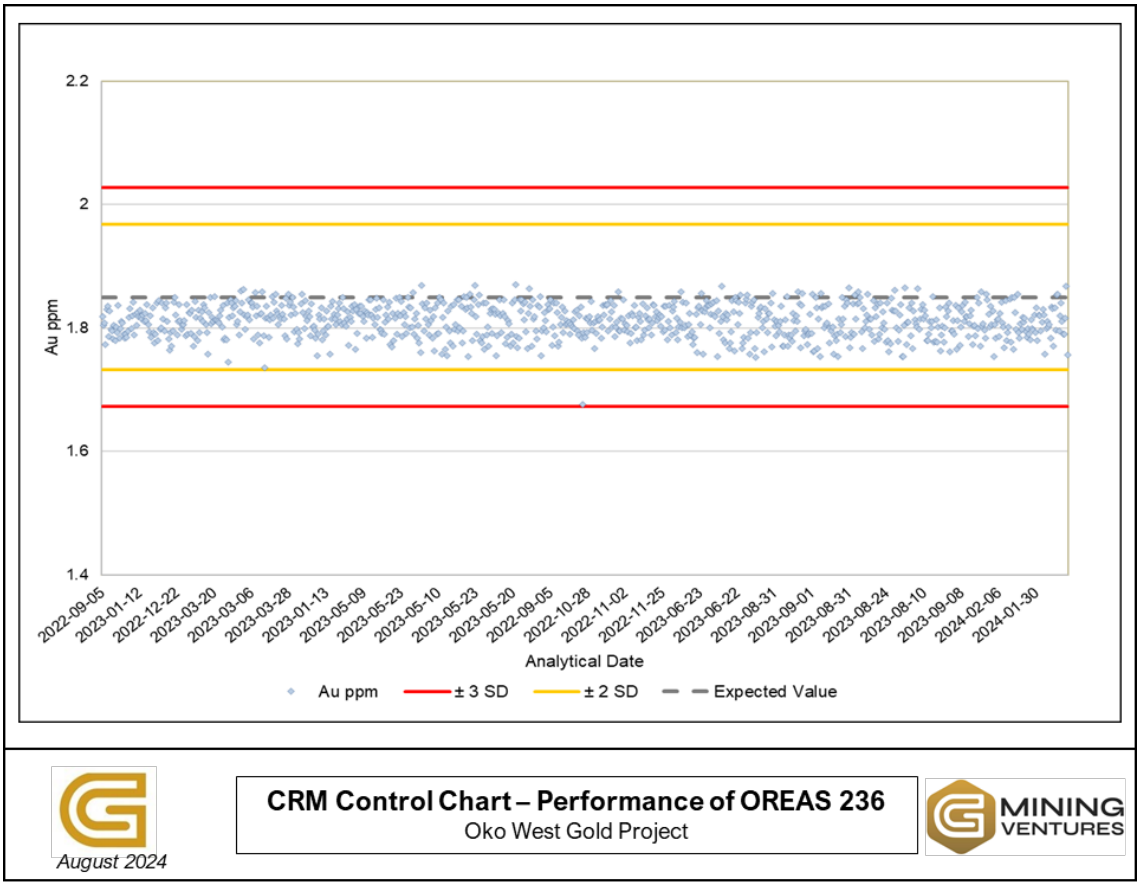
The performance of the CRMs was also validated on time-series control charts to monitor for analytical drift and abnormal assay batches. After verification, the CRMs generally performed well over time with the majority of the results within the control limits of ± 3 times standard deviation (± 3SD) of the certified recommended value. Some of the control limits of CRMs are illustrated in Figure 11.10 to Figure 11.14.

Figure 11.10: Sample Control Chart of CRM OREAS 240



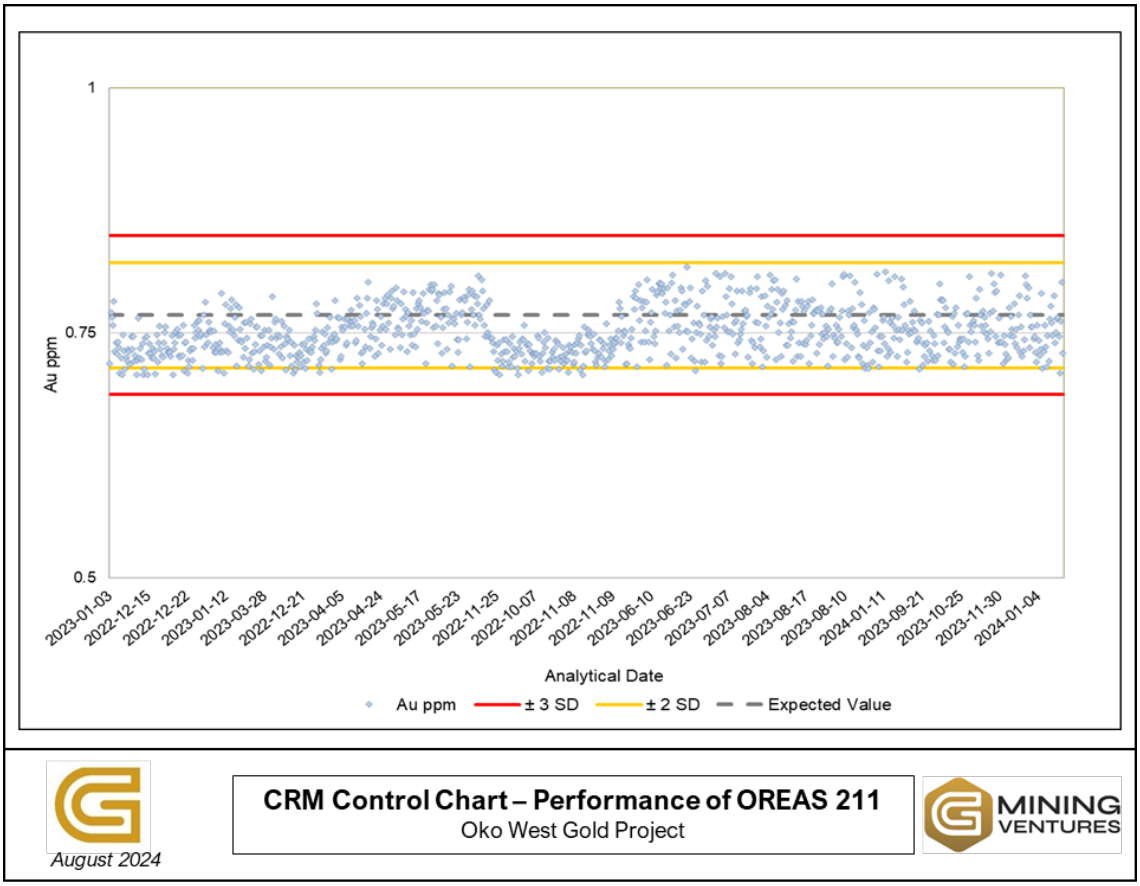
Source: GMS, 2024

Figure 11.11: Sample Control Chart of CRM OREAS 236



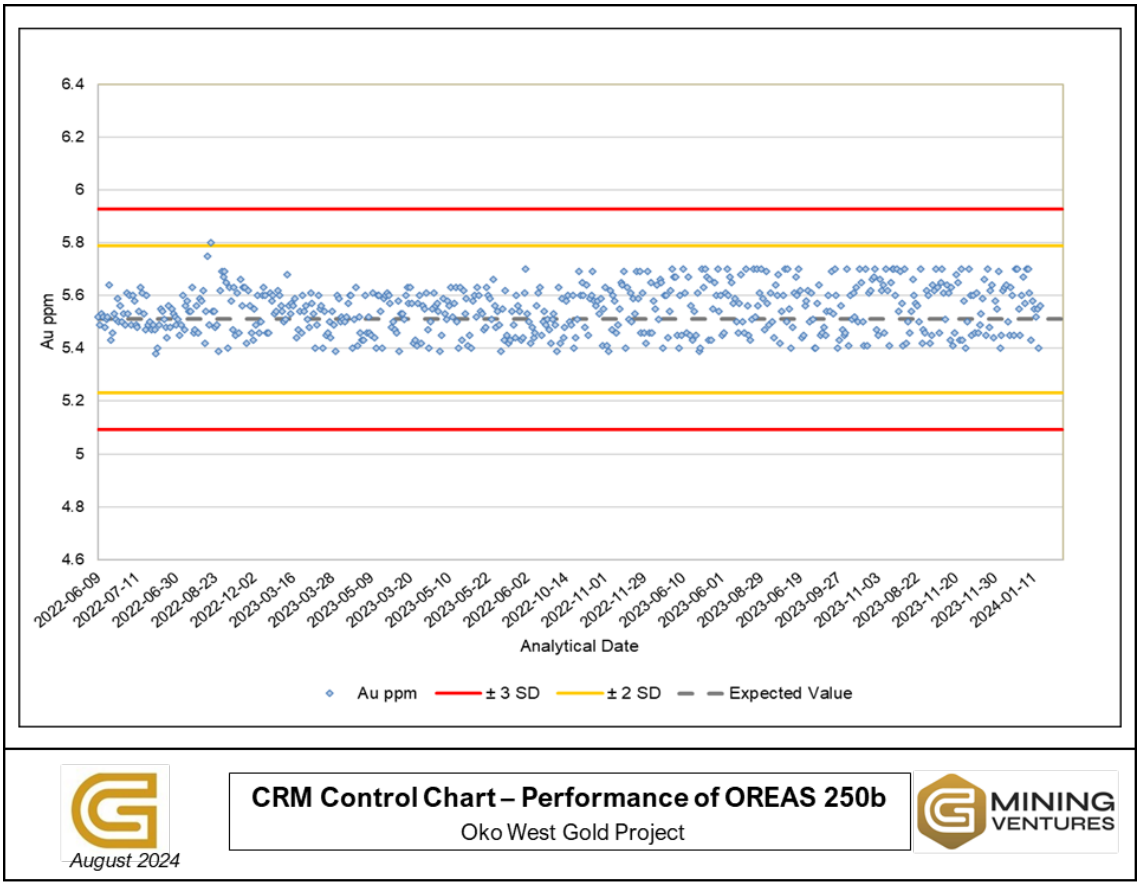
Source: GMS, 2024

Figure 11.12: Sample Control Chart of CRM OREAS 211



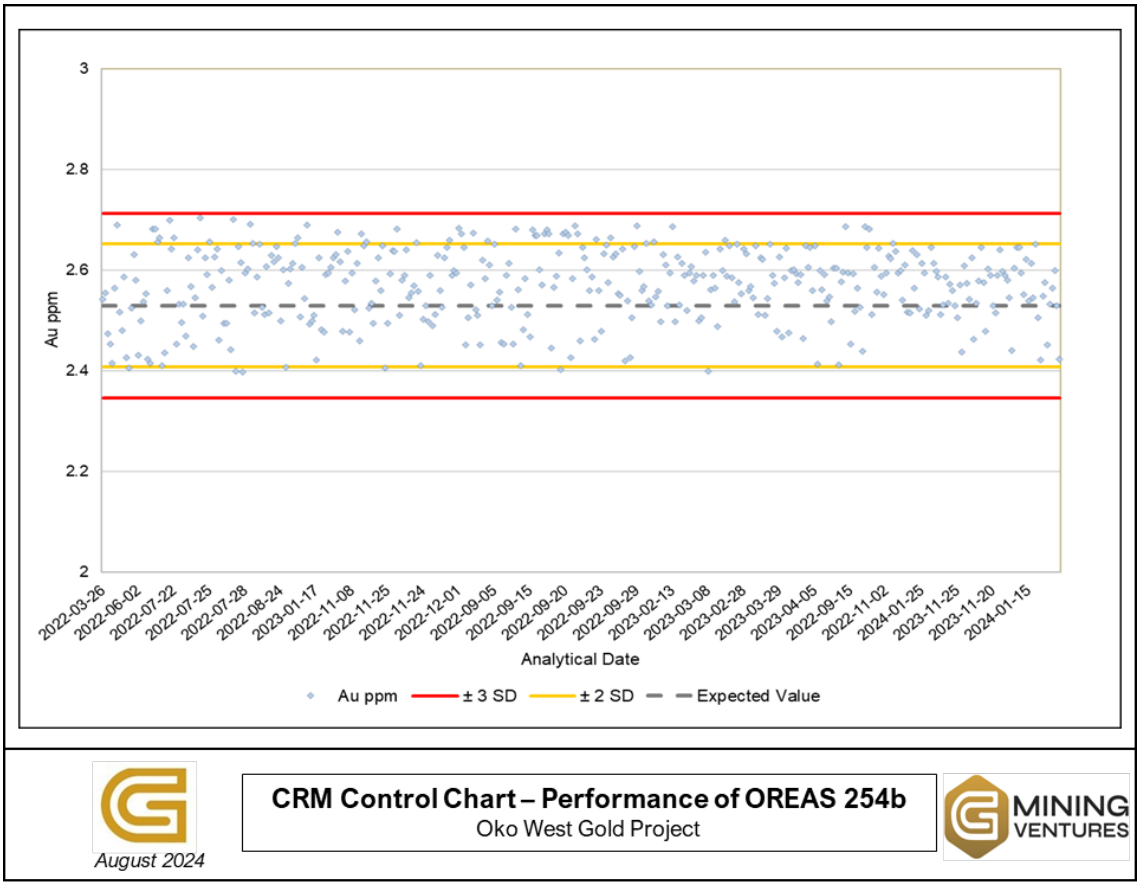
Source: GMS, 2024

Figure 11.13: Sample Control Chart of CRM OREAS 250b



Source: GMS, 2024

Figure 11.14: Sample Control Chart of CRM OREAS 254b



Source: GMS, 2024

11.7.3 Duplicates

As part of the QA/QC program, duplicates were incorporated as QC samples to assess grade variability were introduced among assayed samples in addition to monitoring laboratory consistency covering the 2020-2024 drilling programs.

A total of 2,086 field duplicates from RC holes, trenches, and diamond drill holes (DH) were collected and submitted for analysis using methods 1A2-50 and 1A3-50. For the RC duplicates, 934 out of 1,654 reported above 0.05 ppm and below 3.00 ppm gold for method 1A2, and 20 duplicate pairs reported above 0.3 ppm gold for method 1A3. For the trench duplicates, 176 duplicate pairs out of 221 reported above 0.05 ppm and below 3.00 ppm gold for method 1A2, and 7 duplicate pairs reported above 0.3 ppm gold for method 1A3. For the DH field duplicates, 113 out of 211 pairs reported above 0.05 ppm and below 3.00 ppm gold for method 1A2, and 89 pairs reported above 0.3 ppm gold for method 1A3.

Table 11.2 summarizes the number of samples by duplicate type submitted by the Company to assess the reproducibility of assays and identify any sampling bias.

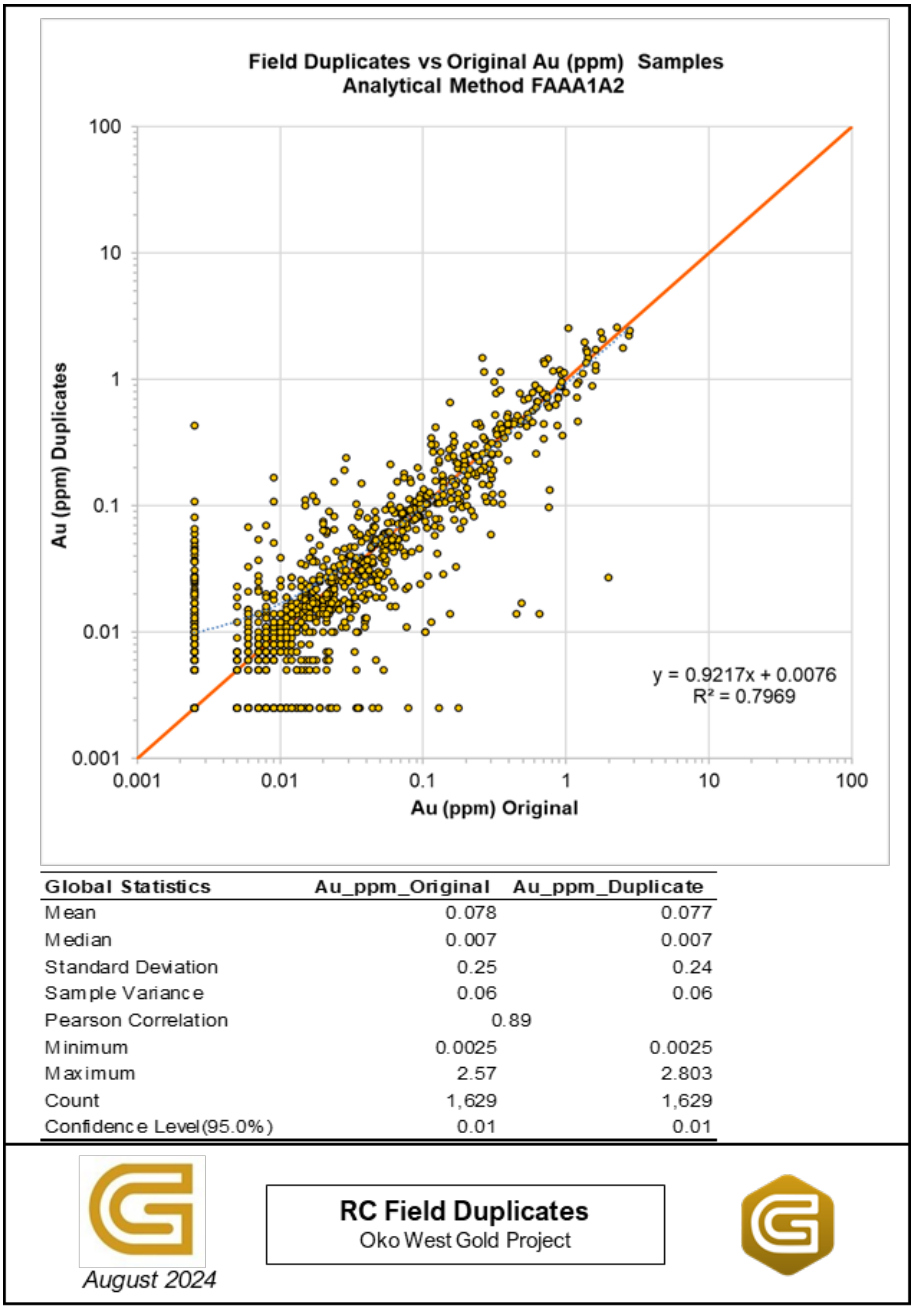
Table 11.2: Quality Control (QC) Duplicates Submitted by the Company

QC Sample Type	Total of QC Samples (FAAA-1A2)
Field Duplicate (RC)	1,629
Field Duplicate (Trench)	214
Field Duplicate (DH)	101
Coarse Reject Duplicate	1,673
Pulp Duplicate	1,933
Total	**5,550**

Analytical Method	Umpire Pulp Duplicates
FAS-121	1,676
FAS-425	118
Total	**1,794**

RC field duplicates are sent to the primary laboratory to calculate field, preparation, and analytical precision. The results of RC field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.15. The RC field duplicates returned results with a good correlation of determination (R^2) of 80% and a linear regression slope of 0.92. The RC field duplicates results are similar in average when compared to the original assays showing a relative difference of 2.1% (Figure 11.15).

Figure 11.15: RC Field Duplicates – 2020-2024 Drilling Programs



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.078	0.077
Median	0.007	0.007
Standard Deviation	0.25	0.24
Sample Variance	0.06	0.06
Pearson Correlation	0.89	
Minimum	0.0025	0.0025
Maximum	2.57	2.803
Count	1,629	1,629
Confidence Level(95.0%)	0.01	0.01

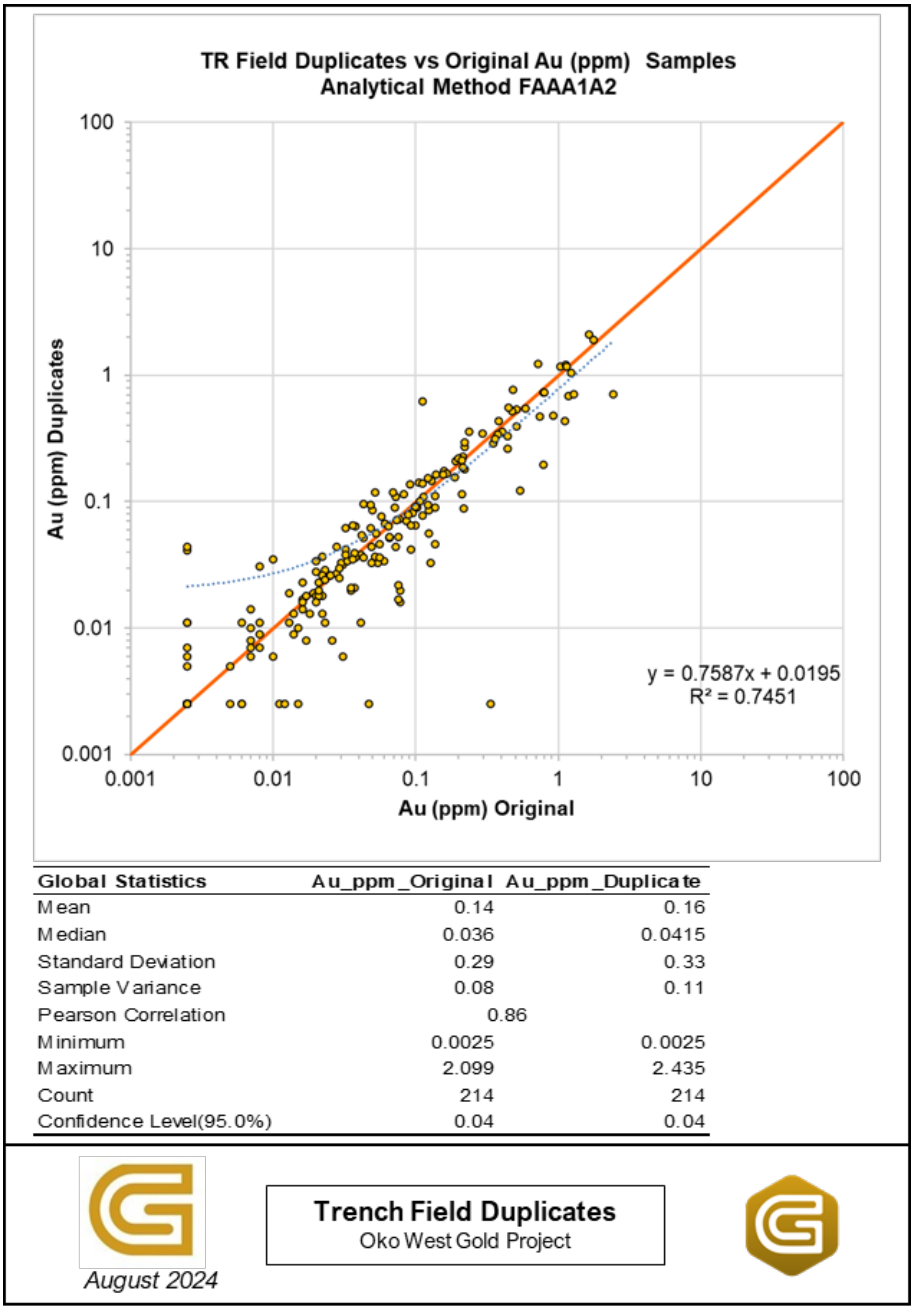
August 2024

RC Field Duplicates
Oko West Gold Project

Source: GMS, 2024

The results of trench field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.16. The trench field duplicates returned results with a good correlation of determination (R^2) of 74% and a linear regression slope of 0.76. The trench field duplicates are on average 14% higher when compared to the original assays (Figure 11.16).

Figure 11.16: Trench Field Duplicates – 2020-2024 Drilling Programs

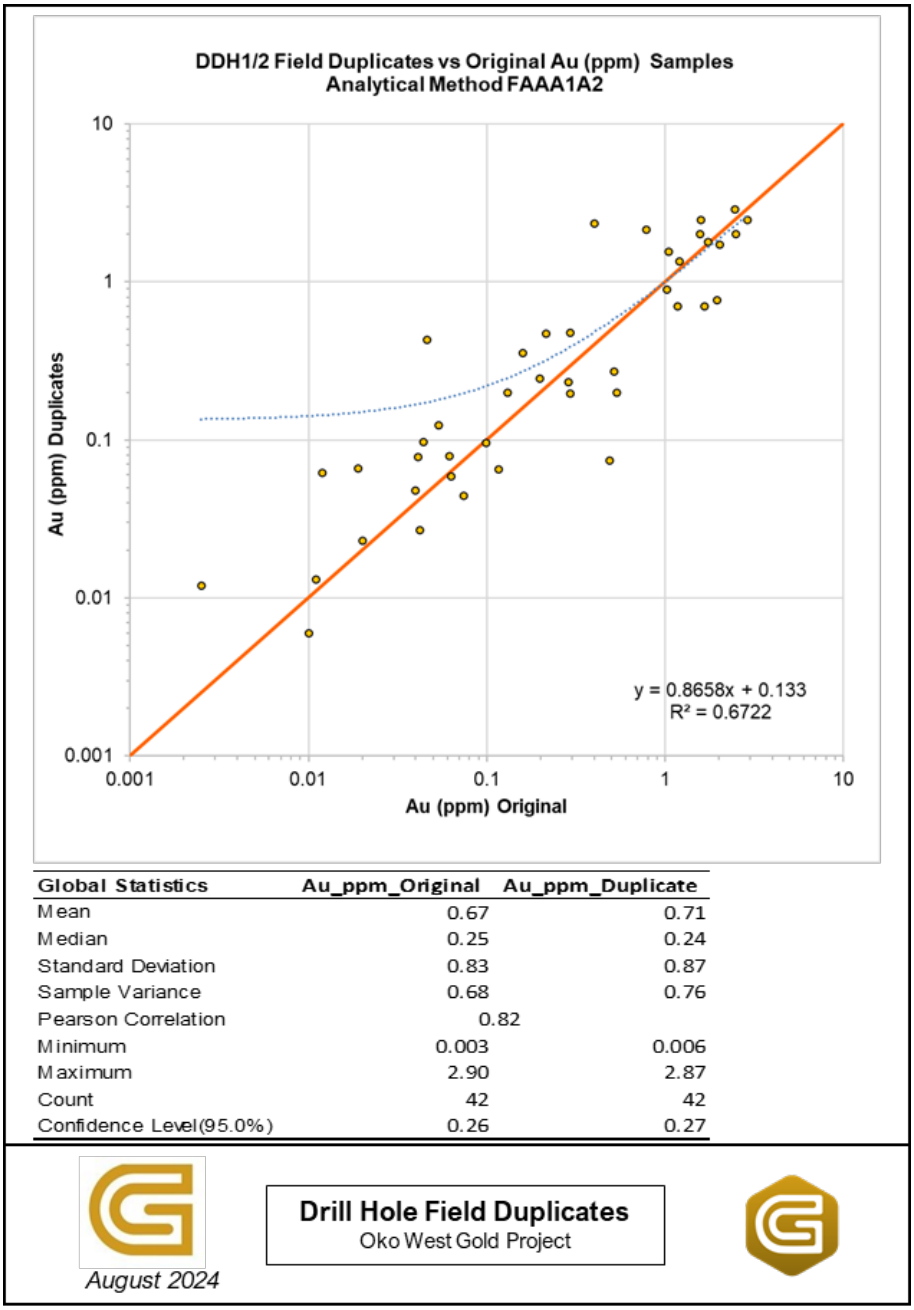


Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.14	0.16
Median	0.036	0.0415
Standard Deviation	0.29	0.33
Sample Variance	0.08	0.11
Pearson Correlation	0.86	
Minimum	0.0025	0.0025
Maximum	2.099	2.435
Count	214	214
Confidence Level(95.0%)	0.04	0.04

Source: GMS, 2024

The results of half and quarter core drill hole field duplicates vs. original gold values assayed by FAAA-1A2 are presented in Figure 11.17. The trench field duplicates returned results with a good correlation of determination (R^2) of 67% and a linear regression slope of 0.86. The core field duplicates are on average 7% higher when compared to the original assays (Figure 11.17).

Figure 11.17: Diamond Drill Hole (½) Field Duplicates – 2020-2024 Drilling Programs



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.67	0.71
Median	0.25	0.24
Standard Deviation	0.83	0.87
Sample Variance	0.68	0.76
Pearson Correlation	0.82	
Minimum	0.003	0.006
Maximum	2.90	2.87
Count	42	42
Confidence Level(95.0%)	0.26	0.27

Source: GMS, 2024

Actlabs also conducts an internal quality control program, which includes a routine selection of coarse rejects and pulp duplicates. The preparation duplicates are taken from the coarse crushed material before the pulverizing process. The pulp duplicates are taken after the sample has been pulverized, and two (2) same-weighted pulps are analyzed by the same analytical method.

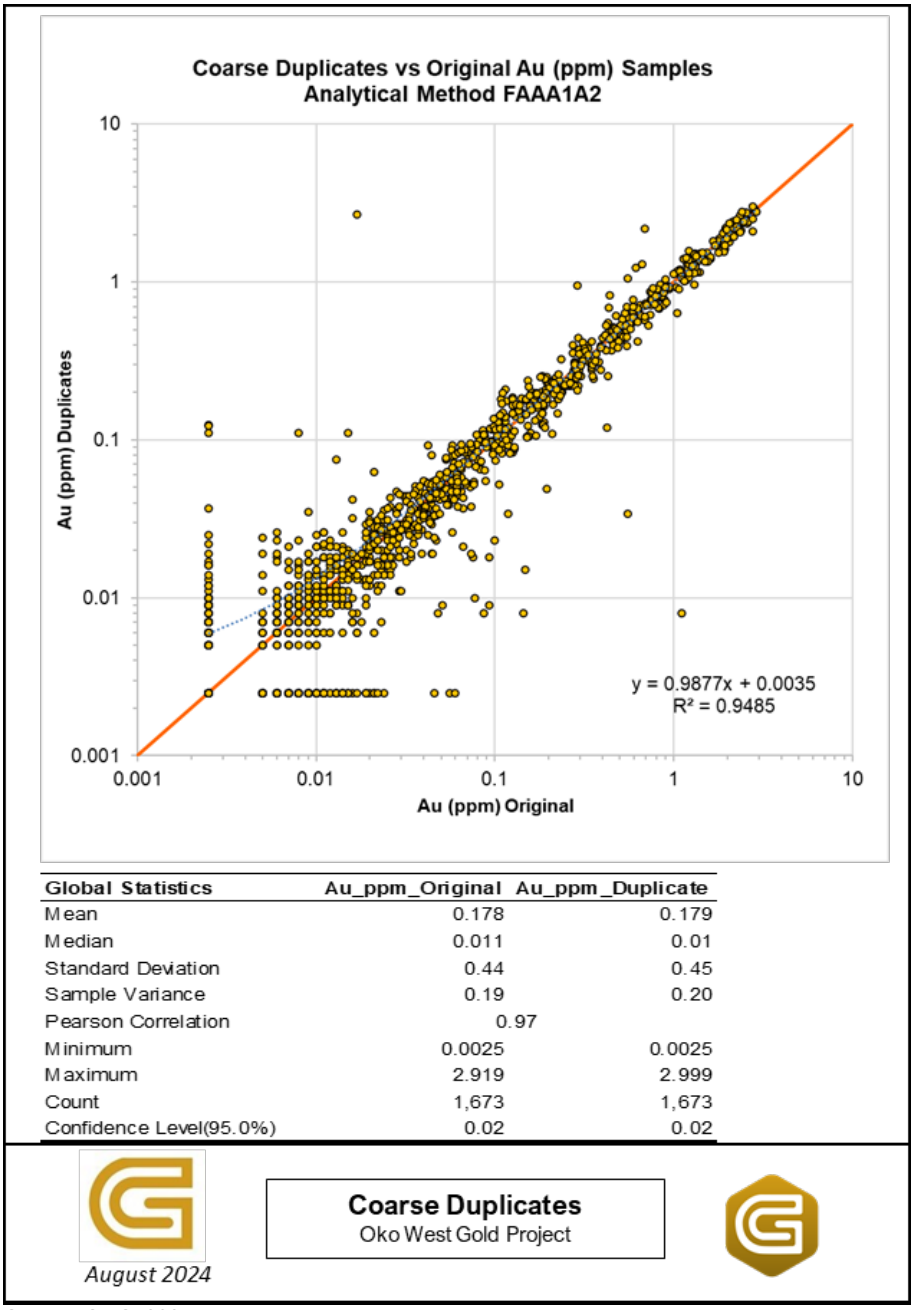
Coarse reject duplicates are created by splitting a second cut of the crushed sample following the same protocol and for the same weight as the original sample. The main objective is to determine if splitting procedures are applied consistently or modifications to sample preparation procedures, as the crush size of the samples is required. Due to their particle sizes, preparation duplicates are expected to be less similar than the pulp duplicate samples.

A total of 1,673 coarse reject duplicates were analyzed for gold by fire assay with AA finish (FAAA-1A2) at Actlabs. A total of 507 duplicates reported gold results above 0.05 ppm and below 3.00 ppm gold for method FAAA-1A2 (Figure 11.18). The preparation duplicates for gold have 64% of the duplicate pairs reporting within ±20% tolerance limit. GMS judges this acceptable since most pairs ±20% difference is below 0.1 g/t Au. The coarse reject duplicates produced an excellent correlation of 97%. These internal QC results indicate that the duplicate grades are close to the original grade values, and good reproducibility is obtained at the primary laboratory regarding coarse reject material at the preparation stage of the analysis process.

Commercial laboratories routinely assay a second aliquot of the sample pulp, usually for one (1) in ten (10) samples. The data are used by the laboratory for their internal quality control monitoring. These data are provided to the clients at no additional cost.

A total of 1,933 pulp duplicates were analyzed for gold by fire assay with AA finish (FAAA-1A2). 613 duplicate pairs sent to Actlabs reported gold results above 0.05 ppm and below 3 ppm gold for method FAAA-1A2. Figure 11.19 shows a good correlation of 98% between the pulp duplicates and the original gold values.

Figure 11.18: Coarse Reject Duplicates – 2020-2024 Drilling Program



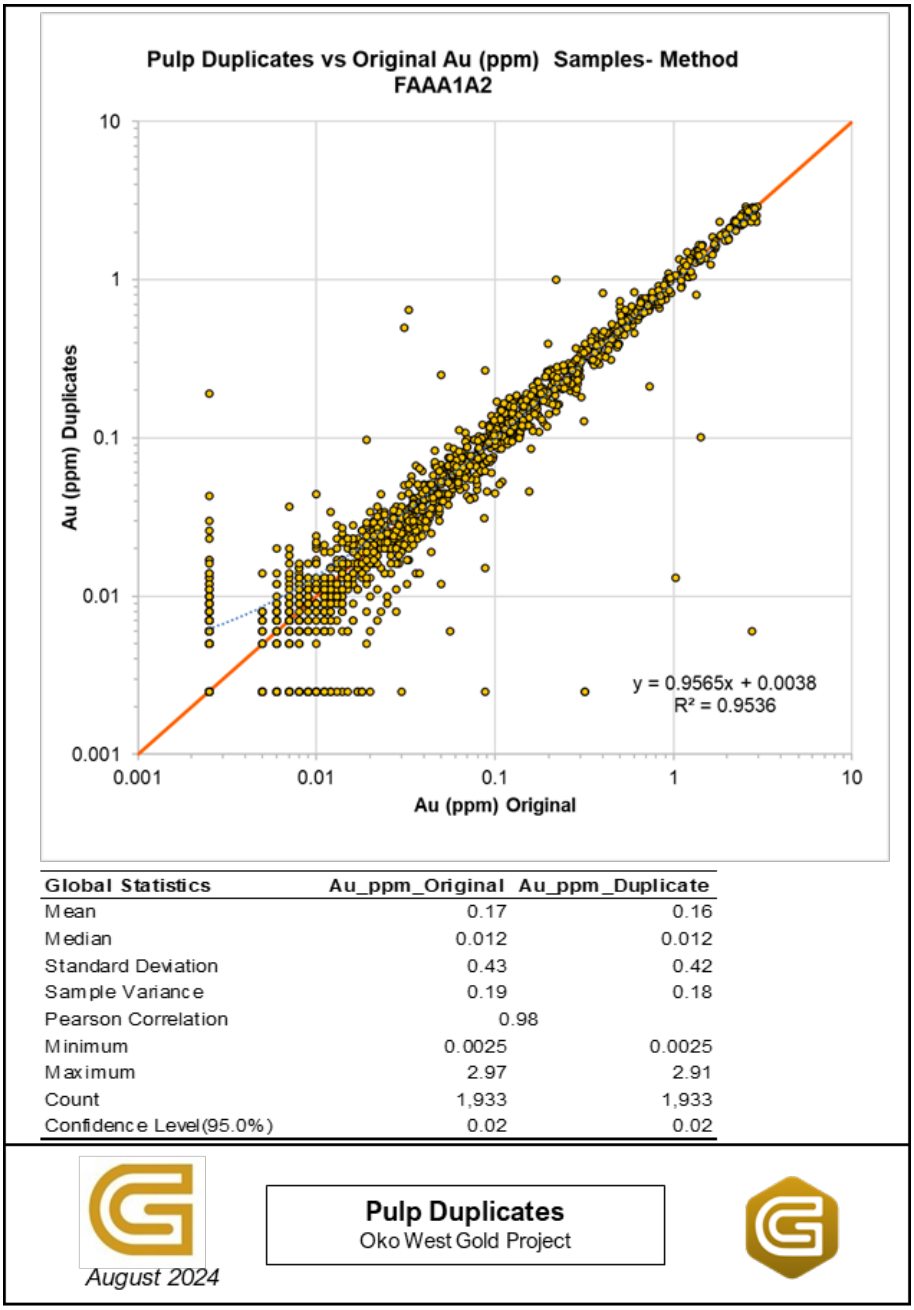
Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.178	0.179
Median	0.011	0.01
Standard Deviation	0.44	0.45
Sample Variance	0.19	0.20
Pearson Correlation	0.97	
Minimum	0.0025	0.0025
Maximum	2.919	2.999
Count	1,673	1,673
Confidence Level(95.0%)	0.02	0.02

August 2024

Coarse Duplicates
Oko West Gold Project

Source: GMS, 2024

Figure 11.19: Pulp Duplicates Check – 2020-2024 Drilling Programs



Global Statistics	Au_ppm_Original	Au_ppm_Duplicate
Mean	0.17	0.16
Median	0.012	0.012
Standard Deviation	0.43	0.42
Sample Variance	0.19	0.18
Pearson Correlation	0.98	
Minimum	0.0025	0.0025
Maximum	2.97	2.91
Count	1,933	1,933
Confidence Level(95.0%)	0.02	0.02

Source: GMS, 2024

11.7.4 Umpire Check Assays

Umpire check assays are often used to assess the possibility of intra-laboratory bias.

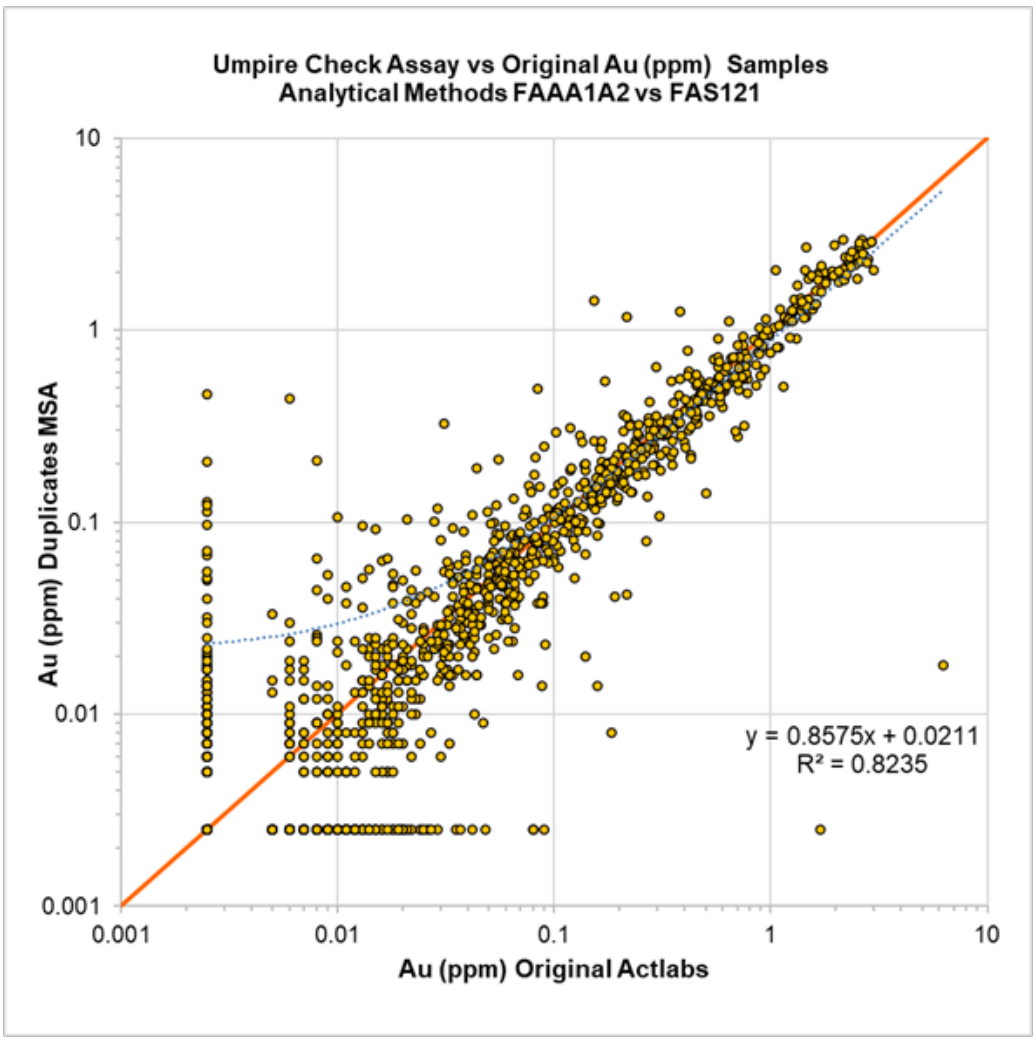
The geology team selects the samples to be submitted as check assays at a frequency of one (1) in 15 to one (1) in 20 samples (5%). The primary laboratory, Actlabs, prepares a second pulp and sends it to the

check laboratory, MSA Labs, in Georgetown, Guyana, with additional reference materials. The analytical methods at the check laboratory are comparable to the original laboratory. There is a preference for samples from unweathered rocks to avoid potential discrepancies between both laboratories. The secondary lab thoroughly homogenizes the samples and analyses them using a method as close to the primary lab as possible. The results between the primary and secondary labs are compared for discrepancies.

Reunion selected 1,486 samples for check assaying at the MSA Labs. The umpire check assays with AA finish (FAS-121) at MSA Labs have 67% of the duplicate samples reporting within ± 25%. As shown in the global statistical analysis (Figure 11.20), the mean gold grades obtained by MSA Labs pulp duplicates are slightly higher than the primary laboratory assay results. The average relative percent difference between both laboratories is 3.4%. GMS considers the check assay results as acceptable.

Figure 11.20 presents the results of umpire pulp duplicates compared to the original Au values from Actlabs.

Figure 11.20: Actlabs vs MSA Labs Check Assays



Global Statistics	Au_ppm_Oginal	Au_ppm_Duplicates
Mean	0.186	0.192
Median	0.016	0.017
Standard Deviation	0.45	0.48
Sample Variance	0.20	0.23
Pearson Correlation	0.91	
Minimum	0.0025	0.0025
Maximum	2.96	6.28
Count	1,486	1,486
Confidence Level(95.0%)	0.02	0.02

Source: GMS, 2024

11.8 External Audit – Qualitica Consulting Inc, September 2022

In September 2022, Qualitica Consulting Inc. was engaged to undertake an external audit of QA/QC results. The report contained a summary of QA/QC results up to September 2022, and made the following conclusions:

- There is no evidence of systematic contamination; however, it is likely that a portion of the blank failures is due to mislabelling or CRM / blank mix-ups.
- The CRM failure rate is less than 1%, and the results from CRMs are acceptable.
- A mix-up of OREAS250b and OREAS250 CRMs was investigated and resolved.
- The RC field duplicate results are acceptable.
- The coarse reject preparation duplicate results are acceptable.
- The pulp duplicate results are acceptable.
- The umpire check assay results are acceptable; however, it is recommended that MSA does not homogenize the pulps by mat rolling before running the analysis. As of 2024, mat rolling is still in in practice.

11.9 QP Conclusions and Recommendations

The QP concludes that the sample preparation, analysis, and security procedures applied by the company are acceptable. Documentation of sampling procedures used to support the diamond and reverse circulation drilling programs is considered by GMS as best industry practice.

At this time, there is no re-assay procedure for duplicates that significantly exceed the original laboratory value. GMS recommends implementing a procedure to the QA/QC protocols for outliers greatly exceeding the expected value.

In the opinion of the QP, sample preparation, analysis, and security procedures implemented by the company are comparable with the best industry standards, and robust controls are in place to ensure the integrity of the assay database. A statistical analysis of the quality control data from the 2020 to 2024 sampling programs did not expose any significant analytical issues. The QP believes that the drilling database is robust and reliable.

12 DATA VERIFICATION

12.1 Site Visits

Pascal Delisle, P.Geo., Director of the Geology and Resource department at G Mining Services (GMS), as well as Émile Boily-Auclair, Mineral Resources Estimation Engineer at GMS visited the project between January 30, 2024, and February 2, 2024. A selection of drill collars was visited and independently verified using a handheld GPS. The comparison between the results and the database is shown in Table 12.1. Some examples of drill collars are shown in Figure 12.1.

Mr. Christian Beaulieu, P.Geo., consulting geologist for GMS and qualified person (QP) under NI 43-101 visited the project on April 18 and April 19, 2023. During this site visit, the QP inspected mineralized intervals, alteration assemblages and QAQC protocols and conducted field checks of trenches and to validate drill collars. Some trenches were cleaned, and sampling protocols were assessed directly in the field with site geologists; drill core was reviewed with Reunion personnel. Drill core review permitted to observe clear relationships between gold grades (or presence of mineralization) and rock alteration / strain within mineralized domains LDZ, AU_2, AU_3, AU_3A. It is the opinion of the QP that work practices at Oko West are in line with the CIM Best Practice Guidelines (2019).

During all site visits, drilling activities were ongoing at the time of the visit. Core processing and storage facilities located on-site were toured (Figure 12.2), and the drill core from Oko West was reviewed. Outcropping mineralization and trenches were visited and compared with the provided LiDAR topographic surface for survey accuracy. Sampling protocols were also reviewed with field geologists working on the Oko West Project. Independent quarter core samples were collected, and pulps sent to an independent laboratory (Section 12.2).

GMS also reviewed sampling and QAQC procedures on site and visited the preferred independent laboratory (Actlabs, Georgetown, Guyana) and the umpire laboratory (MSALABS, Georgetown, Guyana) to inspect the sample preparation facilities. During all site visits, chain-of-custody and sample security protocols were also reviewed and were found to be robust and transparent. The protocols in place are judged to be very robust: several instances of redundancy are integrated, and the staff is well trained to perform several tasks, which both contribute to reducing the risks of errors and lost samples.

Prior to the above site visit, one (1) other visit was performed by James Purchase, P.Geo., geologist at GMS, between May 9, 2022, and May 13, 2022. During this visit, Mr. Purchase was able to visit drill rigs, inspect drill core, collect QP samples, validate sampling protocols, review drilling procedures with drilling contractors and field geologists working for the Project and visited the preferred independent laboratory

(Actlabs, Georgetown, Guyana) and the umpire laboratory (MSALABS, Georgetown, Guyana). Mr. Purchase believed that work practices at Oko West are in line with the CIM Best Practice Guidelines (2019) (Oko West Technical Report; 2022).

Table 12.1: Validation of Drill Collar Coordinates (PSAD 1956 UTM Zone 21N)

Site Visit	Hole ID	Database		G Mining		Difference	
		X	Y	X	Y	X	Y
2022	OKWD21-41	272777.8	701902.2	272777.0	701904.0	-0.8	1.8
	OKWD21-42	272786.5	701994.8	272784.0	701997.0	-2.5	2.3
	OKWD22-73	272833.3	701651.8	272835.0	701652.0	1.7	0.2
	OKWR22-123	272705.2	702248.2	272704.0	702254.0	-1.2	5.8
	OKWR22-132	272821.3	702648.3	272819.0	702654.0	-2.3	5.7
	OKWR22-77	272900.3	701735.8	272901.0	701739.0	0.7	3.2
2023	OKWD21-015	272669.5	701604.0	272667.0	701603.0	2.1	1.2
	OKWT21-044A	272676.5	701688.0	272677.0	701687.0	-1.0	0.7
	OKWD21-057	272691.5	701656.0	272687.0	701655.0	4.2	0.9
	OKWD22-153	272703.5	701692.0	272698.0	701694.0	5.1	-1.8
	OKWD22-087	272779.5	701860.0	272777.0	701857.0	2.6	2.6
	OKWD21-051	272792.5	701795.0	272788.0	701793.0	5.0	1.8
	OKWD22-064	272792.5	701656.0	272787.0	701657.0	5.1	-1.3
	OKWD23-221	273003.5	702151.0	273002.0	702153.0	1.5	-2.0
	OKWD22-175	273045.5	701876.0	273045.0	701875.0	0.5	0.6
	OKWD22-182	273092.5	701721.0	273089.0	701722.0	3.4	-0.9
	OKWD22-178	273094.5	701719.0	273091.0	701720.0	3.9	-1.2
2024	OKWD22-173	272813.0	701982.0	272812.5	701985.0	0.5	-3.3
	OKWD22-088	272886.9	702135.5	272886.5	702143.0	0.4	-7.4
	OKWD22-163	272758.4	701622.6	272760.5	701622.0	-2.1	0.6
	OKWD22-166B	272795.9	702072.5	272795.5	702077.0	0.4	-4.5
	OKWD23-220	272815.3	702020.6	272815.5	702025.0	-0.2	-4.4
	OKWD23-348	273089.7	701716.8	273085.5	701719.0	4.2	-2.2
	OKWD23-343	273108.1	701792.3	273105.5	701794.0	2.6	-1.7

Figure 12.1: Verification of Drill Collar – OKWD23-220



Source: GMS, 2024

Figure 12.2: Sample Reject Storage in Georgetown (top) & Core Storage Facilities Onsite (bottom)



Source: GMS, 2024

Figure 12.3: Outcrop Inspection (top) and Closeup of Outcrop OKWT20-001.



Source: GMS, 2024
**Note: Primary textures are still visible from surface altered material (saprolite).*

Figure 12.4 : Core Cutting Facility (top) and Core Sampling Facility (bottom)



Source: GMS, 2024

12.2 QP Duplicate Samples

During Mr. Purchase's site visit, 20 quarter-core duplicates were taken from mineralized intervals derived from four (4) drillholes distributed through the strike length of gold mineralization. Both oxidized and unoxidized mineralization was sampled. The 20 duplicates were hand-delivered by the QP to Actlabs in Georgetown and were subsequently analyzed using the standard analysis protocol described in Section 11. QAQC samples were also inserted into the sample stream, and all standards and blanks returned gold values within the expected ranges of error. The entire sampling process was supervised.

Another nine (9) quarter-core duplicates were collected by Mr. Delisle during the 2024 site visit. Samples were collected from mostly fresh material ranging from 0.01 g/t Au to 15.56 g/t Au, across the deposit strike and depth. Selected core samples were sampled manually and cut under the supervision of GMS' team in quarter core. All samples were bagged and sealed with a security tag by GMS' personnel (Figure 12.5). The samples were sent to MSALABS in Georgetown. Laboratory technicians made sure nobody tempered with the security tags when samples were received. Nine (9) pulp samples from the reject storage facility previously analysed at Actlabs in Georgetown were hand-delivered to MSALABS in Georgetown by Mr. Delisle for external umpire check assays.

The results and a comparison with the original assays are presented in Table 12.2 and Figure 12.6 for 2022 and 2024 core duplicates and in Table 12.3 for 2024 pulp re-assays.

Table 12.2: Independent Core Duplicate Results

Site Visit	Hole ID	From	To	Original Sample ID	QP Sample ID	Au ppm Original	Au ppm QP
2022	OKWD21-053	193.0	194.5	577154	791858	6.52	5.89
	OKWD21-053	194.5	196.0	577156	791860	0.79	0.52
	OKWD21-053	196.0	197.5	577157	791861	0.75	0.75
	OKWD21-053	197.5	199.0	577158	791862	4.55	3.11
	OKWD21-053	199.0	200.5	577159	791863	2.66	4.53
	OKWD21-031	66.0	67.5	1074952	791864	1.13	1.58
	OKWD21-031	67.5	69.0	1074953	791865	2.05	2.61
	OKWD21-031	69.0	70.5	1074954	791866	1.70	2.04
	OKWD21-031	70.5	72.0	1074955	791868	0.85	2.17
	OKWD21-031	72.0	73.5	1074956	791869	2.05	4.66
	OKWD22-062	280.0	281.0	576235	791870	0.71	1.54
	OKWD22-062	281.0	282.0	576236	791871	3.05	2.56
	OKWD22-062	282.0	283.0	576237	791873	1.89	2.50
	OKWD22-062	283.0	284.0	576238	791874	3.30	2.76
	OKWD22-062	284.0	285.0	576239	791875	15.62	6.65
	OKWD22-066	180.8	182.0	1013343	791877	1.14	0.97
	OKWD22-066	182.0	183.0	1013344	791878	6.16	6.65
	OKWD22-066	183.0	183.8	1013345	791879	13.83	10.17
	OKWD22-066	183.8	185.0	1013346	791881	1.28	0.68
	OKWD22-066	185.0	186.0	1013347	791882	0.41	1.49
2024	OKWD23-237A	317.0	318.0	549446	B00333475	0.01	0.01
	OKWD23-237A	305.0	306.0	549433	B00333476	15.56	8.65
	OKWD23-237A	306.0	307.4	549434	B00333477	0.13	0.39
	OKWD23-237A	293.0	294.0	549419	B00333478	0.85	0.95
	OKWD21-046	112.5	114.0	569800	B00333479	0.57	0.51
	OKWD23-243	443.3	444.5	553142	B00333480	1.54	2.17
	OKWD23-243	524.0	524.5	553240	B00333481	8.80	25.90
	OKWD21-046	175.5	177.0	569847	B00333483	4.98	8.62
	OKWD21-046	63.0	64.5	569764	B00333484	2.75	0.63

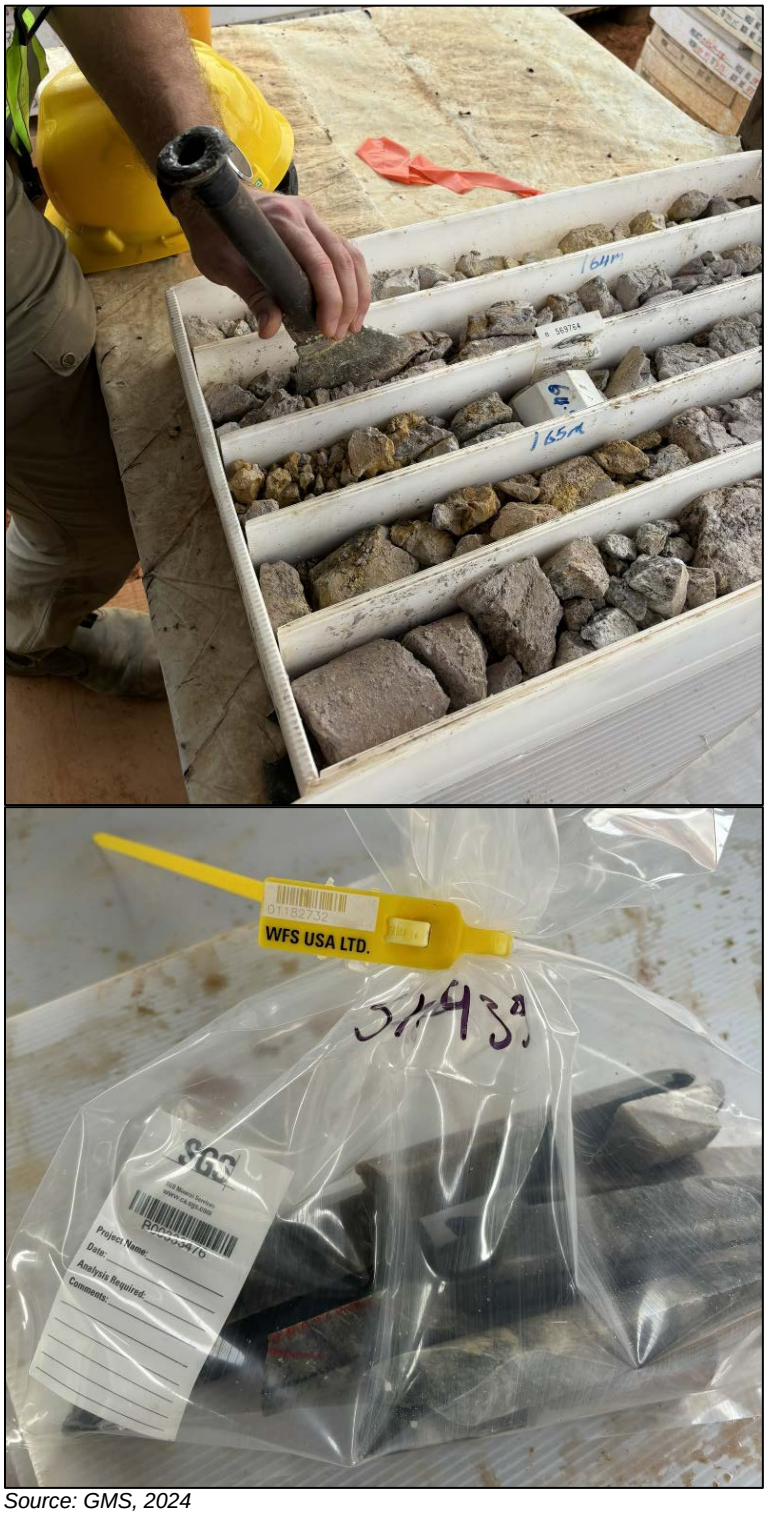
Table 12.3: Pulp Re-assay Results

Site Visit	Hole ID	From	To	Original Sample ID	Au ppm	Au ppm
					Original	Umpire
2024	OKWD22-129	223.0	224.0	A02208	0.61	0.69
	OKWD22-129	256.8	258.6	A02242	1.06	0.98
	OKWD22-187	471.1	472.4	A08865	1.62	1.39
	OKWD22-187	479.3	480.7	A08872	6.11	5.74
	OKWD23-223	236.0	237.0	545705	0.85	0.81
	OKWD23-223	266.7	267.6	545737	4.19	3.12
	OKWD23-223	250.0	251.8	545719	1.44	1.27
	OKWD23-243	534.0	535.0	553259	1.48	1.35
	OKWD23-243	525.5	526.0	553244	40.84	44.60
	OKWD23-243	516.5	517.0	553225	20.30	19.90

The assay results from the QP duplicates sent to Actlabs (2022 site visit) show a good correlation with the original assay values found in the database. For duplicates sent to MSALABS (2024 site visit), a good correlation is also observed between assays from the database and re-assays. Some variability is expected due to the nugget effect of the gold mineralization, and the difference in sample size (½ core for the original assays, ¼ core for the QP duplicate assays). No bias was identified.

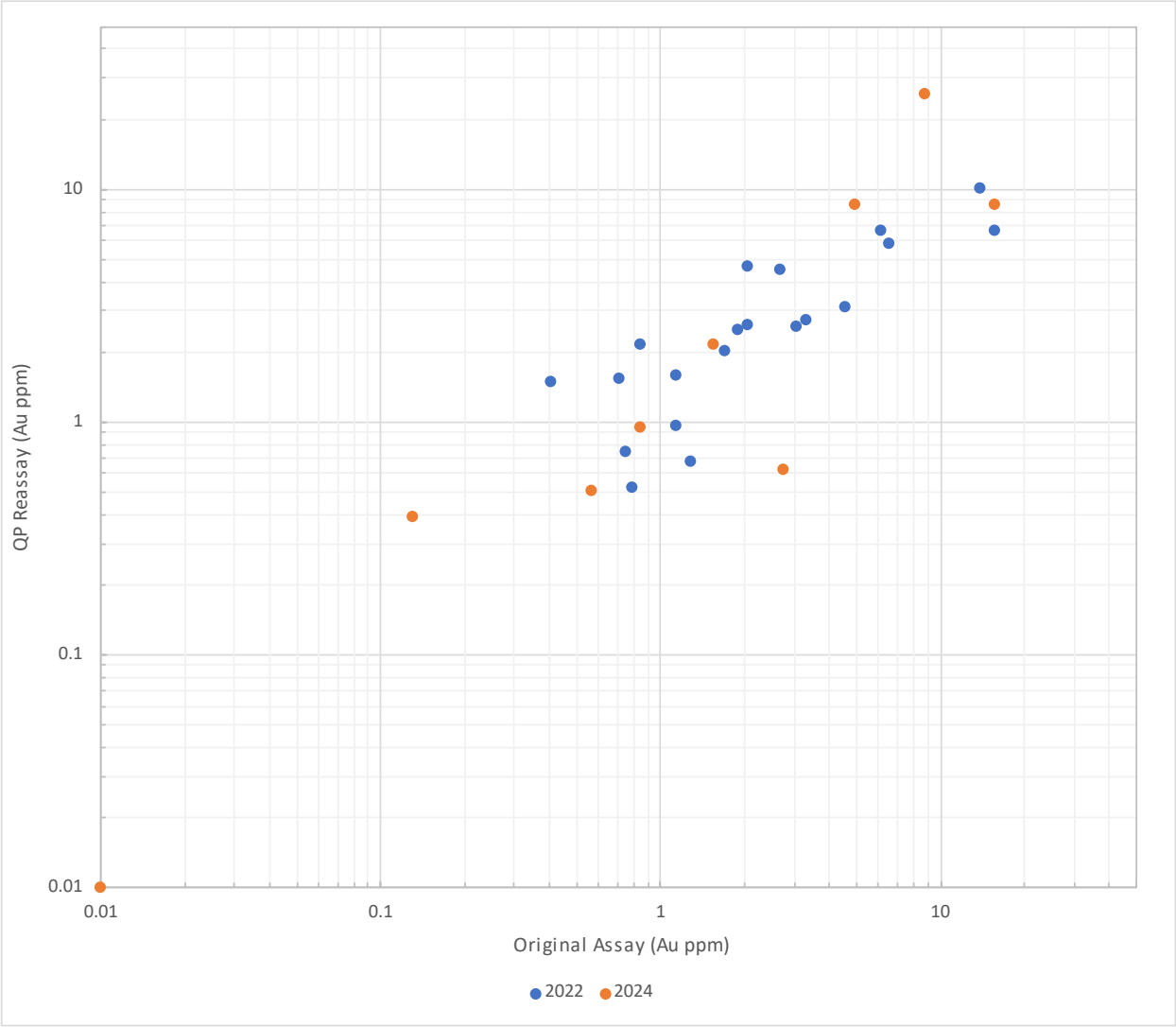
For the pulp re-assay, a 2 – 26% difference is observed between original assays completed at Actlabs and pulp re-assays submitted to MSALABS. MSALABS pulp duplicate tend to be slightly lower than the primary laboratory (i.e. Actlabs). As explained in Section 11, this could be due to the homogenization done by MSALABS on pulps.

Figure 12.5: QP Sampling (Top) and QP Sample with Security Tag (Bottom).



Source: GMS, 2024

Figure 12.6: Scatter Plot Showing Original Assays (X-axis) vs. QP Duplicate Assays (Y-axis)



Source: GMS, 2024

12.3 Drill Core Inspection

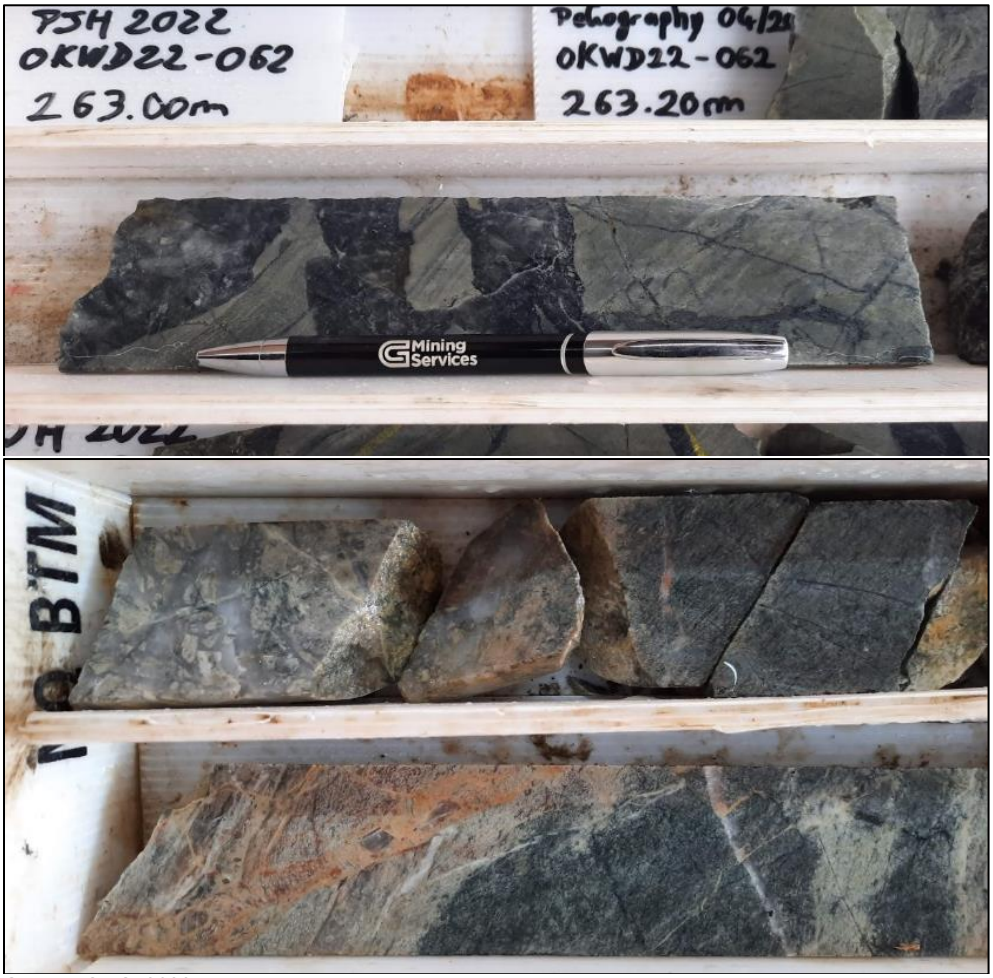
GMS reviewed numerous mineralized intersections from 16 holes distributed evenly throughout the strike length of gold mineralization during the 2024 site visit. The QP personally inspected 12 holes distributed evenly throughout the strike length of the deposit. A clear relationship was visible between gold grades and alteration / veining intensity. The frequency of smoky quartz brecciation and silicification, and sericitic "ghosting" of host rock appear to be directly relatable to the gold grades observed in assays. Visible gold was observed on numerous occasions. GMS notes that sulphide content is generally low compared to other orogenic deposits; however, pyrite accumulations were observed in mineralized intervals hosted within mafic rocks, although these occurrences are uncommon. Figure 12.7 shows some typical examples of alteration assemblages associated with mineralization. In trenches, a clear distinction between alluvium or

colluvium sediments with saprolite material could be observed (Figure 12.8). Primary textures in saprolite are also very well preserved (Figure 12.8).

In oxidized intervals, gold grades were found to be directly related to the abundance of quartz fragments hosted with clays.

Unmineralized rocks in the hanging wall were found to be relatively unaltered sediments or volcaniclastics with intact primary textures exhibiting weaker foliation / shearing. The footwall rocks (also unmineralized) were also found to be weakly altered by metasomatic assemblages unrelated to mineralization, with primary igneous textures evident. These observations correlate with low-gold grades observed in the assays.

Figure 12.7: Typical Examples of Gold Mineralized Intervals Displaying Various Alteration Assemblages and Veining



Source: GMS, 2022.
**Note: Top: Typical smoky-quartz vein with fragments of the sericite-altered host rock sediments. Bottom: Brecciation and silicification within volcaniclastic host rock with sericitic alteration emanating from fracture planes. Fe--carbonate veining (orange) is mostly barren.*

Figure 12.8: Contact Between Overburden Material and Saprolite (top, water bottle as scale), and Primary Textures in Saprolite (bottom)



Source: GMS, 2023

12.4 Drillhole Database Verification

GMS reviewed the original assay certificates for Oko West drilling programs performed between 2020 and the closure of the database (February 07, 2024). A script was used to validate all assays recorded in the database with the gold assays reported by the laboratory in the original certificates. Four (4) discrepancies were noted and addressed to the Company database management team. Only one (1) sample (OKWD23-

264: 558705) was ignored from the database due to an upper limit detection value (≥ 3g/t Au) recorded in the database. This sample should have been re-assayed using gravimetric analysis method.

GMS also reviewed collar location, downhole surveys and diamond drill holes (DDH) twin holes. The drill holes collar locations are recorded in UTM PSAD56 Zone 21 North coordinate system. The collar elevations were validated using a 50-cm digital elevation model (DEM) provided to GMS by Reunion Gold. One (1) collar elevation was corrected (OKWD21-005) after differences of more than seven (7) metres were observed between the database and the DEM. Other collar elevations in the database were consistent with the DEM.

The consistency of drillhole surveys was assessed using LeapFrog Geo™ 3D viewer and GMS' validation script. Any discrepancies with variations exceeding 5 degrees per 100 min dip and direction were identified and subsequently examined for incoherence. Following the investigation, only two (2) surveys (DDH OKWD22-134 at 42.63 m downhole and DDH OKWD22-174 at 119.00 m) were discarded from the resource database.

Finally, DDH twin holes were conducted to examine the possibility of gold smearing in Reverse Circulation (RC) drillholes, as recommended by GMS. Upon careful evaluation, the Qualified Person (QP) expresses confidence in the accuracy of gold grades obtained through RC drilling and approves the use of RC drill holes for the resource estimation of the Oko West deposit. One (1) RC drillhole (OKWR22-128) was however excluded from the resource database due to potential gold grade smearing identified from the neighbouring DDH twin hole (OKWD22-127) assay result. Furthermore, a single trench (OKWT20-006) was removed from the database as it was entirely sampled in unconsolidated alluvial material.

12.5 Database Validation and Verification by the Qualified Person

An independent validation and verification of the database was undertaken by the Qualified Person for the Mineral Resource Estimation (Mr. Christian Beaulieu). The inspection of the database used in the estimation consisted of inspecting all relevant drilling information in Leapfrog Geo® 2023.2, such as collars, deviation, geology, weathering, assays, composite tables, interpolation profiles and integration into the block model, and checked for inconsistencies. A special attention was paid to more recent data, as the current QP was also acting as a QP for the initial Mineral Resource in 2023 (Technical Report, 2023).

A validation of the assay database was conducted by comparing original assay certificates (PDF) provided by Actlabs with the database used for the estimation. No errors were found.

12.6 QP Commentary and Conclusions

In the opinion of the QP, the drilling, sampling and QAQC procedures meet best industry practices. Sampling equipment and logging facilities were found to be of sufficient quality, and the sample storage facilities were adequate. Although the site is remote, the company has established a robust chain-of-custody in relation to the storage and transportation of samples. Inspection of drill core demonstrated a good correlation between alteration and gold grades, and the independent duplicate assays correlated with the original assay database with acceptable levels of error. The QP has no concerns relating to the validity of the drilling database.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

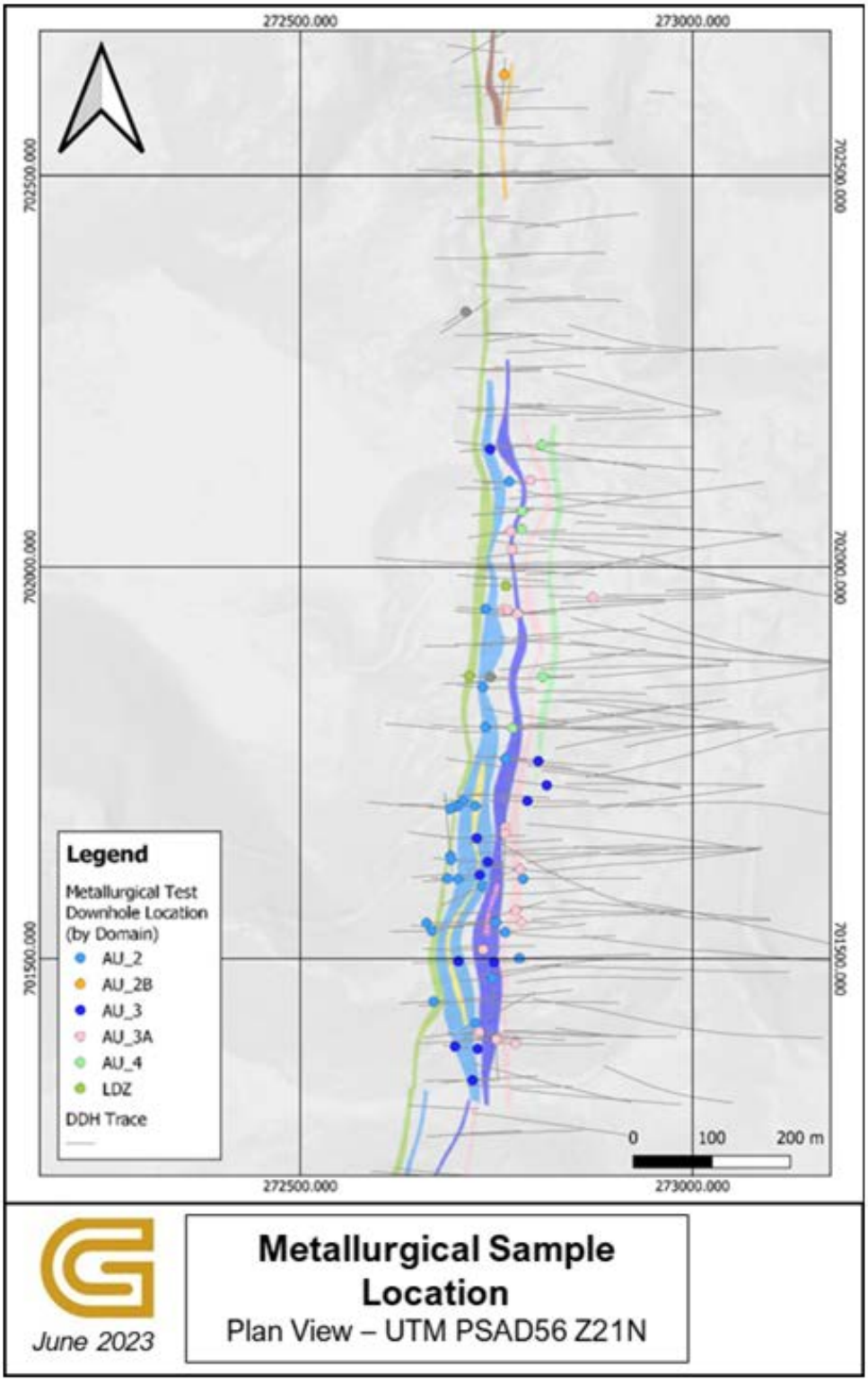
13.1 Introduction

A metallurgical test work program was completed during May-September 2023 at Basemet Laboratories (BML) in Kamloops, British Columbia, Canada, with the objective of determining the preliminary metallurgical response of material domains within the Oko West deposit, establishing initial metallurgical recoveries for the mineral resource estimate and developing an initial flowsheet. The metallurgical test work scope included chemical analysis, mineralogy, comminution, gravity, leach, cyanide detoxification and acid base tests. A variability metallurgical test work program was initiated in April 2024 and is on-going. Relevant metallurgical data and information from the 2023 program is summarized in this section.

13.2 Sample Selection

Samples were selected from three (3) weathering zones, i.e., saprolite, transition and fresh rock (open pit) and main geological units, i.e., volcanics, metasediments and carbonaceous sediments. Samples were also selected for two (2) gold grades, at approximately 1 g/t Au and 2 g/t Au. As a result, 18 master composites were formed. Representative core samples were selected and are illustrated in Figure 13.1.

Figure 13.1: Metallurgical Sample Location



Source: G Mining Services, 2023

13.3 Intensive Cyanidation Test Work

Samples from the 18 master composites were initially subjected to an intense cyanidation test to determine preliminary gold extractions. Test conditions were a primary grind size of P80 of 75 µm, pH 10.5, 10,000 ppm NaCN, oxygen sparged and 48-hour leach time. Intensive leach results are summarized in Table 13.1.

Table 13.1: Intensive Leach Results

Test	Sample ID	Weathering Type	Geological Unit	% Au Extraction at 48 h	Measured Head	Calculated Head	Tails
					Au Assay	Au Assay	Au Assay
					g/tonne	g/tonne	g/tonne
CN-01	MET23-HFC-01	Fresh Rock	Carbonaceous	97.1	1.81	3.06	0.09
CN-02	MET23-HFS-02	Fresh Rock	Metasediments	95.7	1.16	1.16	0.05
CN-03	MET23-HFV-03	Fresh Rock	Volcanics	83.2	1.57	1.04	0.18
CN-04	MET23-HSC-04	Saprolite	Carbonaceous	99.0	1.62	2.09	0.02
CN-05	MET23-HSS-05	Saprolite	Metasediments	95.7	1.34	1.29	0.05
CN-06	MET23-HSV-06	Saprolite	Volcanics	99.4	1.44	1.61	0.01
CN-07	MET23-HTC-07	Transition	Carbonaceous	96.3	1.81	1.89	0.07
CN-08	MET23-HTS-08	Transition	Metasediments	92.1	2.04	2.02	0.16
CN-09	MET23-HTV-09	Transition	Volcanics	94.2	1.81	1.89	0.11
CN-10	MET23-LFC-10	Fresh Rock	Carbonaceous	94.7	1.12	1.22	0.07
CN-11	MET23-LFS-11	Fresh Rock	Metasediments	92.9	1.00	1.20	0.09
CN-12	MET23-LFV-12	Fresh Rock	Volcanics	89.7	1.05	0.87	0.09
CN-13	MET23-LSC-13	Saprolite	Carbonaceous	96.1	0.40	0.77	0.03
CN-14	MET23-LSS-14	Saprolite	Metasediments	98.1	0.59	0.67	0.02
CN-15	MET23-LSV-15	Saprolite	Volcanics	98.5	0.54	0.67	0.01
CN-16	MET23-LTC-16	Transition	Carbonaceous	95.1	0.84	1.21	0.06
CN-17	MET23-LTS-17	Transition	Metasediments	90.4	0.93	0.73	0.07
CN-18	MET23-LTV-18	Transition	Volcanics	89.4	0.28	0.28	0.03

The average leach results per weathering type are summarized in Table 13.2.

Table 13.2: Average Intensive Leach Gold Recoveries

Weathering Type	Average % Au Extraction
Saprolite	97.8%
Transition	92.9%
Fresh Rock	92.2%

13.4 Comminution Test Work

The Volcanic and Metasediment samples were subject to a series of comminution testing, i.e. SAG Mill Comminution (SMC), Bond Abrasion (Ai), Bond Rod Work Index (BRWi) and Bond Ball Work Index (BBWi) determinations. Three (3) of the samples were not physically competent for comminution test work. The BBWi was performed at a closing screen size of 106 microns. Comminution test results are summarized in Table 13.3. No Bond low-energy impact (CWi) testing or unconfined compressive strength (UCS) testing was completed and a calculation to infer a CWi value based on Axb and SG data was completed as a conservative design criteria.

Table 13.3: Comminution Summary

Sample ID	SMC Axb	Ai	Bond Rod Mill Work Index				Bond Ball Mill Work Index @ 106 µm CSS			
			F80 µm	P80 µm	Gpr	BRWI kWh/t	F80 µm	P80 µm	Gpr	BBWI kWh/t
MET23-HFS-02	31.6	0.164	9944	916	7.63	16.4	2916	71	1.32	13.3
MET23-HFV-03	35.4	0.123	9988	921	6.63	17.9	3253	70	1.08	15.5
MET23-HSS-05	390	0.054	8699	847	42.5	5.5	1716	71	1.90	10.5
MET23-HSV-06	-	0.049	-	-	-	-	-	-	-	-
MET23-HTS-08	116	0.055	8649	876	29.5	7.0	1669	76	3.33	7.0
MET23-HTV-09	198	0.016	7672	854	31.0	6.9	1370	73	2.74	8.1
MET23-LFS-11	33.3	0.114	9740	920	7.40	16.8	2897	73	1.33	13.5
MET23-LFV-12	35.0	0.129	9537	917	8.32	15.7	2849	73	1.28	14.0
MET23-LSS-14	-	-	-	-	-	-	-	-	-	-
MET23-LSV-15	-	-	-	-	-	-	-	-	-	-
MET23-LTS-17	228	0.028	9021	871	33.27	6.4	1817	76	3.40	6.8
MET23-LTV-18	193	0.002	8115	860	43.1	5.6	1302	72	4.34	5.6

As some of the transition samples appeared to be softer than the saprolite samples, both transitional and saprolite test results were grouped together for material characterization analysis. The saprolite BBWi of 10.5 kWh/tonne was excluded from the material interpretation. The resulting combined saprolite and transition 85th percentile BBWi of 7.6 kWh/tonne was determined to be in good agreement with similar saprolite oxide ores from other projects in the Guiana Shield.

In general, the fresh rock is more competent, hard and abrasive in comparison to the saprolite and transition material. The fresh rock exhibits competent material (Axb 15th percentile of 32.4), hard grindability (85th percentile BBWi of 14.8 kWh/tonne) and is mildly abrasive (Ai of 0.133).

The saprolite and transition material is very soft (Axb 15th percentile of 162), with soft grindability (85th percentile BWi of 7.6 kWh/tonne) and is mildly abrasive (Ai of 0.034).

Comminution design criteria is based on processing 100% fresh rock and material characteristics are summarized in Table 13.4.

Table 13.4: Comminution Design Parameters (Fresh Rock)

Parameter	Value	Notes
Ai	0.133	Average
CWi	22.2 kWh/tonne	Inferred calculation
BRWi	17.4 kWh/tonne	85th percentile
BBWi	14.8 kWh/tonne	85th percentile
Axb	32.4	15th percentile
Specific Gravity (sg)	2.83	Average

13.5 Chemical Analysis

Duplicate head cuts were removed and assayed for elements of interest in the project using standard assaying techniques. A multi-element ICP analysis was performed on a single head cut from each sample.

Gold content in the samples varied between 0.50 and 2.48 g/tonne (as expected) and silver ranged between 0.1 and 1.8 g/tonne. Sulphur in the samples measured between 0.01 and 0.77 percent, indicating a relatively small sulphide mineral component.

Base metal content in the samples were low, i.e. 67 ppm Cu, 88 ppm Zn, 7 ppm Pb (average). Arsenic and mercury levels were low at 8.5 ppm As and <1 ppm Hg.

The Preg-Robbing Value (PRV) of select samples was measured and indicated near negligible values according to the Preg-Robbing Index (PRI). The samples were subject to a 48-hour intensive leach test to determine the cyanide soluble gold content. Results are provided in Table 13.5.

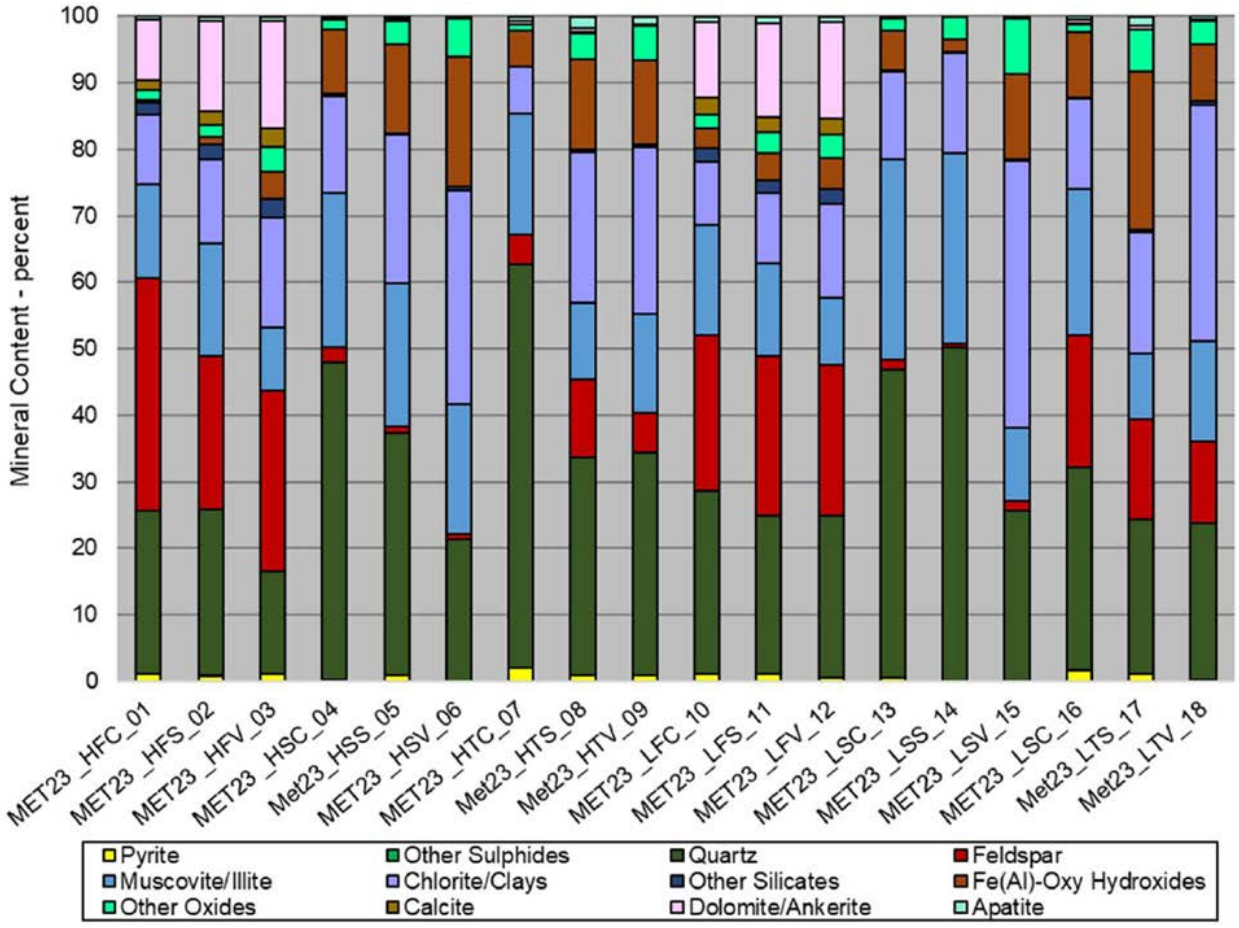
Table 13.5: Chemical Analysis

Sample ID	Au g/t	Ag g/t	Fe %	S %	C %	TOC %	PRV	PRI	CN Soluble Au%
MET23-HFC-01	1.81	0.2	3.94	0.56	1.89	0.01	-0.02	0	97
MET23-HFS-02	2.11	0.2	5.69	0.30	2.56	0.03	-	-	96
MET23-HFV-03	1.81	0.3	9.50	0.56	3.23	0.01	-	-	83
MET23-HSC-04	2.48	0.3	4.83	0.21	0.69	0.16	0.05	0.01	99
MET23-HSS-05	1.75	1.1	7.70	0.25	0.44	0.07	-	-	96
MET23-HSV-06	1.99	0.4	13.5	0.07	0.15	0.05	-	-	99
MET23-HTC-07	2.08	0.3	3.60	0.77	0.57	0.04	0.05	0.01	96
MET23-HTS-08	2.25	0.4	10.7	0.44	1.09	0.03	-	-	92
MET23-HTV-09	2.09	1.1	11.4	0.55	0.51	0.02	-	-	94
MET23-LFC-10	1.12	0.1	5.66	0.51	2.72	0.01	-0.03	0	95
MET23-LFS-11	1.03	0.3	7.90	0.76	2.90	0.02	-	-	93
MET23-LFV-12	1.24	0.3	8.90	0.42	3.02	0.01	-	-	90
MET23-LSC-13	1.00	0.1	3.01	0.12	0.39	0.11	-0.10	0	96
MET23-LSS-14	0.91	1.8	1.42	0.01	0.04	0.02	-	-	98
MET23-LSV-15	0.70	0.3	11.5	0.02	0.09	0.03	-	-	99
MET23-LTC-16	0.91	0.2	5.89	0.77	0.93	0.06	-0.07	0	95
MET23-LTS-17	1.29	0.4	13.5	0.72	1.63	0.03	-	-	90
MET23-LTV-18	0.50	0.3	10.6	0.11	0.38	0.02	-	-	89

13.6 Mineral Analysis

Bulk Mineral Analysis (BMA) using QEMSCAN, was conducted on the samples. This assessment provides an unsized measure of mineral composition (refer to Figure 13.2). This analysis also provides information regarding sulphur distribution in the samples (refer to Figure 13.3).

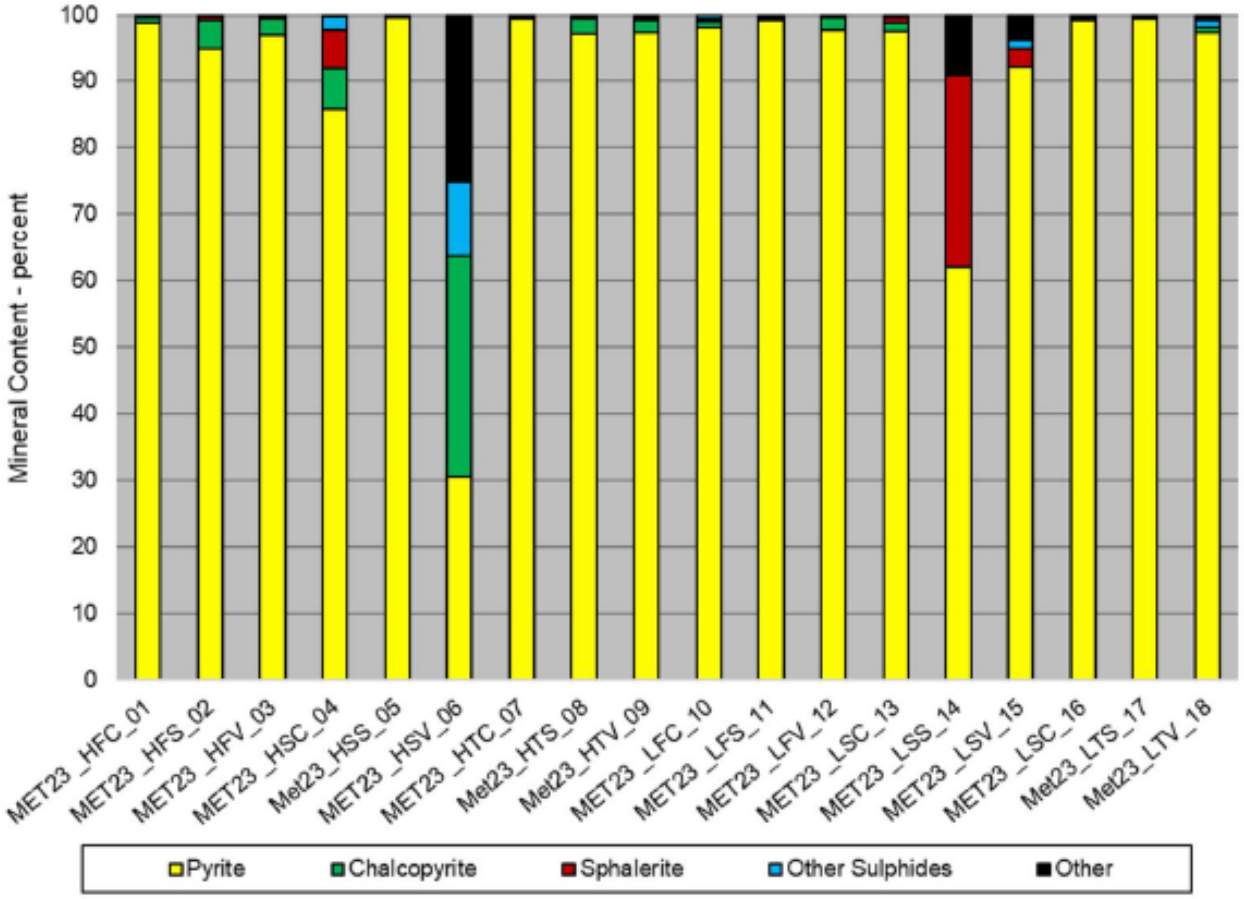
Figure 13.2: Mineral Content



Pyrite accounted for the main sulphide mineral in almost all the samples. Sulphide in sample MET23-HSV-06 (saprolite sample) was dominated by chalcopyrite, and MET23-LSS-14 (saprolite sample) had a higher concentration of sphalerite. Chalcopyrite, sphalerite and other sulphides were also detected in lower concentrations in the majority of the other samples.

The non-sulphide suite of minerals varied, consisting mainly of quartz, feldspars, muscovite / illite and chlorite and clays.

Figure 13.3: Mineral Analysis – Sulphur Distribution



13.7 Gravity Test Work

Gravity Recoverable Gold (E-GRG) tests were performed on all samples. These tests were performed by first preparing 10 kilograms of the feed to 100% passing 1.7 mm and passing the entire crushed material through a Knelson MD-3 concentrator at a force of 60-Gs. The concentrate was retained and sized for assay; the tailings were sub-sampled for sizing.

The tailings were ground in a laboratory rod mill to a grind target of P80 of 212 µm and processed through the Knelson Concentrator a second time (Pass 2), the concentrate and tailings were sampled for assay and sizing as per the initial pass before regrinding the tailings to a final target of P80 of 75 µm K80 and repassing through the Knelson concentrator (Pass 3). The final tailings were sampled, sized, and assayed by size. All assaying included gold by fire assay with concentrate fractions assayed to extinction.

For MET23-LSV-15, the feed sizing was much finer than P80 of 1.7 mm and the gravity recoverable test was performed with only two (2) passes.

Gravity gold recovery for the fresh rock samples ranged between 36% and 63% with gold concentrates assaying between 40 and 102 g/tonne. The high-grade samples resulted in higher gravity recoverable gold. Gravity gold recovery for the saprolite samples ranged between 27% and 46% with gold concentrates assaying between 23 and 65 g/tonne. For these samples, the low-grade samples resulted in higher overall gravity recoverable gold.

Based on the favourable GRG results, it is recommended to include a gravity circuit in the flowsheet.

13.8 Whole-of-Ore Leach Tests

A series of whole-of-ore leach (WOL) tests were performed on the 18 samples, evaluating primary grind size of P80 of 75 µm, 88 µm and 105 µm. Test conditions for these tests consisted of 1,000 ppm NaCN, pH 10.5 and sparged with oxygen. Tests were performed at a feed density of 40 percent solids.

Overall, the samples responded very well to cyanide leaching with gold extraction ranging between 88.0% and 99.7%, averaging 94.7% (refer to Table 13.6). Silver extraction ranged between 28% and 95%, averaging 65%.

Results indicated that for most of the samples, an increase in both gold and silver extraction was obtained, as primary grind size decreased. Generally, higher sodium cyanide consumption was also measured at finer primary grind size.

13.9 Gravity-Leach Tests

A series of subsequent gravity-leach tests were performed on the 18 samples, evaluating primary grind size of P80 of 75 µm, 88 µm and 105 µm. Test conditions for these tests consisted of 1,000 ppm NaCN, pH 10.5 and sparged with oxygen. Tests were performed at a feed density of 40 percent solids.

For these tests, similarly, gold extraction was high, ranging between 90.1% and 99.2%, averaging 95.4% (refer to Table 13.6). Gold recovery to the gravity concentrate varied, ranging from 6.1% to 77.3%, averaging 31.7%. Gold extraction slightly increased with finer grind sizes.

13.10 Carbon-in-Leach Tests

The 18 samples were subject to a series of carbon in leach (CIL) tests at a primary grind size of K80 of 75 µm. Test conditions consisted of 500 ppm NaCN, pH 10.5, sparged with air and with an addition of 15 g/L carbon. Tests were performed for a duration of 48 hours.

For these tests, gold extraction ranged between 90.7% and 97.9%, averaging 94.6% (refer to Table 13.6).

13.11 Gravity-CIL Tests

The next set of tests evaluated a gravity circuit ahead of CIL testing, with the same CIL test conditions as the previous set of direct CIL tests. Primary grind size of P80 of 75 µm was used.

For these tests, overall gold extraction was similar. Gold on average was 23.0% recovered into the gravity concentrate and overall gold extracted was between 90.9% and 97.9%, averaging 94.0% (refer to Table 13.6).

Table 13.6: Overall Gold Extraction Results

Sample ID	Overall Gold Extraction per Flowsheet (%) at 48 Hours							
	Direct Leach			Gravity-Leach			CIL	Gravity-CIL
	105 µm	88 µm	75 µm	105 µm	88 µm	75 µm	75 µm	75 µm
MET23-HFC-01	90.9	92.3	92.7	93.2	90.1	92.7	91.8	91.3
MET23-HFS-02	95.5	96.4	96.5	96.7	96.3	97.5	95.5	96.2
MET23-HFV-03	88.9	89.7	92.5	92.8	92.9	93.3	93.0	92.5
MET23-HSC-04	96.2	98.7	98.9	98.5	99.0	98.4	97.9	97.9
MET23-HSS-05	93.3	92.5	94.1	91.5	91.9	94.3	93.8	90.9
MET23-HSV-06	97.7	98.1	98.3	94.8	98.4	99.2	97.3	97.4
MET23-HTC-07	94.8	96.4	97.9	97.7	97.2	98.4	97.2	95.4
MET23-HTS-08	91.8	92.7	93.6	92.6	94.7	95.2	92.8	93.1
MET23-HTV-09	94.8	95.4	95.9	97.6	96.3	95.4	94.8	94.6
MET23-LFC-10	93.1	96.3	96.3	95.3	94.6	94.1	92.6	93.2
MET23-LFS-11	90.2	88.0	90.6	91.3	92.3	94.3	90.7	92.0
MET23-LFV-12	93.9	92.8	95.1	94.2	94.6	94.5	92.5	92.8
MET23-LSC-13	91.8	94.5	98.1	91.4	95.8	95.2	95.2	93.4
MET23-LSS-14	99.7	98.7	98.5	98.1	98.7	98.9	97.5	96.8
MET23-LSV-15	96.5	97.2	98.4	98.2	97.0	98.9	97.4	95.6
MET23-LTC-16	94.2	92.9	93.9	93.2	96.4	96.1	93.1	93.6
MET23-LTS-17	90.9	91.0	93.1	92.9	93.2	95.4	93.1	91.7
MET23-LTV-18	95.9	96.8	92.7	96.3	96.6	98.2	97.3	94.5

13.12 Cyanide Destruction Tests

A total of five (5) blended composites were constructed from the low- and high-grade samples from each geological unit, except for the carbonaceous samples. These composites were subject to a series of bulk gravity-CIL tests to generate products for downstream cyanide destruction testing (via SO_2 / Air method) as well as mineralogy and environmental assay analysis on the detox tailings. The cyanide destruction results are summarized in Table 13.7.

The target CN_{WAD} value of <1 ppm was achieved for all six (6) composites at an SO_2:CN_{WAD} ratio of 4. Copper addition was not required for some samples and up to 100 mg/L was required for other samples.

Table 13.7: Cyanide Destruction Summary

Composite Name	Detox Test	Test Parameters				Test Length		Feed / Detox Solution Assays (ppm)				
		pH	Time (min)	SO_2 g/g CN_{MP}	Cu mg/L	Time (min)	No. of Displ.	CN_{MP}	Cu	Fe	Ni	Zn
SAP-Volcanic T171CNTI	Feed	9.5	-	-	-	-	-	147	1.96	0.5	0.28	0.93
	C1	9.5	120	4	25	240	2	5.02	4.21	3.1	0.27	<0.01
	C2	9.5	120	4	50	240	2	0.53	0.49	4.4	0.23	<0.01
	C3	9.5	120	4	25	240	2	6.01	5.47	4.0	0.31	<0.01
	C4	9.5	120	4	50	480	4	0.25	0.20	5.5	0.21	<0.01
SAP-Metasediments T172CNTI	Feed	10.0	-	-	-	-	-	208	2.89	15.6	0.02	0.84
	C1	8.6	120	4	25	600	5	0.74	0.28	9.7	<0.01	<0.01
	C2	8.6	120	4	0	120	1	1.10	0.29	10.9	<0.01	<0.01
	C3	8.5	120	4	25	600	5	0.95	0.08	11.6	<0.01	<0.01
TRANS-Volcanic T173CNTI	Feed	10.3	-	-	-	-	-	251	5.82	0.5	0.17	0.50
	C1	9.6	120	4	25	240	2	0.38	0.28	0.5	0.02	0.01
	C2	9.5	120	4	0	240	2	0.51	0.29	0.5	0.05	<0.01
TRANS-Metasediments T174CNTI	Feed	10.2	-	-	-	-	-	228	10.1	6.1	0.46	1.09
	C1	9.2	120	4	25	240	2	0.29	0.10	4.5	0.01	0.01
	C2	9.2	120	4	0	60	2	0.36	0.11	4.8	0.01	0.01
FR-Volcanic T175CNTI	Feed	10.0	-	-	-	-	-	179	6.45	55.4	0.26	0.46
	C1	8.8	120	4	25	240	2	0.33	0.32	48.6	0.02	<0.01
	C2	8.7	120	4	0	240	2	0.34	0.30	66.7	0.09	<0.01

Composite Name	Detox Test	Test Parameters				Test Length		Feed / Detox Solution Assays (ppm)				
		pH	Time (min)	SO_2 g/g CN_{MP}	Cu mg/L	Time (min)	No. of Displ.	CN_{MP}	Cu	Fe	Ni	Zn
FR-Metasediments T176CNTI	Feed	10.0	-	-	-	-	-	170	10.6	49.9	0.89	0.25
	C1	8.5	120	4	25	120	1	1.90	1.42	32.4	0.10	<0.01
	C2	8.5	120	4	50	240	2	1.37	0.39	37.7	0.19	<0.01
	C3	8.8	120	5	50	120	1	1.46	0.24	37.2	0.30	<0.01
	C4	8.7	120	4	100	90	0.75	0.56	0.31	34.5	0.04	<0.01

13.13 Acid Base Accounting Tests

Acid Base Accounting (ABA) tests were completed on the five (5) blended composites (following cyanide destruction testing) and on waste rock samples (hanging wall (HW) and footwall (FW)) and the results summarized in Table 13.8.

All samples, with the exception of one of the transition samples, had net neutralizing potential (NNP) greater than zero, indicating that these materials are potentially acid neutralizing. All samples, except for transition samples, had a neutralizing potential ratio (NPR) greater than 4.1, indicating no potential for ARD. It is recommended to further evaluate the transition samples as part of the variability test work program.

Table 13.8: Acid-Base Accounting Summary

Sample Type	Cyanide Destruction Test No.	Net Neutralizing Potential (NNP)	Neutralizing Potential Ratio (NPR)
Saprolite Tails	171	8.2	53.4
Saprolite Tails	172	3.3	22.3
Trans Tails	173	8.0	2.0
Trans Tails	174	-6.8	0.7
Fresh Rock Tails	175	190.0	16.8
Fresh Rock Tails	176	176.0	13.8
FW	-	156.0	99.9
FW	-	127.0	98.2
FW	-	115.0	237.0
HW	-	102.0	41.2
HW	-	69.0	443.0
HW	-	86.9	133.0

13.14 Gold Recoveries

Gold extractions from the gravity-leach tests produced the best gold extraction. Test conditions were primary grind size of P80 of 75 µm, pH 10.5, 1,000 ppm NaCN, oxygen sparged and 48-hour leach time.

The average gold extraction results per weathering type are summarized in Table 13.9. A discount factor was applied to establish gold recoveries. No fresh rock sample from the potential underground mine was tested, and the recoveries are assumed to be same as the open pit fresh rock samples.

Table 13.9: Gold Recoveries

Weathering Type	Lab Scale Average % Au Extraction	Plant Scale Average % Au Recovery Assumptions
Saprolite	97.5%	96.0%
Transition	96.5%	95.0%
Fresh Rock	94.4%	92.5%

13.15 Recommendations

It is recommended to continue variability metallurgical test work of the main material domains to confirm the metallurgical response across material zones, which includes the following scope of work:

- Head assays and ICP analysis
- Quantitative mineralogy tests
- Comminution tests
- Gravity tests
- Grind-leach determination tests
- Pre-robbing tests
- Gravity and gravity tails leach and CIL tests
- Cyanide destruction tests
- Sequential triple contact carbon lading tests
- Oxygen uptake tests
- Static and dynamic settling tests
- Flocculant screening tests
- Viscosity (shear-rate) tests
- Acid-base accounting

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following chapter presents Mineral Resource Estimates (MRE) for the Oko West deposit. The MRE was prepared by Pascal Delisles, P.Geo., Director of the Geology and Resources at G Mining Services (GMS), and Émile Boily-Auclair, P.Eng. in Mineral Resources Estimation at GMS. The MRE presented in this section, along with all steps leading to its completion, has been revised and approved by Mr. Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services and independent Qualified Person (QP) as defined in the National Instrument 43-101.

GMS personnel and the QP visited the Oko West Project twice, between May 2022 and January 2024, to review the geological data, drilling program, and sampling protocols. Independent verification samples from drill cores were collected during the first and last site visit by GMS personnel (see Section 12.

The MRE was prepared following the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves. The effective date of the mineral resource estimation is February 7, 2024, and the MRE statement is listed in Table 14.1.

Table 14.1: In-pit and Underground Updated Mineral Resource Estimate at Oko West

Category	Updated MRE Tonnage (kt)	Updated MRE Au grade (g/t)	Updated MRE Contained Gold (koz)
Pit Constrained Resource			
Indicated	64,115	2.06	4,237
Inferred	8,107	1.87	488
Underground Constrained Resource			
Indicated	**491**	**1.85**	**29**
Inferred	**11,510**	**3.01**	**1,116**
Total Open Pit and Underground			
Indicated	**64,606**	**2.05**	**4,266**
Inferred	**19,617**	**2.54**	**1,603**

**Notes on Mineral Resources:*
The Mineral Resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).

1. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
2. *The effective date of the Mineral Resource Estimate is February 7, 2024.*
3. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g/t Au in saprolite and alluvium / colluvium, 0.313 g/t Au in transition, and 0.37 g/t Au in fresh rock.*
4. *Underground Mineral Resources are reported inside potentially mineable volume and include below cut-off material (stope optimization cut-off grade of 1.38 g/t Au):*
    a. *A change in the reporting method for the underground part of the deposit explains the differences in tonnage and average grade between this PEA and the MRE published in February 2024. Tonnage of potentially mineable material stated below cut-off (i.e., must take material) is declared for this constrained underground Mineral Resource Estimate. Blocks have been reclassified inside each stope based on deposit knowledge and continuity and reflect the existing classification. No changes in total ounces are observed.*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g/t Au was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 0.5 m x 2.5 m, and has been reported inside an optimized pit shell and optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Inverse Distance for domains AU_2A, AU_2B and AU_5, and Ordinary Kriging for all other domains. Capping was applied on eight domains, ranging from 5 g/t Au to 80 g/t Au.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$14.51/t for saprolite and alluvium/colluvium, with a 96.0% processing recovery US$17.16/t for transition with a 95.0% processing recovery and US$19.80/t for fresh rock based on 92.5% processing recovery.*
    c. *Inter-ramp angles of 30° in saprolite and alluvium / colluvium, 40° in transition and 50° in fresh rock.*
    d. *Royalty rate of 8%.*
10. *UG optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$73.26/t for fresh rock.*
    c. *Stope height of 30 m, strike length of 20 m, maximum width of 25 m and minimum width of 2 m.*
    d. *The Deswik.SO (DSO) was used to constrain the Resources.*
    e. *Royalty rate of 8%.*
11. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces. The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*

12. *These Mineral Resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*
13. *These Mineral Resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

The total pit constrained Indicated Mineral Resource is reported at 64,115 kt grading 2.06 g/t Au, for a total of 4,237 koz of gold. The total pit constrained Inferred Mineral Resource is reported at 8,107 kt grading 1.87 g/t Au, for a total of 488 koz of gold. The underground Resources are estimated from zones outside the constrained Resources of the open pit. The total constrained underground Indicated Mineral Resource is reported at 491 kt grading 1.85 g/t Au, for a total of 29 koz of gold. The total constrained underground Inferred Mineral Resource is reported at 11,510 kt grading 3.01 g/t Au, for a total of 1,116 koz of gold. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The QP has determined that there are no known factors or issues that could significantly impact the Mineral Resource Estimate (MRE), other than the typical risks associated with mining projects, such as environmental, permitting, taxation, socio-economic, marketing, and political factors, as well as additional risk factors related to indicated and inferred mineral resources.

It was determined that the database used for estimation is reliable, and that the current drilling information is of sufficient quality for interpreting the boundaries of gold mineralization with confidence. Additionally, the assay data used for the mineral resource estimation and block modelling is considered reliable by the QP. The mineral resource estimation methodology and key assumptions considered for the MRE are described in the following sections.

14.2 Estimation Methodology

The mineral resources presented in this Report have been estimated through interpolation into a sub-block model using the modelled mineralized zones of the deposit.

The estimation methodology is summarized below:

- Drillhole database validations and selection of the drillholes to be included in the mineral resource estimation;
- 3D modelling of host units (lithological model) based on available geological data (drill logs, surface geophysics surface plan maps, drill core photography, etc.) using Leapfrog Geo™ 2023.2.1;
- 3D modelling of gold-bearing domains based on geology model, strain, alteration (type and intensity), Sulphide content, assay results and drill core photography using Leapfrog Geo™ 2023.2.1;

- Geostatistical analysis for data conditioning: mineralization domain validation, density assignment, capping assumptions, compositing and variography using Leapfrog Geo™ 2023.2.1 and Supervisor™ v. 8.15.1 ;
- Block modelling and grade estimation using Leapfrog Edge™ 2023.2.1;
- Resource classification and grade interpolation validations; and
- Grade and tonnage sensitivities to different cut-off grade scenarios.

14.3 Resource Database

On February 7, 2024, GMS received from the company an extract of the database in the form of a series of comma-separated spreadsheets containing information on the Oko West deposit. The database includes information such as collar locations, drillhole types, downhole surveys, assay results, drill logs, density measurements and various geological interpretations.

Drillhole collar elevations were verified using a 50-cm resolution digital elevation model (DEM) provided to GMS. One (1) drillhole collar (OKWR22-289) was pressed to the DEM after differences of more than 2 m were observed between the collar elevation and the DEM. All other drillhole collar elevations were consistent with the DEM and were left unchanged. Drillhole surveys were verified for inconsistencies using Leapfrog Geo™ 3D viewer and a GMS validation script. Variations in dip and direction of more than 5 degrees per 100 m were flagged and investigated for consistency. Following the investigation, only two (2) surveys (DDH OKWD22-134 at 42.63 m downhole and DDH OKWD22-174 at 119.00 m) were discarded from the resource database.

Diamond drillhole (DDH) twin holes were completed to validate the inclusion of Reverse Circulation (RC) drillholes in the MRE. After review, the QP is confident that the RC drilling is a good representation of gold grades. Only one (1) RC drillhole (OKWR22-128) was discarded from the resource database after potential smearing of gold grades was observed from the adjacent DDH twin hole (OKWD22-127). Finally, one (1) trench was taken out of the database because it was completely sampled in unconsolidated alluvial material (OKWT20-006).

Some drillholes from previous drill campaigns on the Oko West Project have been drilled downdip of the mineralized shear zones resulting in local clusters of assays not reflecting the current drill spacing of the Project. This inconsistency in distribution is affecting the accuracy of the resource estimation. To address this issue, drillholes from the original database were excluded from the resource estimate if the assayed intervals of mineralization were more than three (3) times the average length of the mineralized domains.

This step ensures that the resource estimation is based on data that better reflects the current drill spacing and orientation of mineralized lenses of the Project.

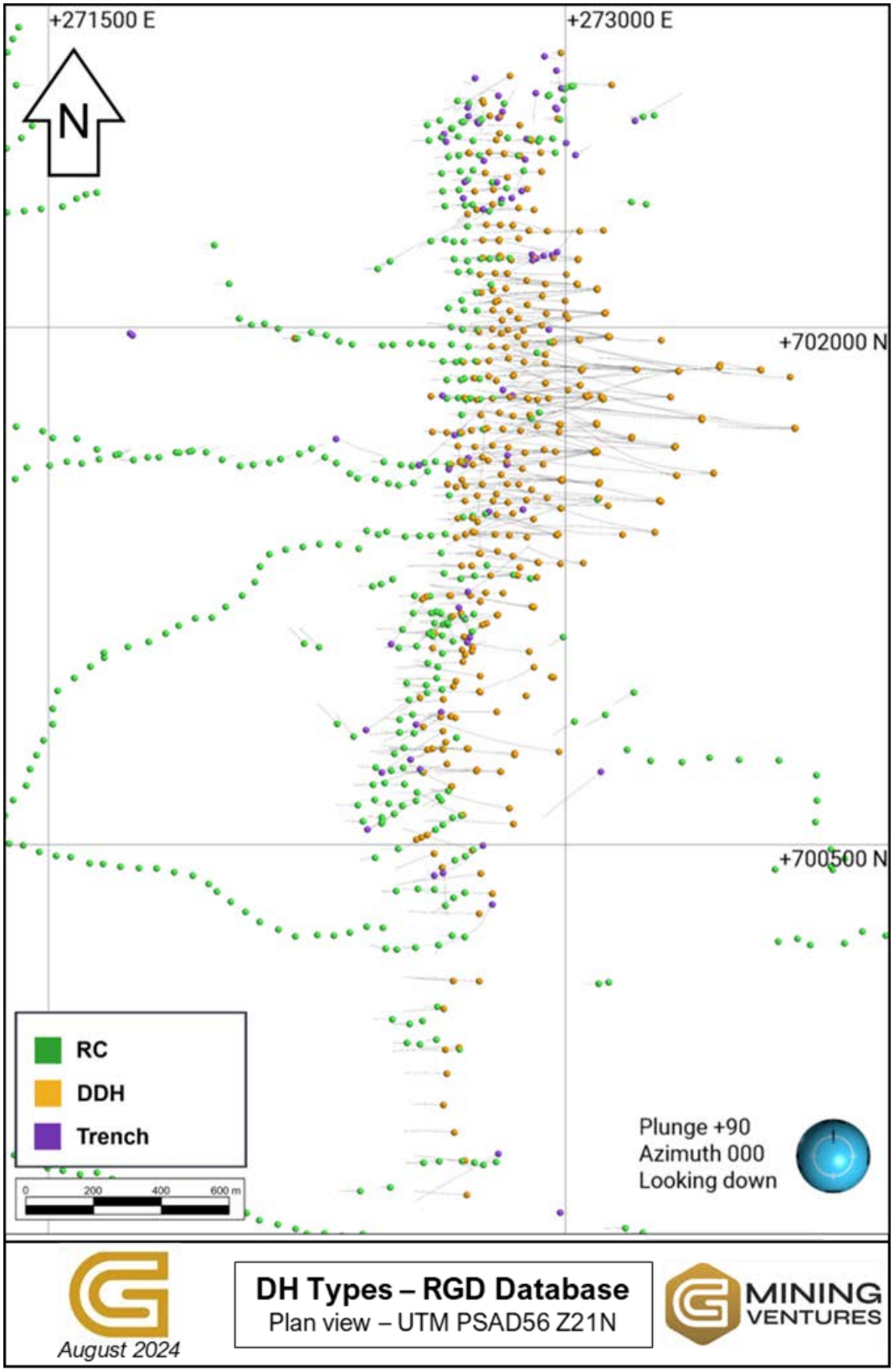
Table 14.2 summarizes the original drillhole database received from the company and the filtered one used for the resource estimation of the Oko West deposit. Figure 14.2 presents a plan view of the Oko West drillhole collars used in the estimate.

Table 14.2: Summary of Drillholes and Assays Used in the Oko West Resource Estimate

Bore Hole Types	Original Database				Resource Estimate Database			
	Total Number of Drillholes	Total Drilled Length (m)	Total Assayed Length (m)	Total Number of Assays (#)	Total Number of Drillholes	Total Drilled Length (m)	Total Assayed Length (m)	Total Number of Assays (#)
Trenches	85	8,735.3	7,803.7	4,048	59	6,707.5	6,002.5	3,132
RC	1,760	52,926.0	45,089.0	42,279	292	21,808.5	21,208.5	21,159
DD	414	124,837.7	79,284.1	64,836	382	121,154.9	76,927.2	63,336
Wedges (DD)	19	6,542.1	3,317.3	3,136	15	5,321.9	3,215.7	3,049

**Note: Despite different resource database numbers from previous MRE, the exact same database was used. Only the methodology to estimate the database statistics changed.*

Figure 14.1 : Drillhole Database, Coloured by Drillhole Type



Source: GMS, 2024

14.4 Geological Models

14.4.1 Lithology and Weathering Models

A 3D geological model was provided by the company and validated by GMS. All units are separated by an interpreted east-south-east subvertical fault. This fault separates Block 4 to the North from Block 5 to the South. The modelled units are presented below, from the footwall (West) to the hanging wall (East) of the mineralized domains:

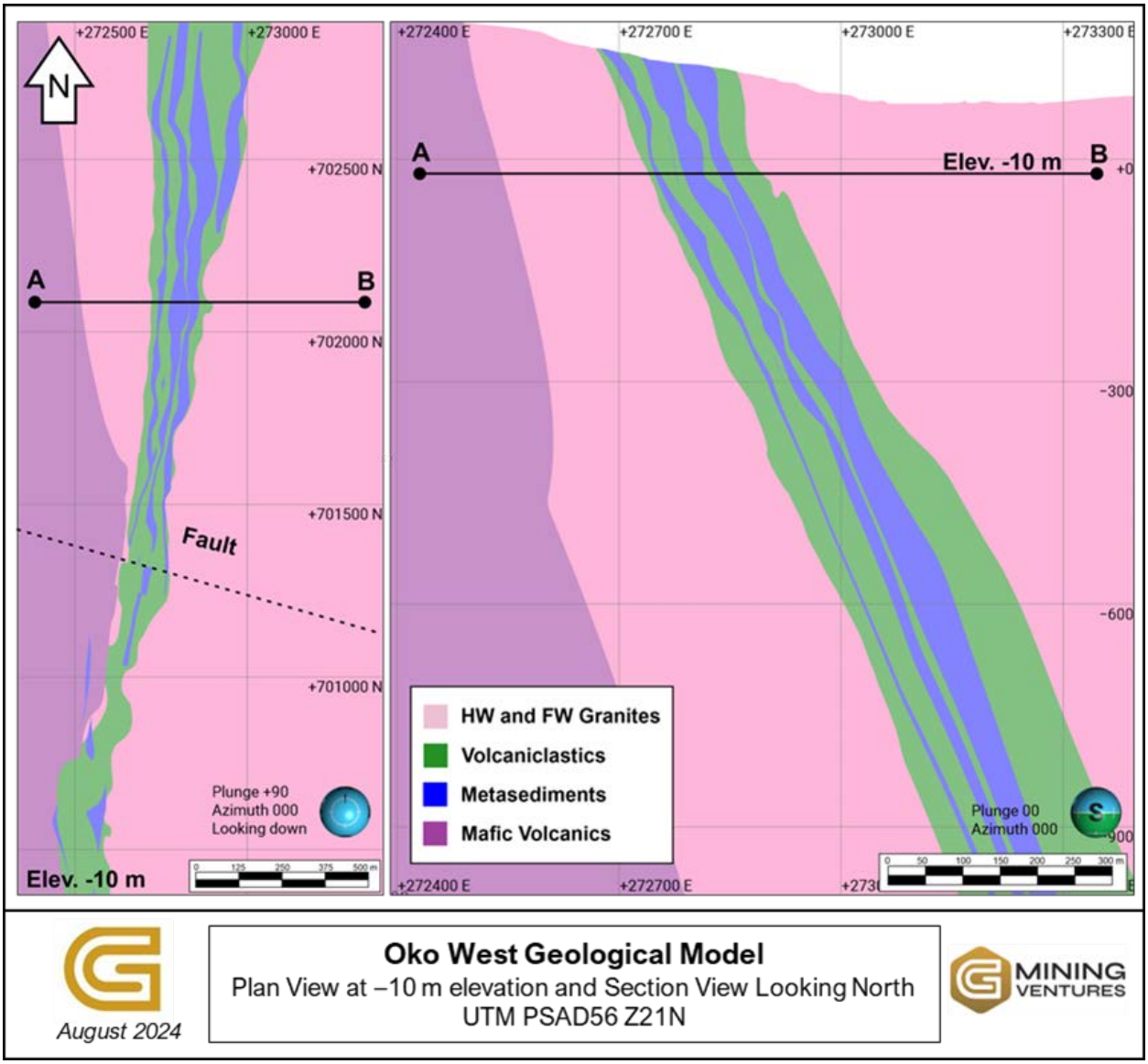
1. Mafic volcanics (MBAS)
2. Footwall granite (GRAFW)
3. Volcanoclastic units (VOLC)
4. Sedimentary package, including, carbonaceous sediments (MSED)
5. Hanging wall granite (GRAHW)

A weathering model was created by GMS using simplified drillhole logs. An alluvium / colluvium model was integrated to the weathering model and was created using lithology and alteration logs. Manual selections and editing were completed to smooth surfaces and avoid over-estimation of the alluvium model. The weathering model units are presented below:

1. Alluvium / colluvium (OVBN)
2. Saprolite (SAP)
3. Transition material (TRANS)
4. Fresh rock (FRSH)

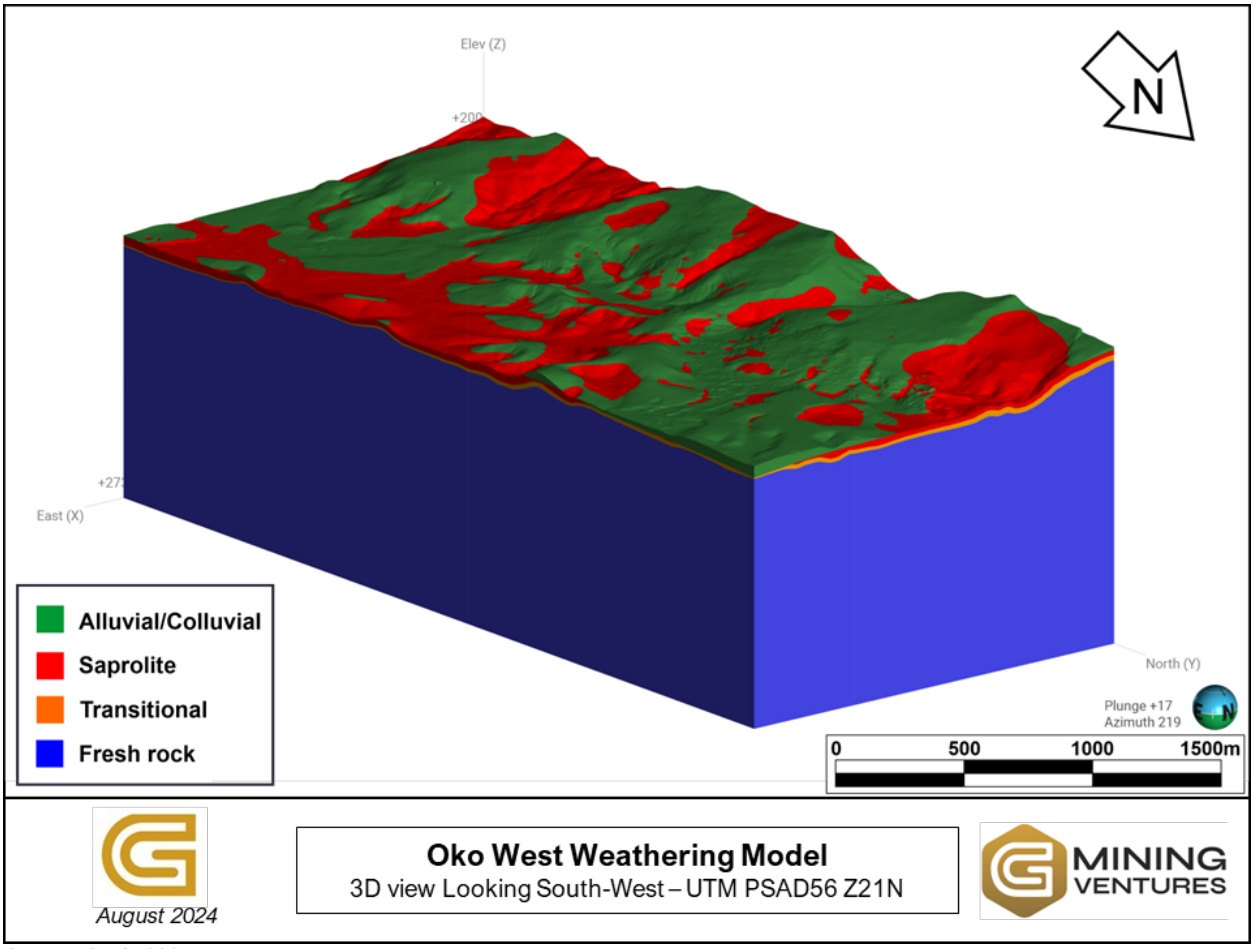
Both models are created in Leapfrog Geo™ and are used to assign density in the block model. The geological model was also used to guide the mineralization domain interpretation. Figure 14.2 and Figure 14.3 presents the geological and weathering models used for this resource estimate.

Figure 14.2: Geological Model Plan View and Vertical Section



Source: GMS, 2024

Figure 14.3: Weathering Model Isometric View



Source: GMS, 2024

14.4.2 Mineralization Model

The drillhole assay intervals were used to model eight (8) distinct gold-bearing mineralized domains using Leapfrog Geo™ interval selection method. Most domains are separated by the interpreted east-southeast subvertical fault (Figure 14.4 and Figure 14.5). The mineralization model was built based on several parameters or inputs, such as: lithologies (geological model), shear and brecciation intensity, alteration types and intensity, strain, Sulphide content, vein types, vein density and gold content to properly assign mineralized intervals to their respective domains to preserve domain stationarity. Most of the main zones are running parallel to the footwall granite. A cut-off of 0.30 g/t Au and a minimum true thickness of three (3) metres were used to constrain the mineralized domains. Within the main zone (AU_2), a high-grade sub-domain was modelled using a cut-off of 5.00 g/t Au, resulting in 109 continuous drill intercepts averaging 9.41 g/t Au over an estimated true thickness of 6 m. This high-grade ore domain was modelled using grade, strain intensity, Sulphide content and lithological domain (Figure 14.5). Additionally, a total of six (6) internal waste sub-domains were modelled where continuity of low-grade / unmineralized

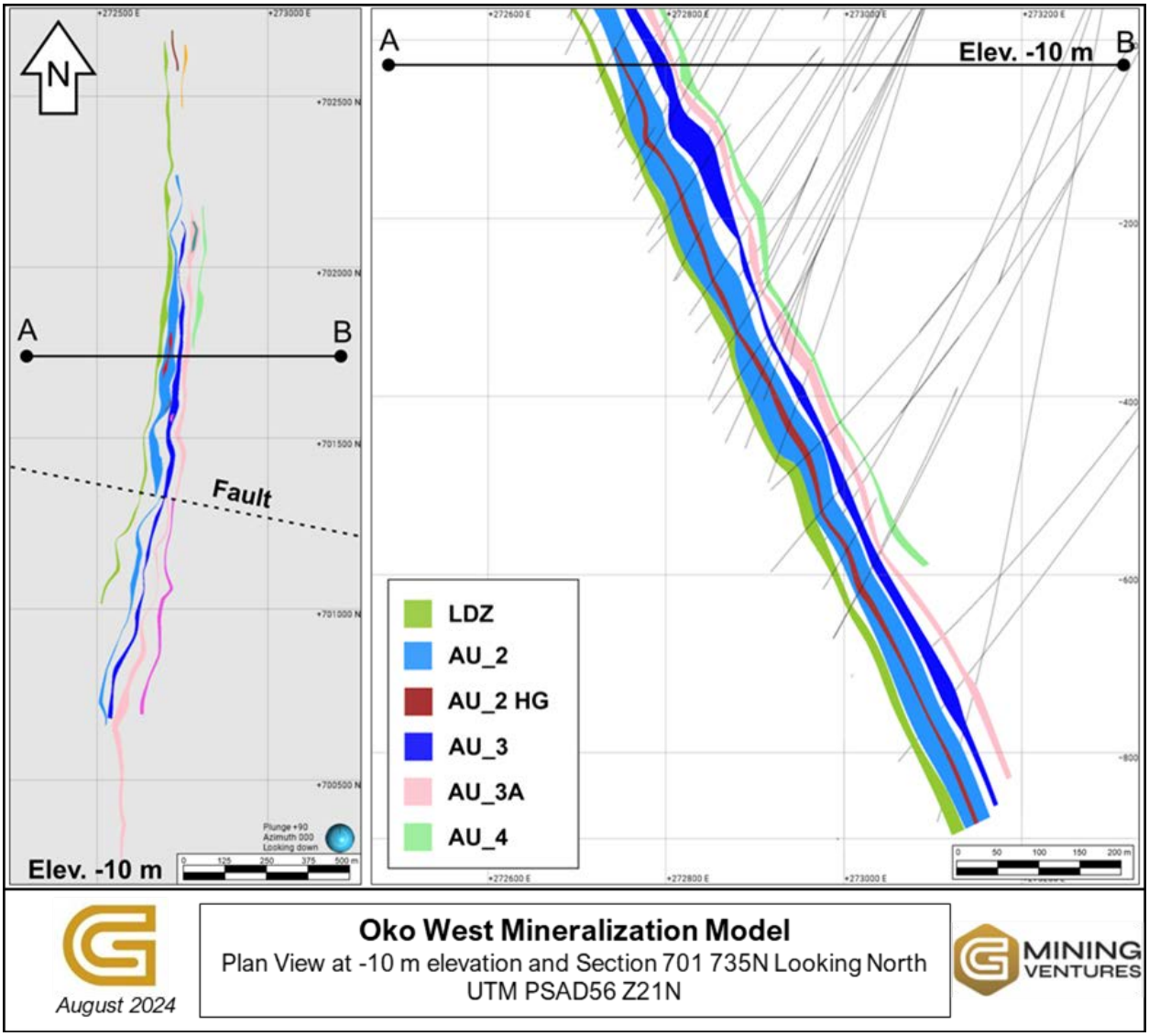
intervals could be demonstrated on more than one (1) section. Sub-domains were created to reduce smearing between grade populations where clear boundaries could be interpreted.

The models were improved by using manual editing such as insertions of points, polylines and structural data. Figure 14.4 presents a plan and cross-section view of the major mineralized domains. The modelled domains are listed below, from footwall to hanging wall by fault block:

- Fault Block 1 (Block 4, northern domain):
  - Lower Deformation Zone (LDZ)
  - AU_2:
    - AU_2_HG
  - AU_2A
  - AU_2B
  - AU_3
  - AU_3A
  - AU_4
- Fault Block 2 (Block 5, southern domain):
  - Lower Deformation Zone (LDZ)
  - AU_2
  - AU_3
  - AU_3A
  - AU_5.

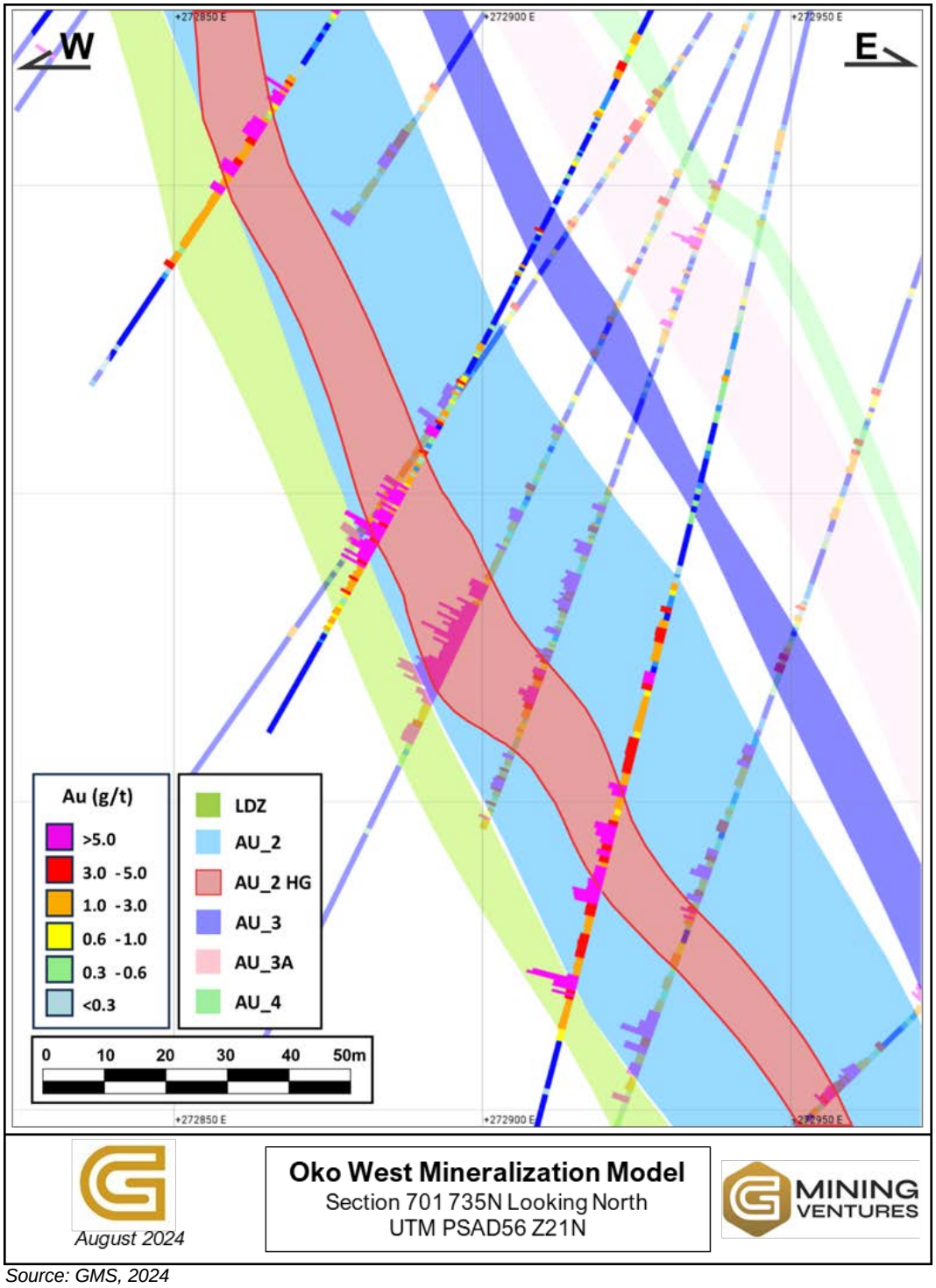
The main mineralized domains (*i.e.*, LDZ, AU_2, AU_3, AU_3A) span over 1,600 m in strike-length at surface and extend to depths of approximately 900 m. All mineralized zones are characterized by a consistent N5° orientation and a general 60° dip to the east.

Figure 14.4: Oko West Mineralization Model



Source: GMS, 2024

Figure 14.5: Oko West Mineralization Model - Zoom In



Source: GMS, 2024

14.5 Assays, Capping and Compositing

14.5.1 Raw Assays

Statistics of the Oko West deposit assays are presented below. The assay values reported below detection limits were assigned half the detection limit for statistical analysis and grade estimation purposes. Missing analysis (3,917) within the database were assigned a value of 0.00 g/t Au. Table 14.3 presents the descriptive statistics of gold assays used for the resource estimations of the Oko West deposit.

Table 14.3: Oko West Gold Assays Statistics (length weighted)

Oko West Assays	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Standard Deviation	CV	Median (g/t Au)	Length (m)
LDZ	3,060	0.00	61.89	1.73	3.66	2.12	0.74	3,257.5
AU_2	6,920	0.00	157.29	1.80	4.16	2.31	0.78	7,092.5
AU_2 HG	824	0.05	84.67	8.99	9.61	1.07	6.40	708.2
AU_2A	44	0.00	42.57	1.18	5.26	4.45	0.37	45.5
AU_2B	94	0.00	4.91	0.69	0.85	1.22	0.42	103.0
AU_3	2,609	0.00	1,106.05	2.13	18.10	8.52	0.63	2,737.3
AU_3A	2,618	0.00	65.84	1.38	3.13	2.27	0.53	2,765.0
AU_4	950	0.00	129.09	1.57	6.06	3.86	0.53	962.5
AU_5	179	0.00	3.09	0.49	0.57	1.15	0.30	218.0
Total	**17,298**	**0.00**	**1,106.05**	**2.02**	**8.35**	**4.14**	**0.71**	**17,889.5**

14.5.2 Capping

Capping is a technique used to mitigate the impact of outliers, specifically extremely high-grade values, on the estimation of mineral resources. It involves establishing a threshold or limit on the maximum value that can be utilized in the estimation process.

Capping analysis was conducted independently for each mineralized domain, examining assay statistics (including, but not limited to, coefficient of variation), histograms, cumulative probability plots, and conducting decile analyses to identify domains for potential grade capping. Additionally, the spatial distribution of outliers was assessed in three dimensions (3D) to identify any high-grade clusters or localized high-grade areas within the mineralized domains that would warrant higher capping.

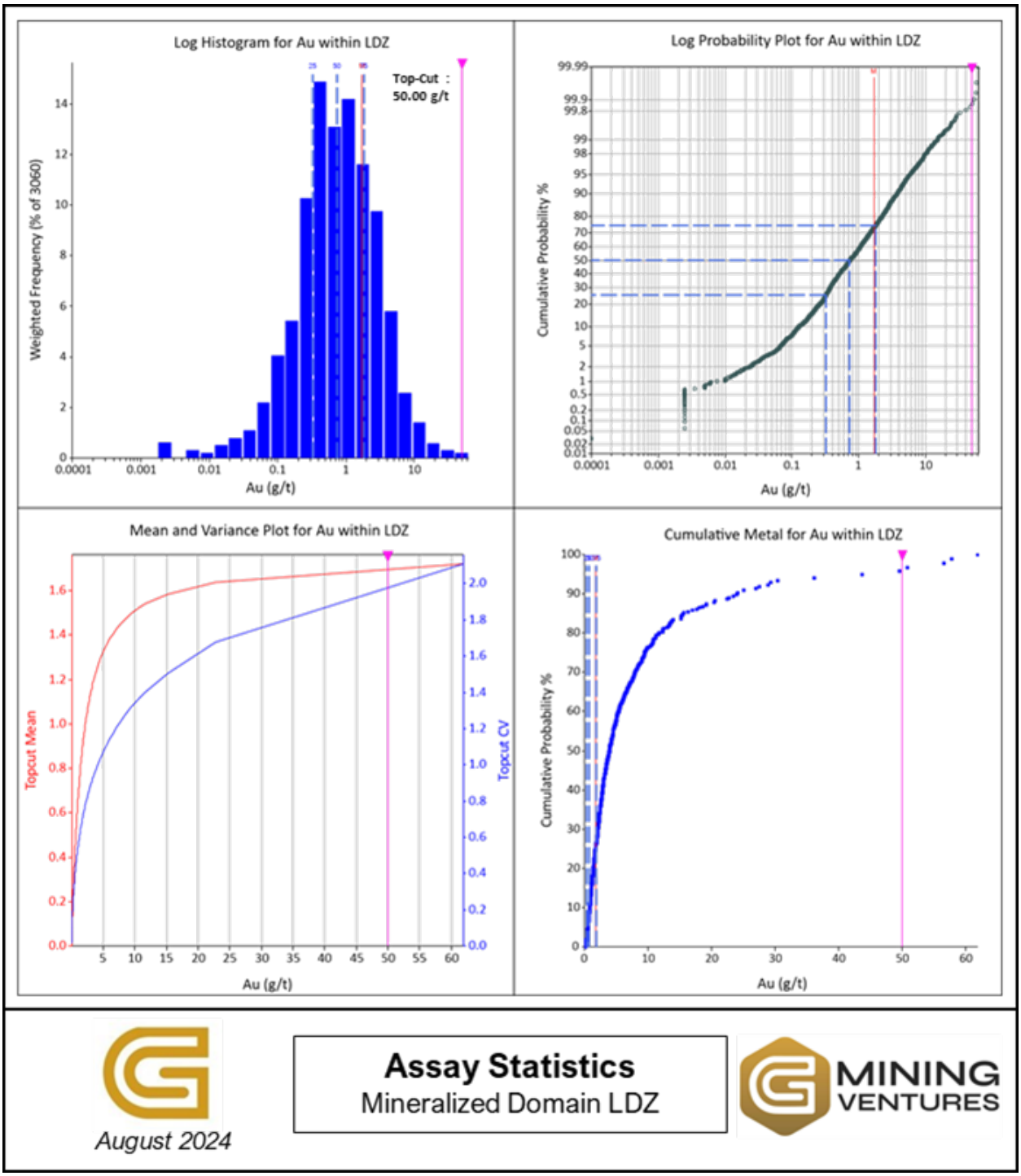
Capping assumptions for Oko West mineralized domains are presented in Table 14.4.

Table 14.4 : Capping Applied to Oko West Mineralized Domains

Oko West Domains	Capping (Au g/t)
LDZ	50.0
AU_2	45.0
AU_2 HG	No capping
AU_2A	8.5
AU_2B	No capping
AU_3	55.0
AU_3A	35.0
AU_4	40.0
AU_5	No capping

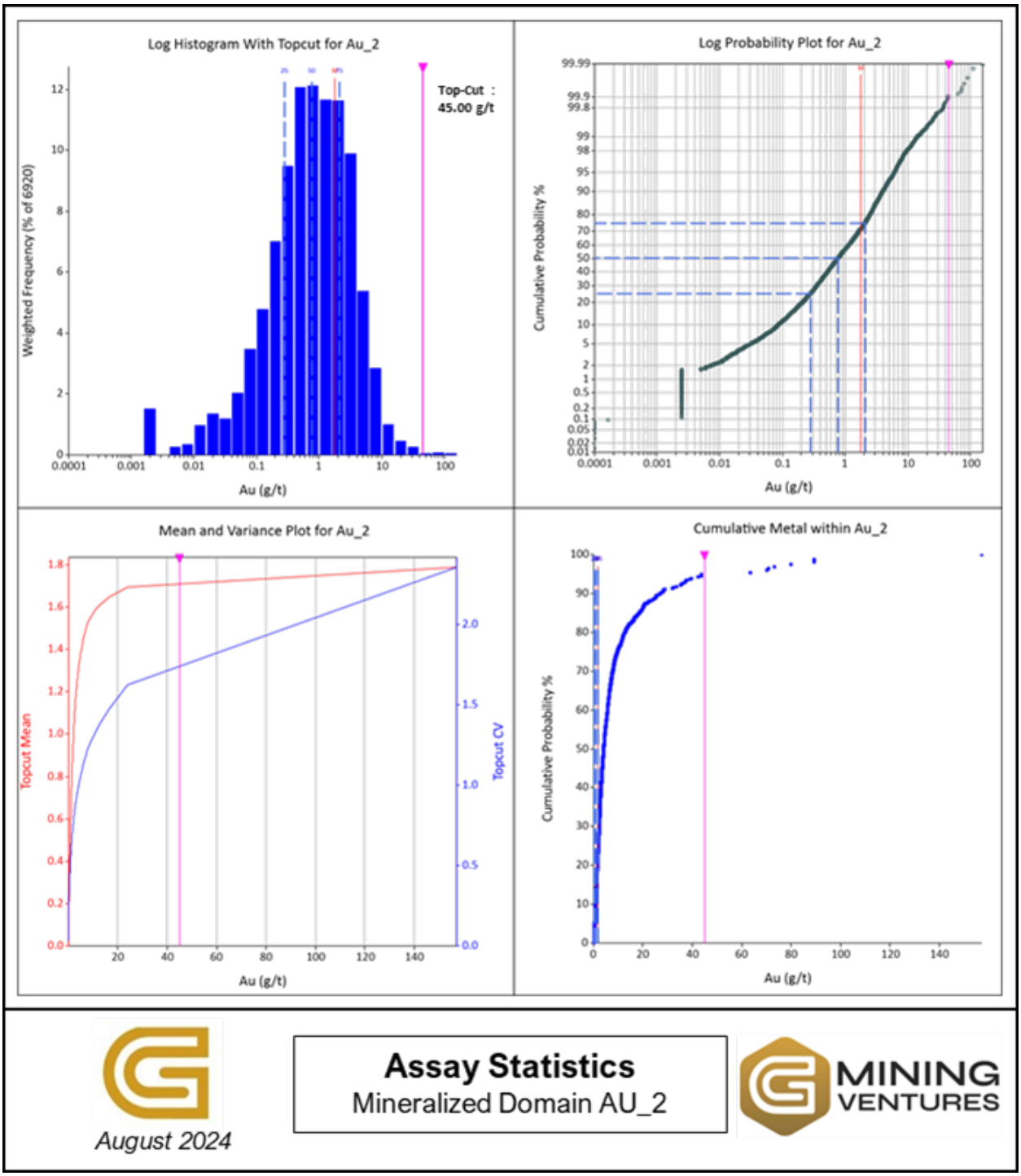
Figure 14.6 to Figure 14.8 present histograms, log probability plots, mean and variance plots, as well as cumulative metal plots for gold within the Oko West LDZ, AU_2 and AU_2HG mineralized domains respectively. Table 14.5 compares statistics for uncapped and capped assays of the Oko West deposit, per domain. Domain AU_3 is highly influenced by only a few anomalous very high-grade samples.

Figure 14.6: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the LDZ Mineralized Domain



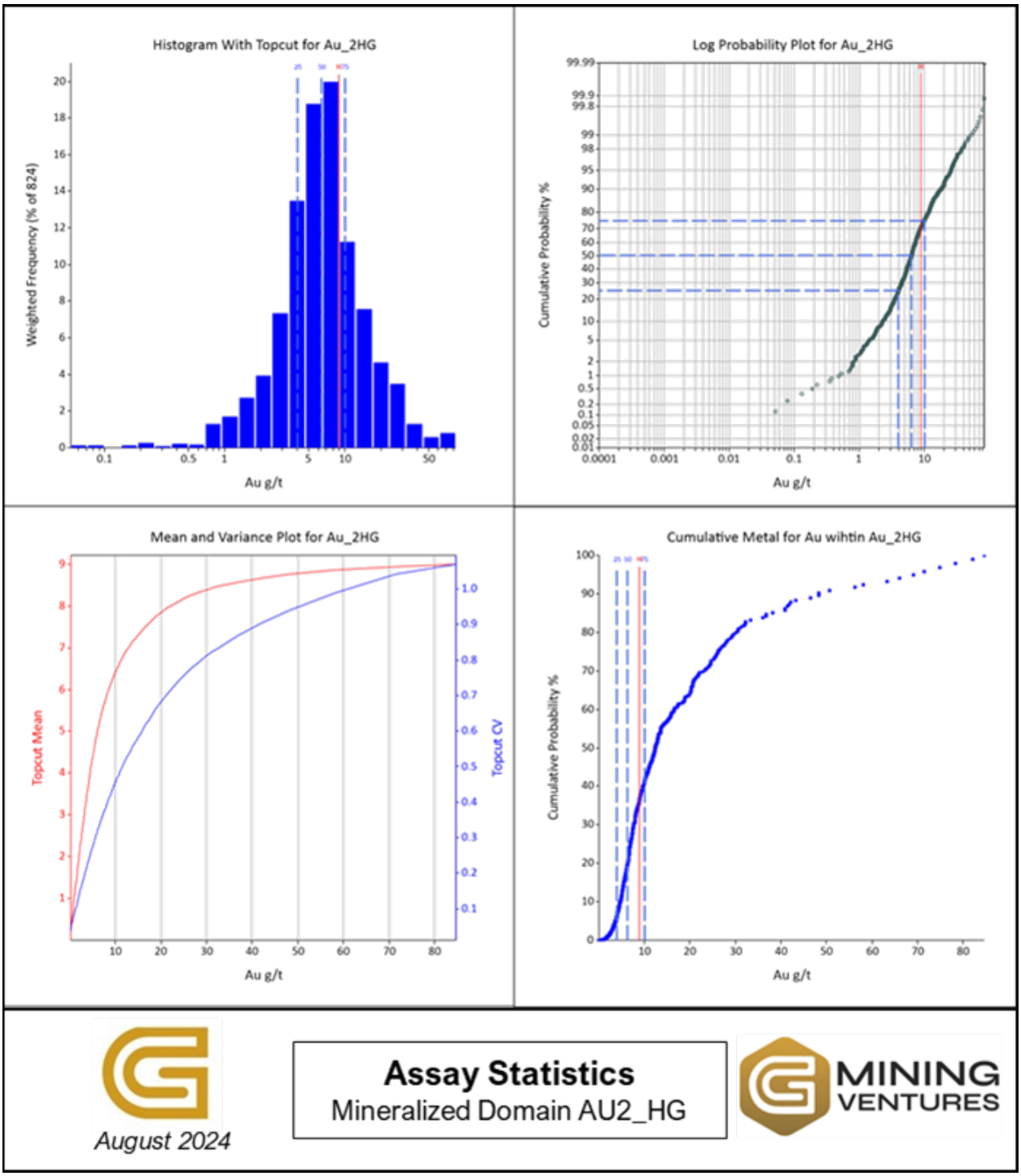
Source: GMS, 2024

Figure 14.7: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2 Mineralized Domain



Source: GMS, 2024

Figure 14.8: Histograms, Log Probability Plots, Mean and Variance Plots, and Cumulative Metal Plots for Gold Within the AU_2HG Mineralized Domain



Source: GMS, 2024

Table 14.5: Statistics of Uncapped and Capped Assays of Oko West, per Domain (length weighted)

Domain	Num. of Assays	Au Uncapped (g/t Au)			Num. of Assays Capped	Au Capped (g/t Au)			Metal Cut (%)
		Max	Mean	CV		Max	Mean	CV	
LDZ	3,060	61.89	1.73	2.12	4	50.00	1.71	2.05	0.6
AU_2	6,920	157.29	1.80	2.31	7	45.00	1.76	1.83	2.4
AU_2 HG	824	84.67	8.99	1.07	0	84.67	N/A	N/A	N/A
AU_2A	44	42.57	1.18	4.45	1	8.50	0.66	1.66	44.4
AU_2B	94	4.91	0.69	1.22	0	4.91	N/A	N/A	N/A
AU_3	2,609	1,106.05	2.13	8.52	8	55.00	1.72	2.37	19.2
AU_3A	2,618	65.84	1.38	2.27	4	35.00	1.36	2.06	1.5
AU_4	950	263.27	1.57	3.86	4	40.00	1.40	2.41	11.0
AU_5	179	4.07	0.49	1.15	0	4.07	N/A	N/A	N/A

14.5.3 Compositing

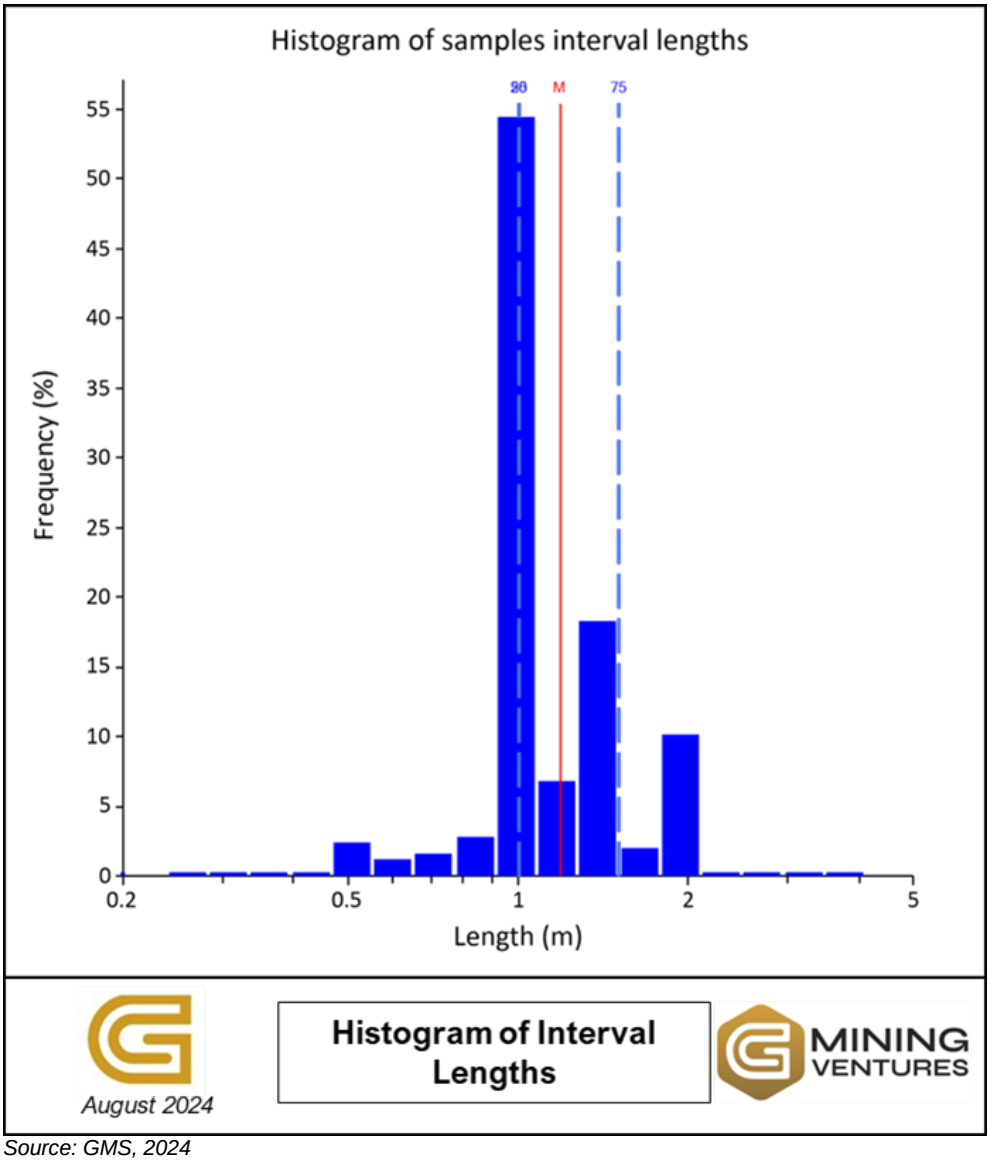
Following the application of assay capping, the samples were composited downhole within the boundaries of each mineralized domain. The length of the composites was determined through statistical analysis of the sample lengths, taking into consideration factors, such as the most sampled interval length (i.e. mode), block sizes, and modelled mineralized domain sizes.

Composites of 1 m were retained, with residuals of less than 0.3 m distributed equally through the interval length. A sample coverage of each composite interval of at least 50% was needed for composites to be created. Table 14.6 presents the statistics for uncomposited and composited samples, per mineralized domain. Figure 14.9 presents the histogram of interval length on the Oko West Project, while Figure 14.10 compares values before and after compositing.

Table 14.6: Uncomposited and Composited Statistic by Mineralized Domain

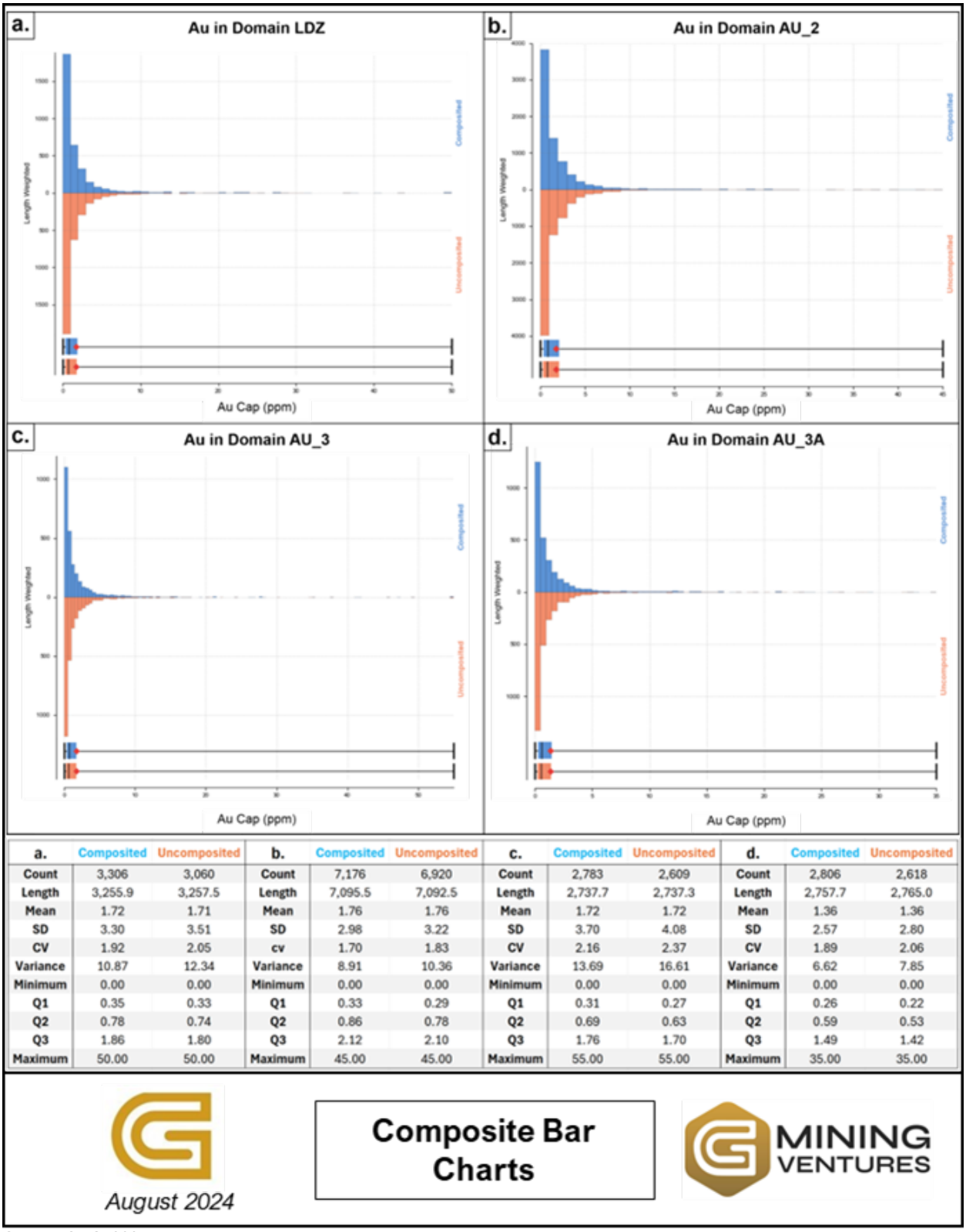
Zone	Uncomposited (length-weighted)				1 m Composites			
	Number Samples	Length	Average	CV	Number Samples	Length	Average	CV
LDZ	3,060	3,257.5	1.71	2.05	3,306	3,255.9	1.72	1.92
AU_2	6,920	7,092.5	1.76	1.83	7,176	7,095.5	1.76	1.70
AU_2 HG	824	708.2	8.99	1.07	729	708.2	8.99	0.95
AU_2A	44	45.5	0.66	1.66	47	45.5	0.66	1.38
AU_2B	94	103.0	0.69	1.22	104	103.0	0.69	1.18
AU_3	2,609	2,737.3	1.72	2.37	2,783	2,737.7	1.72	2.16
AU_3A	2,618	2,765.0	1.36	2.06	2,806	2,757.7	1.36	1.89
AU_4	950	962.5	1.40	2.41	985	963.0	1.40	2.08
AU_5	179	218.0	0.49	1.15	225	218.5	0.49	1.07

Figure 14.9: Oko West Database Histogram of Sampled Interval Lengths



Source: GMS, 2024

Figure 14.10: Composited and Uncomposited Assays Comparative Bar Charts



a.	Composited	Uncomposited	b.	Composited	Uncomposited	c.	Composited	Uncomposited	d.	Composited	Uncomposited
Count	3,306	3,060	Count	7,176	6,920	Count	2,783	2,609	Count	2,806	2,618
Length	3,255.9	3,257.5	Length	7,095.5	7,092.5	Length	2,737.7	2,737.3	Length	2,757.7	2,765.0
Mean	1.72	1.71	Mean	1.76	1.76	Mean	1.72	1.72	Mean	1.36	1.36
SD	3.30	3.51	SD	2.98	3.22	SD	3.70	4.08	SD	2.57	2.80
CV	1.92	2.05	cv	1.70	1.83	CV	2.16	2.37	CV	1.89	2.06
Variance	10.87	12.34	Variance	8.91	10.36	Variance	13.69	16.61	Variance	6.62	7.85
Minimum	0.00	0.00	Minimum	0.00	0.00	Minimum	0.00	0.00	Minimum	0.00	0.00
Q1	0.35	0.33	Q1	0.33	0.29	Q1	0.31	0.27	Q1	0.26	0.22
Q2	0.78	0.74	Q2	0.86	0.78	Q2	0.69	0.63	Q2	0.59	0.53
Q3	1.86	1.80	Q3	2.12	2.10	Q3	1.76	1.70	Q3	1.49	1.42
Maximum	50.00	50.00	Maximum	45.00	45.00	Maximum	55.00	55.00	Maximum	35.00	35.00

Source: GMS, 2024
**Note: a. LDZ, b. AU_2, c. AU_3, d. AU_3A.*

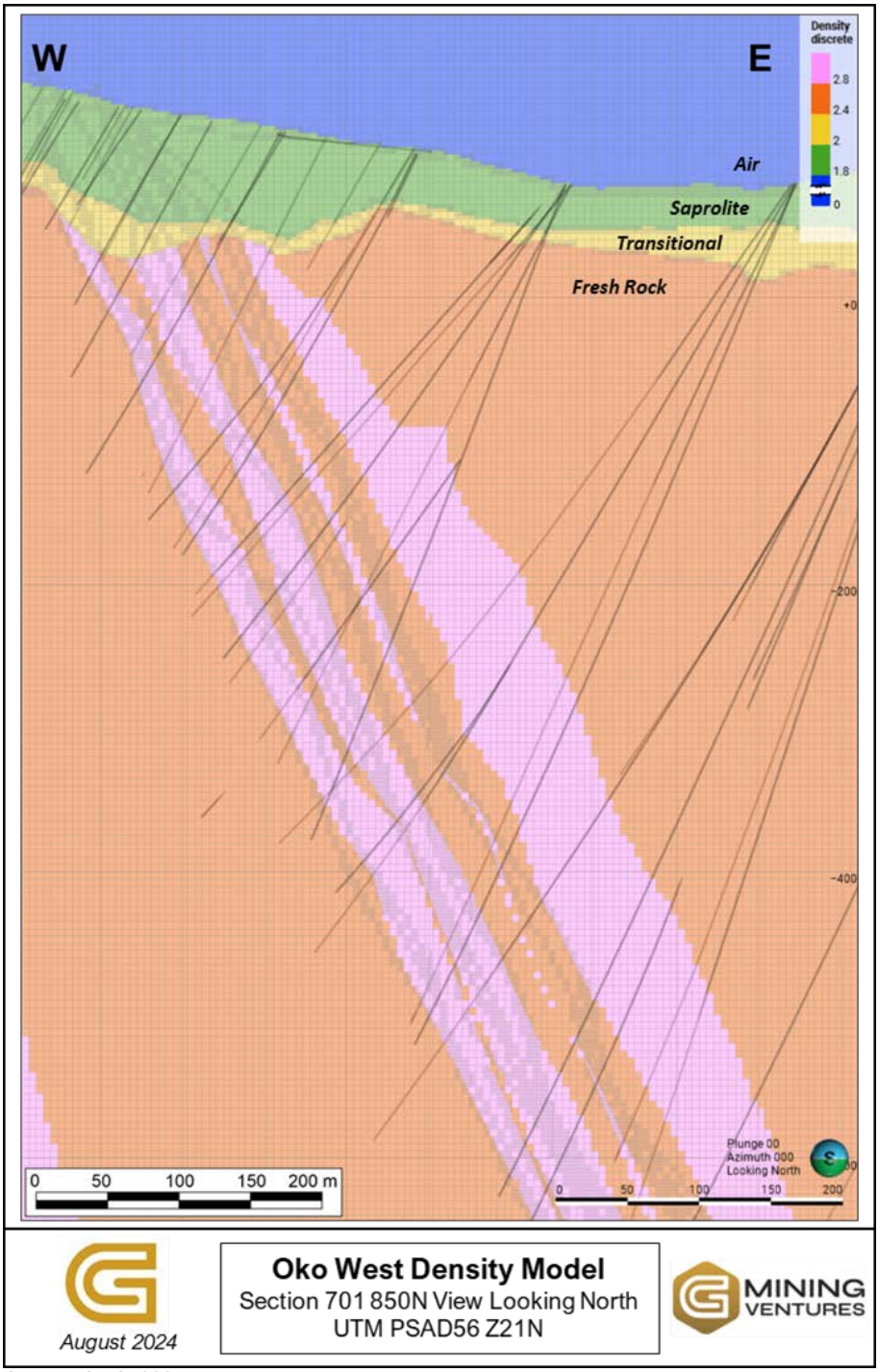
14.6 Density Measurements

Density measurements were collected systematically in DDH every five (5) to ten (10) metres or at every geological domain transition (i.e., weathering and/or lithology shift) (Procedures, 2022). The weight in water, weight in air method was used by the personnel to measure the density of chosen core samples. These data were used to assign median specific gravity to each modelled geological and weathering domain. Table 14.8 presents the number of measurements and median density used in the resulting 16 density domains. Figure 14.11 presents a cross-section of the Oko West density model.

Table 14.7: Host Rocks Density Statistics by Weathering Profile

Weathering Domain	Geological Domain	Count	Mean	Minimum	Median	Maximum
Alluvium / Colluvium		105	1.91	1.49	1.85	2.94
Saprolite	Granite FW	42	1.98	1.55	1.97	2.60
	Granite HW	1,771	1.88	1.06	1.88	2.84
	Metasediment	450	1.93	0.64	1.91	3.06
	Mafic Volcanic	0	NA	NA	NA	NA
	Volcaniclastic	409	1.93	0.68	1.91	2.76
Trans	Granite FW	24	2.33	1.85	2.35	2.73
	Granite HW	621	2.25	1.26	2.23	3.41
	Metasediment	98	2.27	1.83	2.23	2.96
	Mafic Volcanic	3	2.65	2.47	2.60	2.87
	Volcaniclastic	206	2.21	1.05	2.16	5.25
Fresh Rock	Granite FW	803	2.71	1.33	2.75	3.88
	Granite HW	3,637	2.71	1.07	2.73	5.38
	Metasediment	1,172	2.74	1.33	2.77	3.47
	Mafic Volcanic	259	2.82	1.43	2.87	3.12
	Volcaniclastic	2,223	2.78	1.01	2.82	6.05

Figure 14.11: Density Model Coloured by Density Value

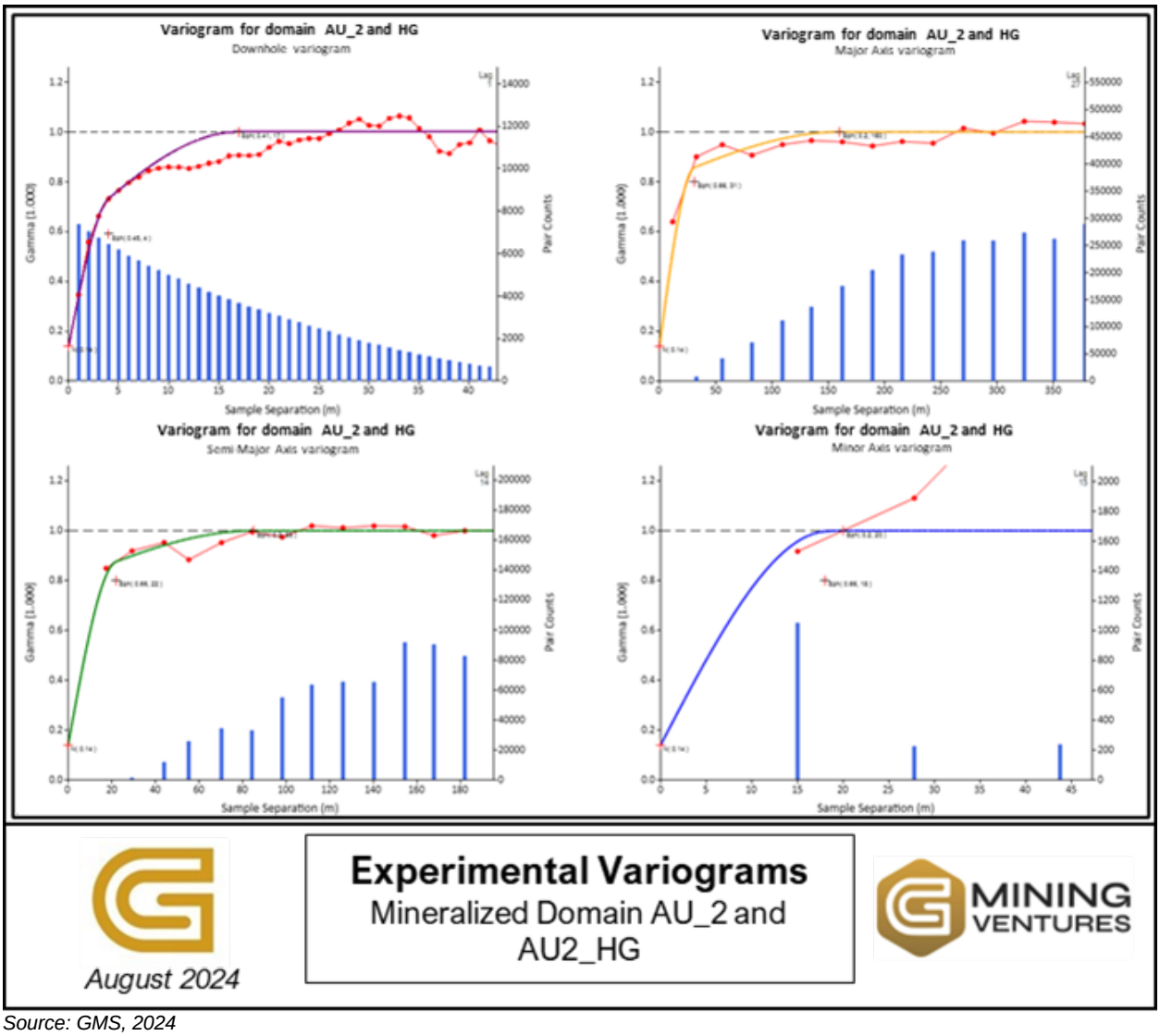


Source: GMS, 2024
**Note: Weathering materials can be distinguished between saprolitic material (green), transitional material (yellow) and fresh rocks (orange and pink).*

14.7 Variography

Variography is a statistical tool used in resource estimation to evaluate the spatial distribution of grades within a mineralized domain. Experimental variograms were produced for each mineralized domain, based on the 1-m composites presented above. Some similar domains were merged to obtain more coverage and facilitate variogram interpretation (e.g. domain AU_2 and AU_2HG). Variograms for AU_2A, AU_2B and AU_5 could not be adequately interpreted due to the insufficient number of composites. Figure 14.12 presents variogram parameters, while Table 14.8 presents an example of variography for the Oko West AU_2 mineralized domain (combined with the high-grade domain, fault block 1 only).

Figure 14.12: Oko West Experimental Variograms for Mineralized Domain AU_2 and AU_2_HG Combined



Source: GMS, 2024

Table 14.8: Variogram Parameters Used for the Mineral Resource Estimation per Domain

Domain	Direction			Nugget	Structure 1				Structure 2			
	Dip	Dip Azimuth	Pitch		Sill 1	Major	Semi-Major	Minor	Sill 2	Major	Semi-Major	Minor
LDZ	66	94	70	0.18	0.64	41	34	8	0.18	150	129	12
AU_2	65	96	73	0.22	0.67	31	22	18	0.11	160	85	20
AU_3	65	96	73	0.22	0.67	31	22	18	0.11	160	85	20
AU_3A	66	95	70	0.27	0.55	30	26	8	0.18	95	40	10
AU_4	67	92	70	0.25	0.64	40	42	10	0.11	100	50	12

**Note: All parameters are normalized with a total Sill of 1.0.*

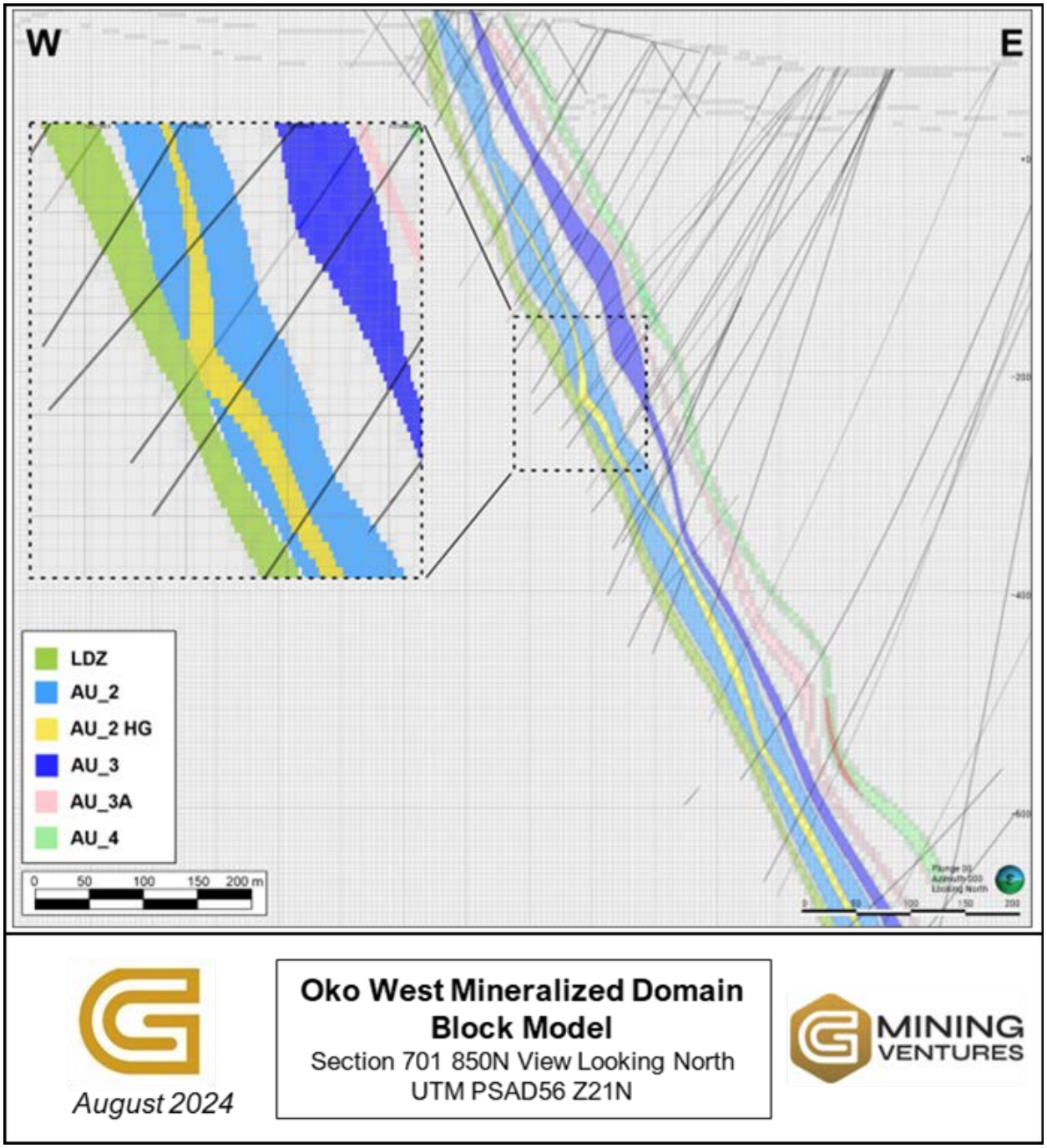
14.8 Block Modelling

The modelling of each block was carried out by adopting a parent block size of 5.0 m x 5.0 m x 5.0 m and a sub-block count of 2 x 10 x 2 for a minimum block size of 2.5 m x 0.5 m x 2.5 m. The block size was chosen based on the width of the mineralized zones, the nominal drill spacing and the anticipated open pit and underground mining methods. To validate the accuracy of the block size against the mineralization wireframe volumes, GMS compared the volumes. The blocks for each domain are good representations of their respective 3D model volumes. Table 14.9 presents block model parameters, while Figure 14.13 presents a cross-section of the block model mineralization domain trigger.

Table 14.9: Oko West Block Model Parameters

Description	Easting (m)	Northing (m)	Elevation (m)
Origin Coordinates	2,734,000.0	700,000.0	250.0
Parent / Sub-block Count	5/2	5/10	5/2
Minimum Block Size	2.5	0.5	2.5
Number of Blocks	588	308	264
Rotation	275°		

Figure 14.13: Mineralized Domain Block Model



Source: GMS, 2024

14.9 Block Model Interpolation

The Ordinary Kriging (OK) interpolation method was used to interpolate block grades based on the variogram models presented in Section 14.7. Inverse Squared Distance (ID^2) interpolation method was used in areas with limited data and where robust variogram models were not achievable (see Section 14.7).

For domains interpolated using OK, a discretization of 5 x 5 x 5 in X, Y and Z was applied to the blocks. For OK and ID^2, a four-pass approach applied by domains was used, with an increasing ellipsoid size after each pass estimation. A fifth pass was added for a minority of blocks that were not evaluated within the first four passes (search distance up to 160 m and minimum of 4 composites). Blocks falling outside mineralized domains, but within the mineralized volcanoclastic units were interpolated using ID^2 and a four-pass approach, similar to the estimation used for this MRE. This dilution model was used by the engineering team for Whittle and stope optimizations. However, for mineral resource statement, all blocks falling outside the mineralized domains were assigned a value of 0.00 g/t Au.

For ellipsoid orientation, Leapfrog Edge's "dynamic anisotropy" (DA) was used based on the geometry of each domain. The variable ellipsoid orientation (or DA) was validated for each domain and no inconsistencies were observed. Table 14.8 to Table 14.11 present the parameters, restrictions and search criteria used for the Oko West resource estimation.

Table 14.10: Search Ellipsoids by Mineralized Domains and Search Passes

Domain	Interpolation Method	Ellipsoid Ranges (m)											
		Pass 1			Pass 2			Pass 3			Pass 4		
		Max	Int	Min	Max	Int	Min	Max	Int	Min	Max	Int	Min
LDZ	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_2	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_2 HG	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_2A	ID2	60	50	15	80	65	20	120	100	30	160	120	30
AU_2B	ID2	60	50	15	80	65	20	120	100	30	160	120	30
AU_3	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_3A	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_4	OK	60	50	15	80	65	20	120	100	30	160	120	30
AU_5	ID2	60	50	15	80	65	20	120	100	30	160	120	30

Table 14.11: Sample Search Criteria by Passes

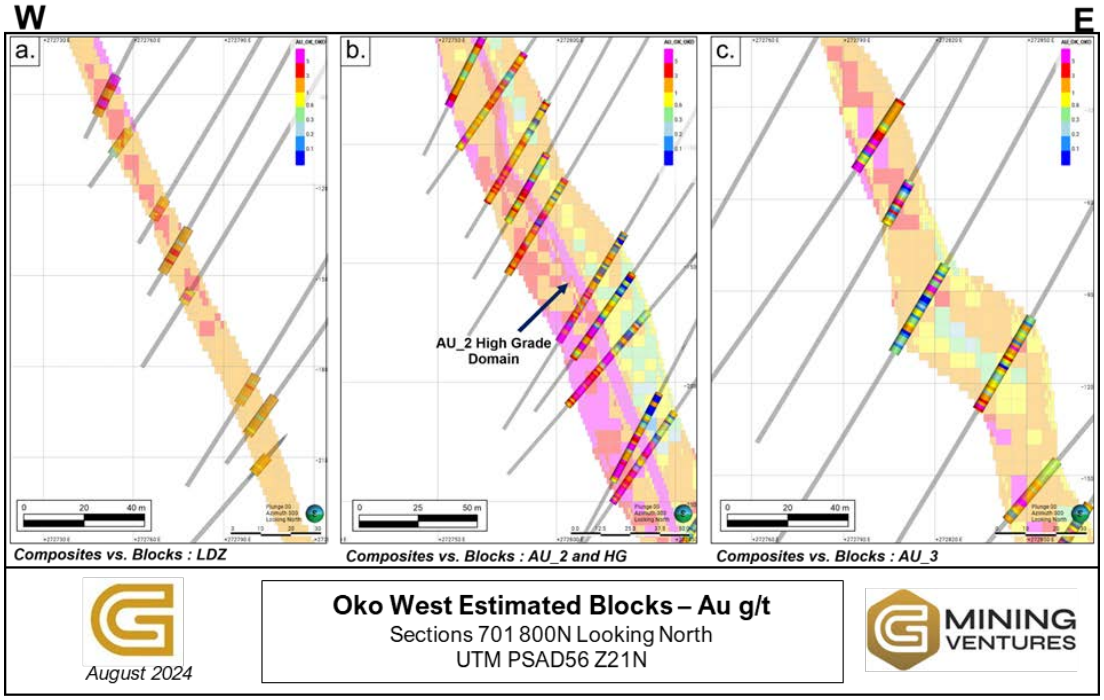
Pass	Composites			Minimum DDH
	Min.	Max	Max/DDH	
Pass 1	7	12	3	3
Pass 2	7	12	3	3
Pass 3	7	12	3	3
Pass 4	4	12	3	2

14.10 Grade Estimation Validation

14.10.1 Visual Validation

A visual validation was conducted to confirm that the ellipsoid orientation matches the orientation of the modelled veins and the distribution of grades. To ensure that the estimated blocks are a robust interpretation of the composites, various validation methods were used. Visual checks of the block model, per vertical section and plan view, were used as validation of the interpolation outputs. Figure 14.14 presents global cross-sections of the interpolated block models against composites. In general, the estimated block gold grades are good representations of composites gold grades.

Figure 14.14: Oko West Block Model and Composites Visual Validation



Source: GMS, 2024
**Note: a. LDZ, b. AU_2, c. AU_3. The High-Grade domain is constrained within AU_2.*

14.10.2 Global Statistical Validation

To ensure proper composite representation in each domain, a statistical comparison was made between the global composite mean and global interpolated block means for various interpolation methods. Table 14.12 shows a summary of global means, per domain. Based on the results, the interpolation using OK is judged to be valid and a good representation of composite grades, and no important bias is observed between the interpolation methods.

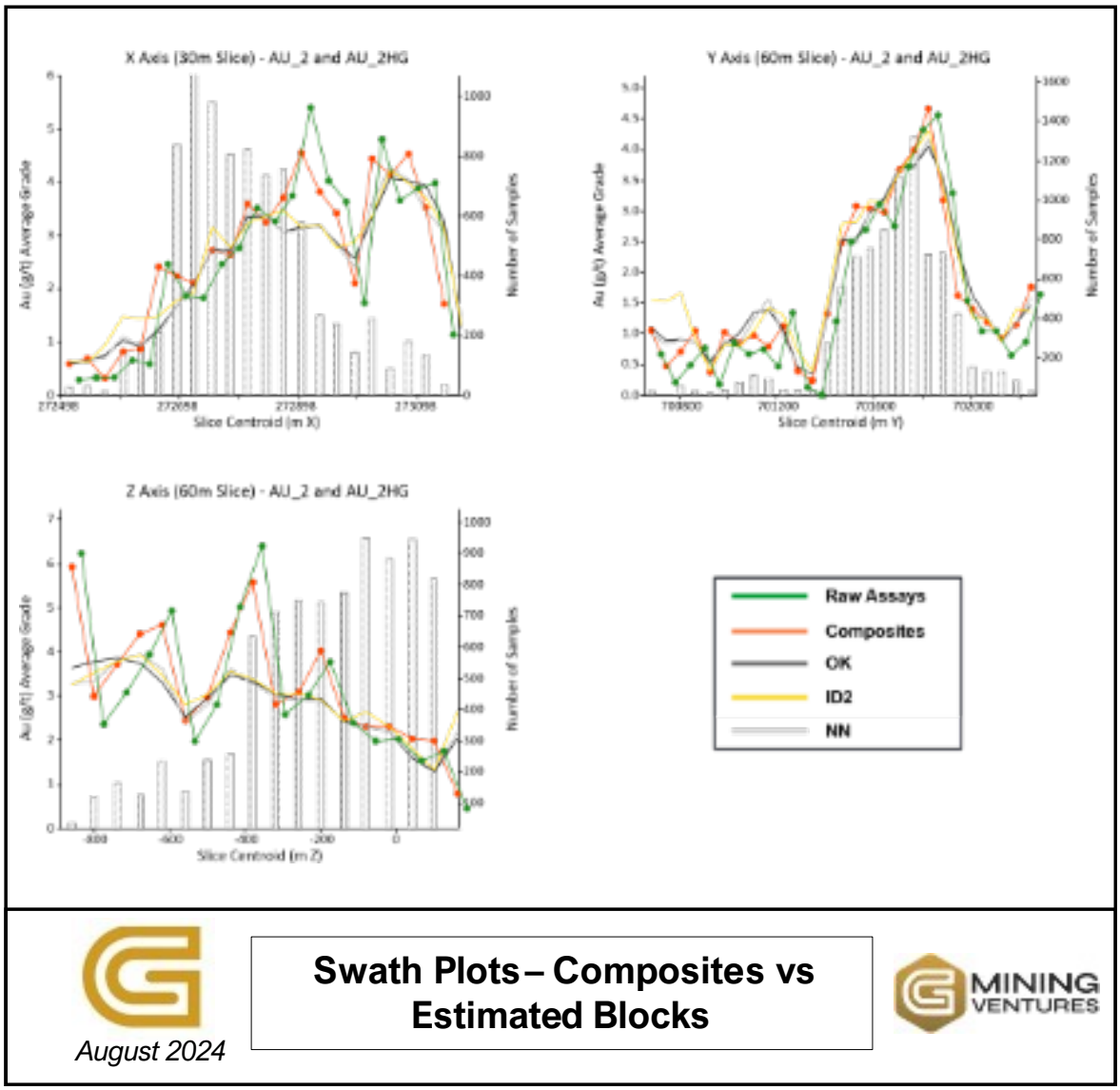
Table 14.12: Mean Grade Comparison Between Composites and Blocks, per Domains (volume weighted)

Oko Domains	Comp (g/t Au)	Block OK (g/t Au)	Block ID^2 (g/t Au)	Block NN (g/t Au)
LDZ	1.72	**1.55**	1.54	1.60
AU_2	1.76	**1.66**	1.65	1.67
AU_2 HG	8.99	**8.58**	8.62	8.97
AU_2A	0.66	0.66	**0.63**	0.68
AU_2B	0.69	0.62	**0.61**	0.71
AU_3	1.72	**1.69**	1.72	1.65
AU_3A	1.36	**1.25**	1.24	1.23
AU_4	1.40	**1.39**	1.33	1.39
AU_5	0.49	0.44	**0.45**	0.55

14.10.3 Local Statistical Validation – Swath Plot

Finally, swath plots were created to validate local estimation. The method involves comparing the predicted values of a block from the interpolation model to the actual values obtained from drillhole samples (i.e., composites). When enough samples were available for swaths plot analysis, peaks and troughs in composite grades generally follow peaks and troughs in block grades. Figure 14.15 presents swath plots along the X, Y and Z-axis. In general, composite gold grades are well represented within estimated block gold grades and the smoothing inherent to OK appears reasonable.

Figure 14.15: Swath Plots for X (along strike), Y (along cross-strike) and Z for Domain AU_2 and AU_2HG Combined



Source: GMS, 2024

14.11 Mineral Resources

14.11.1 Mineral Resources Classification

The estimated blocks were classified according to CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and adhere to the CIM "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (2019). The mineral resources at Oko were classified as Indicated and Inferred mineral resources.

As stated in CIM's "Definition Standards for Mineral Resources and Mineral Reserves":

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, density, shape and physical characteristics are estimated 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."

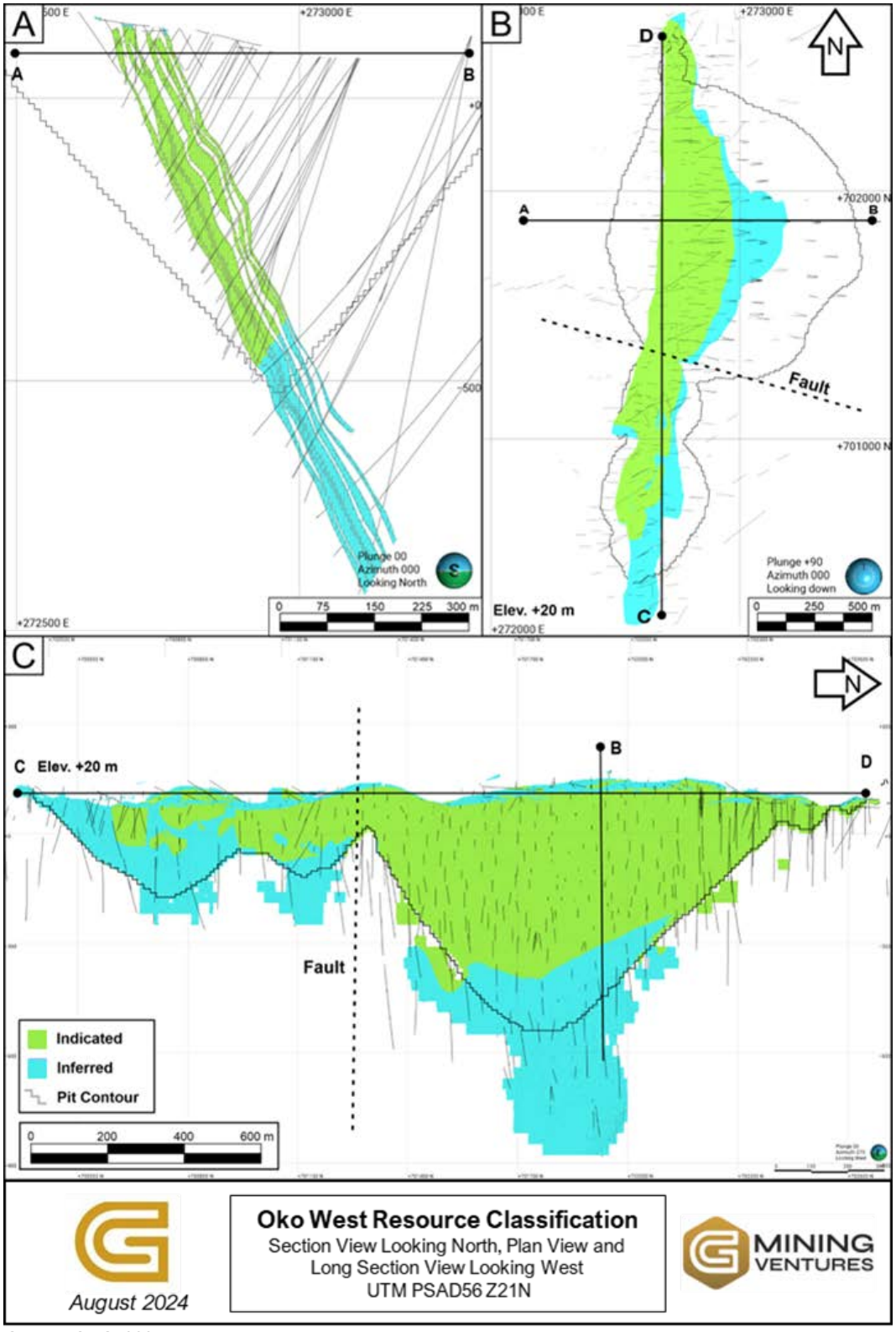
"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity."

GMS considered variogram ranges, drillhole spacing, confidence in the geological interpretation and recovery methods to determine parameters that will define the resource categories. The final mineral resource classification is mostly based on average drillhole spacing and manual editing to avoid isolated blocks. The principal assumptions to classify the Mineral Resources as Indicated and Inferred are summarized below:

- No Measured Mineral Resources are defined at Oko West at this stage of the Project;
- Indicated Mineral Resources are defined where blocks have an average distance to the nearest three (3) drillholes of less than 45 m;
- Inferred Mineral Resources are defined where blocks have an average distance to the nearest three (3) drillholes of less than 80 m. This limit corresponds to sectors with sparse drilling and less lateral and horizontal continuity; and
- Final categories of all domains were manually edited to avoid isolated clusters of blocks.

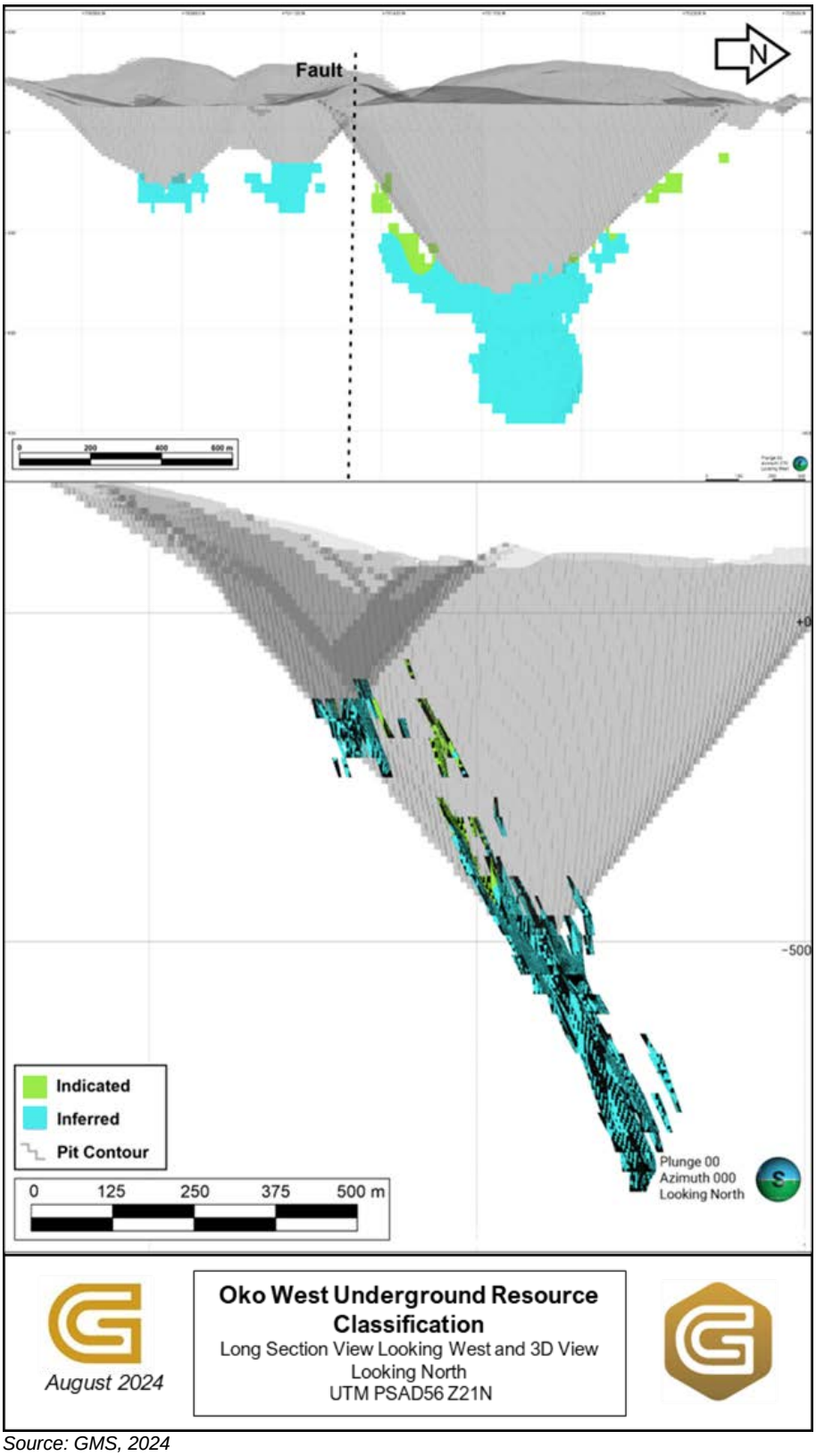
The classification of blocks in Blocks 5 and 6 (domain south of the interpreted fault zone) appears more discontinuous between parallel, stacked domains. This is mainly caused by some drilling, not piercing, through all zones. The final classification of mineral resources is displayed in Figure 14.16 for the in-pit resource and Figure 14.17 for the underground resource constrained within stopes optimized using Deswik Stope Optimizer (DSO).

Figure 14.16: Mineral Resource Classification with Pit Outline Optimized Using Whittle



Source: GMS, 2024

Figure 14.17: Underground Mineral Resource Classification Constrained Within Stopes Optimized from DSO



Source: GMS, 2024

14.11.2 Reasonable Prospects of Eventual Economic Extraction (RPEEE)

The Oko West deposit is constrained by a Whittle pit shell for the open-pit part, and stopes modelled using Deswik Stope Optimizer (DSO) for the underground part of the deposit. Whittle pit shell and underground stopes were modelled by GMS mine engineering personnel. To define the resource pit and underground stopes, different parameters were selected according to the surface alteration intensity (i.e., weathering) of the host rocks. The parameters for pit and stopes optimization and cut-off grade assumptions are presented in Table 14.13 and Table 14.14. The optimization of the mineral resource pit is presented in Figure 14.16, while the optimization of the underground mineral resource stope is presented in Figure 14.17.

Table 14.13: Parameters Used for Open Pit Whittle Optimization and Open-pit Cut-off Grade Assumptions

Optimization Parameters		Resources Parameters		
		Fresh Rock	Trans	Saprolite
Electricity Cost	*USD/kWh*	0.136	0.136	0.136
Discount Rate	*%*	5%	5%	5%
Gold Price	*USD/oz*	1,950.00	1,950.00	1,950.00
Payable Metal	*%*	99.95%	99.95%	99.95%
Transport & Refining Cost	*USD/oz*	8.00	8.00	8.00
Royalty Rate	*%*	5.39%	5.39%	5.39%
Royalty Cost	*USD/oz*	102.40	102.40	102.40
Net Ore Value	*USD/oz*	1,788.60	1,788.60	1,788.60
Nominal Milling Rate	*t/d*	16,438	16,438	16,438
Plant Throughput	*kt/y*	6,000	6,000	6,000
Recovery	*%*	92.5	95.0	96.0
Total Ore Based Cost	*USD/t milled*	14.51	11.87	9.23

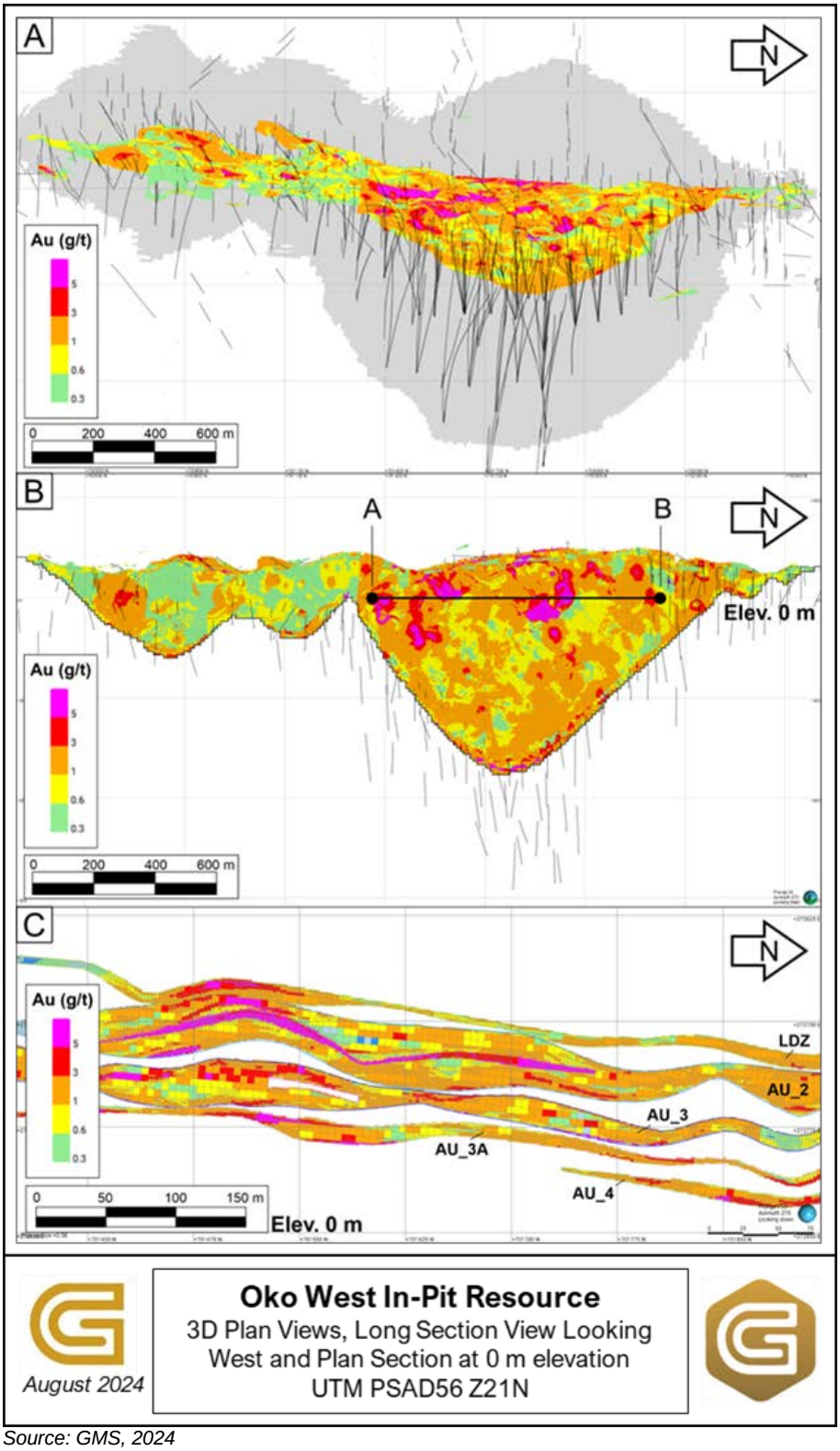
Optimization Parameters		Resources Parameters (Open-Pit)		
		Fresh Rock	Trans	Saprolite
Mining Dilution	%	-	-	-
Mining Loss	%	-	-	-
Total Mining Reference Cost	*USD/t mined*	2.69	2.56	2.05
Incremental Bench Cost	*USD/10 m bench*	0.04	0.04	0.04
IRA SLOPE Hanging Wall (East)	*degree*	50	40	30
IRA SLOPE Footwall (West)	*degree*	55	40	30
Gram Value	*USD/g*	57.50	57.50	57.50
Total Ore Based Cost	*USD/t*	19.80	17.16	14.51
Used COG (Inc. Proc. Rec.)	*g/t*	0.37	0.31	0.30

Table 14.14: Parameters Used for Stope Optimization and Underground Cut-off Grade Assumptions

Optimization Parameters		Resources Parameters (Underground)		
		Fresh Rock	Trans	Saprolite
Mining Dilution	*%*	-	-	-
Mining Recovery	*%*	100%	-	-
Total Mining Reference Cost	*USD/t mined*	48.32	-	-
Stope Height	*m*	30	-	-
Strike Length	*m*	20	-	-
Maximum Width	*m*	25	-	-
Minimum Mining Width	*m*	2	-	-
HW Dilution	*m*	-	-	-
FW Dilution	*m*	-	-	-
Minimum Pillar Width	*m*	5	-	-
Minimum Dip	*degree*	50	-	-
Stope Optimizer Sub-Shapes	*Elev. & Dir.*	Yes	-	-

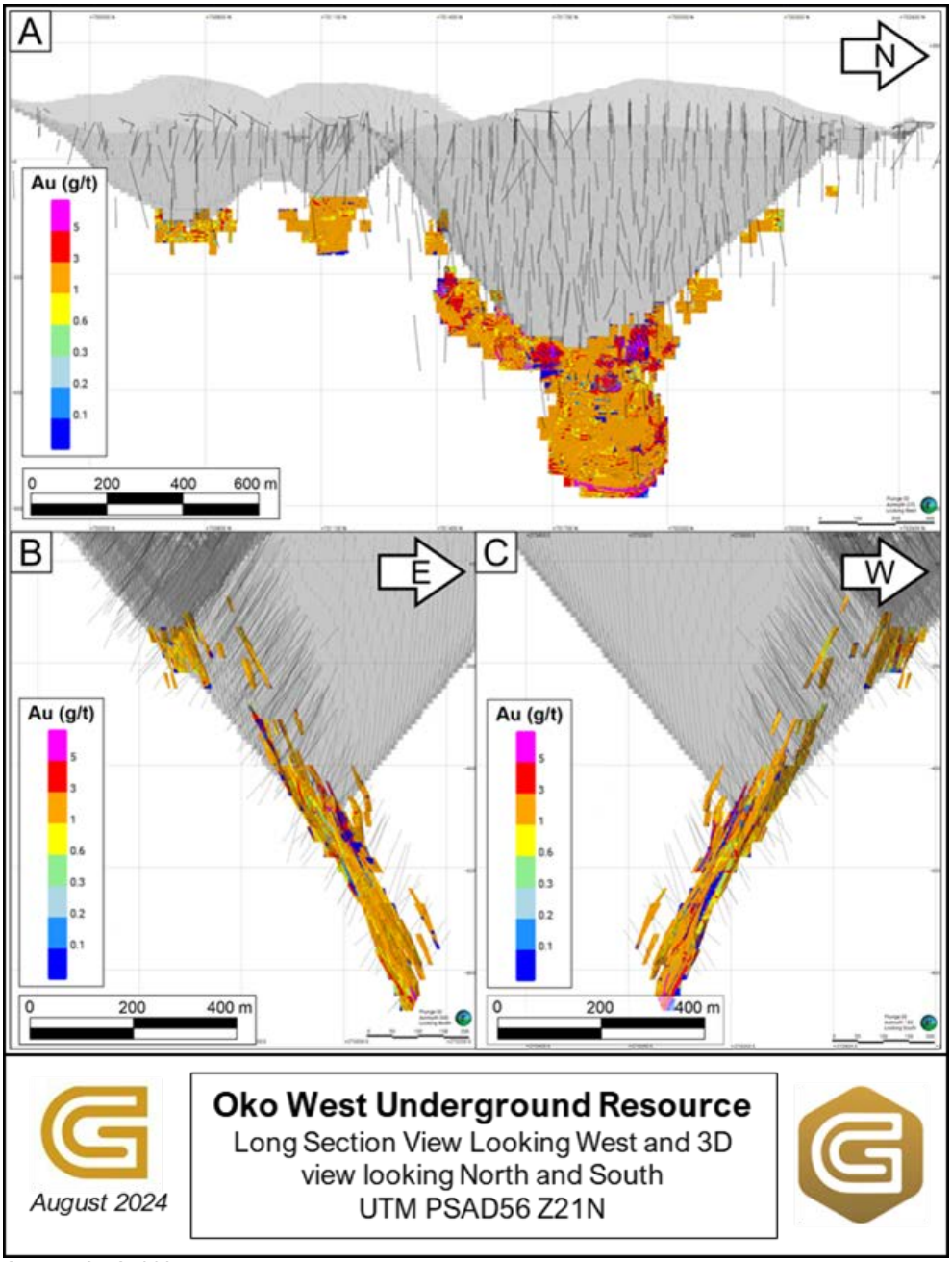
Optimization Parameters		Fresh Rock	Trans	Saprolite
Gram Value	*USD/g*	57.50	-	-
Total Ore Based Cost	*USD/t*	73.26	-	-
Used COG (Inc. Proc. Rec.)	*g/t*	1.38	-	-

Figure 14.18: Open-Pit Optimization with Block Model Coloured by Gold Grades (g/t)



Source: GMS, 2024

Figure 14.19: Underground Stope Optimization with Block Model Coloured by Gold Grades (g/t)



Source: GMS, 2024

14.12 Mineral Resource Statement

The Oko West deposit open-pit Mineral Resource is stated using a lower cut-off of 0.30 g Au/t in alluvium / colluvium and saprolite, 0.31 g Au/t in transitional material, and 0.37 g Au/t in fresh rocks. The resources are constrained within the resource pit. Results for the open-pit part of the deposit are presented by weathering profile in Table 14.15. Open pit Indicated Mineral Resources are estimated at 61,115 kt,

grading 2.06 g Au/t for 4,237 koz Au. Open pit Inferred Mineral Resources are estimated at 8,107 kt, grading 1.87 g Au/t for 488 koz Au.

Table 14.15: Oko West Deposit In-pit Mineral Resource Estimate – Effective Date February 7, 2024.

Resource Classification	Weathering Profile	Tonnage (kt)	Grade (g Au/t)	Gold Content (koz Au)
Indicated	Alluvium / Colluvium	0	-	0
	Saprolite	5,714	1.86	342
	Transition	2,859	1.85	170
	Fresh Rock	55,542	2.09	3,726
	Total	**64,115**	**2.06**	**4,237**
Inferred	Alluvium / Colluvium	627	1.52	31
	Saprolite	214	0.75	5
	Transition	47	0.83	1
	Fresh Rock	7,219	1.94	451
	Total	**8,107**	**1.87**	**488**

**Notes:*

1. *The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).*
2. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
3. *The effective date of the Mineral Resource Estimate is February 7, 2024.*
4. *The lower cut-offs used to report open pit Mineral Resources is 0.30 g Au/t in saprolite and alluvium / colluvium, 0.31 g Au/t in transition, and 0.37 g Au/t in fresh rock.*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g Au/t was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 0.5 m x 2.5 m, and has been reported inside an optimized pit shell. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Inverse Distance for domains AU_2A, AU_2B and AU_5, and Ordinary Kriging for all other domains. Capping was applied on eight domains, ranging frm 5 g Au/t to 80 g/t.*
9. *Open pit optimization parameters and cut-off grades assumptions are as follows:*
   a. *Gold price of US$1,950/oz.*
   b. *Total ore-based costs of US$14.51/t for saprolite and alluvium/colluvium, with a 96.0% processing recovery US$17.16/t for transition with a 95.0% processing recovery and US$19.80/t for fresh rock based on 92.5% processing recovery.*
   c. *Inter-ramp angles of 30° in saprolite and alluvium/colluvium, 40° in transition and 50° in fresh rock.*
   d. *Royalty rate of 8%.*
10. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces.*
11. *The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*
12. *These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.*
13. *These mineral resources assume no mining dilution and losses.*

Stope-constrained underground mineral resources are presented in Table 14.16. The underground Resources are estimated from zones outside the constrained Resources of the open pit. The underground stope optimization was designed using a cut-off grade of 1.38 g Au/t – all material within the stopes is reported. Underground Indicated Mineral Resources are estimated at 491 kt, grading 1.85 g/t for 29 koz Au. Underground Inferred Mineral Resources are estimated at 11,510 kt, grading 3.01 g/t for 1,116 koz Au. The bulk of the ounces are located below the deepest limits of the open pit MRE, which represents the extension of the high-grade zone in Block 4 at depth.

Table 14.16: Oko West Deposit Underground Mineral Resource Estimate – Effective Date February 7, 2024.

Category	Tonnage (kt)	Au grade (g/t)	Contained Gold (koz)
Indicated	491	1.85	29
Inferred	11,510	3.01	1,116

**Notes:*

1. *The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019).*
2. *The qualified person (QP) for this Mineral Resource Estimate (MRE) is Christian Beaulieu, P.Geo., Consulting Geologist for G Mining Services Inc.*
3. *The effective date of the Mineral Resource Estimate is February 7, 2024.*
4. *Underground Mineral Resources are reported inside potentially mineable volume (i.e., must take material) and include below cut-off material (stope optimization cut-off grade: 1.38 g Au/t).*
    a. *A change in the reporting method for the underground part of the deposit explains the differences in tonnage and average grade between this PEA and the MRE published in February 2024. Tonnage of potentially mineable material stated below cut-off (i.e., must-take material) is declared for this constrained underground Mineral Resource Estimate. Blocks have been reclassified inside each stope based on deposit knowledge and continuity and reflect the existing classification. No change in total ounces is observed.*
5. *The Oko West Deposit has been classified as Indicated and Inferred Mineral Resources according to drill spacing. No Measured Mineral Resource has been estimated.*
6. *The density has been applied based on measurements taken on drill core and assigned in the block model by weathering type and lithology.*
7. *A minimum thickness of 3 metres and minimum grade of 0.30 g Au/t was used to guide the interpretation of the mineralized zones.*
8. *This MRE is based on a subblock model with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 0.5 m x 2.5 m, and has been reported inside optimized stope shapes. Gold grades in fresh rock, transition and saprolite were interpolated with 1 m composites using Inverse Distance for domains AU_2A, AU_2B and AU_5, and Ordinary Kriging for all other domains. Capping was applied on eight domains, ranging from 5 g Au/t to 85 g/t.*
9. *UG optimization parameters and cut-off grades assumptions are as follows:*
    a. *Gold price of US$1,950/oz.*
    b. *Total ore-based costs of US$73.26/t for fresh rock.*
    c. *Stope height of 30 m, strike length of 20 m, maximum width of 25 m and minimum width of 2 m.*
    d. *The Deswik.SO (DSO) was used to constrain the resources.*
    e. *Royalty rate of 8% payable to the Government of Guyana.*
10. *Tonnage has been expressed in the metric system, and gold metal content has been expressed in troy ounces.*
11. *The tonnages have been rounded to the nearest 1,000 tons, and the metal content has been rounded to the nearest 1,000 ounces. Totals may not add up due to rounding errors.*
12. *These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.*
13. *These mineral resources assume no mining dilution and losses, however must-take material is accounted for in underground stopes.*

Mr. Christian Beaulieu, P.Geo., is not aware of any factors or issues that materially affect the mineral resource estimate other than normal risks faced by mining projects in the province in terms of

environmental, permitting, taxation, socio-economic, marketing, and political factors, and additional risk factors regarding indicated and inferred resources.

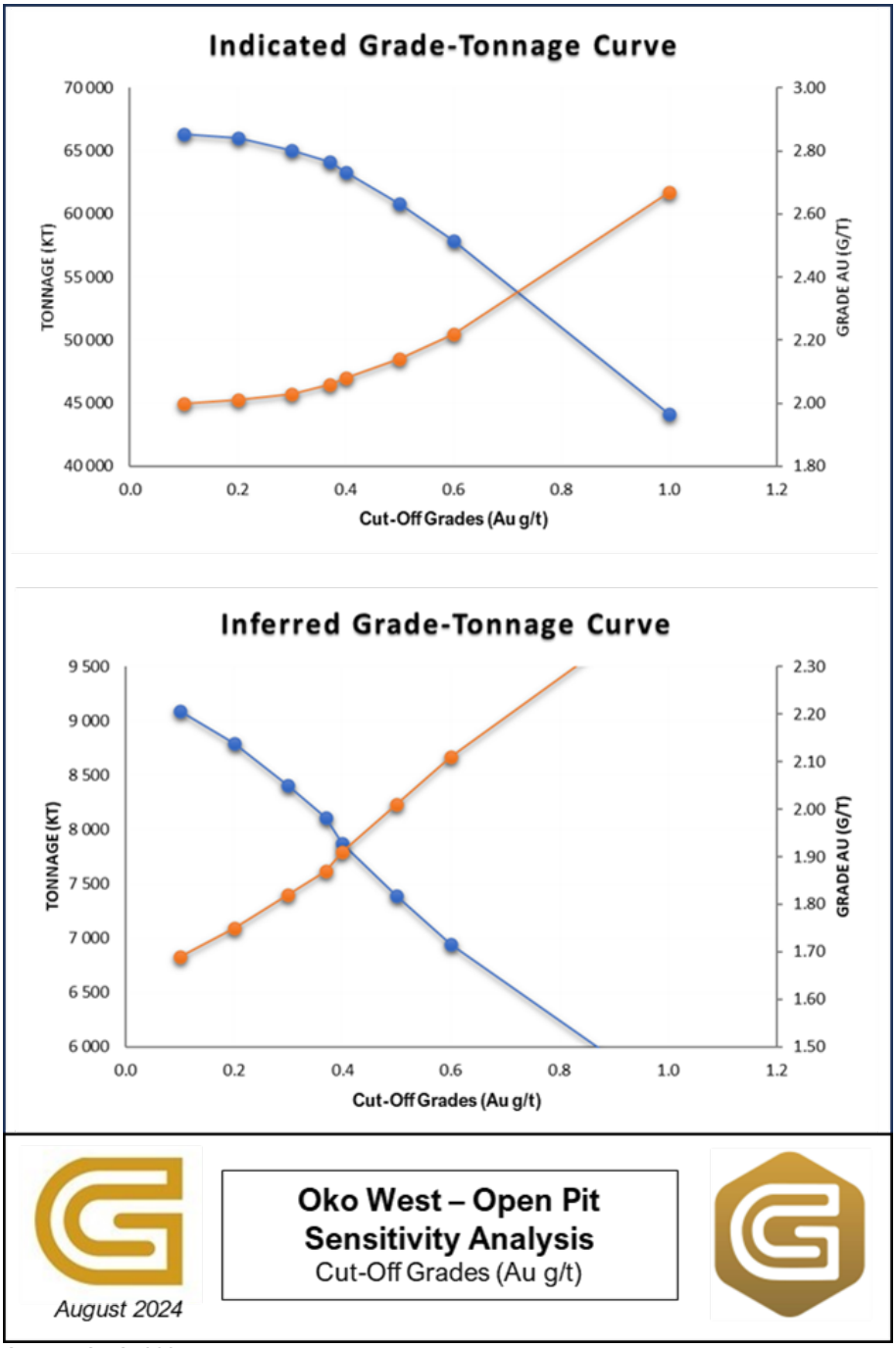
These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this Report are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

14.12.1 Cut-Off Grade Sensitivities

The sensitivity of the open pit and underground resources to different cut-off grades scenarios are summarized in Table 14.17 and Table 14.18. Figure 14.20 presents the grade-tonnage curves for varying gold cut-offs of the indicated and inferred open-pit mineral resource, Figure 14.21 presents the grade-tonnage curves for varying gold cut-offs of the indicated and inferred stopes-constrained underground mineral resource. The tonnages and grade at differing cut-offs shown below are for comparison purposes only and do not constitute an official Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

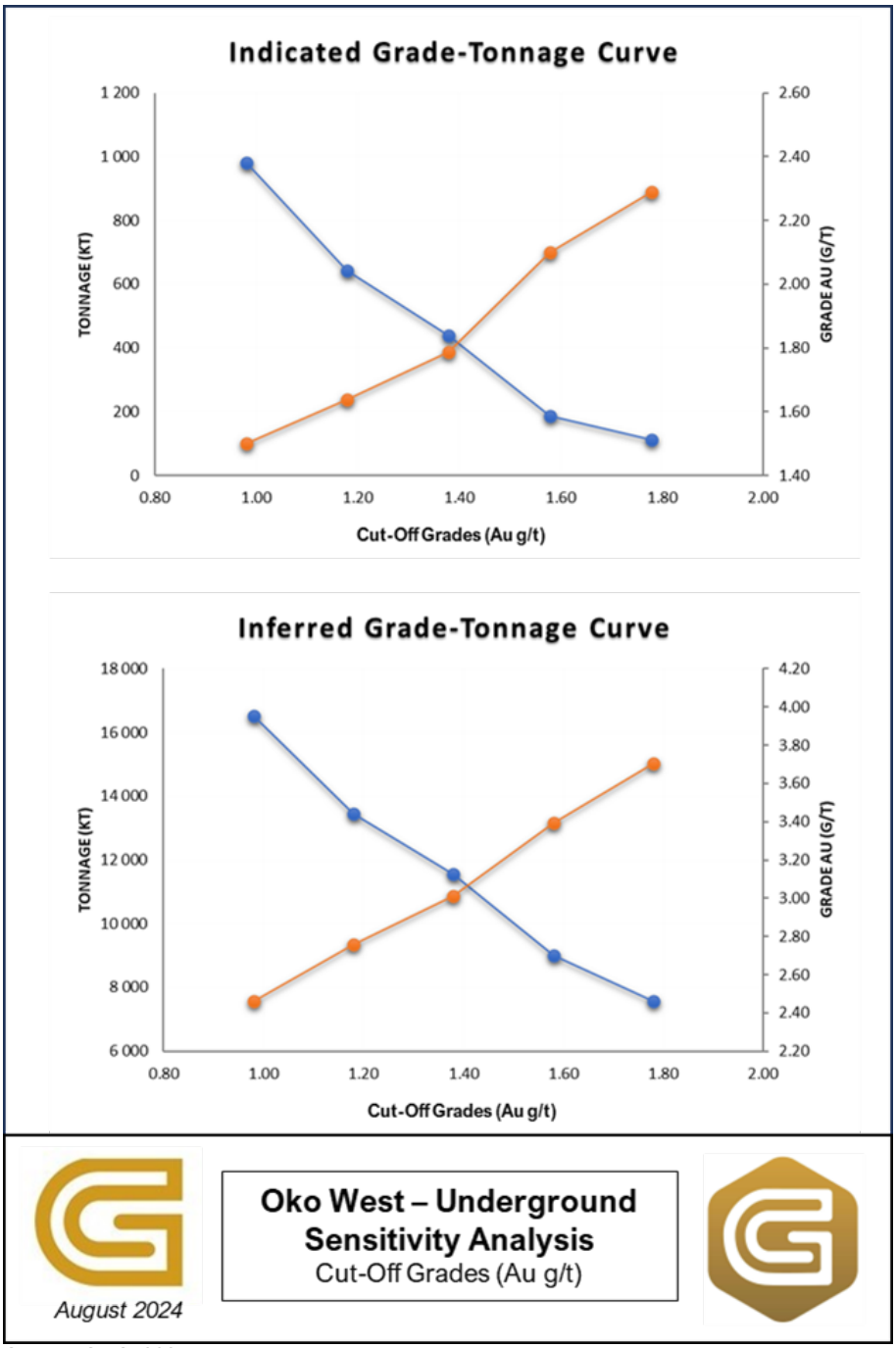
As seen from the following table and graphs, the Oko West open-pit deposit shows a low sensitivity to cut-off grades. Gold content in the Indicated category remains stable with increasing cut-offs below 0.60 g Au/t. The Inferred category constrained within the pit does not reflect the sensitivity of the deposit and only accounts for less than 10% of the total gold content. As expected, the underground part of the deposit is more sensitive to increasing cut-off. However, the gold content remains relatively stable with increasing cut-offs below 1.40 g Au/t.

Figure 14.20: Indicated and Inferred Grade-Tonnage Curves for In-pit Resource



Source: GMS, 2024

Figure 14.21: Indicated and Inferred Grade-Tonnage Curves Within Underground Stopes Modelled at Different Cut-off Grades Using DSO



Source: GMS, 2024

Table 14.17: Oko West In-Pit Cut-off Grade Sensitivity

Cut-off Grade (g/t)	Indicated			Inferred		
	Tonnage (kt)	Grade (g/t)	Gold Content (koz)	Tonnage (kt)	Grade (g/t)	Gold Content (koz)
0.10	66,311	2.00	4,257	9,092	1.69	495
0.20	65,990	2.01	4,255	8,794	1.75	494
0.30	65,022	2.03	4,247	8,410	1.82	491
COG*	**64,115**	**2.06**	**4,237**	**8,107**	**1.87**	**488**
0.40	63,291	2.08	4,227	7,875	1.91	485
0.50	60,832	2.14	4,191	7,392	2.01	478
0.60	57,872	2.22	4,139	6,941	2.11	470
1.00	44,108	2.67	3,784	5,540	2.44	434

**Note 1: COG tonnage and grades are calculated at the MRE cut-off grade (0.30 g Au/t in colluvium / alluvium and saprolite, 0.31 g Au/t in transition and 0.37 g Au/t in fresh rock).*
**Note 2: The tonnages and grade at differing cut-offs shown above are for comparison purposes only and do not constitute an official Mineral Resource Estimate.*

Table 14.18: Oko West Underground Cut-off Grade Sensitivity

Cut-off Grade (g/t)	Indicated			Inferred		
	Tonnage (kt)	Grade (g/t)	Gold Content (koz)	Tonnage (kt)	Grade (g/t)	Gold Content (koz)
0.98	1,093	1.54	54	16,372	2.47	1,300
1.18	705	1.69	38	13,371	2.76	1,188
COG*	**491**	**1.85**	**29**	**11,510**	**3.01**	**1,116**
1.58	225	2.18	16	8,967	3.40	980
1.78	143	2.38	11	7,528	3.71	898

**Note 1: COG tonnage and grades are calculated at the MRE cut-off grade (1.38 g/t for underground material in fresh rock).*
**Note 2: The tonnages and grade at differing cut-offs shown above are for comparison purposes only and do not constitute an official Mineral Resource Estimate.*
**Note 3: The tonnages and grade at differing cut-offs shown above are constrained within optimized stope.*

15 MINERAL RESERVE ESTIMATES

This Preliminary Economic Assessment (PEA) of the Oko West gold deposit is based on indicated and inferred resources. Because of the inclusion of inferred resources, it is not applicable to determine reserves at this stage of the project. Ore zones will be classified as mineralized materials only.

16 MINING METHODS

16.1 Summary

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The initial milling rate is set at 6 Mtpa for processing hard rock, increasing to 7 Mtpa when incorporating saprolite, following a 5-month ramp-up during the open pit phase. The milling process is designed to operate for 13 years, with stockpiles peaking at 4.4 Mt by Year 2 to maintain consistent mill feed. A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and to be used in an economic analysis except as allowed in PEA studies. There is no guarantee that Inferred Mineral Resources can be converted to Indicated or Measured Mineral Resources and, as such, there is no guarantee the Project economics described herein will be achieved.

The OP will utilize a fleet of diesel-powered equipment, including drills, haul trucks, and hydraulic shovels. The Project consists of a main pit that is deeper and centered on Block 4, with two (2) smaller sub-pits positioned on the southern extension to the main one. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LoM) of 13 years. A total of 60.7 Mt of mineralized material will be mined at an average diluted gold grade of 1.72 g Au/t. A total of 364.6 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.0 tonnes of waste per tonne of mineralized material. The primary production equipment includes 22 m^3 diesel-hydraulic shovels paired with 136-t off-highway mining trucks for the mineralized material and waste. The mining operation is planned to be fully owner-operated, with pre-production mining scheduled over approximately 24 months to secure construction material and to remove overburden to allow access to the mineralized material. A total of 28.4 Mt of waste and overburden as well as 3.5 Mt of mineralized material will be mined in the pre-production and ramp-up period.

The UG operation consists of one (1) mine separated in three (3) zones: the main zone and two (2) satellites zones, all accessible from a surface mine portal through the main decline ramp. The selected mining method is long hole open stoping (LHOS), including transverse stoping and longitudinal stoping variations.

The LoM for the UG mine is expected to be 13 years including construction, development, pre-production and the full production period. Over this LoM, the UG mine is expected to be in production for 11 years,

including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed and sustain full production. Initially, a contract mining period is anticipated for the construction and development of the mine followed by a transition to full owner-operated mining activities.

The UG mine is expected to achieve an average production rate of 4,250 tpd of mineralized material, with 4,000 tpd derived from stope production and 250 tpd from lateral development. Development of the UG mine includes approximately 47.0 km of lateral and 3.2 km of vertical development to be excavated. A total of 14.5 Mt of mineralized material is expected to be mined at an average diluted gold grade of 3.19 g Au/t. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

16.2 Geotechnical Considerations

The preliminary geotechnical work for the Oko West Project was conducted by two (2) consulting firms:

- NewFields, which handled the OP portion, and
- Alius Mining, responsible for the UG portion.

16.2.1 Rock Mass Characterization

NewFields (2022) conducted a rock mass characterization as part of the PEA geotechnical guidelines for the OP. The Oko West team provided data for pit slope analysis through a series of core hole logs containing geology and geotechnical information such as lithology, Rock Quality Designation (RQD), rock strength index per ISRM (1981), structure type (e.g., fault, foliation, veins), and structure orientation (dip and dip direction). From these, a selection of core holes was made to represent the rock mass conditions and structure of the hanging wall and footwall. The selection process involved reviewing available logs and data distribution to ensure that the chosen core holes met specific criteria: providing geotechnical data that covers most of the proposed open pit, piercing a significant proportion of the pit wall rock, and representing the rock mass in both the hanging wall and footwall. Based on these criteria, 12 core holes were selected for the slope analysis.

The RQD and ISRM Strength Index indicates that the hanging wall rock mass is of high quality and very strong, whereas the footwall rock mass is of lower quality with variable strength, though the majority is classified as Strong to Very Strong Rock.

16.2.2 Open Pit

Given the overall good rock quality and strength, the performance of the pit slope is expected to be dominated by structural controls rather than rock mass strength.

The structural features logged in the core holes by the project team included:

- Bedding
- Foliation
- Veins
- Faults
- Shears and fractures

The structural features for each core hole were individually plotted on stereonets to evaluate the dominant structures in the hanging wall and footwall. Subsequently, the combined data from hanging wall and footwall were plotted on separate stereonets to define the primary structural controls for the pit wall rock

No information regarding the characteristics of the saprolite was provided for the Oko West Project. As a result, the experience of NewFields experts with saprolite pit slopes in Venezuela and Suriname was applied. Generally, well-drained saprolite soil slopes can be reliably developed using an IRA of 30 degrees. However, it is important to implement aggressive water management measures to prevent slope instabilities and excessive erosion. Such measures could include diversion channels graded to ensure proper drainage and the use of bench-sumps to help control water on the saprolite slopes.

16.2.3 Slope Stability

Newfields delivered preliminary slope stability analysis conducted on generic cross-sections for the hanging wall and footwall of the Oko West pit and analysis of the discontinuities from drillhole cores. The analysis concluded that global stability would be achieved with the following key parameters:

- Inter-Ramp Angle (IRA) = 55 degrees on footwall and 50 degrees on hanging wall
- Maximum IRA slope height on hanging wall = 180 m
- Maximum IRA slope height on the footwall = 120 m
- Minimum geotechnical bench = 20 m

- Average unconfined intact rock strength of hanging wall rock = 100 MPa (estimated from ISRM Index)
- Average Geological Strength Index (GSI) of hanging wall rock = 80 (estimated from RQD data)
- Average Geological Strength Index (GSI) of footwall rock = 65 (estimated from RQD data)
- Fully drained, depressurized slope
- Maximum highwall height = 450 m

16.2.4 Underground

Alius Mine Consulting (Alius) provided PEA-level geomechanical recommendations and guidelines to support the underground operation of the Oko West Project. The rock mass characterization conducted by NewFields (2022), was subsequently reviewed by Alius.

The Geological Strength Index (GSI) as defined by Hoek (1994), was estimated from RQD data of 12 drillholes. The hanging wall and footwall GSI were estimated at 80 and 65, respectively. A review of the geological database was conducted to derive Q' rock mass classifications (Barton et al., 1974), accordingly.

$$Q = Q' \times \frac{J_w}{SRF} = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

Where:

- RQD: rock quality designation as defined by Deere et al. (1967).
- Jn: joint set number.
- Jr: joint roughness number.
- Ja: joint alteration number.
- Jw: joint water reduction factor.
- SRF: stress reduction factor.

The rock mass classifications were determined using lower and upper bound values RQD, along with average values of Jn and Jr/Ja. The lower and upper limits correspond to the 25th and 50th percentiles of the cumulative distribution function (CDF). Note that all drill holes used in the study were filtered for depths exceeding 150 m. The resulting rock mass classifications for the three (3) stope-relevant units are shown in Table 16.1.

Table 16.1: Q' Rock Mass Classifications

Lithology	Q'25	Sap
Footwall Granite (GRAFW)	14.1	16.3
MetaSediment (MSED1)	16.7	19.6
Mafic Volcanic (VMAF)	14.2	17.6

**Note that Q classifications are equal to those of Q' as the Jw/SRF ratio is assumed to be one (1).*

16.2.5 Stope Sizing

The stability analysis for each stope surface utilized the Potvin (1988) stability chart and the ELOS chart proposed by Clark (1998). Additionally, probabilistic assessments were performed employing the approach outlined by Mawdesley et al. (2001). These empirical methods incorporate the N' stability number along with the hydraulic radius (HR), which are defined as follows:

$$N' = Q' \times A \times B \times C$$

And:

$$HR = \frac{Stope\ surface\ area\ (m^2)}{Stope\ surface\ perimeter\ (m)}$$

Where:

- A: rock stress factor.
- B: joint orientation adjustment factor.
- C: gravity adjustment factor.

Furthermore, considering that no back support is planned for stopes narrower than 15 metres, two (2) distinct cases were examined: stopes ranging from 5 to 15 metres in width (mined longitudinally), and stopes ranging from 15 to 25 metres in width (mined transversely).

The following assumptions are made to assess the stability numbers and hydraulic radius:

- The hanging walls (HW), footwalls (FW), and backs / endwalls (BACKS/EW) are assumed to be in VMAF, GRAFW and MSED1 lithologies, respectively. Since backs (BACKS) and endwalls (EW) are in both VMAF and MSED1 lithologies; VMAF was chosen because of its lower rock mass characterization.

- The Q' rock mass classifications are those presented in Table 16.1.
- The rock stress factors (A) neglect induced stresses. The uniaxial compressive strengths were estimated from field intact rock strength values (R-index obtained from hammer tests on the core).
- Three (3) joint sets were assumed to assess B and C factors. These were obtained using lower stereographic projections of the structural data. The resulting joint sets are (dip/dip direction): 75/086°, 40/318° and 39/258°.
- Stope Vertical height 30 m.
- Stope strike length 20 m.
- Minimum FW–HW width 5 m.
- Minimum dip 50°.

From Alius analysis, the following conclusions can be drawn regarding stability and the probability of failure (PoF):

- Longitudinal stopes with mining widths between 5 and 15 m:
  - HW Stable without support PoF = 10–20%.
  - FW Transition: Stable–Stable with support PoF = 20–30%.
  - BACK Stable without support PoF = 0–10%.
  - EW Stable without support PoF = 0%.
- Transversal stopes with mining widths between 15 and 25 m:
  - HW Stable without support PoF = 10–20%.
  - FW Transition: Stable–Stable with support PoF = 20–30%.
  - BACK Stable without support PoF = 10–20%.
  - EW Stable without support PoF = 0%.

The surfaces of stopes typically fall within stable zones on stability charts, with probabilities of failure considered acceptable at below 20%. Currently, there are no plans to use cable bolting on hanging walls and footwalls; however, the existence of a lower deformation zone in the footwall may necessitate a reconsideration of this assumption. Additionally, some stopes have shallow dips (e.g., <60°), which may require adherence to specific support standards.

16.2.6 Dilution

Similarly to the stope sizing analyses, dilution was estimated using the Equivalent Linear Overbreak Slough (ELOS) method. From the current stope dimensions, the following conclusions can be made:

- Longitudinal stopes with mining widths between 5 and 15 m:
  - HW ELOS < 1.0 m
  - FW ELOS < 1.0 m
  - BACK ELOS < 0.5 m
  - EW ELOS < 0.5 m
- Transverse stopes with mining widths between 15 and 25 m:
  - HW ELOS < 1.0 m
  - FW ELOS < 1.0 m
  - BACK ELOS < 1.0 m
  - EW ELOS < 0.5 m

16.2.7 Ground Support

The ground support planned for the underground mine considers the following specifications:

- Back: 2.4 m long resin rebars on a 1.2 m x 1.2 m square pattern with 4” x 4” 6-gauge mesh screen.
- Walls: 1.8 m long friction bolts on a 1.2 m x 1.2 m square pattern with 4” x 4” 6-gauge mesh screen. Friction bolts of 18” are also installed as needed to prevent bagging in the mesh screen.
- Intersections: In addition to the standard ground support, 3.6 m long inflatable bolts will be installed on a 1.2 m x 1.2 m diced pattern to increase the ground support capacity. Some larger intersections will require cable bolting.
- Stopes: long support consisting of 8.0 m cables on a square pattern of 2.0 x 2.0 m.

No ground support has been selected at this stage of the Project for the portal development. These will be addressed in detail at the FS stage of the Project.

16.3 Hydrogeology

No hydrogeological studies have been completed to date to assess groundwater conditions at this stage of the Project. The hydrogeological site investigation program, which is currently being conducted by ERM and GMS is ongoing as part of the FS. Hydrogeological recommendations and guidelines at the FS level will be provided based on these findings. GMS assumes that groundwater infiltrations into the open pit will add an additional 15% to the volume of precipitation expected. The assumptions considered for groundwater infiltration into the underground mine are presented in Chapter 16.5.8.1.

16.4 Open Pit Mining

16.4.1 Pit Optimization

The Mineral Resource block model (Oko West 2.5_0.5_2.5) was imported to the Deswik CAD™ software as a single block model. The model provided was regularized and reblocked into a 5 m x 5 m x 5 m block model. The evaluation of the Potentially Extractable Portion of the Mineral Resource Estimate, referred as mineralized material mined, in the Oko West PEA includes all categories of Mineral Resources: Measured, Indicated, and Inferred.

A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred resources. Inferred Mineral Resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and to be used in an economic analysis except as allowed for in PEA studies. There is no guarantee that Inferred Mineral Resources can be converted to Indicated or Measured Mineral Resources, and as such, there is no guarantee the Project economics described herein will be achieved.

Open pit optimization was conducted in GEOVIA Whittle™ version 2022 to determine the optimal economic shape of the open pit and guide the pit design process. This task was performed utilizing Whittle's pseudoflow algorithm, which operates on a block model of the mineralized material. The algorithm progressively constructs lists of blocks that should or should not be mined, based on their economic value. The optimization process defines a pit outline that maximizes total economic value while adhering to the required pit slopes and other parameters.

The pit optimizations performed to generate optimal pit limits to guide ultimate pit design were based on valuing Measured, Indicated, and Inferred Mineral Resource category blocks.

16.4.2 Slope Recommendations

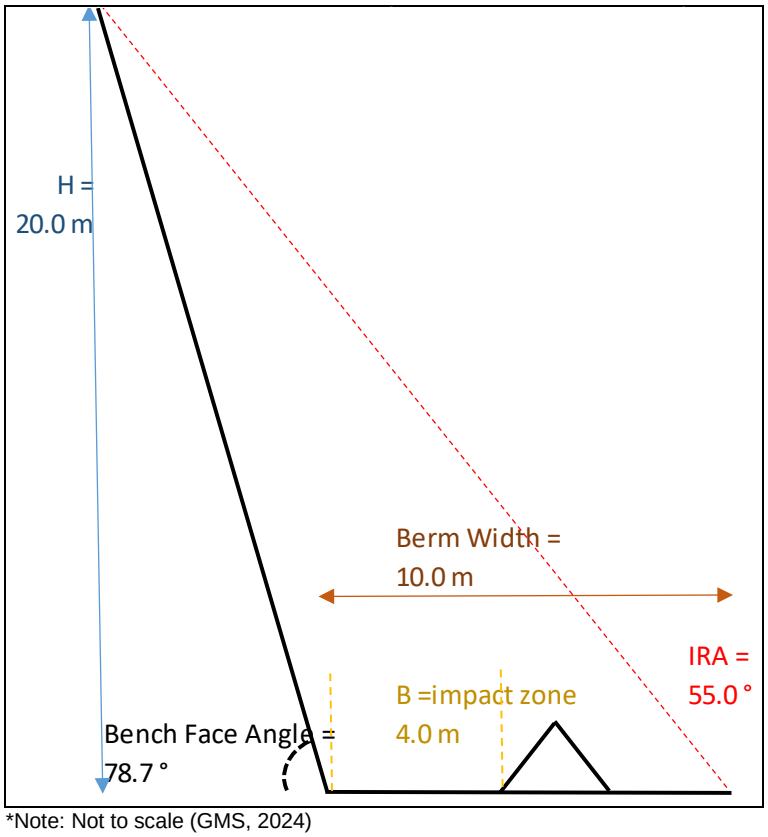
Based on the updated NewFields PEA-Level Pit Slope Design Report (June 2023), the recommended slope angles, as detailed below in Table 16.2 have been adopted. Figure 16.1 illustrates the typical slope configuration.

Table 16.2: Open-Pit Slopes Angles per Rock Type and Face Orientation

Final Berm Width for Pit Design				
Rock TYPE	**H = Bench Height (m)**	**Berm Width (m)**	**Bench Face Angle, degrees**	**Inter-Ramp Angle "IRA", degrees**
Fresh Rock West (Footwall)	20.0	10.0	78.7	55.0
Fresh Rock East (Hanging wall)	20.0	10.0	71.3	50.0
Transition	20.0	10.0	55.3	40.0
Saprolite and Overburden	20.0	10.0	39.0	30.0

**Note: Length in metres (m); Update to be determined by geotechnical study.*

Figure 16.1: Slope Configuration



*Note: Not to scale (GMS, 2024)

Depending on the face orientations (from geotechnical discontinuities), the IRA in fresh rock varies from 55 to 50. Due to the depth of the pit, geotechnical berms must be added and will be designed as described in Table 16.3.

Table 16.3: Geotechnical Berms

Foot Wall (West) Geotechnical Berm*	Hanging Wall (East) Geotechnical Berm*
20 m width every 120 vertical metre elevation	20 m width every 180 vertical metre elevation

**Note: Ramp can be considered as a geotechnical berm.*

Overall Slope Angle (OSA) results consider both IRA and geotechnical berms. Table 16.4 shows the OSA criteria which have been used for pit optimization.

Table 16.4: Overall Slope Angle for Open-Pit

		Footwall		Hanging wall	
Design Sector		**Fresh**	**Sap**	**Fresh**	**Sap**
Number of Benches (n) to Stack	#	25.0	10.0	25.0	10.0
Final Vertical Bench Height	m	20.0	20.0	20.0	20.0
Bench Face Angle	degree	78.7	39.0	71.3	39.0
Avg. Catch Berm Width	m	10.0	10.0	10.0	10.0
Inter-ramp Angle IRA (crest-to-crest)	degree	55.0	30.0	50.0	30.0
Ramp Width	m	33.7	33.7	33.7	33.7
Ramp Segments	#.	2.0	1.0	2.0	1.0
Geotechnical Bench Width	m	20.0	20.0	20.0	20.0
Number	#.	1.0	0.0	1.0	0.0
Interval	m	120.0	120.0	180.0	180.0
OSA (crest-to-crest)	degree	48.8	27.7	44.6	27.7

16.4.3 Pit Optimization Parameters and Cut-Off-Grade

A summary of the pit optimization parameters for a nominal processing rate of 6.0 Mtpa is presented in Table 16.5. The gold price is set at $1,750/oz, and the Project is subject to an 8% royalty.

Reference mining unit costs are based on a mining cost for a block usually located near the average pit surface elevation and incrementally increased as mining depth progresses, due to the increased haulage cycle time. The reference mining cost is estimated from a previous study at $2.69/t for fresh rock, $2.56/t for transition, and $2.05/t for saprolite. The incremental cost comes from the same analysis and is set at $0.04/t per 10 m bench.

The total mineralized material cost, which includes processing, general and administrative expenses, rehabilitation, and sustaining capital, varies from $19.80/t milled for fresh rock, $17.16/t milled for transition, to $14.51/t milled for saprolite.

The Overall Slope Angles, as presented in Table 16.4, range from 27.7 to 48.8 degrees and are incorporated into the optimization parameters.

Table 16.5: Economics Optimization Parameters

Optimization Parameters		Potential Mineable Resource Parameters		
		FRESH ROCK	TRANSITION	SAPROLITE
Electricity Cost	USD/kWh	0.136	0.136	0.136
Discount Rate	%	5%	5%	5%
Gold Price	USD/oz	1,750	1,750	1,750
Payable Metal	%	99.95%	99.95%	99.95%
Transport & Refining Cost	USD/oz	8.0	8.0	8.0
Royalty Rate (Calculated)	%	8.00%	8.00%	8.00%
Royalty Cost	USD/oz	140.0	140.0	140.0
Net Mineralized Material Value	USD/oz	1,601.1	1,601.1	1,601.1
Nominal Milling Rate	t/d	16,438	16,438	16,438
Plant Throughput	kt/y	6,000	6,000	6,000
Recovery	%	92.5	95.0	96.0
Plant Labour Unit Rate	USD/t milled	0.57	0.57	0.57
Variable Power Consumption	kWh/t	37.11	29.34	21.57
Power Cost	USD/t milled	5.05	3.99	2.93
Consumables (reagents & media)	USD/t milled	8.00	6.42	4.83
Plant Maintenance & Lab.	USD/t milled	0.9	0.9	0.9
Total Processing Cost incl. Power	USD/t milled	14.51	11.87	9.23
Total Processing Cost incl. Power	USD/t milled	14.51	11.87	9.23
General & Administration Costs	USD/t milled	4.78	4.78	4.78
Rehabilitation & Closure	USD/t milled	0.26	0.26	0.26
Sustaining Capital	USD/t milled	0.25	0.25	0.25
Total Mineralized Material-based Cost	USD/t milled	19.80	17.16	14.51
Mining Dilution	%	13.00%	16.00%	9.00%
Mining Loss	%	6.00%	9.00%	4.00%
Total Mining Reference Cost	USD/t mined	2.69	2.56	2.05
Incremental Bench Cost	USD/10 m bench	0.040	0.040	0.040
IRA SLOPE Hanging Wall (East)	degree	50	40	30
IRA SLOPE Footwall (West)	degree	55	40	30
OSA Hanging Wall (East)	degree	44.7	35.4	27.8
OSA Footwall (West)	degree	48.9	36.4	27.8
Gram Value	USD/g	51.48	51.48	51.48
Total Mineralized Material-Based Cost	USD/t	19.80	17.16	14.51
Used COG (Inc. Proc. Rec.)	g/t	0.42	0.35	0.30

The Whittle nested shell results are presented in Table 16.6. These consider Measured, Indicated, and Inferred Mineral Resources.

The shell selection is presented in Table 16.7 and in Figure 16.2. Pit shell 26 was selected as the optimum final pit shell which corresponds to a revenue factor of 0.72 ($1,260/oz gold price). This shell has a total tonnage of 390 Mt, including 60 Mt of mineralized material. The pit shell was selected as having a good combination of best-case and specified-case scenario in Whittle™ while minimizing the risk when comparing the worst-case scenarios.

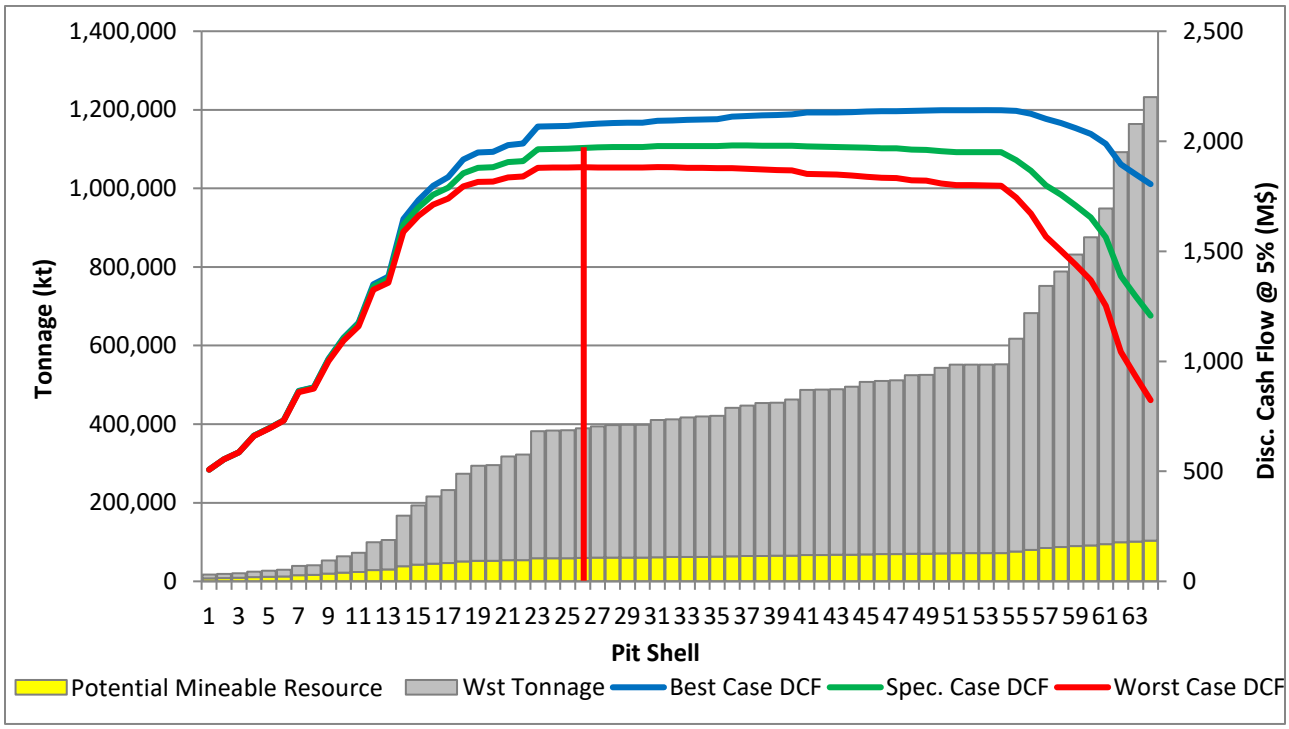
Table 16.6: Whittle Shell Results

	Best Case	Specified	Worst Case	Total	Mineralized Material	Strip	Grade	In-Situ	Waste	Revenue	Gold
Pit	Disc. @ 5%	Disc. @ 5%	Disc. @ 5%	Tonnage	Tonnage	Ratio		Gold	Tonnage	Factor	Price
Shell	(M$)	(M$)	(M$)	(kt)	(kt)	(W:O)	(g/t)	(k oz)	(kt)		(US$/oz)
1	507	507	507	16,894	7,723	1.19	3.07	348	9,171	0.30	525
2	554	554	554	18,911	8,643	1.19	3.00	386	10,267	0.31	542
3	587	587	587	20,629	9,432	1.19	2.89	418	11,197	0.32	560
4	662	662	661	24,823	11,034	1.25	2.81	482	13,789	0.33	579
5	695	695	694	26,895	11,844	1.27	2.73	520	15,051	0.34	600
6	732	732	730	29,439	12,751	1.31	2.64	562	16,689	0.36	622
7	865	864	859	39,639	15,853	1.50	2.53	692	23,786	0.37	646
8	882	880	875	41,165	16,279	1.53	2.46	725	24,886	0.38	672
9	1,011	1,008	1,001	53,211	19,374	1.75	2.39	872	33,836	0.40	700
10	1,107	1,104	1,094	63,895	21,952	1.91	2.31	998	41,943	0.42	730
11	1,176	1,171	1,160	72,591	23,872	2.04	2.25	1,099	48,719	0.44	764
12	1,350	1,340	1,324	99,486	28,774	2.46	2.21	1,344	70,712	0.46	800
13	1,385	1,374	1,357	105,442	30,047	2.51	2.14	1,419	75,395	0.48	840
14	1,647	1,620	1,588	166,913	38,862	3.29	2.15	1,852	128,051	0.51	884
15	1,733	1,699	1,661	193,376	42,140	3.59	2.10	2,046	151,235	0.53	933
16	1,797	1,757	1,712	215,602	44,813	3.81	2.05	2,207	170,789	0.56	988
17	1,837	1,790	1,739	232,237	46,680	3.98	2.00	2,328	185,557	0.60	1,050
18	1,917	1,855	1,794	273,772	50,508	4.42	1.96	2,562	223,264	0.64	1,120
19	1,949	1,880	1,815	293,965	51,903	4.66	1.96	2,658	242,062	0.65	1,138
20	1,952	1,882	1,817	295,706	52,170	4.67	1.95	2,676	243,535	0.66	1,155
21	1,982	1,905	1,836	317,819	53,679	4.92	1.95	2,777	264,140	0.67	1,173
22	1,990	1,910	1,840	322,915	54,252	4.95	1.93	2,810	268,664	0.68	1,190
23	2,066	1,964	1,880	381,947	58,763	5.50	1.94	3,072	323,184	0.69	1,208
24	2,068	1,965	1,880	383,392	58,972	5.50	1.93	3,087	324,420	0.70	1,225
25	2,070	1,966	1,881	384,947	59,079	5.52	1.92	3,101	325,868	0.71	1,243
26	2,076	1,970	1,882	389,815	59,701	5.53	1.90	3,131	330,114	0.72	1,260
27	2,081	1,972	1,880	394,749	60,288	5.55	1.89	3,157	334,462	0.73	1,278
28	2,083	1,973	1,881	397,487	60,503	5.6	1.87	3,176	336,984	0.74	1,295
29	2,084	1,973	1,881	398,076	60,632	5.6	1.86	3,186	337,443	0.75	1,313
30	2,084	1,974	1,881	398,678	60,669	5.6	1.85	3,195	338,009	0.76	1,330
31	2,093	1,978	1,883	410,485	61,456	5.7	1.85	3,246	349,029	0.77	1,348
32	2,095	1,978	1,882	412,352	61,719	5.7	1.84	3,260	350,632	0.78	1,365
33	2,098	1,978	1,880	417,428	62,217	5.7	1.82	3,284	355,211	0.79	1,383
34	2,099	1,978	1,880	419,318	62,355	5.7	1.82	3,297	356,963	0.80	1,400
35	2,100	1,978	1,878	421,083	62,531	5.7	1.81	3,309	358,552	0.81	1,418
36	2,111	1,981	1,878	441,455	63,680	5.9	1.80	3,384	377,775	0.82	1,435
37	2,115	1,981	1,875	447,640	64,300	6.0	1.79	3,413	383,340	0.83	1,453
38	2,118	1,980	1,872	453,522	64,830	6.0	1.78	3,440	388,692	0.84	1,470
39	2,118	1,980	1,870	455,004	65,028	6.0	1.77	3,450	389,976	0.85	1,488
40	2,122	1,980	1,867	463,029	65,469	6.1	1.76	3,482	397,560	0.86	1,505
41	2,130	1,976	1,851	487,503	67,214	6.3	1.75	3,568	420,289	0.87	1,523
42	2,131	1,976	1,851	487,852	67,290	6.3	1.75	3,575	420,562	0.88	1,540
43	2,131	1,974	1,848	489,155	67,504	6.2	1.74	3,585	421,651	0.89	1,558
44	2,133	1,972	1,844	495,353	68,051	6.3	1.73	3,611	427,302	0.90	1,575
45	2,136	1,970	1,838	507,724	68,718	6.4	1.72	3,654	439,006	0.91	1,593
46	2,136	1,968	1,834	510,375	69,057	6.4	1.71	3,667	441,318	0.92	1,610
47	2,136	1,967	1,832	511,407	69,145	6.4	1.70	3,675	442,262	0.93	1,628
48	2,139	1,962	1,823	524,636	70,260	6.5	1.69	3,723	454,377	0.94	1,645
49	2,139	1,961	1,821	525,640	70,369	6.5	1.68	3,731	455,271	0.95	1,663
50	2,141	1,955	1,808	543,224	71,329	6.6	1.68	3,785	471,896	0.96	1,680
51	2,141	1,951	1,800	551,121	72,005	6.7	1.67	3,815	479,115	0.97	1,698
52	2,141	1,951	1,800	551,229	72,011	6.7	1.66	3,819	479,218	0.98	1,715
53	2,141	1,951	1,800	551,322	72,030	6.7	1.66	3,823	479,292	0.99	1,733
54	2,141	1,950	1,798	552,270	72,141	6.7	1.65	3,830	480,130	1.00	1,750
55	2,139	1,914	1,744	617,236	76,242	7.1	1.61	4,035	540,995	1.05	1,838
56	2,125	1,866	1,671	682,466	80,203	7.5	1.55	4,238	602,264	1.15	2,013
57	2,101	1,799	1,567	751,559	84,945	7.8	1.48	4,436	666,614	1.25	2,188
58	2,083	1,757	1,502	788,545	87,452	8.0	1.43	4,545	701,093	1.35	2,363
59	2,060	1,708	1,438	831,377	89,598	8.3	1.39	4,652	741,779	1.45	2,538
60	2,034	1,654	1,369	875,413	91,680	8.5	1.36	4,754	783,733	1.55	2,713
61	1,988	1,563	1,253	948,822	94,815	9.0	1.32	4,900	854,007	1.65	2,888
62	1,896	1,388	1,043	1,091,956	99,237	10.0	1.30	5,136	992,719	1.75	3,063
63	1,850	1,297	932	1,163,956	101,465	10.5	1.27	5,260	1,062,491	1.85	3,238
64	1,806	1,207	824	1,232,316	103,554	10.9	1.25	5,370	1,128,761	1.95	3,413

Table 16.7: Pit Shell Selection

Shell Selection		Max NPV	Selection
Shell Number	**53**	**36**	**26**
Shell RF	0.99	0.82	0.72
Shell Price	1,733	1,435	1,260
Total Tonnage (kt)	551,322	441,455	389,815
Waste Tonnage (kt)	479,292	377,775	330,114
Strip Ratio (W:O)	6.65	5.93	5.53
Mineralized Material Tonnage (kt)	72,030	63,680	59,701
Grade (g/t)	1.66	1.80	1.90
In-situ Gold (k oz)	3,823	3,384	3,131
DCF @ 5 % (M$)	1,951	1,981	1,970
LOM (Y)	13.2	11.6	10.9

Figure 16.2: Pit by Pit Graph @ USD 1,750/oz Gold Price



Source: GMS 2024

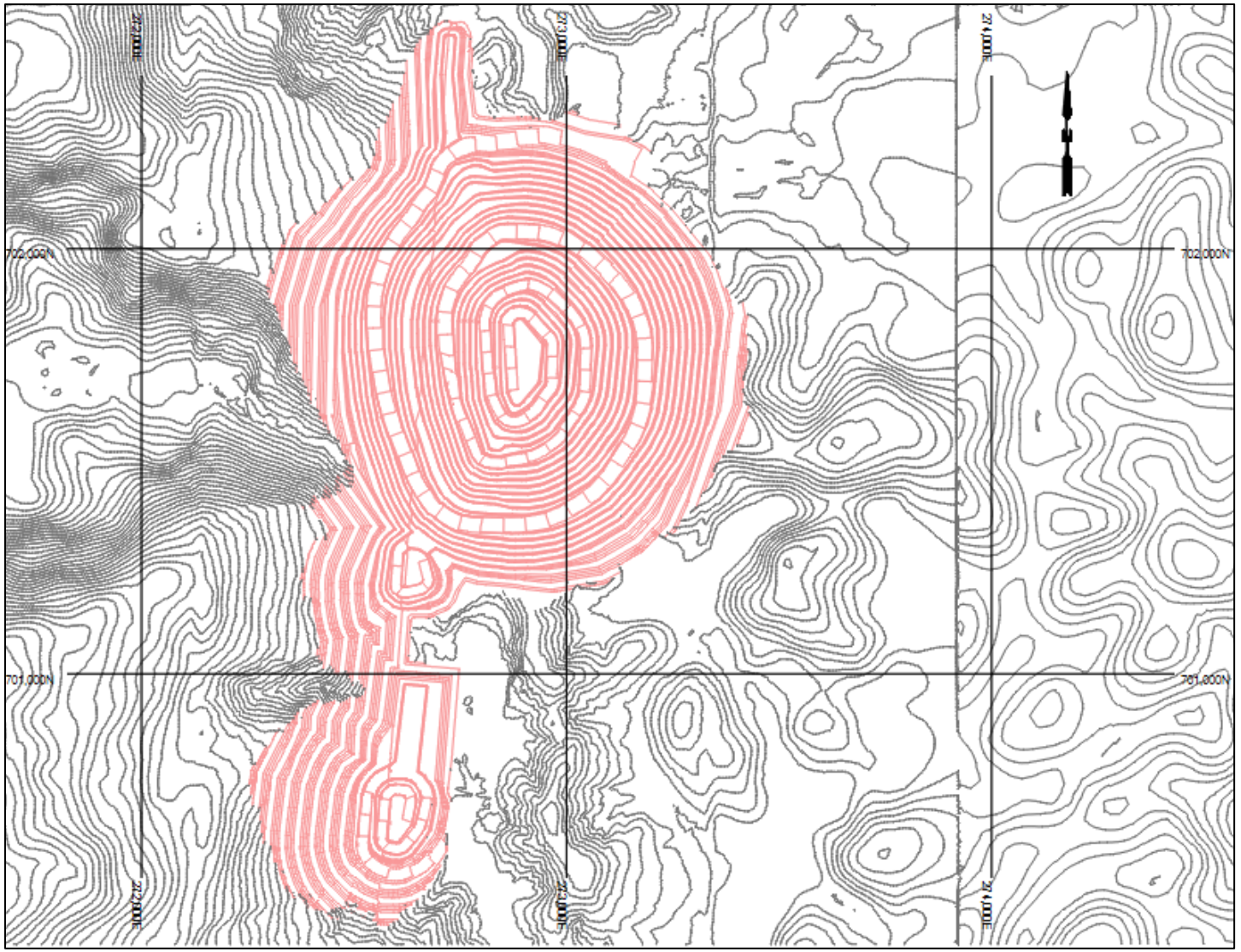
16.4.4 Mine Phases

The open pit mine designs were developed using optimal Whittle™ shells. An optimized mining sequence was computed to identify the optimal shells for defining pushbacks and aiming to maximize the project Net Present Value (NPV). Shells 9, 15 and 26 were subsequently selected. The mining of the Oko West open pit is planned with five (5) phases. Phase 1 was guided by pit shell 9, Phase 2 was guided by shell 15, and Phases 3 and 4 were guided by the optimal pit shell 26. Phase 0 represents early works: within the Phase 1, to access to hard rock for construction, and within the Phase 2, to build the UG portal. The mining data of each mining phase are summarized in Table 16.8. The final configuration of the pit is presented in Figure 16.3. Figure 16.4 represents the configuration of the different pit phases.

Table 16.8: Mining Resources by Phase

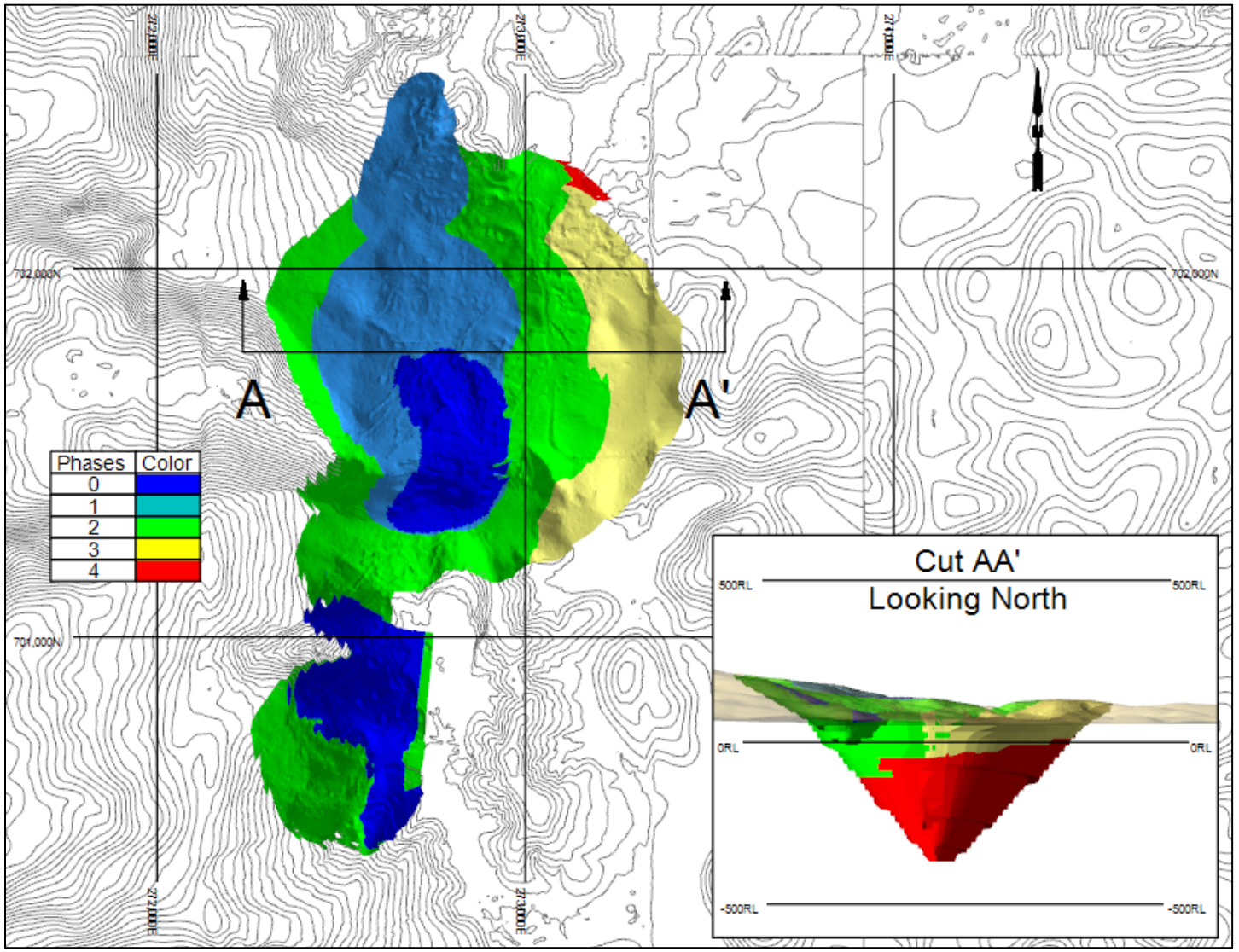
Mining Resources by Phase		Phase 0	Phase 1	Phase 2	Phase 3	Phase 4	TOTAL
Total tonnage	k-tonnes	10,910	48,886	186,117	65,149	114,283	425,345
Waste Tonnage	k-tonnes	9,003	32,440	161,836	63,753	97,611	364,643
Saprolite Waste	k-tonnes	6,792	17,128	38,150	16,562	135	78,767
Transition waste	k-tonnes	767	4,072	14,199	8,387	391	27,816
Fresh Rock Waste	k-tonnes	1,444	11,239	109,486	38,805	97,085	258,060
Mineralized Material Tonnage	k-tonnes	1,907	16,447	24,281	1,396	16,671	60,702
Saprolite Mineralized Material Tonnage	k-tonnes	1,631	4,113	1,916	0	0	7,660
Saprolite Mineralized Material Grade	g/t	1.51	1.62	0.84	-	0.43	1.40
Trans Mineralized Material Tonnage	k-tonnes	139	2,252	1 018	3	0	3 411
Trans Mineralized Material Grade	g/t	0.97	1.80	0.81	0.49	-	1.47
Fresh Rock Mineralized Material Tonnage	k-tonnes	137	10,082	21,348	1,393	16,671	49,631
Fresh Rock Mineralized Material Grade	g/t	0.79	1.77	1.62	0.97	2.10	1.79
Strip Ratio (W:O)		4.72	1.97	6.67	45.68	5.86	6.01

Figure 16.3: End of LOM Pit Layout



Source GMS, 2024
**Note: Not to scale.*

Figure 16.4: Phase Limits



Source GMS 2024
**Note: Not to scale.*

16.4.5 Ramp Designs

The pit is situated on the east side of the property, northwest of the planned crusher and mill location, positioned on a hillside. The pit will expand from its centre to the final wall over four (4) main phases. In each phase, the haulage ramp transitions from a double lane to a single lane at the latest benches, optimizing the capture of mineralized material pockets at the bottom of the pit. Pit exits are planned to the north to allow for the shortest haulage distance to the crusher and the waste dump. Ramp designs for the mining phases are as follows:

Phase 0 includes two (2) small pits: one located within the Phase 1 area, designed to access fresh rock as quickly as possible for construction purposes, and a second pit designed to minimize excavation volume while reaching the underground (UG) portal. Phase 0 pits feature single-lane haulage ramps.

Phase 1 is the initial operation phase, starting at the centre of the deposit. The ramp is located on the east wall, with a pit exit near the previous location, allowing for the closest route to the crusher. Phase 1 measures 1,200 m in length, 550 m in width, and reaches a maximum depth of 270 m from the top of the hill to the bottom.

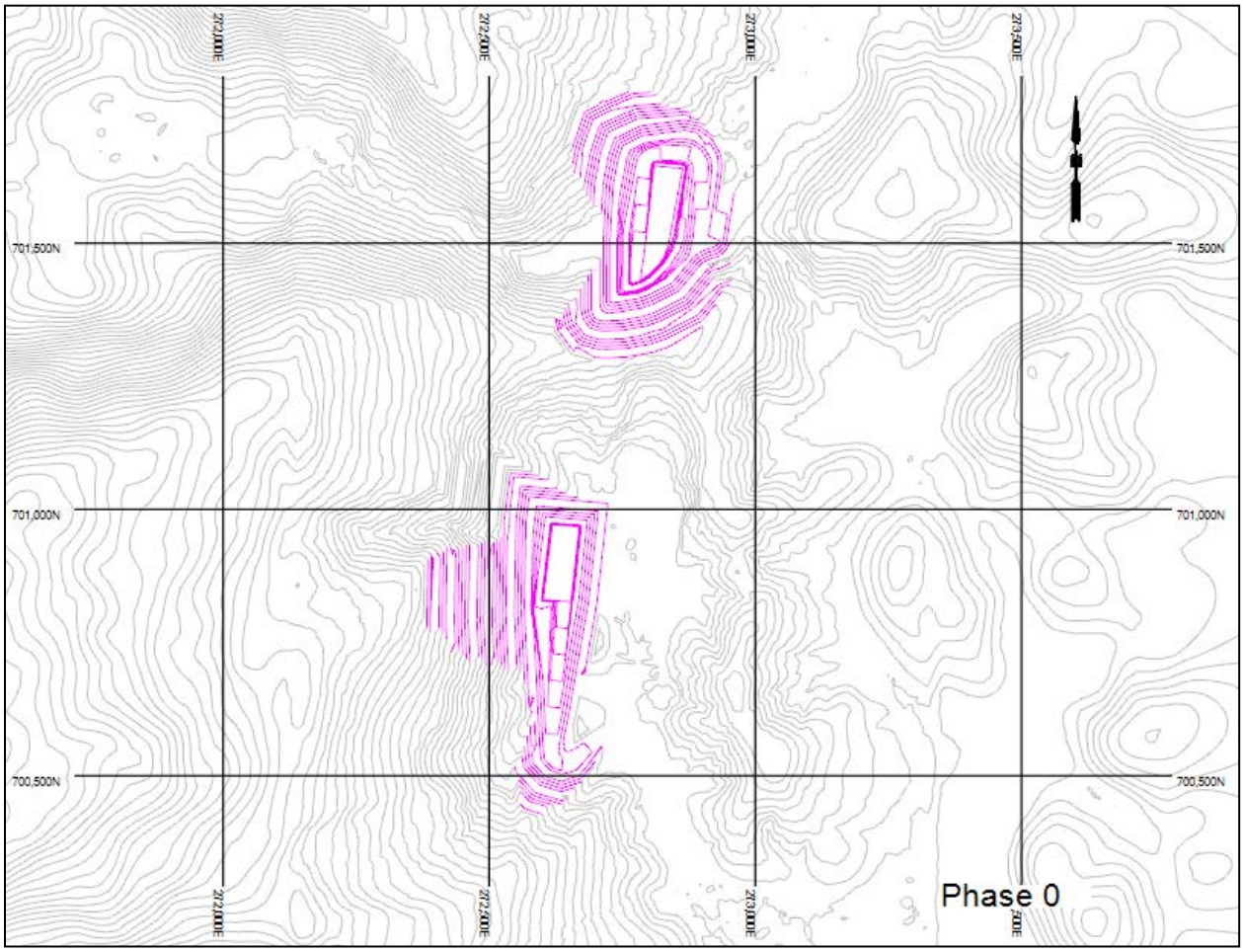
Phase 2 starts at the final wall on the west side with ramp exit to the North. Phase 2 is 2,100 m in length, 950 m in width, and reaches a maximum depth of 410 m.

Phase 3 involves mining a temporary access ramp located on the east wall, which is required since the final ramp is not yet available.

Phase 4 is the final nested pit. This phase starts at the final wall on the east side and ramps in a spiral. Phase 4 is 2,100 m in length, 1,150 m in width, and reaches a maximum depth of 570 m.

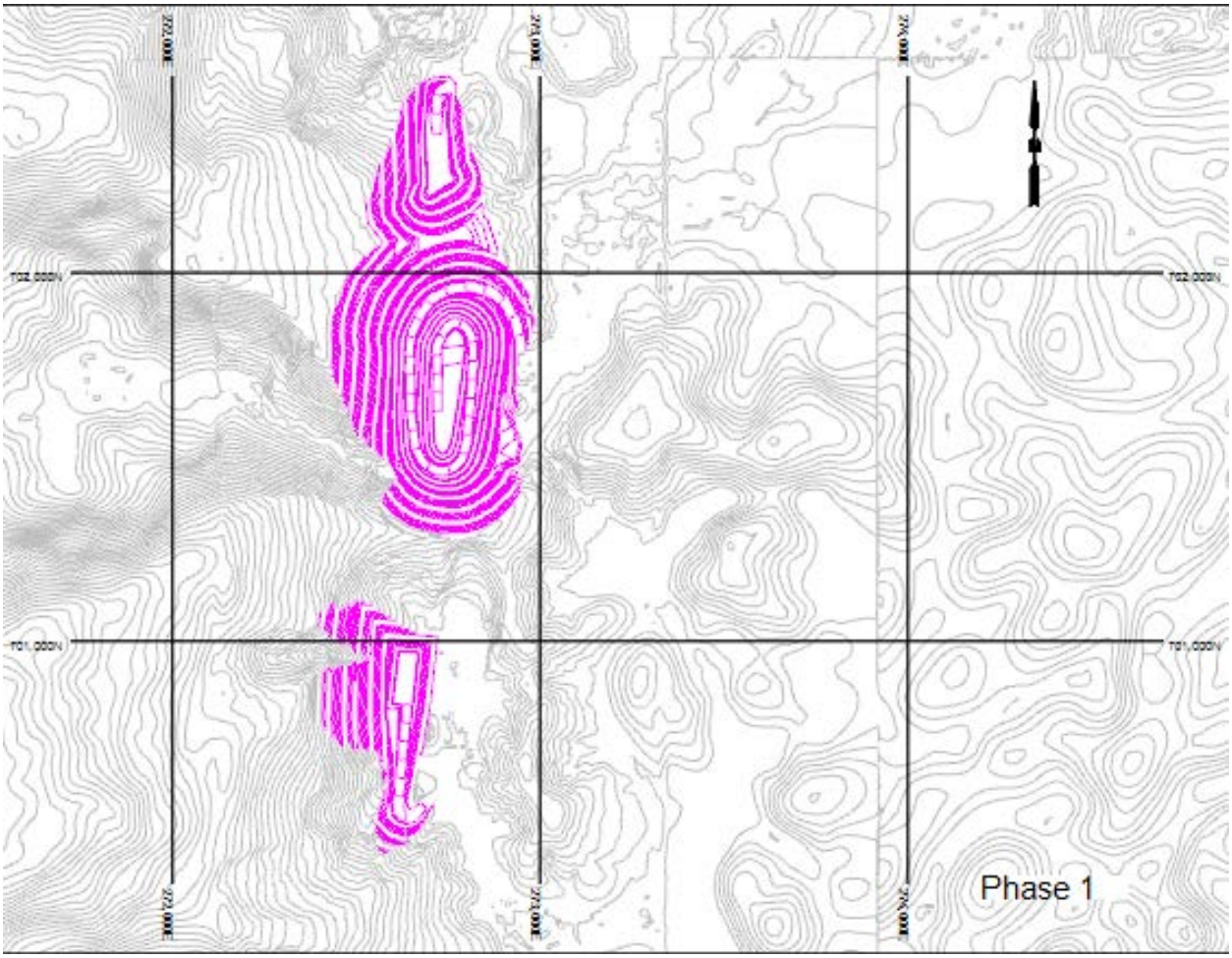
The different Pit phases are presented in Figure 16.5 to Figure 16.9.

Figure 16.5: Phase 0



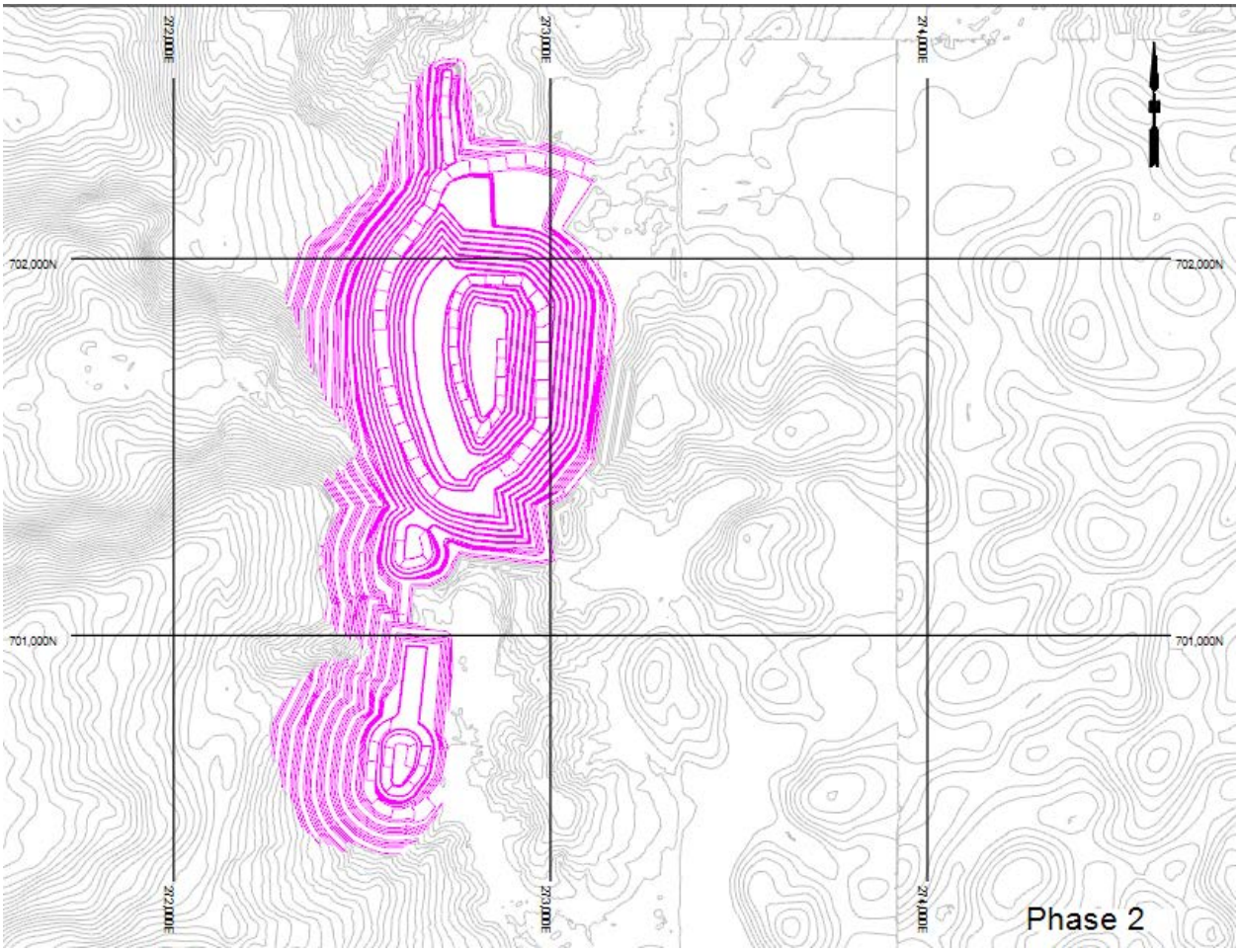
Source GMS, 2024
**Note: Not to scale.*

Figure 16.6: Phase 1



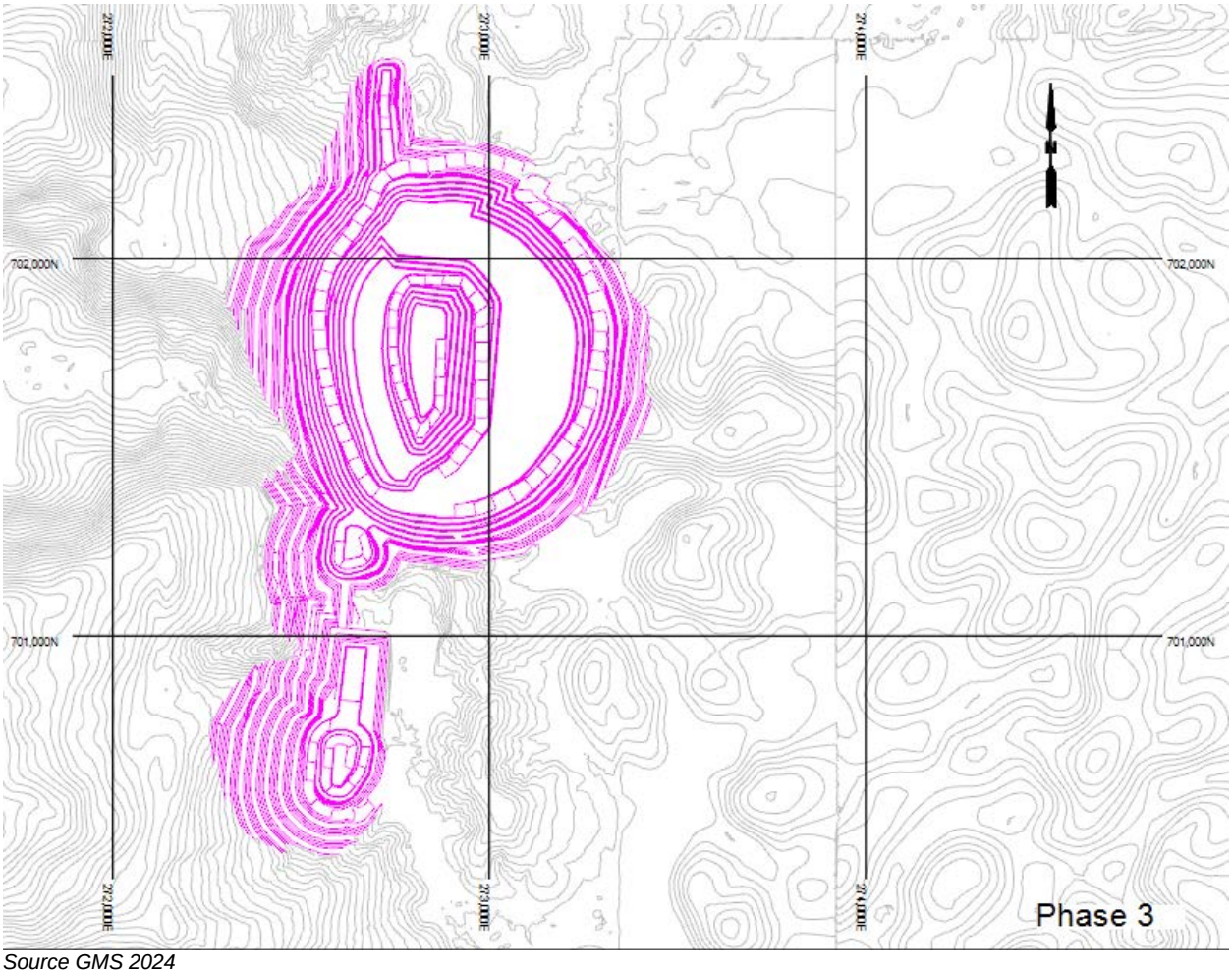
Source GMS 2024
**Note: Not to scale*

Figure 16.7: Phase 2



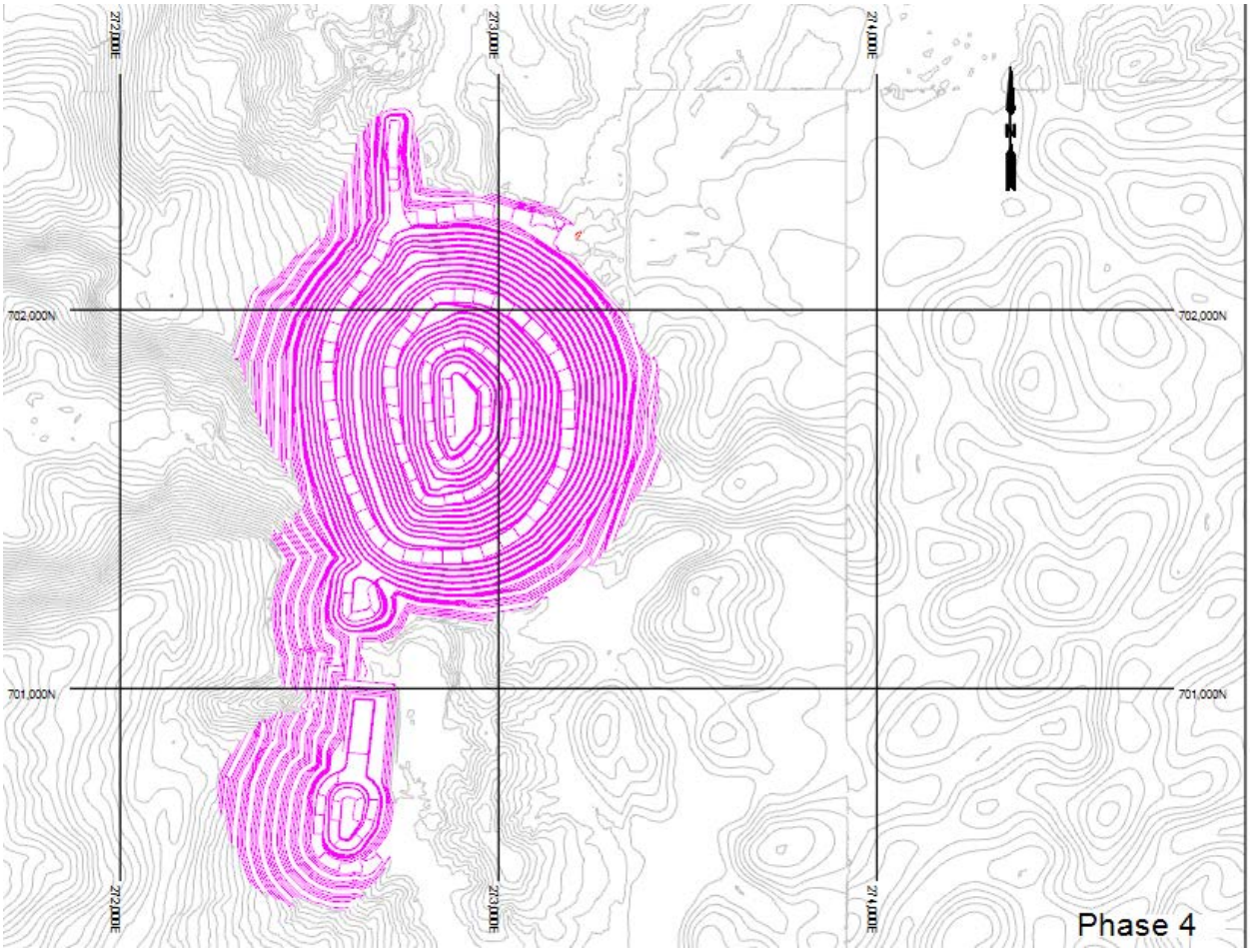
Source GMS 2024.
**Note: Not to scale*

Figure 16.8: Phase 3



Source GMS 2024
**Note: Not to scale*

Figure 16.9: Phase 4



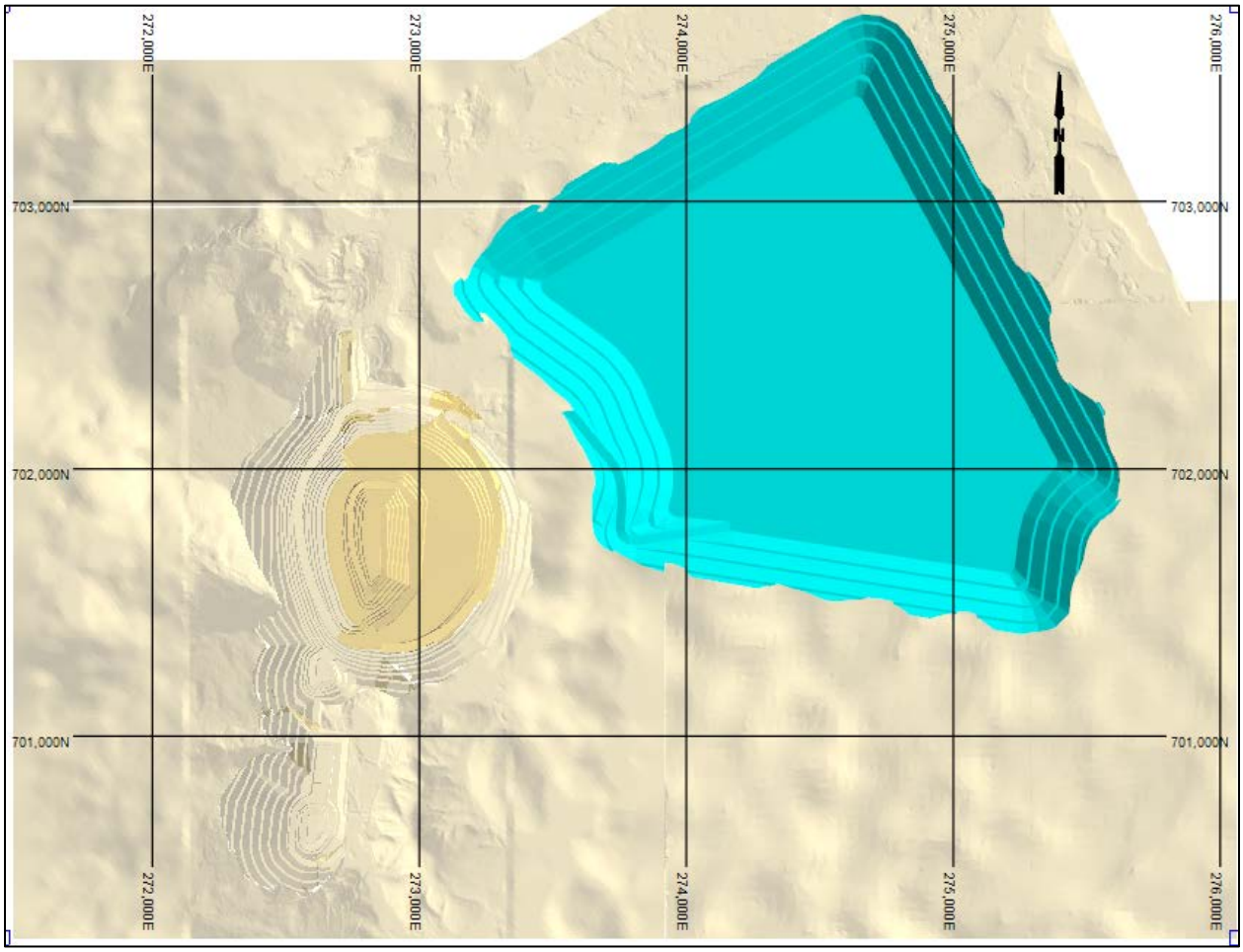
Source GMS 2024
**Note: Not to scale*

16.4.6 Waste Rock Storage Facility

A total of 365 Mt of waste rock, transition and saprolite will be mined in the open pit over the LOM. All materials will be hauled to the waste rock storage facility and stacked using a track dozer. The material pile will be constructed in layers oriented in the north-south and west-east axes. Figure 16.10 illustrates the location of the waste storage. Table 16.9 presents the capacities and general design considerations.

A total of 1 Mt of waste rock is allocated for construction activities, including the initial tailings dam, site laydowns, haulage roads, and concrete.

Figure 16.10: Waste Storage Facility



Source GMS 2024.
**Note: Not to scale*

Table 16.9: Waste Storage Facility Capacity and Design Parameters

Facility	Capacity (Mm^3)	Capacity Used (Mm^3)	% Filled	OSA (Degrees)
Waste Storage Facility	216	182	84	22

16.4.7 Mineralized Material Stockpile

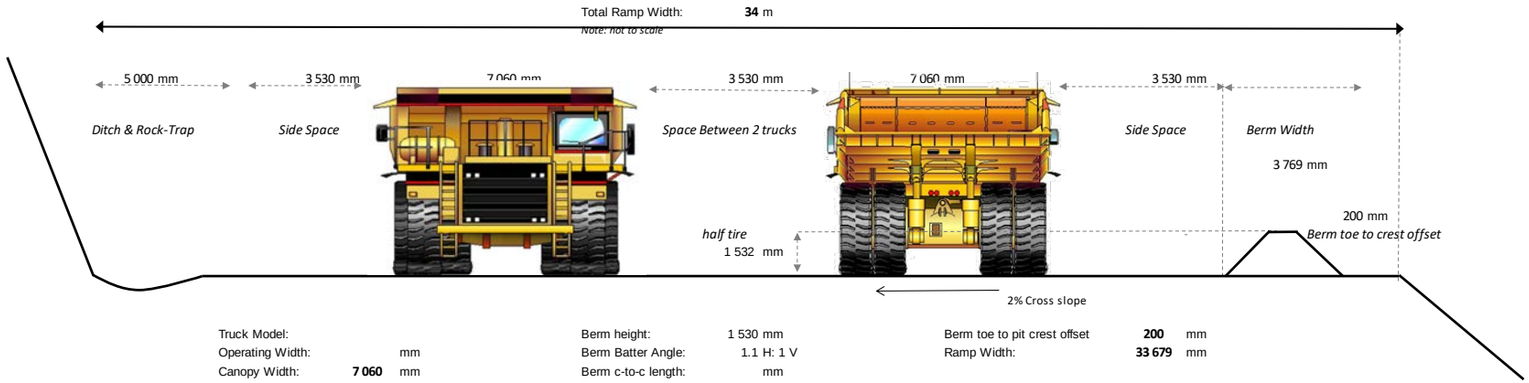
Mineralized material will be stockpiled south of pit, next to the crusher, to ensure a steady flow of material to the mill.

16.4.8 Mine Haulage Roads

The ramp was designed for double lane traffic to accommodate 139-t class off-highway trucks, with the exception of the final benches of the pits, where it transitions to one-way access. Single lane roads and ramps are 23 m wide and the double lane roads, and ramps are 34 m wide. Figure 16.11 shows the configuration of the double lane planned roads and ramps, and Figure 16.12 shows the single lane planned road and ramp configuration.

The ramp gradient is 10% with a minimum turning radius of 25 m. Pit exit is located towards the north end to minimize distance to the mill, stockpile, and waste dump.

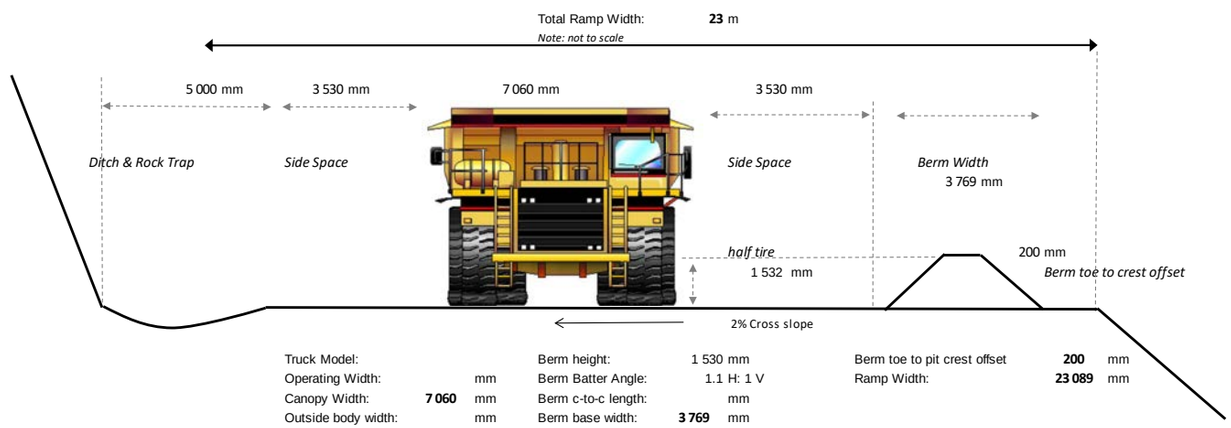
Figure 16.11: Double Lane Ramp Design Criteria



Truck Model:			Berm height:	1 530	mm	Berm toe to pit crest offset	**200** mm
Operating Width:		mm	Berm Batter Angle:	1.1 H: 1 V		Ramp Width:	**33 679** mm
Canopy Width:	**7 060**	mm	Berm c-to-c length:		mm		
Outside body width:		mm	Berm base width:	**3 769**	mm		
Tire Height:	**3 063**	mm					

Source GMS 2024.
**Note: Not to scale*

Figure 16.12: Single Lane Ramp Design Criteria



Truck Model:			Berm height:	1 530	mm	Berm toe to pit crest offset	**200** mm
Operating Width:		mm	Berm Batter Angle:	1.1 H: 1 V		Ramp Width:	**23 089** mm
Canopy Width:	**7 060**	mm	Berm c-to-c length:		mm		
Outside body width:		mm	Berm base width:	**3 769**	mm		
Tire Height:	3 063	mm					

Source GMS 2024.
**Note: Not to scale*

16.4.9 Open Pit Production Schedule

The LOM production schedule for the open pit mines was optimized using Minemax[tm] Scheduler, which is an industry leading schedule optimizer using best in class CPLEX technology. Minemax[tm] Scheduler is an automated mine scheduling tool which leverages multi-period optimization to determine maximum net present value ("NPV") while imposing various physical constraints and targets. The optimization includes mine sequencing and mining rate, stockpile usage and rehandling, and fleet usage. The strategic optimal plan from Minemax[tm] on an annual basis was then further detailed by year using Deswik[tm] to track material movements, stockpile inventory, mill blending, waste movements, and equipment usage.

Open pit mining activities are planned over a duration of 15 years, which include two (2) years of pre-production. The mining rate will ramp up to reach a maximum range of 40-44 Mtpa on Year 1 until Year 6, and it will decrease afterwards. Figure 16.13 presents the open pit mining schedule by material type (without stockpile reclaim movement). Variations in the quantities of mineralized material mined are due to three (3) effects:

- The contribution of underground mine production to mill feed is variable, ramping up in years 3 and 4, before remaining relatively stable for the balance of the LoM, decreasing the requirement for mineralized material from the open pit in later years.
- Mineralized saprolite availability in the earlier years allows for a higher mill throughput.
- Availability of mineralized material from stockpile (especially Y4).

Figure 16.14 presents the yearly tonnage mined by phase. In any given year, there are up to three (3) phases mined simultaneously, with one phase typically serving as the primary source of mineralized material and the others focusing on waste removal. The mine plan kept a maximum sinking rate of 60 m per year per phase.

Details of mine production showing mined grades and material movement are presented in Table 16.10. Figure 16.16 through Figure 16.19 depict the progression of the pit and waste storage facility over the years and at the end of the open pit at Year 13.

Figure 16.13: Open Pit Mine Production by Material Type (without stockpile reclaim)

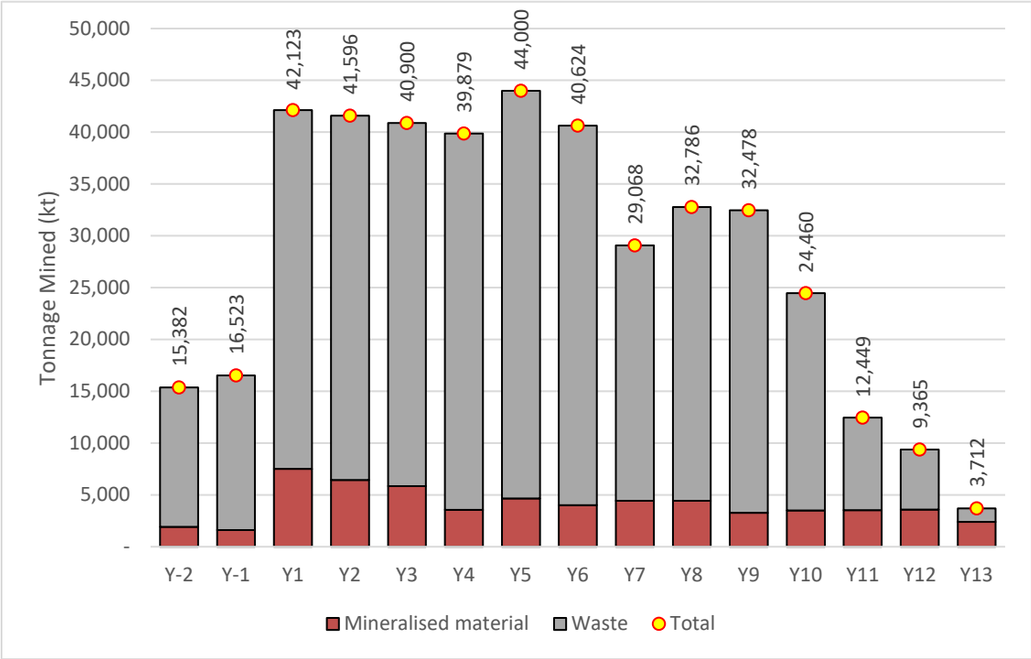


Figure 16.14: Open Pit Mine Production by Phase

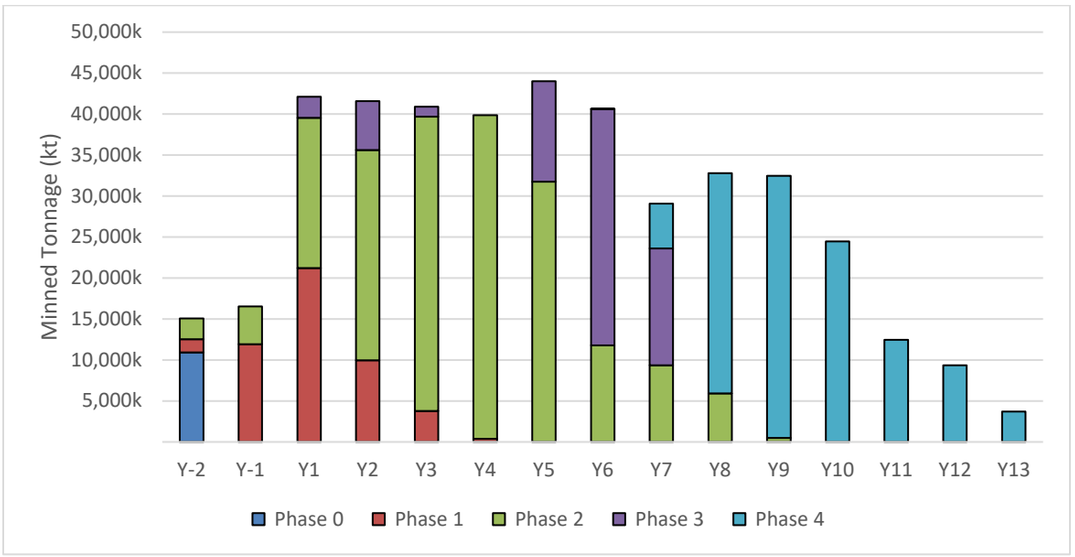


Figure 16.15: Mineralized Material Type Production Mined

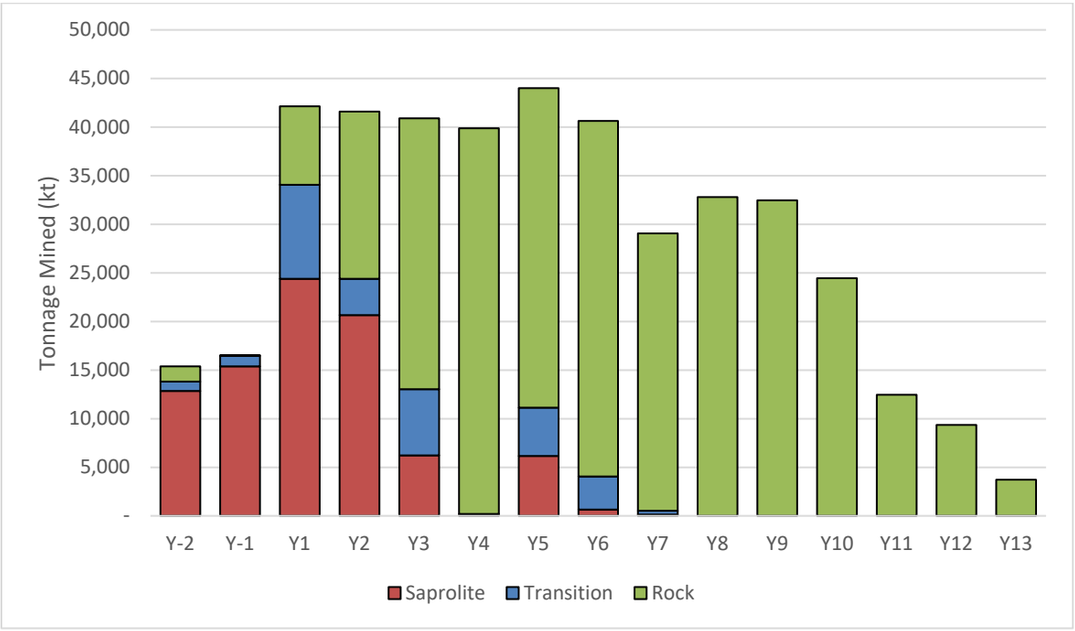
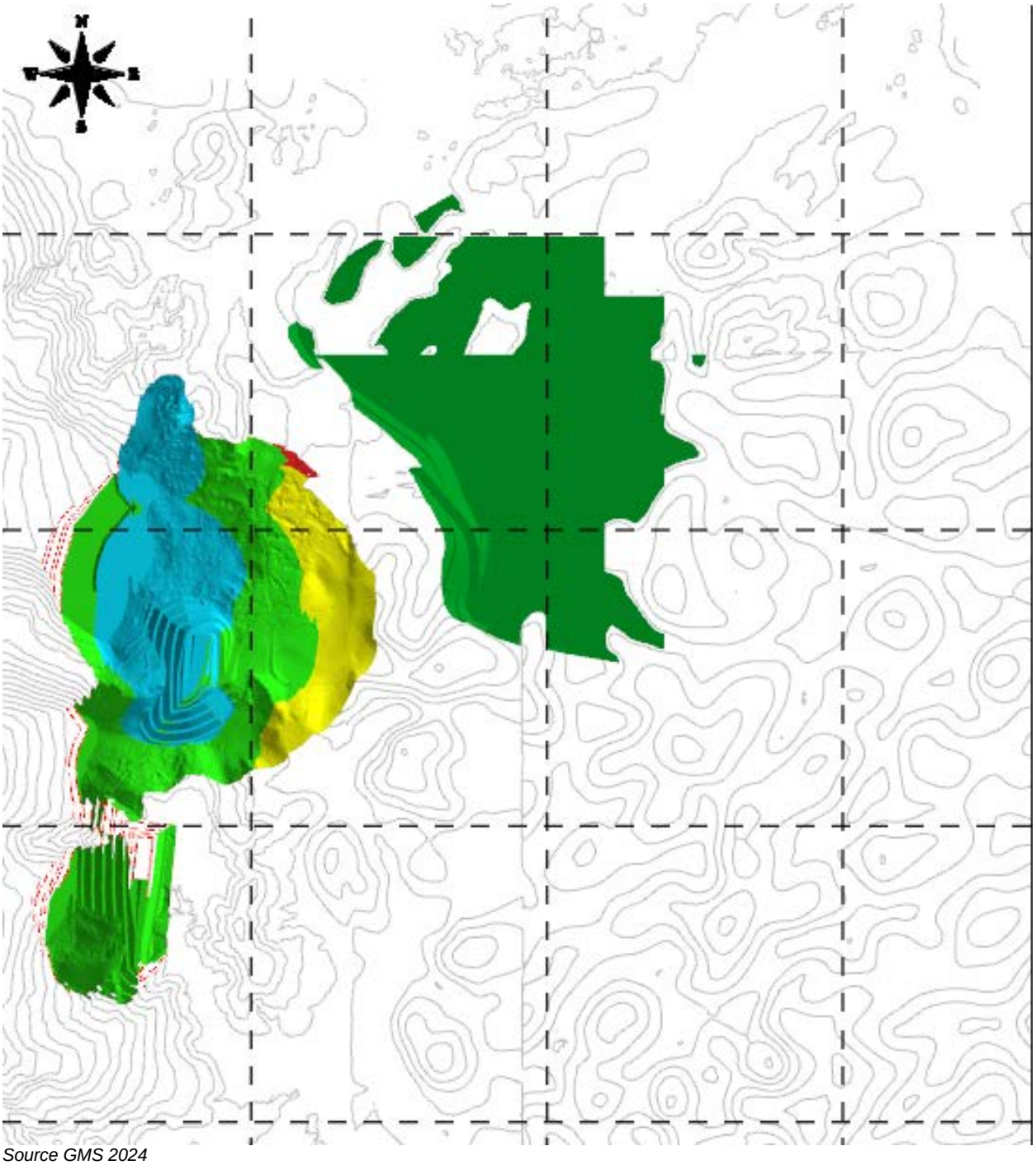


Table 16.10: Open Pit Mining Schedule Summary

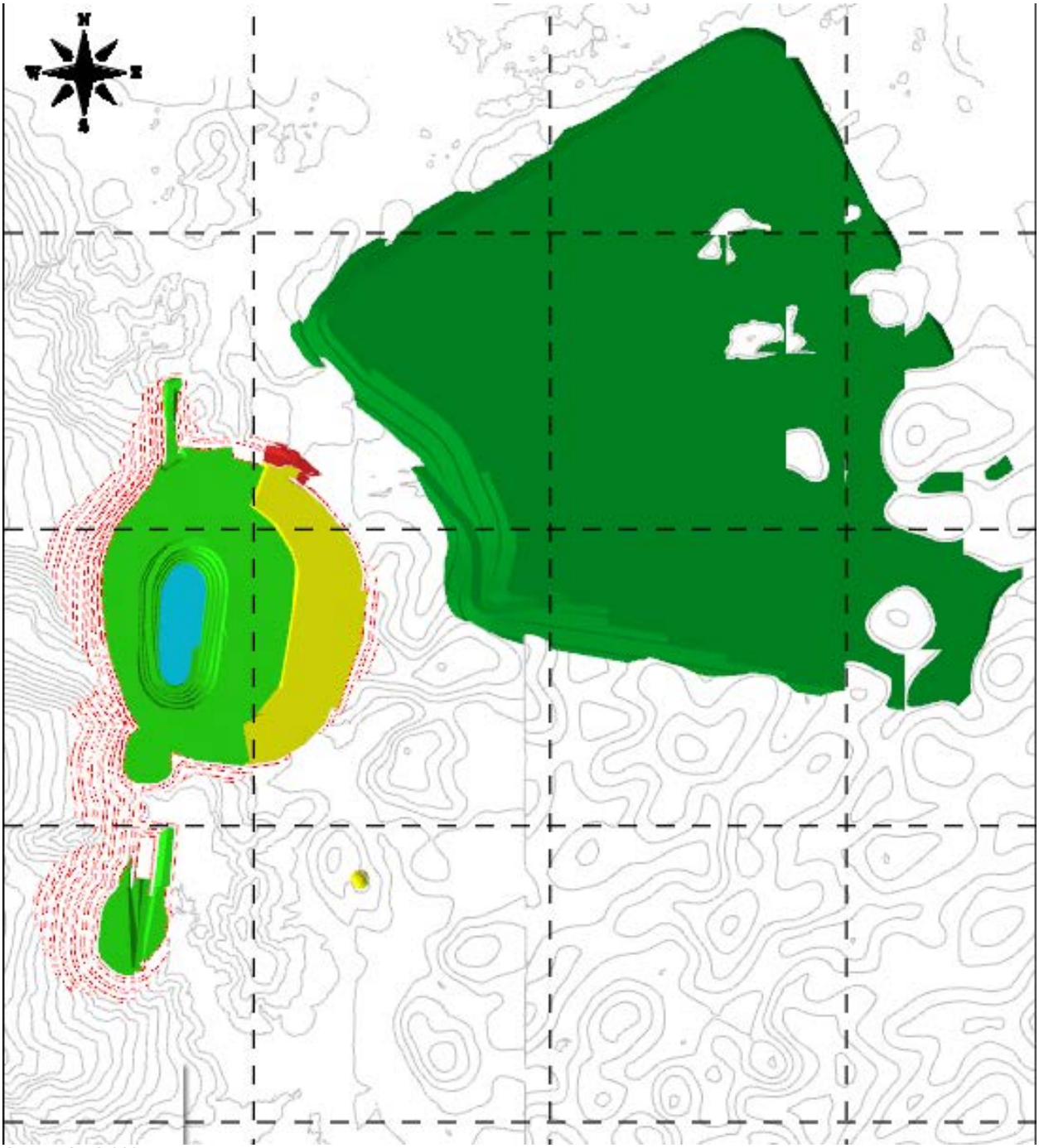
Description	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
MINED - OPEN PIT																	
OP Waste	k-tonnes	364,643	13,466	14,911	34,614	35,148	35,042	36,308	39,345	36,613	24,637	28,348	29,201	20,971	8,931	5,793	1,315
Overburden	k-tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Saprolite Waste	k-tonnes	78,767	11,211	13 899	21,107	19,710	5,906	-	6,152	647	135	-	-	-	-	-	-
Trans. Waste	k-tonnes	27,816	810	968	7,630	3,192	6,259	203	4,967	3,395	391	-	-	-	-	-	-
Fresh Rock Waste	k-tonnes	258,060	1,444	43	5,876	12,246	22,877	36,105	28,227	32,571	24,111	28,348	29,201	20,971	8,931	5,793	1,315
OP MM Tonnage	k-tonnes	60,702	1,916	1 612	7,509	6,448	5,858	3 571	4,655	4,011	4,432	4,438	3,278	3,489	3,518	3,572	2,397
OP MM Grade	g/t	1.7	1.4	1.6	1.6	1.5	1.6	1.5	1.5	1.5	1.5	1.9	1.7	1.9	2.3	2.2	3.0
OP MM Ounces	koz	3,362	87	82	385	305	298	168	219	196	208	266	181	214	261	256	234
Saprolite MM Tonnage	k-tonnes	7,660	1,641	1 476	3,285	945	313	-	-	0	-	-	-	-	-	-	-
Saprolite MM Grade	g/t	1.4	1.5	1.6	1.5	0.9	0.8	-	-	0.4	-	-	-	-	-	-	-
Saprolite MM Ounces	koz	345	79	75	157	26	8	-	-	0	-	-	-	-	-	-	-
Trans. MM Tonnage	k-tonnes	3,411	139	132	2,040	544	553	-	-	3	-	-	-	-	-	-	-
Trans. MM Grade	g/t	1.5	1.0	1.8	1.8	1.0	0.9	-	-	0.5	-	-	-	-	-	-	-
Trans. MM Ounces	koz	161	4	8	116	17	16	-	-	0	-	-	-	-	-	-	-
Fresh Rock MM Tonnage	k-tonnes	49,631	137	4	2,183	4,959	4,991	3,571	4,655	4,008	4,432	4,438	3,278	3,489	3,518	3,572	2,397
Fresh Rock MM Grade	g/t	1.8	0.8	1.3	1.6	1.6	1.7	1.5	1.5	1.5	1.5	1.9	1.7	1.9	2.3	2.2	3.0
Fresh Rock MM Ounces	koz	2,856	3	0	112	262	274	168	219	196	208	266	181	214	261	256	234
Strip Ratio (W:O)	-	6.0	7.0	9.3	4.6	5.5	6.0	10.2	8.5	9.1	5.6	6.4	8.9	6.0	2.5	1.6	0.5
OP Waste & MM Tonnage	k-tonnes	425,345	15,382	16,523	42,123	41,596	40,900	39,879	44,000	40,624	29,068	32,786	32,478	24,460	12,449	9,365	3,712

Figure 16.16: Mine Development Y-1



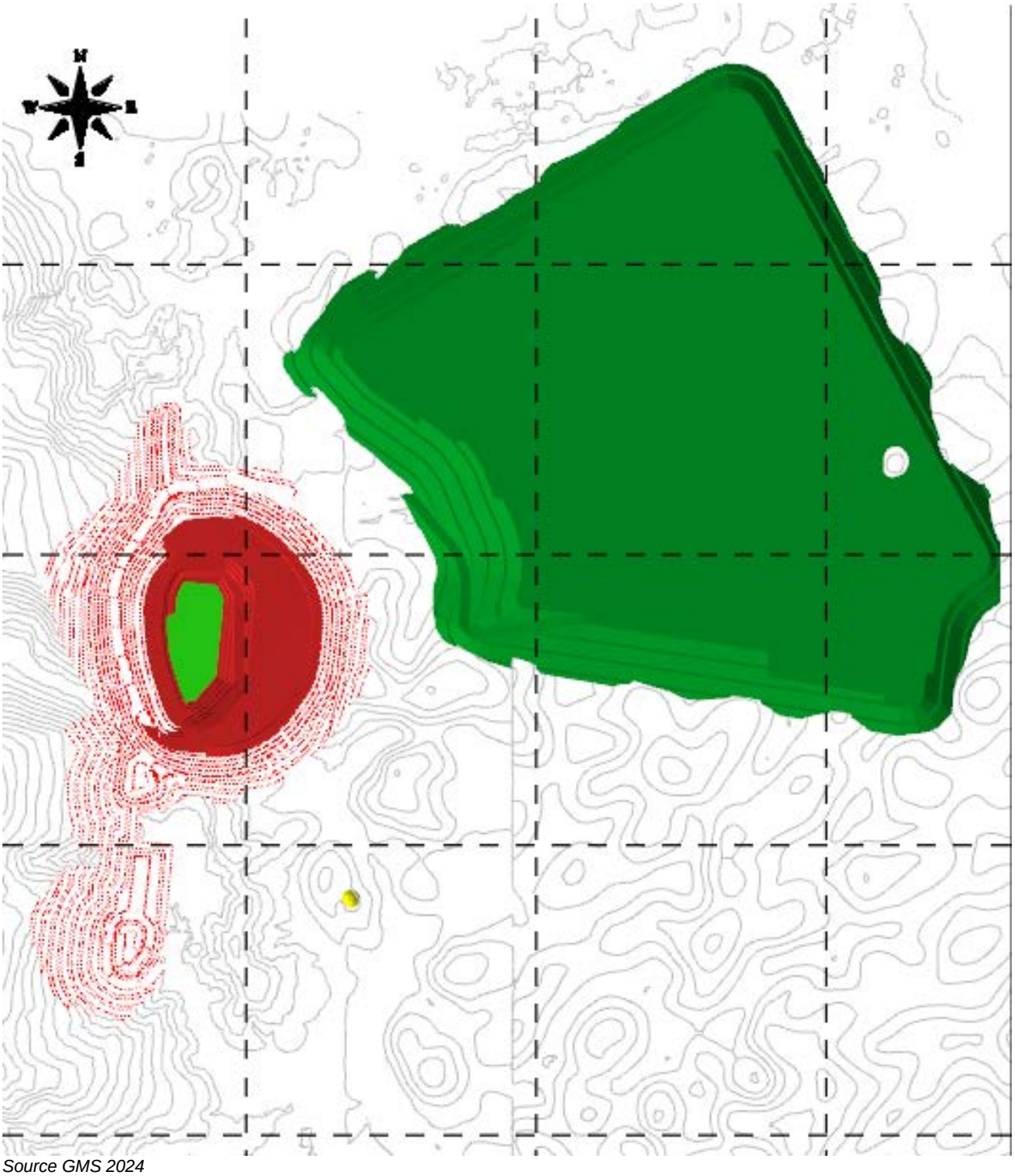
Source GMS 2024
**Note: Not to scale*

Figure 16.17: Mine Development Y4



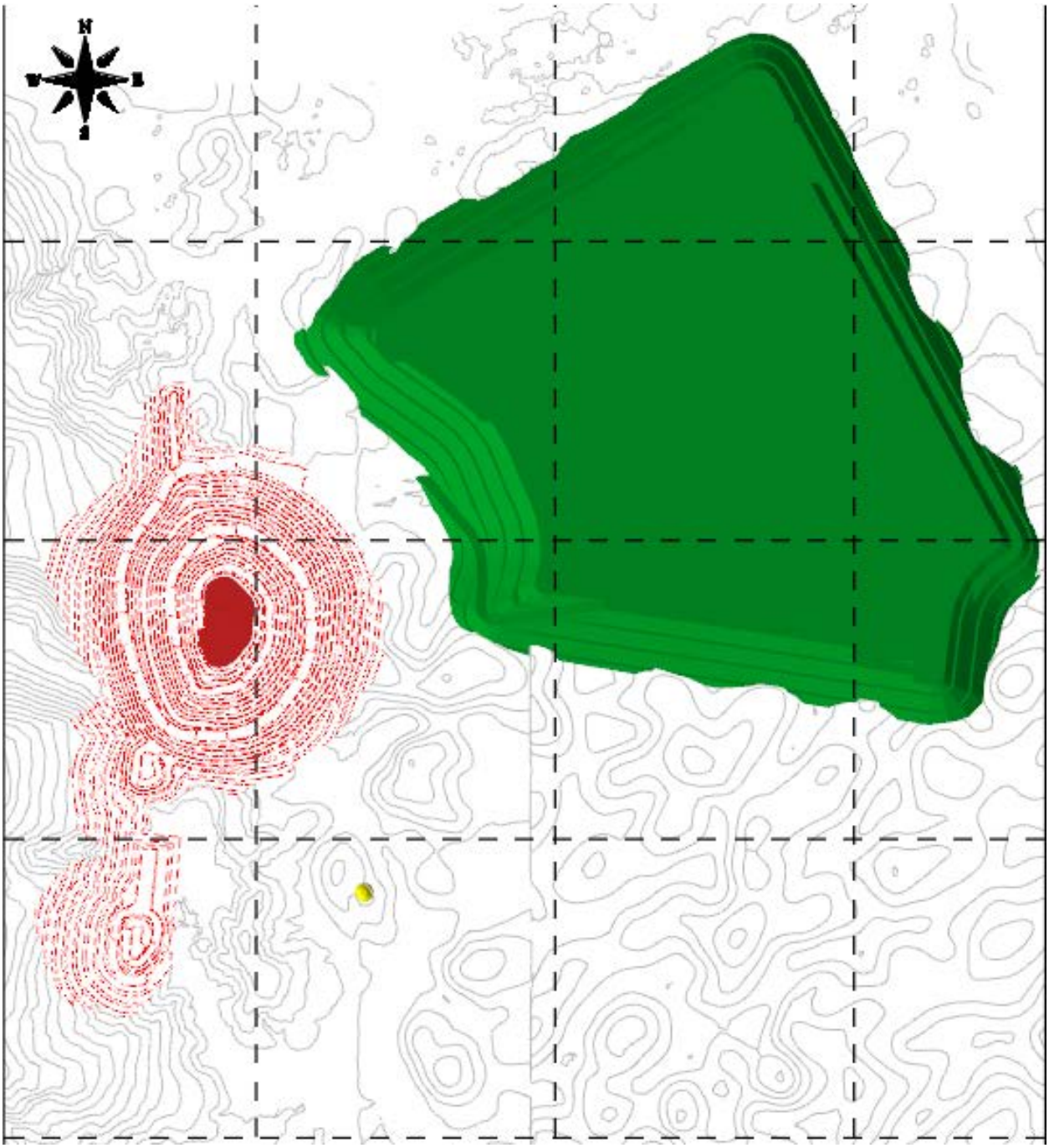
Source GMS 2024
**Note: Not to scale*

Figure 16.18: Mine Development Y9



Source GMS 2024
**Note: Not to scale*

Figure 16.19: Mine Development – Y13 (End of LOM)



Source GMS 2024
**Note: Not to scale.*

16.4.10 Mine Operations and Equipment Selection

16.4.10.1 Drilling and Blasting

Four (4) different drilling patterns will be used depending on the hardness of the rock and the destination of the material (mineralized material or waste). Single pass drills will be chosen to ensure optimal productivity.

Table 16.11 describes all the different parameters considering materials and destination. Saprolite will be drilled for grade control purposes only. Nevertheless, 25% of mined saprolite will be assumed to have been blasted, as the limit between saprolite and transition is not easily defined. The saprolite and transition will be drilled using 6.75 inches holes and rotary drills. For the transition, a subdrill of 2.0 m will be added to maintain good pit floors.

For fresh rock, two (2) different patterns are planned. The DTH drilling method will be the most efficient method on this harder rock with 8.5 inches holes for waste and 6.75 inches holes for mineralized materials. A subdrill of 1.5 m will be considered. For waste material, a wider drilling pattern will be used, with 5.5 m burden and 6.25 m spacing. For mineralized material, to optimize feed fragmentation to the crusher, a smaller pattern will be used, with 4.8 m burden and 5.5 m spacing. Powder factors will be 0.32 kg/t in mineralized material and 0.31 kg/t in waste, respectively.

Table 16.11: Drill and Blast Parameters

Drill & Blast Parameters		Mineralized Material Fresh Rock	Waste Fresh Rock	Saprolite (25%)	Transition
Drill Type		**DTH**	**DTH**	**Rotary**	**Rotary**
Explosive Density	g/cm^3	1.20	1.20	1.20	1.20
Hole Diameter	in	6.75	8.50	6.75	6.75
Diameter (D)	m	0.171	0.216	0.171	0.171
Burden (B)	m	4.80	5.50	6.00	6.00
Spacing (S)	m	5.50	6.25	7.00	7.00
Subdrill (J)	m	1.50	1.50	-	2.00
Stemming (T)	m	4.00	4.70	4.00	4.00
Bench Height (H)	m	10.0	10.0	10.0	10.0
Blasthole Length (L)	m	12.53	11.5	10	12
Pattern Yield					

Drill & Blast Parameters		Mineralized Material Fresh Rock	Waste Fresh Rock	Saprolite (25%)	Transition
Rock Density	t/bcm	2.77	2.77	1.90	2.20
BCM / Hole	bcm/hole	264	344	420	420
Yield per Hole	t/hole	731	952	798	924
Yield per Metre Drilled	t/m drilled	58	83	80	77
Explosive Column (LE)	m	8.53	6.80	6	8
Volume of Explosives / Hole	m^3	0.20	0.25	0.14	0.18
Weight of Explosives / Hole	kg	235.18	298.73	166.23	221.63
Powder Factor	kg/t	0.32	0.31	0.21	0.24
Powder Factor	kg/bcm	0.89	0.87	0.40	0.53
Drill Productivity					
Re-Drills	%	5.0%	5.0%	5.0%	5.0%
Pure Penetration Rate	m/min	1.15	1.15	3.00	2.1
Pure Penetration Rate	m/hr	69.0	69.0	180.0	124.5
Overall Drilling Factor (%)	%	40.00%	40.00%	50.00%	50.00%
Overall Penetration Rate	m/hr	27.6	27.6	90.0	62.3
Drilling Efficiency	t/hr	1 610	2 285	7 182	4 793
Drilling Efficiency	holes/hr	2.20	2.40	9.00	5.19

Most of the blast holes will be initiated with NONEL detonators paired with two (2) prime boosters of 450 g.

According to our primary studies, it can be expected to have a better drilling production rate with rotary drills for saprolite and transition material, with 90 and 62.3 m/engine hour, respectively. For fresh rock it is expected to have better productivity with DTH drills, with 27.6 m/engine hour.

Controlled blasting techniques will be used including buffer blasts and pre-splits. The pre-split consists of closely spaced holes along the designed excavation limit. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. A pre-split drill rig was selected for this application.

Explosives will be supplied by a third-party provider who will be responsible for supplying and delivering explosives into emulsion trucks. An owner-operated blasting team will oversee the loading and blasting activities. The mine engineering department will be responsible for designing blast patterns.

16.4.10.2 Loading

The loading fleet consists of four (4) 22 m^3 diesel hydraulic shovels and one (1) 12.2 m^3 diesel front-end loader. The 22 m^3 shovel will be utilized for both waste and mineralized material loading, while the front-end loader will be primarily used for rehandling the mineralized material stockpile and to supplement mineralized material loading as needed. Table 16.12 presents productivity assumption for the loading fleet. The number of units is represented in whole numbers for the purpose of these productivity assumptions.

Table 16.12: Loading Fleet Productivity Assumptions

		Waste Rock 1	Waste Rock 2	Waste Rock 3	MM Rock 1	MM Rock 1	MM Rock 2	MM Rock 2	MM Rock 3	MM Rock 3
Loading Unit		**22 m³ Shovel SAP**	**22 m³ Shovel TRAN**	**22 m³ Shovel FRESH**	**22 m³ Shovel SAP**	**12 m³ Loader SAP**	**22 m³ Shovel TRAN**	**12 m³ Loader TRAN**	**22 m³ Shovel FRESH**	**12 m³ Loader FRESH**
Haulage Unit		**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**	**139-t Truck**
Rated Truck Payload	t	139	139	139	139	139	139	139	139	139
Heaped Tray Volume	m³	85	85	85	85	85	85	85	85	85
Bucket Capacity	m³	24.0	24.0	20.0	24.0	12.9	24.0	12.9	20.0	12.9
Bucket Fill Factor	%	87%	84%	83%	87%	90%	84%	88%	83%	85%
Bucket Weight	t	45.60	45.60	38.00	45.60	24.51	45.60	24.51	38.00	24.51
In-situ Dry Density	t/bcm	1.90	2.20	2.77	1.90	1.90	2.20	2.20	2.77	2.77
Moisture	%	8%	5%	3%	8%	8%	5%	5%	3%	3%
Swell	%	35%	40%	40%	35%	35%	40%	40%	40%	40%
Wet Loose Density	t/lcm	1.52	1.65	2.04	1.52	1.52	1.65	1.65	2.04	2.04
Actual Load per Bucket	t	31.74	33.26	33.83	31.74	17.65	33.26	18.73	33.83	22.35
Passes (decimal)	#	4.38	4.18	4.11	3.54	6.59	3.54	7.42	4.11	6.22
Passes (whole)	#	4.00	4.00	4.00	4.00	7.00	4.00	7.00	4.00	6.00
Actual Truck Wet Payload	t	127	133	135	127	124	133	131	135	134
Actual Truck Dry Payload	t	118	127	131	118	114	127	125	131	130
Actual Heaped Volume	m³	84	81	66	84	81	81	79	66	66
Payload Capacity	%	91%	96%	97%	91%	89%	96%	94%	97%	96%
Heaped Capacity	%	98%	95%	78%	98%	96%	95%	93%	78%	77%
Cycle Time										
Hauler Exchange	min	0.60	0.60	0.60	0.60	0.70	0.60	0.70	0.60	0.70
First Bucket Dump	min	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.56	0.60	0.64	0.56	0.80	0.60	0.80	0.64	0.80
Load Time	min	2.38	2.50	2.62	2.38	5.60	2.50	5.60	2.62	4.80
Cycle Efficiency with Wait Time	%	75%	75%	75%	75%	75%	75%	75%	75%	75%
Number of Trucks Loaded per Hr	#	18.91	18.00	17.18	18.91	8.04	18.00	8.04	17.18	9.38
Production / Productivity										
Productivity Dry Tonnes / Op. Hr	t/h	2,223	2,281	2,256	2,223	919	2,281	1,003	2,256	1,220
Effective Hours per Year	h/y	5,214	5,214	5,214	5,214	5,540	5,214	5,540	5,214	5,540
Dry Annual Production Capacity	kt/y/unit	11,588,758	11,893,473	11,765,816	11,588,758	5,092,072	11,893,473	5,559,164	11,765,816	6,760,874

16.4.10.3 Hauling

Haulage will be performed by 139 t class off-highway mining trucks for waste and mineralized material. The mineralized material will be hauled to the crusher located outside of the pit, while the waste will be hauled to the waste storage facilities as shown in Figure 16.10.

The truck requirements have been calculated in Deswik.LHS (Landform and Haulage) software. This software links the mining schedule to the material movements and determines optimal haulage routes and simulates them using Rimpull data from the fleet. The following assumptions were used when running the simulations.

- Max site speed limit of 50 km/h on roads and 30 km/h on bench.
- Max speed loaded and downhill of 30 km/h.
- Average rolling resistance of 2%.
- Time for loading and spotting time is variable depending on the loader, truck, and material. This value ranges from 2.4 – 5.6 minutes (see Table 16.12).

Figure 16.20 depicts the average cycle time for mineralized material and waste over time. Note that cycle time increases as pits get deeper due to increased uphill haulage required. Plateaus or dips in the cycle time represent transitions to new pushbacks starting from the surface, temporarily reducing cycle time. The cycle times presented do not include loading times.

Figure 16.21 depicts the total truck requirements over the years. Twenty-four (24) 139 t trucks will be required to maintain production at peak mining rate. This hauling fleet is expected to be sufficient for the life of the mine.

Figure 16.20: Average Cycle Times by Material Type

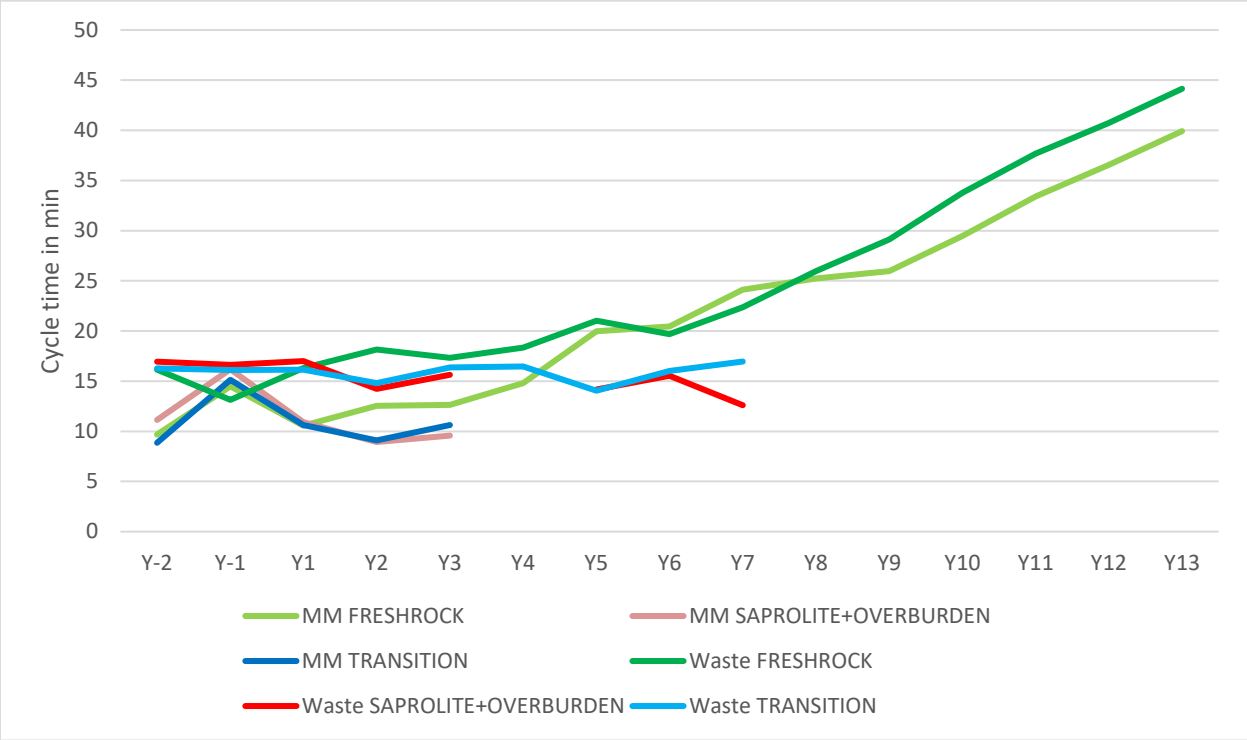
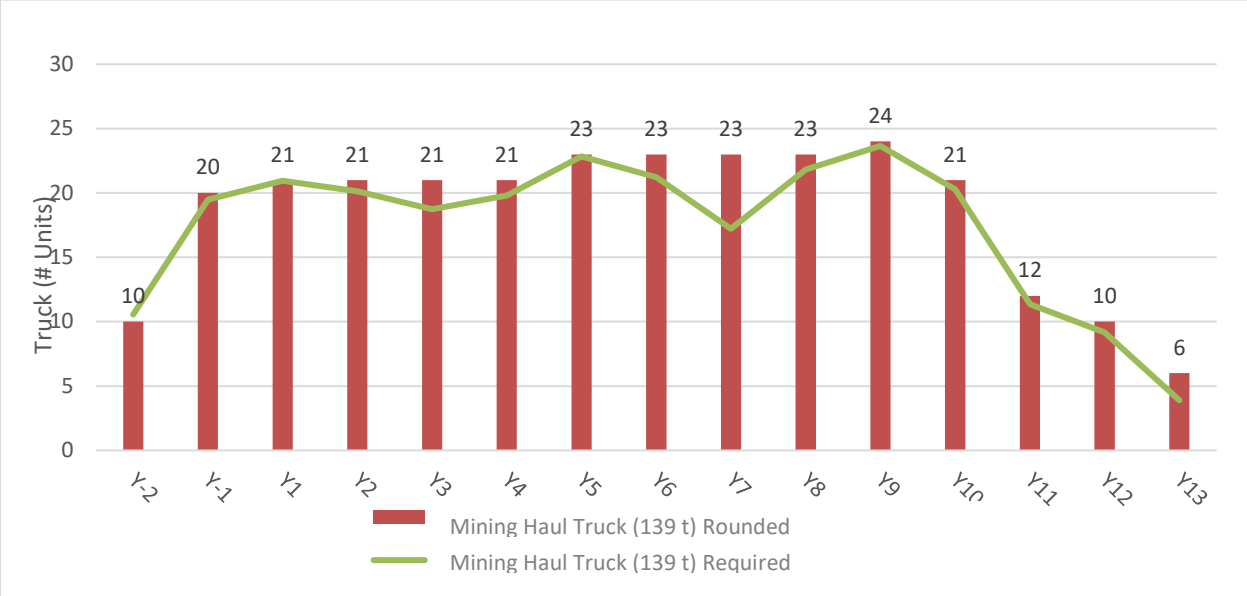


Figure 16.21: Truck Requirements



16.4.10.4 Support Operations

Support equipment requirements are based on typical open pit mine operation and maintenance requirements to safely support the loading, hauling, and drilling fleets.

Support equipment is planned for maintaining dump areas, stockpiles, pit floors, ditches and mine roads. The fleet of support equipment consist of the following:

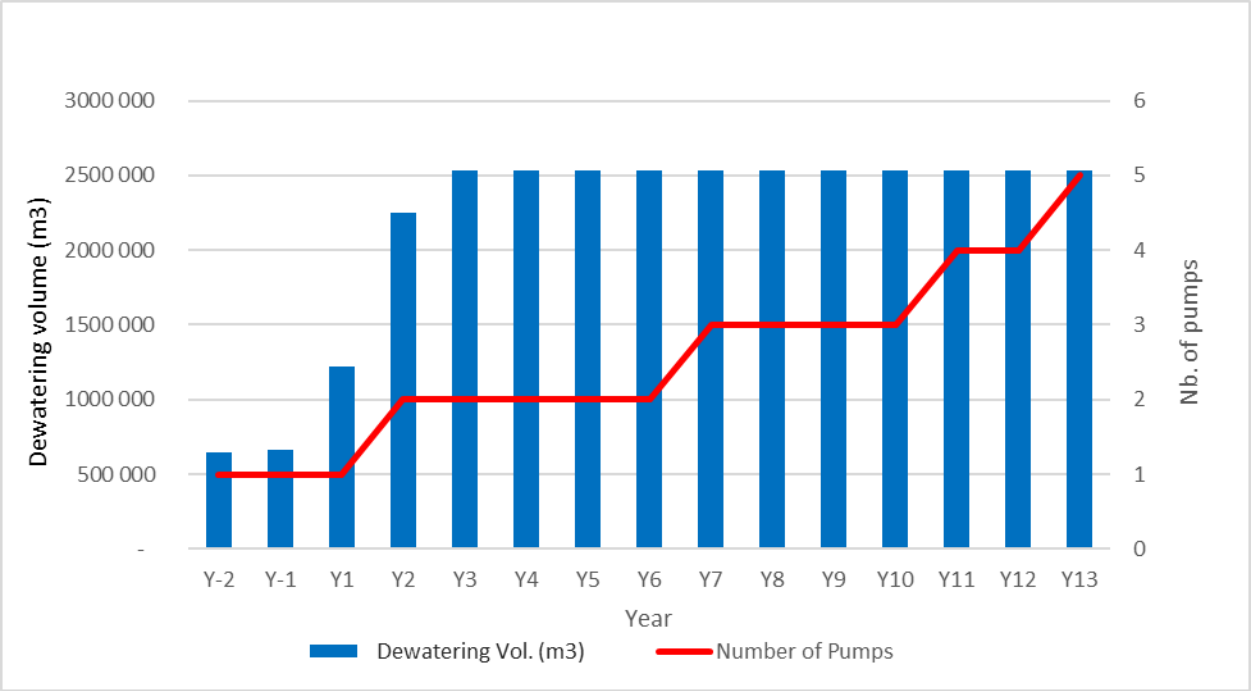
- 5 x 600 HP dozers for dump maintenance
- 4 x 16 ft blade motor graders for road upkeep
- 1 x water / sand trucks for dust suppression
- 2 x 496 HP wheel dozers

16.4.10.5 Mine Dewatering

Minimal initial dewatering for the open pit mining is expected and will be performed with six (6) inches diesel pumps.

Dewatering during the operations will reach a maximum of 2.5 Mm^3 per year of water requiring up to five (5) umps in series. Dewatering volumes and quantity of pumps over time are presented in Figure 16.22.

Figure 16.22: Dewatering Volumes and Quantity of Pumps Over Time



16.4.11 Mining Fleet Requirements

Table 16.13 summarizes the gross operating hours considered for calculating equipment fleet requirements. The mine is expected to operate 22 hours per day, 355 days per year. This accounts for shift changes and 10 days of delay related to weather. Additional delays and applied factors are described in productivity calculations for each fleet as calculated in Table 16.13Table 16.12.

Additional equipment will be procured to facilitate the maintenance activities and support the operation, such as fuel and lube trucks, a forklift, a telehandler, a low-boy trailer and a tractor for moving the tracked equipment. Other small equipment such as mechanic service trucks, generators and welding machines are also included.

Table 16.14 and Table 16.15 present the equipment purchase schedule for the LOM.

Table 16.13: Equipment Usage Assumption

		Shovels	Loaders	Trucks	Drills	Pumps
Days in Period	days	365	365	365	365	365
Weather, Schedule Outages	days	10.0	10.0	10.0	10.0	10.0
Shifts per Day	shift/day	2.0	2.0	2.0	2.0	2.0
Hours per Shift	h/shift	12.0	12.0	12.0	12.0	12.0
Availability	%	85.0	85.0	85.0	85.0	90.0
Use of Availability	%	90.0	90.0	90.0	85.0	95.0
Utilization	%	76.5	76.5	76.5	72.25	85.5
Effectiveness	%	80.0	85.0	87.0	85.0	90.0
OEE	%	61.2	65.0	66.6	61.4	77.0
Total Hours	hours	8,760	8,760	8,760	8,760	8,760
Scheduled Hours	hours	8,520	8,520	8,520	8,520	8,520
Down Hours	hours	1,278	1,278	1,278	1,278	852
Delay Hours	hours	1,304	978	847	923	728
Standby Hours	hours	724	724	724	1,086	383
Operating Hours	hours	6,518	6,518	6,518	6,156	7,285
Ready Hours	hours	5,214	5,540	5,670	5,232	6,556

Table 16.14: Major Equipment Purchase Schedule

Major Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Production Drill (6-10")	5	1	0	0	2	1	0	1	0	0	0	0	0	0	0	0
Auxiliary Pre-split Drill (4.5-8")	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Diesel Hydraulic Excavator (22 m^3)	4	2	0	1	1	0	0	0	0	0	0	0	0	0	0	0
Wheel Loader (12.2 m^3)	1	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0
Auxiliary Pre-split Drill (4.5-8")	1	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0
Mining Haul Truck (150 t)	24	7	3	10	1	0	0	2	0	0	0	1	0	0	0	0
Track Dozer (600 HP)	7	1	0	3	1	0	0	0	0	0	1	1	0	0	0	0
Motor Grader (16 ft)	8	1	0	2	1	0	0	0	0	2	2	0	0	0	0	0
Water / Sand Truck (76 kL tank)	2	1	0	0	0	0	0	0	0	0	0	0	1	0	0	0
Wheel Dozer (496 HP)	3	0	0	2	0	0	0	0	0	0	0	1	0	0	0	0

Table 16.15: Support Equipment Purchase Schedule

Major Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Emulsion Truck	6	0	0	1	2	0	0	0	1	2	0	0	0	0	0	0
Stemming Loader	3	1	0	0	0	1	0	0	1	0	0	0	0	0	0	0
Articulated Dump Truck (45 t)	6	3	0	0	0	0	0	0	0	0	0	3	0	0	0	0
Excavator (49 t)	2	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Excavator (90 t)	2	1	0	0	1	0	0	0	0	0	0	0	0	0	0	0
Hydraulic Hammers for Excavator 49 t	6	1	0	1	0	0	0	1	1	0	0	0	1	1	0	0
Wheel Loader 311 HP	2	0	0	1	0	0	0	0	0	0	1	0	0	0	0	0
Telehandler	4	0	0	2	0	0	0	0	0	0	0	2	0	0	0	0
Mechanic Service Truck	7	1	0	3	0	0	0	0	0	0	1	1	1	0	0	0
Tire Handler Truck	2	0	0	1	0	0	0	0	0	0	0	0	1	0	0	0
Mining Haul Truck (240 t)	1	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0
Dollie	4	0	0	0	4	0	0	0	0	0	0	0	0	0	0	0
Fuel & Lube Truck (7.5 kL) ADT	4	1	0	1	0	0	0	0	2	0	0	0	0	0	0	0
Tow Haul Truck 150 t	1	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0
Trailer Lowboy 150 t	1	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0
Multipurpose Truck - Vacuum Tank	3	0	0	1	0	0	0	1	0	0	0	0	1	0	0	0
Pick-up Truck	64	8	0	16	0	0	0	24	0	0	0	0	16	0	0	0
Pit Bus	7	1	0	1	1	0	0	0	0	1	3	0	0	0	0	0
Welding Machine Diesel 400 A	2	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0
Crane Rough Terrain 130 t	1	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0
Water Pump 10 in - Diesel	14	1	0	0	0	1	1	1	0	2	1	0	2	2	0	3
Water Pump - Decantation	2	0	0	1	0	0	0	0	0	0	0	1	0	0	0	0
10" Pipe – 230 psi (m)	1993	0	1024	0	100	375	454	40	0	0	0	0	0	0	0	0
Wheel Loader 425 HP	3	1	0	0	0	0	1	0	1	0	0	0	0	0	0	0

16.4.12 Mobile Crushing Plant

The production of crushed material will be necessary for blasthole stemming purposes and for road maintenance. It is assumed that the required aggregate material production will occur internally with a mobile crusher on site. Waste rock to feed the small crushing plant will come from pit pre-production.

Additionally, the UG backfill will require waste material from the OP mine. This material will be crushed to minus 6 inches and sent UG with trucks. A loader will be added to the fleet to meet the 1 Mtpa fill requirement starting in Year 5.

16.4.13 Mine Maintenance

Maintenance will be performed by the owner's personnel. The maintenance department and personnel requirement has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support will be key for the initial years of the project and major component rebuilds will be supported by the equipment dealer throughout the LOM. Tire monitoring, rotation and/or replacement will also be conducted internally.

16.4.14 Mine Management & Technical Services

The operation team is responsible for achieving production targets in a safe and efficient manner. The engineering and geology team will support the operations team by providing short-term and long-term planning, grade control, surveying, mining resources estimation and other technical functions. The mine dispatch system is included in this PEA and will be managed by the mine operation team.

16.5 Underground Mining

16.5.1 Underground Mining Method

The selected underground mining method is the long hole open stoping (LHOS) mining method with either transverse or longitudinal stoping. The stoping sequence will be ascending from an initial undercut. Generally, stopes will be mined using an upper access for drilling and a lower access for mucking. Stopes within the sill pillars will be mined exclusively from the lower access level, which will be utilized for both drilling and mucking operations.

LHOS is a commonly used underground mining method for competent hard rock orebodies. The first phase of LHOS is the mine development phase. During this phase, the following excavations are developed:

decline, level access, haulage drifts, draw point and other infrastructures. These excavations provide access to the stoping area and support the production activities. The development phase also includes the development of both an overcut drift and an undercut drift.

The overcut drift is developed to allow production drilling of the stoping area, while the undercut allows for ore extraction from the stope. Once the development of the drilling drift and extraction drift is completed, a slot raise, typically raise bored or conventionally drilled and blasted, is created to provide an initial void for production drilling and blasting activities.

The production drilling phase involves drilling long vertical or inclined holes, at regular intervals along the length and width of the stoping area. The blasting phase begins by loading the production drill holes with bulk explosives and stemming once production drilling is completed.

During the blasting phase, the controlled use of explosives allows to fracture the rock surrounding the slot raise. Once the rock is blasted and the blasting gases cleared from the mine, the fragmented material is removed from the stope through the undercut drift with a load-haul-dump (LHD) unit. Depending on the size and geometry of the stopes, it can take up to 3 or 4 blasting and mucking cycles to extract all the material from the stope. The broken material from the stopes is loaded into haul trucks and then transported to surface.

LHOS is a non-entry mining method, since the stoping area is not accessible to personnel once production begins. The use of some remote-controlled load-haul-dump (LHD) units is required to completely remove the blasted material from the stope. The final phase of LHOS is the backfilling phase. Depending on multiple factors like the variant of LHOS used and the mining sequence, stopes can be filled with cemented material, uncemented material or a combination of both. Cemented backfill can include paste fill, hydraulic fill and cemented rockfill (CRF) while uncemented backfill is typically rockfill.

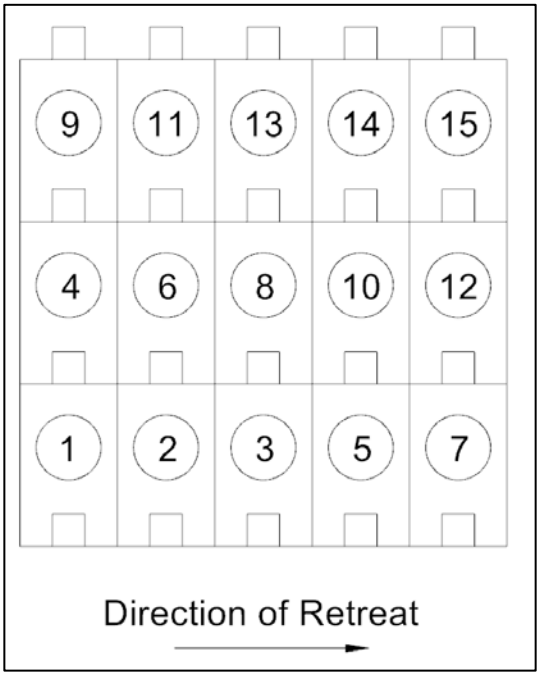
This non-entry mining method offers several advantages including, but not limited to high productivity, high operational flexibility, low operating costs, efficient mineralized material extraction and improved workers safety.

The longitudinal variant of LHOS is used for the following situations:

- Stoping thickness is narrower than 8 m.
- Adjacent to longitudinal stopes where transverse LHOS is not practical.

The longitudinal mining areas are accessed by developing the overcut and undercut ore drifts inside the stoping area along the strike of the orebody. Once the development is completed to the extremity of the longitudinal mining area, the production cycle of the initial stope can start. Subsequent stopes will be mined in the same cycle while retreating towards the main access located either at an extremity or near the middle of the orebody. Figure 16.23 illustrates the typical longitudinal mining sequence.

Figure 16.23: Typical Longitudinal Stoping Sequence



Source: GMS 06-06-2024 (not to scale)

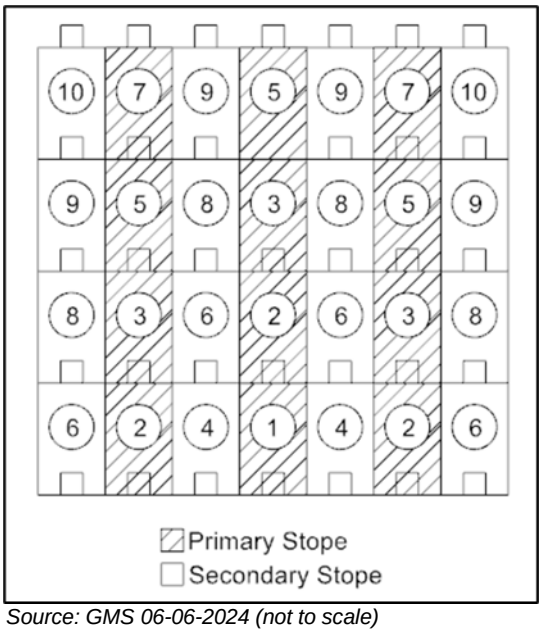
The transverse variant of LHOS is used for the following situations:

- Stoping area is wider than 8 m.
- Adjacent to transverse LHOS where longitudinal LHOS is not practical.

The transverse mining area will be accessed by developing a haulage drift parallel to the strike of the orebody. This haulage drift shall maintain a reasonable distance from the stoping area to preserve its integrity during the production phase. Perpendicular to the haulage drift, a series of parallel draw points evenly spaced along the haulage drift, will be used to access the undercut and overcut ore drifts developed over the full width of the orebody. Once the haulage drift, the draw points and the ore drifts are developed, the production cycle of the initial primary stope can begin. Subsequent primary stopes are mined in the exact same cycle while retreating towards the extremities of the orebody. Stopes will be sequenced in an overhand approach so that when two (2) lifts of primary stopes are mined, the secondary stope between these two (2) primaries can be mined on the first lift. This primary-secondary transverse mining approach

is advantageous in terms of production and operational flexibility as multiple stopes can be in operation simultaneously. One disadvantage of the method is that a haulage drift must be excavated along the entire length of the mineralized zone. However, the production cycle can still begin even if the haulage drift is not completely excavated to the extremities of the orebody. Figure 16.24 illustrates the typical sequence of transverse stoping.

Figure 16.24: Typical Transverse Stoping Sequence



Source: GMS 06-06-2024 (not to scale)

16.5.2 Cut-Off Grade

The cut-off grade (COG) is the minimum mineral or metal content in the mineralized material for its extraction and processing to be profitable. It represents the grade at which the costs of extraction, processing, and marketing would be equal to the revenues derived from the selling price of the valued commodity. To evaluate the Potentially Economical Portion of the Mineral Resource Estimate, a cut-off grade was calculated for the selected mining method (LHOS). Table 16.16 identifies the parameters used to estimate the mine's cut-off grade and Table 16.17 identifies the detailed mining operating cost (OPEX) estimation for underground mining at Oko West.

Table 16.16: Underground Mine Cut-offs Calculation Parameters

Parameters	Units	Value
Gold Price	*$/oz*	1,750
Exchange Rate	*CAD/USD*	1.33
	GYD/USD	208
Discount Rate	*%*	5.0
Royalty Rate	*%*	8.0
TC & RC		
Transportation	*$/oz*	8.00
Refining	*$/oz*	Incl.
Payables		
Payable Metal Au	*%*	99.95
Mill Recovery		
Average Mill Au Recovery	*%*	92.5
Mineralized Material Based Costs		
Mine Operating Cost - LHOS	*$/t milled*	55.00
Processing Cost (Fresh Rock)	*$/t milled*	14.51
General & Administration Cost	*$/t milled*	10.43
Sustaining Capital	*$/t milled*	5.00
Cut-Offs		
Cut-Off Value (No Sustaining Capital)	***$/t milled***	**79.94**
Cut-Off Grade (No Sustaining Capital)	***g/t***	**1.75**

The cut-off value without sustaining capital was used for the stope optimization process. A zone-by-zone approach was used to validate the economical viability of the various zones by including their specific sustaining capital cost in the economical analysis.

Table 16.17: Underground Mine Detailed LHOS Mining Cost

Parameters	Units	Value
Definition Drilling & Geology	$/t	2.80
Stope Preparation	$/t	5.10
Drilling & Blasting	$/t	6.25
Mucking & Hauling	$/t	9.80
Backfilling	$/t	6.20
Mine Supervision	$/t	4.80
Technical Services	$/t	4.15
Mine Services	$/t	9.00
Maintenance Services	$/t	6.00
Surface Services	$/t	0.90
Total Mining Cost – LHOS	$/t	55.0

16.5.3 Potentially Economical Portion of the Mineral Resource Estimate

The Mineral Resource block model was provided by GMS and was imported to Deswik CAD™ software as a singular block model. The model provided was a 5 m x 5 m x 5 m parent block model and sub-blocked to a minimum of 2.5 m x 0.5 m x 2.5 m block size. The evaluation of the Potentially Economical Portion of the Mineral Resource Estimate in the Oko West PEA includes all categories of Mineral Resources: Measured, Indicated, and Inferred.

A PEA is preliminary in nature and is intended to provide only an initial high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are too geologically speculative to apply the economic considerations that would allow them to be categorized as mineral reserves and used in economic analysis, except as permitted in PEA studies. There is no guarantee that Inferred resources can be converted to Indicated or Measured Mineral Resources, and as such, there is no guarantee the Project economics described herein will be achieved.

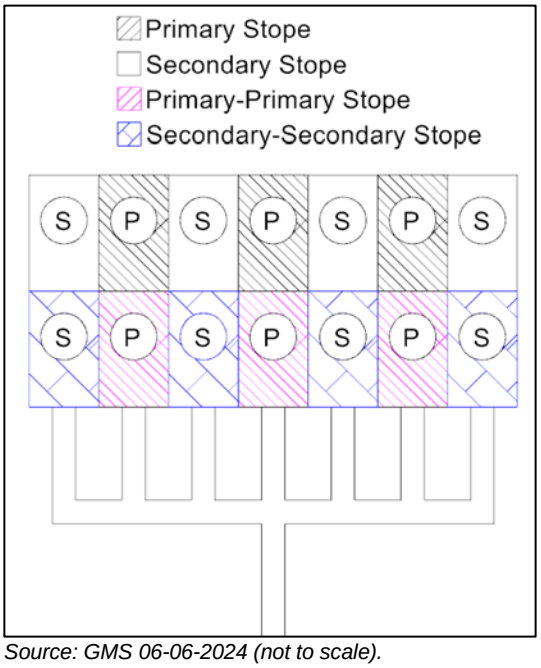
Dilution parameters were assigned to each stope to estimate the additional dilution experienced during mining operations. A 1.0 m and 0.5 m Equivalent Linear Overbreak Slough (ELOS) was applied to the stope hanging wall and footwall, respectively. To reflect dilution caused by backfill, an additional dilution factor along with a mining recovery of 95% were applied after the stope optimization process. Table 16.18 shows

the various backfill dilution factors used depending on the stope type while Figure 16.25 illustrates the different stope types for transverse stoping to reflect the multiple possible situations.

Table 16.18: Underground Mine Backfill Dilution Parameters

Parameters	Units	Value
Sublevel Transverse Stoping		
Primary Stope	%	2.0
Secondary Stope	%	11.0
Primary - Primary Stope	%	5.0
Secondary - Secondary Stope	%	15.0
Sublevel Longitudinal Stoping		
Primary Stope	%	7.0

Figure 16.25: Underground Mine Transverse Stope Types



Source: GMS 06-06-2024 (not to scale).

A series of iterations with the stope optimizer tool of Deswik™ software were performed to obtain the best possible stope shapes. The stope geometry and cut-off grade parameters used in the stope optimizer tool are summarized in Table 16.19.

Table 16.19: Underground Mine Stope Optimizer Parameters

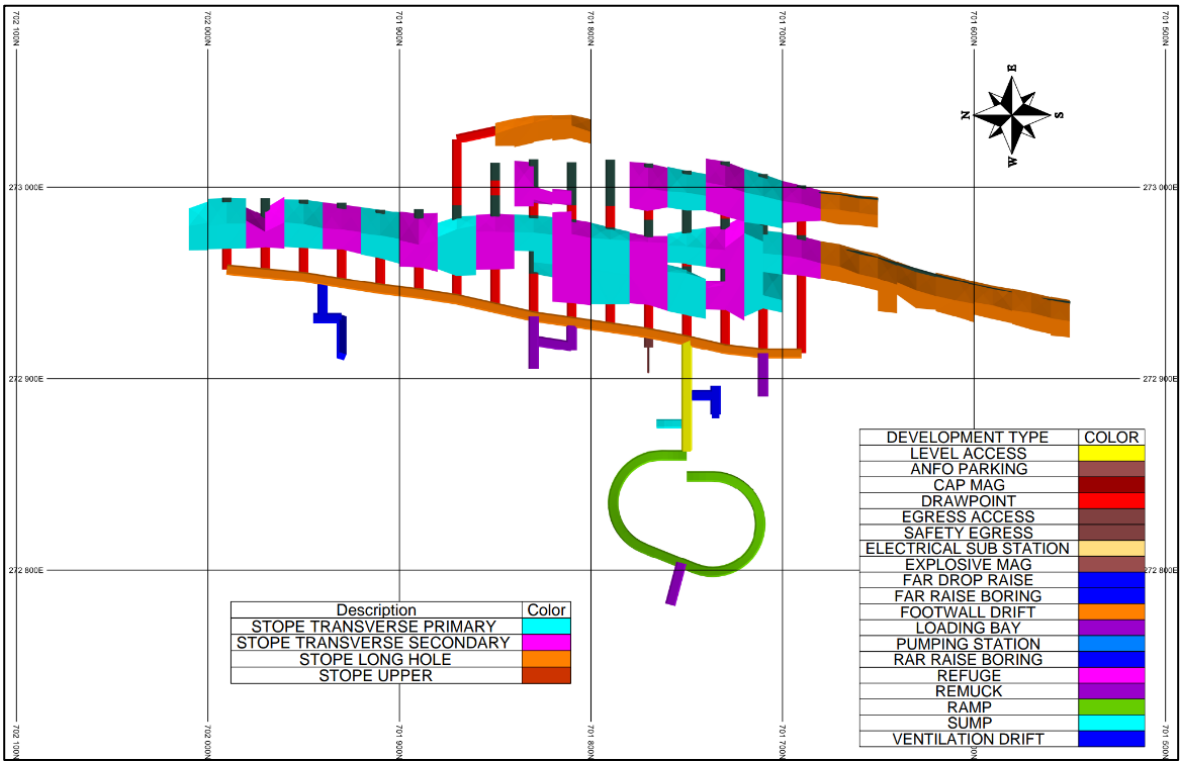
Parameters	Units	Value
Deswik Stope Optimizer Parameters		
Stope Height	m	30
Strike Length	m	20
Maximum Mining Width (HW to FW)	m	25
Minimum Mining Width (HW to FW)	m	5.0
HW Dilution	m	1.0
FW Dilution	m	0.5
Minimum Dip	°	50
Crown Pillar Thickness	m	30
Side Ratio (Top-Bottom)	-	2.25
Side Ratio (Front-Back)	-	2.25
Cut-Off Grade	g/t	1.75

16.5.4 Underground Mine Design

16.5.4.1 Development Design

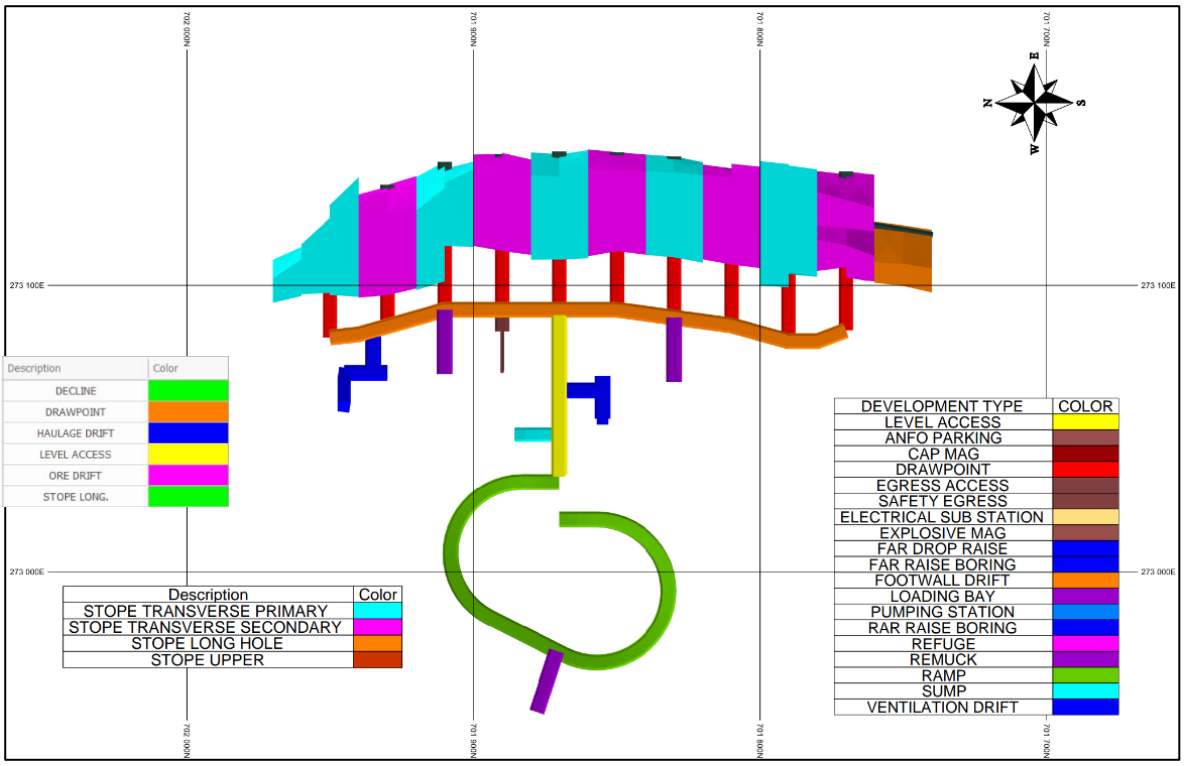
The Oko West UG mine will be accessed by a single decline with a portal located inside an initial phase of the open pit. Each production level will be accessed by a level access and a haulage drift that leads to the crosscuts that are either driven perpendicular to the ore body for transverse stoping or within the longitudinal direction of the orebody for longitudinal stoping. For a typical level access, the following infrastructures must be included: sump, electrical bay, remuck, loading bay, fresh air access, return air access and a safety egress. Figure 16.26 and Figure 16.27 show plan views of a typical underground mine production levels while Figure 16.28 and Figure 16.29 show a longitudinal and a section view of the lateral and vertical development of the mine.

Figure 16.26: Typical Larger Level Plan View



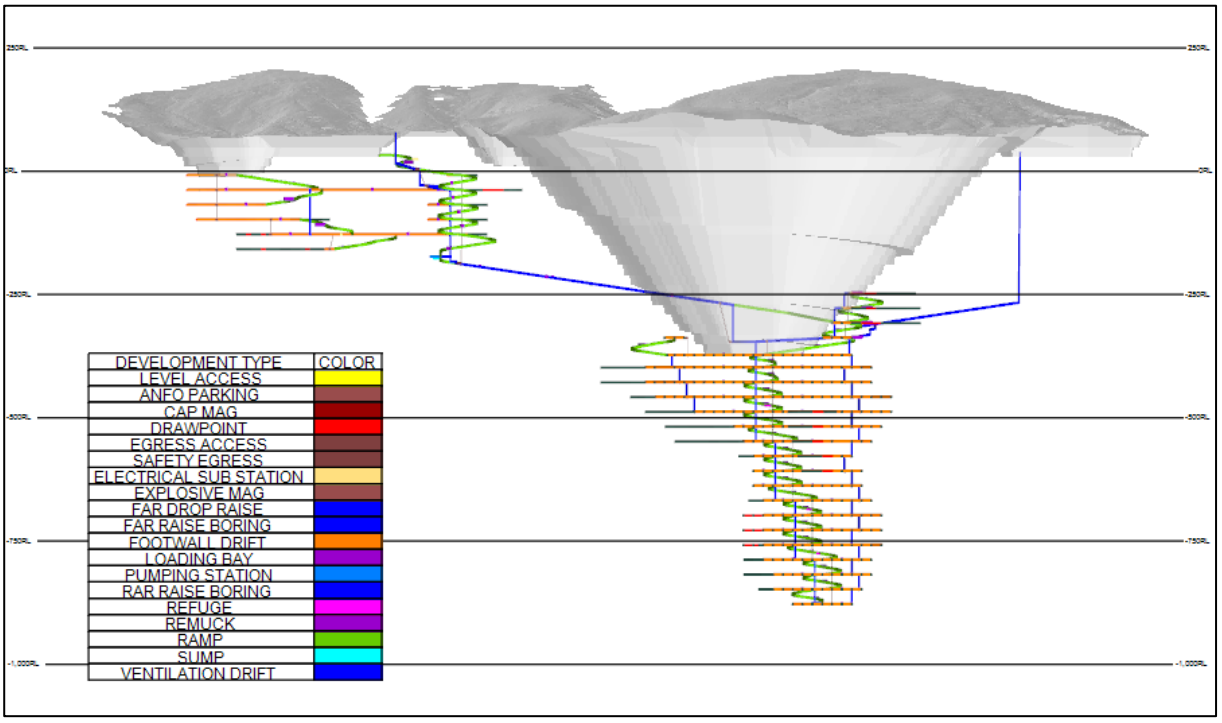
Source: GMS 06-06-2024 (not to scale)

Figure 16.27: Typical Smaller Level Plan View



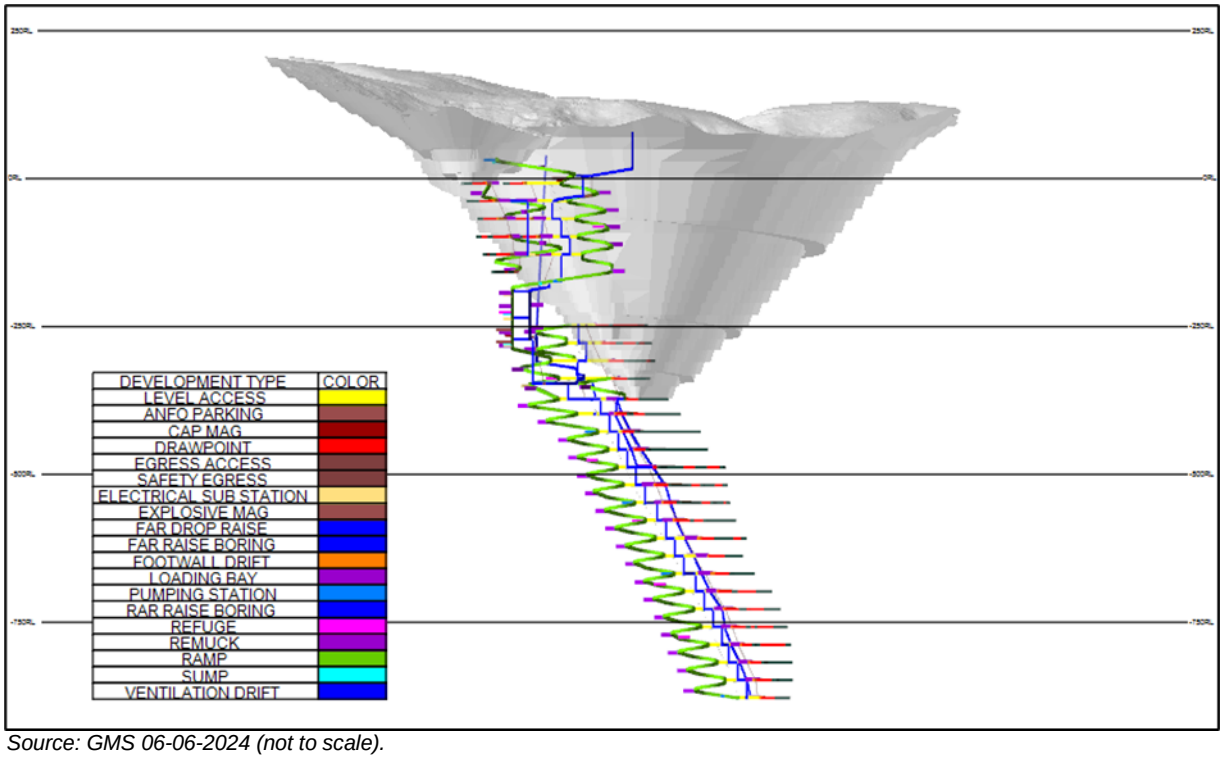
Source: GMS 06-06-2024 (not to scale)

Figure 16.28: Mine Development Longitudinal View - Looking West



Source: GMS 06-06-2024 (not to scale).

Figure 16.29: Underground Mine Development Longitudinal View - Looking North



Source: GMS 06-06-2024 (not to scale).

Table 16.20 lists some of the most relevant parameters used for the mine design, while

Table 16.21 lists the various development factors applied to the mine design. Table 16.22 lists the different development design criteria present in the mine and their respective dimensions.

Table 16.20: Underground Mine Development Parameters

Item	Measurements / Specifications
Ramp & Lateral Development	
Ramp Gradient	Nominal +/- 5%
Ramp Gradient at Intersections	Reduced to 10%, then 5% & Flat at Intersections
Turn Radius (Ramp)	25 m
Ramp - Orebody Offset (minimum)	60 m
Footwall Drift - Orebody Offset (minimum)	20 m

Table 16.21: Underground Mine Development Factors

Parameters	Units	Value
Development Allowances		
Ramp (e.g., safety bays, TDB for fans, intersection slashes, cut-outs for pump boxes)	%	5
Lateral Development (Capex) (e.g., safety bays, TDB for fans, intersection slashes)	%	5
Lateral Development (Opex) (e.g., TDB for fans, cutouts for remote mucking)	%	2.5
Infrastructure (e.g., slashes, TDB for fans and services)	%	2.5
Sills (Ore Drift)	%	2.5
Raises	%	0
Overbreak		
Development in Waste	%	10
Development in Mineralized Material	%	10
Raises	%	0

Table 16.22: Underground Mine Development Type and Dimensions

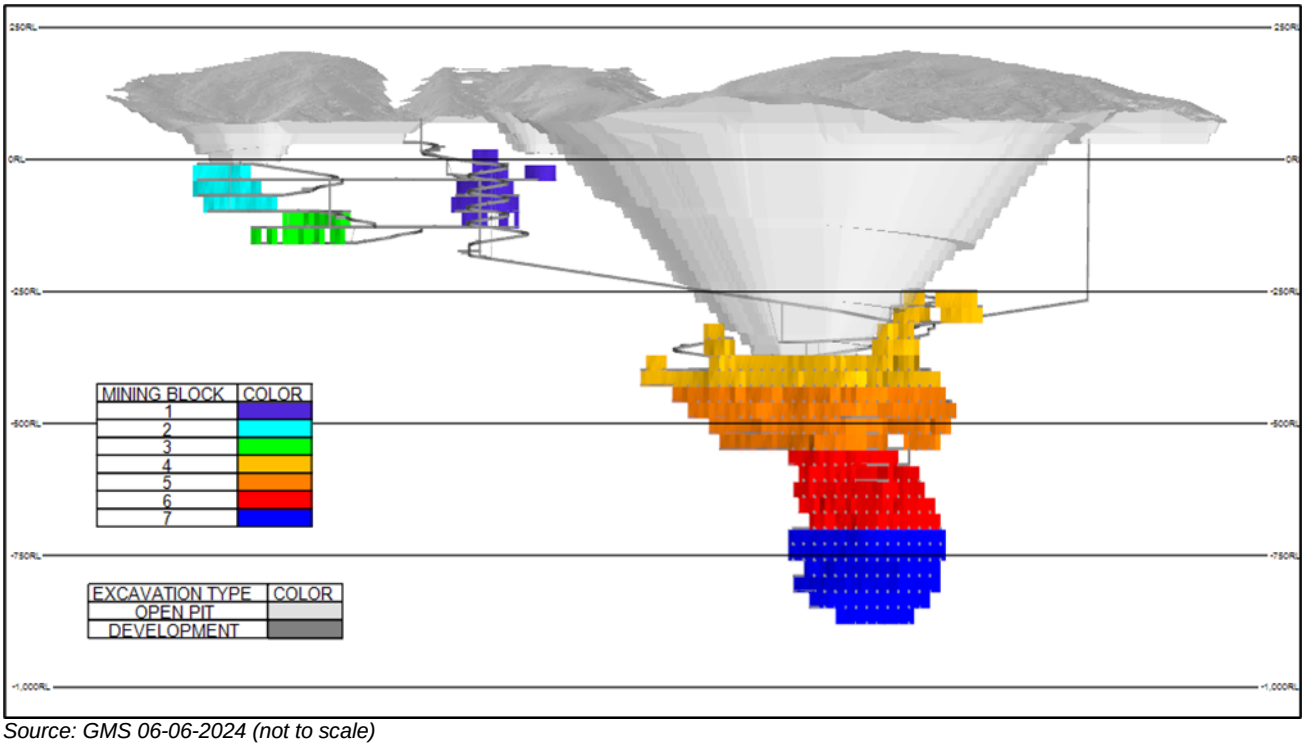
Development Type	Width (m)	Height (m)	Length (m)
CAPEX Development			
Decline	5.5	6.0	Variable
Level Access	5.5	6.0	Variable
Haulage Drift	5.5	6.0	Variable
Remuck	5.5	7.5	20
Safety Bay	1.5	2.0	2.0
Sump	5.0	5.0 (Flat Back)	12.5
Settlement Bay	5.0	5.0 (Flat Back)	18.0
Ventilation Drift	5.5	6.0	Variable
Ventilation Raise Access	5.5	5.5	Variable
Refuge	5.0	5.0	20.0
Gear Bay	5.5	7.5	20.0
Explosive Mag	8.0	5.5	25.0
Cap Mag	6.5	5.0	10.0
Electrical Sub Station	5.0	5.0	12.5
Safety Egress Access	5.0	5.0	Variable
Loading Bay (LHD)	5.5	5.5	11.5
Loading Bay (TRUCK)	5.5	7.5	16.0
Pumping Station	6.5	5.0	20.0
ANFO Parking	8.0	5.5	22.5
Fuel & Lube Bay	5.5 to 9.5	6.0	20.0
Wash Bay (Garage)	7.0	6.5	20.0
Maintenance Bay (Garage)	8.0	6.5	20.0
Tool Crib (Garage)	5.5	6.5	10.0
Electric Cut-out (Garage)	5.0	5.0	3.5
Office Bay (Garage)	5.0	5.0	3.5
Welding Bay (Garage)	5.0	5.0	3.5
OPEX Development			
Drawpoint	5.0	5.0	20
Ore Drift	5.0	5.0	Variable

16.5.4.2 Stope Design

The Oko West orebody is divided into three (3) distinct mining zones, which includes the main zone, located directly under the open pit, and two (2) smaller satellites zones located south of the main zone. The underground mine has a total of 34 production levels. To increase operational flexibility and achieve the production target, these zones were divided in several mining blocks. Figure 16.30 and Figure 16.31 show the seven (7) different mining blocks of the underground mine.

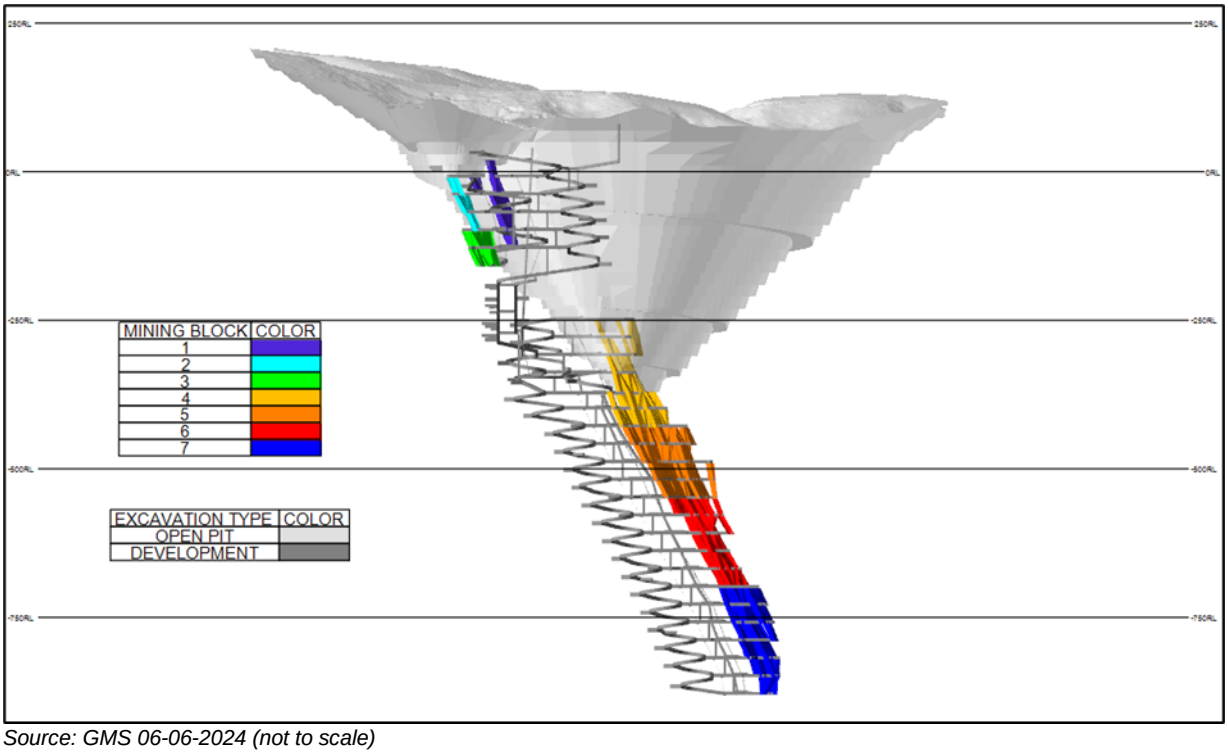
Development and production activities will occur simultaneously in two (2) or three (3) mining blocks to ensure that multiple development headings and stopes are available. The orebody is sub-vertical, with the stope thickness averaging 16.5 m and stope tonnage averaging 26,000 t. This is strongly reflected in the fact that 93% of the orebody is mined with the transverse stoping variant of the LHOS mining method and only 7% is mined with the longitudinal stoping variant of the LHOS mining method. Figure 16.32 and Figure 16.33 both show the distribution of the mining methods in the mine. Finally, Figure 16.34 shows a plan view of the underground mine with the open pit.

Figure 16.30: Underground Mine Longitudinal View by Mining Blocks - Looking West



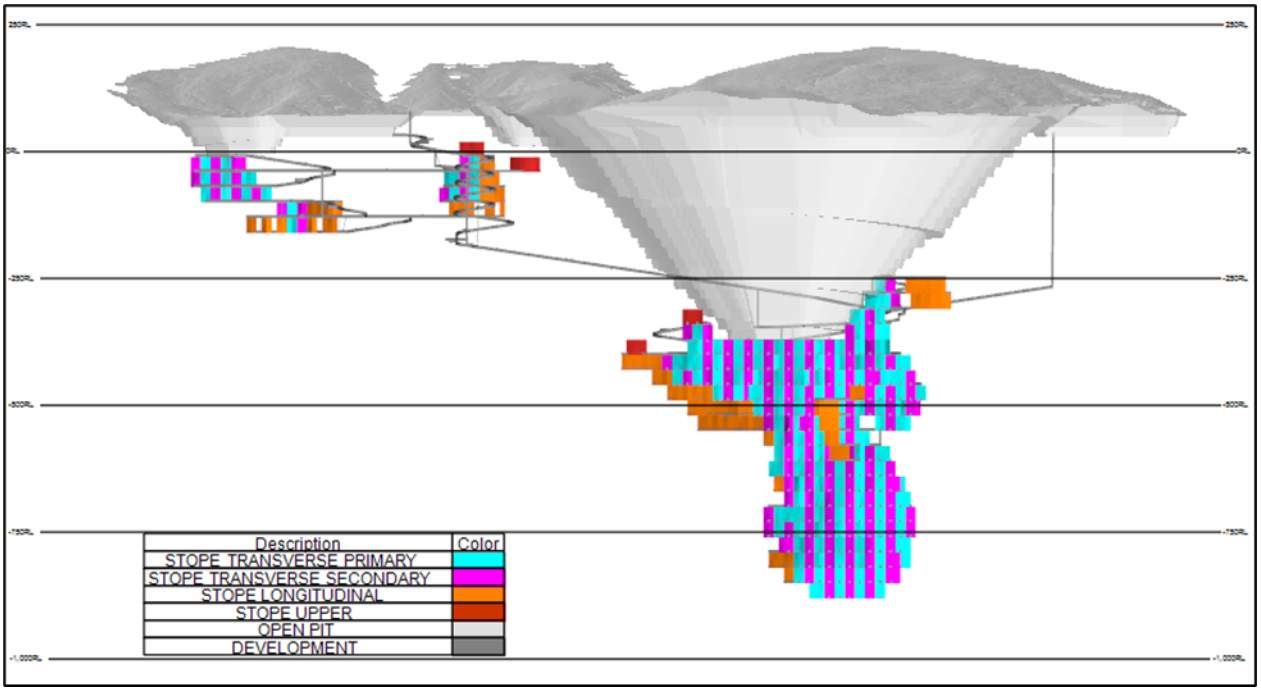
Source: GMS 06-06-2024 (not to scale)

Figure 16.31: Underground Mine Longitudinal View by Mining Blocks - Looking North



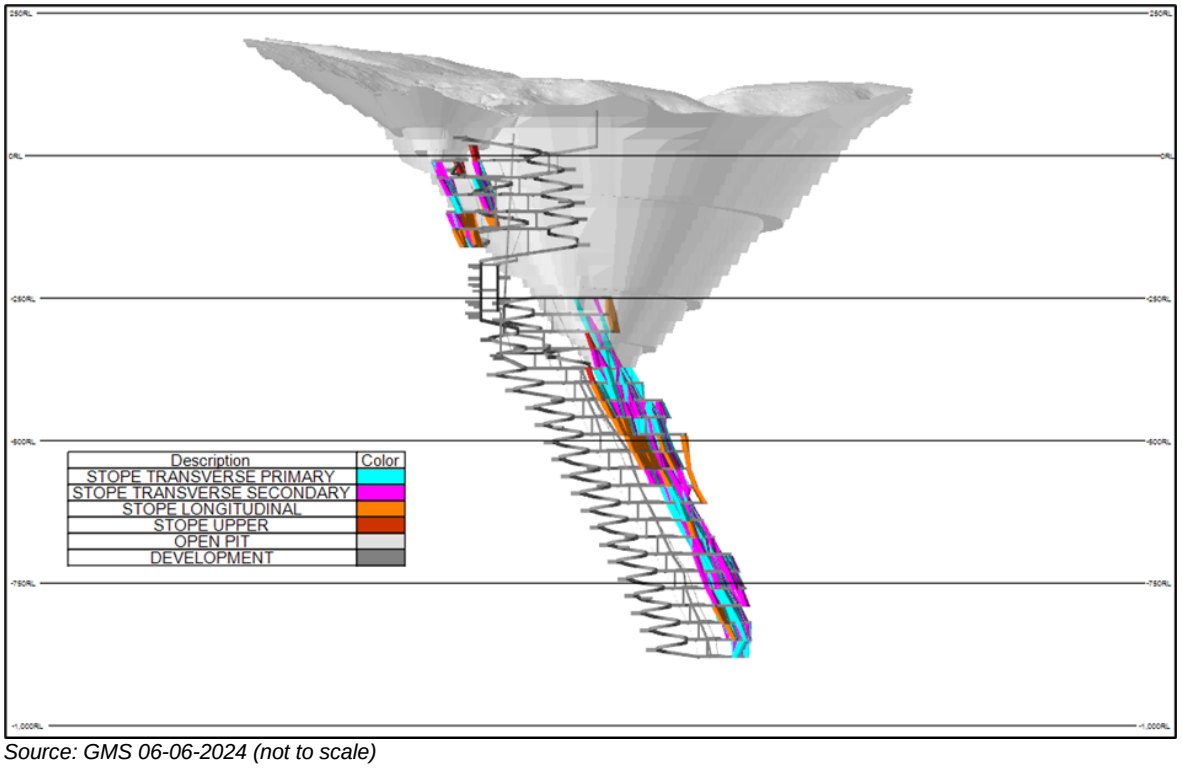
Source: GMS 06-06-2024 (not to scale)

Figure 16.32: Underground Mine Longitudinal View by Mining Method - Looking West



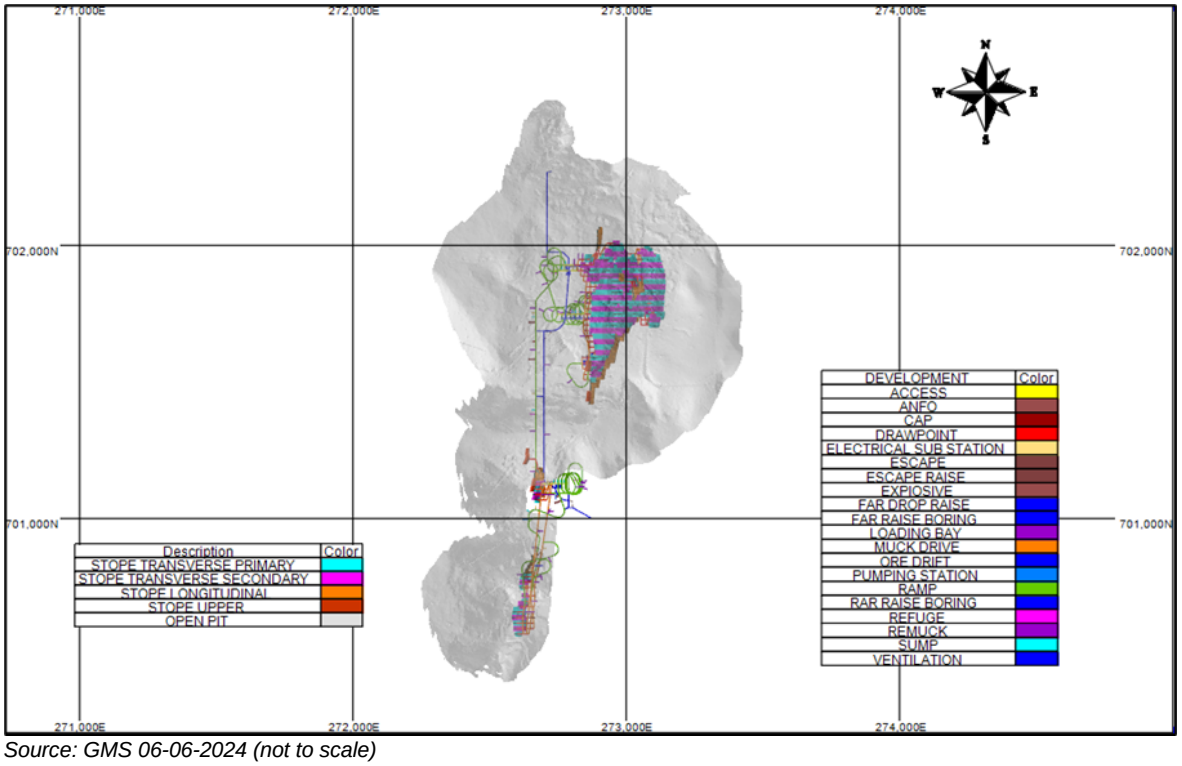
Source: GMS 06-06-2024 (not to scale)

Figure 16.33: Underground Mine Longitudinal View by Mining Method - Looking North



Source: GMS 06-06-2024 (not to scale)

Figure 16.34: Underground Mine Plan View with the Open Pit



Source: GMS 06-06-2024 (not to scale)

16.5.4.3 Physicals Summary

Table 16.23 summarizes the total development statistics of the mine. The stoping and development tonnages and gradients are presented in Table 16.24, including both mineralized material and waste development quantities.

Table 16.23: Underground Mine Design Summary

Development Type	Unit	Value
Lateral Development		
Main Decline	m	8,290
Level Access	m	1,668
Haulage Drift	m	7,167
Infrastructures	m	7,634
Sub-total CAPEX Development	m	24,759
OPEX Development	m	22,287
Total Lateral Development	m	47,047
Vertical Development		
Raise Bore	m	375
Drop Raise	m	1,690
Emergency Egress	m	1,138
Total Vertical Development	m	3,203

Table 16.24: Underground Mine Physicals Summary

Item	Unit	Value
Development Physicals		
Development Mineralized Material	t	1,020,684
	g/t	2.68
Development Waste	t	2,882,102
Production Physicals		
Stoping Mineralized Material	t	13,479,927
	g/t	3.22
Total Mine Physicals		
Total Underground Mineralized Material	t	14,500,611
	g/t	3,19

16.5.4.4 Development and Production Rates

The targeted underground mine production rate is set at 4,250 tpd or 1.55 Mt of mineralized material per year including stope production (4,000 tpd) and lateral development within mineralized material (250 tpd). The production rate varies slightly since the quantity of development ore produced is not constant over the LOM. Multiple mining blocks are mined simultaneously to maintain the targeted underground mine production rate. The production rate for the Oko West UG mine is calculated using the Deswik™ mining sequence, considering the different rates shown in Table 16.25.

Table 16.25: Underground Mine Scheduler Rates

Parameter	Units	Rate
Single Face Development Rate	m/d/unit	5.0
Multi Face Development Rate	m/d/unit	9.5
Stope Preparation	d	3
Stope Cables	m/d	120
Slot Raise Drilling Rate	m/d	10
Production Drilling Rate	m/d	200
Production Drilling Factor - Transverse	t/m drilled	14.1
Production Drilling Factor - Longitudinal	t/m drilled	11.7
Blasting Delay	d	3
Mucking Rate	t/d	1,400
Rockfill Rate	t/d	1,200
Cemented Rockfill Rate	t/d	1,000
Short Cure Time	d	7
Long Cure Time	d	28
Maximum Stoping	t/d	4,000

16.5.5 Development and Production Sequencing

Once the excavation and construction of the portal is completed, the development phase will begin with the development of the main decline towards the satellite zones. This approach reduces the initial project CAPEX and ensures that these two (2) zones enter production quickly. The development of the primary ventilation and safety egress networks are a priority since they are essential to allow stoping to begin. Although the initial focus is to enter in production rapidly in the satellite zone, the priority remains the development of the ramp to access the main zone, and particularly the bottom of the first mining block in the main zone. Access to the main zone is critical for transitioning from initial production to full production, as it is the largest zone and essential for sustaining the full production rate.

Stoping in the satellite zones begins approximately two (2) years after the start of the development. The satellite zones will be in production for 19 months and the main zone will be in production through the entire LOM. The mineralized material production profile (mined material) of the underground mine is summarized by zone in Figure 16.35, and the production plan (mined material) for the underground mine is presented in Table 16.26.

Figure 16.35: Underground Mine Mineralized Material Production by Zone

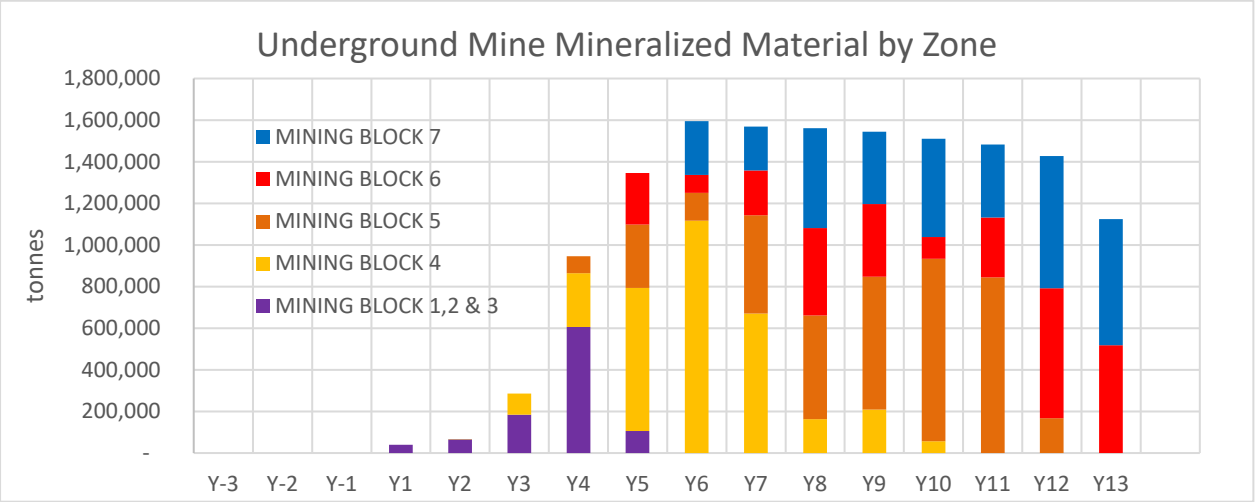


Table 16.26: Underground Mine Production Plan

Oko West Underground Mine	Units	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Development Mineralized Material	kt	1,021	-	-	40	67	102	124	155	135	108	102	81	51	22	33	-
	g/t	2.7	-	-	2.0	2.1	2.9	2.4	2.6	3.2	2.7	2.8	2.3	2.7	3.5	3.2	-
	koz	88	-	-	3	4	10	9	13	14	9	9	6	4	3	3	-
Stoping Mineralized Material	kt	13,480	-	-	-	-	184	822	1,190	1,460	1,460	1,460	1,464	1,460	1,460	1,395	1,125
	g/t	3.2	-	-	-	-	2.5	2.4	3.3	3.5	3.2	3.2	3.1	3.0	3.4	3.5	3.7
	koz	1,398	-	-	-	-	15	63	125	162	150	151	147	140	158	155	132
Total Underground Mineralized Material	kt	14,501	-	-	40	67	286	946	1,345	1,595	1,568	1,562	1,545	1,511	1,482	1,428	1,125
	g/t	3.2	-	-	2.0	2.1	2.6	2.4	3.2	3.4	3.2	3.2	3.1	3.0	3.4	3.4	3.7
	koz	1,485	-	-	3	4	24	73	138	176	159	160	153	144	161	158	132

16.5.6 Underground Mine Equipment

The requirements in terms of underground equipment were determined based on the number of operating hours needed to achieve the projected production and development rates outlined in the mine plan.

During the production years, haulage cycles consider the distances from the loading point in the footwall drifts to the level access, followed by the ramp ascent to the surface stockpile. Mucking and hauling cycles are determined based on a fixed distance between stopes and the loading point of trucks.

The quantities of auxiliary equipment were estimated based on the size of the operation or derived from other equipment requirements. Table 16.27 shows the results of the equipment requirements for the mine pre-production and the full-production stages of the LOM.

Table 16.27: Underground Mine Mobile Equipment Fleet

Equipment Type	Qty Pre-Production	Qty Production
Jumbo – 2 Boom	2	2
Bolter (plate form bolter)	4	4
Production Drill – Hydraulic Top Hammer	-	2
Production Drill – ITH	1	1
Cable Bolter	-	1
LHD – 21T	2	9
LHD – 10T	1	2
Truck – 63T	2	12
Explosive Truck - Development	1	1
Explosive Truck - Production	-	2
Scissor Lift - Development	1	2
Scissor Lift - Construction	-	2
Attachment - Fan Handler	-	1
Boom Truck	1	1
Personnel Carrier	-	1
Fuel & Lube Truck	1	1
Water Truck	-	1
Block Holer	-	1

Equipment Type	Qty Pre-Production	Qty Production
Cassette Truck	1	3
Cassette - Crane	-	1
Cassette - Emulsion Loading Deck	-	1
Cassette - Flat Deck	-	1
Cassette - Fuel Tanker	1	1
Cassette - Fuel & Lube	1	1
Cassette - Personnel Carrier	-	1
Cassette - Transmixer	-	1
Shotcrete Sprayer	-	1
Shotcrete Mixer	-	3
Grader	-	1
Light Vehicle	5	25
Light Vehicle – Mine Rescue	1	1
Tractor - Mechanics	1	2
Tractor - Electricians	0	2
Mobile Air Compressor	1	2
Backhoe Loader	-	2
Shotcrete and Concrete Transport	-	1
V-30	-	1

16.5.7 Underground Mine Ventilation and Cooling

Ventilation requirements for the underground mine are primarily based on diesel emissions from the equipment fleet. The minimum ventilation standard of 0.06 m^3/kW was used to determine the required airflow per equipment taking into account some attenuation factors applied based on the estimated equipment utilization. Table 16.28 illustrates the typical ventilation fresh air requirements per equipment used underground. Preliminary Ventsim designs have been created and simulations performed for the maximum production scenario.

Table 16.28: Underground Mine Fresh Air Requirements per Equipment

Equipment	Engine	HP	CFM/EQUIP	Utilization Factor	CFM Requirement
Jumbo – 2 Boom	Cummins QSB 4.5	170	16,117	40%	6,447
Bolter	Cummins QSB 4.5	170	16,117	40%	6,447
Production Drill – Top Ham.	Mercedes OM904 Series	148	13,985	40%	5,594
Production Drill - ITH	Mercedes OM904 Series	148	13,985	40%	5,594
Cable Bolter	Mercedes OM904 Series	148	13,985	40%	5,594
LHD – 21T	Volvo TAD1344VE	472	44,751	80%	35,801
LHD – 10T	Volvo TAD1340VE	343	32,546	80%	26,037
Truck – 63T	Volvo TAD1643VE-B (Tier 2)	758	71,830	80%	57,464
Explosive Truck	Cummins QSB 4.5	170	16,117	60%	9,670
Scissor Lift	Cummins QSB 4.5	170	16,117	60%	9,670
Personnel Carrier	Cummins QSB 4.5	170	16,117	60%	9,670
Deck Truck	Cummins QSB 6.7	193	18,297	60%	10,978
Water Truck	Cat C7 ACERT™ TIER 2	145	13,746	60%	8,248
Block Holer	Cummins QSB 4.5	170	16,117	60%	9,670
Scaler	Cummins QSB 4.5	170	16,117	60%	9,670
Shotcrete Sprayer	Cummins QSB 4.5	170	16,117	60%	9,670
Shotcrete Mixer	Cummins QSB 4.5	170	16,117	60%	9,670
Grader	Cat C7 ACERT™ TIER 2	145	13,746	60%	8,248
Light Vehicle	1HZ PCNA	127	12,040	60%	7,224
Light Vehicle - Mine Rescue	1HZ PCNA	127	12,040	60%	7,224
Tractor - Mechanics	Cummins QSB 6.7	193	18,297	60%	10,978
Tractor	Mercedes OM904 Series	148	13,985	60%	8,391
Shotcrete and Concrete Transport	Cummins QSB 6.7	193	18,297	60%	10,978

Table 16.29 illustrates the ventilation fresh air requirement quantities estimated for the Oko West UG mine.

Table 16.29: Underground Mine Fresh Air Requirements

Equipment	Qty Total	CFM Requirement
Jumbo – 2 Boom	2	12,893
Bolter	4	25,786
Production Drill – Top Ham.	2	11,188
Production Drill - ITH	1	5,594
Cable Bolter	1	5,594
LHD – 21T	9	322,205
LHD – 10T	2	52,074
Truck – 63T	12	603,372
Explosive Truck	3	29010
Scissor Lift	4	38680
Boom Truck	1	10,978
Personnel Carrier	1	9,670
Fuel & Lube Truck	1	10,978
Water Truck	1	8,248
Block Holer	1	9,670
Cassette Truck	3	29,010
Shotcrete Sprayer	1	9,670
Shotcrete Mixer	3	29,010
Grader	1	8,248
Light Vehicle	25	180,600
Light Vehicle - Mine Rescue	1	7,224
Tractor - Mechanics	2	21,956
Tractor - Electricians	2	16,782
Backhoe Loader	2	6,509
Shotcrete and Concrete Transport	1	10,978
Fresh Air Requirements (Sub-Total)	-	1,475,926
Contingency (10%)	-	147,593
Fresh Air Requirements (Total)	-	1,623,519

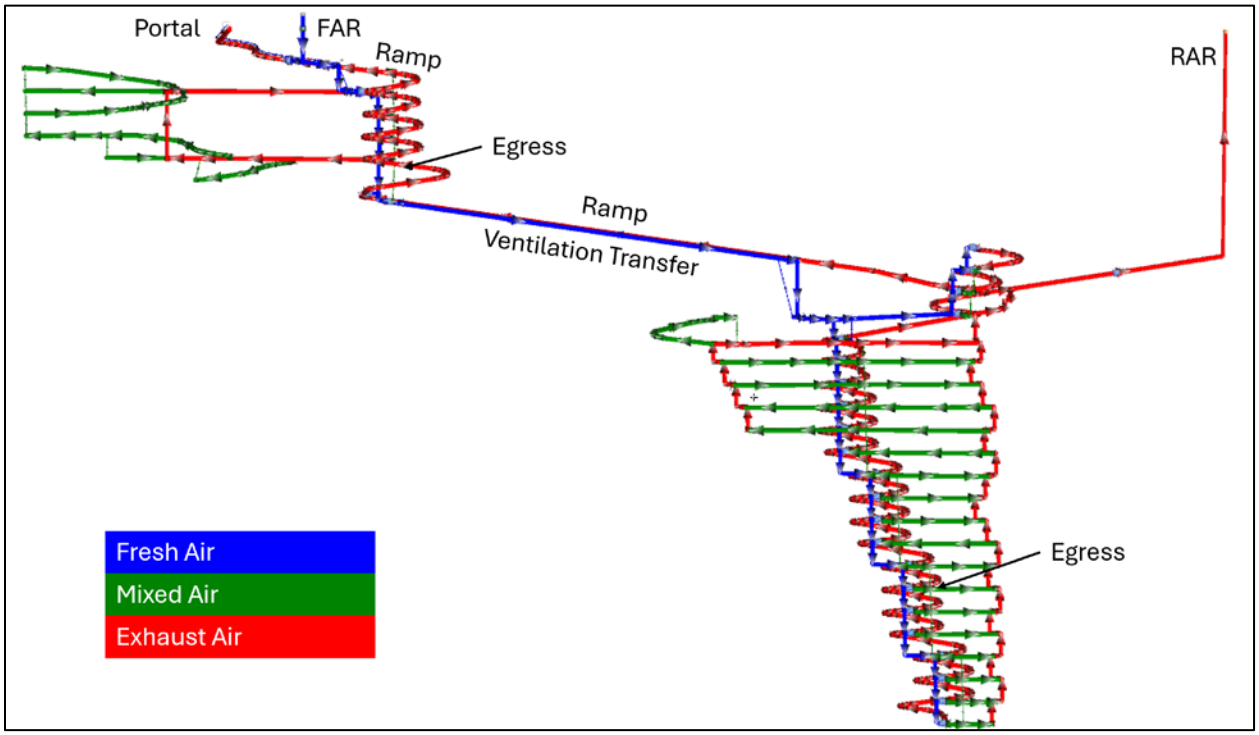
A phased approach will be used to implement the ventilation system. Initially, a temporary system will be put in place to supply fresh air and allow the development of the main decline until the permanent ventilation network is developed and commissioned.

The temporary ventilation system is designed to provide fresh air requirements for one (1) LHD and two (2) haul trucks. This entire fresh air volume will be provided by two (2) 54” rigid ventilation ducts. Fresh air is supplied to each ventilation line by a pair of axial fans installed in series.

The permanent system is designed to accommodate the fresh air requirements of the initial pre-production phase and the ramp-up to full production. The proposed permanent ventilation system is a push ventilation system that consists of one (1) fresh air raise (FAR) intake with one (1) return air raise (RAR) and the main decline acting as exhausts. Two (2) main fans are planned for installation in parallel on the surface with both fans pushing air into the mine through the FAR. A total of 1.6M cfm is required to provide sufficient air for manpower and equipment underground.

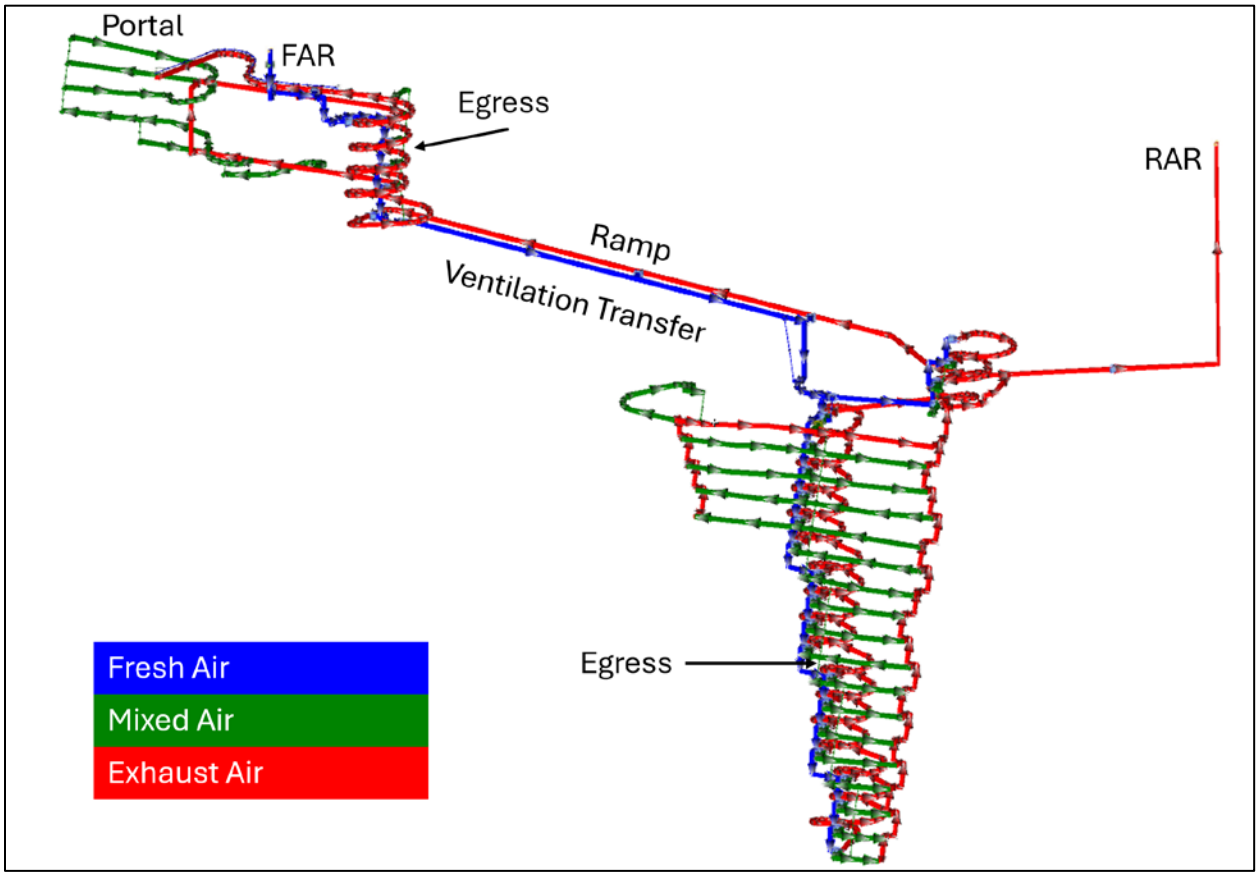
Figure 16.36 and Figure 16.37 both illustrate the permanent underground mine ventilation network.

Figure 16.36: Underground Mine Ventilation Network – Looking West



Source: GMS 06-06-2024 (not to scale).

Figure 16.37: Underground Mine Ventilation Network – Isometric View



Source: GMS 06-06-2024 (not to scale).

Both the main FAR and RAR ventilation raises will be excavated with a raise-boring machine from surface and are varying in length and diameter. The FAR is then converted into a series of drop raises to deliver fresh air at depth.

This permanent ventilation system will be operating at a variety of pressures and flows to suit the various operating conditions of the mine. The installation of ventilation louvers at the fresh air raise access on every level will ensure that an adequate amount of fresh air is distributed to the correct workplace. At smaller production levels, access to the RAR network is located north of the level access to ensure flow-through ventilation over this longer portion of the production level. On larger production levels, there is also access to the RAR network on the south of the level access to ensure flow-through ventilation on both sides of the level. Table 16.30 summarizes the different fan design parameters.

Table 16.30: Underground Mine Ventilation System Details

Main Fans	Value	Unit
Fans	2 x 1,035	kW
Pressure	16.0	In. wg.
Airflow	1,600,000	cfm
Temporary Fans	**Value**	**Unit**
Fans	4 x 300	kW
Pressure	12.0	In. wg.
Airflow	150,000	cfm

16.5.8 Underground Mine Services

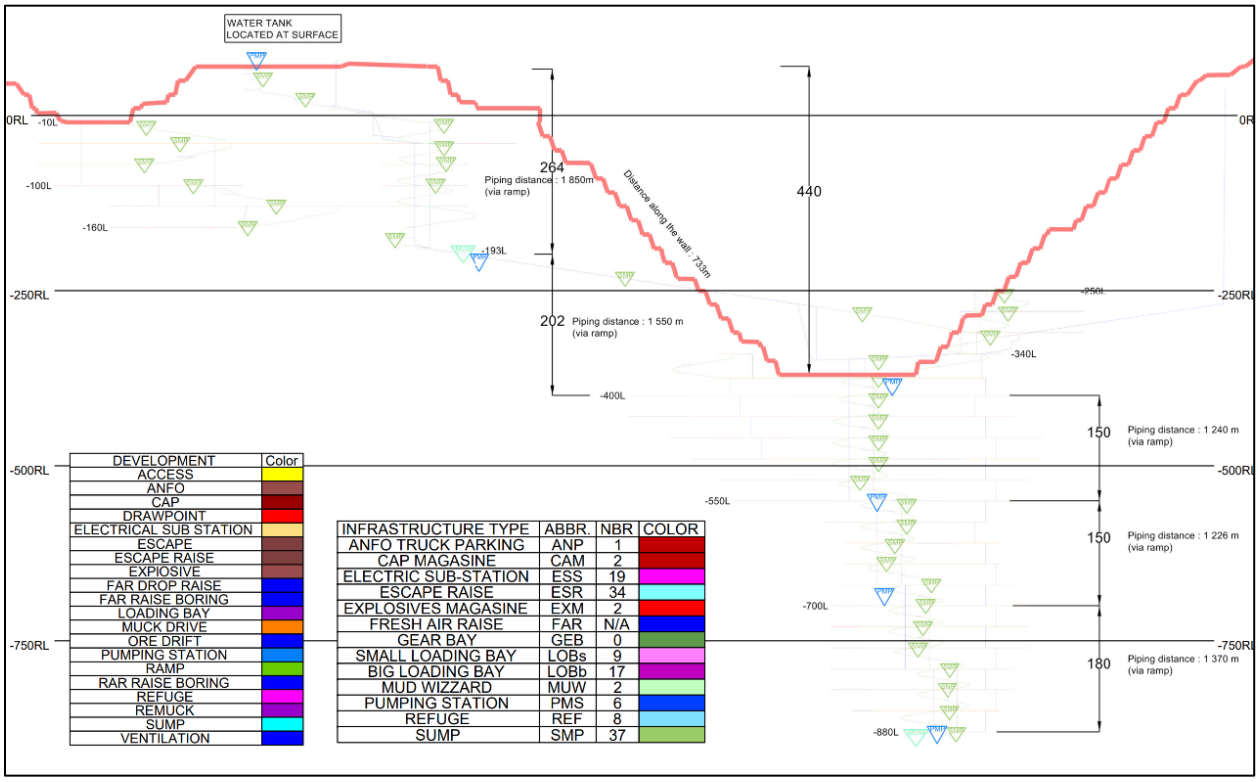
16.5.8.1 Dewatering

A mine water balance was completed for the entire mine and the overall flows were attributed to each zone of the mine, based on the production rate and natural groundwater inflow assumptions. Mine operations water consumption was calculated based on the equipment lists and their respective water consumption. Water from the underground mine will be pumped to the surface by a series of pumping stations to a surface pond. Water stored in the surface pond will then be reused for the mine operations water supply. For each zone, a pumping system will be installed near the deepest production level and some intermediate pumping stations will be required. The number of intermediate pumping stations varies depending on the depth and expected flows of each zone. Table 16.31 shows the dewatering assumption, while Figure 16.38 illustrates the dewatering network, and Table 16.32 details the pumping required.

Table 16.31: Underground Mine Dewatering Assumption

Water Operation	Value	Unit
Mine Operations Water	798	USGPM
	3,019	(l/min)
Natural Ground Water	1,000	USGPM
	3,785	(l/min)
Total Dewatering	1,798	USGPM
	6,805	(l/min)

Figure 16.38: Underground Mine Dewatering Network



Source: GMS 06-06-2024 (not to scale).

Table 16.32: Underground Mine Pumping Requirements Details

Pump Capacity		
Mining Block #1-2-3		
Pipe Length	1,850	m
Dewatering Flow Capacity	3,595	USGPM
Vertical Head	264	m
Pump Efficiency	75%	%
Pump Power	1,500	hp
Mining Block #4		
Pipe Length	1,550	m
Dewatering Flow Capacity	3,195	USGPM
Vertical Head	202	m
Pump Efficiency	75%	%
Pump Power	1,000	hp
Mining Block #5		

Pump Capacity		
Pipe Length	1,240	m
Dewatering Flow Capacity	2,795	USGPM
Vertical Head	150	m
Pump Efficiency	75%	%
Pump Power	650	hp
Mining Block #6		
Pipe Length	1,226	m
Dewatering Flow Capacity	2,395	USGPM
Vertical Head	150	m
Pump Efficiency	75%	%
Pump Power	500	hp
Mining Block #7		
Pipe Length	1,370	m
Dewatering Flow Capacity	1,595	USGPM
Vertical Head	180	m
Pump Efficiency	75%	%
Pump Power	400	hp

16.5.8.2 Cemented Rockfill Plant

Cemented rockfill (CRF) has been selected as the cemented fill material for the underground mine. CRF is a mix of waste rock and cement slurry. In this case, the cement slurry will be produced on surface at the same concrete batch plant used for construction purposes. Cement bulk bags will be transported to the plant by surface trucks to feed the batch plant and produce cement slurry. Cement slurry will be transported underground to the desired location by trans mixers and/or agitator trucks. The waste rock for rockfill and cemented rockfill will primarily be sourced from development waste, while any additional waste required will be obtained from surface operations. Waste material sourced from surface operations will be crushed to minus 6 inch and backhauled underground in the same trucks that transported the mineralized material to the surface. Waste rock will then be mixed with the cement slurry from the trans mixer / agitator truck directly in the bucket of the LHD used for backfilling activities. The LHD will then dump the CRF directly in the stope to fill. The planned cement content is 5% for all underground stopes.

16.5.8.3 Compressed Air

The compressed air supply will be provided by a series of electrical compressors installed on the surface. The compressed air piping network will be installed along the ramp, in the main drifts and in the escapeways throughout the mine. Compressed air will provide power to the dewatering pumps of the development headings, to handheld drills, to some air powered actuators as well as any other air-powered equipment. The compressors will also provide an emergency air supply to the refuge stations.

16.5.8.4 Communications

The underground communication network consists of an LTE system that will be installed on site and will be expanded over the LOM. Mobile equipment operators, light vehicles, and supervisors will be equipped with LTE phones to communicate with personnel on the surface. LTE will also be used to control the underground ventilation network and will allow the use of the automation features of some equipment.

16.5.8.5 Fuel Storage and Distribution

Fuel will mostly be stored on the surface. However, there will be a small size underground fuel distribution system to provide fuel for equipment that rarely returns to the surface. A fuel truck is also planned as part of the mobile equipment fleet to distribute the fuel to underground equipment that cannot travel quickly to the underground fuel station for refueling.

16.5.8.6 Explosives Storage and Handling

Two (2) underground explosive and detonator magazines will be installed in designated locations: one (1) near surface in the satellite zone and one (1) in the main zone. Explosives will be delivered to the portal by the selected explosive supplier, then will be transported to the underground magazines by flatbed service truck for later use.

16.5.8.7 Personnel and Underground Material Transportation

Supplies and personnel will access the underground mine via the main ramp. A series of personnel carriers, such as land cruisers will be used to transport workers from the surface to the underground mine. Supervisors and technical services will also use some light vehicles for transportation underground. The construction team as well as mechanical and electrical personnel will use maintenance tractors. A flatbed truck equipped with a service boom will be used to move supplies from the surface to the underground active headings, stopes and material storages.

16.5.8.8 Equipment Maintenance

Most of the major mechanical maintenance will be performed in the surface workshop. Some major repairs will be completed underground in a garage excavated and equipped for this purpose. All minor maintenance and small emergency work will be performed underground by mobile maintenance teams.

16.5.9 Underground Mine Safety Measures

16.5.9.1 Emergency Exits

The main decline will provide primary egress from the underground workings. In the case of a secondary exit, the egress raises will be excavated between production levels for most of the mine. It is important to note that some of the ventilation raises, like the FAR, will be equipped with manways, and some of the ventilation transfer drifts between the different zones will also serve as secondary egress. The independent safety egress raises will be equipped with prefabricated modular Laddertube^{tm} systems.

16.5.9.2 Refuge Stations

Refuge stations will be positioned so that all employees can access a refuge in less than 10 minutes from the moment they leave their workplace or at every 1,000 metres. Refuge stations used for the underground mine will be portable or pre-built refuge stations.

Each refuge station will be equipped with the following:

- Telephone or radio for communication with surface, independent of mine power supply
- Compressed air, water lines, and water supply
- Emergency lighting
- Hand tools and sealing material
- Plan of the underground work showing all exits and the ventilation plans
- All other necessary items according to the applicable regulation
- Fire Protection

Underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with best practice. Fire extinguishers will be provided and maintained in accordance with regulations and best practices at electrical installations, pump stations, gear bays, fuel station, service garages and wherever a

fire hazard exists. Every vehicle will carry at least one (1) fire extinguisher of adequate size and proper type.

16.5.9.3 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with applicable regulations. Mine rescue equipment, including a dedicated underground emergency vehicle and a foam generator, will be available on site.

Rescue teams will be trained for surface and underground emergencies. An emergency response plan will be developed and continuously updated as the mine and regulations evolve.

16.5.9.4 Emergency Stench System

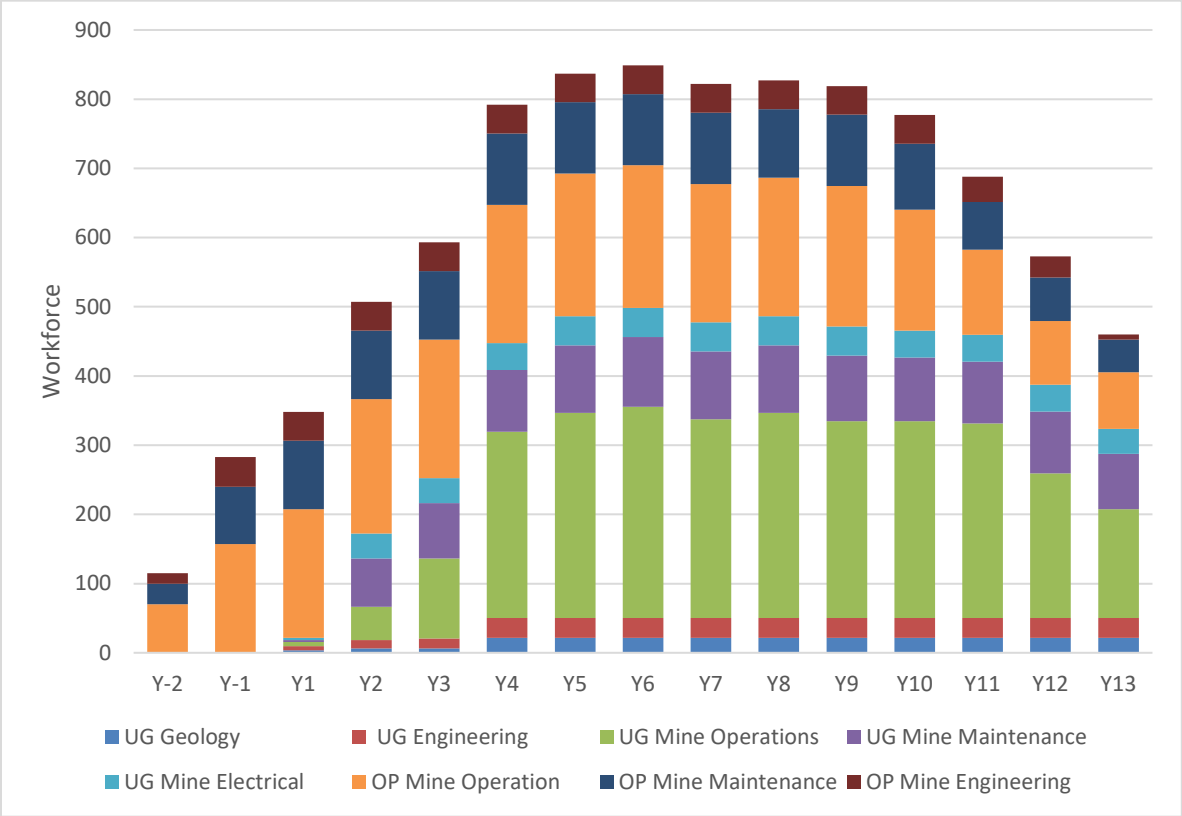
A mine stench gas warning system will be installed in the main surface ventilation system, initially in the temporary system and later in the permanent system. This system is designed to inject a specific dosage of stench gas, calculated based on airflow quantity. This gas with a particular smell would alert the workers of an emergency as soon as they smell the gas. Another mine stench gas warning system will be installed at the mine compressed air system as a second means to alert underground workers in the event of an emergency.

16.6 Mine Manpower

Mine personnel were divided into hourly and staff positions and divided between mine operations, mine maintenance, mine engineering and geology. Hourly positions were mostly associated with a shift roster of 14-days-on and 7-days-off, and as such, each unit of equipment requires three (3) operators hired in hourly positions.

Staff positions in management, supervision or technical services roles that require a continuous 7 days a week presence will also be on roster schedule. Most of the staff positions are considered expats on a 23/19 schedule. The combined UG and OP mine workforce peaks at 849 individuals in Year 6. Figure 16.39 shows the workforce variation over the life of mine.

Figure 16.39: OP and UG Mine Workforce



16.6.1 Open Pit Mine Manpower Requirements

Table 16.33 to Table 16.36 show the estimated OP workforce requirements over the LOM. The mine workforce peaks at 351 individuals in Year 5. Note that some positions are shared with the UG mine workforce (ex: Chief Engineer, Chief Geologist, etc.).

Table 16.33: Open Pit Mine Operations Workforce

Mine Operations	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Superintendent	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Ops. General Foreman	0	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Supervisor	2	8	8	8	8	8	8	8	8	8	8	8	4	2	0
Mine D&B Supervisor	0	2	2	4	4	4	4	4	4	4	4	4	2	2	2
Dispatcher	0	4	4	4	4	4	4	4	4	4	4	4	2	2	2
Training Supervisor	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Driller	2	3	9	9	12	12	12	12	12	12	12	9	6	6	4
Auxiliary Drill Operator for Mineralized Material (4.5-8")		4	4	4	4	4	4	4	4	4	4	4	4	4	4
Auxiliary Drill Operator - Auxiliary Pre-split Drill (4.5-8")	3	3	3	3	3	3	3	3	3	3	3	3	3	3	4
Blaster	1	2	3	3	4	4	4	4	4	4	4	2	1	1	1
Blaster Helper	2	4	6	6	8	8	8	8	8	8	8	4	2	2	2
Explosives Truck Operator	0	2	3	3	4	4	4	4	4	4	4	2	1	1	1
Explosives Plant Mechanic	0	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Explosives Plant Batchmen	0	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Utility Equip. Operator - Small Stemming Loader (95 HP)	1	2	2	2	4	4	4	4	4	4	4	2	1	1	1
Laborer	0	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Shovel/Excavator Operator LU 1	6	9	12	12	12	12	12	12	9	9	9	9	6	3	4
Loader Operator LU 4	0	3	3	6	3	3	3	3	3	3	3	3	3	3	4
Haul Truck Operator 1	30	60	63	63	63	63	69	69	69	69	72	63	36	30	24
Dozer Operator - Track Dozer 1	4	12	15	15	15	15	15	15	12	12	12	12	9	6	4

Mine Operations	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Grader Operator	2	6	9	12	12	12	12	12	12	12	12	12	9	6	8
Water Truck Operator – Water / Sand Truck	0	3	3	3	3	3	3	3	3	3	3	3	3	3	4
Dozer Operator - Wheel Dozer	0	3	6	6	6	6	6	6	6	6	6	3	3	3	4
Utility Equip. Operator - Articulated Truck – CAT 740	6	6	6	6	6	6	6	6	6	6	6	6	6	0	0
Shovel/Excavator Operator	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Shovel/Excavator Operator	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Utility Equip. Operator - Wheel Loader	0	2	4	4	4	4	4	4	4	4	4	4	4	4	4
Sub. Total Mine Operations	70	157	186	194	200	200	206	206	200	200	203	175	123	92	82

Table 16.34: Open Pit Maintenance Workforce

Mine Maintenance Workforce	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
General Foreman	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Supervisor	2	8	8	8	8	8	8	8	8	8	8	8	4	2	2
Senior Planner	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Planner	0	2	2	2	2	2	2	2	2	2	2	2	0	0	0
Mechanical Engineer	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Trainer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Clerk	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mobile Mechanic	8	36	48	48	48	52	52	52	52	48	52	44	32	28	20
Electrician	4	8	8	8	8	8	8	8	8	8	8	8	4	4	4
Welder / Machinist	4	8	8	8	8	8	8	8	8	8	8	8	4	4	4
Fuel & Lube Technician	2	4	8	8	8	8	8	8	8	8	8	8	8	8	4
Tool Crib Attendant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Helper	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Sub. Total Mine Maintenance	30	83	99	99	99	103	103	103	103	99	103	95	69	63	47

Table 16.35: Open Pit Mine Engineering and Geology Workforce

Mine Geology and Engineering Workforce	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Geologist	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Senior Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Resource Geologist	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Production Geologist	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Junior Geologist	1	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Geology Technician	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Grade Control Labourers / Samplers	2	8	8	8	8	8	8	8	8	8	8	8	8	8	1
Chief Mine Engineer	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Assistant Chief Mining Engineer	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Long-Term Planning Engineer	1	2	2	2	2	2	2	2	2	2	2	2	1	0	0
Short-Term Planning Engineer	1	2	2	2	2	2	2	2	2	2	2	2	2	1	0.5
Drill & Blast Engineer	0	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Senior Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Geotechnical Engineer	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Junior Mine Engineer	0	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Dispatch system coordinator	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Mining Technician	0	4	4	4	4	4	4	4	4	4	4	4	2	2	0
Senior Surveyor	1	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Surveyor	2	4	4	4	4	4	4	4	4	4	4	4	2	2	0
Clerk	1	1	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Sub Total	15	43	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	36.5	30.5	7.5

Table 16.36: Open Pit Mine Total Workforce

Total Manpower Open Pit Mine	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operation	70	157	186	194	200	200	206	206	200	200	203	175	123	92	82
Mine Maintenance	30	83	99	99	99	103	103	103	103	99	103	95	69	63	47
Mine Engineering and Geology	15	43	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	41.5	36.5	30.5	7.5
Total Manpower	115	283	326.5	334.5	340.5	344.5	350.5	350.5	344.5	340.5	347.5	311.5	228.5	185.5	136.5

16.6.2 Underground Mine Manpower Requirements

All site-based positions will be on a rotational schedule. A total workforce of 499 locals / nationals and expats are expected to be employed for the Oko West underground mine. Note that some positions are shared with the open pit mine workforce and are thus not accounted for in this section. The underground mine labour is described in Table 16.37 through Table 16.40. Finally, Table 16.42 provides a summary of both the OP and UG mines workforce.

Table 16.37: UG Engineering Workforce

UG Engineering	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Mine Engineer	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Assistant Chief Mining Engineer	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Long-Term Planning Engineer	-	-	0.5	1	1	2	2	2	2	2	2	2	2	2	1
Short-Term Planning Engineer	-	-	0.5	1	1	2	2	2	2	2	2	2	2	2	1
Production Engineer	-	-	-	-	-	2	2	2	2	2	2	2	2	2	1
Drill & Blast Engineer	-	-	-	-	-	2	2	2	2	2	2	2	2	2	1
Junior Mine Engineer	-	-	-	-	-	2	2	2	2	2	2	2	2	2	-
Ventilation Engineer	-	-	-	-	1	2	2	2	2	2	2	2	2	2	1
Senior Geotechnical Engineer	-	-	-	-	-	1	1	1	1	1	1	1	1	1	1
Geotechnical Engineer	-	-	-	1	1	1	1	1	1	1	1	1	1	1	-
Construction Engineer	-	-	-	1	2	2	2	2	2	2	2	2	2	2	-
Drill & Blast Technician	-	-	-	-	-	2	2	2	2	2	2	2	2	2	1
Geotech. Technician	-	-	-	1	1	2	2	2	2	2	2	2	2	2	1
Ventilation Technician	-	-	-	1	1	2	2	2	2	2	2	2	2	2	1
Senior Surveyor	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor	-	-	2	4	4	4	4	4	4	4	4	4	4	4	4
Clerk	-	-	-	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5

Table 16.38: UG Geology Workforce

UG Geology	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Chief Geologist	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Senior Geologist	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Resource Geologist	-	-	-	-	-	1	1	1	1	1	1	1	1	1	-
Exploration Geologist	-	-	-	-	-	1	1	1	1	1	1	1	1	1	-
Production Geologist	-	-	2	2	2	4	4	4	4	4	4	4	4	4	2
Junior Geologist	-	-	-	-	-	2	2	2	2	2	2	2	2	2	1
Grade Control Technician	-	-	-	3	3	6	6	6	6	6	6	6	6	6	3
Grade Control Labourers / Samplers	-	-	-	-	-	6	6	6	6	6	6	6	6	6	3

Table 16.39: UG Mine Operations Workforce

UG Mine Operations Workforce	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Superintendent	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Ops. General Foreman	-	-	1	1	1	2	2	2	2	2	2	2	2	2	1
Clerk	-	-	-	-	-	1	1	1	1	1	1	1	1	1	1
Supervisor	-	-	3	3	9	18	21	21	18	21	18	18	18	15	12
Mine D&B Supervisor	-	-	-	-	-	2	2	2	2	2	2	2	2	2	2
Trainer	-	-	1	6	8	12	12	12	12	12	12	12	12	6	-
Planner	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2
Miner 1	-	-	-	-	4	30	30	30	30	30	30	30	30	15	15
Miner 2	-	-	-	-	4	30	30	30	30	30	30	30	30	15	15
Miner 3	-	-	-	-	4	30	30	30	30	30	30	30	30	15	15

UG Mine Operations Workforce	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Long-Hole Driller	-	-	-	3	6	6	9	9	9	9	9	9	9	9	9
Blasters	-	-	-	-	6	12	12	12	12	12	12	12	12	6	6
Scoop Operators	-	-	-	3	6	15	24	27	24	27	24	24	24	21	18
Truck Operators	-	-	-	5	9	18	30	36	36	36	36	36	33	36	27
Jumbo Operator	-	-	-	3	6	6	6	6	3	3	3	3	3	3	-
Rockbolter	-	-	-	5	9	9	9	9	6	6	3	3	3	3	-
Cable Bolter	-	-	-	3	3	3	3	3	3	3	3	3	3	3	-
Scissor Lift Operator-Development	-	-	-	6	12	12	12	12	6	6	6	6	6	6	-
Level Services	-	-	-	-	3	6	6	6	6	6	6	6	6	6	6
Grader Operator	-	-	-	-	3	3	3	3	3	3	3	3	3	3	3
U/G Constructions	-	-	-	6	8	12	12	12	12	12	12	12	12	6	6
Sumps and Services Labour	-	-	-	-	-	6	6	6	6	6	6	6	6	6	3
Boom Truck Operator	-	-	-	1	2	6	6	6	6	6	6	6	6	6	3
Fuel Truck Operator	-	-	-	1	1	6	6	6	6	6	6	6	6	6	3
Lamps-Dry	-	-	-	1	3	3	3	3	3	3	3	3	3	3	3
Labour - Lunch Room, UG Tool Crib, etc.	-	-	-	-	3	6	6	6	6	6	6	6	6	6	3
Labour Spare	-	-	-	-	4	12	12	12	12	12	12	12	12	6	3

Table 16.40: UG Mine Maintenance Workforce

UG Mine Maintenance Workforce	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Superintendent	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Superintendent Assistant	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1
General Foreman	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Reliability Engineer	-	-	-	2	2	2	2	2	2	2	2	2	2	2	1
Mechanical Engineer	-	-	-	2	2	2	2	2	2	2	2	2	2	2	1
Trainer	-	-	1	3	3	3	3	3	3	3	3	3	3	3	-
Senior Planner	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	-	-	-	3	3	3	3	3	3	3	3	3	3	3	1
Administrative Assistant	-	-	-	1	1	1	1	1	1	1	1	1	1	1	-
Leader Mechanics - Mobile Equipment	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Mechanics - Mobile Equipment	-	-	-	27	36	45	54	57	54	54	51	48	45	45	33
Welders - Mobile Equipment	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Mechanics - Fixed Equipment	-	-	-	9	9	9	9	9	9	9	9	9	9	9	3
Welder - Fixed Equipment	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Labour - Fixed Equipment	-	-	-	9	9	9	9	9	9	9	9	9	9	9	3

Table 16.41: Total UG Workforce

Total Manpower UG Mine	Y -2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
UG Engineering	-	-	6	12	14	29	29	29	29	29	29	29	29	29	16
UG Geology	-	-	3.5	6.5	6.5	21.5	21.5	21.5	21.5	21.5	21.5	21.5	21.5	21.5	10.5
UG Mine Operations	-	-	6	48	116	269	296	305	287	296	284	284	281	209	157
UG Mine Maintenance	-	-	3	70	80	89	98	101	98	98	95	92	89	89	57
UG Mine Electrical	-	-	3	36	36	39	42	42	42	42	42	39	39	39	19
Total Manpower	-	-	21.5	172.5	252.5	447.5	486.5	498.5	477.5	486.5	471.5	465.5	459.5	387.5	259.5

Table 16.42: Total Mining Workforce

Total Manpower Mines	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total Underground	0	0	21.5	172.5	252.5	447.5	486.5	498.5	477.5	486.5	471.5	465.5	459.5	387.5	323.5
Total Open Pit	115	283	326.5	334.5	340.5	344.5	350.5	350.5	344.5	340.5	347.5	311.5	228.5	185.5	136.5
Total Manpower	115	283	348	507	593	792	837	849	822	827	819	777	688	573	460

16.7 Combined Production

The open pit mine is scheduled for a pre-production period lasting 25 months. During this time, the mine will produce a total of 25.6 million tonnes (Mt) of waste and overburden, along with 2.9 Mt of mineralized material. Additionally, 1 Mt of material will be produced during the pre-production phase, which will be used for the construction of the mine and to prepare the portal for the underground mine. The focus during this phase will be on efficiently removing non-valuable material to access the mineralized zones, preparing the site for full-scale operations. In the last five (5) months of the pre-production period, the processing plant will gradually ramp up its production capacity, aiming to achieve a full output of 7 million tonnes per annum (Mtpa). This capacity is made possible by the significant amount of saprolite being sent to the mineral processing plant. Saprolite is much easier to process than other material due to its softer nature, allowing for quicker and more efficient extraction and processing. This gradual increase will ensure a smooth transition to full production, allowing for the testing and optimization of processes to maintain high efficiency and reliability as the mine moves into full-scale operations. This phased approach is critical to ensure the seamless integration of all operational components and to maximize the economic potential of the mineral resource.

As the processing plant begins production, the underground mine will advance the decline development from the portal excavated by the open pit mine. Following a two-year development period, the underground mine is expected to reach full production capacity, further enhancing the overall output and efficiency of the mining operation.

Table 16.43 shows annual production data for the Life-of-Mine (LOM). Additionally, Figure 16.41 shows mine production by type of material; Figure 16.40 shows total gold production; and Figure 16.41 shows gold production by mine.

Table 16.43: Yearly LOM Production Details

Description	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
MILL FEED																	
Saprolite MM Tonnage	kt	7,660	-	30	2,195	1,378	1,169	2,483	-	395	0	0	-	9	0	0	0
Saprolite MM Grade	g/t	1.4	-	1.5	1.5	1.4	1.3	1.3	-	1.3	1.3	1.3	-	1.3	1.3	1.3	1.3
Saprolite MM Ounces	Koz	345	-	1	108	62	50	106	-	17	0	0	-	0	0	0	0
Trans. MM Tonnage	Kt	3,411	-	271	1,989	596	553	-	-	3	-	-	-	-	-	-	-
Trans. MM Grade	g/t	1.5	-	1.4	1.8	1.0	0.9	-	-	0.5	-	-	-	-	-	-	-
Trans. MM Ounces	koz	161	-	12	114	20	16	-	-	0	-	-	-	-	-	-	-
OP Fresh Rock MM Tonnage	Kt	49,631	-	140	2,183	4,959	4,991	3,571	4,655	4,008	4,432	4,438	3,278	3,489	3,518	3,572	2,397
OP Fresh Rock MM Grade	g/t	1.8	-	0.8	1.6	1.6	1.7	1.5	1.5	1.5	1.5	1.9	1.7	1.9	2.3	2.2	3.0
OP Fresh Rock MM Ounces	Koz	2,856	-	4	112	262	274	168	219	196	208	266	181	214	261	256	234
Total OP Processed MM	Kt	60,702	-	441	6,368	6,933	6,714	6,054	4,655	4405	4,432	4,438	3,278	3,498	3,518	3,572	2,397
Total OP Processed Grade	g/t	1.7	-	1.2	1.6	1.5	1.6	1.4	1.5	1.5	1.5	1.9	1.7	1.9	2.3	2.2	3.0
Total OP Processed Ounces	Koz	3,362	-	17	334	343	340	275	219	213	208	266	181	214	261	256	234
UG MM Tonnage	Kt	14,501	-	-	40	67	286	946	1,345	1,595	1,568	1,562	1,545	1,511	1,482	1,428	1,125
UG MM Grade	g/t	3.2	-	-	2.0	2.1	2.6	2.4	3.2	3.4	3.2	3.2	3.1	3.0	3.4	3.4	3.7
UG MM Ounces	Koz	1,485	-	-	3	4	24	73	138	176	159	160	153	144	161	158	132
Total UG & OP Processed MM	Kt	75,203	-	441	6,408	7,000	7,000	7,000	6,000	6,000	6,000	6,000	4,822	5,009	5,000	5,000	3,522
Total UG & OP Processed Grade	g/t	2.00	-	1.20	1.63	1.55	1.62	1.54	1.85	2.02	1.91	2.21	2.15	2.22	2.62	2.58	3.24
Total UG & OP Processed Ounces	Koz	4,848	-	17	336	348	365	347	357	389	368	426	334	358	422	415	367

Figure 16.41: Processing by Rock Type

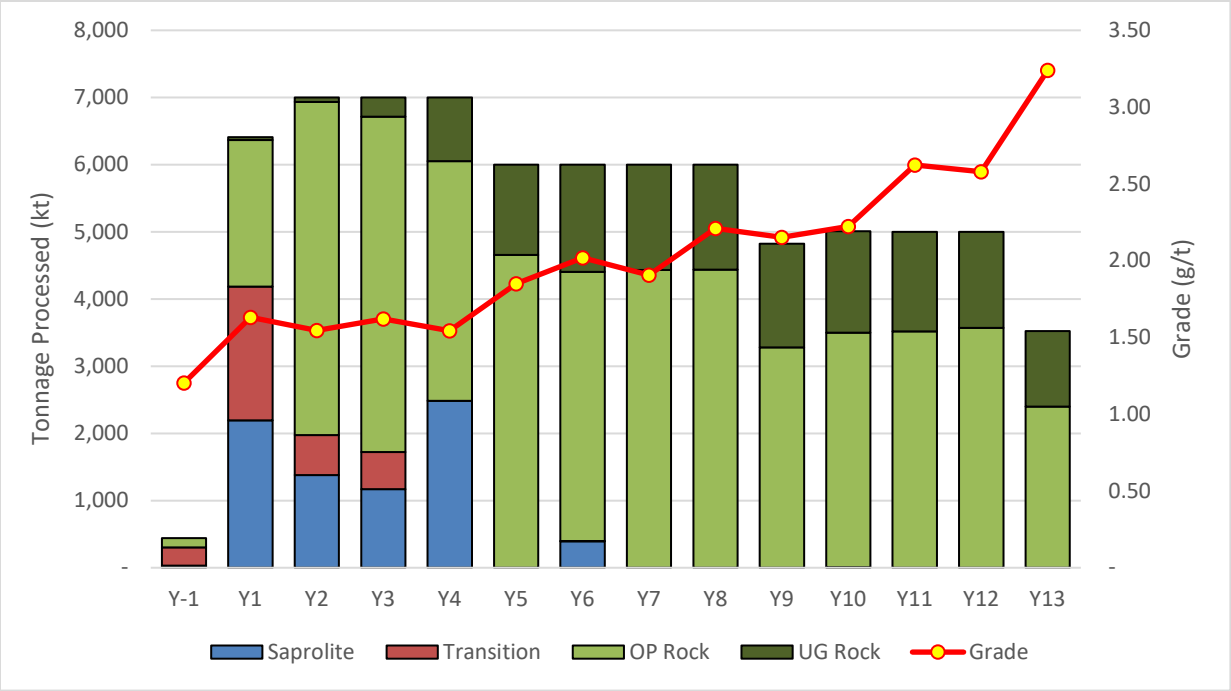


Figure 16.40: Gold Processed

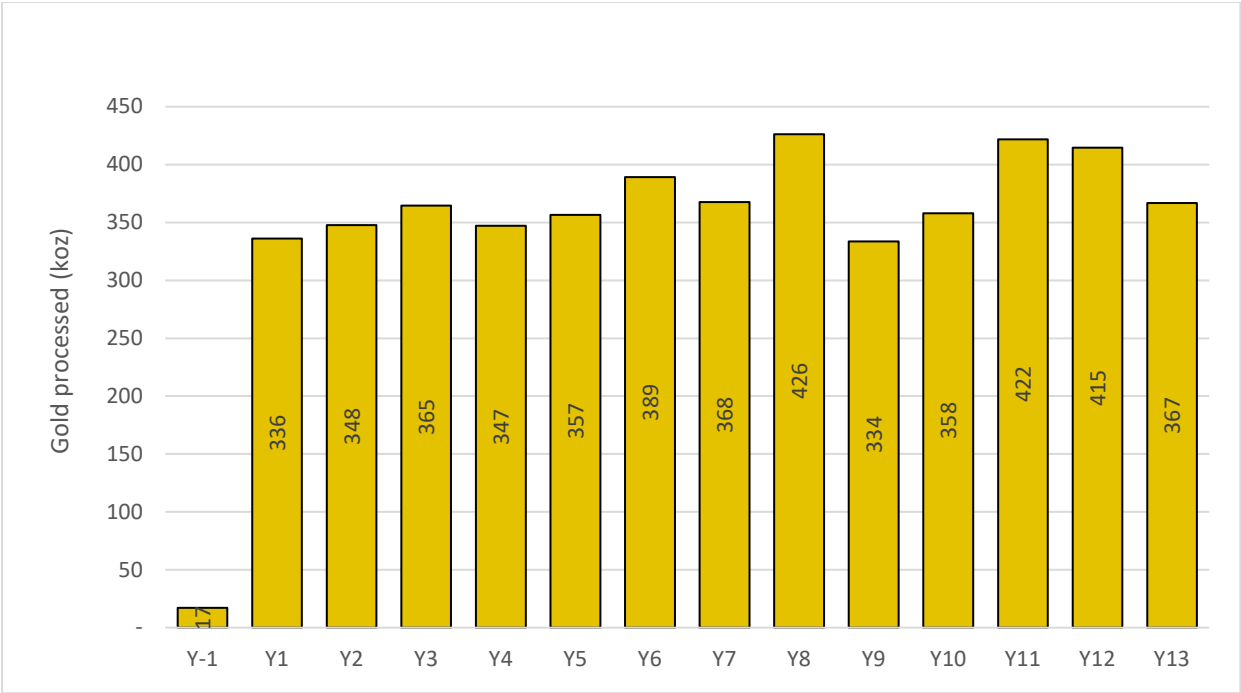
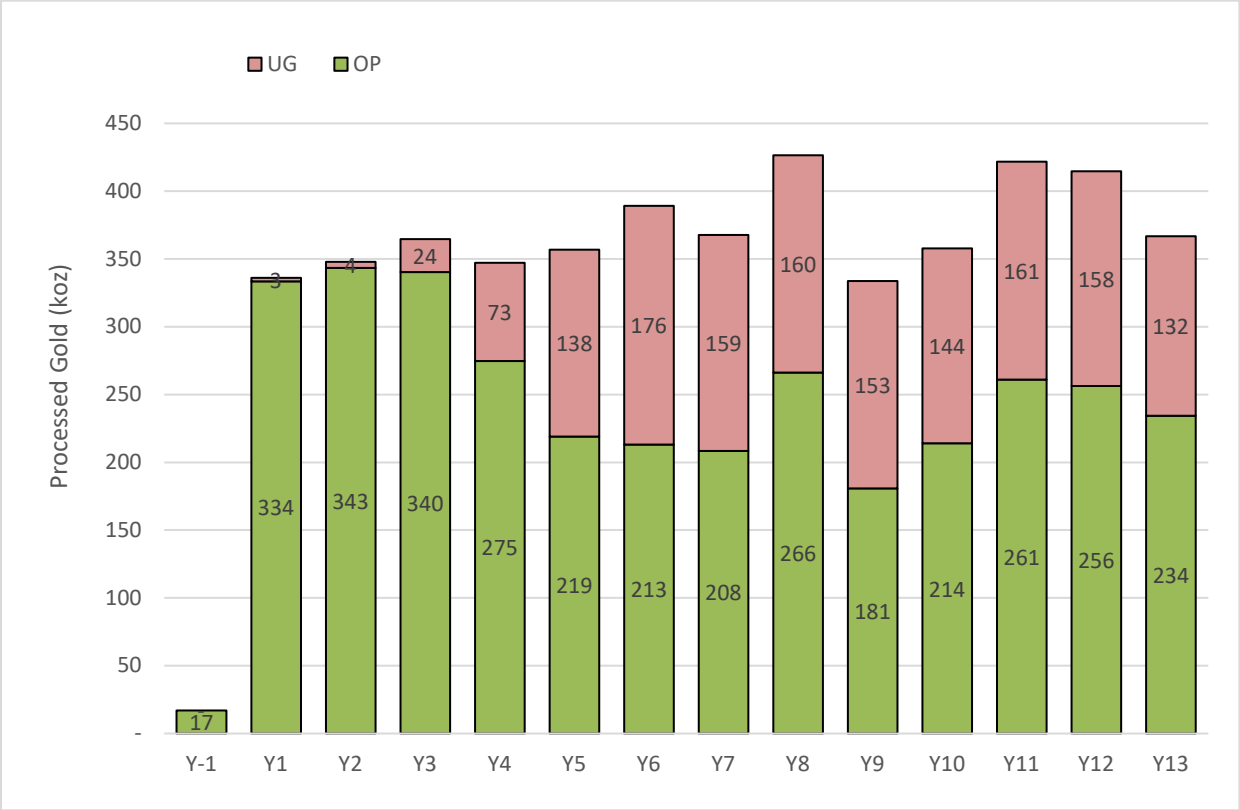


Figure 16.41: Gold Production per Mine



17 RECOVERY METHODS

17.1 Introduction

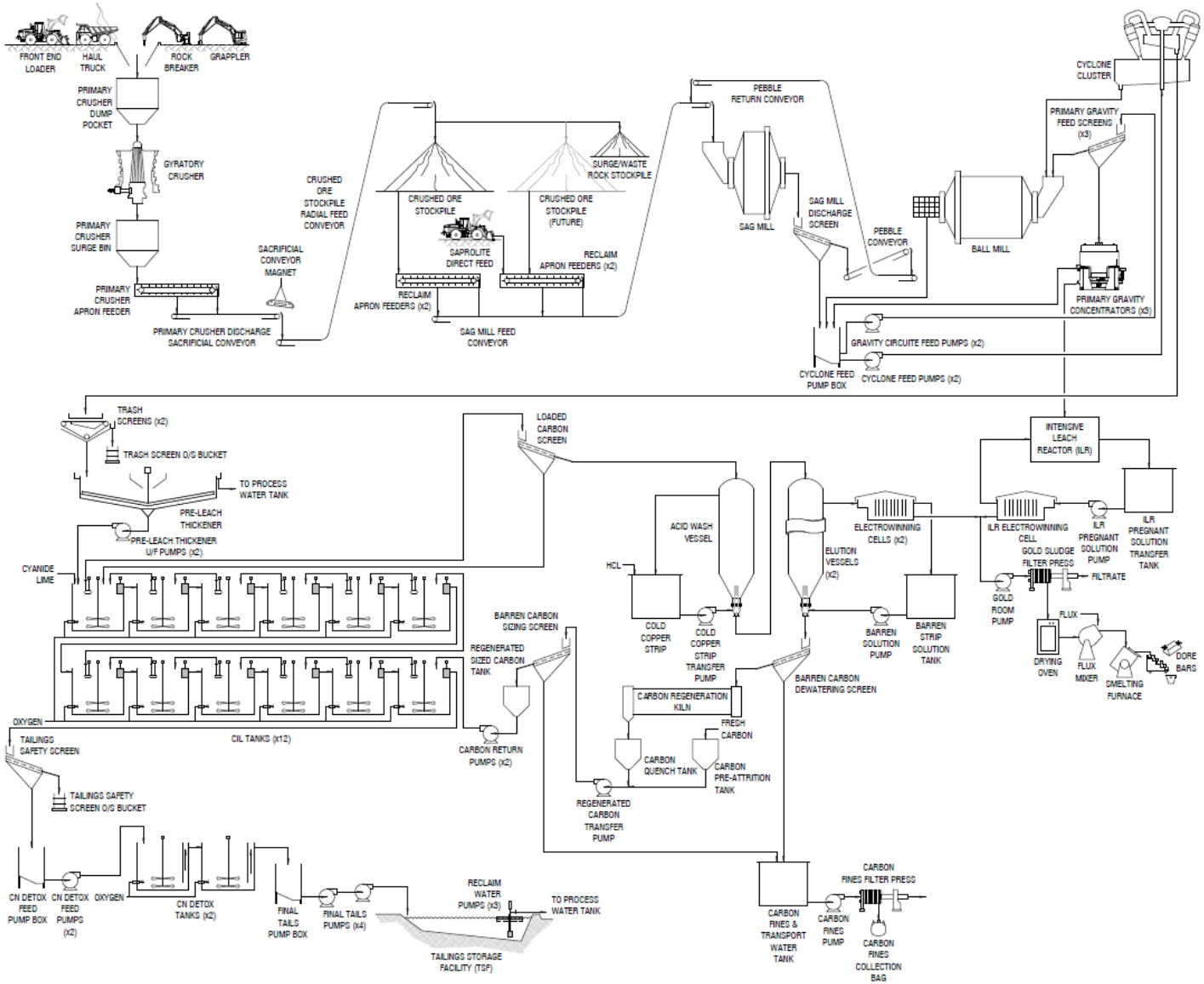
The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing material to produce doré. The flowsheet is based on metallurgical test work described in Section 13, industry standards and conventional unit operations.

The process plant is designed to nominally treat 6 Mtpa of fresh rock and will consist of comminution, gravity concentration, cyanide leach and adsorption via carbon-in-leach (CIL), carbon elution and gold recovery circuits. CIL tailings will be treated in a cyanide destruction circuit and pumped to a tailings storage facility. Figure 17.1 presents the overall flowsheet for the Oko West Project.

The key project design criteria for the process plant are listed below:

- Nominal throughput of 6 Mtpa of fresh rock.
- Crushing plant availability of 70%.
- Grinding, gravity, CIL, gold recovery and tailings handling circuit availability of 92% through the use of standby equipment in critical areas, inline crushed material stockpile and reliable power supply.
- Comminution circuit to produce a primary grind size of (P80) 80% passing 75 µm.
- CIL residence time of 48 hours to achieve optimal gold extraction.
- Cyanide destruction circuit to produce weak acid dissociable (WAD) cyanide levels of less than 10 ppm.
- Sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control if and when required.
- Equipment selection based on suitability for the required duty, reliability, and ease of maintenance.
- Plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

Figure 17.1: Overall Flowsheet



17.2 Process Design Criteria

The proposed process plant will consist of the following unit operations:

- Primary crushing of material.
- Coarse material stockpile and reclaim.
- Grinding consisting of semi-autogenous (SAG) mill and ball mill with hydrocyclones producing a final product P80 of 75 µm.
- Gravity concentration to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening
- Cyanide leaching, and carbon adsorption via a Carbon-in-Leach (CIL) circuit.
- Carbon elution via Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIL tailings using SO_2 / air process.
- Tailings pumping to a tailings storage facility.
- Air and oxygen circuits.
- Water systems (potable water, raw water, gland seal water and process water).

Key process design criteria are summarized in Table 17.1.

Table 17.1: Key Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Nominal Annual Throughput (fresh rock)	t/y	6,000,000
	Nominal Daily Throughput	t/d	16,500
	Crusher Plant Availability	%	70
	Process Plant Availability	%	92
	Average Gold Head Grade	g/t	1.6
	Average Fresh Rock Gold Recovery	%	92.5%
	Average Transition Material Gold Recovery	%	95.0%
	Average Saprolite Gold Recovery	%	96.0%
Crushing & Storage	Crusher Work Index	kWh/t	22.2
	Run of Mine (ROM), Maximum Size	mm	910
	Crusher Circuit Product Size (P80)	mm	125
	Stockpile Capacity (live)	h	12
Grinding	SMC A x b (15^{th} percentile) – Fresh Rock	-	32.2
	Bond Ball Mill Work Index (85^{th} percentile) – Fresh Rock	kWh/t	14.8
	Bond Rod Mill Work Index (85^{th} percentile) – Fresh Rock	kWh/t	17.4
	Grinding Circuit Product Size (P80)	µm	75
Gravity Concentration	Type	-	3 x KC-QS48
	Intensive Leach Reactor	-	CS6000
Pre-Leach Thickening	Thickener Underflow Density	%w/w	42 (high sap) 45 (fresh rock)
	Solids Loading	t/m^2h	0.6
CIL	Residence Time	h	48
	CIL Tanks	-	12
DR	Elution Batch Size (Carbon)	t	10
Cyanide Destruction	Cyanide Destruction Technology	-	SO_2 / air
	Number of Tanks	-	2
	Total Retention Time	h	2

17.3 Process Plant Description

17.3.1 Primary Crushing

Material from the open pit will be transported to the plant by rear dump trucks. The trucks will tip directly to the primary crusher dump pocket. However, if the trucks are not permitted to directly tip into the dump pocket, then the truck load will be dumped onto the ROM pad. The ROM pad will be primarily utilized for short term or emergency storage and material blending as required by the mine plan. ROM material will be reclaimed to the dump pocket by a front-end loader.

Material from the primary crusher dump pocket will feed a gyratory crusher. A rock breaker will be installed to assist in breaking down oversize material retained above the gyratory crusher. Crushed material from the gyratory crusher will discharge to the primary crusher surge bin. An apron feeder and sacrificial conveyor will withdraw the crushed material. A belt magnet at the sacrificial conveyor discharge will recover any trash metal. The sacrificial conveyor will convey crushed material to the stockpile feed conveyor, which will convey material to the crushed material stockpile. The stockpile feed conveyor will be fitted with a weightometer to monitor the primary crusher throughput and to control the apron feeder variable speed drive (VSD).

The crushing circuit will be serviced by a single dust collection system consisting of multiple extraction hoods, ducting, and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor.

17.3.2 Material Stockpiles

The stockpile area has provision to store crushed fresh rock from the gyratory crusher and waste material for construction purposes. The stockpile feed conveyor will be a radial stacker that can feed multiple stockpiles. The fresh rock stockpile will have a live capacity of approximately 9 kt (equivalent to 12 hours of mill feed). Two (2) reclaim apron feeders located underneath the stockpile will be installed with variable speed drives (VSDs) to control the reclaim rate feeding the grinding circuit. Two (2) additional apron feeders will be installed for a second stockpile. Each apron feeder will be sized for 500 t/h. During the initial production years, saprolite material will be directly fed to the mill via these apron feeders. When the operation transitions to 100% fresh rock feed, both stockpiles will store crushed fresh rock and have a total live capacity of 24 hours.

17.3.3 Grinding

Reclaimed material from the stockpile will feed a 9.75 m diameter by 5.7 m effective grinding length (EGL) SAG mill via the SAG mill feed conveyor. The SAG mill will be installed with a 10,800-kW synchronous motor and a VSD to control the speed of the SAG mill. A belt-scale on the SAG feed conveyor will monitor the feed rate. Process water will be added to the SAG mill to maintain a 70% slurry discharge density. SAG mill discharge will pass through a screen to remove grinding media scats and a small number of pebbles. The SAG screen undersize will report to the cyclone feed pump box, combining with ball mill discharge. SAG screen oversize will be conveyed back to the SAG mill feed conveyor.

Slurry from the cyclone feed pump box will be pumped to a cyclone cluster of 18 (15 operating / 3 standby) 508 mm hydrocyclones for size classification. The cyclone overflow, at a final target product P80 of 75 µm, will flow via gravity to the trash screens prior to the CIL circuit. The hydrocyclones have been designed for a 350% circulating load.

Cyclone underflow will feed a 7.32 m diameter by 10.20 m EGL ball mill with an installed 10,800 kW fixed speed motor. Slurry will overflow from the ball mill to a trommel screen, attached to the ball mill discharge end. Trommel undersize will discharge into the cyclone feed pump box.

A portion of the cyclone feed will feed a gravity separation circuit for coarse gold recovery.

17.3.4 Gravity Gold Recovery

The gravity recovery will consist of three (3) centrifugal gravity concentrator units equipped with a feed screen and an intensive cyanidation unit. The gravity gold recovery unit will be located in a secured area within the grinding area structure.

Gravity feed slurry will be screened using a vibrating screen to remove +2 mm material. The oversized material will overflow directly to the ball mill feed. The undersize product will feed a KC-QS48 or equivalent gravity concentrator. Gravity concentrator tailings will discharge into the cyclone feed pump box.

Periodically, the centrifugal concentrator will be bypassed and switched to flushing mode using fresh water to recover the collected concentrate. The collected concentrate will be pumped to the intensive leach reactor unit (ILR).

The gravity concentrate will be batch processed in the intensive cyanidation unit in 24-hour intervals. The gravity concentrate will be leached to dissolve gold in a leach solution that includes sodium cyanide, caustic

solution, and a leach accelerant. After the leach cycle is complete, the pregnant solution will be pumped to the electrowinning circuit while the intensive cyanidation unit residue will be pumped to the pre-leach thickener.

17.3.5 Pre-Leach Thickening and CIL

Cyclone overflow will flow to two (2) trash screens in parallel and then gravity fed to a 28 m diameter pre-leach feed thickener to increase slurry density for the downstream cyanidation process. Flocculant will be added to the thickener feed to promote the settling of solids. The thickener overflow will report to a pre-leach thickener overflow tank which is then pumped to the process water tank.

The thickener underflow at 42% w/w solids (for high saprolite feed) or 45% w/w solids (for fresh rock feed) will be pumped to the CIL circuit consisting of 12 x 6,048 m^3 tanks (live volume). The CIL tanks will provide a total retention time of 48 hours, and the tanks will be sparged with oxygen. The CIL tanks will be equipped with inter-stage screens and pumps to advance the loaded carbon upwards to the next CIL tank. Activated carbon will be added into the CIL tanks 11 and 12 and loaded carbon will leave the CIL circuit from the first and second CIL tanks. Activated carbon concentrations will vary between 10 and 20 g/L slurry within the CIL tanks.

Sodium cyanide will be added to the CIL circuit to dissolve the gold and lime slurry will be added to maintain the slurry pH of approximately 10.5 - 11.0.

The loaded carbon will be transferred to the carbon stripping circuit, while the leach residue from the last tank will be sent to a carbon safety screen to recover any carbon fines. The screen undersize will be pumped to the cyanide destruction circuit.

17.3.6 Cyanide Detoxification

The cyanide destruction circuit will consist of two (2) 1,150 m^3 mechanically agitated tanks, providing a total retention time of 2 hours. The conventional SO_2 / air process will be used for cyanide destruction. Treated slurry will flow by gravity to the cyanide destruction tailings pump box for pumping to the tailings storage facility.

The cyanide destruction circuit will treat CIL tailings, process spills from various contained areas and process bleed streams: cold cyanide barren solution effluent, acid wash effluent and area sump pump discharge.

Oxygen will be sparged into the cyanide destruction tanks. Hydrated lime will be added to maintain the pH of 8.5 and copper sulphate will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a source of SO_2. The process will reduce WAD cyanide in solution to 10 mg/L. The total cyanide and WAD cyanide level in solutions will eventually drop to lower levels via natural degradation and dilution from rainwater in the tailings storage facility.

17.3.7 Acid Wash and Elution

Loaded carbon from the CIL circuit will be pumped and screened to the acid wash column where it will be treated with hydrochloric acid to remove inorganic foulants such as calcium, magnesium, sodium salts, and silica. The carbon will first be rinsed with fresh water. Acid will then be pumped from the acid wash circulation tank to the acid wash column and then pumped upward through the acid wash vessel and overflow back to the acid wash circulation tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities. Fresh acid will be pumped from drums into the acid wash tank when required.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into one of the elution vessels using recycled carbon transfer water. Carbon slurry will discharge directly into the top of one of the elution vessels.

The carbon stripping (elution) cycle will utilize barren solution to strip gold rich carbon to create a pregnant solution. The strip circuit will be equipped with two (2) strip columns that can hold 10 t of carbon each to allow more than one (1) strip per day, depending on the feed to the plant. During the strip cycle, solution containing approximately 2.0% hydroxide and 0.2% sodium cyanide, at a temperature of 150°C and 500 kPa will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution. The heated barren solution will then be heated again through the primary heat exchanger using heated water to bring the solution to its final temperature.

The hot barren solution will then be pumped into the elution column through the carbon bed and recirculated multiple times creating a pregnant solution. A barren solution tank will store barren solution, and a pregnant solution tank will store pregnant solution. The elution column can also be used as a cold strip circuit to remove copper from carbon if copper levels are too high.

17.3.8 Carbon Regeneration

Once stripped of gold, transport water will transfer the carbon from the elution vessel to the carbon dewatering screen. The screen acts as both a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. A 600 kg/h diesel fired kiln will be utilized to treat 10 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge will be transferred to the carbon quench tank by gravity, cooled by process water and stored in the regenerated sized carbon tank prior to being pumped back into the CIL circuit.

To compensate for carbon losses by attrition, fresh carbon is added to the carbon pre-attrition tank along with fresh water to mix and activate the carbon. The fresh carbon will then drain into the regenerated carbon tank.

17.3.9 Electrowinning and Gold Room

The pregnant solution generated from the elution column will be pumped to two (2) electrowinning cells from the pregnant solution tank. These cells will operate on a single-pass basis to produce a gold sludge. The barren solution will be collected in the barren solution pump box where it will be pumped to the barren solution tank.

The primary flow from the barren solution pump box returns the solution to the elution circuit where it will be reused as barren stripping solution for the elution column.

The pregnant solution generated by the intensive cyanidation unit in the gravity circuit will be pumped to a separate pregnant solution tank and then be pumped to a dedicated electrowinning cell. Pregnant solution will be recirculated through the dedicated electrowinning cell until all gold is deposited onto the electrowinning cathodes.

The electrowinning cathodes will be manually transferred from the electrowinning cells to the cathode washing tank where a high-pressure washer will be used to dislodge gold sludge from the cathode surface. The sludge will be filtered by a filter press. The resulting filter cake will be dried in a drying oven and the resulting filtrate will be pumped back to the barren solution pump box within the refinery.

The dried filter cake will then be transferred manually into the electric smelting furnace with flux materials where it will be batch smelted into gold doré bars and stored in a secure vault.

17.3.10 Tailings Storage Facility

The tailings storage facility will receive tailings from the cyanide destruction circuit. The cyanide content in the tailings will be further reduced by natural cyanide degradation and dilution from rainwater to meet IFC standard prior to release in the environment. Tailings pond supernatant (reclaim water) will be pumped back to the process water tank using vertical pumps on a barge.

17.4 Reagents

Reagents consumed within the process plant will be prepared on site and distributed via various reagent handling and makeup systems. These reagents include sodium cyanide, hydrated lime, hydrochloric acid, sulphuric acid, sodium hydroxide, copper sulphate, sodium metabisulphite, antiscalant, flocculant, and activated carbon.

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate more than the content of the largest tank. Where required, each reagent system will be located within its own containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eye wash stations and showers, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

The reagents will be mixed, stored, and then delivered to the ILR, pre-leach thickener, CIL, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks will be sized to typically handle one (1) day of production. The reagents will be delivered in dry form, except for hydrochloric acid and antiscalant, which will be delivered as solutions.

17.4.1 Sodium Cyanide

Sodium cyanide will be used as a gold lixiviant. The cyanide will be shipped in briquette form by road to site in 18-t ISO containers and stored in the cyanide mixing facility; separate from the reagent storage and mixing facility. The sodium cyanide will be mixed with fresh water to form a cyanide solution for use in the CIL circuit.

17.4.2 Hydrated Lime

Hydrated lime will be used as a pH modifier and will be supplied in dry form in bulk bags. Hydrated lime will be added into a mix tank to prepare a milk of lime slurry before addition into the process.

17.4.3 Copper Sulphate

Copper sulphate ($CuSO_4$) will be used as a catalyst for cyanide destruction. The copper sulphate will be supplied as a dry flake in one (1)-tonne bulk bags and stored in the reagent storage area adjacent to the reagents mixing facility. The copper sulphate will be mixed with fresh water to form a copper sulphate solution ready for use in the processing facility.

17.4.4 Sodium Metabisulphite

Sodium metabisulphite ($Na_2S_2O_5$), also known as SMBS, will be the source of SO_2 for the cyanide destruction process and will be supplied in one (1)-tonne bulk bags as a dry reagent. SMBS will be stored in the reagent storage area where it will be transferred to the mixing facility to produce a SMBS solution prior to use in the cyanide destruction process.

17.4.5 Sodium Hydroxide

Sodium hydroxide (NaOH), also known as caustic soda, will be used as a pH modifier and will be supplied as solid beads in one (1)-tonne bulk bags. Caustic soda will be mixed with fresh water prior to being used in the ILR, gold elution circuit, and cyanide mixing tank.

17.4.6 Hydrochloric Acid

Hydrochloric acid (HCl) will be used to remove inorganic carbonates from carbon in the acid wash process within the elution plant. They will be supplied in drums and stored in the reagent storage area adjacent to the reagent mixing facility.

17.4.7 Flocculant

Flocculant is a liquid polymer that will be used in the thickener to settle solids. It will be supplied in 25 kg bulk bags as a dry reagent. Flocculant will be shipped by road to site, offloaded by forklift, and stored in the reagent storage area adjacent to the reagents mixing facility. Flocculant will be diluted using fresh water and further diluted using an inline mixer with process water prior to being added into the processing facility.

17.5 Plant Services

17.5.1 Plant & Instrumentation Air

Three (3) air compressors will provide plant and instrument air for the process plant. Plant air receivers will act as a buffer storing air to account for variations in demand prior to being distributed throughout the process plant including the oxygen generation plant. Instrument air will be dried before being stored in the instrument air receivers and distributed throughout the plant.

17.5.2 Oxygen Generation

An oxygen generation plant will be used to provide industrial grade oxygen for the CIL and cyanide destruction circuit. The plant air compressors will supply air to the oxygen generation circuit. The oxygen generation plant will include an oxygen plant air drier, a Pressure Swing Adsorption (PSA) oxygen generator, and an oxygen plant receiver.

17.5.3 Fresh and Fire Water

Fresh water will be pumped to the plant fresh / fire water tank by vertical turbine pumps. The plant fresh / fire water tank will serve as a combined storage for both fresh and fire water supply. Fresh water will draw from part way up the tank while the lower section of the tank is held in reserve for a dedicated fire water supply.

The fire water portion of the tank will have minimum capacity of 108 m^3 and will feed the plant and permanent camp fire suppression systems; fire hydrants and hose reels via a fire water ring main. Fresh water in the tank will be used to supply the following services:

- Primary crushing circuit dust suppression water.
- Reagent preparation water.
- Slurry pumps gland seal water.
- Cooling water systems: i.e., elution circuit, mill motor cooling.
- High pressure wash water in the refinery.
- Make-up water for the process water system.

Fresh water will be pumped to the fresh / fire water tank through multimedia filter to remove particulates.

17.5.4 Potable Water

Feed to the potable water system is supplied from wells using vertical well pumps. The water will be treated in a vendor-supplied potable water plant to produce potable water for the process plant and camp facilities distribution. The potable water will be used in the process plant for safety showers and washrooms.

17.5.5 Gland Seal Water

Water for the gland seal water system will be supplied by fresh water from the fresh / fire water tank and cooling water returning from the elution circuit cooling heat exchanger. The gland seal water tank will store and distribute gland water to the plant with gland seal water pumps in a duty-standby configuration.

To prevent particulates from causing damaged gland seals throughout the plant, the water feeding the gland water tank will pass through 25-micron particulate filters.

17.5.6 Process Water

Process water with comprise of pre-leach thickener overflow and tailings reclaim water. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty – standby configuration.

17.6 Metallurgical Accounting

Several samplers will be provided throughout the plant to generate composite shift samples from key process streams. Two (2) types of sampling will be performed, metallurgical and process control sampling.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams will be equipped with metallurgical samplers:

- Primary cyclone overflow.
- CIL Tailings.

The metallurgical samplers will sample feed and tailings product which will allow an accurate metal balance of the plant to be completed.

Process control sampler will generate samples used to monitor unit processes in the plant. The process control samplers will be used to generate shift composite samples on process streams that will provide plant operation performance data.

The following process streams will be equipped with process control samplers:

- Leach feed.
- Leach tailings.
- Final tailings.
- Pregnant solution to electrowinning.
- Barren solution after electrowinning.

All samplers will produce 5-10 L of slurry that can be transported to the assay laboratory for further analysis.

A weightometer on the stockpile feed conveyor will measure primary crushed ore tonnage, and a weightometer on the SAG mill feed conveyor will determine mill feed tonnage.

A manual belt cut sampling point on the SAG mill feed conveyor will allow for the collection of a mill feed head grade sample for cross-checking with the calculated head grade. This sample will also be utilized to establish the moisture content of the mill feed.

Regular surveys of the gold and silver in circuit will allow a reconciliation of precious metals in the feed compared to doré production.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates to unit operations will be measured and accumulated using flowmeters.

17.7 Plant Control System

The following provides a broad overview of the control strategy that will be employed for the process plant.

The general control philosophy for the process plant will be one with a moderate level of automation and remote-control facilities to allow critical process functions to be carried out with minimal operator

intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, located in the process plant, will house PC-based operator interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

Additional OITs will be provided for data logging and engineering / programming functions.

A field touch panel will be installed in the feed preparation area to allow local operator control of the crushing plant to facilitate ease of operation for rock breaking and stockpiling if required. A second field touch panel will be installed in the elution area to allow local operator control of the elution sequence. A third field touch panel will be supplied for the grinding circuit area.

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA-based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run, and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons that will be hard-wired to the drive starter. Plant drives will predominantly be started by the control room operator after the equipment has been inspected by an operator in the field.

The OITs will allow drives to be selected to Auto, Local, Remote, Maintenance or Out-of-Service modes via the drive control popup. Statutory interlocks, such as emergency stops and thermal protection, will be hardwired and will apply in all modes of operation. All PLC-generated process interlocks will apply in Auto, Local and Remote modes. Process interlocks will be disabled or bypassed in Maintenance mode with the exception of critical interlocks, such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton, which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. Maintenance selection will allow each drive to be operated by maintenance personnel in the field via the local start pushbutton, which is connected to a PLC input. A

PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor-supplied packages will use vendor standard control systems as required throughout the project. Vendor packages will generally be operated locally with limited control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

The use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the PCS.

The PCS will perform all digital and analogue control functions, including PID control, for all non-packaged plant. Faceplates on the PCS displays will facilitate the entry of set-points, readout of process variables (PVs) and controlled variables (CVs), and entry of the three (3) PID parameters (proportional, integral and derivative).

The majority of equipment interlocks will be software configurable. However, selected drives will be hard wired to provide the required level of personal safety protection (e.g., the emergency stop buttons associated with every motor and the pull wire switches associated with conveyors).

All alarm and trip circuits from field or local panel-mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of mains power supply, the equipment will return to a de-energized state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated for non-packaged plant equipment, except for the elution circuit. However, in any process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g., trip of SAG mill feed conveyor will result in stop of the upstream apron feeder). Standard vendor packages may include automatic sequence start / stop controls within the vendor package only.

17.8 Plant Consumption

17.8.1 Energy

The power demand for the process plant, along with the rest of the project, will be provided by a dedicated power plant. The power demand for the future operation is discussed in Section 18.7.

17.8.2 Reagents & Consumables

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Reagents and consumables usage are summarized in Table 17.2 and Table 17.3, assuming 100% fresh rock.

Table 17.2: Reagents Consumption

Description	Delivered Form	Average Usage (kg/t)
Sodium Cyanide	Briquette ISO tank	0.40
Lime (@90% CaO)	1T bag	1.35
Hydrochloric Acid (32% strength)	1,000 L IBC	0.06
Sodium Hydroxide	1 t bags (dry)	0.03
Copper Sulphate	1 t bags (dry)	0.10
SMBS	1 t bags (dry)	1.0
Flocculant	25 kg bags (dry)	0.01
Activated Carbon	500 kg bags (dry)	0.04

Source: GMS, 2024

Table 17.3: Consumables Consumption

Description	Delivered Form	Usage
Gyratory Crusher – Mantle	lot	1.5 sets / year
Gyratory Crusher – Concave Segments	lot	1.5 set / year / segment
SAG Mill Liners	lot	2.0 sets / year
Ball Mill Liners	lot	1.5 sets / year
Cyclone – Body	lot	1.0 sets / year
Cyclone – Vortex & Spigot	lot	4.0 sets / year
Trash Screen Panels	lot	2.0 sets / year
Loaded Carbon Screen Panels	lot	1.3 sets / year
Barren Carbon Screen Panels	lot	1.3 sets / year
Carbon Safety Screen Panels	lot	2.0 sets / year
Interstage Screens Panels	lot	2.0 sets / year
SAG Mill Grinding Media (125 mm)	bulk	0.45 kg/t
Ball Mill Grinding Media (50 mm)	bulk	0.60 kg/t

Source: GMS, 2024

17.9 Process Plant Personnel

The personnel for the process plant will consist of management, operations, maintenance, and laboratory. Operating staff will work 11-hour days and night shifts on a 2-week on-1-week off rotation cycle and management will work 12-hour days.

Annual process plant personnel requirements are provided in Table 17.4.

Table 17.4: Process Plant Personnel

Department	Position	Compliment
Process Management	Process Manager	1
	Process Superintendent	1
	Assistants	2
Process Operations	General Foreman	2
	Trainers	2
	Supervisors	6
	Lead Operators	15

Department	Position	Compliment
	Operators	39
	Labourers	12
Process Maintenance	General Foreman - Mechanical	2
	General Foreman - Electrical	2
	Planner - Mechanical	2
	Planner - Electrical	2
	Supervisor - Mechanical	4
	Supervisor - Electrical	2
	Technicians - Electrical	3
	Lead Mechanical	21
	Mechanics	42
	Labourers	12
Metallurgical & Assay Laboratory	Senior Metallurgist	1
	Metallurgist	1
	Technicians	3
	Chief Chemist	1
	Chemists	1
	Assayers	6
	Sample Preparation	15
Total		**200**

Source: GMS, 2024

17.10 Recommendations

The following are recommendations related to the process plant:

- Finalize SAG and ball mill sizing once further comminution test work has been completed.
- Finalize gravity, CIL and cyanide destruction circuit sizing once further metallurgical test work has been completed.
- Optimize process plant reagent consumption by material type once further metallurgical test work has been completed.

18 PROJECT INFRASTRUCTURE

The Project infrastructure is designed to support the operation of an open pit (OP) and underground (UG) mine feeding a process plant with a nominal 6 Mtpa, operating on a 24-hour per day, 7-day per week basis. It is designed in consideration of local conditions and topography.

18.1 Site Layout

Figure 18.1 illustrates the project site plan, presenting the onsite infrastructure. This plan is designed to minimize environmental impacts, provide secure site access, minimize construction costs, and enhance operational efficiency.

18.1.1 Roads and Drainage

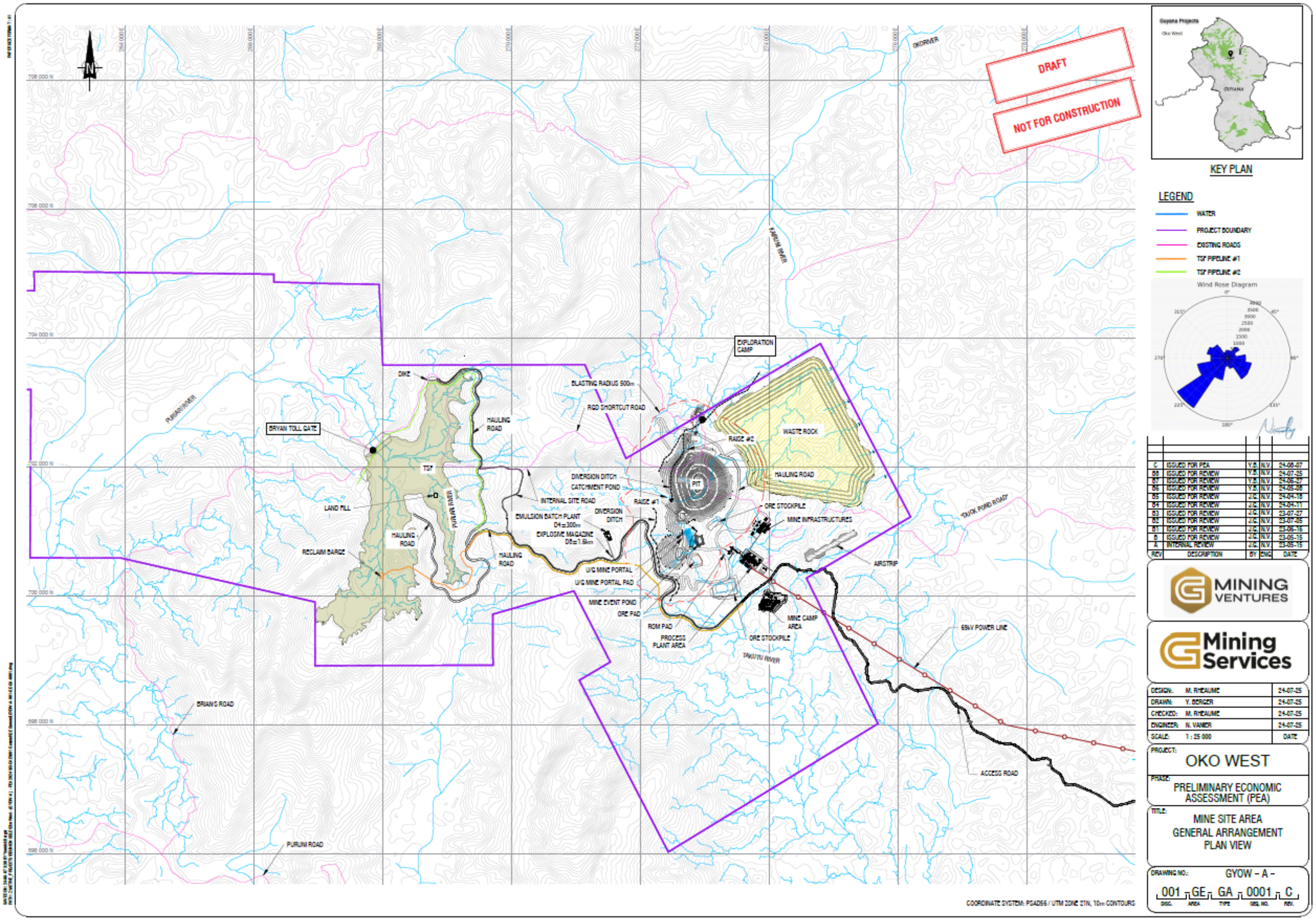
Currently, there are existing roads and a trail network accessible by light vehicles or all-terrain vehicles (ATVs). A new network of gravel-surfaced roads will be constructed to accommodate both light and heavy vehicles. The heavy haul roads surrounding the mine and construction quarry will provide access for large trucks to key facilities, including the truck shops, fuel bay, wash bay, crusher, waste storage area, and tailing dam areas. All other site areas will be accessible via roads designed for light vehicles.

The project site will be accessed mostly with the existing Puruni Road with the addition of a planned 13 km laterite-surfaced road. Upgrades to specific segments of the Puruni road are designed to improve gradients, surface quality, and drainage systems. A short connecting road from the barge landing to the Puruni road will complete the land access to site. To ensure reliable communication along the access route, a comprehensive radio communication system will be implemented, supported by strategically placed repeater towers.

The site will feature approximately 20 km of service roads, interconnecting key infrastructures such as the airstrip, explosives storage facility, tailings storage facility, operations sites, and camp site.

Drainage channels and culverts will be installed to divert water away from critical infrastructure, such as the plant site, process plant, open pit, and waste storage. Plant site pads are designed with a slope directing water away from buildings towards catchment ditches. Surface water from the vicinity of the truck facilities that are suspected to carry hydrocarbons will be collected and diverted to designated oil / water separators.

Figure 18.1: General Site Plan



18.2 Site Infrastructure

The primary buildings have been strategically positioned to facilitate efficient construction access and to leverage the existing topography, thereby minimizing the volume of bulk earthworks. This placement also adheres to geotechnical recommendations, ensuring structural stability and safety.

Infrastructure buildings have been designed as modular prefabricated structures combined with steel and cladding where required. Buildings will be equipped with smoke, carbon monoxide and heat detectors, as well as appropriate chemical fire extinguishers looped together to one (1) main fire alarm panel. All infrastructure will comply with the national building codes.

The camp buildings, such as the Kitchen, Administration Office, Laundry, Recreational Centre, Gym and Ablution Units, will be placed within walking distance. This will optimize the electrical and piping network and create a PPE-free zone, away from the industrial area.

An access gate and guard facility including a search and site access control building will be installed at the site entrance. Preliminary screening of all traffic entering and leaving the property will be conducted at this location.

The control building will house the security access control office and provide control of all personnel entering and leaving site. Only site approved and security-cleared vehicles will be allowed to proceed beyond this point.

A new 850 m-long airstrip, classified as category 2, will be constructed to accommodate most aircrafts in the country. This airstrip will facilitate the transport of senior personnel, critical supplies, medical emergencies and the export of doré.

18.3 Camp Accommodations

18.3.1 Dormitory

The permanent camp is designed to accommodate a construction peak of 1,500 individuals on site reducing to 900 individuals during operations and will cover an estimated area of 9,000 square metres. The buildings will be divided into several units, also referred to as dormitories, each consisting of rooms on a single level. Table 18.1 illustrates the camp capacity.

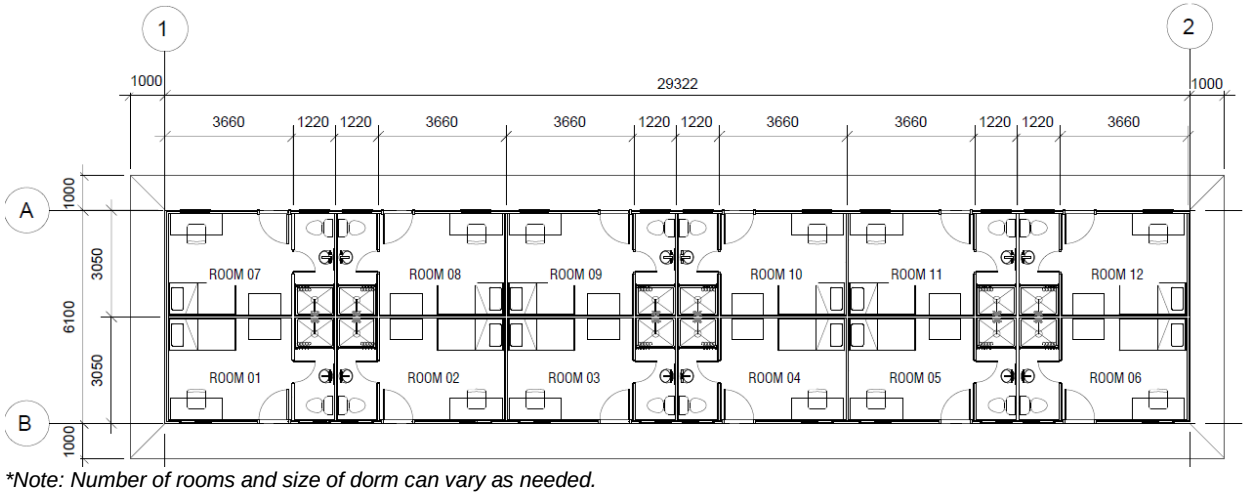
Table 18.1: Camp Capacity

CAMP CAPACITY						
Type	**People**	**%**	**Beds / Building**	**Buildings**	**Rooms / Building**	**Total Beds**
A	72	5%	12	6	12	72
B	240	15%	24	10	24	240
C	1,248	80%	96	13	24	1,248
TOTAL	**1,560**	**100%**		**40**		

**Note: Distribution of beds and number of rooms per building may vary. Quantities showed in the table are based on typical type of dorms.*

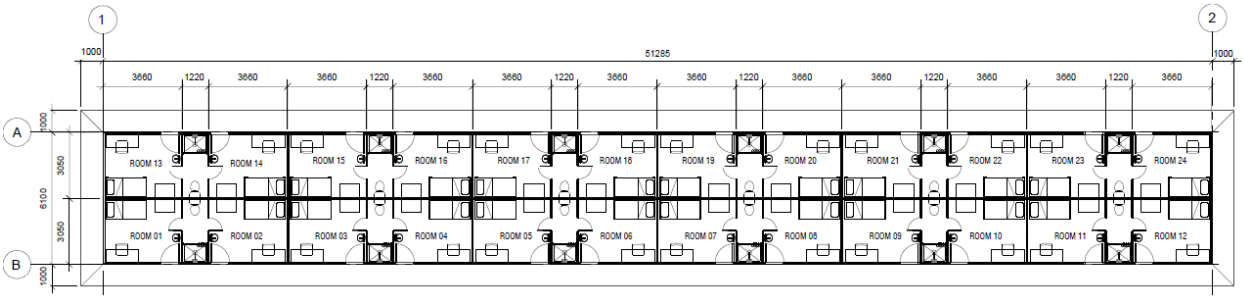
For a Type A building, the capacity is 12 people, each with an individual bathroom, along with one (1) janitor room and one (1) E-room. Type B buildings consist of 24 individual rooms, with one (1) bathroom for every two (2) rooms, plus one (1) janitor room and one (1) E-room. Type C buildings have a maximum capacity of 96 people accommodated in bunk beds with four (4) people per room, one (1) janitor room and one (1) E-room per block. After the construction peak, the bunk beds are eliminated, and the capacity of the Type C building is reduced to 48 beds. Figure 18.2, Figure 18.3 and Figure 18.4 illustrate the typical camp dorms.

Figure 18.2: Typical Camp Dorm – Type A



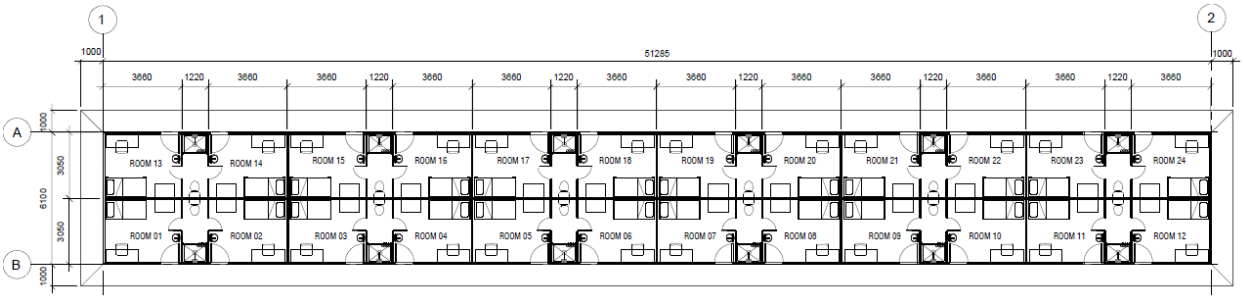
**Note: Number of rooms and size of dorm can vary as needed.*

Figure 18.3: Typical Camp Dorm – Type B



**Note: Number of rooms and size of dorm can vary as needed.*

Figure 18.4: Typical Camp Dorm – Type C



**Note: Number of rooms and size of dorm can vary as needed.*

18.3.2 Kitchen & Lunchroom

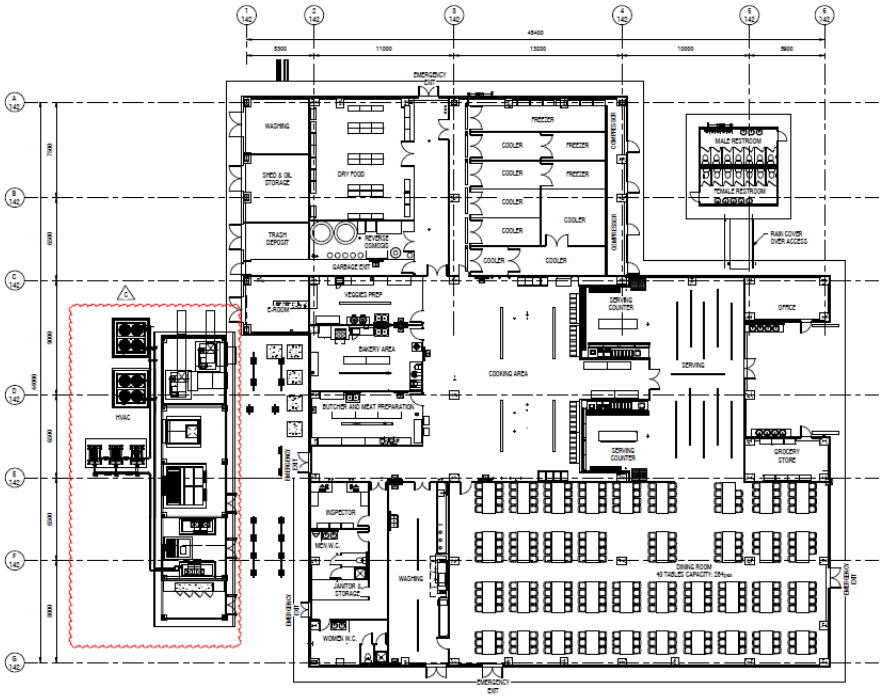
The kitchen and lunchroom design assumes a service for 1,500 people, including cooking areas, serving areas, hygienist zones, staff offices, freezer rooms and a storage area. Cooking stations will have commercial-grade appliances and proper ventilation. Designated meal preparation and serving spaces will manage high volumes, and cleaning zones will comply with health regulations. Offices for management and staff meeting rooms are included. Freezer rooms ensure food safety, and a well-organized storage area is provided.

HVAC equipment will be outside the main building, insulated to minimize noise and heat transmission. A maintenance road will encircle the complex, providing back access to avoid disrupting pedestrian traffic at the main entrance. This layout ensures efficient, safe, and comfortable operations with strategic placement and circulation planning supporting optimal flow. Figure 18.5 and Figure 18.6 illustrate the typical kitchen and lunchroom.

Figure 18.5: Typical Kitchen & Lunchroom – 3D VIEW



Figure 18.6: Typical Kitchen & Lunchroom – PLAN VIEW



**Note: Building capacity can vary as needed.*

18.3.3 Camp Office Welcome Centre and Laundry

The camp office is a single-story building with an approximate footprint of 1,100 m^2. It is located adjacent to the laundry room and features a shared corridor. The building will house offices and open areas for the

Transportation Supervisor and Camp Supervisor, meeting rooms, a janitorial room, a storage room and a washroom. Figure 18.7 and Figure 18.8 illustrate the camp office and laundry area.

Figure 18.7: 3D View of Camp Office and Laundry Area

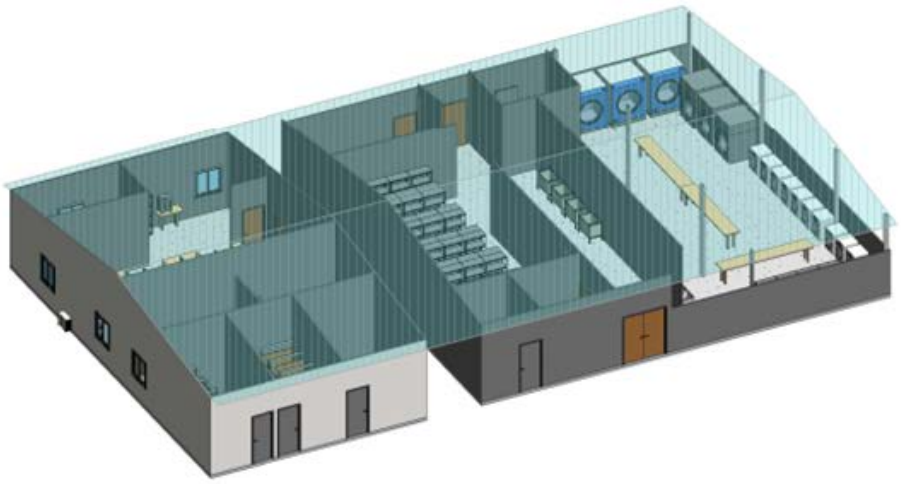
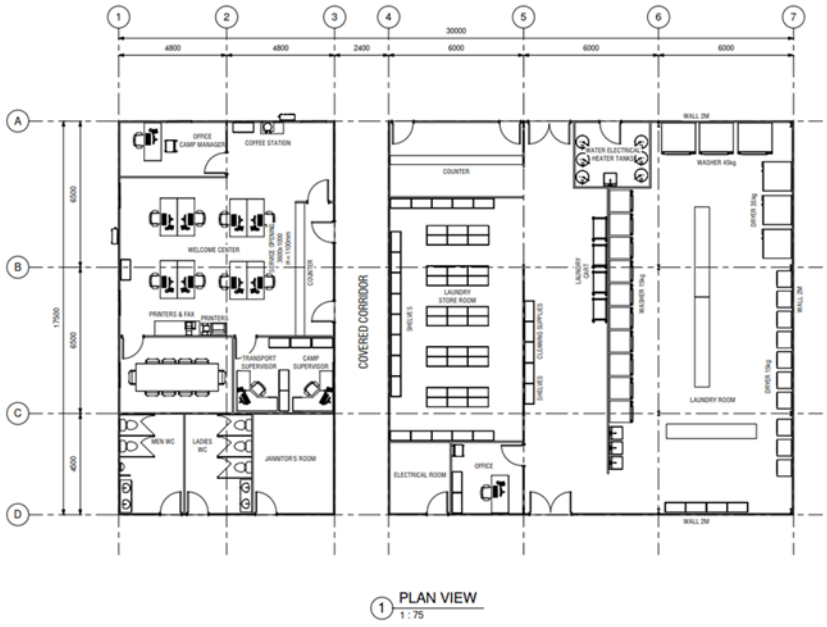


Figure 18.8: Camp Office / Welcome Centre - Plan View



18.3.4 Recreational Centre

The Recreational Centre facilities will include a game area, internet access with shared computers, a TV room, lecture rooms and other amenities.

The gym will feature ablution units, exercise equipment area, aerobic exercise spaces and other fitness activities. Furthermore, recreational areas will include sports fields to provide sufficient space for outdoor activities.

All these facilities will be located in close proximity to the main accommodation area, in order to maintain a prudent distance that minimizes undesirable noise and pedestrian traffic disruptions.

18.3.5 Greenhouse and Nursery

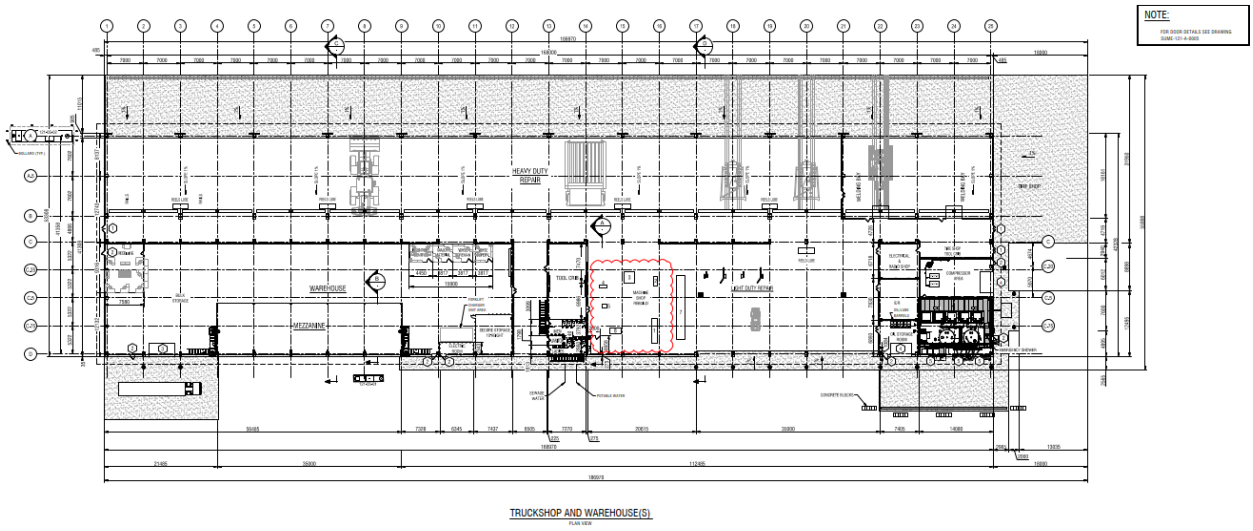
The greenhouse and plant nursery will be constructed using vernacular methods that adhere to local standards and requirements. The primary goal is to offer shade and natural sunlight as needed for optimal plant growth. The nursery will feature multiple external areas distributed across terraces, designed to accommodate various plant species. One of its key services will include an animal rescue area and a veterinary room, emphasizing its role in animal care alongside plant cultivation. Adequate water services will also be provided to ensure efficient irrigation and maintenance of the nursery's ecosystem.

18.4 Mine Infrastructure

18.4.1 Mine Maintenance Facility & Warehouse area.

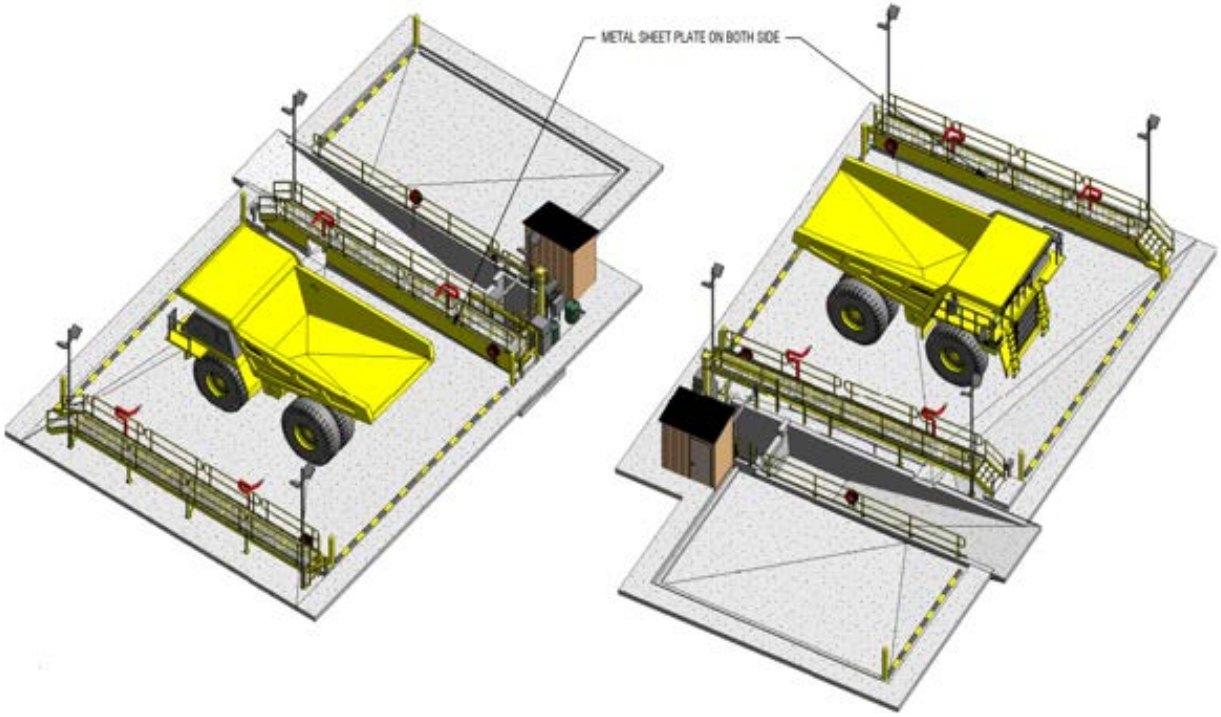
The Mine Maintenance Facility and Warehouse (Figure 18.9) will be located in the Balance of Plant (BOP) area. The mining machinery will have easy access to its restricted sector. The Facility includes ten (10) service bays sized for haulage trucks, five (5) light vehicle bays and two (2) maintenance / welding bays. A lubricant and grease storage sector will include the various products which will be distributed to the various bays with fixed piping and pumps. The truck shop will also include office space, a kitting room, tools storage and a mezzanine with meeting rooms, restrooms and lockers. The building construction is based on a conventional steel structure with overhead crane; insulated sandwich panels will be used for the walls and roof of the building. The warehouse area will be contained in this building and will have all the necessary facilities for its operation.

Figure 18.9: Maintenance Facility and Warehouse Area



A wash bay (Figure 18.10), also sized to service haulage trucks will be located near the maintenance facility. Wash water will be captured, settled, oil separated, and water recycled.

Figure 18.10: Wash Bay

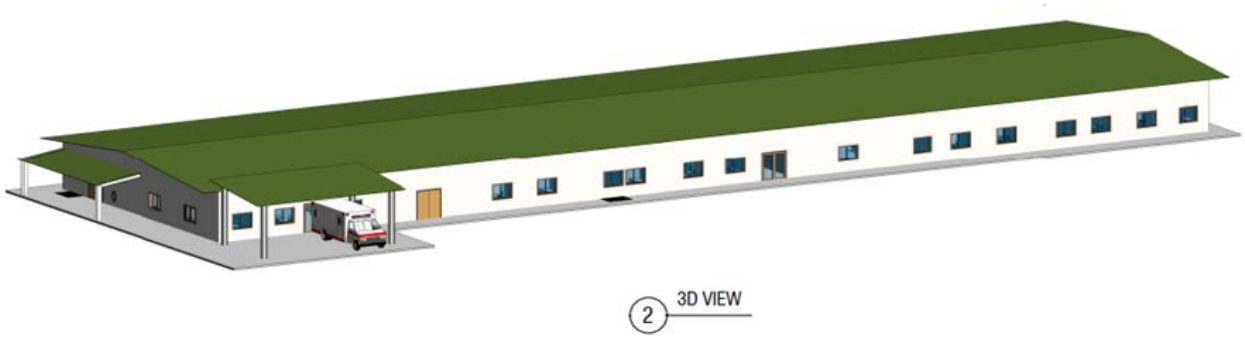


A separate container area will be dedicated to the storage of special lubricants and greases and a controlled area will be provided around flammable products such as solvents and paints.

18.4.2 Main Administration Building

The Main Administration Building, shown in Figure 18.11, has a footprint of 1,762 m^2. The ground slab will be structural concrete supporting a light structure. This one-story building will be used by the site management and general services, mine management and technical services, medical centre and ambulance parking, with other facilities for mid-shift lunchrooms and various meeting rooms.

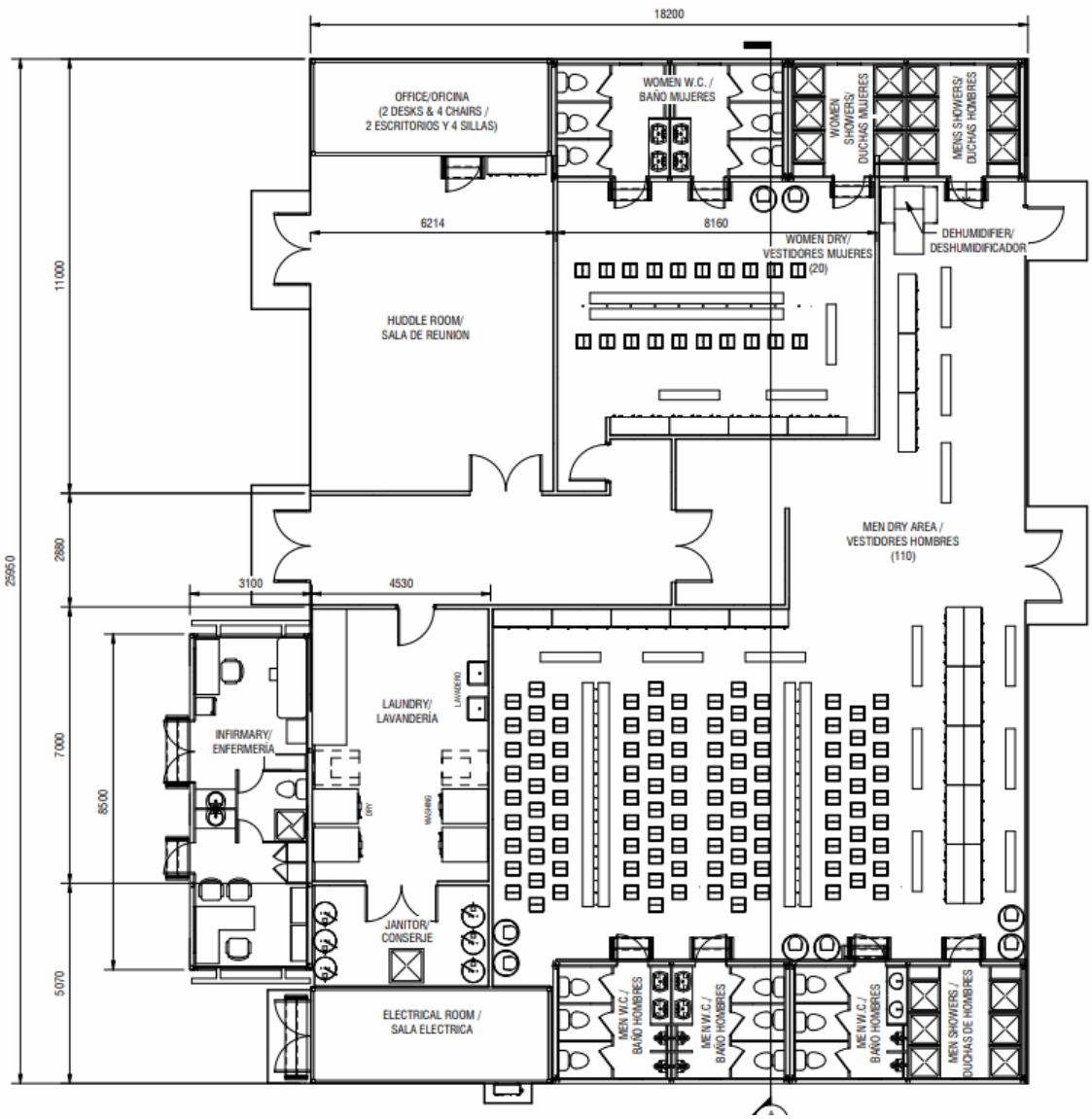
Figure 18.11: Main Administration Building



18.4.3 Mine Dry

The Mine Dry (Figure 18.12) will be built as part of the underground development and will include a dry portion and change room, a mine rescue facility and office space for mine supervision and maintenance.

Figure 18.12: Mine Dry - Plan View



18.4.4 Explosive Storage

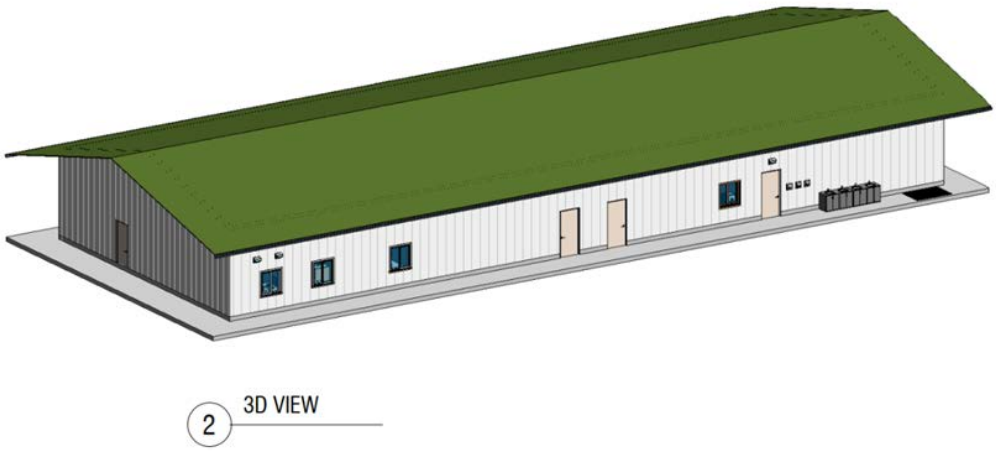
The explosive storage facility is designed for a capacity of 160 t of emulsion using 40 t skid mounted tanks, 18 t of explosives products in a magazine with another magazine for accessories. Storage capacity is sufficient for 30 days at peak consumption.

18.5 Process Infrastructure

18.5.1 Mill Offices

The Mill Office building (Figure 18.13) has an area of 705 m^2. It will have several offices for management, maintenance and operating personnel as well as for the security department. In addition, the building has a meeting room, a lunchroom, and men and women changing rooms and washrooms. The building will have a ground slab of structural concrete and steel structure with insulated panels for walls and roof.

Figure 18.13: Mill Office - 3D VIEW

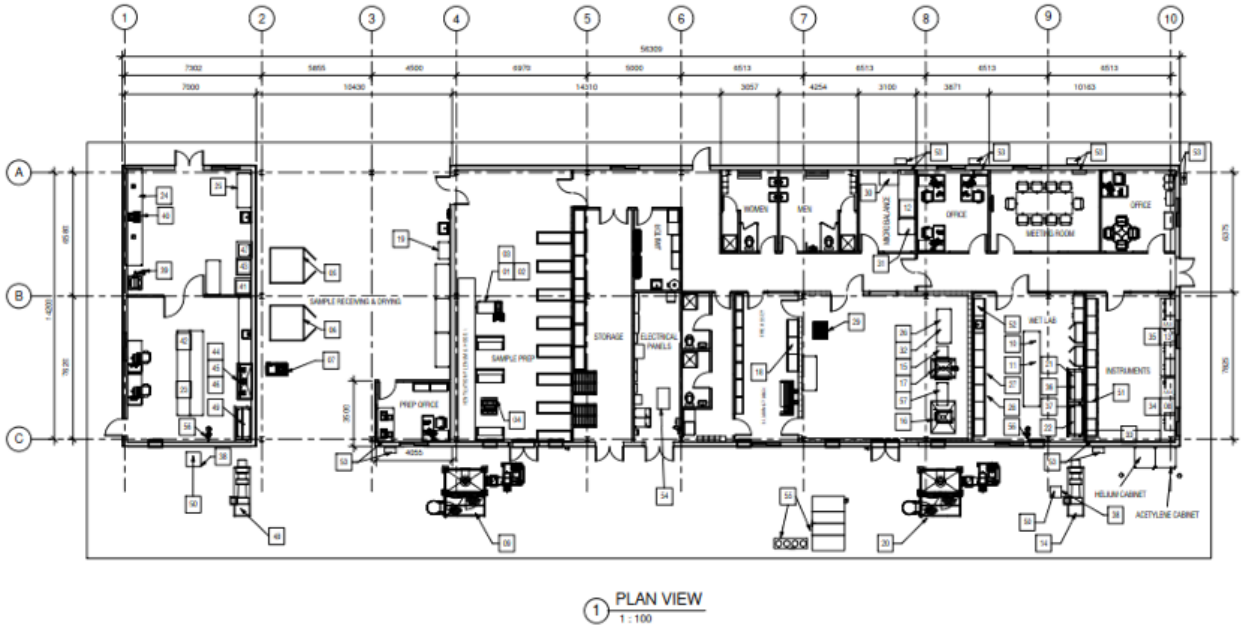


18.5.2 Assay Laboratory

A complete sample preparation and assay laboratory facility (Figure 18.14) will be built adjacent to the process plant. It will include complete lab equipment packages to provide assays for mine grade control, process control, effluent monitoring and other environmental requirements.

This facility will consist of a 14 m x 56 m structural steel building with offices for staff and equipped with an air compressor and storage space for various lab consumables. It will be capable of processing 350 samples per day. The facility will be outfitted with all required lab equipment to perform sample preparation, Fire Assay, XRF, Atomic Absorption (AA), Gravimetry, Leach, settling and wet chemical tests.

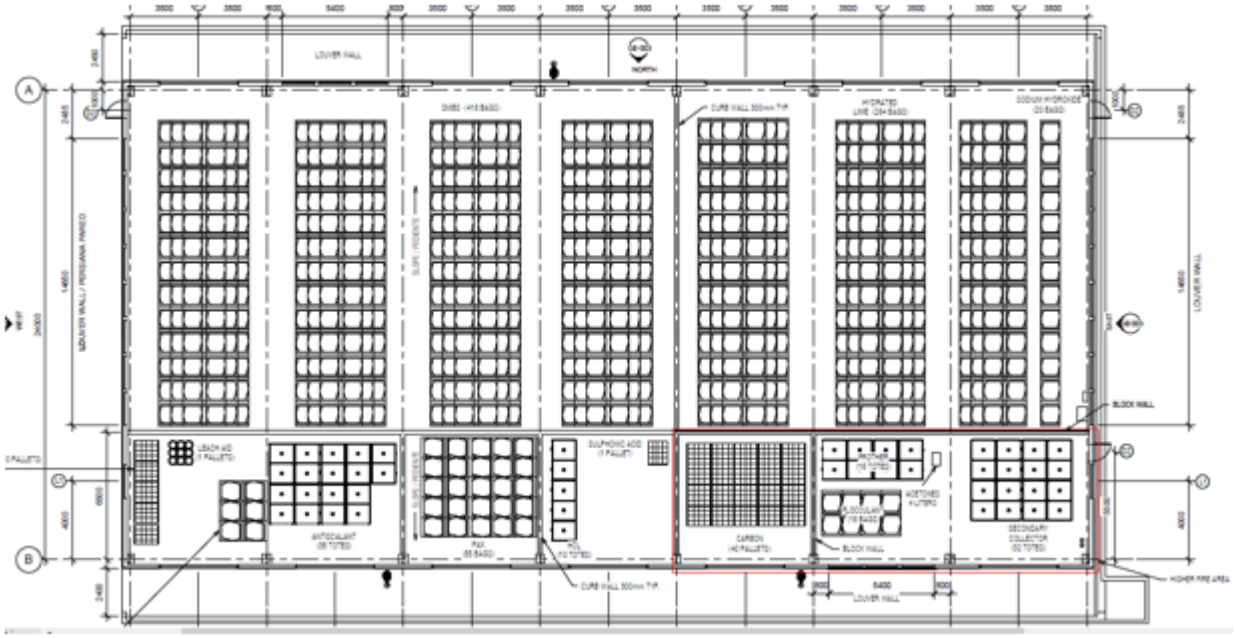
Figure 18.14: Assay Laboratory



18.5.3 Reagent Storage

All process plant reagents, except cyanide and lime, will be stored in a warehouse made of two (2) fabric top buildings on a concrete slab with approximately 1,300 m^2 of covered storage space. The reagents will be segregated by walls or curbs where required to avoid any potential cross-contamination. Liquid spillage in this area will be contained within the building. The floor plan is shown on Figure 18.15.

Figure 18.15: Reagent Storage Floor Plan



A separate area will house the sodium cyanide storage area. Sodium cyanide will be shipped to site and stored in iso-containers. The delivery trucks will return with an empty iso-container for exchange. All contact water in this area will be collected and sent to the process plant.

The reagent storage areas will be within the fenced and gated area of the process plant.

18.6 Waste Storage and Tailings Facilities

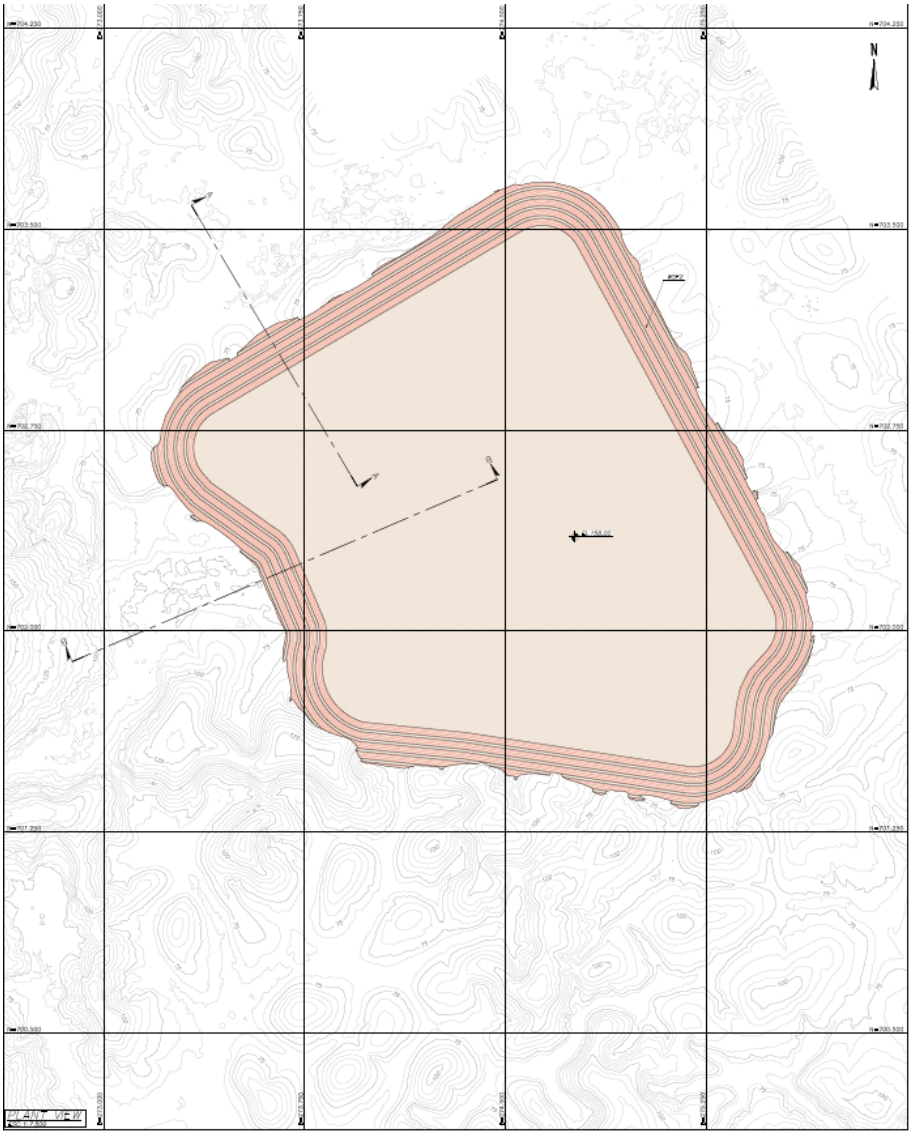
TEC3, a Brazilian geotechnical firm, conducted a comprehensive study to evaluate the technical feasibility of options for the Waste Storage Facility (WSF) and Tailings Storage Facility (TSF). Their objective was to confirm the feasibility of these options and propose design and material balance to determine the required expenditures for the various alternatives.

18.6.1 Waste Storage Facility

Several options for the Waste Storage Facility (WSF) have been evaluated. Based on the proximity of the open pit and the terrain slope for accessing purposes, the current configuration and the main design considerations were selected.

The conceptual arrangement for the WSF was prepared in accordance with the guidelines proposed by Hawley and Cunning (2017) and incorporates good engineering practices. Figure 18.16 illustrates the general arrangement of the WSF.

Figure 18.16: WSF Configuration



The WSF layout extends over an area of 370 ha. The structure has a maximum height of 96 m, with a crest at El. 158 m, resulting in a maximum storage capacity of approximately 257,890,182 m^3. According to the mine waste schedule, the expected volume is 70% of waste rock and 30% of saprolite and transition waste.

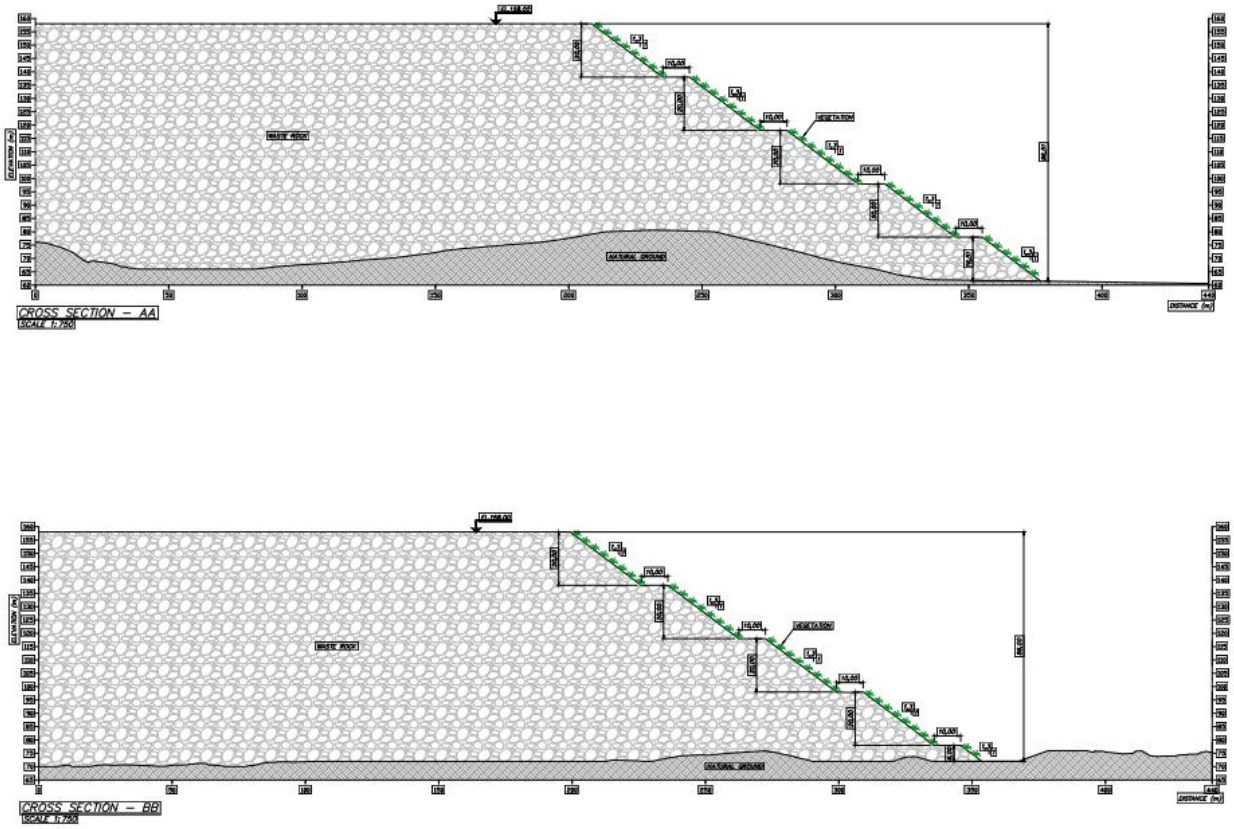
The stack geometry for the final configuration (in waste rock material faces) will consist of 20 m high benches, 10 m wide berms, and a side slope of 1V:1.3H. The main characteristics of the WSF-2 are summarized in Table 18.2, while

Figure 18.17 illustrates the geometry of the WSF for the final configuration.

Table 18.2: Characteristics of WSF

WSF Summary		
Footprint Area (ha)		370.3
Maximum Height (m)		96
Crest Elevation (m)		158
Volumetric Capacity (m^3)	Total	257,890,182
	Soil Sector	77,367,054
	Rock Sector	180,523,127
Bench High (m)	Soil Sector	10
	Rock Sector	20
Berm Width (m)	Soil Sector	10
	Rock Sector	10
Side Slope (H – Horizontal; V – Vertical)	Soil Sector	2,0H:1,0V
	Rock Sector	1,3H:1,0V
Global Slope (o)	Soil Sector	19
	Rock Sector	29

Figure 18.17: Final Configuration of WSF



At closure, the final face of the slope will be revegetated to reintegrate it into the natural surroundings. Slope angles and berm geometries allow for proper revegetation after adding a soil layer over the WSF faces.

The WSF will be constructed using the ascending method with compaction induced by equipment traffic during the waste material disposal and spreading stages. For the surface drainage of the waste rock pile, the characteristics of the materials were considered, most of which were made up of 70% of blocks of rock. These materials generate a surface on which the portion of water originating from precipitation infiltrates almost entirely into the pile. As a result, significant flows are not expected to occur on the surface and no surface drainage structures were considered.

The water infiltrating the waste dump is drained to the foundation land and directed naturally, depending on the topographic surface, to the valley areas where the WSF will be formed. The underdrains were designed considering that 70% of the waste dump is rock and 30% is saprolite waste.

Additionally, there is a sediment retainment system designed to retain sediments generated during waste disposal and other mining activities carried out in their area of contribution. The sediment retention structures must have sufficient volume to store the volumes of precipitation and transported sediment.

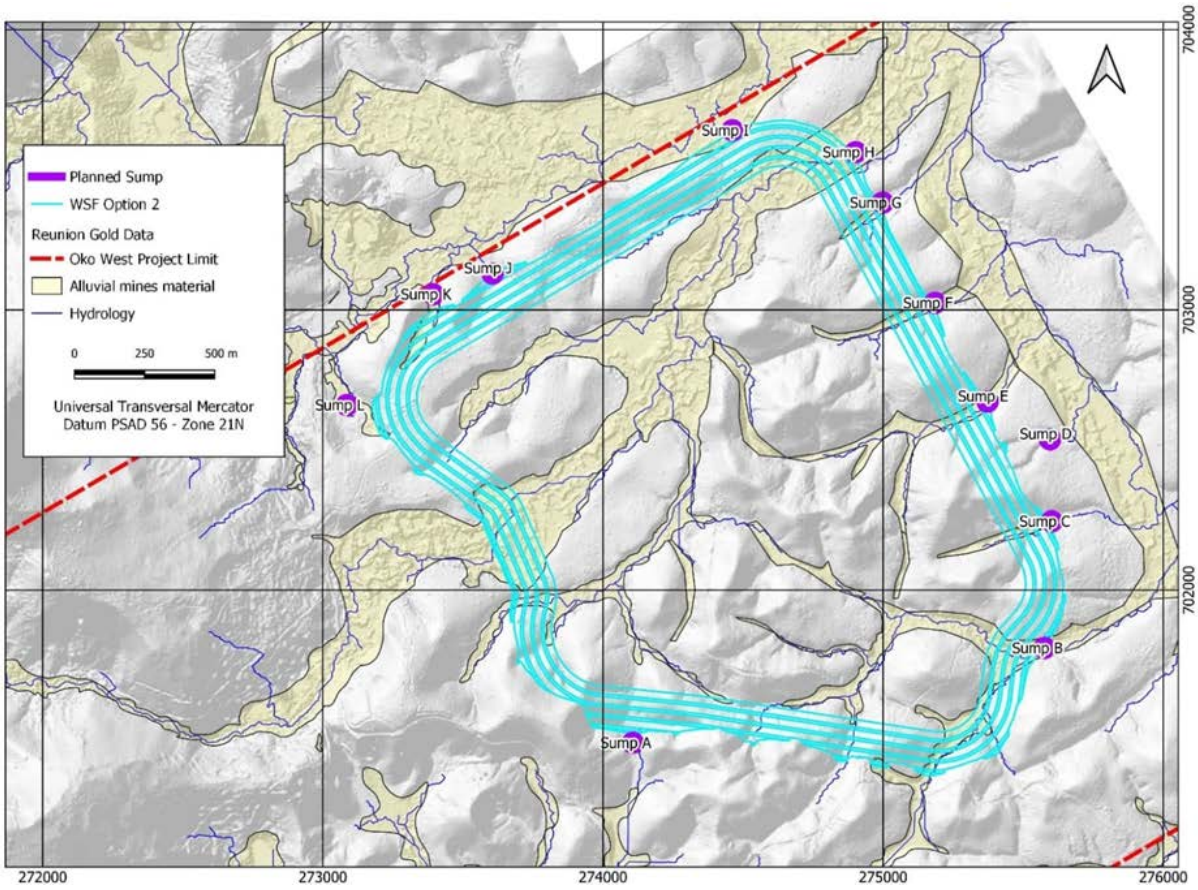
Sediment retention structures will be constructed to retain sediments generated in mining activities carried out in their area of contribution. These structures will consist of sumps excavated in the natural terrain downstream of the pile. The sediment retainment structure will have sufficient volume to store the volumes of sediment generation.

The volume required at the overflow threshold level corresponds to a storm storage capacity corresponding to a 2-year return period and 24-hour duration, plus the volume of sediment generated in one (1) year. The sediment generation rates for surrounding areas adopted for the reservoir design purposes will be as follows:

- Areas with exposed soil in natural terrain: 100 m^3/ha.year.
- Areas with rock stack: 30 m^3/ha.year.
- Areas with remaining vegetative cover: 2 m^3/ha.year.

The sumps outlets will be designed for the 1,000-y storm and the critical duration event for the operation and closure phases. Sumps are in all drainages downstream to the WSF, as shown on Figure 18.18.

Figure 18.18: Sumps Planned Locations of the WSF

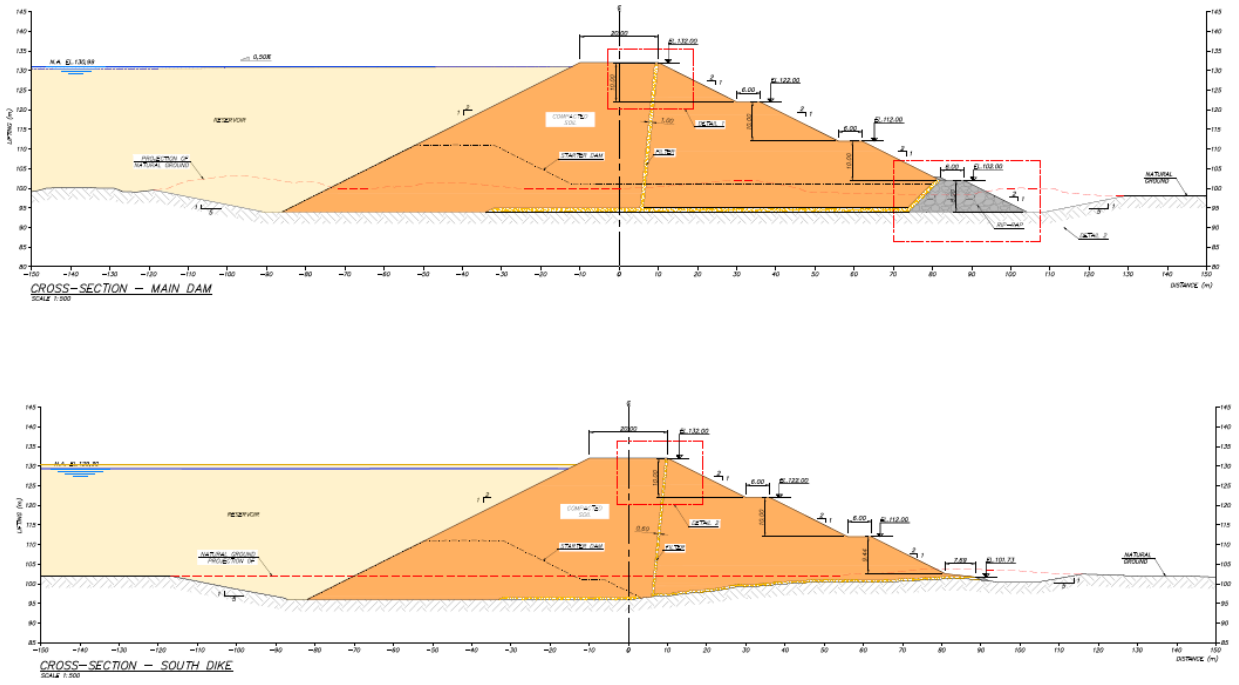


18.6.2 Tailings Storage Facility

Several options for the Tailings Storage Facility (TSF) have been evaluated. Based on the topography, land tenure, environmental impact and proximity to the processing plant, the current configuration and the main design considerations were selected.

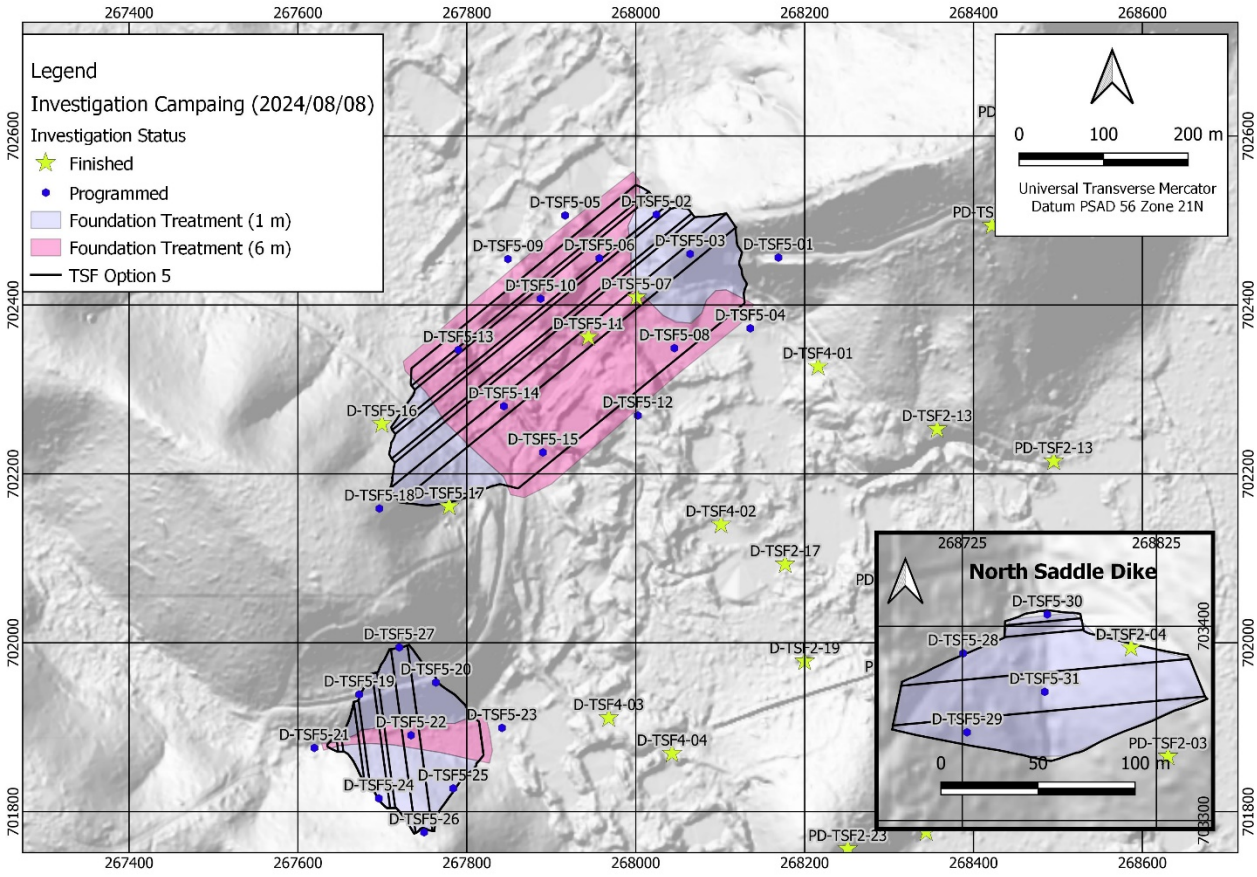
The TSF is composed of three (3) embankments: the main, the secondary, and the saddle dikes. Figure 18.19 shows the typical cross-sections anticipated for these embankments, which were used to estimate the quantities.

Figure 18.19: Typical Dam Sections



Borehole data were used to determine the thickness of materials unsuitable for the TSF foundation. These materials include alluvium / mining tailings, topsoil and residual soils with low nSPT values. Excavation thicknesses of 6 m in the valley area (alluvium) and of 1 m in the abutment regions (topsoil removal) were estimated (Figure 18.20).

Figure 18.20: TSF Embankments and Anticipated Foundation Treatment Areas



It was assumed that the entire foundation treatment area would be replaced with compacted fill for the starter dam fill. This approach is intended to prepare the ground for subsequent raises (using the downstream method) and prevent waterlogged areas from forming at the structure's base.

The accumulated volumes of fill required for the main dam, secondary dam and north saddle dam are shown in (Table 18.3). The volume values reported to elevation 100 m refer to soft soil removal (foundation treatment).

Table 18.3: Dam Crest Elevation Versus Accumulated Fill Volume for the TSF

Elevation (m)	Main Dam Volume (m^3)	Secondary Dam Volume (m^3)	North Saddle Dam Volume (m^3)	Total Volume (m^3)
100	468,884	35,383	-	504,267
111	631,977	60,494	-	692,471
114 (starter dam)	699,350	72,837	-	772,187
118	844,815	99,705	-	944,521
120	966,936	127,984	-	1,094,920
122	1,037,733	143,183	0	1,180,916
124	1,132,162	164,060	3,351	1,299,573
127	1,293,895	202,331	12,183	1,508,409
129	1,416,562	231,641	19,764	1,667,967
132	1,591,988	294,182	31,238	1,917,409

The TSF reservoir elevation x storage table is shown in Table 18.4.

Table 18.4 :Reservoir Elevation vs Storage Volume

Elevation (m)	Area (m^2)	Volume (m^3)[1]	Elevation (m)	Area (m^2)	Volume (m^3)[1]
94.00	0	0	115.00	2,330,150	20,712,824
94.50	352	58	115.50	2,375,633	21,889,365
95.00	992	394	116.00	2,439,192	23,088,956
95.50	1,649	1,053	116.50	2,487,278	24,321,537
96.00	2,297	2,040	117.00	2,530,126	25,575,947
96.50	3,061	3,379	117.50	2,572,673	26,851,603
97.00	3,838	5,103	118.00	2,622,054	28,148,902
97.50	4,628	7,220	118.50	2,667,091	29,471,609
98.00	7,229	9,739	119.00	2,709,208	30,815,715
98.50	9,472	13,898	119.50	2,751,597	32,180,827
99.00	12,047	19,266	120.00	2,803,459	33,567,650
99.50	15,271	26,054	120.50	2,847,061	34,980,551
100.00	192,075	34,749	121.00	2,889,015	36,414,576

Elevation (m)	Area (m^2)	Volume (m^3)[1]	Elevation (m)	Area (m^2)	Volume (m^3)[1]
100.50	276,630	158,871	121.50	2,930,683	37,869,505
101.00	336,951	311,467	122.00	2,977,051	39,345,440
101.50	405,775	496,310	122.50	3,019,375	40,844,778
102.00	626,477	720,695	123.00	3,060,205	42,364,704
102.50	691,965	1,053,449	123.50	3,101,041	43,905,000
103.00	746,821	1,413,122	124.00	3,145,051	45,465,827
103.50	807,776	1,801,026	124.50	3,185,420	47,048,549
104.00	942,007	2,223,704	125.00	3,225,511	48,651,284
104.50	1,004,755	2,713,643	125.50	3,265,421	50,273,962
105.00	1,052,298	3,228,003	126.00	3,308,428	51,916,830
105.50	1,104,729	3,766,776	126.50	3,349,006	53,581,219
106.00	1,239,785	4,334,600	127.00	3,389,350	55,265,804
106.50	1,297,644	4,969,922	127.50	3,429,693	56,970,515
107.00	1,353,751	5,632,645	128.00	3,471,616	58,695,547
107.50	1,413,368	6,323,973	128.50	3,512,872	60,441,977
108.00	1,543,964	7,047,602	129.00	3,552,482	62,208,314
108.50	1,609,445	7,838,789	129.50	3,592,490	63,994,498
109.00	1,662,150	8,656,779	130.00	3,635,679	65,801,065
109.50	1,716,108	9,501,147	130.50	3,674,191	67,628,622
110.00	1,802,665	10,373,546	131.00	3,711,532	69,475,092
110.50	1,858,262	11,289,817	131.50	3,748,456	71,340,086
111.00	1,906,946	12,231,246	132.00	3,787,382	73,223,545
111.50	1,955,194	13,196,699	132.50	3,823,844	75,126,434
112.00	2,020,372	14,186,865	133.00	3,859,925	77,047,377
112.50	2,073,660	15,211,365	133.50	3,895,782	78,986,262
113.00	2,121,686	16,260,282	134.00	3,932,463	80,943,128
113.50	2,168,560	17,332,797	134.50	3,967,354	82,918,183
114.00	2,233,701	18,428,849	135.00	4,001,800	84,910,445
114.50	2,284,457	19,559,054	-	-	-

The preliminary TSF water balance considers the following inputs:

- Mine schedule.
- Monthly rainfall data available in Hydroweb from Mohdia station (1969-2019).
- Annual runoff coefficient of 0.15 from 2019 data of the Oko SWG-02 monitoring point.
- Annual evaporation of 1,540 mm.
- 80% tailings water reclaim.

The annual freeboard volumes are estimated for a 10,000-y annual rainfall event (4,560 mm) to determine the tailings and flood storage elevation; a 1 m additional freeboard was included in the dam crest elevation. This risk criteria adopted refers to the highest classification as per the GISTM recommendations and will be confirmed during the next phase of the project.

Table 18.5 presents the annualized volumes for the TSF construction, and tailings produced, as well as the incremental volume needed for the embankment along the LOM. In the final design, some of the indicated raisings can be merged (e.g. the Final dam raising) considering the compaction volumes indicated.

The dam raise schedule indicated in Table 18.5 considers that in the year Y-1 of the mine schedule, tailings would be available for deposition. The starter dam elevation indicated (114 m) reflects that need.

On Year 10 of the mine schedule, the final dam crest elevation is reached, and a permanent spillway must be constructed. This spillway is designed for 10,000-y rainfall event. The spillway section was sized from reservoir flood routing studies using the elevation x storage tables and a full contribution of upstream catchment (drainage area of 12.294 km^2), keeping a 1 m freeboard over the highest water level produced varying flood durations from 5 min to 30 days. The spillway crest is placed at the El. 129.30 m and considers a rectangular reinforced concrete 3.0 m wide and 2.7 m high was considered.

Table 18.5: Yearly Embankment and Compaction Volumes During LOM

Construction Year	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Raise	Initial Prep.	Starter Dam	1	2	3	4	5	6	7	8	9	10
Tailings Storage Capacity (10 m^3)[1]	765	6,154	11,538	17,596	23,097	28,321	33,475	38,629	43,782	48,936	54,090	59,244
Reservoir Lake Elevation (m)[2]	104.6	108.7	111.7	114.5	116.8	118.8	120.7	122.4	124.1	125.7	127.2	128.7
Dam Crest Elevation (m)	111.0	114.0	116.0	118.0	120.0	122.0	124.0	125.5	127.0	129.0	131.0	132.0
Total Embankment Volume (10 m^3)	692.5	772.2	850.5	944.5	1,094.9	1,212.2	1,327.5	1,427.5	1,527.5	1,679.4	1,836.4	1,917.4
Annual Compaction Volume (m^3)	692,471[5]	79,716	78,313	94,020	150,400	117,234	115,306	100,002	100,002	151,978	156,978	80,989

**Note:*
[1] Tailings storage needs for the following year, assuming tailings density of 1.3 g/cm^3.
[2] Calculated from tailings volume for the following year added of a 2 million m^3 storage for recirculation.
[3] Calculated from tailings volume added of 10,000-y annual flood volume (4,560 mm), added of a 1 m freeboard.
[4] North Saddle was considered completely built in Y4 (31,238 m^3).
[5] From this volume, 504,267 m^3 were estimated for foundation treatment

18.7 Water Management

18.7.1 Industrial / Fire Water

A pumping station will be installed in a non-contact stream within the boundary of the project which will supply raw water. Raw water will be treated at the water treatment plant to remove suspended solids and perform pH adjustment and disinfection. After treatment, this industrial / domestic water will be used at the camp, process plant and mining infrastructures. Industrial / domestic water will be distributed to the site and process plant from a water tank near the process plant. A secondary tank will be dedicated to fire water. The fire water system will supply the building sprinkler systems and hydrants at the process plant and camp complex. The water required to do the initial commissioning and startup of the process plant will be pumped from the water accumulated in the TSF impoundment.

18.7.2 Potable Water

In the early stages of the project, potable water for all facilities on site will be provided by bottles and jugs. Domestic water will be further treated by an osmosis plant to generate potable water for the kitchen and personal consumption. Testing for wells capable of supplying adequate water quality and quantity will be performed as a possible complement for potable water.

18.7.3 Sewage Treatment and Oil-Water Separation

A sewage treatment plant is planned to treat sewage from the plant and camp. Sewage water will be handled by standard septic tank collection systems and treated using natural breakdown bioreactors prior to discharge. Sewage will be treated, separated, and the liquid discharged. Sludge from the sewage treatment plant will be disposed at an approved location.

Oil-water separation systems with be located adjacent to the likely locations of contamination. Water will be collected from the Maintenance Facility and the diesel fueling station to be redirected to an oil-water separator prior to discharging clean water to the environment. At the wash bay, an oil skimmer will capture any oil before water is reused to clean equipment.

18.8 Fuel Storage and Distribution

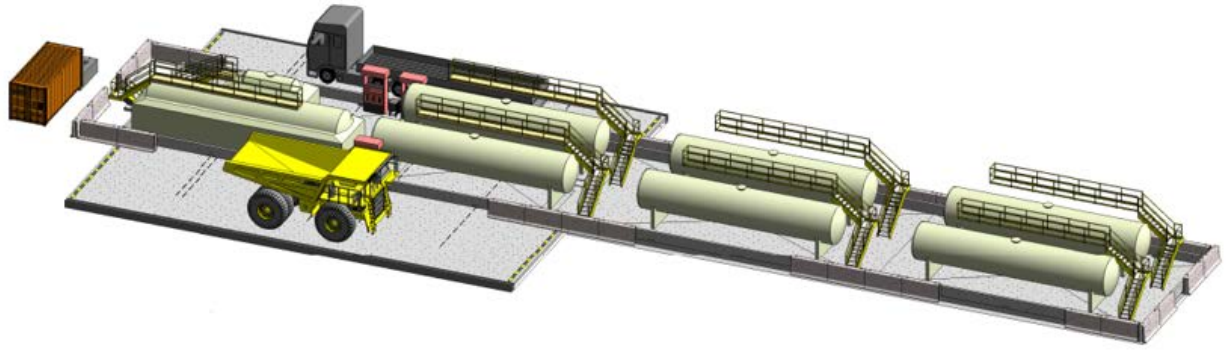
A fuel storage facility will service the mining and site fleet with seven (7) tanks of 60,000 liters diesel fuel for approximately seven (7) days of mobile equipment operations and a 5,000 litres tank of gasoline for light

vehicles. The main tanks will be storing S500 fuel, whereas the last one will store S10 fuel. This infrastructure will service both light and heavy vehicles.

Additional fuel storage will be available at the barge landing, where the power plant is expected to be constructed.

Once the underground mine goes in operation, two (2) additional 60,000 litres tanks will be added to accommodate the increasing needs. One (1) will be for S500 fuel and the other S10. The layout of the fuel storage and distribution is shown on Figure 18.21.

Figure 18.21: Fuel Storage



Source: GMS (2021)

18.9 Power Supply and Distribution

Plant site activities will require an average of 37 MW at its full operation, including the process plant, underground mining, OP mine and BOP infrastructure. Full-power consumption of the plant was benchmarked against similar projects with open pit and underground mining adjusted for processing throughput.

The base case scenario for the project considers the installation of a dedicated HFO-fired power plant. It is anticipated that the power plant will comprise six (6) 9.4 MWe engine generating sets, totalling 56.4 MWe installed capacity and 42.3 MWe running capacity. assuming that one (1) of the generators would be on standby. One (1) additional genset is planned in sustaining capital assuming that one (1) genset would be out of service.

Alternative power supplies will be studied as part of the Feasibility Study.

The base-case power plant will be connected to two (2) 13.8 kV-69 kV step-up transformers, through a medium voltage switchgear. A transmission line of 69 kV will be installed from the HFO-fired power plant at the Barge Landing to the Project site over a distance of 50 km. Two (2) fully redundant on-site transformers will step down the power to 13.8 kV for distribution on site.

The power distribution for the Project will be configured at 13.8 kV, 60 Hertz (Hz) and will consist of substations, step-down transformers, switchgears, motor control centres, powerlines and cableways. The electrical loads of the project will be supplied by voltages of 4.16 kV, 480 V or 208/120 V.

The processing plant will have various satellite electrical rooms. These electrical rooms will serve the following areas:

- Crushing area.
- Grinding / gravity.
- Gold room.
- Cyanide detoxification / plant services.
- Pre-leach / leach / CIL / acid wash / elution / carbon regeneration.
- Ore handling.

Power lines will be used to distribute power to other infrastructure, such as:

- Camp / Communication.
- Administration Building / Assay Lab / Gate House.
- Mine Maintenance Facility / Warehouse / Diesel Fuel Storage / Explosives Storage Facility.
- Various Water Management / Treatment Ponds.
- TSF Tailings and Reclaim Water.
- Water Treatment Plant (WTP).
- Sewage Treatment Plant (STP).

18.10 Communications

Off-site Fibre Optic cable and microwave-based connections with a minimum speed of 1 Gbps will be required to connect the Project. These redundant communications will provide high-speed internet access as required by early works, construction and operation phases. On-site fibre optic cable will connect all site

facilities as the mine is developed, configuring a backbone infrastructure for all computer systems within the Project.

All Core Network and Computer Systems will be allocated at a Data Centre that will require continuous power and air conditioning systems. The Data Centre should be able to allocate at least three (3) 42 u equipment racks.

A trunking-based Radio system will provide robust radio communication with at least 30 Channels that will be mostly used by the various construction sub-teams at the site. A minimum of 250 handheld units and 50 mobile units will be deployed during all stages of construction. Radio and Wireless communications for OP and UG development will be required for safety and tracking purposes. Leaky feeder-based technology and LTE options will be evaluated.

A fibre-based (GPON) network will provide communications to the Camp ensuring communications for all employees and contractors.

Mobile coverage options to ensure 4G / LTE communications will be evaluated during the early stages of construction. In-country ISPs and Mobile Communications vendors will be scrutinized to ensure top quality of service and cost efficiency.

18.11 Offsite Infrastructure

The project infrastructure will also need to be supported by offsite infrastructure as illustrated in Figure 18.22.

Figure 18.22: Offsite Infrastructure Arrangement



The offsite infrastructure critical to the project includes a dedicated landing facility designed to accommodate the barging of equipment and materials from ocean ships to land access and the transportation for the rotating workers, both during construction and operations. The landing facility near the confluent of the Cuyuni and Mazaruni Rivers is strategically positioned to facilitate efficient handling of bulk materials and equipment. It will be designed to accommodate barges on each side of the 70 m long by 20 m wide jetty.

Adjacent to the landing facility, a substantial laydown area will be established to manage and store materials and equipment and aggregate prior to their transportation to the project site. This area will also include dedicated facilities for customs processing, as it will serve as the point of entry in country for all imported materials. By integrating customs processing into the infrastructure, the project aims to minimize potential bottlenecks and delays, ensuring that materials are cleared and available for use with minimal disruption to the project schedule. The streamlined customs process is a critical component in maintaining the overall efficiency of the supply chain.

This laydown area is designed to accommodate large volumes of materials. The layout facilitates efficient inventory management and ease of access, with designated zones for different types of materials and equipment. The area will be surrounded by fencing and equipped with security measures to safeguard assets. It will be supported by a range of lifting and mobile equipment capable of lifting heavy and oversized items, as well as mobile forklifts and loaders for smaller material and routine tasks.

The landing area has been strategically designated for the construction of the power plant and storage of light fuel oil / diesel (LFO) and heavy fuel oil (HFO). The HFO and LFO storage tanks will accommodate 5,000 m^3 and 1,000 m^3, respectively. The power plant and landing workforce will require a small camp and maintenance facility.

19 MARKET STUDIES AND CONTRACTS

19.1 Gold Market

Gold is a freely traded commodity in a well-established and mature market, renowned as a safe haven for investors. It is sold daily by banks and traders at a spot price for immediate delivery.

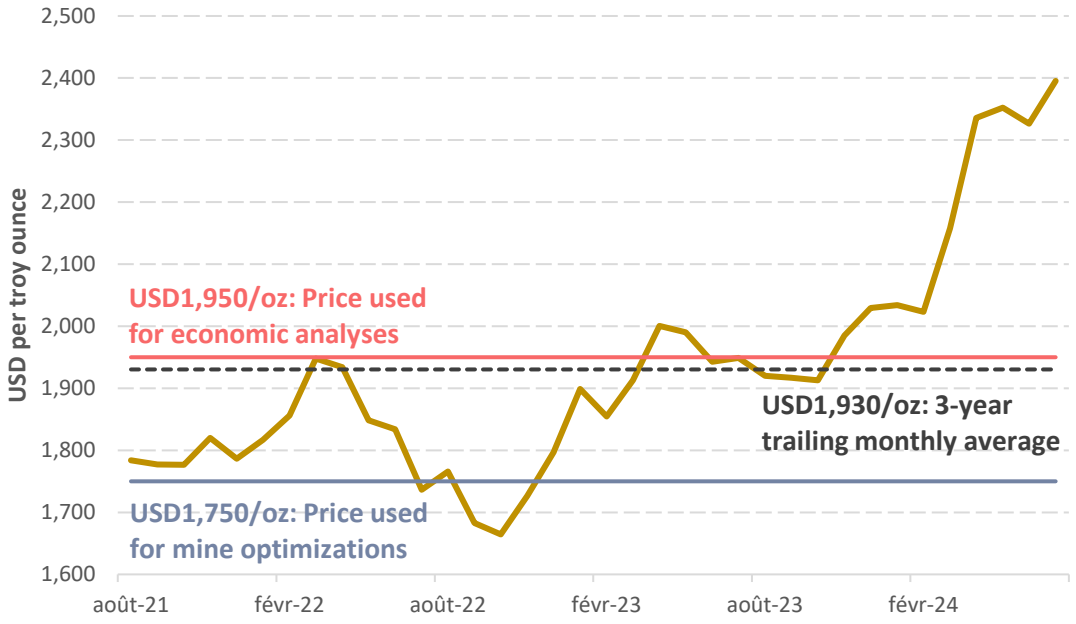
The Oko West Project will produce gold in doré bars. Oko West is authorized to export the gold doré bars from Guyana to an internationally recognized refiner. Prices are typically quoted in US dollars per troy ounce.

19.2 Metal Price

The price of gold is the primary factor in determining the profitability and cash flow from operations. The project's financial performance is closely linked to the gold price. The gold price was determined based on historical prices and the consensus of long-term estimates from banking analysts. The long-term consensus price for August 2024 as published by CIBC Global Mining Group is USD 1,957 per troy ounce. The long-term gold price assumption used in the PEA is USD 1,950/oz Au, in line with analyst consensus commodity price forecasts.

Figure 19.1 shows the historical monthly average value of gold for the last three (3) years. As of July 31, 2024, the five (5)-year average stands at USD 1,847 per troy ounce, while the three (3)-year trailing average is USD 1,930 per troy ounce.

Figure 19.1: Monthly Average Gold Price



Source: World Gold Council

19.3 Contracts

Transportation and refining contracts for gold doré bars will be negotiated and finalized during the construction phase of the project. Preliminary costs for transportation and refining are quoted in the USD 8-10 per ounce range.

G Mining Ventures has entered into a Master Service Agreement with G Mining Services, a Canadian engineering and mine development firm specializing in the mining sector, for the engineering and construction management of the Oko West Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Environmental Resources Management (ERM) is the author of Section 20 of the Technical Report and Mr. Derek Chubb, Senior Partner at ERM, is the QP responsible for this section. This Section discusses the relevant information on the environmental, permitting, and social or community factors related to the Oko West Project. The Oko West Project is in the early stages of exploration and development planning and, as such, this section focuses on recommended future work required to further advance the identification and effective management of potential material environment and social risks and opportunities as Project design advances.

Reunion has engaged the international consultancy Environmental Resource Management (ERM) - which has a registered Guyana entity - to conduct environmental baseline studies (EBS) of the Project area and to prepare the environmental and social impact assessment (ESIA) in accordance with local requirements in Guyana. ERM's team includes several subject matter experts from the University of Guyana's Centre for Study of Biological Diversity and professionals from other Guyanese environmental consultancies. Reunion also hired the firm Sustainability Frameworks, LLP of Washington, DC to act as a peer reviewer of the work completed by ERM and to provide advisory services to Reunion.

The information contained in this Section is informed by the studies completed to date, including: a desktop review of available information provided by Reunion, environmental and social baseline data collection, regulatory permitting, as well as recommended work plans for future environmental and social studies.

20.1 Environmental and Social Conditions

Reunion holds a 100 percent interest in the Prospecting Licence (PL) referred to as the "Project" for the purposes of this section. The Project covers approximately 4,400 hectares. Reunion has expressed it has all of the required permits and approvals for its current stage of exploration and these are in good standing. ERM has not undertaken a compliance assessment of the current exploration program.

The Project area has not been identified as a priority area of conservation interest by the Government of Guyana nor does it fall in or near a Guyana Protected Area, a World Heritage Site, an International Union for Conservation of Nature Key Biodiversity Area or an Alliance for Zero Extinction site. The Government of Guyana has not granted formal title to any Amerindian (or Indigenous) lands within the PL; however, there are Amerindian land titles in very close proximity to Project ancillary facilities.

According to the Guyana Geology and Mines Commission, the PL is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders and one (1) group of medium-scale mining and prospecting permits controlled by G2 Goldfields. The potential for cumulative environmental and social impacts will need to be taken into consideration as Project planning advances.

The Project area straddles the Cuyuni-Mazaruni Mining Districts (Guyana administrative Region 7) in northcentral Guyana, South America. The Project area is located approximately 100 kilometres southwest of Georgetown, the capital city of Guyana, and approximately 60 kilometres from Bartica, the capital city of Region 7. Bartica is accessible by a 20-minute direct flight from Ogle airport in Georgetown or by road to Parika and then by boat from Parika to Bartica or Itaballi on the Essequibo River. There are regular boat services between Parika and Bartica.

The Project area is accessible by the Puruni and Aremu laterite roads from the town of Itaballi at the confluence of the Cuyuni and Mazaruni rivers and then along any of several trails that connect the Project area to these two roads. The Project area is also accessible by helicopter. The Project area is situated at elevations ranging from between approximately 60 and 400 metres above sea level. The Puruni Road is the planned main access road for the Project. The electrical line power supply to the Project line is also assumed to be constructed along the Puruni Road corridor. Ancillary infrastructure is in the very early stages of planning.

20.1.1 Baseline Studies

Physical and biological baseline studies have been undertaken between 2022 and 2024 and social baseline studies between 2023 and 2024 with environmental studies completed during both, the dry and wet seasons. The overall purpose of these studies has been to collect relevant data needed to inform Project planning, including the identification of potential issues of concern and recommended actions. Studies are ongoing to support continued Project design and regulatory permitting.

The focus to date for environmental baseline studies has been the collection of information in the immediate area of the expected Project footprint. Environmental baseline studies of the ancillary facilities, including the power plant and barge landing area, the main access road, and the electrical power supply corridor were initiated in 2024 with results currently pending. Social baseline studies to date have considered community and other stakeholders who could reasonably be expected to be impacted by the Project and its activities.

The findings of the environmental and social baseline studies completed to date are summarized in the below sections.

20.1.1.1 Physical Baseline

Ambient Air

ERM conducted onshore ambient air quality monitoring program in the 2023 wet and dry seasons.to characterize existing ambient air quality conditions in the Project area.

In the absence of applicable Guyana regulatory guidance, ERM used standard methods for selecting air pollutants to monitor, selecting the monitoring site, and establishing other program parameters. Resulting measurements were compared to the World Health Organization's (WHO) Air Quality Guideline Values (WHO, 2021).

The program was designed and implemented to assess background concentrations of NO_2, SO_2, PM_{10}, and $PM_{2.5}$ during a four-week period during the wet and dry seasons at the proposed Project site. While data recovery was high, field and trip blanks had a wide range of non-zero results that were indistinguishable from the exposed sample media. While the results of the field blanks make it impossible to provide an exact background value, the study demonstrated that exposed sample media did not acquire any measurable NO_2 or SO_2 above that of the blanks. This is expected for this location due to no significant or consistent sources of NO_2 or SO_2 in the vicinity of the project site.

Monitoring for PM_{10} and $PM_{2.5}$ showed background values were well below the WHO's 2021 Air Quality Guidelines and were representative of expected conditions at the project site.

GHG and Carbon Stock Analysis

To summarize the baseline carbon stocks, surveys regarding vegetation and soils were conducted during the dry season in 2023. It is important to note that this was not a full baseline and is instead a preliminary report for Reunion Gold's internal understanding. The Project area was represented by the average of 11 plots established to perform a Forest Inventory. The results of the survey appear consistent with the descriptions of the Forestland in Guyana presented in the REDD+ report (Sampling Design and Implementation Plan for Guyana's REDD+ Forest Carbon Monitoring System (FCMS): Version 2). ERM recommends this baseline study be followed by a more detailed study considering improvement measures to allow for the design of mitigation and restoration measures for the Project.

Ambient Noise

Noise surveys were completed in the 2023 dry season to establish background noise conditions in the vicinity of the future mine site and nearby noise receptors. Noise monitoring procedures for this assessment were developed based on International Finance Corporation (IFC) Environmental, Health, and Safety (EHS) Guidelines for baseline noise assessments (IFC 2007). The measurement program consisted of two (2) one-hour measurements during daytime hours (on separate days) and two (2) one-hour measurements during nighttime hours (on separate nights) at each receptor location. Average measured noise measurements were found to be above IFC nighttime and daytime guidance levels at nearby noise receptor locations. This finding is not unexpected as these are areas of existing human activity.

Soils

Soil Surveys were completed at six (6) locations on the Project during wet and dry seasons in 2022, and at 11 locations during the dry season in 2023. The latter was from undisturbed locations within the potential mine development area, as well as from areas with evidence of disturbance from activities including exploration and prior artisanal mining. Due to the lack of data on soil productivity and environmental quality for the Project site, additional soil samples to assess the chemical characteristics of the soils were collected for laboratory analysis.

Despite the often-favorable physical conditions of the soils (e.g., the soil texture of the soils are generally loams intermixed with gravel), the soil fertility appears generally low. The average pH of the soils in the samples collected showed little variability, with pH ranging from 4.5 to 5.6, indicating acidic soil conditions. The reported concentrations for most heavy metals, polycyclic aromatic hydrocarbons (PAHs), total petroleum hydrocarbons (TPHs), and xylene (BTEX) were below the method detection limits. In the few cases where parameters were detected, the parameters were very low, or the reported concentrations were below the USEPA screening levels for residential and industrial reference benchmarks. Soil quality will need to be taken into consideration for activities such as revegetating disturbed areas as part of mine closure.

Groundwater and Hydrogeology

A hydrogeology survey was conducted during the dry season of 2023, with a purpose to expand on the existing groundwater data collection which was originally established in 2022 as the initial monitoring network and in support of the baseline studies program. Groundwater samples were collected from two (2) sampling points during the wet season and four sampling points during the dry season in the 2022 survey.

A total of 14 monitoring wells were drilled and installed around the Project area during 2023. Groundwater samples were collected from seven (7) of those monitoring wells and seven (7) monitoring wells were observed to be dry upon arrival. A datalogger was installed in each of the 14 monitoring wells and a barometric pressure logger installed in one (1) well to collect continuous water level measurements.

Groundwater sampled at most wells exhibited neutral pH (6.65 to 8.62). More basic conditions have been observed in the deep wells OKWW23-003 (11.12 pH) and OKWW23-009 (10.4 pH), though these pH measurements may be due to influences of cement grout rather than natural groundwater conditions. Sampled groundwater temperatures have ranged from 25.7°C to 29.4°C. Total suspended solids (TSS) ranged between 5.7 and 47.2 mg/L at most wells but were higher and above the IFC Effluent guideline (50 mg/L) at OKWW23-004 and OKWW23-009 (169.8 and 951 mg/L, respectively). Cyanide concentrations were below detection limits at all sites except for OKWW23-002, where free cyanide (0.01 mg/L), which is representative of the bioavailability, is present above the CONAMA FW guideline (0.005 mg/L) in comparison to a total cyanide detection (0.012 mg/L) at this well.

Approximately a third of dissolved metals results were below detection limits. Dissolved metals that were elevated relative to guidelines include one (1) sample of dissolved chromium and two (2) samples of dissolved iron. Results for polycyclic aromatic hydrocarbons, petroleum hydrocarbons, and oil and grease were all reported below the laboratory's reported detection limits with one exception.

The results from continued baseline studies will be used to inform water quality and numerical groundwater flow and contaminant migration modelling and form the basis for developing effective strategies for managing water inflow into the open pit and underground workings during the life of mine as well as post-closure, and to mitigate the potential adverse effects to groundwater quality in the receiving environment (both natural and man-made) from Project activities. Further modelling studies will also be used to validate current assumptions used in the conceptual mine closure plan. The PEA presents a conceptual water management strategy that will need to be refined in later stages of study. There is a potential for additional and/or differing management strategies needed to be deployed than those considered in the PEA.

Surface Water Quality and Hydrology

While there are many sub-basins, there are four (4) primary watersheds in the Project area consisting of minor river systems. The main rivers flowing through the Project area are the Puriari River, Takutu River, Kairuni River and Oko River. The Puriari River watershed drains the western half of the concession area and flows southwest into the Mazaruni River. The Takutu River watershed drains the southern arm of the concession area and flows southeast into the Mazaruni River. The Kairuni River watershed drains the

northeastern portion of the concession area and flows North into the Oko River approximately 4 km North of the project concession area. The Oko River watershed drains a large area North of the project concession and flows northeast into the Cuyuni River. While all major watersheds in the Project area have been impacted to some degree by artisanal mining, the Oko River is the current site of the most extensive artisanal mining activities.

Three (3) surface water hydrology monitoring stations were installed and automated stage monitoring initiated for the wet and dry season surveys in 2023. Surface water quality and hydrology monitoring stations were co-located to begin establishing a body of knowledge for water resources within the Project area.

Over the course of seven (7) days during both, wet season (August 2023) and dry season (October 2023), field measurements and water quality samples were taken / collected at 20 surface water quality monitoring stations near key mine site features (i.e., the location of the open pit plus established and proposed infrastructure areas). Sites that represent basin drainage and areas outside of the PL were also established at that time. Due to natural (i.e., stream was dry) and logistical constraints, field measurements and water quality samples could not be taken at two (2) stations. Similarly, the initial hydrology monitoring network focused on the characterization of streamflow through the eastern section of the Project area, where the open pit and key infrastructure would be located. Monthly stage-discharge monitoring continued through 2023 and 2024 apart from December 2023 and January 2024. After reviewing the field measurements and laboratory results, between one (1) to six (6) stations were deemed to exceed applicable CONAMA specifications / standards for three (3) parameters (turbidity: SW23-8; dissolved oxygen: SW23-4, SW23-6 and SW23-16; and total nickel: SW23-5 and SW23-12). Sediment samples were co-located with surface water samples and indicated exceedances of chromium, copper, iron and nickel. (CONAMA 2005). Impacts of artisanal mining was evident in the baseline results where biologically toxic parameters, such as mercury (measured as methylmercury) and cyanide were detected in the streams surrounding the project area and also in the Cuyuni River water column and sediments. Artisanal mining has resulted in extensive stream and watershed alterations to the north of the Project which has altered the natural hydrology of the area, especially where old pits and tailings ponds now act as retention ponds and wetlands. To date, the presence of artisanal miners has also restricted ready access to this area to collect baseline field data.

In 2024, an additional six (6) surface water hydrology stations were installed to expand the monitoring network to the western section of the Project area and to locations outside the concession to monitor downstream impacts. These new sites were co-located with new sediment and water quality sites to ensure sufficient spatial coverage required for the assessment of potential impacts across the entire project area. One such site was also co-located with an aquatic biology site on a natural creek with limited mining impacts in order to establish a reference site for baseline conditions.

Through baseline studies, it has been determined that project infrastructure will overprint a number of streams and large sections of existing watersheds. This will require the diversion of upstream flows around project infrastructure for non-contact water, specifically around the open pit and TSF. Contact water produced by runoff from the infrastructure footprint and dewatering of the pit will require treatment through sedimentation ponds to protect water quality in downstream receiving environments and reduce sediment loading. A key design consideration will be the choice of discharge watershed for treated TSF, contact water and wastewater outflows. This will require a risk-benefit analysis of factors such as the benefits of maintaining water within its natural watershed compared to discharging contact waters to watersheds with relatively good water quality and/or sensitive aquatic species of interest.

ERM has recommended a monthly sampling program and a more robust quality assurance / quality control program at a subset of baseline sites relevant to future operations. These data will be essential for water quality modelling and to support continued engineering design. In terms of surface water hydrology, ERM has recommended bi-monthly monitoring to continue in 2024 and into the future in order institute long-term monitoring within the project area to better inform project planning. The existing baseline data and ongoing long-term monitoring will be crucial to inform the design of water management infrastructure and the development of an integrated water management plan.

Geochemistry

A geochemistry baseline program is underway to evaluate the metal-leaching and acid rock drainage (ML/ARD) characteristics of future waste materials at Oko West. The program's objectives are to characterize the ML/ARD properties of anticipated waste materials (waste rock, tailings, and overburden) through static and kinetic environmental testing. The program structure is based on global best practice (Price,1997; MEND, 2009) for environmental geochemical characterization. The results will be used to inform water-quality modelling and to develop effective management strategies for these materials throughout the project's lifecycle.

The comprehensive baseline geochemical program includes samples that are representative of the material masses expected to be disturbed as well as reflective of the various geochemical material-type domains that are present at Oko West, as outlined below. The waste rock and tailings samples were subject to acid-base accounting (ABA) to determine acid and neutralization potentials (AP and NP, respectively), shake flask extraction (SFE) tests, sulfur- and carbon-speciation analysis, and four-acid ICP-MS multi-element analysis.

A total of 75 variability samples were selected to represent the main material types, as represented by multi-element geochemistry (as captured by pXRF analysis) and identified by geological logging. In the

fresh domain, that is footwall granitoid, hanging-wall granitoid, footwall basalt, the central volcaniclastic and siliciclastic sediments (silt- and sandstones), and carbonaceous mudstone. Transitional (partially oxidized / weathered) rock is characterized by a depletion in calcium (~one order of magnitude lower than fresh rock). Compositionally, the saprolite domain (strongly oxidized / weathered rock) can be subdivided into two (2) types, Ox 1 and Ox 2. Compared to fresh and transitional rock, Ox 1 is strongly depleted in calcium (typically below 1%) and iron, and Ox 2 is very strongly depleted in calcium (typically below 0.1%), enriched in iron (typically above 10%) and associated transition metals (titanium, manganese, vanadium). The increasing depletion in calcium from fresh over transitional to saprolite material is interpreted as progressive carbonate weathering. Both unmineralized samples and mineralized samples (above Au cut-off) were tested to make sure materials adequately represented if classification of ore and waste changes over the course of the project. Furthermore, testing mineralized samples allows to assess the ML/ARD potential of ore stockpiles and process tailings.

Acid-production and metal-leaching over time are currently evaluated in ongoing kinetic testing for which the 75 variability test samples were blended into 14 composite samples undergoing humidity cell tests (HCTs). Four (4) additional HCTs are also being carried out on composite samples prepared from mine tailings material generated by the 2023 metallurgical test work. The four (4) composites comprise tailings produced from processing of fresh, transitional and saprolite material as well as a combination of those three (3) based on the preliminary 2023 mine plan, respectively. The kinetic testing program also includes ABA, particle size analysis, multi-element analysis and mineralogical analysis of each composite sample.

Based on the ABA results, most samples tested are not potentially acid generating (NPAG), due to high a neutralization potential ratio (NPR = AP – NP) and/or low total sulfur content (<0.1%). In particular, the main waste-rock units (hanging- and foot-wall granites, footwall basalt) are of low concern, as confirmed by the initial ABA test work on footwall and hanging wall granites, in which all samples showed high NPR with total sulfur values below 0.1%.

Potentially Acid-Generating (PAG) waste was identified in the transitional domain located between fresh rock and saprolite, where the carbonates are partially leached but not all pyrite has been oxidized. Most of the PAG material; however, appears to be mineralized above the current Au cut-offs and therefore unlikely to be waste rock. Nevertheless, based on the currently available data waste rock from the transitional domain is most at risk to be PAG, depending on Sulphide and carbonate contents. Static testing of individual tailings has highlighted something similar; namely, that tailings samples generated from processing of transitional material are most at risk to be PAG (or near-PAG) based on their NPR between 1 and 2, which classifies them as “uncertain”.

Fresh siliciclastic sedimentary rocks of the central rock unit are most variable regarding their Net Neutralization Potential (NNP = NP – AP) and NPR and may classify as PAG or “uncertain” locally, depending on Sulphide content. This is especially suspected of carbonaceous mudstones that may contain diagenetic pyrite unrelated to gold mineralization. Additional static test work and increased sulfur data coverage across the deposit are planned to be carried out for further characterization. The volcaniclastics of the central volcano-sedimentary rock unit appear not to be at risk to be PAG due to their high carbonate contents.

Using the static testing results, an NPR proxy was developed from linear regression of calcium / sulfur (Ca/S) ratio against the experimentally determined NPR. Using this proxy materials can be flagged as at risk to be PAG when the NPR proxy is smaller than 2, and total sulfur is larger or equal to 0.1%. Reunion Gold has recently increased the coverage of four-acid ICP-MS multi-element data across the deposit, which was used for geostatistical estimation of calcium and sulfur data into the resource block model to estimate volumes of PAG material based on the Ca/S ratio by G Mining Services. However, confidence in the geostatistical models is insufficient for operations and Reunion Gold plans analysis of C and S in blast holes for short-term mine planning and waste management during operations.

Despite most tested samples being considered NPAG, potential for neutral metal leaching cannot be excluded, which is evaluated in the ongoing kinetic testing. To date, HCT leachate results show consistency with static testing prediction. Most HCT leachates have circumneutral pH. Only the HCT leachates from saprolite samples are moderately acidic, which is natural for the project area and consistent with baseline groundwater data. The HCTs and SFE tests demonstrate that low-sulfur / low-gold samples in the siliciclastic sediments show the potential to leach Arsenic (As), which will continue to be monitored, but seems to only affect a relatively small proportion of the site’s total waste rock. Notably, the concentrations of arsenic in HCT leachate tests are close to guideline values and have decreased over time. Results from the kinetic testing of waste-rock, overburden and tailings will be used to define geochemical source terms for future water-quality modelling.

In summary, ML/ARD risk in the main waste rock units (hanging- and footwall granites, and footwall basalts) appears to be low. Waste rock from the transitional domain and locally (carbonaceous) sedimentary rocks appear to be at most at risk of being PAG. Similarly, tailings generated from processing of transitional material appear to at most at risk of being PAG as well.

The results of the geochemical characterization program will be used to inform water-quality modelling and form the basis for developing effective strategies for handling these materials throughout the project’s lifecycle. An ML/ARD management plan will need to be developed to mitigate the potential adverse effects on the receiving environment of waste rock and tailings and elevated metal concentrations in surface- and

groundwaters. Based on currently available information, ML/ARD is not expected to present a material environmental risk.

For the purposes of the PEA, Reunion has assumed that mitigation will include isolating PAG materials within the wase rock storage facility, capturing and testing waste rock seepage, and including, as a contingency, the potential for the treatment of excess water from the tailings storage facility (TSF).

20.1.1.2 Biological Baseline

Guyana's biodiversity is characterized by its diverse ecosystems, including tropical rainforests, savannas, wetlands, and mangroves. Located within the Guiana Shield, its stable geological structure contributes to the regions' high levels of endemism and species diversity. Guyana consists of extensive freshwater systems that support a wide array of terrestrial and aquatic life.

The Project Area, though not recognized as a conservation priority by the Government of Guyana, is located within a region of high biodiversity significance. Despite the absence of designated protected areas nearby, the Project Area's ecological value highlights the importance of integrating conservation considerations into project planning and implementation. Understanding the spatial distribution of biodiversity and identifying critical habitats within the Project Area are crucial steps towards responsible environmental stewardship and sustainable development.

The biological baseline for the Project Area has been established through a combination of desktop research and in-field surveys conducted between 2022 and 2024. The methodologies employed have aimed to comprehensively assess the presence and distribution of key species and habitats within the study area.

The significant species diversity and abundance observed across terrestrial and aquatic surveys highlight the ecological richness of the Project Area. However, this also indicates potential regulatory requirements and mitigation measures to ensure protection during project development. Fluctuations in species abundance and richness across seasons suggest the need for adaptive management strategies. Projects must account for these seasonal dynamics in planning and mitigation efforts. The presence of species of conservation concern, such as endangered (EN), vulnerable (VU) or near threatened (NT) IUCN Red List species and CITES-listed species, and high biodiversity in the Project Area will require robust environmental management plans and potentially increased mitigation costs.

The findings of these surveys highlight the need for a comprehensive conservation strategy that integrates habitat protection, species monitoring, and adaptive management practices. This approach is crucial for

minimizing project impacts and ensuring long-term sustainability. The following recommendations are suggested:

- Mitigation strategies should focus on avoiding, minimizing, and compensating for potential impacts on these species and their habitats.
- Implement a Wildlife Sightings & Records Book and database for ongoing monitoring of wildlife sightings and habitat use patterns.
- Establish a Biodiversity Monitoring Program to periodically assess species populations and habitat conditions.
- Conduct detailed assessments to identify critical habitats, especially for species like giant otters, and integrate findings into project planning and mitigation strategies.

By addressing these recommendations and continuing to monitor biodiversity, the Project can proactively manage environmental risks and enhance its conservation outcomes while ensuring compliance with regulatory requirements.

Terrestrial Ecology

Terrestrial biodiversity surveys conducted during the dry and wet seasons have provided a comprehensive description of the existing mammal, bird, amphibian and reptile species located in the Project Area. The 2023 surveys significantly expanded both the geographic coverage and methodological approaches compared to the previous year, resulting in a more robust examination of the area. In 2023, additional Auditory Encounter Surveys (AES) and Visual Encounter Surveys (VES) along transects, mist netting, pitfall traps, camera traps, and environmental DNA (eDNA) sampling ensured a more comprehensive assessment, validating species presence and significantly increasing the understanding of species abundance across all taxonomic groups. In 2024, baseline data collected has expanded to include areas outside of the immediate mine footprint required for planned ancillary infrastructure, such as the barge site, power line, and access road.

The mammals identified during the completed wet and dry season surveys offer valuable insights into the biodiversity dynamics within the Project Area. A total abundance of 354 individuals and 23 species were identified during the 2022 to 2024 surveys. In 2022, a total of 81 mammals were recorded, with 34 individuals observed during the dry season and 47 during the wet season. The following year, in 2023, mammal abundance increased to 200 individuals, consisting of 80 mammals observed during the dry season and 120 during the wet season. During 2024, the total number of mammals for the dry season was 73. These observations highlight fluctuations in mammal populations over the three-year period, reflecting

variations influenced by seasonal factors, sampling effort, and updated methodology. Several species identified were on the IUCN Red list (EN, VU, or NT) and CITES.

Giant otters (*Pteronura brasiliensis*) are an IUCN red-listed endangered species and were physically noted as being present in the Project area during the baseline field campaign in 2023. In response, more detailed studies are underway, aimed at gaining an understanding of the giant otter population size, distribution and habitat use within the Project Area and surrounding river catchments, including the Puriari River, Takutu River (tributaries of the Mazaruni River), and Oko River (tributary of the Cuyuni River) (collectively referred to as the Regional Survey Zone (RSZ). The ultimate purpose of the survey will be to determine the criticality of the Project area to the Giant Otter population in the region so that appropriate mitigation measures can be implemented. Depending on the results of this survey, a Biodiversity Management Plan, including the need for habitat offsets may be required to be implemented. The cost of mitigation measures and potential constraints on Project design and operation are currently unknown.

Across the wet and dry season bird surveys of 2022 and 2024, there was a noticeable increase in both total abundance and species richness. From 2022 to 2024, surveys identified a total abundance of 4,588 individuals across 274 species. In 2022, 1,187 birds were recorded, with 616 individuals observed in the dry season and 571 in the wet season. The following year, 2023, saw an increase in bird abundance to 2,564 individuals, comprising 1,358 birds during the dry season and 1,206 during the wet season. By 2024, bird numbers during the dry season totalled 837. These observations highlight fluctuations in bird populations over the three-year period, influenced by seasonal dynamics, survey efforts, and updated methodologies.

Similar to the mammal survey efforts, the use of different sampling methods across survey seasons offers additional insights into bird species detection and survey efficiency. While AES and VES methods were consistently used across all seasons, the integration of mist netting alongside AES/VES during the wet seasons of 2023 led to a substantial increase in species detection and identification. The additional survey methods likely contributed to the identification of a higher number of endemic bird species, IUCN Red List species, and CITES-listed species during the wet season of 2023 compared to previous surveys.

In the dry season of 2022, amphibian abundance surpassed 370 individuals with 17 species and 6 families recorded, while reptile counts totalled 121 individuals spanning 9 species across 4 families. Subsequent wet seasons of 2022 and 2023 saw notable increases: amphibian counts exceeded 268 individuals with 15 species and 6 families in 2022, while reptile counts reached 120 individuals across 12 species and 6 families. By the wet season of 2023, amphibian counts exceeded 470 individuals with 26 species observed, and reptile counts rose to 174 individuals spanning 18 species and 9 families. In the 2024 dry

season, combined amphibian abundance from transect and pitfall surveys surpassed 452 individuals across 23 species, with reptile surveys identifying 116 individuals across 26 species.

The findings of the surveys highlight the dynamic nature of amphibian and reptile populations across different seasons and note the importance of using diverse sampling methodologies, such as AES/VES and pitfall traps, for comprehensive biodiversity assessments. Some species identified were CITES-listed species. The observed fluctuations in abundance and diversity emphasize the need for continued monitoring and conservation efforts to ensure the preservation of amphibian and reptile species in the Project Area. It is noted that the 2024 surveys have commenced for ongoing aquatic ecology surveys in the wet and dry season.

Aquatic Ecology

A total of 25 sites were surveyed in the wet and dry seasons in 2022 and 2023 within and surrounding the prospect licence area to establish baselines for macroinvertebrates, fish, and physical habitat conditions. An additional seven (7) sites were also surveyed in 2024 to include areas that may be impacted by development of ancillary facilities, as identified above.

The results of the physical habitat survey indicate widespread impacts of historical and current small scale gold mining, as well as associated land clearing, road construction, and channel disturbances. Some streams, particularly in the southern portion of the licence area, retain aspects of a natural channel morphology, but the nearly all streams within and surrounding the licence area with the exception of some extreme headwaters have been impacted by mining or timbering activities to some degree.

Seasonality significantly contributed to differences in the abundance and diversity of macroinvertebrates in the dry versus the wet seasons. The dry season showed greater abundance of terrestrial and aquatic macroinvertebrates. However, the wet season has greater diversity and evenness than the dry season, with the majority of sites containing better water quality in the wet season when compared to the dry season.

The wet season fish survey identified a total of 66 species of fish within the mine's prospect licence boundary; these species were distributed across five (5) orders, 19 families and 41 genera. The number of species captured across all sites was slightly higher in the dry season than in the wet season (45 species in the dry season vs 41 species in the wet season, or a 10% increase in the dry season over the wet season), but abundance increased by a larger margin (48% increase in the dry season over the wet season).

None of the species recorded in these surveys to date are currently listed as being of conservation concern according to IUCN nor CITES. However, several species hold significant ecological and economic value. For example, some species are key food or protein sources, and others are sought after in the aquarium trade. This highlights their potential vulnerability to either fisheries overexploitation and or habitat loss. Several important fish species recorded for the combined seasons were the *Pseudoplatystoma fasciatum, Hoplias aimara* and *Hoplias malabaricus*, which are at the top of the food chain and can serve as bioindicators species. These species play very important roles in maintaining balance within the food webs, by regulating and ensuring that many smaller sized fish and aquatic animal species do not become dominant; hence, they maintain the biodiversity and promote stability within these ecosystems. These species also provide an indication of the health of their ecosystems due their ability to accumulate and bio-magnify environmental contaminants (such as mercury) within their tissues. As bioindicators, monitoring their populations and health can provide pertinent but early indication of pollution and environmental degradation.

The Project area encompasses a wide range of aquatic habitats, including high gradient headwater / ephemeral streams, larger perennial bottomland streams, ponds, and swamps. This diversity in habitat preferences underscores the ecological complexity within these freshwater and headwater systems in tropical watersheds. The historical anthropogenic and current environmental degradation described above poses a significant threat to these habitats, potentially impacting the biodiversity and ecological balance of these freshwater streams. Therefore, the Project's contribution to conservation efforts, habitat restoration and continuous monitoring will be important to help maintain the ecological integrity of these freshwater and headwater ecosystems.

20.1.1.3 Social Baseline

Socioeconomics

As part of the socioeconomic baseline studies, a scoping visit was conducted by ERM personnel in May 2023 to establish an initial understanding of the Project, geographic context, socioeconomic and cultural dynamics, and to define a preliminary Area of Influence (AoI) for a potential future mine. The first scoping visit was followed by longer fieldwork visits in October 2023 and May 2024. All fieldwork trips included a team of international and Guyanese Subject Matter Experts (SMEs). In May 2023, this was a team of three (3) people; in October 2023, the team consisted of six (6) people, as well as Reunion Gold's Country Manager; and in May 2024, the team was composed of five (5) people.

Settlements within the preliminary AoI are:

- Georgetown, the capital city.
- Parika, a key port town on the Essequibo River, located west of Georgetown and accessible by road from the capital.
- Bartica, a regional centre located at the confluence of the Cuyuni, Mazaruni, and Essequibo Rivers.
- Agatash and Dagg Point, Amerindian communities that share boundaries and infrastructure with Bartica.
- Batavia, an Amerindian settlement and titled area at the confluence of the Essequibo and Cuyuni Rivers.
- Kartabo (sometimes spelled Karatabo), an Amerindian settlement on the Mazaruni River, in the vicinity of the proposed barge landing facilities. Kartabo is not formally titled.
- Karrau (or Karau), an Amerindian settlement and land title located across the Essequibo River from Bartica.
- Itaballi (or Itaballi Landing, sometimes spelled Itabali), a port community at the northeastern end of the Puruni Road (or Itaballi-Puruni Road).
- Takatu (or Takutu), an informal landing located approximately halfway along the Puruni Road, and currently acting as the land commercial and service centre between Itaballi and the Project.
- Bryan's Gate, a private toll road used to access the Project site between Takatu and the Prospecting Licence Area.
- Oko West Camp and nearby landings known as Sand Hill, Sand Hill 2 and Oko Landing as well as Blackwater Crusher and Blackwater.
- Puruni (or Puruni Landing), a landing area at the southwestern end of the Puruni Road, on the Puruni River.

Gold and diamond mining is central to livelihood strategies throughout Region 7, and specifically alluvial gold mining in the Project area. To buffer against the risks inherent to boom-bust cycles associated with gold mining, most households practice diversified livelihood strategies, including a mix of formal and informal activities, such as artisanal mining, logging, and vending or commerce. Households are typically male-led, and males are the primary—if not often the sole—income earners. Community members, both men and women, do also maintain livelihoods as shopkeepers, buying and selling gold, inn keepers and brothel owners, cultivating kitchen gardens, and conducting ad hoc repair and construction work. Interviews with community members suggest formal positions with established mining operations are attractive

because salaries tend to be much higher than the national average, but that these positions are very difficult to secure.

Mining in Region 7 includes artisanal and small-scale mining, large-scale exploration, and operations. There is currently a moderate to high level of artisanal mining activity in the region and preliminary AoI. Local trails and tracks cross the Project area, although the previous title holder has maintained the PL largely free of artisanal miners. Now, only pork-knockers using metal detectors continue mineral prospecting along roads and paths within the PL. There are active dredging and shaft operations to the north (Blackwater Crusher, Blackwater) and west (towards Puruni) of the Project area. Occupants and operators are granted land usage and extraction rights by legal owners who receive a percentage of mineral earnings.

Primary health concerns in communities throughout the preliminary AoI are malaria, dengue, and typhoid, with access to clean (potable) water a common problem, particularly in landing and backdam areas, as well as smaller interior communities. Food insecurity and nutritional issues are also common within the preliminary AoI, as available cash or spending power are strongly associated with boom-bust mining cycles in the backdams; correspondingly, successfully harvesting animals (fish and wildlife) for consumption is reportedly more challenging in areas impacted by mining and forestry activities. Access to medical care (including at posts, health centres, and hospitals) is similarly limited in more isolated communities. As these dynamics show, an existing economy underpinned by artisanal and small-scale mining requires key consideration in developing a future mine and is a common risk to projects in many parts of the world, including Guyana.

There are both formally titled and non-titled Amerindian (or Indigenous) land titles in the preliminary AoI, and in close proximity to proposed ancillary facilities associated with the Project, including Batavia and Karrau (titled), and Kartabo (untitled). Surface rights on Amerindian titled lands are ultimately granted by Amerindian communities, with agreements managed and overseen by the Government of Guyana. Although ancillary facilities do not currently overlap with titled Amerindian lands, further work to determine the potential socioeconomic impacts of ancillary facilities may be required after the final location of any associated infrastructure is confirmed.

Stakeholder Engagement

Reunion Gold has established strong relationships with some key stakeholders in the Project area, including formal titleholders, organizations, businesses, and/or individuals. Engagement to date has included long-term and informal relationship-building, public scoping consultation meetings in December 2023 as required by the EPA, and increasingly focused efforts that align with mine planning and development, such

as job fairs in nearby population centres like Bartica, as well as community investment and development initiatives in Amerindian communities.

Engagement planning is a central component of Reunion's recent Sustainability Strategy, and a formal Stakeholder Engagement Plan is being finalized that is commensurate with advancement of mine planning and permitting. Stakeholder 'mapping' conducted in early 2024 as part of the development of a Stakeholder Engagement Plan identified key stakeholders by representative group or category, including:

- Governmental bodies (e.g., Guyana Forest Commission, Ministry of Amerindian Affairs);
- Businesses and private sector organizations (e.g., Willems Timber, Boom Blast Driving, Mekdeci Machinery and Construction (MMC));
- Civil society (e.g., Guyana Gold and Diamond Miners Association, Conservation International); and
- Local and Indigenous communities and population centres (e.g., Itaballi, Batavia, Bartica).

Feedback and perspectives on the Project collected as part of stakeholder engagement thus far centre on the following themes:

- Livelihood diversification and income-generating opportunities (especially for Indigenous women) with the mine and/or supported by the Oko West Project.
- Education, including programs to support ongoing school attendance and limit secondary school drop-out rates by adolescents who leave school to support their families by finding work in mining.
- Safety, health, and wellbeing, including concerns about impacts on Amerindian communities as a result of any potential increases in traffic and non-community visitors with mine development (including ancillary facilities).

As part of ongoing work to formalize communications and engagement with all stakeholders, Reunion keeps records of all engagement activities undertaken.

Cultural Heritage

Between September and October 2023, a cultural heritage baseline study that employed background research, archaeological field surveys, and community interviews were conducted within the Project area and surrounding communities to assess what cultural resources (tangible and intangible cultural heritage) may be affected by a potential future mine. Background research did not identify any cultural heritage sites in the study area, but archaeological field surveys documented three (3) cultural resources – one (1) archaeological site and two (2) isolated finds locations. Three (3) additional cave sites were

identified in the Project area, but field surveys in May 2024 did not reveal any cultural materials present and as such have been discounted as potential sites. There are no built heritage sites in the current proposed Project area.

In two (2) areas of the project, (the TSF area and west of the Pit) a number of colonial-era bottle scatters were observed on the surface of the jungle floor. Investigations did not reveal any buried deposits associated with the bottles, so they are regarded as an isolated finds and not significant.

One (1) archaeological site was discovered within the footprint of the planned open pit. The site, a rock shelter (designated Cave-1), had pottery and lithic debitage and stone tools visible on the surface. Interviews with residents of communities adjacent to the Project area or in the broader region, indicated that while some people are aware of caves in the area, people are not venturing into them and are not aware of the caves of this region having ever served as habitation or camp locations. In May 2024, a joint team from ERM and the University of Guyana undertook a series of 3D digital surveys and archaeological test excavations at Cave-1 to further understand the nature, extent and significance of the site in order to feed into the regulatory permitting process. This work was under permit from the EPAG. Significant quantities of archaeological artefacts were recovered from the surface of the cave, but there was no evidence of funerary activity within the cave or surrounding area – no burials, human bone or votive offerings. There is no evidence of petroglyphs (rock carvings) or Pictographs (rock paintings). Post-excavation analysis is currently being undertaken in Georgetown under the direction of Louisa Daggers of the University of Georgetown.

The preliminary results confirmed that Cave-1 is an important archaeological site both regionally and in the wider Guyana context, and that it can be classified as ‘Non-Replicable Cultural Heritage’ as per the IFC Performance Standard for Cultural Heritage (PS8). Given its location within the pit, Cave-1 will be removed if the project proceeds. Preservation in-situ is not deemed feasible as it would severely impact the viability of the project. It is deemed that the overall benefits of the project to the surrounding population and Guyana as a whole would outweigh the anticipated cultural heritage loss through removal of the site. Since preservation is not deemed feasible, a suitable mitigation measure is required under PS8 to minimize this anticipated loss of Cultural Heritage. In line with the Equator Principles, Best International Industry Practice and more specifically IFC Performance Standard 8 (Cultural Heritage), it is recommended that Preservation by Record (full archaeological excavation followed by an appropriate level of post excavation analysis) using modern scientific techniques is undertaken at Cave-1. The mitigation of Cave-1 would not represent a significant cost item to the project and if appropriately scheduled, would not cause schedule implications.

20.1.2 Future Environmental and Social Baseline Studies

A comprehensive, Project specific baseline study program was initiated in 2022, and information collected to date has helped build an understanding of the local and regional environmental and social conditions that have informed the PEA. Additional environmental and social data collection will need to be collected, and additional studies will be required to contribute to the next stages of Project design, the identification and mitigation of potential impacts on its receiving environment, and the submission of an environmental impact assessment (EIA) for regulatory purposes. These studies will need to further incorporate ancillary project components such as power supply and site access roads. ERM has recommended the following continued environmental and social data collection, studies, and actions:

Topic	Recommendation
All	• Conduct wet and dry season sampling in 2024 in the Project Area and for the associated facilities (transmission line, port, etc.). • Conduct temporal and spatial analysis of the 2022 to 2024 baseline data and incorporate this into the EIA.
Ecology	• Terrestrial Ecology: Conduct additional surveys to expand on the existing vegetation baseline throughout the Project Area in order to: • Enhance survey accuracy to allow for a more robust baseline. • Capture additional spatial variability. • Continue to document species of conservation concern. • Monitor changes over time. • Inform management decisions and future planning. • Community (i.e. informal settlements) engagement is recommended to determine the local uses of the vegetation within the area, as many were recorded to have economic, cultural, or medicinal values. • Establish a comprehensive Wildlife Sightings & Records Book (and database) to enable field-based staff to document significant wildlife sightings, enhancing monitoring of relative and seasonal species abundance to inform project area management. • Implement a Biodiversity Monitoring Program conducted by experts to conduct seasonal assessments, providing essential data for conservation efforts. • Prioritize surveys on the giant otter to contribute valuable data for critical habitat determination, including distribution and habitat surveys during both wet and dry seasons. • As the Project advances, address potential impacts on mammal, bird, amphibian, and reptile species, including habitat loss, illegal hunting, fishing, and poaching, through conservation strategies, such as habitat restoration, enforcement of protected area regulations, and community-based initiatives to promote sustainable land use practices. • Establish set-asides to protect key habitats critical for threatened species identified under the IUCN Red List, regulated under CITES, defined as endemic or with restricted habitat ranges.

Topic	Recommendation
	• Sustain surveys to gather essential data for determining critical habitat, particularly for the *Pteronura brasiliensis* along the Puriari River during both wet and dry seasons. • Conduct further surveys for the Guyana endemic, *Stefania evansi* to determine this species' presence in the area, focusing on both wet and dry seasons to ensure comprehensive data collection. • Use additional survey locations along clipped forest lines (200 to 300 m) to actively sample leaf litter and debris for frog and reptile species, contributing to monitoring of threatened *herpetofauna*. • Continuously monitor breeding areas for amphibians, particularly focusing on water quality parameters, especially in new mining areas, to ensure the preservation of critical breeding habitats. • Conduct a preliminary investigation for each new eDNA survey to evaluate detection probabilities for target species based on sampling and analysis protocols, including spatial sampling design, sample volume and collection method. • Expand sampling efforts across multiple seasons (wet and dry) and increase sample size to identify or validate the presence of elusive and rare species. • Aquatic Ecology: Continued conservation efforts, habitat restoration and monitoring for maintaining the ecological integrity of the freshwater and headwater ecosystems. Sampling larger downstream rivers / streams / channels should yield higher diversity and richness and provide insight into the potential effects of the Project on downstream aquatic ecology.
Physical	• Air Quality – Conduct monitoring and sampling for gaseous compounds and particulate matter (PM), $N0_2$ and SO_2 along with existing information to characterize local baseline concentrations in the Project area and for the ancillary facilities. • Noise – Conduct additional baseline noise level measurements. • Climate Change and Greenhouse Gases – Conduct GHG characterization based on the Project available information to calculate the GHG emissions during construction phase (e.g. assumed average fuel consumption for construction equipment) and operation phase (e.g. assumed average power consumption for operating equipment). • Hydrology – Further characterization of streams in and around the Project Area and ancillary facilities using LiDAR topography data where available, surface water monitoring gauge station data, a site survey, and stream discharge collection. • Surface Water and Sediment Quality – Further characterization of streams and sediment quality in and around the deposit area and the expected location of major associated infrastructures (pit, WRD, TSF) during both wet and dry seasons, building on existing data from previous baseline assessments. • Groundwater – Conduct additional groundwater investigations to expand on the existing groundwater quality and quantity database and advance conceptual groundwater flow and contaminant migration models. Collect additional information to characterize the groundwater baseline conditions in areas of key domains such as potential waste rock and tailings facility locations.

Topic	Recommendation
	• Geochemistry – Collection of samples representative of shallow overburden in the direct Project area and ancillary facilities area to determine the predicted geochemistry of the anticipated waste rock and tailings storage facilities, exposed pit and disturbed areas of the project and build in the aquatic baseline data. • Terrestrial Ecology – Conduct additional seasonally-based surveys to characterize the terrestrial biodiversity for areas expected to be impacted by the Project, including ancillary infrastructure such as access roads and power lines. Establish an Ecologically Appropriate Area of Analysis (EAAA) for the survey sampling locations for terrestrial vegetation and wildlife. In addition to the terrestrial biodiversity survey, complete a targeted distribution survey of Giant Otter in and around the Project area. • Aquatic Ecology - Additional aquatic species composition and distribution surveys will be required across the Project site and ancillary facilities.
Socio-Economic & Cultural Heritage	• Stakeholder Engagement: Finalization and implementation of a Stakeholder Engagement Plan for the Project, including development of commitments register and associated grievance mechanism. • Potential Resettlement and/or Economic Displacement Comprehensive assessment of the potential socioeconomic impacts of the Project in alignment with IFC Performance Standards, including impacts on informal settlers (those without recognizable legal right to the land they occupy) and Indigenous peoples. • Human Rights – Development of a human rights assessment framework, which will translate human rights regulations and international standards into clear requirements. Building on previous work conducted to-date, an External Factor Review (EFR) will be required to identify local and operational inherent risks associated with the site as well as an initial identification of rightsholders, and related potential and actual impacts which will guide the fieldwork and stakeholder engagement step. • Cultural Heritage – Additional cultural heritage field surveys will focus on the wider portions of the project footprint including ancillary facilities and core project extension areas and will focus on identifying any previously undocumented cultural heritage sites, significant historic structures or any other locations associated with intangible cultural heritage. In addition, full archaeological excavation of Cave-1, in line with IFC PS8, is recommended as a mitigation measure (under permit from the EPAG) prior to any construction activities in that area, to mitigate the impact of the project on the site.

ERM is of the opinion the above additional information will be sufficient to inform the next stage of engineering study and project permitting. Data collected to date and reviewed by ERM has not identified any material issues of concern that would, at this stage, deem a project not viable in this location. Key findings to date include the presence of Giant Otters on-site, archaeological caves within the Project area, and the extent to which artisanal mining is present in the area.

20.2 Environmental Permitting

20.2.1 Guyana Environmental Protection Act

In 1996, the Environmental Protection Act (EP Act) was enacted to implement the environmental provisions of the Constitution. The EP Act is Guyana's most significant piece of environmental legislation because it articulates national policy on important environmental topics, such as pollution control and the requirements for environmental review of projects that could potentially affect the environment. It also provides for the establishment of an environmental trust fund. Most importantly, the EP Act authorized the formation of the Environmental Protection Agency (EPA) and established the EPA as the lead agency on environmental matters in Guyana, including the issuance of environmental authorizations with appropriate conditions. The EP Act mandates the EPA to oversee the effective management, conservation, protection, and improvement of the environment (EPA 2021). It also requires the EPA to take the necessary measures to prevent and control of pollution, assess the impact of economic development on the environment and sustainable use of natural resources.

Regulations on hazardous waste management, water quality, air quality and noise management were established in 2000, pursuant to the EP Act. These pollution management regulations were developed to regulate the activities of development projects during Construction and Operations stages. The following are regulations applicable to the Project under the EP Act.

20.2.1.1 EPA's Role in EIAs

The EP Act mandated four functions for the EPA, which relate to environmental assessment. The EPA has determined that development of a mine at this location will likely have significant environmental impacts. Four (4) functions of the EPA are consequently applicable to this Project, which are:

- To take such steps as necessary for the effective management of the natural environment to ensure the conservation, protection and sustainable use of natural resources;
- To promote the participation of members of the public in the process of integrating environmental concerns in planning for development on a sustainable basis;
- To ensure that any development activity which may cause an adverse effect on the natural environment is assessed before such activity is commenced and that such adverse effect is taken into account in deciding whether or not such activity should be authorized; and
- To give development consent which entitles the developer to proceed with the Project.

The EPA is required to implement several environmental management principles as part of this process. These principles are:

- The “polluter pays principle”: the polluter should bear the cost of measures to reduce pollution.
- The “precautionary principle”: where there are threats of serious or irreversible damage, lack of full scientific certainty shall not be used to postpone measures to prevent environmental degradation.
- The “strict liability” legal principle: any person who contravenes this Act or regulations shall be liable to the penalties prescribed thereafter.
- The “avoidance” principle: it is preferable to avoid environmental damage, as it can be impossible or more expensive to repair rather than prevent damage.
- The “state of technology” principle: the measures protecting the environment are restricted by what is technologically feasible, and as technology improves, the improved technology should be used.
- To prevent and repair environmental damage.

Reunion Gold was required to obtain an environmental authorization (also commonly referred to as an Environmental Permit) from the EPA to conduct its current exploration activities. An Application for Environmental Authorization was filed with the EPA on the October 17, 2022. After submission and review, the EPA issued a no-objection letter on January 18, 2023, and reconfirmed its decision by letter dated July 4, 2023. Reunion Gold will be required to obtain a new Environmental Permit to eventually develop the Project. In September 2023, the Company filed an initial application and subsequently collaborated with the EPA to establish the Terms and Scope (ToS) of a future environmental impact assessment. As part of this process, the Company conducted meetings with both government agencies and local communities in the last quarter of 2023 to determine the essential elements to be incorporated into the ToS. The approval of the ToS was required for the Company to move forward with work on an environmental and social impact assessment (ESIA). The ESIA is currently under preparation.

Table 20.1 describes the legal framework for national environmental management related to mining operations and Table 20.2 provides a summary listing of the major anticipated permits which are currently expected to be required based on the mine defined by the PEA.

Table 20.1: Legal Framework for National Environmental Management in relation to Mining

Title	Objective(s)	Implementing Agency	Comments	Applicability
The Environmental Protection Air Quality Regulations	Establishes parameter limits with respect to emission of smoke, solid particles, sulfuric acid mist or sulfuric trioxide, fluoride compounds, hydrogen chloride, chlorine, hydrogen Sulphide, nitric acid or oxides of nitrogen and carbon monoxide.	Environmental Protection Agency		The construction, operation and closure phases of the project may impact air quality.
The Environmental Protection Water Quality Regulations	Establishes guideline to minimize impacts of discharges on water quality.	Environmental Protection Agency	If the level of contaminants in effluent discharge from the project exceeds tolerable limits, water quality will be impacted. These regulations require registration and environmental authorization by any person whose construction, installation, operation, modification, or extension of any facility cause the discharge of effluents.	The project will entail discharges to surface water. The most stringent of the water quality standards outlined below will be applied during project execution.
The Environmental Protection Noise Management Regulations	Ensures effective implementation of health and maintain safety measures during operations and to minimize impacts on receptors in the site vicinity	Environmental Protection Agency	Under these regulations operations that emit noise in the execution of various activities such as construction, transport, industry, commerce, and any institution are required to apply to the Agency for an environmental authorization.	The project will generate noise. The regulations are consequently applicable to the project. Further the most stringent of the following noise regulations will be applied during project execution.
The Environmental Protection Hazardous Wastes Management Regulations	Ensures that trough the environmental authorization process, that all operations that generate, transport, treat, store, and dispose of hazardous wastes are managed in a manner that protects human health and the environment.	Environmental Protection Agency	There are no regulations for the management of hazardous substances. It can be inferred that these regulations will apply to reagents to be used by the project.	Gold will be beneficiated by cyanidation. This will entail the utilization of cyanide and other hazardous reagents in the processing operations. Petroleum products will be used for power generation and equipment operations. The chemical reagents and petroleum products will have to be managed in conformance with the EPA Hazardous Waste Management Regulations, 2000.
The Environmental Protection Authorizations Regulations	Mandated to keep records of all environmental monitoring (not less than 3 years), sampling procedures, maintenance and calibration procedures and all problems and malfunctions.	Environmental Protection Agency	Before an application for Environmental Authorization is made, an Environmental Impact Assessment or any other relevant document should be submitted to the EPA for approval.	The records should be project specific and shall include: • Steps Guyana Goldfields should follow to avoid, minimize, mitigate, and compensate for impacts. • Records of monitoring information such as date, place, and time of measurement etc. • Proper maintenance of facilities and appropriate quality assurance procedures. • Establishment of environmental monitoring programme.
National Environmental Action Plan (NEAP) developed in 1994	Aims for sound management of the environment and natural resources.	Environmental Protection Agency		The project must be undertaken based on sound environmental management practices. The National Environmental Action Plan is consequently considered applicable to this project.
Use of Poisonous Substances	Mandates for the use of cyanide and require that no operation in which cyanide or any preparation containing cyanide is used in the treatment of gold or other minerals may be commenced until the necessary buildings, structures, rooms, appliances and other arrangements to carry on these operations have been inspected and approved by a mines officer.	Guyana Geology and Mines Commission		The poisonous substances regulations are applicable primarily to the use of mercury and cyanide in the gold mining arena and stipulate that no person shall use elemental mercury or any form of mercury. Contravention of the regulations results in a Cease Work Order being placed on the operations on the claim or area until cleanup is done to the satisfaction of the Commissioner.

Title	Objective(s)	Implementing Agency	Comments	Applicability
Mining Regulations	Provisions for the sanitary management of human waste and for burial of animals and any person who expires at the mine.	Guyana Geology and Mines Commission	The Mining (Amendment) regulations 2005 contain mandates for the following: • Use of Poisonous Substances • Requirements for Environmental Management for Large and Medium Scale Mining • Requirements for Environmental Management for Small Scale Mining • General Requirements • Protected Areas • Pollution Control • Offences and Penalties	
Guyana Geology and Mines Commission Act of 1979 and Mining Act of 1989	Provisions for a system of mineral agreements and licences for regulating prospecting. Provisions for the disposal of sanitary waste and the storage of poisonous substances in mining areas.	Guyana Geology and Mines Commission	Gives the Commission the responsibility for establishing regulations for mining and quarrying operations.	Mining licence is required to mine any mineral and is issued at the discretion of the Commissioner of Guyana Geology and Mines Commission with the Minister responsible for mining.
Occupational Safety and Health in Mining	Contains provisions applicable to the regulation of health and safety in mines.	Ministry of Labour, Human Services and Social Security with the assistance of the International Labor Organization (ILO).		Draft OSH Regulation covers duties of employers and self-employed persons, owners of mines, manufacturers and workers. Reporting of dangerous situations and occurrences, administrative requirements such as the preparation, checking and maintenance of mine development plans in advance of mine development, as well as those for mine operations, inspections by mine operators, minimum age of workers and the maximum duration of work shifts, testing of shaft ropes and notice of death or injury are also covered.
Wild Birds Protection Act Chapter 71:01	Ensures the protection of certain wild birds.	Ministry of Legal Affairs		There are no provisions for the protection of wild birds relative to mining operations; however, conservation of biological resources is applicable to the project.
Town and Country Planning Act Chapter 20:01	Provisions for the orderly and progressive development of land (which is specifically defined), Cities, Towns and other areas be they urban or rural, for their preservation and improvement and for other related matters.	Central Housing and Planning Authority established under the Housing Act Chapter 36:20.	This act would regulate any new settlement associated with operations.	
Pesticides and Toxic Chemicals Control Act No. 13 of 2000	The registration of Pesticides and Toxic Chemicals. The issuance of licence to persons to import or manufacture registered pesticides and toxic chemicals.	Ministry of Agriculture	Regulations for the health and safety of workers against the risk of poisoning by controlled products when working in connection with the use of controlled products or when working on land or in any premises on or in which controlled products have been or are being used, stored or manufactured. Regulations imposing duties on the employers of workers, on the workers themselves and on others using controlled products and controlling and prohibiting sale, use, disposal and transportation of any toxic chemical or pesticide.	This project will entail the importation of cyanide and other chemicals reagents for use in beneficiation, some of which can be classified as toxic chemicals.

Title	Objective(s)	Implementing Agency	Comments	Applicability
National Biodiversity Action Plan	To integrate the implementation of the Convention on Biological Diversity into national development.	Ministry of Natural Resources and the Environment	The overall goal of the NBAP was “to promote and achieve the conservation of Guyana’s biodiversity, to use its components in a sustainable way, and to encourage the fair and equitable sharing of benefits arising out of the use of Guyana’s biodiversity”.	The mining operation and access roads will be in forested areas and impinges on biological diversity. Therefore, according to NBAP 1 consideration should be given to the adoption of the four approaches stated below when planning for management of biodiversity.
Public Health Act Cap. 145	Sanitary control of mining districts or parts thereof, and the health and welfare of the inhabitants.	Ministry of Health		The provisions of the Public Health Act will be factored into the employee health and safety plan for the mine site operations.
Forest Act 2008	Governs all activities that are carried out in the forest including mining and associated activities.	Guyana Forestry Commission	Under this Act, no person shall engage in the following unless a mineral prospecting or mining licence has been granted under the Mining Act 1989: • Enter and occupy state forest. • Cut, damage, or take any forest produce or carry on any kind of forest operation. • Carry out any kind of exploratory operation in a state forest.	
Guyana Forestry Commission Act 2007	To provide advice on various forest related issues and on formulation of forest policy. To prepare plans, codes of practice, and guidelines for the conservation and management of forests. To research, collate, analyse, prepare, and disseminate data, statistics, and other information about forests and all aspects of forestry including forest ecology and the use of forest produce. To inspect, certify and accredit services for quality control of forest produce.	Guyana Forestry Commission		The project will entail management and reclamation of forest resources, and this act must therefore be adhered to.
Code of Practice for Timber Harvesting 2002	Provides guidelines and standards on what constitutes satisfactory road building and other related forestry activities.	Guyana Forestry Commission		One important component of this project is the construction of an access road to the mine site. This road will at a minimum have to confirm to GFC Road Planning guide as outlined in the Code of practice for Timber Harvesting.
Low Carbon Development Strategy 2030	Intends to create a new low-carbon economy in Guyana by establishing incentives which value the world’s ecosystem services and promoting these as an essential component of a new model of global development with sustainability at its core. In Guyana’s case, harnessing the value of the country’s ecosystem services can build a long term, low-carbon diversification opportunity	Office of the President	The LCDS 2030 sets out how to accelerate the achievement of these four objectives, Forest Climate Services and other Ecosystem Services Stimulate future growth through clean energy and sustainable economic activities. Protect against climate change. Align with global climate goals in determining how to do this, planning for sustainable development is the core principle that guides the LCDS 2030	The LCDS 2030 will ensure that Guyana creates an ecosystems economy which balances the following: low-impact mining and forestry to enhance employment and income generation opportunities; forest climate services through which the value provided by Guyana’s forests to the world is recognised; Guyana’s next generation of ecosystem services, such as water management, and biodiversity protection.

Title	Objective(s)	Implementing Agency	Comments	Applicability
Land Permit on private lands	The owner of any private lands, granted before the passing of the Mining Ordinance, 1903, shall hold and enjoy all metals other than gold and silver therein or thereon, and may after obtaining a licence or permit under this Act search or mine for them in accordance with this Act and the licence and, when found, take and appropriate them to his own use.	Guyana Geology and Mines Commission	The Commission may, with the approval of the Minister and subject to section 8, grant a licence or permit under this Act authorizing the holder of the licence to enter on private lands and their search or mine for, take and appropriate, any minerals.	Before the Mining Ordinance 1903, private landowners can use any metal, other than gold or silver, found in valuable minerals on their lands. However, they must purchase the metal from the landowner at an agreed price or pay the Commission's stated net value, without deductions for gold or silver or both. Owner must comply with current laws for recording gold or silver extracted from mineral, removing it, and paying royalty.
Electrical Installation and Inspection Licence	set up to protect users of electricity against the hazard of unsafe and unsound electrical installation	Government Electrical Inspectorate	Issuance of Permits to Perform Electrical Installations. Issuance of Certificates for code compliant electrical installations. Inspection and Testing of Electrical Installations. Licensing of Qualified electrical contractors.	The mission of the Government Electrical Inspectorate (GEI) is to promote the awareness of the Electricity Sector Regulations Act (ESRA 2008), the National Electrical Code (NEC) as well as good Electrical safety practices, while providing electrical contractors and other personnel with practical advice in carrying out electrical works in compliance to the relevant codes and/or regulations.
Bulk Fuel Storage Licence / Consumer Installation Licence	Tasked with enforcing the Laws and Regulations governing all activities involving petroleum and petroleum products in Guyana.	Guyana Energy Authority (GEA)	Licences are issued after a site visit is conducted to ensure compliance with the Regulations and approved standards.	The Bulk transportation Licence is issued to a person who owns a vehicle capable of transporting an aggregate quantity of 2,000 litres of Petroleum and petroleum products.
Well Installation Permit	For the installation of water wells.	Guyana Water Inc, Hydromet (Dept of Ministry of Agriculture)		
Wharf Facility Permit	For the utilization or commencement of the construction of marine structure for the purpose specified.	Maritime Administration Department (Regional Democratic Council)	Renewed annually	
Tower Installation		Guyana Civil Aviation Authority, Guyana Lands and Survey Commission		
Explosives - Use Permit	This is covered under the Blasting Operations Act which states that someone, the holder of a Certificate of Competency granted by the GGMC to any person deemed by the commission after examination.	Guyana Geology Mines Commission	Blasting Operations Act (Laws of Guyana)	
Radio Frequency Licence (Ground to Aircraft communication)	Regulation in the term of spectrum management is necessary to ensure that this resource is managed and utilised most efficiently and effectively for the country's benefit.	Telecommunications Agency		Renewed Annually at the on or before the Anniversary
Frequency Allocation Permit (Licence for the Installation and Operation of Radio		Telecommunication Agency		

Table 20.2: Required Permit Overview

No.	Permit	Regulatory Body	Legislation
1	Environmental Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
2	Hazardous Waste Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
3	Operation Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
4	Mining Licence	GGMC	Mining (Amendment) Regulations 2005
5	Cyanide Permit	GGMC	Mining (Amendment) Regulations 2005
6	Water Permit (Water Quality Regulation)	EPA	Guyana Water Authority Act (Amendment) 1997
7	Land Permit	GLSC, GGMC	Guyana Mining Act (Amendment 2010) 1989
8	Electrical Installation and Inspection Licence	GEI	Electricity Sector Regulations Act (ESRA 2008)
9	Aerodrome Licence	GCAA	CIVIL AVIATION ACT (Cap. 53:01)
10	Export Licence - Gold	Ministry of Business Investment, Guyana Gold Board	Guyana Gold Board Act Cap. 66:01
11	Bulk Fuel Storage Licence / Consumer Installation Licence	Guyana Energy Authority (GEA)	a) Guyana Energy Agency Act 1997 b) Guyana Energy Agency (Amendment) Act 2004
12	Well Installation Permit	GWI, Hydromet (Dept. of Min. of Agriculture)	
13	Wharf Facility Permit	MARAD – (RDC)	
14	Wharf Facility - Lease Land	GLSC	
15	Tower Installation	GCAA. GLSC	
16	Explosives - Use Permit	GGMC	Blasting Operations Act (Laws of Guyana)
17	Radio Frequency Licence (Ground to Aircraft communication)	Telecommunications Agency	
18	Frequency Allocation Permit (Licence for the Installation and Operation of Radio Equipment)	Telecommunications Agency	
19	Watercourse diversion permit	EPA	Environmental Protection (Authorizations) Regulations 2000

No.	Permit	Regulatory Body	Legislation
20	Radiation Permit	EPA	Environmental Protection (Authorizations) Regulations 2000
21	Transmission Line	Ministry of Natural Resources	Component of building the mine, covered under the Mineral Agreement
22	Deforestation No Objection (No permit needed)	GFC	

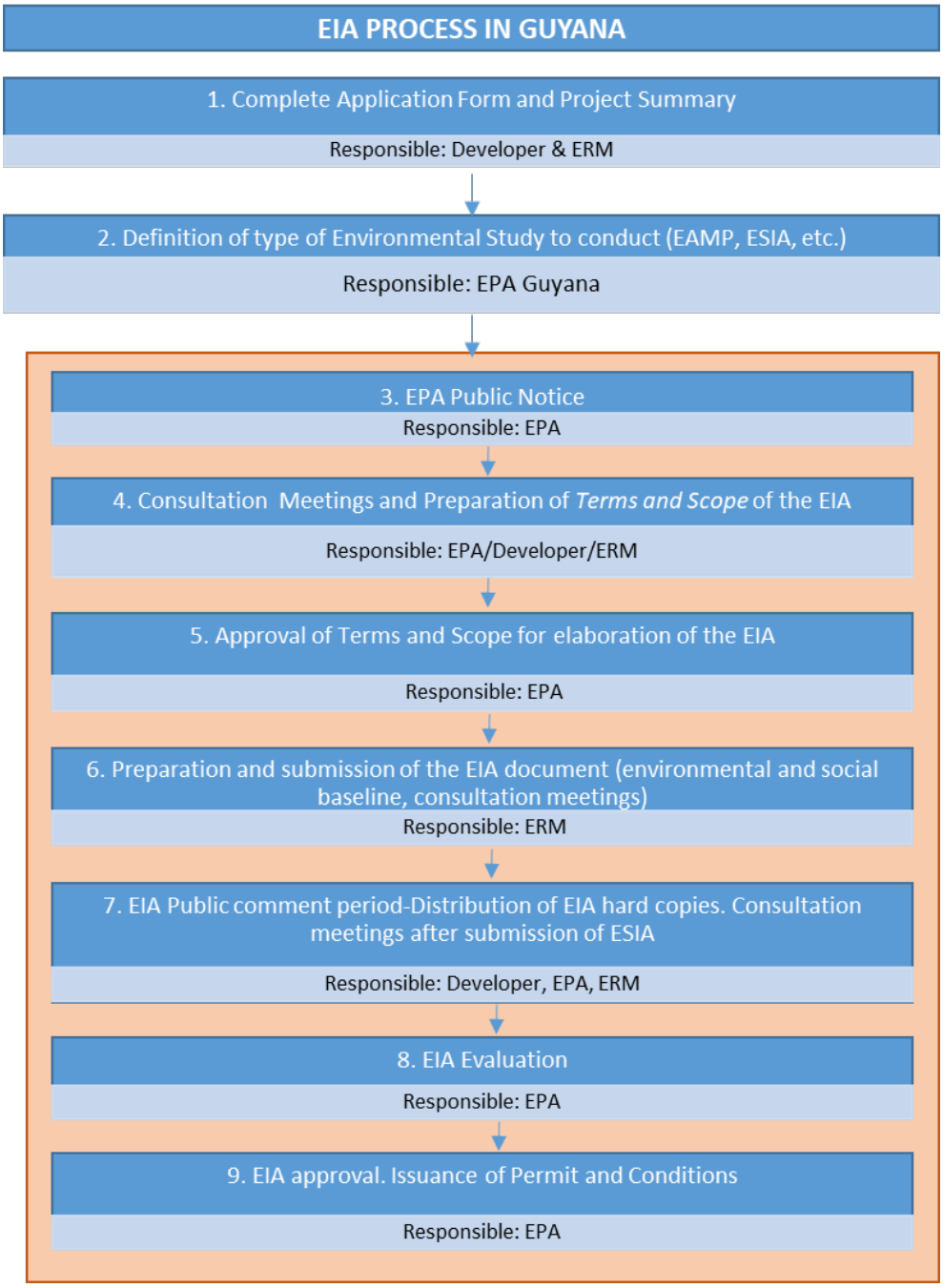
**Note:*

EPA Environmental Protection Agency
GCAA Guyana Civil Aviation Authority
GEA Guyana Energy Authority
GEI Government Electrical Inspectorate
GGMC Guyana Geology and Mines Commission
GLSC Guyana Lands and Surveys Commission
GWI Guyana Water Inc.
MARAD Maritime Administration Department
RDC Regional Democratic Councils.

20.2.2 Regulatory Timelines

Figure 20.1 is a schematic representation of the EIA process for Guyana, governed by the Environmental Protection Agency (EPA). It is ERM's understanding that regulatory timelines are not legally mandated. Reunion Gold has completed steps 3 to 5 of this process and is currently in the process of preparing an EIA. It is ERM's further understanding there is an intent to file the EIA by the end of 2024. Reunion Gold has indicated they are anticipating a possible EIA approval and permit and conditions issuance within 6 months after submission.

Figure 20.1: EIA Process for Guyana



20.3 Mine Closure

The project is currently at the PEA stage and considering this, initial conceptual closure planning has been initiated, and a conceptual closure cost estimate has been developed. Financial assurance for closure and reclamation of the existing program (current footprint) is not a regulatory requirement in Guyana and is not discussed here.

A Mine Closure Plan is being developed and will be advanced during future stages of study. A Mine Closure Plan (MCP) is an all-encompassing document that describes the nature of the operations that will be carried out, current baseline environmental conditions, potential effects on the environment together with appropriate mitigation measures, and the Company's plan for rehabilitating the site to its natural state at the end of operations.

Currently, closure planning has consisted of identifying closure risks and progressive closure opportunities, setting parameters around how each domain should be closed within the site context, and using this as input into the PEA design. It should be noted that the closure case that has been costed is reliant on there being sufficient materials for closure activities available, such as clean waste rock and topsoil. Another key factor that requires further consideration include waste rock, ore, and tailings geochemistry, and how these materials will be managed, as well as how the geochemistry may affect the water quality of the expected post-closure pit lake. In terms of social factors, closure planning should be advanced in proactive consultation with various government ministries, Amerindian and local communities, who will drive the environmental, social, and economic vision for the land post-closure.

The Guyana Geology and Mines Commission has developed several Codes of Practice related to closure planning. These are listed below and should be considered as the project progresses:

- Mine Reclamation and Closure Plans
- Contingency and Response Plans
- Mercury
- Tailings Management
- Use of Small Dams – Water / Tailings Management
- Waste Management

In addition, in accordance with the EP Hazardous Waste Management Regulations (Environmental Protect Agency), the ESIA under development will need to contain information regarding the programme for rehabilitation and restoration of the environment following closure of the project.

To support the EPA’s decision to issue an environmental permit, Reunion will have to demonstrate conditions which are reasonably necessary to protect human health and the environment and will therefore be obligated to restore and rehabilitate the environment following mine closure.

Mine closure activities will need to include:

- Rendering the site both environmentally and physically stable.
- Capping, covering, and revegetating disturbed land and constructed landforms.
- Decommissioning, decontamination and demolition of site infrastructure.
- Maintenance and monitoring activities post-closure.

A conceptual closure cost estimate has been developed in consultation with ERM for the purposes of the PEA. The estimate is included in Section 21.1.

21 CAPITAL AND OPERATING COSTS

Life-of-mine project capital costs are estimated to total USD 1.510 billion consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 6 Mtpa of fresh rock. Initial capital costs total USD 936.2M (including $100.3 million for contingency and net of $28.8 million in pre-production revenue), which will be expended over 32-month of engineering, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations and the underground development. Sustaining capital costs are estimated to be $537.5 million including indirect costs but excluding a contingency.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs $36.6 million and includes a 20% contingency. The capital and sustaining expenditures are summarized in Table 21.1 according to the Level 1 work breakdown structure ("WBS").

Table 21.1: Initial and Sustaining Capital Expenditures Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
100 – Infrastructure	70,763	5,091	**75,854**
200 - Power and Electrical	118,243	25,598	**143,841**
300 – Water Management	16,318	11,267	**27,585**
400 – Surface Operations	45,952	-	**45,952**
500 - Mining	128,910	447,518	**576,428**
600 - Process Plant	190,010	22,000	**212,010**
700 - Construction Indirect	107,496	-	**107,496**
800 - General Services / Owner's Cost	111,432	-	**111,432**
900 - Pre-production, Start-up, Comm.	46,746	26,020	**72,766**
990 - Contingency	100,304	-	**100,304**
Total	**936,174**	**537,494**	**1,473,668**

21.1 Initial Capital Expenditures

21.1.1 Basis of Estimate

The capital cost estimate is according to AACEI standard Class 4 and is accurate to a -10% / +20% range. The base date of the CAPEX estimate is Q2-2024. The initial capital expenditure ("CAPEX") duration is planned over a period of 32 months, assumed from May 2025 to end of December 2027. The CAPEX estimate is aligned with an owner-managed project delivery model.

Expenditures are planned in few other currencies with the native currencies retained as part of the estimate. The initial CAPEX estimated is presented in US dollars and no allowance for escalation or exchange rate fluctuation were used. An exchange rate of 0.73 (C$:US$) has been applied as necessary. Expenditures are planned in few other currencies with the native currencies retained as part of the estimate.

The capital cost estimate is a detailed, bottoms-up, built-up effort by major facility and discipline based on G Mining's in house database of executed projects and studies as well as experience from similar operation. In some cases, a detailed cost build-up by cost type consisting of labour, material, construction equipment, consumables, construction materials, and services costs was completed based on material takeoffs from drawings and concepts. According to standards established at the outset of the Project, pricing of equipment, material and labour were estimated according to the following guidelines:

- Infrastructures were developed from GMS database in 2024 US dollars.
- The concrete and structure cost in these infrastructures were based on the executed MTO with unit cost per cubic metre and tonnes of steel respectively. The cost per cubic metre of concrete was revised from benchmark to material quotation received in Guyana
- Earthworks quantities and deforestation were based on take offs from the general arrangement drawings with benchmarked unit costs from executed projects.
- Equipment proposals for major, high value and long lead items were specified and quoted specifically for the Project; equipment prices for minor items were derived from recent projects or from GMS' current database.
- Specific material prices were based on quotations received from suppliers for sheet piling and aggregate for example.
- Labour rates were derived from the TZ operation, the GMS expatriate database, and wage surveys conducted in Guyana during the PEA study. The labor strategy and construction equipment costs for the Project were developed through separate analyses, specifically for the indirect costs.

21.1.2 Initial CAPEX Summary

The initial CAPEX is estimated at USD 936.2M. The capital expenditure is summarized in

No residual value was estimated at this stage of the study.

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at USD 36.6M OPEX and is summarized in Table 1.3. The OPEX includes mining, processing, general services and administration (G&A), transportation, refining and royalties. Power costs are separated from processing costs. The average OPEX is USD 853/oz Au or USD 51.15/t milled over the LOM. The all-in-sustaining costs (AISC) which includes closure, reclamation and sustaining capital costs averages USD 986/oz Au or USD 59.13/t milled.

Table 1.3: Opex Costs

Operating Costs per Tonnes Milled		
Open Pit Mining Cost	t/milled	13.13
Underground Mining Cost	t/milled	10.76
Material Rehandling	t/milled	0.15
Processing Cost	t/milled	9.04
Power Cost	t/milled	5.93
General & Administration Cost	t/milled	4.14
Refining Cost	t/milled	0.48
Total Site Cost	**t/milled**	**43.62**
Royalty Cost	t/milled	7.53
Total OPEX Cost	**t/milled**	**51.15**
Sustaining Capital	t/milled	7.19
Closure Cost	t/milled	0.49
Land Payments	t/milled	0.30
All-In Sustaining Costs (AISC)	**t/milled**	**59.13**

Operating Costs per Payable Ounce		
Open Pit Mining Cost	t/oz	219
Underground Mining Cost	t/oz	179
Material Rehandling	t/oz	2
Processing Cost	t/oz	151
Power Cost	t/oz	99
General & Administration Cost	t/oz	69
Refining Cost	t/oz	8
Total Site Cost	**t/oz**	**728**
Royalty Cost	t/oz	126
Total OPEX Cost	**t/oz**	**853**
Sustaining Capital	t/oz	120
Closure Cost	t/oz	8
Land Payments	t/oz	5
All-In Sustaining Costs (AISC)	**t/oz**	**986**

according to the Level 1 work breakdown structure ("WBS"). WBS Areas 100 to 600 include the Project's direct costs, while WBS Areas 700 to 900 cover indirect costs, owner's costs, and pre-production costs. This amount includes pre-production revenues of approximately USD 31.5M for 16 koz of gold recovered during commissioning.

The CAPEX includes a contingency of USD 100.3M, which is 12% of the total before contingency or 18% of the direct costs for WBS areas 100 to 600. The total hours for construction for the initial CAPEX phase is estimated at 8.5M hours.

Table 21.2: Initial Capital Expenditures Summary

Capital Expenditures (k USD)	Initial Capital Cost
100 – Infrastructure	70,763
200 - Power and Electrical	118,243
300 – Water Management	16,318
400 – Surface Operations	45,952
500 - Mining	128,910
600 - Process Plant	190,010
700 - Construction Indirect	107,496

800 - General Services / Owner's Cost	111,432
900 - Pre-production, Start-up, Comm.	46,746
990 - Contingency	100,304
Total	**936,174**

21.1.3 Infrastructures

The CAPEX estimates for WBS 100 – Infrastructure is summarized in Table 21.3. the detailed description of infrastructures is presented in Section 18.

Table 21.3: Infrastructures Capital Expenditures

Capital Expenditures (k USD)	
100 – EL - Electrical Distribution	5,943
110 – Roads, Bridges and Fencing	**11,998**
111 – General Earthwork (DEFORESTATION)	4,899
112 – Site Roads	1,572
113 – External Site Roads	2,192
114 – Bridges	600
116 – Fencing	762
117 – Airstrip	1,972
120 – Mine Infrastructure	**15,906**
121 – Mine Dry (sustaining)	-
122 – Mine Maintenance Facility	10,000
123 – Wash Bay	1,344
124 – Explosive Magazine	3,649
126 – Mine Dispatch Office	914
130 – Support Infrastructure	**8,429**
131 – Administrative Building	2,336
132 – Site Guard House	326
135 – Laydown	398
137 – Assay Lab	5,368
140 – Camp Facilities	**19,016**
141 – Camp Dorms	14,245

Capital Expenditures (k USD)	
142 – Kitchen	3,306
143 – Camp Office / Welcome Centre	287
144 – Laundry	568
145 – Recreational Room	338
146 – Recycling / Sort Facility	147
147 – Domestic Waste	43
148 – Sports Field	81
160 – Process Plant Infrastructure	**4,801**
161 – Process Search House	336
164 – Reagents Storage Building	1,157
165 – Mill Office	1,393
166 – Work Shop	1,914
170 – Fuel Systems Storage	**-**
190 – Offsite Infrastructure	**4,669**
Total	**70,763**

The fuel storage systems, both on site and at the barge landing, were included in the power generation costs, covering fuel transfer from the main storage tanks to the small day tanks of the temporary generators, which will be used for emergency power during operations.

21.1.4 Power Supply and Electrical

The CAPEX estimates for WBS Area 200 – Power Supply and Electrical is summarized in Table 21.4.

The main electrical costs for power generation were derived using a direct cost-based strategy based on genset quotations received, with ancillaries such as fuel tanks, concrete, and structural buildings included. The costs for the power transmission line were based on a cost-per-kilometre benchmark from other executed projects. For the main substation, pricing was sourced from the GMS database.

All other electrical costs were benchmarked based on the cost per tonne of material mined and processed. This includes distribution costs as well as the costs for electrical rooms and their equipment. Additionally, the automation network, including process plant instrumentation, was factored from the mechanical equipment costs.

Table 21.4: Power Supply and Communications Capital Expenditures

Capital Expenditures (k USD)	
200 – POWER & ELECTRICAL	118,243
210 – Main Power Generation	81,222
211 – Offsite Substation / HFO Power Plant	62,790
212 – Power Transmission Line 69 kV, 50 km	9,375
213 – Site Main Substation step down TRF	9,056
230 – Water Management Electrical Rooms	300
240 – Infrastructure distribution	4,343
260 – Process Plant Electrical	19,358
270 – Medium Voltage (MV) Distribution O/H Line	2,869
280 – Automation Network	10,151

21.1.5 Water Management

Capital expenditures summary for water management is presented in Table 21.5.

Tailings storage facility construction costs are based on a material take-off (MTO) approach, supported by TEC3, using benchmarked unit costs from recently executed projects. The remaining water management costs are factorized from mechanical equipment and direct infrastructure costs.

Table 21.5: Water Management Capital Expenditures

Capital Expenditures (k USD)	
300 – WATER MANAGEMENT	16,318
310 – Fresh water Intake / Wells	188
320 – Water Ponds and Water Management	3,495
330 – Domestic Water	847
340 – Sewage Water	1,174
350 – Fire Protection	4,332
360 – Effluent Water Treatment	939
370 – Tailings Storage Facility (TSF)	5,343
371 – Tailings Storage Facility - Bassin Preparation (Foundation)	1,738
373 – Tailings Storage Facility - Starter Dam (Elevation: 114.00)	2,927
375 – Tailings Storage Facility - South Dike (Elevation: 114.00)	677

21.1.6 Surface Equipment

A summary for the capital expenditures for mobile equipment, concrete batch plant and aggregate plant is presented in Table 21.6.

Construction mobile equipment includes purchasing costs for lifting equipment, utility vehicles and specialized construction equipment.

Table 21.6: Surface Equipment Capital Expenditures

Capital Expenditures (k USD)	
EL - Surface Ops Electrical distribution	820
410 - Surface Operations Equipment	**40,752**
411 - Construction Mobile Equipment	10,784
412 - Process Plant Mobile Equipment	47
414 - G&A Mobile Equipment	3,320
415 - Off-Site Surface Ops Equipment	8,901
416 - Marine Equipment	17,700
430 - Concrete Batch Plant	**390**
480 - Aggregate Plant	**3,990**
Total	**45,952**

21.1.7 Mining

The capital costs estimate for the mining areas are presented in Table 21.7. The costs are based on an owner-operated mine fleet.

Table 21.7: Mining Capital Expenditures

Capital Expenditures (k USD)	
540 - Mine Infrastructure	12,991
541 - Haul Road	5,775
542 - Electrical Distribution - Earthwork	7,215
550 - Mine Equipment	115,920
551 - Primary Mining Equipment	91,731
554 - Other Mining Support Equipment	15,289
559 - Dispatch / Comm. / Software / Survey	8,899
Total	128,910

21.1.8 Process Plant

The estimated capital costs for the processing area are presented in Table 21.8. The detailed description of process plant is presented in Section 17. Direct costs include all direct labour, permanent equipment and materials but exclude mobile equipment and freight that are captured in the indirect costs (700s) and G&A costs (800s) respectively.

Each major process area was estimated separately addressing the following disciplines:

- Earthworks (EW) quantities based on the topographical information.
- Concrete (CI), Structure (ST) & Architecture (AR) factored from GMS database benchmarked against similar executed projects with equivalent process technologies.
- Mechanical (MC) small equipment, platework, tanks and piping (PI) factored from major equipment quoted.
- Electrical material (EL) and instrumentation (IN) as explained in the Power & Electrical area.

Table 21.8: Processing Capital Expenditures

Capital Expenditures (k USD)	
600 - PROCESS PLANT	190,010
EL - Process Electrical distribution	6,762
601 - Site Preparation / Road / Berms	10,480
604 - ROM Pad	1,766
610 - Comminution	82,231
620 - Gravity & Intensive Leach	6,195
640 - Lixiviation	44,394
650 - Reagents	5,999
660 - Refinery	19,384
680 - Tailings Management	5,746
690 - Process Plant Services	7,053

21.1.9 Construction Indirects

Construction Indirect costs are presented in Table 21.9. They are based on GMS's historical executed project costs and from first principles with benchmarked infrastructure and MTOs. each single cost code has a detailed explanation.

Table 21.9: Construction Indirect Capitals

Capital Expenditures (k USD)	
700 - EL - Electrical distribution	3,279
710 - Engineering, CM, PM	**33,117**
711 - Site CM Staff and Consultants	26,534
713 - Surveying	2,275
714 - QA/QC	2,410
715 - Induction	795
716 - Project Control	1,103
720 - Construction Offices, Facilities & Services	**2,509**
721 - Construction Offices / Trailers	260
722 - Temporary Truckshop	367
723 - Temporary Explosive Magazine	103
724 - Temporary Laydown and tool cribs	762
725 - Camp Construction Temporary Facilities	750
727 - Site Toilets / Ablution Units	215
729 - Construction Temp Water and Piping Network	51
730 - Shops	**902**
731 - Fab Shop	210
733 - Carpentry, Rebar and Civil	126
737 - Lifting Equipment	565
740 - Construction Equipment & Tools	**20,908**
741 - Owned Equipment	2,826
743 - Operation and Maintenance	4,001
744 - Construction Tools	11,242
746 - LOTOTO Team	507
747 - EPP for Construction	1,681

Capital Expenditures (k USD)	
748 - Scaffolding	650
760 - Energy	**21,941**
790 - External Engineering	**24,840**
Total	**107,496**

21.1.10 General Services

Part of General Services include the basic services listed in Table 21.10.

Table 21.10: List of Departments Included in General Services

General Services
General Management & Legal
Finance & Accounting
Information Technology (IT)
Sustainability & Permitting
Health & Safety
Supply Chain
Human Resources & Training
Security

The various departments provide support to construction, mining, power and processing and their efficient operations are essential. In the case of a remote project like Oko West, two other sectors are particularly important:

- Site Services:
  Site Services include the camp services (kitchen, housing & laundry) and the regular site maintenance for all building and offices. It also ensures transportation of all workers on site to their home and back according to their rotations.
- Logistics:
  It includes the cost of transportation of goods from the suppliers at offshore or domestic locations to the project sites. For Oko West it is expected that most materials and equipment will be imported

from overseas and transported on ocean ships to Guyana then transferred to barges to reach a dedicated landing and then finally transferred to tractor/trailer for final travel to site.

Capital expenditures for the General Services listed in Table 21.10 during project development will progressively increase to the peak of construction and will reduce afterwards to a steady level for the operating period. Table 21.11 presents the estimated monthly expenditures over the 32 months of the Project.

Table 21.11: General Services Expenditures

Capital Expenditures (k USD)	
810 - G&A Departments	31,950
820 – Logistics / Taxes / Insurance	37,403
822 - Sea Freight	33,434
824 - In-Country Barging Land Freight	3,969
830 - Operating Expenses	42,079
831 - Site Services (Camp OPEX)	29,455
832 - Travel & Transportation	12,264
Total	**111,432**

Expenditures for Site Services and Transportation will vary with manpower on site for project development. Site services costs are estimated at $35 per man-day and the transportation costs at $15 per man-day.

Equipment and materials to be purchased for sustaining capital carry an 8% allocation for logistics.

21.1.11 Pre-production and Commissioning Expenditures

The initial open pit operation contributes for most of the pre-production expenditures with a smaller charge for Processing during its preparation and training for commissioning. The initial underground operation will come fully on stream in the fourth year of the mine commercial production and its pre-production expenditures will be accounted in sustaining capital.

Pre-Production and Commissioning expenditures are listed in Table 21.12.

Table 21.12: Pre-Production & Commissioning CAPEX

Capital Expenditures (k USD)	
910 - Mining Pre-Prod	59,051
912 - Mine Engineering	4,680
913 - Mine Geology	3,261
914 - Mine Operations Pre-prod	37,275
916 - Mine Maintenance	4,620
919 - Mine Support Operations	9,215
950 - Process Plant Pre-Production	-14,655
952 - Process Plant Pre-Production	9,187
953 - Process Plant Commissioning & Construction Support	3,980
954 - Vendor Reps	1,000
955 - Pre-Prod. Revenue (Net)	-28,822
960 - First Fill, Spares & Consumables	2,350
961 - Spare Capital Parts	1,500
965 - First Fill	850
Total	**46,746**

Pre-Production mining costs will be incurred during the two years prior to the start of commercial production; a tonnage of 3,528 kt of material and 28,377 kt of waste will be mined during this 2-year period.

Pre-Production processing costs include a 5-month period with full processing personnel for training and commissioning. The 2-month period prior to commercial production will include all processing costs required to hot commission the plant assuming 25% and 50% of nameplate throughput for those two (2) months before declaring commercial production.

The power costs incurred by the site power plant during commissioning are included in the pre-production expenditures.

Some three (3) months of construction labour in support of commissioning were included in the commissioning costs.

An estimate of $1,000,000 was also included for vendor representatives.

Finally, it is estimated that 16,140 oz of gold will be recovered during hot commissioning. Net revenues of $28,822,000, after deducting royalties and refining costs, are credited against pre-production expenditures.

21.1.12 Contingency

Contingency has also been included to the cost estimate per area basis as a deterministic allowance by assessing the level of confidence of the scope definition and cost, and then applying a Monte Carlo iteration analysis. The overall recommended contingency was 12% of direct and indirect expenditures for an amount of $100.3M.

Estimates relied on our extensive experience with well-known executed activities in similar environments. Also, our self-perform model allows for a lower risk of execution compared to a typical EPCM approach with contractors. This approach has allowed us to effectively manage potential uncertainties and execution issues. The project's scope and schedule have been well-defined and stable, further reducing the likelihood of major changes. By leveraging historical data and lessons learned from analogous projects, optimized the contingency allocation while ensuring robust risk management and project success.

21.2 Sustaining Capital

Sustaining capital is presented in Table 21.13. Sustaining capital for the mine includes additional equipment purchases for the open pit operations and the initial and sustaining equipment for the underground operations for an overall mining requirement of $473.5M. Major equipment repairs were kept in the operating costs.

Additional work is required for raising the main embankment of the tailing’s storage facility (“TSF”) and the construction of the North dike of the TSF to increase capacity. The continued raising of the TSF will be completed by the mine operations team with fill material from the open pit mine or nearby borrow sources. A mine dry building for the underground personnel will be constructed in Year 2 and also all the electrical infrastructures to support the underground development. The sustaining capital is estimated at USD 64M for these other activities.

Table 21.13: Sustaining Cost Summary

Sustaining Cost Summary (k USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
MINING														
Open Pit	**216,261**	39,771	6,510	19,005	23,077	14,602	17,720	20,107	25,957	16,300	10,920	11,968	7,223	3,101
Initial Underground	**124,132**	59,663	64,469											
Sustaining Underground	**133,145**			33,559	30,050	27,146	16,160	6,100	6,650	2,250	6,716	3,550	964	
Total Mining	**473,538**	99,434	70,979	52,564	53,127	41,748	33,880	26,207	32,607	18,550	17,636	15,518	8,187	3,101
OTHER COST														
110 - Roads, Bridges and Fencing	**3,761**	2,212	1,225	324										
120 - Mine Infrastructure	**1,329**		1,329											
210 - Main Power Generation	**14,439**	9,626	4,813											
250 - Mine Electrical Room	**6,960**	1,740	1,740	696	696	696	696	696						
260 - Process Plant Electrical Rooms	**4,200**	4,200												
370 - Tailings Storage Facility (TSF)	**11,267**	1,116	1,149	1,267	1,188	1,042	1,011	1,011	1,120	1,130	1,233			
610 - Comminution	**22,000**	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000		
Total Other Cost	**63,956**	20,894	12,256	4,287	3,884	3,738	3,707	3,707	3,120	3,130	3,233	2,000	0,000	0,000
Grand Total	**537,494**	**120,328**	**83,235**	**56,851**	**57,011**	**45,486**	**37,587**	**29,914**	**35,727**	**21,680**	**20,869**	**17,518**	**8,187**	**3,101**

21.3 Closure Costs

The closure costs are estimated to be USD 36.6M as summarized in Table 21.14. Closure costs would cover the following activities:

- Regrading, covering and revegetating of the WSF.
- De-coupling, dismantling and removal of underground infrastructure and equipment, backfilling and plugging openings, allowing to flood naturally.
- Covering and revegetating the TSF.
- Render open pit physical and geochemically stable, installing access barriers and allowing to flood naturally.
- Full decommissioning, decontamination and demolition of processing infrastructure. Footprints regraded and revegetated.
- Decontamination, decommissioning and demolition of all accommodation camp infrastructure.
- Haul roads scarified and revegetated.
- Post-closure maintenance and monitoring, including cover/revegetation, biodiversity, surface and groundwater monitoring, and cap, cover and revegetation maintenance.
- Implementation of social engagement and transition plans.

Table 21.14: Closure Cost Summary

Closure Cost	Cost (k USD)
WSF	3,495
Underground	250
Open Pit	627
TSF	11,655
Processing Infrastructure	5,150
Processing Infrastructure (Scrap Value)	-515
Accommodation Camp	130
Haul Roads	295
Transmission Lines	1,817
Maintenance and Monitoring	1,810
Sub-Total (Directs)	24,715

Closure Cost	Cost (k USD)
Social / Communities (3%)	741
Studies and Indirects (15%)	3,707
Project Management (10%)	2,471
Contingency (20%)	4,943
Total	36,578

For the purposes of scheduling, closure is broken down into four phases, described below, and visually presented in Table 21.15:

- Phase 1 – Progressive Closure: Progressive closure of the WSF and implementation of social engagement and transition plans in the preceding five (5) years to final mining activity.
- Phase 2 – Safety Measures and Bulk Earthworks: Closure of the underground, open pit, TSF and WSF in the five (5) years following the cessation of Mining activity.
- Phase 3 – DD&D, Scarification and Revegetation: Closure of processing infrastructure, accommodation camp, haul roads and transmission lines in the three (3) years following the cessation of processing activity.
- Phase 4 – Post-closure maintenance and monitoring: Maintenance and monitoring activities carried out for ten (10) years post-closure.

Table 21.15: Closure Schedule

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
Phase 1																												
Phase 2																												
Phase 3																												
Phase 4																												

It should be noted that the costs at this level of study having the following notes, assumptions and exclusions:

- An opportunity exists in the form of transferring the transmission lines to a new owner through strong and proactive stakeholder engagement.
- General and administrative (G&A) costs throughout the closure period are excluded.
- It is assumed that the required closure materials will be available on-site at closure.
- Closure and reclamation of borrow areas are excluded.
- Closure costs for the access road, airstrip and barge landing wharf are excluded, under the assumption that they will all be transferred to a new owner or donated to the local community. If these liabilities were to be realised it is estimated that they would total to US$2.5M.

These opportunities, assumptions and exclusions will need to be further assessed and evaluated at the next stage of study.

21.4 Operating Costs

Operating Costs are summarized in Table 21.16 and presented by year in Table 21.17. The operating costs include mining, processing, power, general services and administration (“G&A”), gold transportation and refining and royalties. The average LOM operating cost is USD 728/oz of gold or USD 43.62/t milled, excluding royalty costs. The average LOM all-in sustaining cost (“AISC”) is USD 986/oz of gold or USD 59.13/t milled.

Table 21.16: Operating Costs Summary

Cost Summary	Total LOM Cost (M USD)	Unit Cost (USD/t milled)	Cost per oz (USD/oz)
Open Pit Mining Cost	981	13.13	219
Underground Mining Cost	804	10.76	179
Material Rehandling	11	0.15	2
Processing Cost	676	9.04	151
Power Cost	444	5.93	99
General & Administration Cost	310	4.14	69
Refining Cost	36	0.48	8
Total Site Cost	3,261	43.62	728
Royalty Cost	563	7.53	126

Cost Summary	Total LOM Cost (M USD)	Unit Cost (USD/t milled)	Cost per oz (USD/oz)
Total OPEX Cost	3,824	51.15	853
Sustaining Capital	537	7.19	120
Closure Cost	37	0.49	8
Land Payments	22	0.30	5
Exploration	-	-	-
All-In Sustaining Costs (AISC)	4,421	59.13	986

Table 21.17: Total Operating Costs Summary by Year

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Production Highlights															
Tonnage Milled	kt	**74,762**	6,408	7,000	7,000	7,000	6,000	6,000	6,000	5,822	5,000	5,000	5,000	5,000	3,531
Tonnage Mined MM OP	kt	**57,174**	7,509	6,448	5,858	3,571	4,655	4,011	4,432	4,438	3,278	3,489	3,518	3,572	2,397
Tonnage Mined MM UG	kt	**14,501**	40	67	286	946	1,345	1,595	1,568	1,562	1,545	1,511	1,482	1,428	1,125
Tonnage Mined Waste OP	kt	**336,266**	34,614	35,148	35,042	36,308	39,345	36,613	24,637	28,348	29,201	20,971	8,931	5,793	1,315
Recovered Gold	koz	**4,484**	317	324	339	325	330	361	340	385	319	331	390	384	340
Payable Gold	koz	**4,482**	317	324	339	325	330	361	340	384	318	330	390	383	340
Operating Costs															
Open Pit Mining	M USD	**981**	76	80	89	93	95	94	80	89	90	78	53	43	22
Underground Mining	M USD	**804**	-	-	36	63	83	87	83	88	83	80	79	72	49
Material Rehandling	M USD	**11**	1	1	1	2	1	1	1	1	0	1	1	1	0
Processing	M USD	**676**	55	61	61	60	54	54	54	53	47	47	47	47	36
Power	M USD	**444**	27	37	38	36	38	37	38	37	33	32	33	32	24
General & Administration	M USD	**310**	24	24	24	24	24	24	24	24	24	24	24	24	24
Refining	M USD	**36**	3	3	3	3	3	3	3	3	3	3	3	3	3
Total Site Cost	M USD	**3,261**	186	205	252	281	297	300	283	295	280	264	239	222	159
Royalty Cost on Revenues	M USD	**563**	49	50	51	44	39	40	39	45	36	38	46	45	41
Total OPEX Cost	M USD	**3,824**	235	255	302	325	336	340	321	340	316	303	285	267	199
Sustaining Capital	M USD	**537**	120	83	57	57	45	38	30	36	22	21	18	8	3
Closure Cost	M USD	**37**	-	-	-	-	-	-	-	-	-	-	-	-	37
Land Payments	M USD	**22**	22	-	-	-	-	-	-	-	-	-	-	-	-
Exploration	M USD	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	M USD	**4,421**	378	339	359	382	382	378	351	376	338	324	302	275	239
Unit Operating Costs															
Open Pit Mining Cost	t/mined	**2.49**	1.81	1.92	2.16	2.33	2.16	2.31	2.76	2.71	2.79	3.19	4.24	4.56	5.97
Underground Mining Cost	t/mined	**55.45**	-	-	126.47	66.24	61.67	54.82	52.72	56.39	54.04	53.07	53.39	50.58	43.65
Material Rehandling	t/milled	**0.15**	0.14	0.19	0.20	0.31	0.11	0.14	0.11	0.11	0.10	0.11	0.11	0.11	0.13
Processing Cost	t/milled	**9.04**	8.64	8.67	8.68	8.60	9.04	9.00	9.04	9.09	9.36	9.36	9.36	9.36	10.17
Power Cost	t/milled	**5.93**	4.21	5.28	5.48	5.19	6.36	6.18	6.35	6.38	6.52	6.50	6.51	6.49	6.89
General & Administration Cost	t/milled	**4.14**	3.72	3.40	3.40	3.40	3.97	3.97	3.97	4.09	4.77	4.76	4.76	4.76	6.75
Refining Cost	t/milled	**0.48**	0.40	0.37	0.39	0.37	0.44	0.48	0.45	0.53	0.51	0.53	0.62	0.61	0.77
Total Site Cost	t/milled	**43.62**	28.98	29.34	35.97	40.09	49.56	49.96	47.09	50.59	56.05	52.90	47.74	44.32	44.90

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13+
Royalty Cost	t/milled	**7.53**	7.66	7.14	7.22	6.28	6.48	6.70	6.42	7.78	7.15	7.68	9.22	9.07	11.58
Total OPEX Cost	t/milled	**51.15**	36.64	36.48	43.18	46.37	56.05	56.66	53.51	58.37	63.20	60.58	56.96	53.39	56.48
Sustaining Capital	t/milled	**7.19**	18.78	11.89	8.12	8.14	7.58	6.26	4.99	6.14	4.34	4.17	3.50	1.64	0.88
Closure Cost	t/milled	**0.49**	-	-	-	-	-	-	-	-	-	-	-	-	10.36
Land Payments	t/milled	**0.30**	3.51	-	-	-	-	-	-	-	-	-	-	-	-
Exploration	t/milled	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	t/milled	**59.13**	58.93	48.37	51.30	54.51	63.63	62.93	58.49	64.51	67.54	64.75	60.47	55.03	67.72

21.4.1 Mining Costs

A detailed mine cost build up was developed from basic cost elements such as remuneration costs, consumable prices, fuel prices and equipment productivities.

Equipment operating costs were estimated for each equipment model, which includes operation and maintenance labour, parts (maintenance and repairs), fuel consumption, lubricant consumption, ground engaging tools or tires if applicable. Equipment operating costs were determined from various sources including primarily information from the major suppliers and benchmarked costs from operations in similar environments.

The diesel fuel price assumed for estimating mining costs is USD 0.8539/L (Table 21.18).

Table 21.18: Diesel Fuel Price

Diesel Fuel Price Parameters	Unit	Value
Product	-	Diesel Low Sulfur (USGC No. 2)
Brent Cost	USD/bbl	80
	USD/L	0.64155
Additional Costs	USD/L	0.21233
Refining	US¢/L	4.95
Freight & Insurance	US¢/L	4.81
Barge to Cujuni Landing	US¢/L	2.38
Land Transport	US¢/L	2.50
Fuel Premium	US¢/L	0.29
Profit Margin	US¢/L	6.30
Total Diesel Cost	USD/L	0.8539

The open pit mining unit cost is $2.49/t mined. The underground mining unit cost is $55.45/t MM mined. Table 21.19 presents the breakdown of the open pit mining costs by department while Table 21.20 presents the underground mining cost elements.

Table 21.19: OP Mining Cost Summary

Mining Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operations	48.5	3.9	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	2.8	2.5	1.2
Mine Geology	24.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.7	0.8
Mine Maintenance Admin.	54.9	4.6	4.7	4.6	4.7	4.7	4.7	4.9	4.6	4.7	4.7	3.2	2.9	1.8
Mine Engineering	46.8	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.6	2.9	1.2
Drilling	101.8	6.2	7.7	10.2	11.0	10.9	11.1	8.7	9.6	9.2	7.3	4.4	3.6	1.9
Pre-Split Drilling and Blasting	14.6	2.0	1.2	1.8	1.3	1.0	1.5	1.0	1.2	1.1	0.9	0.5	0.6	0.3
Blasting	150.1	9.3	10.9	14.9	16.8	16.2	16.7	12.9	14.3	14.2	10.5	6.0	4.9	2.5
Loading	90.0	9.3	9.7	9.6	10.0	9.5	9.0	6.5	7.2	7.0	5.5	3.1	2.4	1.2
Hauling	271.5	21.2	20.5	21.6	23.5	26.8	25.2	21.2	26.7	29.1	25.1	14.1	11.4	5.0
Dewatering	16.5	0.2	0.4	1.0	1.0	1.0	1.0	1.5	1.5	1.5	1.5	2.0	2.0	1.8
Dump Maintenance	35.6	3.0	3.5	3.5	3.5	3.5	3.5	2.8	2.8	2.8	2.8	2.1	1.4	0.7
Road Maintenance	47.1	3.7	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	3.5	3.0	2.3	1.2
Grade Control	6.1	0.7	0.7	0.6	0.5	0.6	0.5	0.5	0.5	0.4	0.4	0.3	0.3	0.2
Support Equipment	70.0	5.6	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.5	5.5	3.7	2.4
Crushing for Backfill and Roadbed	17.2	0.4	0.5	0.3	1.2	1.2	1.4	1.7	1.7	1.8	1.8	1.8	1.8	1.6
Transfer to UG	(13.2)	-	0.1	0.2	(0.9)	(0.9)	(1.0)	(1.4)	(1.5)	(1.5)	(1.5)	(1.5)	(1.6)	(1.6)
Total OP Mining Cost	981.5	76.1	80.0	88.5	92.9	94.8	93.7	80.4	88.9	90.5	78.0	52.8	42.7	22.2
Unit Cost ($/t Mined)	2.49	1.81	1.92	2.16	2.33	2.16	2.31	2.76	2.71	2.79	3.19	4.24	4.56	5.97

Table 21.20: UG Mining Cost Summary

Mining Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total Diamond Drilling & Geology	49.6	-	-	1.0	3.3	4.7	5.5	5.4	5.4	5.3	5.2	5.1	4.9	3.8
Total Stope Preparation	68.0	-	-	7.4	7.4	9.8	9.9	8.3	7.8	6.7	4.9	2.4	3.2	0.3
Total Drilling & Blasting	47.7	-	-	1.0	3.3	4.3	5.1	5.1	5.2	5.1	5.1	5.0	4.7	3.9
Total Mucking & Hauling	84.2	-	-	1.1	3.9	7.0	8.8	8.9	9.6	9.3	9.3	9.2	9.3	7.7
Total Backfilling	184.0	-	-	2.3	9.2	20.5	21.5	18.5	23.0	20.4	19.4	21.2	17.2	11.0
Total Supervision	69.8	-	-	3.8	7.0	7.5	7.5	7.0	7.5	7.0	7.0	7.0	5.3	3.4
Total Mine Services	111.9	-	-	6.9	10.8	11.1	11.1	11.1	11.1	11.1	11.0	11.0	9.2	7.5
Total Maintenance Services	61.5	-	-	5.6	5.7	5.8	5.9	5.8	5.8	5.8	5.7	5.7	5.7	4.1
Total Electrical Services	41.5	-	-	3.9	3.9	4.0	4.0	4.0	4.0	4.0	3.9	3.9	3.9	1.9
Total Technical Services	72.5	-	-	3.4	7.2	7.3	7.2	7.2	7.2	7.3	7.2	7.2	7.2	3.9
Transfer to UG	13.3	-	-	(0.2)	0.9	0.9	1.0	1.4	1.5	1.5	1.5	1.5	1.6	1.6
Total Mining Cost	804.1	-	-	36.2	62.7	83.0	87.4	82.7	88.1	83.5	80.2	79.1	72.2	49.1
Unit Cost ($/t Mined)	55.45	-	-	126.47	66.24	61.67	54.82	52.72	56.39	54.04	53.07	53.39	50.58	43.65

21.4.2 Processing Costs

The processing flowsheet described in Section 17 is a conventional circuit to treat free-milling gold material types. Processing costs will vary with the type of weathered materials fed to the plant (saprolite, transition, hard rock) and their respective proportions. The availability of saprolite and transition material will enable a yearly throughput of 7 Mtpa for the first five (5) years of operations; mill throughput will be limited to 6 Mtpa afterwards when only fresh rock material is fed to the plant. Table 21.21 provides a summary of operating costs per type of material. The average processing cost including materials handling but excluding power cost is USD 8.87 per tonne processed.

Table 21.21: Processing OPEX

Processing OPEX (USD/t milled)			
	Saprolite	**Transition**	**Fresh Rock**
Grinding Media & Lines	0.88	1.34	2.01
Reagents	4.11	3.91	3.56
Labour	1.161	1.61	1.61
Ore Handling	0.3	0.3	0.06
Maintenance & Supplies	1.2	1.2	1.4
Contingency	0.35	0.36	0.37
	8.45	**8.72**	**9.01**

Consumption of grinding media and reagents (Table 21.22) are based on historical data from operations in the Guianas and West Africa and Feasibility Studies of similar projects. Prices were compiled from 2024 quotes adjusted for transportation from suppliers to site (Table 21.23). For some of the minor reagents, historical prices were adjusted accordingly.

Labour costs were based on historical personnel compliments at other sites and total remuneration (salary, benefits, incentives) as established by labor surveys conducted in Guyana and recent actual costs for North American expatriates.

Handling costs for saprolite and transition material at the apron feeders with backhoes are estimated at USD 0.30 per tonne of saprolite and transition material while rehandling of fresh rock material (10% of crusher feed) results in expenses of USD 0.06 per tonne of crusher feed. It is assumed that transition material will be handled directly through the feeders, but some tonnage could be mixed with fresh rock material and handled through the primary crusher.

A contingency was added for supplies at the Assay and Metallurgical laboratories and chemicals to treat water effluents for contaminants unknown at this time.

Logistics costs are included in the costs of consumables, supplies and maintenance parts. Power costs are discussed in Sub-section 21.4.3.

Table 21.22: Grinding Media and Reagent Consumption

Grinding Media and Reagent Consumption				
	Unit	**SAP**	**Transition**	**Fresh**
Grinding Media & Liners				
SAG Grinding Media	kg/t	0.10	0.23	0.45
BM Grinding Media	kg/t	0.40	0.50	0.60
Liners (% of Grinding Media)	%	20.00	24.00	28.00
Reagents				
Cyanide	kg/t	0.40	0.40	0.40
Lime	kg/t	2.25	2.00	1.35
Activated Carbon	kg/t	0.04	0.04	0.04
Flocculant	kg/t	0.05	0.03	0.01
Caustic	kg/t	0.03	0.03	0.03
Hydrochloric Acid	kg/t	0.06	0.06	0.06
Antiscalant	kg/t	0.03	0.02	0.01
Sodium Metabisulphite	kg/t	1.00	1.00	1.00
Copper Sulphate	kg/t	0.10	0.10	0.10
Ferric Sulphate	kg/t	0.15	0.15	0.15
Soda Ash	kg/t	0.04	0.05	0.06

Table 21.23: Consumables Operating Costs

Consumables Operating Costs (USD/t milled)				
	Prices (USD/t)	**SAP**	**Transition**	**Fresh**
Grinding Media & Liners				
SAG Grinding Media	1,587	0.16	0.37	0.71
BM Grinding Media	1,426	0.57	0.71	0.86
Liners (% of Grinding Media)		0.15	0.26	0.44
Sub-Total		0.88	1.34	2.01
Reagents - Process				
Cyanide	3,060	1.22	1.22	1.22
Lime (90% CAO)	400	0.90	0.80	0.54
Activated Carbon	3,800	0.15	0.15	0.15
Flocculant	4,580	0.23	0.14	0.05
Caustic	1,150	0.03	0.03	0.03
Hydrochloric Acid	1,360	0.08	0.08	0.08
Antiscalant	4,060	0.12	0.07	0.02
Soda Ash	4,900	0.20	0.25	0.29
Sub-Total		2.94	2.74	2.39
Reagents - CN Destruction & Other				
Sodium Metabisulphite	680	0.68	0.68	0.68
Copper Sulphate	4,130	0.41	0.41	0.41
Ferric Sulphate	520	0.08	0.08	0.08
Sub-Total		1.17	1.17	1.17
Total		**4.98**	**5.25**	**5.57**

21.4.3 Power Costs

Energy cost was estimated based on a Brent crude oil price of USD 80/barrel and a base HFO price of US cent 41.43/litre plus additional costs of US cent 12.57/litre for transportation, premium and profit margin. This results in a full energy cost of USD 0.1361/kW-h.

Based on historical data at other sites, the following power consumptions are estimated for the various material types. This power consumption is the average power required per tonne processed, including not

only the process plant but the entire complete mine site operation. The power consumption variation is entirely due to the material sources (OP and UG) and material types (saprolite, transition, fresh rock).

	kW-h/t milled
Saprolite	20
Transition	30
Fresh Rock – OP	40
Fresh Rock – UG	66.22

The power consumption for the underground combines the fresh rock at the mill plus the power for the underground mined rock (Material + Waste).

The average total site power operating cost is $5.93/tonne processed. Table 21.24 presents the power operating costs per material type and material source.

Table 21.24: Power Operating Costs

	USD/t milled
Saprolite	2.71
Transition	4.08
Fresh Rock – OP	5.75
Fresh Rock – UG	10.01

21.4.4 General and Administration Costs

A full description of the General Services is provided in Section 21.1.10.

Based on experience and data at other similar projects, the operating cost for the General Services listed in Table 21.10 are estimated as follows:

- $1,100,000/month for Years 1 to 3, inclusively when open pit only.
- $1,500,000/month for Years 4 to 7, inclusively when both open pit and underground mines are in full production.
- $900,000/month for Years 8 to 13 because of reduced open pit and processing production.

Expenditures for site services (camp and site maintenance) and transportation services will vary according to the number of workers on the Project. The site services costs are estimated at $35 per day and the transportation costs at $15 per day.

Logistics expenditures will be at maximum during construction and pre-production. During the production period, the transportation costs for consumables, supplies and parts are included in their respective price estimates. The total workforce is estimated to average 1,130 with a peak at 1,533 in 2033.

22 ECONOMIC ANALYSIS

22.1 Overview

The PEA is preliminary in nature and includes Inferred Mineral Resources, which are considered too geologically speculative to be categorized as Mineral Reserves with economic considerations. Therefore, there is no certainty that the PEA will be realized.

The economic and financial evaluation presented in this Technical Report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The commodity prices used in the evaluation were determined in Section 19. The financial model provides results in terms of NPV, payback period, and IRR for the Oko West Project. The economic analysis is conducted in real terms, without considering inflation factors, using Q2 2024 United-States dollars. The analysis does not take into account project financing.

The economic model estimates cash flows on an annual basis for the life of the Oko West Project, based on the level of engineering and design appropriate for a PEA.

Cash flow projections for the life of the Project are based on Sales revenue, OPEX, CAPEX, and other cost estimates. CAPEX is estimated in four (4) categories: initial capital, sustaining capital, closure and reclamation cost and working capital. OPEX estimates include labour, reagents, maintenance, supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current mine and processing plans.

The economic results are calculated from the start of the initial capital expenditures, treating all prior costs as sunk costs.

22.2 Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as defined under the Canadian securities law. These results are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

The forward-looking information includes, but is not limited to, the following:

- Assumed prices for gold.
- Cost inflation.
- The proposed mine production plan.

- Assumptions regarding mining dilution and mining recovery.
- The recovery rates of gold in the processing plant.
- Proposed sustaining and operating costs.
- Labour and materials availability.
- Labour and materials costs being approximately consistent with the assumptions in the report.
- Assumptions regarding closure costs.
- Assumptions regarding environmental, social, and licensing risks.
- Changes to tax rates.
- Unexpected variations in the amount of mineralized material and material grade.
- Geotechnical or hydrogeological considerations during mining that differ from the assumptions.
- Ability to maintain social licence to operate.
- Unrecognized environmental risks.
- Unforeseen reclamation expenses.
- Failure of plant, equipment, and processes to operate as anticipated.
- The absence of significant disruptions affecting the development and operation of the Oko West Project.
- The availability of certain consumables and services, and the prices for electricity and other key supplies being approximately consistent with the assumptions in the Technical Report.

22.3 Key Assumption

22.3.1 Gold Price

The determination of gold prices is described in Section 19. The long-term gold price assumption used in the PEA is USD 1,950/oz Au, in line with analyst consensus commodity price forecasts.

22.3.2 Fuel Price and Energy

The reference price for diesel fuel used to estimate operating costs is $0.85/L. The price of diesel fuel is for off-road or off-highway use by mining equipment that will not be operated on public roadways. The price of electricity is estimated based on the consumption of the mine and the mill at the prevailing rates.

The electricity cost for the Oko West Project in Guyana is based on the total cost of HFO delivered to the project. The efficiency on HFO used is of 0.215 L/kWh. The cost for operation and maintenance is estimated at 2 USC/kWh. The current forecast price of Brent is 80 USD/bbl.

Table 22.1: Total Power Cost in USD/kWh

Parameters		
Brent	80	$/bbl
Brent	0.5032	$/L
Oko West HFO	0.4143	$/L
Additional Costs	0.1246	$/L
Efficiency on HFO	0.215	L/kWh
Fuel Cost	0.1159	$/kWh
O&M	0.02	$/kWh
Total Energy Cost	**0.1359**	**$/kWh**

22.3.3 Other Assumptions

The other key assumptions used in economic analysis are as follows:

- Discount rate 5%.
- All cost estimates are in constant Q2 2024 United States dollars with no inflation or escalation factors taken into account.

22.4 Metal Production and Revenues

Payable gold over the Project life is 4,498 koz based on an average metallurgical recovery of 92.8% and a payability factor of 99.95%. Payable gold during pre-production is 16.13 koz, generating estimated revenue of USD 28.8M (net of transportation, refining and royalty costs) which offsets pre-production CAPEX. A total of 4,481.83 koz of gold will be payable during operations and will generate gross revenue of USD 8,141M (net of transportation, refining and royalty costs). Table 22.2 shows the Milling Production Schedule Summary, while Figure 22.1 presents the yearly gold production during LOM. Gold production over LOM shows the gold production over the life of the mine and Figure 22.2 shows the split between open-pit and underground production.

Table 22.2: Milling Production Schedule Summary

Process Plant Schedule[1]		Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total Ore																	
Total Ore Milled	000 t	75,203	-	441	6,408	7,000	7,000	7,000	6,000	6,000	6,000	5,822	5,000	5,000	5,000	5,000	3,531
Head Grade	g/t	2.00	-	0.96	1.63	1.55	1.62	1.54	1.85	2.02	1.91	2.28	2.08	2.23	2.62	2.58	3.23
Recovered Gold	000 oz	4,500	-	16	317	324	339	325	330	361	340	385	319	331	390	384	340
Saprolite Ore																	
Saprolite Milled	000 t	7,660	-	30	2,195	1,378	1,169	2,483	-	395	0	0	-	9	-	-	-
Head Grade	g/t	1.40	-	1.53	1.53	1.40	1.33	1.33	-	1.33	1.33	1.33	-	1.33	-	-	-
Recovered Gold	000 oz	331	-	1	104	59	48	102	-	16	0	0	-	0	-	-	-
Transition Ore																	
Transition Milled	000 t	3,411	-	271	1,989	596	553	-	-	3	-	-	-	-	-	-	-
Head Grade	g/t	1.47	-	1.38	1.78	1.02	0.89	-	-	0.49	-	-	-	-	-	-	-
Recovered Gold	000 oz	153	-	11	108	19	15	-	-	0	-	-	-	-	-	-	-
Fresh Rock Ore Open Pit																	
Fresh Rock Milled	000 t	49,631	-	140	2,183	4,959	4,991	3,571	4,655	4,008	4,432	4,260	3,455	3,480	3,518	3,572	2,406
Head Grade	g/t	1.79	-	0.03	1.59	1.64	1.71	1.47	1.46	1.52	1.46	1.94	1.63	1.91	2.31	2.23	3.03
Recovered Gold	000 oz	2,642	-	3	103	242	254	156	203	182	193	236	177	197	241	237	217
Underground																	
Fresh Rock Milled	000 t	14,501	-	-	40	67	286	946	1,345	1,595	1,568	1,562	1,545	1,511	1,482	1,428	1,125
Head Grade	g/t	3.19	-	-	1.97	2.09	2.63	2.39	3.18	3.43	3.16	3.19	3.08	2.96	3.37	3.45	3.66
Recovered Gold	000 oz	1,374	-	-	2	4	22	67	127	163	147	148	141	133	149	146	123

**Notes: 1) Includes pre-production tonnage.*

Figure 22.1: Gold Production Over LOM

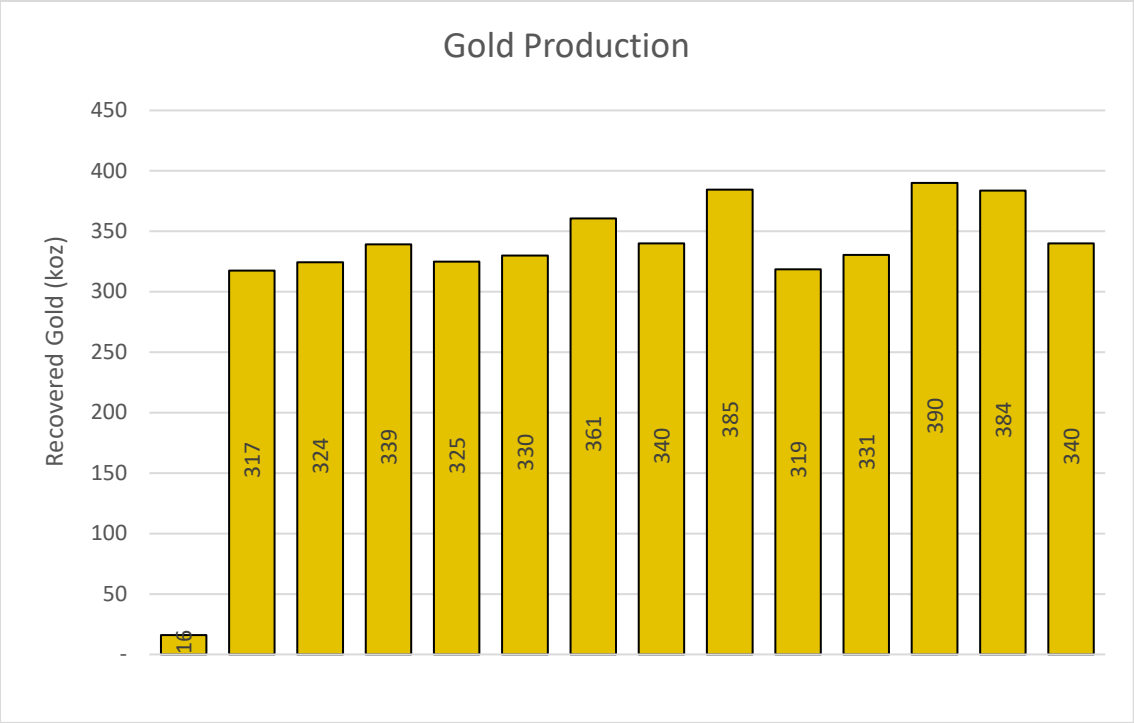
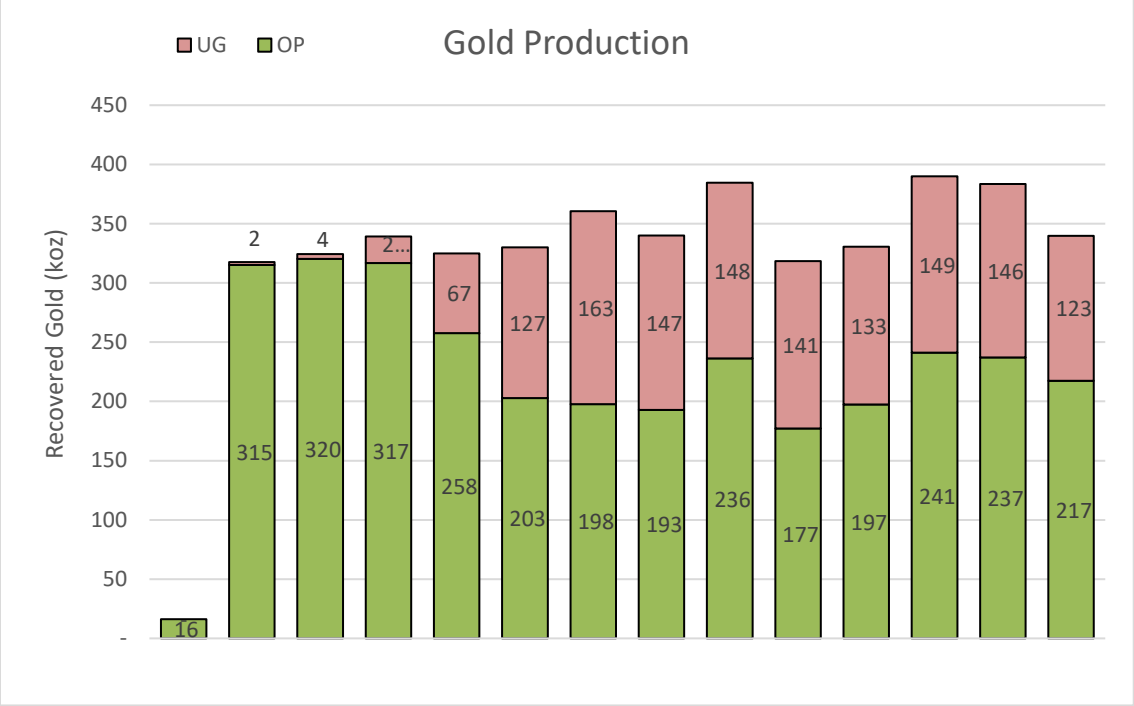


Figure 22.2: Gold Production Over LOM by Mining Method



22.5 Royalties

As agreed in the Mineral Agreement between the Reunion Gold and the Government of Guyana, a royalty is applicable to the gold production from the property. The applicable royalties for gold produced from open pit mining operation within the property shall be 8% of the net smelter return. The applicable royalty for gold produced from underground mining operation within the property shall be 3% of the net smelter return. A payment of $5/oz is payable to the previous owner of the property.

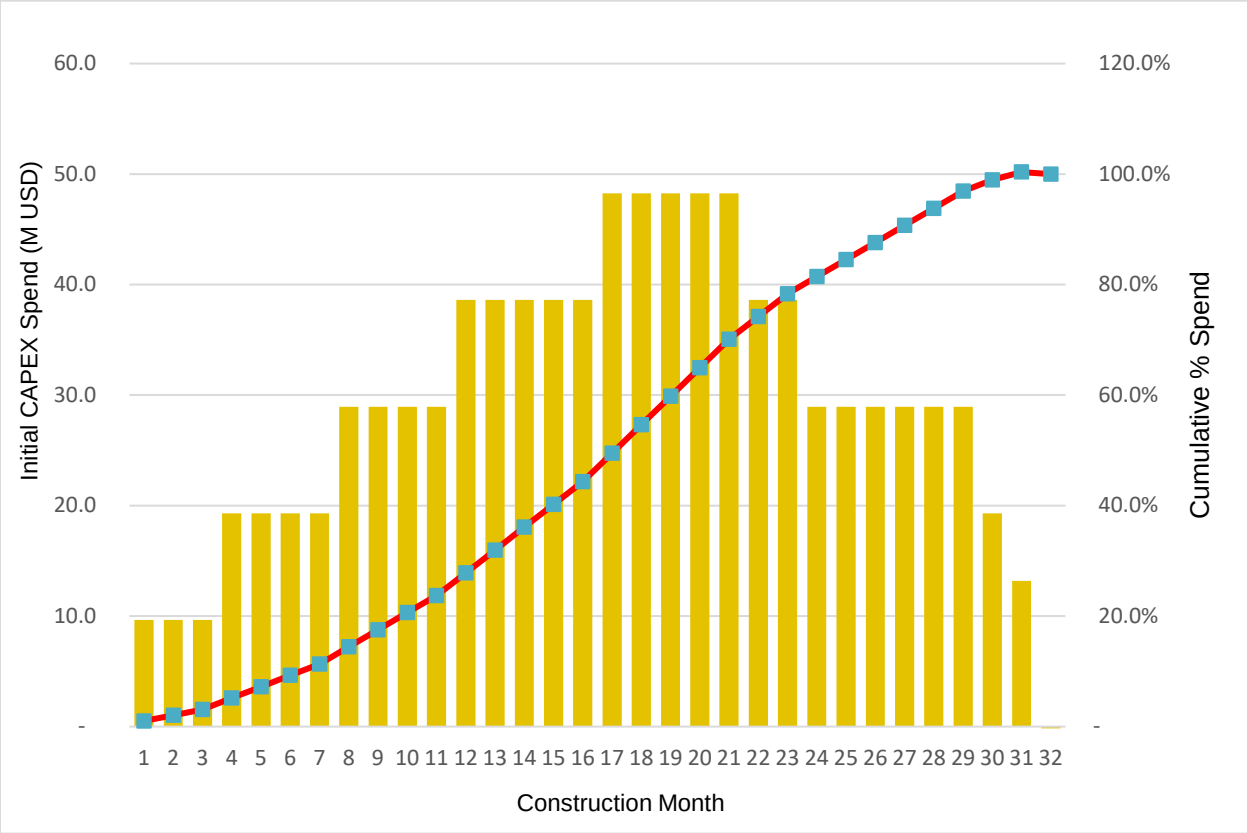
22.6 Capital Expenditures

The capital expenditures include initial capital expenditures as well as the sustaining capital expenditures to be spent after commencement of commercial operations.

22.7 Initial Capital

The initial CAPEX for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated to be USD 936.3M. The total initial Project capital includes a contingency of USD 100.3M, which is 12% of the initial capital expenditures before contingency. The monthly CAPEX estimate is presented in Figure 22.3.

Figure 22.3: Initial CAPEX by Month



Source: GMS, 2024

22.7.1 Sustaining Capital

Sustaining capital is required in specific areas. For the open pit mine, it is necessary to purchase new equipment and major components for the equipment. For the underground mine, sustaining capital includes all development work and the required equipment. Sustaining capital is required during operations for additional equipment purchases for the underground mine. Additional work is required for raising the main embankment of the tailings storage facility ("TSF"). The continued raising of the TSF will be completed by the mine operations team with fill material from the open pit mine. An increase in power requirements is also planned and generators will be added.

The sustaining capital is estimated at USD 537.5M and is detailed in Table 22.3.

22.8 Working Capital

Working capital is required to finance supplies in inventory. It is planned to maintain a 60-day inventory of various consumables. This represents a maximum inventory value of $47.5M. Additionally, it is planned to

pay the various operational suppliers within 30 days, and payables are included in the working capital calculation.

22.8.1 Reclamation & Closure Cost

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at USD 36.7M as discussed in Section 21.

22.8.2 Salvage Value

No residual value was estimated at this stage of the study.

22.9 Operating Cost Summary

Operating costs are presented by year in Table 22.3 and Table 22.4. The operating costs include mining, processing, power, general services and administration (G&A), gold transportation, and refining and royalties. The average LOM operating cost is USD 853/oz of gold or USD 51.15/t milled. The average LOM all-in sustaining cost (AISC) is USD 986/oz of gold or USD 59.13/t milled.

Table 22.3: Operating Cost Summary (USD)

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Production Highlights (excl. Pre-Prod.)																		
Tonnage Milled	kt	74,762	6,408	7,000	7,000	7,000	6,000	6,000	6,000	5,822	5,000	5,000	5,000	5,000	3,531	-	-	-
Tonnage Mined MM OP	kt	57,174	7,509	6,448	5,858	3,571	4,655	4,011	4,432	4,438	3,278	3,489	3,518	3,572	2,397	-	-	-
Tonnage Mined MM UG	kt	14,501	40	67	286	946	1,345	1,595	1,568	1,562	1,545	1,511	1,482	1,428	1,125	-	-	-
Tonnage Mined Waste OP	kt	336,266	34,614	35,148	35,042	36,308	39,345	36,613	24,637	28,348	29,201	20,971	8,931	5,793	1,315	-	-	-
Recovered Gold	koz	4,484	317	324	339	325	330	361	340	385	319	331	390	384	340	-	-	-
Operating Costs (excl. Pre-Prod.)																		
Open Pit Mining	MUSD	981	76	80	89	93	95	94	80	89	90	78	53	43	22	-	-	-
Underground Mining	MUSD	804	-	-	36	63	83	87	83	88	83	80	79	72	49	-	-	-
Material Rehandling	MUSD	11	1	1	1	2	1	1	1	1	0	1	1	1	0	-	-	-
Processing	MUSD	676	55	61	61	60	54	54	54	53	47	47	47	47	36	-	-	-
Power	MUSD	444	27	37	38	36	38	37	38	37	33	32	33	32	24	-	-	-
General & Administration	MUSD	310	24	24	24	24	24	24	24	24	24	24	24	24	24	-	-	-
Refining	MUSD	36	3	3	3	3	3	3	3	3	3	3	3	3	3	-	-	-
Total Site Cost	MUSD	3,261	186	205	252	281	297	300	283	295	280	264	239	222	159	-	-	-
Royalty Cost on Revenues	MUSD	563	49	50	51	44	39	40	39	45	36	38	46	45	41	-	-	-
Total OPEX Cost	MUSD	3,824	235	255	302	325	336	340	321	340	316	303	285	267	199	-	-	-
Sustaining Capital	MUSD	537	120	83	57	57	45	38	30	36	22	21	18	8	3	-	-	-
Closure Cost	MUSD	37	-	-	-	-	-	-	-	-	-	-	-	-	-	12	12	12
Land Payments	MUSD	22	22	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	MUSD	4,421	378	339	359	382	382	378	351	376	338	324	302	275	203	12	12	12

Table 22.4: Operating Cost Summary per Tonne

Operating Costs per Tonnes Mined		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Open Pit Mining Cost	t/MM + waste mined	2.49	1.81	1.92	2.16	2.33	2.16	2.31	2.76	2.71	2.79	3.19	4.24	4.56	5.97
Underground Mining Cost	t/MM mined	55.45	-	-	126.47	66.24	61.67	54.82	52.72	56.39	54.04	53.07	53.39	50.58	43.65
Operating Costs per Tonnes Milled															
Open Pit Mining Cost	t/milled	13.13	11.87	11.43	12.64	13.27	15.81	15.62	13.39	15.27	18.09	15.61	10.55	8.54	6.28
Underground Mining Cost	t/milled	10.76	-	-	5.17	8.95	13.83	14.57	13.78	15.13	16.69	16.04	15.83	14.45	13.90
Material Rehandling	t/milled	0.15	0.14	0.19	0.20	0.31	0.11	0.14	0.11	0.11	0.10	0.11	0.11	0.11	0.13
Processing Cost	t/milled	9.04	8.64	8.67	8.68	8.60	9.04	9.00	9.04	9.09	9.36	9.36	9.36	9.36	10.17
Power Cost	t/milled	5.93	4.21	5.28	5.48	5.19	6.36	6.18	6.35	6.38	6.52	6.50	6.51	6.49	6.89
General & Administration Cost	t/milled	4.14	3.72	3.40	3.40	3.40	3.97	3.97	3.97	4.09	4.77	4.76	4.76	4.76	6.75
Refining Cost	t/milled	0.48	0.40	0.37	0.39	0.37	0.44	0.48	0.45	0.53	0.51	0.53	0.62	0.61	0.77
Total Site Cost	**t/milled**	**43.62**	**28.98**	**29.34**	**35.97**	**40.09**	**49.56**	**49.96**	**47.09**	**50.59**	**56.05**	**52.90**	**47.74**	**44.32**	**44.90**
Royalty Cost	t/milled	7.53	7.66	7.14	7.22	6.28	6.48	6.70	6.42	7.78	7.15	7.68	9.22	9.07	11.58
Total OPEX Cost	**t/milled**	**51.15**	**36.64**	**36.48**	**43.18**	**46.37**	**56.05**	**56.66**	**53.51**	**58.37**	**63.20**	**60.58**	**56.96**	**53.39**	**56.48**
Sustaining Capital	t/milled	7.19	18.78	11.89	8.12	8.14	7.58	6.26	4.99	6.14	4.34	4.17	3.50	1.64	0.88
Closure Cost	t/milled	0.49	-	-	-	-	-	-	-	-	-	-	-	-	-
Land Payments	t/milled	0.30	3.51	-	-	-	-	-	-	-	-	-	-	-	-
All-In Sustaining Costs (AISC)	t/milled	**59.13**	**58.93**	**48.37**	**51.30**	**54.51**	**63.63**	**62.93**	**58.49**	**64.51**	**67.54**	**64.75**	**60.47**	**55.03**	**57.36**

22.10 Taxation

The applicable taxes are included in the economic analysis. The corporate tax rate is 25% and generally allows for straight line depreciation over five (5) years. An initial tax asset pool of USD 161M was considered in the estimate of taxes to be paid.

22.11 Economic Results

The main economic metrics used to evaluate the Project consist of the net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. A 5% discount rate is commonly used as the base case for gold projects.

A summary of the Project economic results is presented in Table 22.5 and the annual Project cash flows are presented in Table 22.6. The total after-tax undiscounted cash flow over the Project life is USD 2,583M, and NPV 5% is USD 1,837M before tax and USD 1,367M after tax. The after-tax Project cash flow results in a 3.8-year payback period from the commencement of commercial operations with an IRR of 23.7% before tax and 20.8% after tax.

Table 22.5: Project Economic Results Summary

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	USD/oz	1,950
Mine Life (operation years)	Mt	12.7
OP Mill Feed Tonnage	Mt	61
UG Mill Feed Tonnage	Mt	15
Total Mineralized Material Mined	***Mt***	***75***
Total Waste Mined (OP and UG)		***367***
Total Tonnage Mined (OP and UG)		***443***
Strip Ratio	***waste: mineralized material***	***6.0***
Average Milling Throughput	***Mtpa***	***6***
Average Milling Throughput	***tpd***	***16,110***
Gold Head Grade	g/t	2.00
OP Head Grade	g/t	1.72
UG Head Grade	g/t	3.19
Contained Gold	koz	4,848
Average Gold Recovery (%)	%	92.8%
Total Gold Production	koz	4,500
Average Annual Gold Production	koz	353
Operating Costs (LOM Average)		
Open Pit Mining Cost	USD/t mined	$2.49
Underground Mining Cost	USD/t milled	$55.45
Processing Cost	USD/t milled	$14.97
G&A Cost	USD/t milled	$4.14
Total Site Costs	USD/t milled	$51.15
Total Site Costs	USD/oz	$728
Government Royalties	USD/oz	$126
Total Operating Cost	USD/oz	$853
AISC	**USD/oz**	**$986**
Capital Costs		
Capital Costs	*USD MM*	*$836*
Contingency	*USD MM*	*$100*
Total Capital Cost	USD MM	$936
Initial UG Capital Costs (Sustaining Capital)	USD MM	$124
OP and UG Sustaining Capital	USD MM	$413
Life of Mine Sustaining Capital	USD MM	$537
Closure Costs	USD MM	$37
Total Capital Costs	USD MM	$1,510
Financial Evaluation		
After-Tax NPV 5%	USD MM	$1,367
After-Tax IRR	%	21%
Payback	years	3.8

Table 22.6: Project Cash Flow Summary

Cash Flow (M USD)	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Gold Price (USD/oz)		1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950	1,950
Rec. Gold (koz)	4,500.2	-	-	16.1	317.5	324.4	339.3	324.9	330.0	360.7	340.1	384.6	318.6	330.5	390.0	383.6	339.9	-	-	-
Payable Gold - Prod	4,498.0	-	-	16.1	317.3	324.3	339.2	324.8	329.8	360.5	339.9	384.4	318.4	330.4	389.8	383.4	339.7	-	-	-
Gross Revenue	8,771.0	-	-	31.5	618.8	632.3	661.3	633.3	643.1	703.0	662.9	749.5	620.9	644.2	760.1	747.6	662.4	-	-	-
Mining OP Cost	1,040.5	0.1	25.0	33.8	76.1	80.0	88.5	92.9	94.8	93.7	80.4	88.9	90.5	78.0	52.8	42.7	22.2	-	-	-
Mining UG Cost	830.1	-	-	-	7.7	18.3	36.2	62.7	83.0	87.4	82.7	88.1	83.5	80.2	79.1	72.2	49.1	-	-	-
Material Rehandling Cost	11.1	-	-	0.1	0.9	1.3	1.4	2.1	0.7	0.8	0.6	0.6	0.5	0.5	0.5	0.6	0.5	-	-	-
Processing Cost	682.8	-	-	7.2	55.4	60.7	60.8	60.2	54.2	54.0	54.2	52.9	46.8	46.8	46.8	46.8	35.9	-	-	-
Power Cost	445.7	-	-	2.0	27.0	37.0	38.4	36.4	38.2	37.1	38.1	37.1	32.6	32.5	32.5	32.5	24.3	-	-	-
General & Administration Cost	309.7	-	-	-	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	-	-	-
Refining Cost	36.0	-	-	0.1	2.5	2.6	2.7	2.6	2.6	2.9	2.7	3.1	2.5	2.6	3.1	3.1	2.7	-	-	-
Royalty Cost on Revenues Cost	565.4	-	-	2.5	49.1	50.0	50.5	43.9	38.9	40.2	38.5	45.3	35.7	38.4	46.1	45.4	40.9	-	-	-
Transfer to Capex	(96.9)	(0.1)	(25.0)	(45.7)	(7.7)	(18.3)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Land Payment	22.5	-	-	-	22.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Operating Costs	3,846.8	-	-	(0.0)	257.3	255.4	302.3	324.6	336.3	340.0	321.1	339.9	316.0	302.9	284.8	267.0	199.5	-	-	-
EBITDA	4,924.2	-	-	31.5	361.5	377.0	359.1	308.7	306.9	363.0	341.8	409.6	304.9	341.3	475.2	480.7	463.0	-	-	-
Initial CAPEX	(965.0)	(135.1)	(472.8)	(357.0)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	(537.5)	-	-	-	(120.3)	(83.2)	(56.9)	(57.0)	(45.5)	(37.6)	(29.9)	(35.7)	(21.7)	(20.9)	(17.5)	(8.2)	(3.1)	-	-	-
Closure Costs	(36.6)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(12.2)	(12.2)	(12.2)
Salvage Value	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital costs	(1,539.1)	(135.1)	(472.8)	(357.0)	(120.3)	(83.2)	(56.9)	(57.0)	(45.5)	(37.6)	(29.9)	(35.7)	(21.7)	(20.9)	(17.5)	(8.2)	(3.1)	(12.2)	(12.2)	(12.2)
Working Capital	-	(0.4)	(5.9)	(18.7)	(19.7)	(3.6)	(1.3)	(8.7)	(0.5)	(0.2)	1.2	(0.8)	0.3	0.9	1.7	1.0	17.4	37.3	-	-
Pre-Tax Cash Flow	3,385.1	(135.5)	(478.7)	(344.3)	221.5	290.2	300.9	243.0	260.9	325.2	313.1	373.1	283.5	321.4	459.4	473.5	477.2	25.1	(12.2)	(12.2)
Taxes	(801.6)	-	-	-	-	(1.9)	(32.6)	(24.0)	(43.1)	(70.8)	(69.8)	(89.2)	(64.4)	(75.2)	(109.6)	(112.0)	(108.9)	-	-	-
After-Tax Cash Flow	2,583.5	(135.5)	(478.7)	(344.3)	221.5	288.3	268.3	219.0	217.8	254.3	243.3	283.9	219.1	246.2	349.8	361.5	368.4	25.1	(12.2)	(12.2)

22.12 Sensitivity Analysis

The Project financial performance is most sensitive to the gold price and much less to the operating costs and capital expenditures. The results of the sensitivity analysis of the project in terms of NPV, IRR and Payback to the gold price are summarized in Table 22.7. Table 22.8 and Table 22.9 summarize the variation on NPV, IRR and payback to the OPEX cost and CAPEX cost. Figure 22.4 to Figure 22.7 show the sensitivity on the After-Tax Total Cashflow, NPV 5%, IRR and Payback.

Table 22.7: Gold Price Sensitivity

	Pre-Tax				After-Tax			
Gold Price (USD/oz)	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (y)	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (y)
1,300	650	57	5.7%	10.4	650	57.3	5.7%	10.4
1,400	1,071	331	9.0%	8.3	905	245.9	8.1%	8.3
1,500	1,492	605	12.0%	6.9	1,158	427.1	10.5%	6.9
1,600	1,912	879	14.8%	5.9	1,475	638.5	13.0%	5.9
1,700	2,333	1,153	17.5%	5.1	1,792	848.5	15.4%	5.2
1,800	2,754	1,426	20.0%	4.4	2,108	1,057.0	17.6%	4.5
1,900	3,175	1,700	22.5%	3.8	2,425	1,264.0	19.8%	4.0
1,950	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
2,000	3,596	1,974	24.8%	3.4	2,742	1,470.6	21.8%	3.6
2,100	4,016	2,248	27.1%	3.0	3,059	1,677.1	23.7%	3.3
2,200	4,437	2,522	29.3%	2.0	3,375	1,883.5	25.6%	3.0
2,300	4,858	2,796	31.4%	2.0	3,692	2,090.0	27.4%	2.0
2,400	5,279	3,070	33.5%	2.0	4,009	2,296.4	29.2%	2.0
2,500	5,700	3,343	35.5%	2.0	4,326	2,502.8	30.9%	2.0
2,600	6,120	3,617	37.4%	2.0	4,642	2,708.9	32.5%	2.0

Table 22.8: OPEX Sensitivity

	Pre-Tax				After-Tax			
OPEX	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**
80%	4,030	2,260	27.1%	3.1	3,067	1,685.0	23.7%	3.3
90%	3,708	2,049	25.4%	3.3	2,825	1,526.2	22.3%	3.6
Base Case	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
110%	3,063	1,626	21.8%	4.0	2,342	1,208.3	19.2%	4.2
120%	2,740	1,414	19.9%	4.4	2,100	1,049.1	17.6%	4.6

Table 22.9: Initial CAPEX Sensitivity

	Pre-Tax				After-Tax			
CAPEX	**NPV 0% (M USD)**	**NPV 5% (M USD)**	**IRR**	**Payback Period (y)**	**NPV 0% (M USD)**	**NPV5% (M USD)**	**IRR**	**Payback Period (y)**
80%	3,578	2,019	28.8%	2.0	2,728	1,511.0	25.2%	3.1
90%	3,482	1,928	26.0%	3.2	2,656	1,439.2	22.8%	3.5
Base Case	3,385	1,837	23.7%	3.6	2,584	1,367.4	20.8%	3.8
110%	3,289	1,746	21.7%	4.0	2,511	1,295.2	19.0%	4.2
120%	3,192	1,655	19.9%	4.4	2,439	1,222.8	17.5%	4.5

Figure 22.4: After-Tax Total Cash Flow Sensitivity

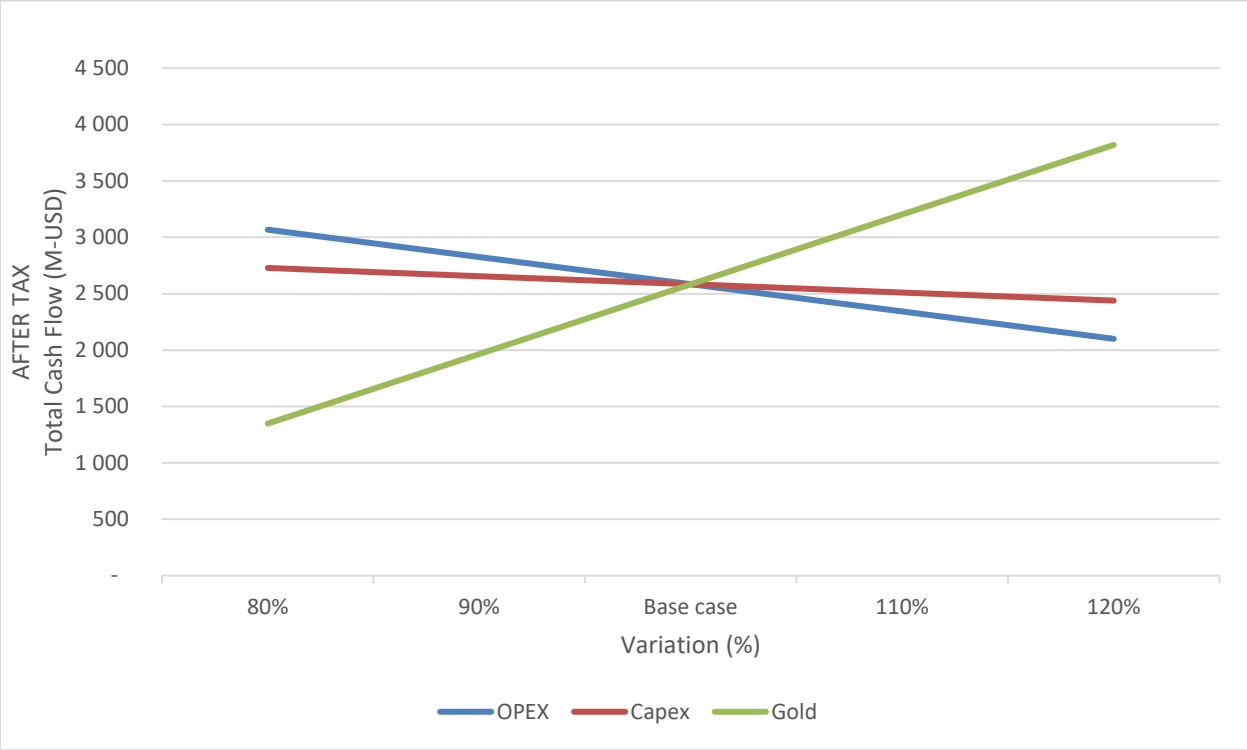


Figure 22.5: After-Tax NPV (5%) Sensitivity

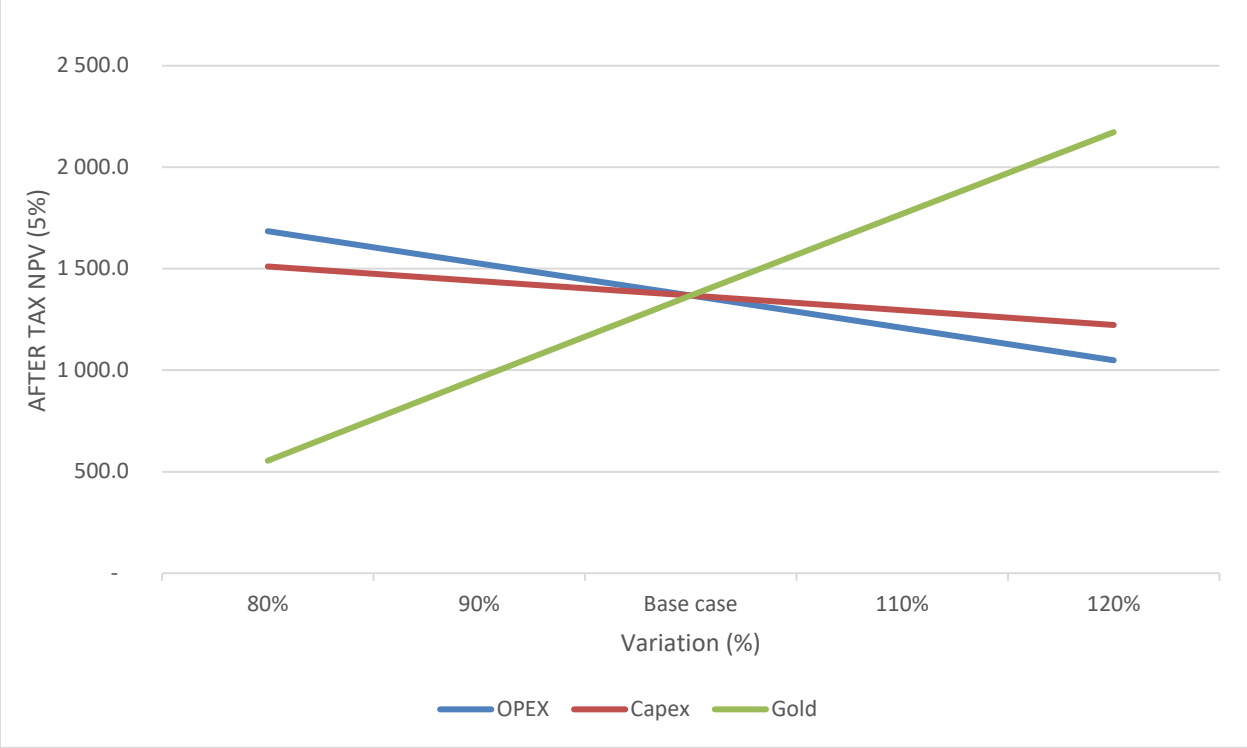


Figure 22.6: After-Tax Internal Rate of Return Sensitivity

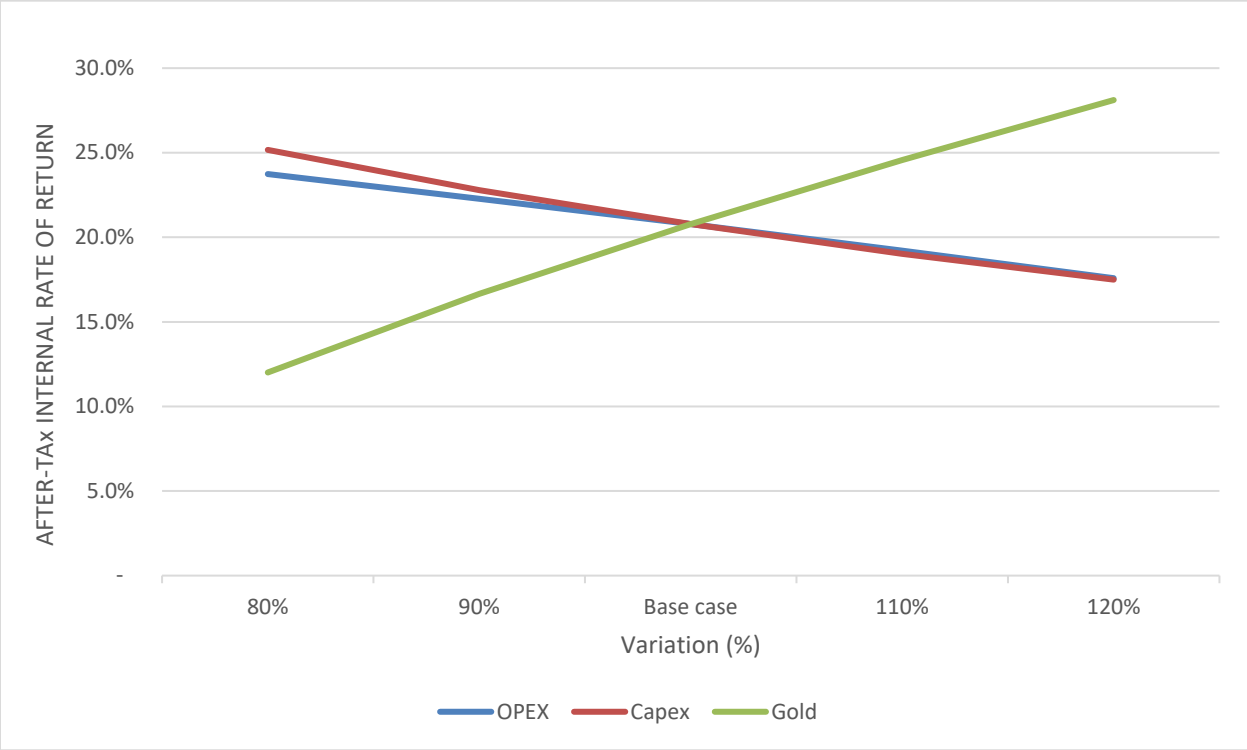
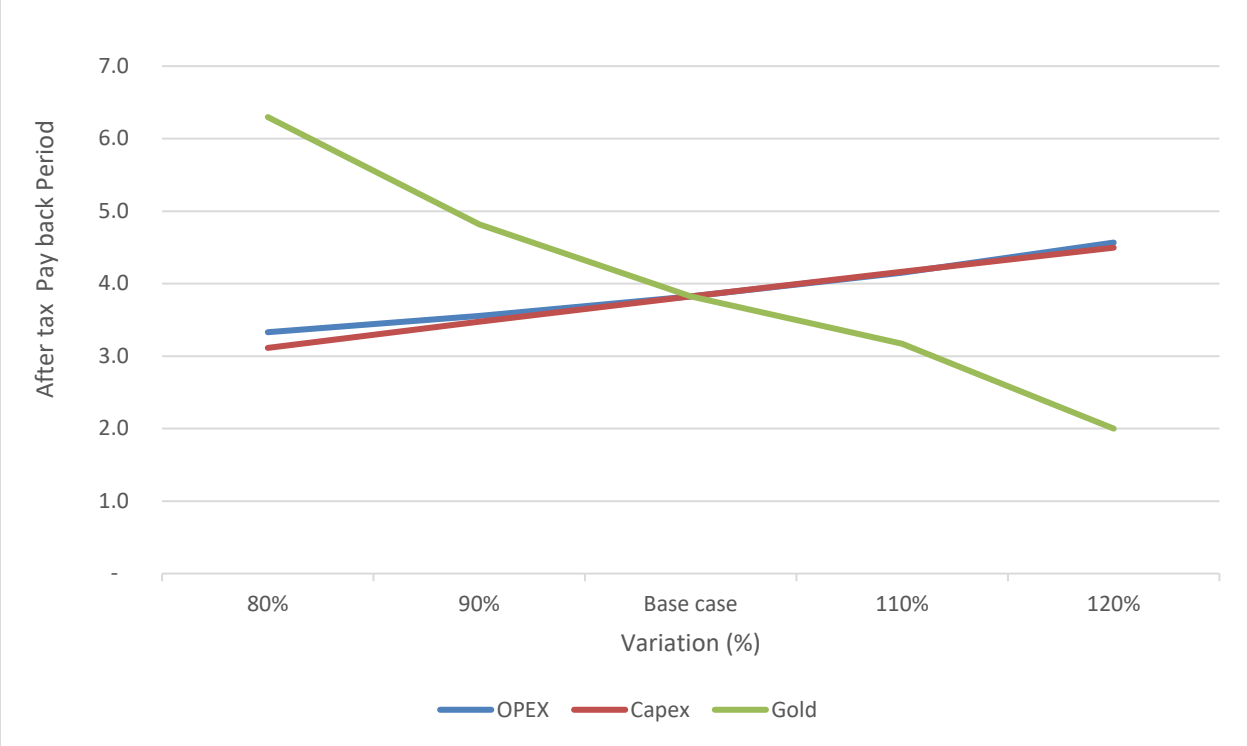


Figure 22.7: After-Tax Payback Period Sensitivity



23 ADJACENT PROPERTIES

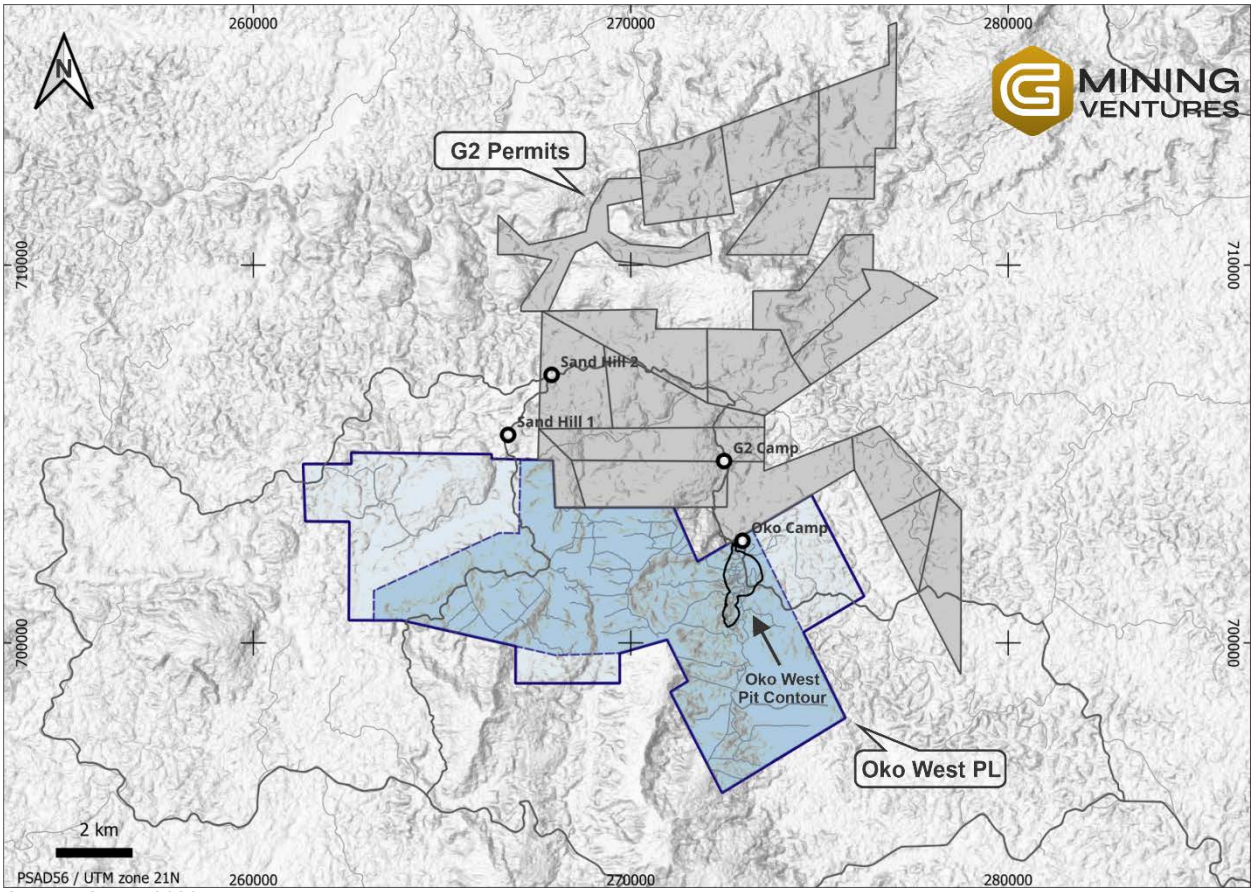
According to the GGMC, the Oko West Prospecting Licence is surrounded by 13 medium-scale mining and prospecting permits held by various Guyanese title holders and one group of medium-scale mining and prospecting permits controlled by G2 Goldfields Inc. (Figure 23.1).

The Company is unaware of any ongoing exploration work on the 13 medium-scale permits held by local entrepreneurs. Significant artisanal mining activity is present on the permits near the community of Sand Hill and within the permits controlled by G2 Goldfields. There is no record of historical artisanal gold production from these permits.

The section below describes the exploration work being done in the permits controlled by G2 Goldfields. A Technical Report in relation to the G2 Goldfields property was filed on SEDAR on June 1, 2022, and is available at www.sedar.com.

The QP has been unable to verify the information on the adjacent property and the information provided herein is not necessarily indicative of the mineralization on the Oko West Project.

Figure 23.1: Oko West Prospecting Licence (PL) and Adjacent Mineral Permits



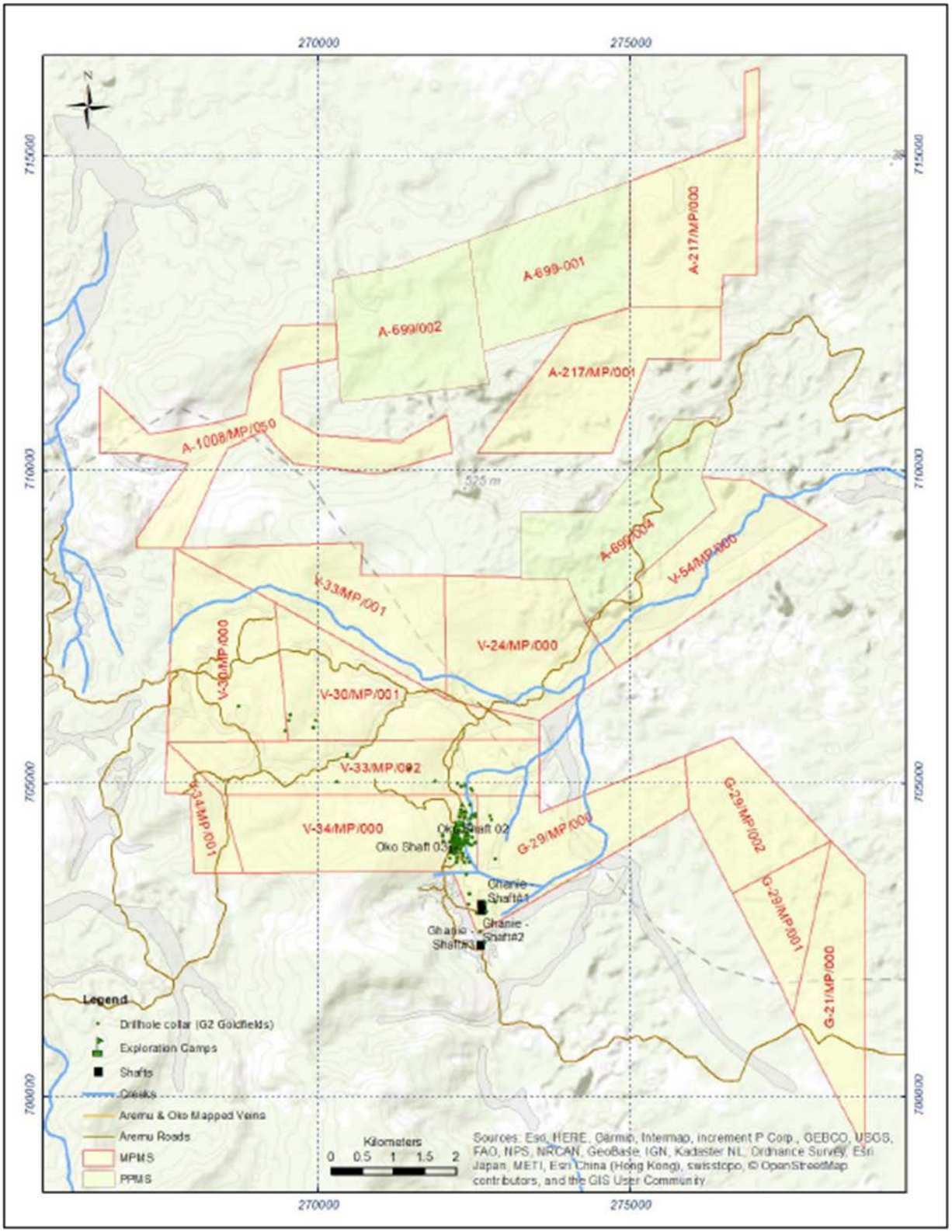
Source: GMIN, 2023

23.1 G2 Goldfields Inc.

23.1.1 Mineral Rights

In 2016, Sandy Lake Gold Inc. collected grab samples at Crusher Hill, a gold prospect north of Oko West, reporting high gold grades in shaft stockpiles associated with quartz, and quartz-carbonate veins in narrow mineralized zones (Ilieva, 2018). This reconnaissance became the basis for Sandy Lake Gold changing its name to G2 Goldfields (G2), to seek an option agreement with the title holder. In December 2017, G2 acquired an interest in additional medium-scale mining permits. G2's permit portfolio has since increased to 18 medium-scale prospecting and mining permits covering 18,837 acres (7,623 ha) (Figure 23.2). There are several artisanal gold operations located on the permits, including two (2) shafts.

Figure 23.2: Mineral Title Held by G2 Goldfields (Ilieva et al., 2022)



Source: Micon International, 2022

23.1.2 Exploration Work

In 2018 and 2019, G2 Goldfields completed a soil geochemical survey and geological mapping, and the results were used for outlining soil anomalies and drill-hole targeting. Since September 2019, G2 has conducted diamond drilling programs targeting areas with known small-scale mining operations, particularly below the Crusher Hill pit. From September 2019 to March 2022, G2 drilled 116 drillholes numbered from OKD 01 to OKD 116 for a total of 28,809 m in three (3) areas called Oko Main zone, Oko Northwest and Oko South (Ghani zone). A second drilling campaign was launched in March 2022 and is still ongoing at the time of writing. This program aims to identify new gold-bearing geological structures and better define known ones. Some 303 surface drillholes for a total of 68,385 m have been drilled between March 2022 and March 2024.

23.1.3 Mineral Resources

G2 Goldfields contracted Micon International Ltd. to undertake an update of the mineral resource estimate of the Oko Gold Project. The mineral resource estimate includes the "Oko Main" deposit, as well as a newly defined area south of the Oko Main called the "Ghanie" zone. The mineral resource estimate is based on the drilling information available up to March 2024 (Lewis et al., 2024), with an effective date of March 27, 2024. Micon classified the mineral resources at the Oko Gold Project into the Indicated and Inferred categories, assuming both open pit and underground operations. Micon used a gold price of US$1,900/ounce, recovery of 85%, mining cost of US$2.5/t in saprolite, US$2.75/t in fresh rock and US$75.0/t in underground, as well as a processing cost of US$12/t for saprolite and US$15/t for fresh rock. Cut-off grades of 0.33 g Au/t in saprolite, and 0.39 g Au/t in fresh rock for open pit operations and 1.80 g Au/t for underground operations were used. Open pit and Underground Indicated Mineral Resources are estimated at 2,364 kt, grading 9.03 g/t Au for 686 koz Au for the Oko Main zone and 3,344 kt, grading 2.20 g Au/t for 236 koz Au for the Ghanie zone. Open pit and Underground Inferred Mineral Resources are estimated at 2,413 kt, grading 6.38 g Au/t for 495 koz Au for the Oko Main zone and 12,216 kt, grading 1.54 g Au/t for 236 koz Au for the Ghanie zone.

24 OTHER RELEVANT DATA AND INFORMATION

This chapter presents other relevant data and information that are pertinent to the understanding and evaluation of the mineral project but are not covered in the preceding sections. The information herein aims to provide a comprehensive overview of the project execution and ensure that all relevant aspects are considered in the assessment of the project.

24.1 Project Execution Plan

An integrated project management team (IPMT) will be created to lead the execution of the Oko West Project using a self-perform approach. The project team will be supplemented by contractors working within the IPMT for both specialized needs and peak manpower requirements. The plan is for the IPMT to lead the project execution and construction of all on-site infrastructure and the process plant. Mine development will also be self-performed by the project team. Off-site infrastructure, including the access road and powerline, will be built or upgraded by a contractor under the supervision of the IPMT.

The project team will work in unison to achieve project objectives through the effective use of the Owners' equipment, material and personnel, and to minimize difficulties common to commissioning and start-up. The IPMT will share the responsibility for the planning and execution of the project.

The project team will use a quality assurance / quality control (QA/QC) system in all phases of the project (engineering, procurement, construction, commissioning and start-up). The system will include audits, certification, factory inspection, and destructive and non-destructive testing during construction, as appropriate. It will also provide traceability from origin for each component of the project. The detailed procedures and practices will be developed with the project QA/QC team integrating the project document control process.

The Mine and Mill operations team will be recruited during construction and will work within the IPMT during construction, pre-commissioning, commissioning, start-up, and handover of the facilities. Handover will be a structured and planned process that will require strong coordination between the project personnel and operations team for a gradual transfer of responsibility.

All departments included in General Services will be staffed with owner employees to service the construction efforts. These departments will become fully operational early in the construction period, with a trained workforce and established programs and service providers. As construction is demobilized, the service departments will seamlessly continue into operations.

The Owner mining team will consist of operations, maintenance and technical services and will be a critical component of the overall project team as it will execute a great deal of the mass earth movement.

The project team will be headed by the project director who will provide overall project management during construction. The Owner operations team will take full control of the operation after commercial production is achieved. Commercial production is defined as the first day of a period of 30 days when the ore tonnage processed averaged at least 60% of nameplate.

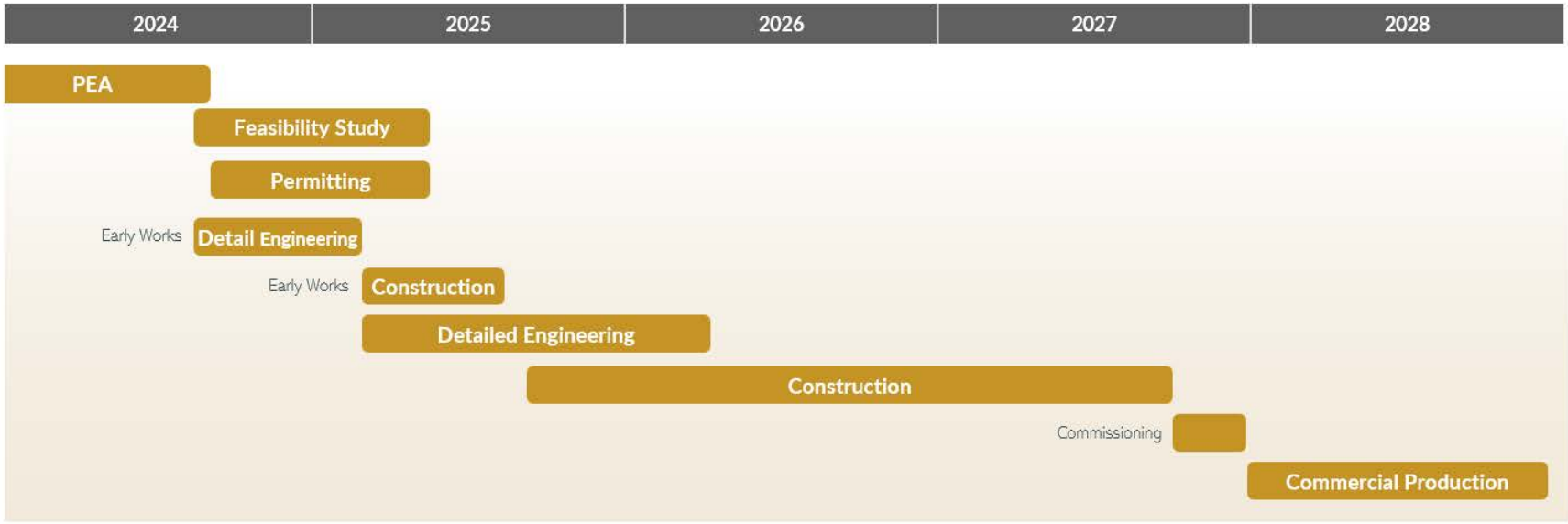
24.2 Project Schedule

Schedule development begins at the management level and drills down through the project / control levels. The management level schedule is used to establish work goals and overall time frames for the scope of work. Project objectives are detailed in Figure 24.1. This Level 1 schedule contains the least amount of detail but provides a high-level tool for management to evaluate and track the main project milestones. The Project schedule will define the detailed tasks and the duration of each task.

A preliminary project schedule is provided in Figure 24.1. It will be further detailed and optimized as the Project goes through the next stages of Feasibility Study and Project Execution.

As the Project is under pressure by the authorities to be fast tracked, the project schedule assumes that the detailed engineering and execution of the Early Works will start as early as September 2024 and be completed in August 2025, at which time full construction will be initiated for a period of 24 months. Commissioning is scheduled for four (4) months, including a one-month contingency to enable the start of commercial production in January 2028. This is obviously an aggressive schedule.

Figure 24.1: Oko West Project Schedule – Level 1



Source: GMS (2024)

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this PEA Report is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an open pit and underground mine thereat, including processing facilities and related infrastructures.

This NI 43-101 Technical Report confirms the potential technical feasibility and potential economic viability based on an open pit mining and underground operation with average gold production at 353 koz per year over a 12.7-year life-of-mine ("LOM"). It is recommended to advance the Project to the Feasibility Study phase. Table 22.1 shows the Preliminary Economic Assessment.

Table 25.1: Oko West Preliminary Economic Assessment Highlights

Technical Report Feasibility Study Update Life-of-Mine Results		
Gold Price – Base Case	USD/oz	1,950
Mine Life (operation years)	Mt	12.7
OP Mill Feed Tonnage	Mt	61
UG Mill Feed Tonnage	Mt	15
Total Mineralized Material Mined	Mt	75
Total Waste Mined (OP and UG)		367
Total Tonnage Mined (OP and UG)		443
Strip Ratio	waste: mineralized material	6.0
Average Milling Throughput	**Mtpa**	**6**
Average Milling Throughput	**tpd**	**16,110**
Gold Head Grade	g/t	2.00
OP Head Grade	g/t	1.72
UG Head Grade	g/t	3.19
Contained Gold	koz	4,848
Average Gold Recovery (%)	%	92.8%
Total Gold Production	koz	4,500
Average Annual Gold Production	koz	353
Operating Costs (LOM Average)		
Open Pit Mining Cost	USD/t mined	$2.49
Underground Mining Cost	USD/t milled	$55.45
Processing Cost	USD/t milled	$14.97
G&A Cost	USD/t milled	$4.14
Total Site Costs	USD/t milled	$51.15
Total Site Costs	USD/oz	$728
Government Royalties	USD/oz	$126
Total Operating Cost	USD/oz	$853
AISC	USD/oz	$986
Capital Costs		
Capital Costs	USD MM	$836
Contingency	USD MM	$100
Total Capital Cost	USD MM	$936
Initial UG Capital Costs (Sustaining Capital)	USD MM	$124
OP and UG Sustaining Capital	USD MM	$413
Life of Mine Sustaining Capital	USD MM	$537
Closure Costs	USD MM	$37
Total Capital Costs	USD MM	$1,510
Financial Evaluation		
After-Tax NPV 5%	USD MM	$1,367
After-Tax IRR	%	21%
Payback	years	3.8

25.2 Geology and Mineral Resources

As of February 7, 2024, sufficient gold mineralization to define a Mineral Resource Estimate (MRE) has been intercepted as part of drilling at the Oko West Project over a strike length of 2.2 km with a down-dip extent greater than 1,000 m. Additional gold mineralization has been defined in other areas within the Project extents that warrant further exploration follow up, both along strike and up to 4 km south of the MRE, and in the western portions of the Prospecting Licences (PL).

Gold mineralization is visually associated with carbonitization-albitization, silicification and sericitic alteration of a sequence of sediments and volcaniclastics. Elevated gold assays are often visually associated with strong alteration, brittle deformation and shearing and sulfidation of the host rock. The litho-structural setting of gold mineralization is relatively well understood, and controls on mineralization are also well understood.

Drilling methods employed at the Oko West Project are typical and follow industry standards used to delineate gold mineralization. Diamond drilling is the principal method, and core recovery is considered excellent. Reverse circulation (RC) drilling is used primarily for scout and reconnaissance drilling. Sampling methods and QA/QC practices are in accordance with industry standards and sufficient controls are in place to ensure a robust drilling database. The drilling database, verified and validated by the QP for the MRE, consists of approximately 80.1 km of assayed diamond drilling (including wedged holes), 21.2 km of assayed RC drilling and 6.0 km of assayed trenches.

Independent sampling reproduced original assay values present in the database within acceptable limits of error. Subsequent database checks have demonstrated that the drilling database is robust and error-free.

The total pit constrained Indicated Mineral Resource is reported at 64,115 kt grading 2.06 g/t Au, for a total of 4,237 koz of gold. The total pit constrained Inferred Mineral Resource is reported at 8,107 kt grading 1.87 g/t Au, for a total of 488 koz of gold. The underground Mineral Resources are estimated from zones outside the constrained Mineral Resources of the open pit. The total constrained underground Indicated Mineral Resource is reported at 491 kt grading 1.85 g/t Au, for a total of 29 koz of gold. The total constrained underground Inferred Mineral Resource is reported at 11,510 kt grading 3.01 g/t Au, for a total of 1,116 koz of gold. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The Mineral Resource described have been prepared in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and follow the CIM Mineral Reserve and Mineral Resource Guidelines (2019).

25.3 Mining

The Oko West Project is planned as a mining operation that integrates both conventional open pit (OP) mining and mechanized long hole open stoping for the underground (UG) mine. The initial milling rate is set at 6 Mtpa for processing hard rock, increasing to 7 Mtpa when incorporating saprolite, following a 5-month ramp-up during the open pit phase. The milling process is designed to operate for 13 years, with stockpiles peaking at 4.4 Mt by Year 2 to maintain consistent mill feed. A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options.

The OP will utilize a fleet of diesel-powered equipment, including drills, haul trucks, and hydraulic shovels. The Project consists of a main pit that is deeper and centered on Block 4, with two (2) smaller sub-pits positioned on the southern extension to the main one. The OP operation will be executed in four (4) phases. The OP peak mining rate is 44.0 Mtpa over a Life-of-Mine (LoM) of 13 years. A total of 60.7 Mt of mineralized material will be mined at an average diluted gold grade of 1.72 g/t Au. A total of 364.6 Mt of combined waste and overburden will be extracted, resulting in a strip ratio of 6.0 tonnes of waste per tonne of mineralized material. The primary production equipment includes 22 m^3 diesel-hydraulic shovels paired with 136-t off-highway mining trucks for the mineralized material and waste. The mining operation is planned to be fully owner-operated, with pre-production mining scheduled over approximately 24 months to secure construction material and to remove overburden to allow access to the mineralized material. A total of 28.4 Mt of waste and overburden as well as 3.5 Mt of mineralized material will be mined in the pre-production and ramp-up period.

The UG operation consists of one (1) mine separated in three zones: the main zone and two (2) satellites zones, all accessible from a mine portal through the main decline. The selected mining method is long hole open stoping (LHOS), including transverse stoping and longitudinal stoping variations.

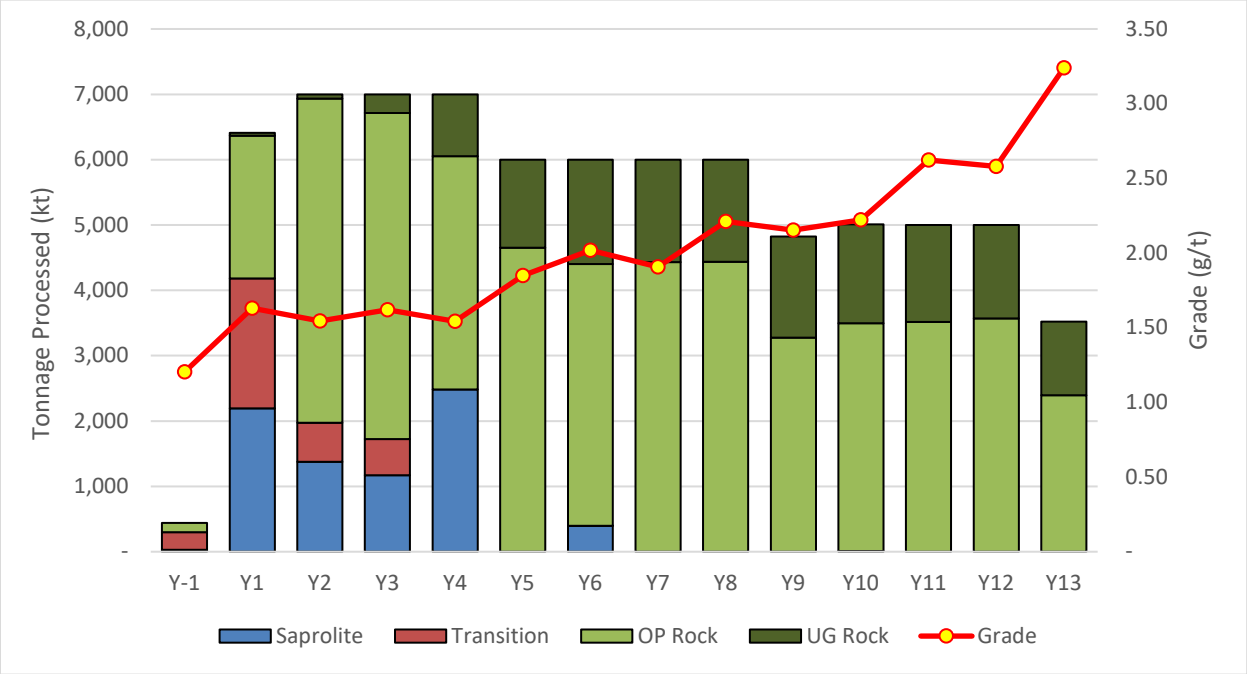
The LoM for the UG mine is expected to be 13 years including construction, development, pre-production and the full production period. Over this LoM, the UG mine is expected to be in production for 11 years, including a 2-year ramp-up period. A two-year pre-production period is planned to allow sufficient underground development to be completed and sustain full production. Initially, a contract mining period is anticipated for the construction and development of the mine followed by a transition to full owner-operated mining activities.

The UG mine is expected to achieve an average production rate of 4,250 tpd of mineralized material, with 4,000 tpd derived from stope production and 250 tpd from lateral development. Development of the UG mine includes approximately 47.0 km of lateral and 3.2 km of vertical development to be excavated. A total of 14.5 Mt of mineralized material is expected to be mined at an average diluted gold grade of

3.19 g/t Au. The primary production equipment includes 21-t diesel-powered load-haul-dump machines (LHD) coupled with 63-t underground mining trucks to handle all mined material.

The planned production from the underground and open-pit mines is expected to be 75.2 Mt of mineralized material. Figure 25.1 shows the processing plan for the material based on its type and origin.

Figure 25.1: Processing by Rock Type



25.4 Metallurgical Testing and Mineral Processing

A metallurgical test work program conducted from May to September 2023 at Basemet Laboratories (BML) aimed to assess the metallurgical response of material domains within the Oko West deposit, determine initial metallurgical recoveries, and develop an initial flowsheet. The scope included chemical analysis, mineralogy, comminution, gravity, leach, cyanide detoxification, and acid-base tests.

Samples were selected from three weathering zones (saprolite, transition, and fresh rock) and main geological units (volcanics, metasediments, and carbonaceous sediments), resulting in 18 master composites. Gold content in the samples varied between 0.50 and 2.48 g/tonne (as expected) and silver ranged between 0.1 and 1.8 g/tonne. Sulphur in the samples measured between 0.01 and 0.77 percent, indicating a relatively small sulphide mineral component. Base metal content in the samples were low, i.e., 67 ppm Cu, 88 ppm Zn, 7 ppm Pb (average). Arsenic and mercury levels were low at 8.5 ppm As and

< 1 ppm Hg. The Preg-robbing value (PRV) of select samples was measured and indicated near negligible values of preg-robbing index (PRI).

Bulk Mineral Analysis (BMA) using QEMSCAN, was conducted on the samples. Pyrite accounted for the main sulphide mineral in almost all the samples. Chalcopyrite, sphalerite and other sulphides were also detected in lower concentrations in the majority of the other samples. The non-sulphide suite of minerals varied, consisting mainly of quartz, feldspars, muscovite / illite and chlorite and clays.

From a material hardness point of view, fresh rock is more competent, hard and abrasive in comparison to the saprolite and transition material. The fresh rock exhibits competent material (Axb 15th percentile of 32.4), hard grindability (85th percentile BBWi of 14.8 kWh/tonne) and is mildly abrasive (Ai of 0.133).

Gravity recoverable gold tests were performed on all samples using a Knelson concentrator. Gravity gold recovery for the Fresh Rock samples ranged between 36% and 63%. Gravity gold recovery for the saprolite samples ranged between 27% and 46%.

Acid base accounting (ABA) tests were completed on blended composites (following cyanide destruction testing) and on waste rock samples. All samples, except for one of transition samples had net neutralizing potential (NNP) greater than zero indicating those materials are potentially acid neutralizing. All samples except for the transition samples had neutralizing potential ratio (NPR) greater than 4.1, indicating no potential for ARD.

Whole-of-ore leach tests showed high overall gold extraction rates, with finer primary grind sizes resulting in higher extraction but also higher cyanide consumption. Subsequent gravity-leach tests and carbon-in-leach (CIL) tests showed consistently high gold extraction rates.

Overall, gold recoveries from gravity-leach tests yielded the best results, with average Au recovery of 96.0% for saprolite, 95.0% for transition and 92.5% for fresh material types.

25.5 Recovery Methods

The proposed process plant design for the Oko West Project is based on a standard metallurgical flowsheet to treat gold bearing material to produce doré. The flowsheet is based on metallurgical test work, industry standards and conventional unit operations.

The process plant is designed to nominally treat 6 Mtpa of fresh rock and includes the following unit operations:

- Comminution circuit (primary gyratory crusher, SAG and ball mill) to produce a primary grind size of P80 of 75 µm.
- Gravity concentration circuit to produce a gold-rich concentrate for intensive leaching and subsequent gold recovery via electrowinning.
- Pre-leach thickening.
- Cyanide leaching, and carbon adsorption via a Carbon-in-Leach (CIL) circuit. CIL residence time of 48 hours to achieve optimal gold extraction.
- Carbon elution via 10-t Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.
- Cyanide destruction of CIL tailings using SO_2 / air process to produce weak acid dissociable (WAD) cyanide levels of less than 10 ppm.
- Tailings pumping to a tailings storage facility.
- Air and oxygen circuits.
- Water systems (potable water, raw water, gland seal water and process water).

25.6 Environmental, Social and Permitting Considerations

- A Project specific baseline study program was initiated in 2022, and information collected to date has helped build an understanding of the local and regional environmental and social conditions that have informed the PEA.
- Data collected to date and reviewed by ERM has not identified any material issues of concern that would, at this stage, deem a project not viable in this location.
- The Project area has not been identified as a priority area of conservation interest by the Government of Guyana, nor does it fall in or near a Guyana Protected Area, a World Heritage Site, an International Union for Conservation of Nature Key Biodiversity Area, or an Alliance for Zero Extinction site. Despite the absence of designated protected areas nearby, the Project Area's ecological value highlights the importance of integrating conservation considerations into project planning and implementation.

- Giant Otters (Pteronura brasiliensis) are an IUCN red-listed endangered species and were physically noted as being present in the Project area during the baseline field campaign in 2023. In response, more detailed studies are underway aimed at gaining an understanding of the giant otter population size, distribution, and habitat use within the Project Area and surrounding river catchments. The ultimate purpose of the survey will be to determine the criticality of the Project area to the Giant Otter population in the region so that appropriate mitigation measures can be implemented. Depending on the results of this survey, a Biodiversity Management Plan, including the need for habitat offsets may be required to be implemented. The cost of mitigation measures and potential constraints on Project design and operation are currently unknown.
- Archaeological field surveys have identified a regionally and in the wider Guyana context important archaeological cave site (Cave-1), that can be classified as 'Non-Replicable Cultural Heritage' as per the IFC Performance Standard for Cultural Heritage (PS8). The cave is located within the footprint of the proposed open mine pit and as such will need to be removed if the project proceeds. The mitigation of Cave-1 is not expected to represent a significant cost item to the project and if appropriately scheduled, would not cause schedule implications.
- A geochemistry baseline program is underway to evaluate the metal-leaching and acid rock drainage (ML/ARD) characteristics of future waste materials at Oko West. The program's objectives are to characterize the ML/ARD properties of anticipated waste materials (waste rock, tailings, and overburden) through static and kinetic environmental testing. The program structure is based on global best practice (Price,1997; MEND, 2009) for environmental geochemical characterization. The results will be used to inform water-quality modelling and to develop effective management strategies for these materials throughout the project's lifecycle. Based on currently available information ML/ARD is not expected to present a material environmental risk.
- The site will interact with surface and subsurface waters, requiring diversion of non-contact waters and management of contact waters to mitigate against environmental impacts. A preliminary water management plan has been developed for the PEA and regular supernatant discharges of excess water inventory from the tailings storage facility (TSF) is expected. Treatment of TSF discharge may be required for specific parameters of concern to meet regulatory requirements and/or mitigate against environmental impacts. Active treatment of contact water may also be required for some time post mine closure.
- Initial conceptual closure planning has been initiated, and a conceptual closure cost estimate has been developed to support the PEA.
- Reunion Gold has established strong relationships with some key stakeholders in the Project area, including formal titleholders, organizations, businesses, and/or individuals. Engagement to date has included long-term and informal relationship-building, public scoping consultation meetings in

December 2023 as required for regulatory purposes, and increasingly focused efforts that align with mine planning and development. Feedback and perspectives on the Project collected as part of stakeholder engagement thus far center on the following themes:

  - Livelihood diversification and income-generating opportunities (especially for Indigenous women) with the mine and/or supported by the Oko West Project.
  - Education, including programs to support ongoing school attendance and limit secondary school drop-out rates by adolescents who leave school to support their families by finding work in mining.
  - Safety, health, and wellbeing, including concerns about impacts on Amerindian communities as a result of any potential increases in traffic and non-community visitors with mine development (including ancillary facilities);
  - The Government of Guyana has not granted formal title to any Amerindian (or Indigenous) lands within the PL; however, there are Amerindian land titles in very close proximity to Project ancillary facilities.

- Reunion Gold was required to obtain an environmental authorization (also commonly referred to as an Environmental Permit) from the EPA to conduct its current exploration activities. In September 2023, the Company filed an initial application and subsequently collaborated with the EPA to establish the Terms and Scope (ToS) of a future environmental impact assessment. As part of this process, the Company conducted meetings with both government agencies and local communities in the last quarter of 2023 to determine the essential elements to be incorporated into the ToS. The approval of the ToS was required for the Company to move forward with work on an environmental impact assessment (EIA). The EIA is currently under preparation.

25.7 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total USD 1.510 billion consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 6 Mtpa of fresh rock. Initial capital costs total USD 936.2M net (including $100.3 million for contingency and $29 million in pre-production revenue), which will be expended over a 32-month design, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations and also the underground mining development. Sustaining capital costs are estimated to be $537.5 million including indirect costs and do not include contingency.

- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs a total of $36.6 million and includes a 20% contingency. The capital and sustaining expenditures are summarized in Table 25.2 according to the Level 1 work breakdown structure ("WBS").

Table 25.2: Initial and Sustaining Capital Expenditures Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
100 – Infrastructure	70,763	5,091	**75,854**
200 – Power and Electrical	118,243	25,598	**143,841**
300 – Water Management	16,318	11,267	**27,585**
400 – Surface Operations	45,952	-	**45,952**
500 – Mining	128,910	447,518	**576,428**
600 – Process Plant	190,010	22,000	**212,010**
700 – Construction Indirect	107,496	-	**107,496**
800 – General Services / Owner's Cost	111,432	-	**111,432**
900 – Pre-production, Start-up, Comm.	46,746	26,020	**72,766**
990 – Contingency	100,304	-	**100,304**
Total	**936,174**	**537,494**	**1,473,668**

The operating costs include mining, processing, general services and administration ("G&A"), gold transportation and refining and royalties. The average LOM operating cost is USD 853/oz of gold or USD 51.15/t milled. The average LOM all-in sustaining cost ("AISC") is USD 986/oz of gold or USD 59.13/t milled.

25.8 Economic Analysis

The economic analysis is carried out in real terms (i.e., without inflation factors) in Q3 2024 United-States dollars.

The base case economic analysis using a gold price of USD 1,950/oz has an after-tax NPV 5% of USD 622M, IRR of 24.2% and payback of 3.2 years after the start of production.

The Project financial performance is most sensitive to the gold price and to a lesser degree to the capital and operating costs.

25.9 Risks and Opportunities

25.9.1 Risks

The main Project risks are:

- Very few drillholes are oriented along the mineralization and may miss East-West trending faults shifting mineralization, especially at depth for the underground Mineral Resource. This is considered as a low risk since continuity between drilling sections is generally excellent throughout the mineralization domains.
- Tension with Venezuela (dispute of land ownership).
- Barge landing lease delays.
- Piracy targeting in-country logistics.
- Limited labor availability, notably on account of competition from the oil and gas industry.
- Limited room available at Georgetown port.
- Delays in procurement from China.
- Very dry climate limiting barging to barge landing.
- Completion of permit.
- Continued inflationary pressure.
- Availability of goods and services in a remote location.
- Limited information on hydrogeology and geotechnical for mining.
- Recent changes of laws and regulations in country.

25.9.2 Opportunities

The main Project opportunities are:

- Increased Mineral Resources at depth and positive Mineral Resource conversion from Inferred to Indicated.
- Exploration potential property wide.

- Increase the size of underground stopes (level spacing).
- Reduce the amount of cement required for the CRF by modifying certain stopes (e.g., Avoca method, further reducing the size of primary stopes and increasing the size of secondary ones).
- Reduction of ventilation by adopting Canadian legislation.
- Optimization of electrical production (e.g., LNG genset).
- Toll milling.
- Sale of aggregates generated from mining of waste rock.
- Accelerate UG development.

26 RECOMMENDATIONS

The results of the financial analysis presented in this Preliminary Economic Assessment (PEA), demonstrate positive project economics. GMS recommends that additional work be undertaken to generate a Feasibility Study for the project. Table 26.1 summarizes the proposed budget to advance the project to the Feasibility Study stage, considering the recommendations discussed in this section. The proposed Feasibility Study budget totals $31.7M.

Table 26.1: Cost Estimate Associated with Recommendations

Description	Amount (USD 000)
Infill Drilling and Extensions	15,000
Resources & Mining Engineering	622
Metallurgical Testing Program	610
Geotechnical Studies for Mining	1,065
Geotechnical Drilling & Testing	1,500
Environment – Baseline Survey and ESIA	5,000
Project Engineering	5,000
Contingency at 10%	2,900
Total	**31,697**

26.1 Geology and Mineral Resources

Based on the results of the Preliminary Economic Assessment described in this report, GMIN plans to continue its exploration program in 2024, in parallel with the preparation of a Feasibility Study ("FS"). GMIN intends to pursue infill drilling programs to upgrade the Inferred Mineral Resource to the Indicated Mineral Resource category and to conduct exploration drilling below the OP Mineral Resource to potentially increase the current underground mineral resources. GMIN also plans to explore other targets on the Project.

GMS makes the following recommendations to advance the Oko West Project:

- Update the geological 3D model using all available information, including the geochemical and geostructural databases.
- Prepare a geostatistical study to determine the optimal drill spacing for Measured Mineral Resource.
- Continue infill drilling to convert Inferred Mineral Resources to Indicated Mineral Resources.

- Complete the estimation of Mineral Resources as part of the Feasibility Study.

26.2 Mining

26.2.1 Geotechnical Studies for Pit Slopes

It is recommended to proceed with a targeted program of geotechnical drilling and analysis for the next phase of project development to better understand potential bench configurations and overall pit slope angles.

26.2.2 Geotechnical Studies for Underground Workings

It is recommended to proceed with a targeted program of geotechnical drilling and analysis for the next phase of project development to optimize stope sizing, and mining infrastructure.

26.2.3 Alternatives Optimization

It is recommended to perform an analysis of different alternatives.

- Optimize the mining sequence and determine the ultimate pit based on the new OP and UG resources.
- Evaluate the potential of a production shaft for the underground mine or other material handling approach.
- Evaluate the potential for automation or electrification of the underground and/or open pit mines.
- Evaluate the potential for contract mining to reduce the initial capex.

26.3 Metallurgical Testing and Mineral Processing

It is recommended to continue variability metallurgical test work of the main material domains to confirm the metallurgical response across material zones, which includes the following scope of work:

- Head assays and ICP analysis
- Quantitative mineralogy tests
- Comminution tests
- Gravity tests

- Grind-leach determination tests
- Pre-robbing tests
- Gravity and gravity tails leach and CIL tests
- Cyanide destruction tests
- Sequential triple contact carbon loading tests
- Oxygen uptake tests
- Static and dynamic settling tests
- Flocculant screening tests
- Viscosity (shear-rate) tests
- Acid-base accounting tests

26.4 Recovery Methods

The following is recommended related to the process plant:

- Finalize SAG and ball mill sizing once further comminution test work has been completed.
- Finalize gravity, CIL and cyanide destruction circuit sizing once further metallurgical test work has been completed.
- Optimize process plant reagent consumption by material type once further metallurgical test work has been completed.

26.5 Project Infrastructure and Plant Design

The following is recommended to be completed during the detailed engineering phase of the future surface plant and required infrastructure.

- Complete geotechnical investigation for the plant foundations. Balance cut and fill requirements.
- Perform geotechnical investigation for off-site infrastructure including barge landing, power plant and logistics hub.
- Complete the LiDAR survey from the off-site infrastructure to define the topography more accurately.
- Engineering for transmission line.
- Complete test work required to confirm adequate source of on-site sand for concrete quality requirements.

- Review hydrology and water management.
- Trade-off study for location of power plant: barge landing or mine site.
- Evaluate scenario where LNG would replace HFO at power plant.
- Perform in-situ and laboratory test work to support design of the TSF dams and water management.

26.6 Environmental, Permitting and Social Considerations

Additional environmental and social data will need to be collected, and additional studies will be required to contribute to the next stages of Project design, the identification and mitigation of potential impacts on its receiving environment and the submission of an environmental impact assessment (EIA) for regulatory purposes. These studies will need to further incorporate ancillary project components such as power supply and site access roads.

- Treatment of TSF discharge may be required for specific parameters of concern to meet regulatory requirements and/or mitigate against environmental impacts. Active treatment of contact water may also be required for some time post mine closure. The following work is required to inform the next stages of design:
  - Completion of initial water resource and related baseline studies (surface water, groundwater, sediment, climate).
  - Preparation of a site-wide water balance and water quality numerical model to support advancement of a water management plan and associated infrastructure design (water and waste management).
- In order to advance closure planning and closure cost estimating, it is recommended further studies take place with regards to:
  - A detailed materials balance, to ensure that an adequate quantity and quality of closure materials will be available at cessation of operations.
  - Studies to further refine the geochemical characterization of the waste rock, ore and tailings, in order to adequately assess closure criteria, objectives and activities.
  - Further studies indicate timings of pit recharge and anticipated water quality.
  - At the next level of study, labor and equipment rates that form the basis of the closure cost estimate should be informed by and aligned with operational rates.
- Detailed recommendations for future environmental and social baseline data collection and studies are provided in Section 20.1.2.

27 REFERENCES

Basemet (2023). *Metallurgical Evaluation of Oko West Project*, by Base Metallurgical Laboratories Ltd, Report BL0949, September 27, 2023.

Bertoni C., Shaw, R.P., Singh, R., Minamoto, J., Richards, J.M., and Belzile, E. (1991). *Geology and Gold Mineralization of the Omai Property, Guyana*. Brazil Gold 91, Balkema, Rotterdam.

Bertoni, C., Garner, D., Tachibana, J., van der Toorn, J., Felmer, M., Abrams, R., McLennan, R., Tsige, Z. (2022). *The Oko West Gold Deposit: A new discovery in the Guyana Shield*. XII Interguiana Geological Conference, Guyana.

Cox, S. F. (2007). *Structural and isotopic constraints on fluid flow regimes and fluid pathways during upper crustal deformation: an example from the Taemas area of the Lachlan Orogen*, *SE Australia*. Journal of Geophysical Research: Solid Earth, 112 (B8).

Cox, S. F., Sun, S. S., Etheridge, M. A., Wall, V. J., & Potter, T. F. (1995). *Structural and Geochemical Controls on the Development of Turbidite-hosted Gold Quartz Vein Deposits, Wattle Gully mine, central Victoria, Australia*. Economic Geology, 90(6), 1722-1746.

Daoust C. (2016). *Caractérisation stratigraphique, structurale et géochimique du district minéralise de Rosebel (Suriname) dans le Cadre de l'évolution géodynamique du Bouclier Guyanais*. PhD Thesis. Université du Québec à Montréal, Montréal, pp. 330.

Delor C., Lahondère D., Egal E., Lafon J.M., Cocherie A., Guerrot C., Rossi P., Truffet C., Theveniaut H., Phillips D., Avelar V.G. (2003a). *Transamazonian crustal growth and reworking as revealed by the 1:500000 scale geological map of French Guiana*. Géologie de la France, 2-3-4, 5-57.

Delor C., Roever E.W.F. de, Lafon J.M., Lahondère D., Rossi P., Cocherie A., Guerrot C., Potrel A. (2003b). *The Bakhuis ultra-high-temperature granulite belt (Suriname): II. Implications for the late Transamazonian crustal stretching in a revised Guiana Shield framework.* Géologie de la France, 2-3-4, 207-230.

ERM, 2022. *Oko West Project interim reports on environmental baseline surveys*.

ERM, 2023. *Oko West Project reports on environmental and social baseline surveys*.

ERM, 2024. *Oko West EIA Scoping Report,* approved by EPA on 29th February 2024.

G Mining Services Inc. (2023). NI 43-101 *Technical Report Oko West Gold Project Cuyuni Mining District, Guyana*. Issued on March 20, 2023, pp. 143.

G Mining Services Inc. (2023). NI 43-101 *Technical Report Oko West Gold Project Cuyuni Mining District, Guyana*. Issued on July 14, 2023, pp. 197.

G Mining Services Inc. (2024). NI 43-101 *Technical Report Oko West Gold Project Cuyuni Mining District, Guyana*. Issued on March 11, 2024, pp. 238.

Gibbs A.K. (1980). *Geology of the Barama-Mazaruni Supergroup of Guyana: unpublished Ph.D*. Thesis. Harvard University, Cambridge, MA, pp. 385.

Gibbs A.K., Barron C.N. (1993). *Geology of the Guiana Shield*. Oxford University Press, Oxford, pp. 246.

Gonzalez, I, Ribeiro C. (2024). *Rainfall Analysis and Probable Maximum Precipitation Studies*.

Granville, J-J. de (1988). *Phytogeographical characteristics of the Guianan forests*. Taxon 37: 578-594.

Hainque, P.J., Lacroix, B., Trap, P., Bertoni, C.H., Eckfeldt, M., Lahondes, D., van der Toorn, J., Le Goff, E. (2024) – *Polyphase deformation and structural controls on gold mineralization at the Oko West deposit, Guyana*. Manuscript in preparation.

Hainque, P-J. & Lacroix, B. (2022). *Geological and structural investigation of the Oko West project*. GexplOre report RP_022_2022.

Heesterman et al. (2003). *Regional Geochemistry Survey, Lower and Upper Puruni Prospects.* GGMC

Ilieva, T. (2018). *NI 43-101 Technical Report for The Aremu-Oko Gold Property, Cooperative Republic of Guyana, South America*. Sand Lake Gold Inc.

Ilieva, T., Martin, A.S., and Gowans, R. (2022). *NI 43-101 Technical report and mineral resource estimate for the Oko gold property, Cooperative* / *Republic of Guyana, South America*. G2 Goldfields report.

Kroonenberg S.B., Roever E.W.F. de, Fraga L.M., Reis N.J., Faraco T., Lafon J.M., Cordani U., Wong T.E. (2016). *Paleoproterozoic evolution of the Guiana Shield in Suriname*: A revised model. Netherlands Journal of Geosciences, Géologie en Mijnbouw, 95-4, 491-522.

Lacroix, B, Hainque, P.J. (2022). *Structural Investigation of the Oko West Project – Guyana.* GexplOre Report RP_001_2022. 61 p.

Lacroix, B, Hainque, P.J. (2024). *Site Visit to Oko West Project – Guyana*. Reunion Gold internal report. 62 p.

Lacroix, B., Hainque, P-J, Hauteville, A., Lahondes, D., L Goff, E., Fournier, D., Bertoni, C. Taravella, St. & Witasse, M., (2022). *The role of polyphase folding in the distribution of gold: Insights from the Guiana Shield*. XII Interguiana Geological Conference. Guyana.

Ledru P., Lasserre J.L., Manier E., Mercier D. (1991) *Le Protérozoïque inferieur nord guyanais: révision de la lithologie, tectonique transcurrente et dynamique des bassins sédimentaires.* Bulletin de la Société Géologique de France, 162 (4), 627-636.

Ledru, P., Lasserre, J.L., Manier, E., Mercier, D., (1991). *Le Protérozoïque inferieur nord guyanais: révision de la lithologie, tectonique transcurrente et dynamique des bassins sédimentaires*. Bulletin de la Société Géologique de France, 162 (4), 627-636.

Llieva, T. (2018). *NI 43-101 Technical Report for The Aremu-Oko Gold Property, Cooperative Republic of Guyana, South America*. Sand Lake Gold Inc.

Llieva, T., Martin, A.S., and Gowans, R. (2022). *NI 43-101 Technical report and mineral resource estimate for the Oko gold property, Cooperative / Republic of Guyana, South America*. G2 Goldfields report.

McPhie J., Doyle M., Allen RL. (1993) *Volcanic Textures: a guide to the interpretation of textures in volcanic rocks*. University of Tasmania. Centre for Ore Deposit and Exploration studies. 196 p.

Milési J.P., Egal E., Ledru P., Vernhet Y., Thieblemont D., Cocherie A., Tegyey M., Martel-Jantin B., Lagny P. (1995). *Les minéralisations du Nord de la Guyane française dans leur cadre géologique. Chronique de la Recherche Minière*. 518, 5–58.

Milési J.P., Lerouge C., Delor C., Ledru P., Billa M., Cocherie A., Egal E., Fouillac A., Lahondère D., Lasserre J., Marot A., Martel-Jantin B., Rossi P., Tegyey M., Théveniault H., Thiéblemont D., Vanderhaeghe O. (2003). *Gold deposits (goldbearing tourmalinites, gold-bearing conglomerates, and mesothermal lodes), markers of the geological evolution of French Guiana: geology, metallogeny, and stable isotope constraints*. Géologie de la France, 2-3-4, 257-290.

Mora, R, Vanier N. (2024). *Airstrip Options Trade-Off Analysis*.

Narain, S. (1985). *Oko Area Gold Prospect*. Guyana Geology and Mines Commission. Georgetown.

P. Ledru, J. L. Lasserre, E. Manier, D. Mercier; (1991) *Le Protérozoïque inferieur nord guyanais: révision de la lithologie, tectonique transcurrente et dynamique des bassins sédimentaires.* Bulletin de la Société Géologique de France, 162 (4), 627-636.

Potvin Y., Thomas, E., Fourie A., (2005). *Handbook on Mine Fill*, Australian center for Geomechanics, pp. 179.

Runge Ian C, (1998). *Mining Economic and Strategy*, Society for Mining and Metallurgy and Exploration, pp. 293.

Saliba, F. & Avila, C, (2024). P*EA Conceptual Design Report – TSF and WRSF*, GYOW-A-001-GT-REP-0112.

Saliba, A. & Albino, U. Rafael Ribeiro, (2024). *Design Criteria Report*, GYOW-A-001-GT-REP-0111.

Saliba, A., Costa, C. (2024). *Preliminary Assessment of WSRF and TSF Alternatives*, GYOW-A-001-GT-MEM-0101.

Saliba, A., Ribeiro, R., (2024). *TSF5-Preliminary Embankment Volume and Raising Estimates*, GYOW-A-001-GT-REP-0117.

Saliba, A., Ribeiro C. (2024). *Rainfall and Probable Maximum Precipitation Studies*, GYOW-A-001-GT-REP-0113.

Santos J.O.S., Hartmann L.A., Gaudette H.E., Groves D.I., McNaughton N.J., Fletcher I.R. (2000) *A new understanding of the provinces of the Amazon Craton based on integration of field mapping and U-Pb and Sm-Nd geochronology*. Gondwana Res. 3, 453-488.

Tachibana, J. and Garner, D., (2019). *Oko West project – reconnaissance geological mapping and sampling*. Reunion Gold Inc. internal memorandum.

Tassinari C.C., Munhá J.M., Teixeira W., Palácios T., Nutman A.P., Sosa C., Santos A.P., Calado B.O. (2004). The Imataca Complex, *NW Amazonian Craton, Venezuela: crustal evolution and integration of*

geochronological and petrological cooling histories. Episodes-Newsmagazine of the International Union of Geological Sciences 27 (1), 3-12.

Tedeschi M.T., Hagemann S.G., Davis J. (2018). *The Karouni Gold Deposit, Guyana, South America: Part I.* Stratigraphic Setting and Structural Controls on Mineralization. Econ. Geol. 113 (8), 1679-1704.

Tedeschi M.T., Hagemann S.G., Kemp A.I.S., Kirkland C.L., Ireland T.R. (2020). *Geochronological constrains on the timing of magmatism, deformation and mineralization at the Karouni orogenic gold deposit: Guyana, South America*. Precambrian Research, 337.

Tedeschi, M., (2021). *Oko Prospect Visit Report. Internal Reunion Gold report*.

Thompson, J. F. H., Thompson, A. J. B., Allen, R. L. (1996). *Atlas of alteration: a field and petrographic guide to hydrothermal alteration minerals*. St. John's, Newfoundland, Geological Association of Canada, Mineral Deposits Division. 119 p.

Thompson, P.H., (2022). *Protoliths, alteration and mineralization of metamorphosed clastic and granitoid samples from Oko West. Reunion Gold internal report*.

Tsige, Z. (2022). *Oko West Project - Alteration, mineralization and key aspect of gold mineralization*. Reunion Gold internal memo.

Vanderhaeghe O., Ledru P., Thieblemont D., Egal E., Cocherie A., Tegyey M., Milési J.P. (1998). *Contrasting mechanism of crustal growth*. Precambrian Research, 92, 165-194.

Wasel, M., Lindsey, A., Donald, P., and Fleming, A., (1997). *Case history of the Gross Rosebel gold project, Suriname*. Golden Star Resources report.