Technical Report

Feasibility Study Update

Loma Larga Project

Azuay Province, Ecuador

Prepared by:

Ryan Telford, P. Eng.
DRA Americas Inc.

David Frost, FAusIMM
DRA Americas Inc.

Daniel Gagnon, P.Eng.
DRA Americas Inc.

Carl John Burkhalter, PE
NewFields Mining & Technical Services LLC

Phil Earl, ACSM BEng (Hons), MEng(sc), MAusIMM (CP) MIEAust
Global Mine Design Ltd.

Dr. Lewis Meyer, ACSM, MCSM, BEng, MSc, PhD, CEng, FIMMM, QMR, MCIM
Global Mine Design Ltd.

Sean McCarthy, P.Eng., PMP
Global Mine Design Ltd.

Leslie Correia, Pr.Eng.
Paterson & Cooke

Katharine Masun, M.Sc., MSA, P.Geo.
SLR Consulting (Canada) Ltd.

Luis Vasquez, M.Sc., P. Eng.
SLR Consulting (Canada) Ltd.

Effective Date: May 28, 2025
Report Date: October 29, 2025

Post-Effective Date Events

After the Effective Date of this Report, events have occurred with respect to the Project. On June 23, 2025, the Government of Ecuador granted to DPM Metals Inc. the Environmental License for exploitation activities for the Project. Subsequently, on October 3, 2025, after the announcement of the results of the Feasibility Study discussed in this Report on September 30, 2025, DPM received notification from the Government of Ecuador that the Environmental License had been revoked.

This Report has been prepared as of the stated Effective Date (prior to both the granting and subsequent revocation of the Environmental License), unless noted otherwise.

It is presently uncertain when, or if, the Ecuadorian government will re-grant an environmental license for the Project. Such uncertainty remains high risk to the Project.

IMPORTANT NOTICE

This Report, following National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) rules and guidelines, was prepared for DPM Metals Inc. (formerly Dundee Precious Metals Inc.) (DPM or the Company) by the Qualified Persons (QPs) noted herein. The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in DRA services, based on:

i) Information available at the time of preparation,

ii) Data supplied by outside sources, and

iii) The assumptions, conditions, and qualifications set forth in this Report. This Report can be filed as a Technical Report with Canadian Securities Regulatory Authorities pursuant to NI 43-101. Except for the purposes legislated under Canadian securities laws, any other uses of this Report by any third party are at that party's sole risk.

This Technical Report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable laws. Forward-looking statements are based upon the responsible QP's opinion at the time they are made but, in most cases, involve significant risks and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this Report, there may be other factors that could cause events or results not be as anticipated, estimated or projected. There can be no assurance that forward-looking information in this Report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements or information, Accordingly, readers should not place undue reliance on forward-looking information. Forward-looking information is made as of the effective date of this Technical Report, and none of the QPs assume any obligation to update or revise it to reflect new events or circumstances, unless otherwise required by applicable laws.

TABLE OF CONTENTS

1 SUMMARY 13
1.1 Mineral Processing and Metallurgical Testing 14
1.2 Geology and Mineral Resources 15
1.3 Mineral Reserve Estimate 17
1.4 Mining 19
1.5 Recovery Methods 22
1.6 Environmental Studies, Permitting and Social or Community Impact 22
1.7 Project Infrastructure 24
1.8 Capital and Operating Costs 25
1.9 Economic Analysis 27
1.10 Other Relevant Data and Information 31
1.11 Interpretations and Conclusions 33
1.12 Recommendations 36
2 INTRODUCTION 39
2.1 Study Participants 40
2.2 List of Qualified Persons 40
2.3 Site Visits 42
2.4 Effective Date 42
2.5 Sources of Information 42
2.6 Units and Currency 43
2.7 Non-GAAP Financial Measures 43
3 RELIANCE ON OTHER EXPERTS 44
4 PROPERTY DESCRIPTION AND LOCATION 45
4.1 Land Tenure 45
4.2 Surface Rights 53
4.3 Land Acquisition Strategy 54
4.4 Permitting 56
4.5 Royalties 56
4.6 Other Significant Factors and Risks 56
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 57
5.1 Accessibility 57
5.2 Ports 57
5.3 Road Transportation 57
5.4 Climate 59
5.5 Local Resources 61

5.6 Infrastructure 61
5.7 Physiography 61
6 HISTORY 63
6.1 Prior Ownership, Exploration, and Development History 63
6.2 Historical Mineral Resource Estimates 65
7 GEOLOGICAL SETTING AND MINERALISATION 66
7.1 Regional Geology 66
7.2 Local and Property Geology 68
7.3 Mineralisation 71
8 DEPOSIT TYPES 76
9 EXPLORATION 79
10 DRILLING 80
10.1 IAMGOLD Drilling 80
10.2 INV Drilling 80
10.3 DPM Drilling 87
10.4 QP's Opinion 88
11 SAMPLE PREPARATION, ANALYSES AND SECURITY 89
11.1 Pre-2012 Programs 89
11.2 INV 2013 and 2016-2017 Drill Program 91
11.3 Results of QA/QC Programs 93
11.4 QP's Opinion 120
12 DATA VERIFICATION 121
12.1 Site Visit 121
12.2 Manual Database Verification 121
12.3 Independent Assays of Drill Core 123
12.4 QP's Opinion 123
13 MINERAL PROCESSING AND METALLURGICAL TESTING 124
13.1 Introduction 124
13.2 Historical Metallurgical Testwork 124
13.3 2022 Metallurgical Testwork Program Update 144
13.4 2023 Jameson Dilution Tests at Base Metallurgical Laboratories 156
13.5 Deleterious Elements 164
13.6 Metallurgical Variability 165
14 MINERAL RESOURCE ESTIMATES 166
14.1 Summary 166
14.2 Mineral Resource Database 168
14.3 Geological Interpretation and 3D Solid 168

14.4 Statistical Analysis 174
14.5 Capping High-Grade Values 175
14.6 Compositing 178
14.7 Density 180
14.8 NSR Cut-Off Value 183
14.9 Variography and Interpolation Values 185
14.10 Block Model 187
14.11 Block Model Validation 188
14.12 Classification 198
14.13 Summary of Mineral Resource Estimate 202
14.14 Comparison to Previous Mineral Resource Estimate 203
14.15 Sulphur Model 209

15 MINERAL RESERVE ESTIMATES 217
15.1 Summary 217
15.2 Factors That May Affect the Mineral Reserve Estimate 218
15.3 Stope Shape Design 219
15.4 Dilution 221
15.5 Ore Recovery 225
15.6 Cut-off Value 225
15.7 Reconciliation 226

16 MINING METHODS 227
16.1 General Description 227
16.2 Mining Method 229
16.3 Geotechnical Considerations 232
16.4 Underground Mine Design 259
16.5 Mine Schedule 265
16.6 Underground Mining Fleet 269
16.7 Workforce Requirements 272
16.8 Mine Ventilation 277
16.9 Underground Service Facilities 296
16.10 Paste Backfill 302

17 RECOVERY METHODS 306
17.1 Introduction 306
17.2 Process Design Criteria 306
17.3 Production Summary 309
17.4 Process Description 311
17.5 Reagents and Consumables 314
17.6 Utilities and Services 316
17.7 Sampling and On-Stream Analysers 317

17.8 Process Control 318
17.9 Recovery Estimates 319
18 PROJECT INFRASTRUCTURE 323
18.1 Introduction 323
18.2 Site Geotechnical Investigations 327
18.3 Access Roads 327
18.4 On-Site Infrastructure and Buildings – Mine Facilities 329
18.5 On-Site Infrastructure and Buildings – Process Plant Area 330
18.6 Tailings Dewatering and Paste Backfill Plant 330
18.7 Filtered Tailings Storage Facility 334
18.8 Waste Rock and Ore Stockpiles 345
18.9 Water Management 347
18.10 Water Treatment and Distribution 350
18.11 Waste Management 354
18.12 Power Supply and Distribution 354
18.13 Communications 356
18.14 Los Pinos Camp 357
18.15 Off-Site Infrastructure 359
18.16 Transportation and Logistics 360
19 MARKET STUDIES AND CONTRACTS 361
19.1 Markets 361
19.2 Treatment Charges and Other Selling Costs 362
19.3 Commodity Prices 363
19.4 Contracts and Off- Takes 364
20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 365
20.1 Introduction 365
20.2 Environmental Setting 365
20.3 Environmental Studies and Management Plans 371
20.4 Environmental Issues 378
20.5 Environmental Permitting 378
20.6 Social or Community Requirements 384
20.7 Mine Closure Requirements 388
21 CAPITAL AND OPERATING COSTS 394
21.1 Capital Cost Estimate 394
21.2 Operating Cost Estimate 411
22 ECONOMIC ANALYSIS 418
22.1 General 418
22.2 Forward Looking Information 418
22.3 Assumptions 419

22.4 Economic Analysis Parameters 420
22.5 Revenue 422
22.6 Mineral Royalties and Agreements 423
22.7 Taxation 424
22.8 Financial Analysis Results 427
22.9 Sensitivity Analysis 430
23 ADJACENT PROPERTIES 433
24 OTHER RELEVANT DATA AND INFORMATION 434
24.1 Project Execution Plan 434
24.2 Operational Readiness 444
25 INTERPRETATION AND CONCLUSIONS 445
25.1 Geology and Mineral Resources 445
25.2 Mineral Processing and Metallurgical Testing 446
25.3 Mineral Reserve Estimate 447
25.4 Mining Methods 449
25.5 Recovery Methods 450
25.6 Project Infrastructure 450
25.7 Environmental Studies, Permitting and Social or Community Impact 450
25.8 Capital and Operating Cost Estimate 451
25.9 Economic Analysis 451
25.10 Risks 452
26 RECOMMENDATIONS 455
26.1 Geology and Mineral Resources 455
26.2 Mineral Processing and Metallurgical Testing 457
26.3 Mineral Reserve Estimate 457
26.4 Mining Methods 457
26.5 Recovery Methods 458
26.6 Project Infrastructure 458
26.7 Market Studies and Contracts 459
26.8 Environmental Studies, Permitting and Social or Community Impact 459
26.9 Economic Analysis 459
26.10 Other Relevant Data and Information 460
27 REFERENCES 461
28 ABBREVIATIONS 475
29 CERTIFICATES OF QUALIFIED PERSONS 484

LIST OF TABLES

Table 1.1 – Mineral Resource Estimate Summary, Effective as of September 29, 2023 16
Table 1.2 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023....18
Table 1.3 – Capital Cost Summary (US$ '000) 26
Table 1.4 – Project LOM Operating Costs by Major Area 27
Table 1.5 – Economic Analysis Parameters....28
Table 1.6 – Summary of Project Financial Metrics 30
Table 1.7 – Proposed Drilling Program 36
Table 2.1 – Qualified Persons and their Respective Sections....40
Table 2.2 – Site Visit 42
Table 4.1 – Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project)....45
Table 4.2 – Political and Administrative Location Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project)....45
Table 4.3 – Property Size for Economic Evaluation Stage....47
Table 4.4 – Size of the Mining Concessions as Result of the Material Division 48
Table 4.5 – Cristal Concession (Exploitation Phase – Loma Larga Project) 49
Table 4.6 – Size of the Mining Concessions as Result of Accumulation 49
Table 4.7 – Current Surface Rights Agreements (INV Metals Inc. – Loma Larga Project)....54
Table 5.1 – Estimated Mean Monthly and Annual Precipitation at Loma Larga Site 59
Table 5.2 – Mean Monthly Potential Evaporation Estimates at Loma Larga Site 60
Table 10.1 – Metallurgical and High-Grade Zone Drill Results 81
Table 10.2 – Geotechnical and Hydrogeological Drillholes Completed in 2016 – 2017 82
Table 10.3 – Selected Drill Results from the 2017 Geotechnical and Hydrogeological Drilling 83
Table 10.4 – Summary of 2017 Loma Larga Drill Program 86
Table 10.5 – 2017 Selected Loma Larga Drill Program Results 86
Table 11.1 – QA/QC Review Summary – 2013 Drilling Program 96
Table 11.2 – CRMs Used in 2013 Drilling Program....98
Table 11.3 – Summary of CRM Results....98
Table 11.4 – Summary of Field Duplicate Results 102
Table 11.5 – Summary of Reject Duplicate Results....103
Table 11.6 – Summary of Pulp Duplicate Results....105
Table 11.7 – Summary of Pulp Replicate Results....106
Table 11.8 – Summary of 2016-2017 Drilling Program 108
Table 11.9 – QA/QC Review Summary – 2017 Drilling Program 109
Table 11.10 – Gold CRMs Used in 2017 Drilling Program 110
Table 11.11 – Summary of CRM Results....110
Table 11.12 – Summary of Reject Duplicate Results....114
Table 11.13 – Summary of Pulp Duplicate Results....117
Table 12.1 – Drillhole Collar Errors 122
Table 13.1 – Ore Type Categories....127
Table 13.2 – LCT #2 Recovery to Bulk Second Cleaner Concentrate 131
Table 13.3 – LCT #1 Results 134
Table 13.4 – LCT #2 Stream Assays 136
Table 13.5 – LCT #3 Sample Head Grade....137
Table 13.6 – LCT #3 Stream Balance Data 138
Table 13.7 – LCT #3 Overall Recoveries Balanced Data....138
Table 13.8 – LCT #3 Stage Recoveries Balanced Data....139

Table 13.9 – LCT #4 Sample Head Grade – 5-Year Composite 140
Table 13.10 – LCT #4 Stream Assays – Balanced Data 141
Table 13.11 – LCT #4 Overall Recoveries – Balanced Data 142
Table 13.12 – LCT #4 Stage Recoveries – Balanced Data 142
Table 13.13 – 2022 Composite Selection Details 145
Table 13.14 – COMP1 and COMP2 Head Assays 146
Table 13.15 – COMP1 Whole Rock Analysis 147
Table 13.16 – COMP1 Head Ore and Final Tails Mineral Content and Liberation 147
Table 13.17– COMP1 Fractional Analysis 148
Table 13.18 – Jameson Dilution Test Results Summary 151
Table 13.19 – Static and Dynamic Thickening Test Results 153
Table 13.20 – Rheology Test Results 154
Table 13.21 – Pressure Filtration Test Results 154
Table 13.22 – Concentrate Detailed Assays 155
Table 13.23 – Head Assays 156
Table 13.24 – Rougher Results Summary 159
Table 13.25 – Cleaner Results Summary 162
Table 13.26 – Deleterious Elements Recovery to Copper Concentrate 164
Table 13.27 – Deleterious Elements Recovery to Pyrite Concentrate 164
Table 14.1 – Mineral Resource Estimate Summary Effective as of September 29, 2023 167
Table 14.2 – Mineral Resource Database 168
Table 14.3 – Rock Codes 169
Table 14.4 – Descriptive Statistics of Resource Assay Values 174
Table 14.5 – Capped Grade Values of Resource Assays 176
Table 14.6 – Descriptive Statistics of Resource Capped Assay Values 176
Table 14.7 – Descriptive Statistics of Resource Capped Composite Values 178
Table 14.8 – Descriptive Statistics of Resource Density Values 180
Table 14.9 – Descriptive Statistics of Composited Density Samples 183
Table 14.10 – Cut-Off Value Assumptions 183
Table 14.11 – Block Estimation Parameters 185
Table 14.12 – Block Model Dimensions 187
Table 14.13 – Block Model Field Descriptions 188
Table 14.14 – Comparison of Grade Statistics for Assays, Composites and Resource Blocks 189
Table 14.15 – Mineral Resource Estimate Effective as of September 29, 2023 202
Table 14.16 – Comparison Between the Previous Mineral Resource Estimate (March 31, 2020) and the Current Mineral Resource Estimate (September 29, 2023) Exclusive of Mineral Reserves 204
Table 14.17 – Sulphur Database Used for Block Grade Estimation 209
Table 14.18 – Descriptive Statistics of Geochemistry Resource Assay Values 211
Table 14.19 – Descriptive Statistics of Sulphur Composite Values 213
Table 14.20 – Sulphur Block Estimation Parameters 213
Table 14.21 – Descriptive Statistics of Sulphur Block Values 215
Table 15.1 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023 (GMD, 2023) 218
Table 15.2 – DSO Parameter 219
Table 15.3 – Summary of Overbreak Results from ELOS Graphs 223
Table 15.4 – Loma Larga NSR Cut-Off Parameters 225
Table 16.1 – Ore Tonnage by Activity Type 228
Table 16.2 – Drilling Parameters 231
Table 16.3 – Summary of GMD Geotechnical Domains 233
Table 16.4 – Summary of Rockmass Classification per Domain 237

Table 16.5 – Summary of Rock Strength Data....237
Table 16.6 – Initial Stress Tensor for Design....238
Table 16.7 – Summary of Fault Information....242
Table 16.8 – Number of Stopes per Domain....245
Table 16.9 – Range of Stope Dimensions per Domain....246
Table 16.10 – Ground Support Types per Domain....248
Table 16.11 – Number of Bolts per Ring....249
Table 16.12 – Support Category by Geotechnical Domains....254
Table 16.13 – Estimated Number of Stopes Potentially Requiring Cable Bolt Support....255
Table 16.14 – Development Dimensions by Type....259
Table 16.15 – Development Quantities by Cost Centre....265
Table 16.16 – Typical Stope Production Rate by Activity....266
Table 16.17 – Advance Rates per Cross Section....266
Table 16.18 – Underground Schedule Summary....268
Table 16.19 – Yearly Backfill Volume Summary....268
Table 16.20 – Required Mobile Equipment by Equipment Type Year....270
Table 16.21 – Annual Net Operating Hours by Equipment Type....271
Table 16.22 – Mine Personnel List....273
Table 16.23 – Drive Dimensions....277
Table 16.24 – Temperatures and Humidity Recordings....279
Table 16.25 – Environmental Ventsim Settings....280
Table 16.26 – Friction Factors....281
Table 16.27 – Shock Losses....281
Table 16.28 – Recommended Threshold Limit Values....282
Table 16.29 – Equipment Airflow Requirements....283
Table 16.30 – Heat Settings....284
Table 16.31 – Air Velocity Limits....284
Table 16.32 – Airflow Requirements in the First Year of Operations....286
Table 16.33 – Airflow Requirements in H1 of the First Year....287
Table 16.34 – Proposed Main Fan Design Parameters....294
Table 16.35 – Main Surface Fan Movements and Installations....294
Table 16.36 – Strengths and Cure Times Requirements....302
Table 16.37 – Target Strengths and Estimated Binder Requirements....303
Table 17.1 – Process Design Criteria Summary....307
Table 17.2 – Concentrate Production Schedule Summary....310
Table 17.3 – Reagents and Consumables....315
Table 17.4 – Concentrator Water Balance Reconciliation....316
Table 17.5 – Sampling Requirements Summary....317
Table 18.1 – Site Electrical Power Demand....356
Table 19.1 – Average LOM Payable Rates per Metal per Concentrate....362
Table 19.2 – RCs for the Different Metals in Each Concentrate....363
Table 19.3 – Commodity Prices Selected....364
Table 20.1 – List of Approved and Pending Environmental Permits....382
Table 21.1 – Capital Cost Summary (US$ '000)....395
Table 21.2 – Foreign Exchange Rates....396
Table 21.3 – Pricing Source Breakdown....406
Table 21.4 – Project LOM Operating Costs by Major Area....412
Table 21.5 – Total Personnel Requirement (Year 5)....413

Table 21.6 – Summary of Mining LOM Average Opex 413
Table 21.7 – Summary of Processing LOM Average Opex 414
Table 22.1 – Economic Analysis Parameters 420
Table 22.2 – Financial Results 427
Table 22.3 – Financial Model Summary (tail of Closure Capex payments not shown) 428
Table 22.4 – Financial Model Summary (tail of Closure Capex payments not shown) 429
Table 22.5 – Sensitivity of After-Tax NPV and IRR to Metal Prices, Capex and Opex 430
Table 22.6 – Sensitivity to a Broader Range of Gold Price (US$/oz) 432
Table 22.7 – Sensitivity of Pre-Tax and After Tax NPV to Discount Rate 432
Table 25.1 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023 (GMD, 2023) 448
Table 25.2 – Risk Register – Critical Risks 453
Table 26.1 – Proposed Drilling Program 456

LIST OF FIGURES

Figure 1.1 – Production Graph from 2023 FSUO Mine Plan 20
Figure 1.2 – Project LOM Unit Operating Costs 27
Figure 1.3 – Sensitivity of After-Tax NPV to Metal Prices, Capex and Opex 30
Figure 1.4 – Sensitivity of After-Tax IRR to Metal Prices, Capex and Opex 31
Figure 1.5 – Project Schedule 33
Figure 4.1 – Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project) 46
Figure 4.2 – Size of the Mining Concessions as Result of the Material Division 48
Figure 4.3 – Size of the Mining Concessions as a Result of Accumulation 50
Figure 4.4 – National System of Protected Areas, Protected Forest and Vegetation Areas, State Forest Heritage 52
Figure 4.5 – Water Recharge Zone 53
Figure 5.1 – Route Options Between Guayaquil / Machala and Girón 58
Figure 7.1 – Major Terranes of Ecuador 67
Figure 7.2 – Regional Geology 68
Figure 7.3 – Property Geology 70
Figure 7.4 – Cross-Section of Loma Larga Deposit, Looking Northeast 73
Figure 7.5 – Longitudinal Section of Loma Larga Deposit, Looking East 73
Figure 8.1 – Zonation of Alteration in High Sulphidation Deposit 78
Figure 8.2 – Schematic Section of a High Sulphidation Deposit 78
Figure 10.1 – Location of INV 2016-2017 Drillholes in 3D Isometric View 84
Figure 10.2 – Location of 2016-2017 Loma Larga Drillholes in 3D Isometric View 85
Figure 11.1 – Gold Control Chart for 2013 Drilling – CRM SG14 99
Figure 11.2 – Gold Control Chart for 2013 Drilling – CRM SI15 100
Figure 11.3 – Gold Control Chart for 2013 Drilling – CRM SN16 100
Figure 11.4 – Gold Field Duplicate Scatterplot 102
Figure 11.5 – Gold Reject Duplicate Scatterplot 104
Figure 11.6 – Gold Pulp Duplicate Scatterplot 105
Figure 11.7 – Gold Pulp Replicate Scatterplot 107
Figure 11.8 – Gold Control Chart – CRM SG66 111
Figure 11.9 – Gold Control Chart – CRM SI15 112
Figure 11.10 – Gold Control Chart – CRM SJ63 112
Figure 11.11 – Gold Control Chart – CRM SN16 113

Figure 11.12 – Gold Control Chart – CRM SN60 113
Figure 11.13 – Gold Reject Duplicate Scatterplot 115
Figure 11.14 – Silver Reject Duplicate Scatterplot 116
Figure 11.15 – Copper Reject Duplicate Scatterplot 116
Figure 11.16 – Gold Pulp Duplicate Scatterplot 118
Figure 11.17 – Silver Pulp Duplicate Scatterplot 119
Figure 11.18 – Copper Pulp Duplicate Scatterplot 119
Figure 13.1 – Drillhole Locations and Ore Types 128
Figure 13.2 – LCT #1 Flowsheet 133
Figure 13.3 – Loma Larga 2017 Metallurgical Testing Program LCT #2 Flowsheet 135
Figure 13.4 – LCT #3 Flowsheet 137
Figure 13.5 – LCT #4 Flowsheet 141
Figure 13.6 – Jameson Dilution Test Grade-Recovery Curves 152
Figure 13.7 – Rougher Tests Schematics 158
Figure 13.8 – Rougher Results 160
Figure 13.9 – Cleaner Tests Schematics 161
Figure 13.10 – Cleaner Results 163
Figure 14.1 – 2018 versus 2016 Wireframe Domains in Plan View (3,600 m Level) 172
Figure 14.2 – 3D View of 2018 High-Grade (2.0 g/t Au) and Low-Grade (0.8 g/t Au) Wireframe Domains, Looking Northeast 173
Figure 14.3 – 3D View of 2018 High-Grade (2.0 g/t Au) and Low-Grade (0.8 g/t Au) Wireframe Domains, Looking West 173
Figure 14.4 – Histogram of Resource Density Samples 181
Figure 14.5 – Box Plot of Resource Density Samples by Domain 182
Figure 14.6 – High Gold Grade and Classification of Selected Block in the Low-Grade Lower Zone 191
Figure 14.7 – Capped Gold Composites and Blocks on 3,600 m Level 193
Figure 14.8 – Capped Silver Composites and Blocks on 3,600 m Level 194
Figure 14.9 – Capped Copper Composites and Blocks on 3,600 m Level 195
Figure 14.10 – High-Grade Main Zone Trend Plots and Histograms of Capped Gold Assays vs. Block Grades 196
Figure 14.11 – Low-Grade Main Zone Trend Plots and Histograms of Capped Gold Assays vs. Block Grades 197
Figure 14.12 – Plan View of Block Classification on the 3,600 m Level 199
Figure 14.13 – Cumulative Histogram of Distance to Nearest Sample Grouped by Class 200
Figure 14.14 – DSO Volumes Used for Underground Resource Reporting in Plan View 201
Figure 14.15 – Waterfall Chart of Tonnage Variance between 2020 and 2023 Measured + Indicated Mineral Resources Exclusive of Mineral Reserves 205
Figure 14.16 – Waterfall Chart of Total Contained Gold Variance between 2020 and 2023 Measured + Indicated Mineral Resources Exclusive of Mineral Reserves 206
Figure 14.17 – Waterfall Chart of Tonnage Variance between 2020 and 2023 Inferred Mineral Resources Exclusive of Mineral Reserves 207
Figure 14.18 – Waterfall Chart of Total Contained Gold Variance between 2020 and 2023 Inferred Mineral Resources Exclusive of Mineral Reserves 208
Figure 14.19 – 3D Isometric View of Drillholes with Sulphur Data 210
Figure 14.20 – Assay Histograms for Sulphur 212
Figure 14.21 – Sulphur Composites and Blocks on 3,600 m Level 216
Figure 15.1 – Transverse Stopes with Areas Marked for Longitudinal Access Investigation 220
Figure 15.2 – ELOS Stability Chart 221
Figure 15.3 – Stopes by Geotechnical Domain 222
Figure 15.4 – Grade Shell 224
Figure 15.5 – Number of Exposed Pastefill Surfaces per Stope 224

Figure 16.1 – Underground Mine Overview 227
Figure 16.2 – Stope Access Types 228
Figure 16.3 – Mining Zones Used for Design 229
Figure 16.4 – Schematic Sections of Transversal Long Hole Stope 231
Figure 16.5 – GMD Geotechnical Domain Map 234
Figure 16.6 – Distribution of Geotechnical Data per Domain 235
Figure 16.7 – Comparison of Rockmass Quality from Drillcore and Block Model Data 235
Figure 16.8 – Combined Geotechnical Block Model and Domain Data 236
Figure 16.9 – General In-Situ Stress Conditions 239
Figure 16.10 – Loma Larga Fault Model 241
Figure 16.11 – Matrix Summary of Stope Stability Assessment Results 244
Figure 16.12 – View of Stopes by Geotechnical Domain 245
Figure 16.13 – Geotechnical Drillhole Data Available Near the Decline 247
Figure 16.14 – Typical Rockbolt Pattern at Loma Larga 249
Figure 16.15 – Ground Support for 3- and 4-way Intersections 251
Figure 16.16 – General Cablebolt Pattern for Stope Support 251
Figure 16.17 – Faults to Be Observed Carefully During Mine Development 253
Figure 16.18 – Development According to Geotechnical Domains 254
Figure 16.19 – Stability Evaluation of the Central Western Raise 257
Figure 16.20 – Portal Location 261
Figure 16.21 – Secondary Drive Types Associated with Decline Development 262
Figure 16.22 – Main Ramp General Arrangement 263
Figure 16.23 – Ore-Drifts General Arrangement 264
Figure 16.24 – Plan View of Lower Level – Infrastructures 264
Figure 16.25 – Production Graph for FSUO Mine Plan 267
Figure 16.26 – Overview Looking North of the Loma Larga Ventsim Model with Raise Names 278
Figure 16.27 – Water Inflow Analysis 280
Figure 16.28 – Simulated Ventilation During Decline Development in the First Year 286
Figure 16.29 – Simulated Ventilation in H1 of Year -1 287
Figure 16.30 – Simulated Ventilation in the Second Half of Year -1 288
Figure 16.31 – Simulated Ventilation in Year 1 289
Figure 16.32 – Simulated Ventilation in Year 2 290
Figure 16.33 – Simulated Ventilation in Year 5 290
Figure 16.34 – Simulated Ventilation in Year 9 291
Figure 16.35 – Simulated Ventilation at End of Mine Life 292
Figure 16.36 – General Arrangement Drawing for Surface Exhaust Fans 293
Figure 16.37 – Refuge Stations Location Overview 297
Figure 16.38 – Dewatering Network 298
Figure 16.39 – Mine Dewatering Process Flow Diagram 299
Figure 16.40 – Electrical Single Line Diagram 300
Figure 16.41 – Underground Electrical Infrastructure Location 301
Figure 16.42 – Underground Paste Distribution System (Plan View) 305
Figure 16.43 – Underground Paste Distribution System (Elevation View) 305
Figure 17.1 – Simplified Overall Process Flow Diagram 312
Figure 17.2 – Metallurgical Data and Model for Copper Recovery and Concentrate Grade 319
Figure 17.3 – Gold Deportment to Copper Concentrate 321
Figure 18.1 – Overall Mine Site Area (right) and Los Pinos Camp Area (left) Layout 324
Figure 18.2 – Overall Site Layout Terrace 325

Figure 18.3 – Overall Off-Site (Los Pinos Camp) Layout 326
Figure 18.4 – On-Site Roads 328
Figure 18.5 – Tailings Dewatering and Paste Backfill 3D Model 331
Figure 18.6 – Tailings Required for Paste and Reporting to FTSF 331
Figure 18.7 – Tailings Dewatering and Paste Simplified Process Flow 333
Figure 18.8 – Tailings Dewatering and Paste Plant Location 334
Figure 18.9 – General Arrangement of Facilities around FTSF 335
Figure 18.10 – FTSF General Layout 340
Figure 18.11 – FTSF Embankment and Seepage Collection Outlet Cross Section 341
Figure 18.12 – Typical Liner System for FTSF Basin and Embankment inside Slopes 342
Figure 18.13 – FTSF Underdrain Collection System 343
Figure 18.14 – Conceptual Closure of FTSF 344
Figure 18.15 – Waste Rock Stockpile 346
Figure 18.16 – Ore Stockpile 347
Figure 18.17 – General Contact Water Management Schematic 349
Figure 18.18 – Water Balance Diagram 350
Figure 18.19 – Off-Site Infrastructure Domestic Water Intake 351
Figure 18.20 – Existing Los Pinos Infrastructure 358
Figure 18.21 – Los Pinos New Installations 359
Figure 21.1 – Project LOM Unit Operating Costs 412
Figure 22.1 – Revenue Profile over the LOM 422
Figure 22.2 – Proportion of Net Revenue Attributable to Concentrate (Left) and Metal (Right) 423
Figure 22.3 – Annual and Cumulative After-Tax Cash Flows over the Project LOM 427
Figure 22.4 – Sensitivity of After-Tax NPV to Metal Prices, Capex and Opex 431
Figure 22.5 – Sensitivity of After-Tax IRR to Metal Prices, Capex and Opex 431
Figure 22.6 – Variations in Pre-Tax and After Tax NPV with Discount Rate 432
Figure 24.1 – Project Schedule 436
Figure 24.2 – Project Management Organisation Chart 441

1 SUMMARY

This section should be read in conjunction with, and is subject to, the section entitled "Post-Effective Date Events" at the beginning of this Technical Report.

DPM Metals Inc. (formerly Dundee Precious Metals Inc.) (DPM or the Company) is a Canadian-based international gold mining company with headquarters in Toronto, Ontario, Canada and operations and projects located in Bulgaria, Bosnia and Herzegovina, Serbia, and Ecuador. DPM is listed on the Toronto Stock Exchange (TSX) (TSX:DPM) and the Australian Stock Exchange as a Foreign Exempt Listing (ASX:DPM) (ARBN: 689370894), with headquarters at 150 King Street West, Suite 902, Toronto, Ontario M5H 1J9. DPM acquired the Loma Larga development project (the Project) through its acquisition of INV Metals Inc. (INV) in July 2021. Subsequent to DPM's acquisition of INV, the Technical Report was reissued to DPM in 2021.

The Loma Larga Project is located in the country of Ecuador, about 30 km southwest of the city of Cuenca and 15 km north of the town of Girón. It comprises concessions covering an area of approximately 7,960 ha.

Loma Larga is a gold-copper-silver deposit classified as a high sulphidation epithermal system and alteration is characterised by multiphase injections of hydrothermal fluids strongly controlled by both structure and stratigraphy. The deposit is a flat lying to gently western dipping, north-south striking, cigar shaped body, which has a strike length of approximately 1,600 m north-south by 120 m to 400 m east-west and up to 60 m thick, beginning approximately 120 m below surface.

In 2022, DPM authorised the preparation of a Feasibility Study Update (2022 Update) for the Project coordinated by Canadian consulting firm G Mining Services (GMS), in an effort to improve project economics through various updates, including but not limited to updates to mine design, process plant and site infrastructure. The 2022 Technical Report performed the mine planning and design, infrastructure, capital estimation and economic evaluation, and coordinated the overall preparation of the 2022 Update.

In 2023, DPM retained DRA to coordinate the optimisation of various project components of the 2022 Update, which led to the preparation of a Feasibility Study Update Optimisation (FSUO), including but not limited to advancing definition of engineering design and development, constructability and operability, completion of trade-off studies, and obtaining updated pricing for equipment and contractor packages.

In 2025, DPM again retained DRA to coordinate the preparation an updated Capex, Opex and financial model based on the latest market conditions, based on the design and scope from the

FSUO, with some minor changes, leading to the preparation of 2025 Feasibility Study Update Technical Report (Technical Report or Report).

1.1 Mineral Processing and Metallurgical Testing

Three (3) separate and distinct phases of metallurgical testwork have been conducted on the Loma Larga deposit. The first two phases of testwork were conducted in 2006 and 2014 and are referred to as historical work. The first program was managed by IAMGOLD in 2006 and the second program managed by RPA (now SLR) in 2014 and was used as the basis of design for the Pre-Feasibility Study (PFS). The third program (2017 metallurgical testwork program) was managed by INV with advisory input from DRA and Promet101. The 2017-2018 metallurgical program formed the basis of the 2020 Loma Larga Feasibility Study (FS).

A significant amount of testwork was conducted to develop a robust and fit for purpose flowsheet for the development of the Loma Larga process plant design. The merits of sequential and bulk flotation flowsheets were examined during the program and analysed. Sufficient testwork has been conducted to support the basis of the FS.

The metallurgical programs concluded that a sequential flotation flowsheet for the recovery of separate gold bearing copper and pyrite concentrates is the preferred processing route. From the testwork, grade and recovery relationships for the copper concentrate and an understanding of the gold-pyrite concentrate recoveries was determined.

In 2022 a metallurgical program was conducted using low-grade composites (COMP1 was used for metallurgical testing, and COMP2 was prepared for tailings testwork) from available drillhole samples collected in 2017.

The 2022 program is considered confirmatory in nature with respect to the flowsheet selected and assessment of process opportunities. Therefore, these tests were considered for reference purposes only and as support to the 2017-2018 program and associated design. Due to the limited quantity of the ore sample used for the 2022 metallurgical program, the 2017-2018 metallurgical program results should still be considered as the basis for the FS.

It must be noted that the observed metallurgical response of the ore used in the 2022 program confirms the ore response and results of the 2017-2018 metallurgical program.

The objectives of the 2022 test program were as follows:

- Review the results from of the 2017 testwork.
- Assess other opportunities with regards to:
  - Primary grind size impact on ore flotation response.
  - Rougher concentrate regrind requirements.

- Number of cleaning stages required for gold-copper, and pyrite flotation circuits.
- Use of Woodgrove, Jameson, and Hydrofloat technologies.
- Evaluation of tailings settling and filtration properties.

- Generate a sample of tailings for Paterson & Cooke tailings testing.

The objectives of the program have been met and have opened up a potential for several process opportunities with regards to the coarser primary grind size (125 µm P_{80}), number of the flotation cleaning stages required, and applicability of the advanced flotation technologies. Use of the Woodgrove, and Jameson flotation cell technologies were found promising.

Considering the attractive Capex for the Jameson cell, in July-August 2023 a 122 kg LOM composite sample was tested at Base Metallurgical Laboratories (BML). The sample was composited using historical drill cores stored on site. The dilution test protocols were applied considering the available quantity of ore. Rougher and cleaner conditions were investigated and compared to the conventional flotation results. It was found that use of the Jameson technology benefits the rougher circuit with improved recoveries and allows for the production of a clean concentrate due to froth washing. Cleaner tests suggested the same performance as it was observed for the conventional flotation. Considering the Capex advantages determined during the process trade off study conducted in April 2023, the use of Jameson technology is recommended for both copper and pyrite flotation. Some optimisation of the rougher and cleaner conditions as well as variability tests are recommended as soon as sufficient fresh sample becomes available.

1.2 Geology and Mineral Resources

SLR estimated Mineral Resources for the Loma Larga Project using all drillhole data available as of October 31, 2018. This Mineral Resource Estimate was previously updated October 31, 2018, and reported in the 2019 Technical Report (DRA, 2019) and readdressed to DPM in 2021 (DRA, 2021). No additional drilling has been completed on the Project since the previous estimate. The current Mineral Resource estimate incorporates updated metal prices and mining costs, is based on an underground mining scenario, and is reported exclusive of Mineral Reserves. In order to ensure that the resources have sufficient spatial continuity, the Mineral Resource estimate was reported within underground resource reporting shapes generated in Deswik Stope Optimizer software, satisfying the minimum mining size, continuity criteria, and using an NSR cut-off value of US$65/t. NSR cut-off values for the Mineral Resources are based on a gold price of US$1,700 per ounce, silver price of US$20.00 per ounce, and copper price of US$3.75 per pound. Mineral Resources exclusive of Mineral Reserves, as of September 29, 2023, are summarised in Table 1.1. The Mineral Resources conform to Canadian Institute of Mining, Metallurgy, Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) as incorporated by reference in NI 43-101.

Table 1.1 – Mineral Resource Estimate Summary, Effective as of September 29, 2023

Resource Classification	Tonnage (Mt)	Au (g/t)	Contained Au (Moz)	Ag (g/t)	Contained Ag (Moz)	Cu (%)	Contained Cu (Mlb)
Measured	0.3	4.80	0.05	20.8	0.2	0.23	1.7
Indicated	5.7	2.79	0.51	21.9	4.0	0.16	19.8
Measured + Indicated	**6.0**	**2.90**	**0.56**	**21.8**	**4.2**	**0.16**	**21.5**
Inferred	2.2	2.54	0.18	37.0	2.7	0.19	9.3

Notes:

- Mineral Resources have been estimated by Katharine Masun, M.Sc., MSA, P. Geo., of SLR, an independent Qualified Person as defined by NI 43-101.
- CIM (2014) definitions were followed for Mineral Resources.
- The Mineral Resources have been reported within underground reporting shapes generated with Deswik Stope Optimizer (DSO) using a net smelter return (NSR) cut-off value of US$65/t.
- Mineral Resources are estimated using a long-term gold price of US$1,700 per ounce, silver price of US$20 per ounce, and copper price of US$3.75 per pound.
- Mineral Resources are exclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Average bulk density is 2.71 t/m³.
- Numbers may not sum precisely due to rounding.

Three-dimensional (3D) grade shell wireframes were constructed at 2.0 g/t Au (High-Grade Zone) and 0.8 g/t Au (Low-Grade Zone). The QP used cross sections, long sections, and plan views to interpret and validate the wireframes.

The Loma Larga High-Grade Zone comprises two (2) mineralised zones: High-Grade Main Zone and High-Grade Upper Zone. The Low-Grade Zone comprises two (2) domains: Low-Grade Main Zone wireframe domain, which encompasses the High-Grade Main Zone, and Low-Grade Lower Zone, which lies below the Low-Grade Main Zone.

Variography was performed on the 2.0 m Au, Ag, Cu, S, and density composites from the High-Grade Main Zone and Low-Grade Main Zone. Block grade interpolation was carried out using Ordinary Kriging (OK) and the gold grade shell wireframe models were used to constrain the grade interpolations. A soft boundary was used between the Low- and High-Grade Main Zones for density block interpolation.

The polymetallic sulphide mineralisation at the Loma Larga deposit contains significant values of Au, Ag, and Cu. Therefore, original assays were converted into NSR values (US$ per tonne). The NSR values account for parameters such as metal price, metallurgical recoveries, smelter terms and refining charges, and transportation costs. For the purposes of developing an NSR cut-off value for an underground operation, a total operating cost of US$65/t milled was assumed, which includes mining, processing, and general and administrative (G&A) expenses.

Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate. The QP understands that DPM is considering all its options to rectify the revocation of the environmental license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project, including the Mineral Resource Estimate. See "Post-Effective Date Events" at the beginning of this Report.

Detailed conclusions related to the Project and deposit geology are contained in Sections 7 and 26.

1.3 Mineral Reserve Estimate

The Mineral Reserve Estimate for Loma Larga consists of selected portions of the indicated and measured resource that are above a US$75.00/tonne Net Smelter Return (NSR) cut-off value. This value was applied when generating the stoping shapes. The mineral reserves for Loma Larga are estimated at 12,585,043 tonnes of recoverable and diluted ore grading 4.70 g/t Au, 28.56 g/t Ag, and 0.29% Cu. The mineral reserves are comprised of 22% in proven category (2,721,800 tonnes grading 6.83 g/t Au, 33.23 g/t Ag and 0.43% Cu) and 78% in probable category (9,863,200 grading 4.12 g/t Au, 27.35 g/t Ag and 0.25% Cu). Reserves are inclusive of dilution and ore loss. The latest NSR value has been added to the DPM-provided block model using Datamine software by Global Mine Design Ltd. (GMD) and used for further work in Deswik.

The Mineral Reserves are based on the mining schedule presented in Section 16. The mineral reserves are the stopes and development tonnage (including material above the cut-off and internal low-grade dilution) to which dilution and recovery factors explained below are added.

Table 1.2 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023

Ore Category	Tonne (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)	Cu Grade (%)	Contained Cu (Mlb)
Proven	2.72	6.83	0.60	33.23	2.91	0.43	25.58
Probable	9.86	4.12	1.31	27.35	8.67	0.25	54.39
Proven and Probable	**12.59**	**4.70**	**1.90**	**28.56**	**11.58**	**0.29**	**79.96**

1. Mineral Reserve is estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (Nov 29, 2019) and CIM Definition Standards for Mineral Resources & Mineral Reserves, May 19, 2014.
2. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the crusher.
3. Mineral Reserves account for unplanned mining dilution and ore loss that varies by orebody dimension and geotechnical properties per mining block area.
4. Mineral Reserves include long hole stopes and development in ore.
5. Mineral Reserves are reported at an NSR cut-off value of US$75/t for long hole stopes and US$55/t for development in ore.
6. Mineral Reserves are estimated using average gold price of US$1,500 per ounce, silver price of US$17.00 per ounce, and copper price of US$3.25 per pound.
7. Average bulk density is 2.71 t/m^3.
8. Numbers may not sum precisely due to rounding.
9. Metallurgical recoveries are not included in the contained values calculation.
10. The mine design and schedule for the Mineral Reserve Estimation was completed under the supervision of Dr. Lewis Meyer, CEng, QMR, FIMMM Associate Principal Engineer at Global Mine Design Ltd., who is a QP as defined under NI 43-101.
11. Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

Mineral Reserves are inclusive of dilution and ore loss. GMD added the latest NSR value to the block model supplied by DPM using Datamine software. GMD conducted further work in Deswik software. However, GMD relies on information as presented in this Technical Report in relation to metallurgical, infrastructure, permitting, legal, title, taxation, socio-economic, marketing or political factors.

Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. The QP understands that DPM is considering all its options to rectify the revocation of the environmental license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project, including the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

Net changes in tonnes and contained metals from the 2020 to the 2023 Mineral Reserves estimate show a decrease of 1,341,4322 in tonnage (1.3 Mt), a reduction of 295,910 ounces of gold (0.3 Moz Au), a reduction of 1,690,402 ounces of silver (1.7 Moz Ag) and a reduction of 8.08 million pounds of copper (8.08 Mlb Cu). The corresponding percentage changes are a 9.6% decrease in tonnes,

and 13.5% reduction in metal content gold, 12.7% in metal content silver and 9.2% in metal contained copper.

1.4 Mining

1.4.1 Mining Method

The Loma Larga deposit, located in the Western Cordillera mountains at an elevation of 3,800 m in Azuay Province, Ecuador, will be developed into an underground operation permitted to mine 3,400 tonnes per day (t/d), which is equivalent to 1.2 million tonnes per year (Mtpa). The mine will be accessed via a surface portal and main decline. There is a total of four (4) ramps connected to the decline and five (5) interlevel ramps linking to five (5) production levels (two (2) main levels and three (3) smaller levels). Typical vertical spacing between levels is 20 m.

GMD has fully revised the mine design and schedule for this Project phase, based upon new economic factors and an extensive geotechnical evaluation (GMD, 2023). The selected mining method is long-hole open stoping, which is a method well suited to the orebody geometry and is known for its flexibility and ability to minimise underground development. Most of the stopes will be mined overhand (bottom-up) in a continuous mining front and backfilled using cemented paste backfill (CPB); however, there are a small number of stopes to be mined under CPB for optimum scheduling. Based on the grade, geometry, and continuity of the orebody, as well as development access efficiency, the design used a combination of stope accesses: 60% of the stopes have transverse access and 40% of the stopes have longitudinal access.

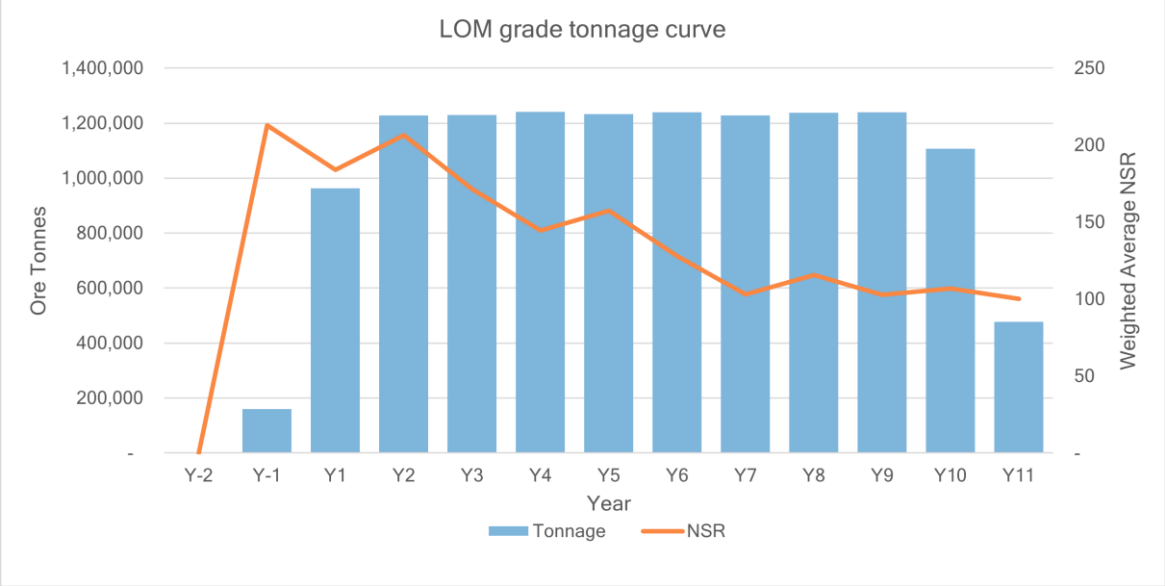
The orebody was split into three (3) main zones: North, Central, and South. Each orebody zone was split further into three (3) mining zones. Dividing the orebody in this manner enables scheduling to focus on mining high-grade material as soon as possible. Combining transverse ore development for maximum sequence flexibility, with longitudinal ore development where geotechnically feasible, has minimised development in waste. Footwall development has also been placed as close as possible to the orebody while maintaining geotechnical rules relating to sequencing and pillar offsets.

The stopes will be formed using long-hole drilling methods, which make use of fan-drilling to maximise ore recovery. The slot will be formed by a raiseboring head. A typical production hole will be loaded with emulsion and two (2) boosters and detonators. Mucking will be performed using 17 tonne Load Haul Dump (LHD) machines remotely controlled. Ore material will be transported to the nearest stockpile (remuck). The LHDs will transfer ore from the stockpile into 51-tonne trucks for transport to the surface ore stockpile.

1.4.2 MINE PLAN

The production graph from the 2023 FSUO mine plan is shown in Figure 1.1 and is described below.

Figure 1.1 – Production Graph from 2023 FSUO Mine Plan



Source: Global Mine Design, 2023

Excavation of the portal and development of the main decline commences the mine plan in Year -2. Development will reach the ore during Q2 of Year -1 and stope production will begin during Q3 of Year -1. The commercial production period is planned to start from Q2 of Year 1 when the mine reaches an ore production rate of 2,000 t/d (60% of nameplate capacity). During the ramp-up period, the higher-grade zones are targeted, and ore tonnes produced will increase to the steady-state production rate of 3,400 t/d by Year 2. The higher production rate will be maintained throughout the LOM until Year 10, when production begins ramping down due to a reduced number of production areas at the end of the mine life.

1.4.3 VENTILATION

The mine ventilation system is a mechanised exhaust system, which uses four ventilation raises in different combinations during the mine life. A forced air ventilation circuit will be required during decline development until the second half of Year -1 when the first ventilation raise is mined. The other raises will be commissioned in Year 1, Year 2, and Year 5. When in full production, two (2) sets of two (2) fans in parallel will be used (one set per raise), which generate a negative pressure underground allowing for fresh air to be drawn through the decline and fresh air raises to supply the required volumetric airflow requirements.

The Loma Larga Project is to use a variety of diesel-powered underground equipment during the life of mine (LOM). The ventilation system is designed to mitigate ventilation risks posed by diesel equipment, such as diesel particulate matter.

1.4.4 Underground Mining Fleet

The underground mining equipment fleet was chosen based on the evaluation of productivity needed to achieve the daily production rate 3,400 t/d in steady-state production (achieved in Year 2), 362.5 days per year, including a mechanical availability of 89% for the trucks, 82% for scoops and 85% for all the other pieces of equipment.

Fleet productivities have been based on first principal calculations, benchmarking, supplier recommendations and practical experience. Equipment requirements were based on the estimated required operating hours in each period and the number of units of each piece of equipment needed to meet those hours.

1.4.5 Mine Infrastructure

Underground mine infrastructure includes the tunnel network, a small underground maintenance bay for minor repairs and daily checks, a fuel bay, mine dewatering system infrastructure including main pump station, ore and waste stockpiles, ventilation raises and access drives, main substation and refuge chamber bays. On surface, there is an explosive magazine, workshop, office, and a dry building. The processing facilities are covered separately.

1.4.6 Paste Backfill

The proposed mining method requires systematic filling of all stope voids for ground stability and extraction of adjacent ore. Cemented Paste Backfill (CPB), Cemented Rock Fill (CRF) and Rock Fill (RF) will be placed within the mine voids to manage closure, stabilise stope spans, and minimise waste haulage to surface. The principal backfilling method will be CPB made of filtered tailings, water, and cement in varying proportions.

Paste produced in the paste plant will be pumped overland and delivered underground via two (2) surface boreholes located directly above the orebody. Once underground, distribution of the paste to the various working areas will be accomplished by main trunk line(s) to the level piping and eventually to the stope piping.

1.5 Recovery Methods

The process plant design for the Project is based on a robust metallurgical flowsheet to treat ore to gold/copper and pyrite concentrates. The flowsheet is based on metallurgical testwork, industry standards, information provided by DPM, DRA, vendors and fit for purpose unit operations.

The Process Design Criteria (PDC) to process a nominal 3,000 t/d of ROM ore from an underground mine (increasing to 3,400 t/d in Year 2).

The process will consist of comminution, gold-copper, and pyrite flotation circuits, concentrate dewatering, and load out circuits. The flotation circuit will use Jameson flotation cell technology which provides a benefit of reduced Capex through a reduced plant footprint versus conventional flotation technology and does not require flotation blowers for air supply.

Pyrite tailings will be dewatered to produce a filtered tailings product for use at the paste plant for the underground mine or stored on surface as filtered tailings.

The crushing and grinding circuit design is based upon the design throughput requirements and ore competency and hardness. The SAG mill sizing is based on achieving the grind size P_{80} of 75 µm required to obtain the required flotation performance as well as the outcomes of the metallurgical testwork and mill simulations.

The design of the flotation circuit is based upon the testwork results of the 2017-2018 SGS testwork program. The 2022 and 2023 testwork programs were designed to confirm the results from of the 2017-2018 testwork and assess a number of the opportunities detailed in Section 13. The results of the 2017-2018 locked cycle testwork provided the basis for recovery and grade calculations, and residence times.

The concentrates are dewatered using conventional plate and frame pressure filtration. The filtration circuit design is based on standard design practices for the concentrate dewatering as no testwork data is yet available. Tailings dewatering will occur in the paste plant described in Section 18.

1.6 Environmental Studies, Permitting and Social or Community Impact

The Project is located in the province of Azuay, in the western Andes Mountains, 30 km southwest of the city of Cuenca and approximately 15 km north of the town of Girón. The concession area of the Project is located in the cantons of Cuenca, Girón and San Fernando. The population in the Project's area of influence is mainly rural (excluding the city of Cuenca). The results of the social baseline study show that the number of people living below the poverty line is above the national average. Most of the people work in agriculture and cattle raising.

The Project site consists of desert plains and rugged valleys, mainly formed by glaciers, with an altitude ranging from 3,500 m ASL to 3,960 m ASL. Vegetation is sparse and typical of the Andean

region above tree line. Much of the property is covered by Andean *páramo*, a type of moorland vegetation. There are stands of pine on hillsides adjacent to the concessions, most of which were planted as part of a forestation project. Terrestrial fauna groups for which baseline data have been collected include mammals, amphibians and reptiles, birds, and insects. Aquatic faunal groups for which baseline data have been collected include fish and macroinvertebrates.

An Environmental Impact Assessment (EIA) was prepared for the Project and submitted to the Ecuadorian government in April 2021 to initiate the review and approval process. The EIA describes the physical, biological, and socio-economic baseline conditions of the Project area. The EIA specified the area of influence of the Project (i.e., the geographical area potentially affected), how various Project activities could affect the physical, biological, and socio-economic aspects within the area of influence, and the potential impacts on the socio-environmental components. Approval of the EIA from a technical perspective was granted in April 2024.

The Environmental Management Plan (EMP) for the Project was developed as part of the EIA. The EMP outlines actions required to prevent, mitigate, control, compensate, correct, and repair the potential adverse impacts resulting from the Project activities, and presents information regarding potential positive impacts of the Project activities. Additionally, the EMP describes the Project's Monitoring Plan, which was developed to track the effectiveness of the mitigation measures proposed in the EIA. The EMP is comprised of nine (9) sub-plans, including the environmental monitoring plan for the Project.

Prior to initiating mining activities, all titleholders must obtain an environmental license, a certificate of no impact on water sources, and the order of priority of access to water. All titleholders must also make an affidavit swearing that there will be no damage to infrastructure or cultural vestiges, among other damages. One (1) year after obtaining the environmental license, the mining companies must carry out an environmental compliance audit that allows the State to monitor, supervise, and verify compliance with the environmental management plans and applicable environmental regulations. Subsequently, environmental compliance audits will be carried out every three (3) years.

The Environmental License resolution for exploitation activities for the Project was issued by the Ministry of Environment, Water, and Ecological Transition (*Ministerio del Ambiente, Agua y Transición Ecológica de la República del Ecuador* – MAATE for its acronym in Spanish) on June 23, 2025 and subsequently revoked by the Government of Ecuador on October 3, 2025.

During the operations phase, mining companies must submit quarterly environmental monitoring and follow-up reports for the exploitation phase, and annual environmental management reports that contain information supporting compliance with the EMP and the applicable legislation.

Due to concerns related to contamination that could be caused by the Project, a legal action was filed in the court of Azuay by members of the communities in 2022 resulting in a court ruling on August 28, 2023 requesting four (4) additional activities to for the Project to proceed to the next

phase. The ruling obliged MAATE and Ministry of Energy and Mines to perform the following activities: i) biotic study in the Quimsacocha national recreation area; ii) water study in the same location; iii) environmental consultation; and iv) Indigenous Consultation in the community of Escaleras. The biotic study, the water study, and the environmental consultation were already finalised and the final reports submitted to the provincial court of Azuay in 2024. The Ministry of Energy and Mines officially notified the Azuay Provincial Court of Justice of the completion of the Indigenous Consultation in the community of Escaleras in May 2025.

According to data obtained in the 2024 social baseline survey, the level of poverty due to unmet basic needs in the communities within the project's area of influence averages 60%. In this socioeconomic context, the population has expectations regarding opportunities associated to local development in areas such as direct employment, supply, and, in general, improvement in quality of life during the construction and operation phases of the project. In addition, it should be noted that migration indicators in the area of influence are, on average, above 33%, which affects community dynamics.

DPM has indicated that it maintains a good relationship with the communities located within the social area of influence. The Company has been implementing its social action and development plan by supporting activities associated with economic production activities, education, health, infrastructure, culture, and recreation.

In Ecuador the mining license holder must provide a bank guarantee or take out an insurance policy, which must remain in force until the total closure of mining operations, to guarantee compliance with the EMP.

1.7 Project Infrastructure

The Loma Larga Mine site is located close to existing infrastructure (at the town of Girón, a significant regional population centre).

The existing access road from the town of San Gerardo to the mine site will be widened and the roadbed upgraded (as part of a separate project). A new 17 km 69 kV power transmission line (power provided by the national grid) will be required to cover the Project power requirements. A new 16.5 MW power plant will be constructed to cover any power outages.

Surface infrastructure (e.g., camp, administrative buildings, water treatment plant) will be required. Design will consider modularised and prefabricated buildings to minimise site installation time.

Tailings production and deposition methods were evaluated, and filtered tailings was the technology chosen as the most appropriate for the site. Approximately half of the tailings produced will be stored on surface in the filtered tailings storage facility (FTSF), while the remainder will be used as cemented paste backfill for the underground mine. Geochemical characterisation of filtered tailings

suggests that the material is potentially acid generating (PAG) and has the potential to leach constituents of potential concern including metals. The FTSF will consist of a ring-shaped downstream expanded embankment that will be fully lined with HDPE geomembrane to avoid contact of tailings material with the environment. Any water draining from the facility, along with contact water, will be collected and treated to meet Local, International and Industry standards prior to release, during operations and closure. The facility will be designed to meet international and local guidelines including the Global Industry Standard on Tailings Management (GISTM).

1.8 Capital and Operating Costs

1.8.1 Capital Cost Estimate

The Capital Cost Estimate (Capex) consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also included.

DRA developed the Capex for the process plant and plant infrastructure for the Project scope described in this Report. External Capex inputs were received as follows:

- GMD prepared the underground mining costs for the Initial and Sustaining Capex.
- NewFields prepared the construction quantities for the Filtered Tailings Storage Facility and DRA applied rates to complete the Capex.
- Paterson and Cooke prepared the cost estimate for the Paste Backfill system, where DRA only estimated the bulk earthworks, concrete, and structural steel.
- DPM provided Owner's General and Administration Costs, Taxes, Duties, and Closure costs.

All costs are expressed in United States Dollars (US$) and are based on Q2 2025 pricing (or Q3 2023 pricing with escalation added where noted). The Capex is deemed to have an accuracy of ±15% and was prepared in accordance with the Association for the Advancement of Cost Engineering (AACE) Class 3 estimating standard.

Table 1.3 presents a summary of the initial Capex and Sustaining Capex as distributed over the LOM, separately indicated from the initial Capex. All costs related to the area within DRA's scope unless otherwise noted. Owner's costs, its contingencies and risk amounts have been included in this Capex.

Table 1.3 – Capital Cost Summary (US$ '000)

Description	Initial Capex	Sustaining Capex	Total
Direct			
1000-Mine General	9,449	-	9,449
2000-Underground Mine	110,692	41,544	152,236
3000-Ore Handling	15,663	-	15,663
4000-Processing Plant	94,252	1,300	95,552
5000-Filtered Tailings / Water Treatment Facilities	61,224	18,977	80,201
6000-On-Site Infrastructure, Site-Wide General	48,945	-	48,945
7000-Off-Site Infrastructure	63,133	4,558	67,691
Subtotal	*403,358*	*66,379*	*469,737*
Indirect			
8000-Operational Readiness and Owner's Costs	66,383	-	66,383
9000-Indirect Costs	75,501	2,014	77,515
8700- Closure and Rehabilitation[2]	-	25,211	25,211
Subtotal	*141,884*	*27,225*	*169,109*
Contingency			
9900-Project Contingency	47,678	-	47,678
9930-Contingency - Sustaining Capital	-	6,638	6,638
9940-Contingency - Closure and Rehabilitation	-	2,521	2,521
Subtotal	*47,676*	*9,159*	*56,837*
Total Major Area Capex	**592,920**	**102,763**	**695,683**

Notes:
1. Initial capital and sustaining capital estimates include import duties.
2. Closure costs do not include non-refundable VAT of $4.2 million.
3. Numbers may not sum precisely due to rounding.

1.8.2 Operating Cost Estimate

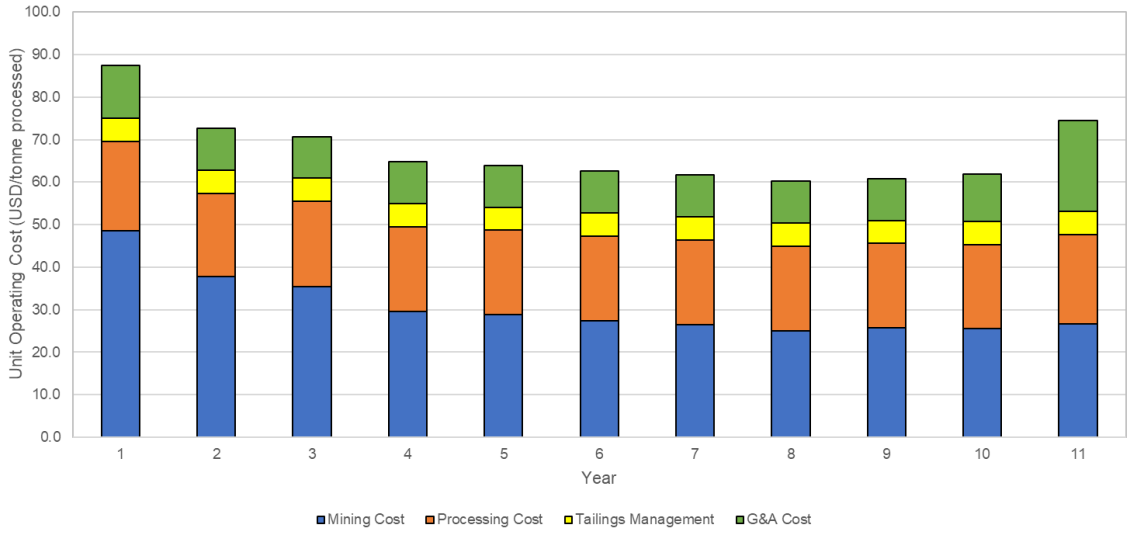
A summary of the overall LOM project operating cost estimate (Opex) is presented in Table 1.4 and the summary of the unit operating costs over life-of-mine is presented in Figure 1.2. The costs presented exclude pre-production operating cost allowances for mining, process and G&A which are considered in the capital cost estimate. Labour rates for the study are provided by DPM in 2025. Direct employment during operations will total approximately 484 people including mine, concentrator, tailings, maintenance, management, and infrastructure.

Table 1.4 – Project LOM Operating Costs by Major Area

Major Project Area	LOM Total (US$ M)	LOM Unit Cost (US$/t)
Mining	385	30.57
Processing	251	19.98
Filtered Tailings Management	68	5.42
G&A	135	10.70
Total Cost	**839**	**66.67**

Numbers may not sum precisely due to rounding.

Figure 1.2 – Project LOM Unit Operating Costs



Source: DRA, 2025

1.9 Economic Analysis

The results of the economic analysis contain forward-looking information as defined by Canadian securities legislation. The results rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here.

The economic analysis is based on the Discounted Cash Flow (DCF) method on a pre-tax and after-tax basis. The key metrics determined in the analysis are the Net Present Value (NPV) at a discount rate of 5%, the Internal Rate of Return (IRR), and the Payback Period. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis with a reference date aligned with a Final Investment Decision (FID) on the Project.

The cash flow model uses inputs from all elements of the Project to provide a comprehensive financial projection for the entire Project on an annual basis. All prices and costs are in US dollars and accurate as of Q2 2025. No provisions have been made for the effects of inflation. Current Ecuadorian tax regulations were used to assess corporate tax liabilities. DPM retained *EY Addvalue Asesores Cia. Ltda.* to review tax calculations for the after-tax economic evaluation of the Project. Table 1.5 summarises the key parameters utilised in the financial analysis.

Table 1.5 – Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Commodity Price		
Gold	US$ per oz	1,900.00
Silver	US$ per oz	27.50
Copper	US$ per lb	4.00
Project Parameters		
Discount Rate	%	5.0
Mine Life	years	11
Mineral Reserve (LOM)	Mtpa	12.6
Grade Mined (LOM average)		
Gold	g/t Au	4.7
Silver	g/t Ag	28.6
Copper	% Cu	0.29
Annual Mill Throughput	Mtpa	1.241
Recovery (LOM average)		
Gold	%	89.7
Silver	%	94.8
Copper	%	95.1
Payability (LOM average)		
Gold	%	96.2
Silver	%	81.8
Copper	%	80.5
Total Payable Metal Production (LOM)		
Gold	Moz	1.6
Silver	Moz	9.0
Copper	kt	27.8
Gold Equivalent[5]	Moz. Au eq.	1.9

Description	Unit	Value
Average Annual Payable Metal Production (LOM)		
Gold	oz	149,170
Silver	oz	814,340
Copper	tonnes	2,527
Gold Equivalent[5]	oz Au eq.	172,685
Government Royalty (NSR) [1]	%	5.0
Capital Cost Estimate [2]		
Initial Capital	US$ M	593
Sustaining Capital (LOM)	US$ M	75
Closure	US$ M	28
LOM Unit Operating Costs (LOM Average) [3]		
Mining	US$/t ore processed	30.6
Processing	US$/t ore processed	20.0
Paste Backfill	US$/t ore processed	5.4
General & Administrative	US$/t ore processed	10.7
Total Opex	US$/t ore processed	***66.7***
Cost Metrics (LOM Average) [4]		
Cash Cost per Tonne of Ore Processed	US$/t ore processed	99
Cash Cost per Ounce of Gold Sold	US$/oz payable gold	809
All-in Sustaining Cost per Ounce of Gold Sold	US$/oz payable gold	873

Notes:

1. The Government Royalty is currently under negotiation between DPM and the Ecuadorian Government. A flat NSR royalty rate has been used in this analysis.
2. Capex is inclusive of applicable import duties and taxes.
3. Unit Opex numbers are exclusive of applicable import duties, taxes and VAT.
4. Cash Cost per Tonne of Ore Processed, Cash Cost per Ounce of Gold Sold, and All-in Sustaining Cost per Ounce of Gold Sold are non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS Accounting Standards and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS. Refer to the “Non-GAAP Financial Measures” in Section 2.7 of this Report for more information, including a detailed description of each of these measures.
5. Gold equivalent calculations are based on the metal prices selected for the project: gold price of $1,900/oz, silver price of $27.50/oz, and copper price of $4.00/lb.
6. Numbers may not sum precisely due to rounding.

Based on the assumptions presented in this Report, the results of the economic analysis indicate a positive after-tax NPV of US$488 M at a discount rate of 5% and an after-tax IRR of 18.1%. A summary of the key financial metrics is presented in Table 1.6.

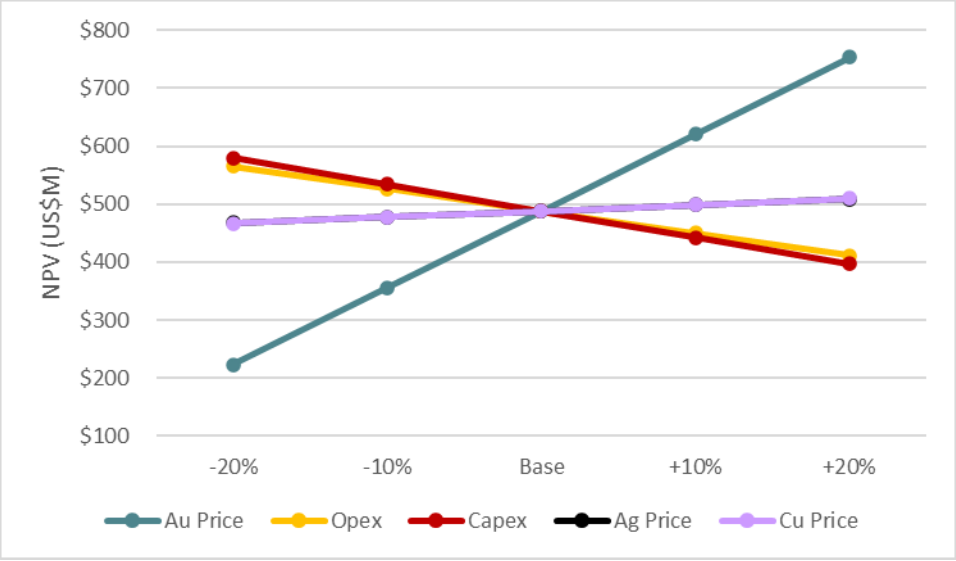
Table 1.6 – Summary of Project Financial Metrics

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	US$ M	627	488
IRR	%	21.3	18.1
Payback Period[1]	Years	2.9	3.4

Notes:
1. The Payback Period provided here is from the start of mill operation.
2. Numbers may not sum precisely due to rounding.

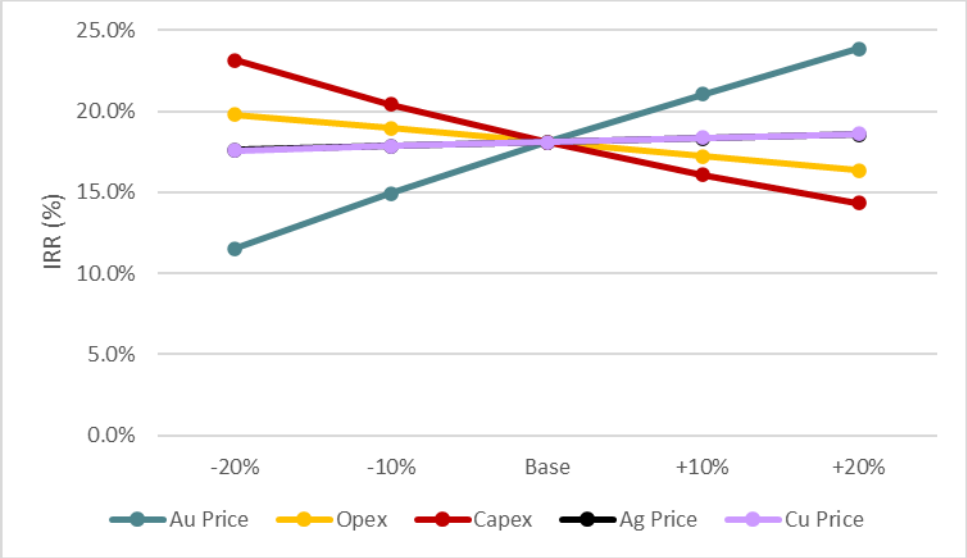
A sensitivity analysis was carried out to assess the impact of variations in metal price, Capex, and Opex on the NPV and IRR. The after-tax results of the sensitivity analysis are shown in Figures 1.3 and 1.4. The NPV and IRR of the Project are most sensitive to variations in the gold price followed by Capex and Opex. The Project retains a positive NPV at the lower limit of the price interval tested. The IRR is more sensitive to variations in Capex than Opex, as evidenced by the steeper slope of the Capex curve in Figure 1.4. The NPV appears to be equally sensitive to variations in Capex and Opex. The Project economics display limited sensitivity to the prices of silver and copper as these metals make up a relatively small proportion of the overall revenue. Overall, within the limits of accuracy of the cost estimates in this study, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs up to 20%, when taken individually.

Figure 1.3 – Sensitivity of After-Tax NPV to Metal Prices, Capex and Opex



Source: DRA, 2025

Figure 1.4 – Sensitivity of After-Tax IRR to Metal Prices, Capex and Opex



Source: DRA, 2025

1.10 Other Relevant Data and Information

The DPM Corporate office in Toronto will remain the executive centre for global oversight of all mining projects, including the Loma Larga Project. DPM has established an Ecuador registered company with the main office in Quito, Ecuador to oversee multiple interests including Loma Larga. A project management office located in Cuenca, Ecuador, will function as DPM Ecuador (DPME) project office.

An engineering services company in Canada will be contracted to provide engineering and procurement services during the detailed engineering phase of the Project. Process equipment and bulk materials will be purchased and competitively sourced with an emphasis placed on bulk materials sourced locally from Ecuador. The DPME Owner's Construction Management (CM) team will accept international shipments at the Ecuador port of entry and provide in-country transport to the mine site. Where feasible, the port at Machala will be the designated as the primary port of entry.

Procurement and contracting for the Project will be directed by DPME to optimise materials and services from Ecuador within Azuay Province, including Girón, and San Gerardo. Other local communities will be researched to identify local suppliers and contractors as much as possible.

DPME will oversee the detailed engineering to ensure design integrity, conformance to the process design and functional specification and to integrate the overall scope into a successfully constructed, commissioned and operating facility.

Contracts administration will be set-up in the starting months for major contracts bidding, and for purchase of long delivery equipment. This Engineering and Procurement (EP) team will develop the

major contracts technical and commercial templates. A DPME contracts administration team has been established in-country to coordinate the local contracts and will receive the packages contracts management from the EP services provider for field administration. Where Spanish language documents are required, the EP services provider will provide those services from their South American affiliate.

DPME CM personnel will mobilise in time to support construction activities and ramp up in accordance with the construction schedule. Emphasis will be placed to find and contract a qualified underground mining contractor in Ecuador to perform the portal and decline construction. An operations camp will be provided at the Project site for early use by selected contractors. All construction contractors will provide their own lodging and catering services off site. And in the later construction phases, as the operations readiness team ramps up their training, contractors will provide their own lodging and associated services from the several nearby towns and the city of Cuenca.

The major activities in the project are illustrated in Figure 1.5.

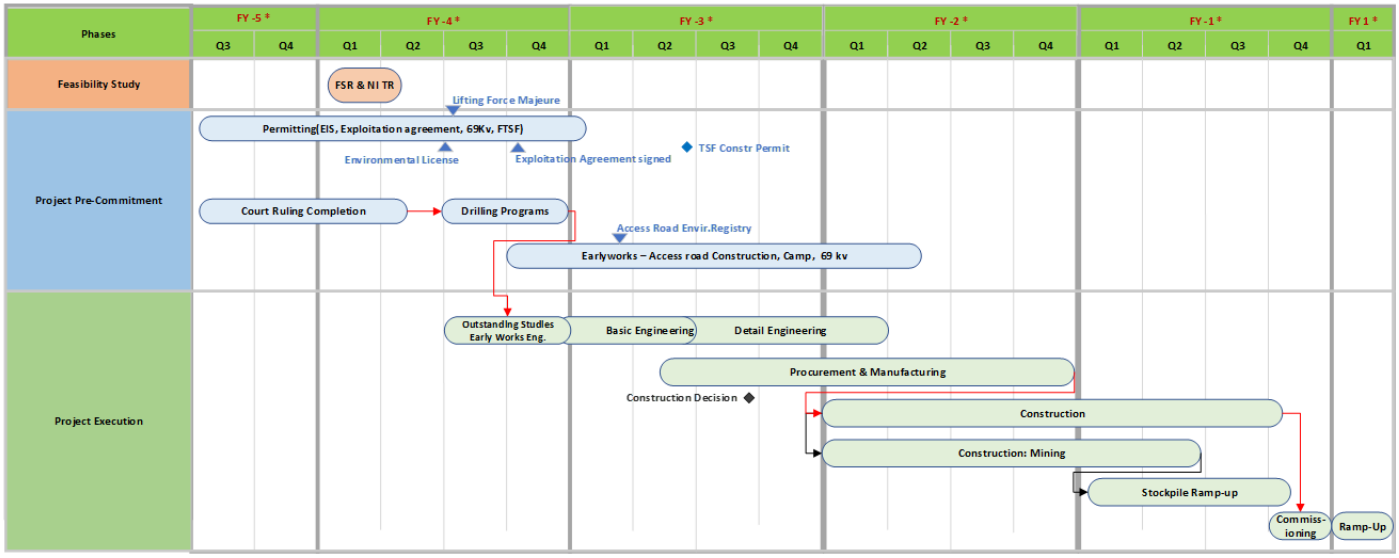
Limited activities will occur ahead of this in the pre-commitment phase. The Project critical path, at the time of preparation of the FS, runs through the Exploitation agreement, FTSF permit, decline and mine development.

In preparing this Technical Report, the risk management process was continued. Project risk review sessions were also concluded. The most significant risks identified for the Project can be summarised as follows:

- Revocation of the environmental license, see “Post-Effective Date Events” at the beginning of this Report.
- Permitting delays.
- Change in energy strategy due to potential energy shortages or crisis.
- Delays and disruption due to backlash from local anti-mining groups.

Risk management activities will continue to be addressed in the execution phase by DPME, and its engineering and management consultants. Management procedures will address project, business, and event risks.

Figure 1.5 – Project Schedule



** Note: The schedule presented above is intended solely to illustrate the general sequence and estimated durations of key Project activities. Estimated finish date of the project and activities are dependent on when, or if, the environmental license will be re-granted. See section entitled "Post-Effective Date Events" at the beginning of this Report.*
Source: DRA, 2025

1.11 Interpretations and Conclusions

Full details on interpretations and conclusions described below are provided in Section 25.

1.11.1 Geology and Resource Database

- The results of the quality control (QC) samples, together with the quality assurance/quality control (QA/QC) procedures implemented by INV at Loma Larga, provide adequate confidence in the data collection and processing, and the assay data is suitable for Mineral Resource estimation.
- Understanding of the Project geology and mineralisation, together with the deposit type, is sufficiently well established to support Mineral Resource and Mineral Reserve estimation.

1.11.2 Mineral Processing and Metallurgical Testing

A significant amount of testwork was conducted to develop a robust and fit for purpose flowsheet for the development of the Loma Larga process plant design. The merits of sequential and bulk flotation flowsheets were examined during the program and analysed. Sufficient testwork has been conducted to support the next phase of the Project.

1.11.3 Mineral Resource

- Block grade interpolation was carried out using Ordinary Kriging (OK) for gold, silver, copper, and density. A 2.0 g/t Au wireframe model (High-Grade Zone) and a 0.8 g/t Au wireframe model (Low-Grade Zone) were used to constrain the grade and density interpolations.
- An NSR cut-off value of US$65/t is appropriate for reporting current Mineral Resources for the Project, which is based on the potential production scenario.
- Mineral Resources are estimated in four (4) zones: the High-Grade Main Zone, which is classified as Measured and Indicated Mineral Resources, the High-Grade Upper Zone, which is classified as Inferred Mineral Resources, and the Low-Grade Main and Lower Zones, which are classified as Indicated and Inferred Mineral Resources.
- At a US$65/t NSR cut-off value, Measured Mineral Resources are estimated to be 0.3 Mt grading 4.80 g/t Au, 20.8 g/t Ag, and 0.23% Cu. Indicated Mineral Resources are estimated to be 5.7 Mt grading 2.79 g/t Au, 21.9 g/t Ag, and 0.16% Cu. Inferred Mineral Resources are estimated to be 2.2 Mt grading 2.54 g/t Au, 37.0 g/t Ag, and 0.19% Cu.
- A sulphur block model was added to the Loma Larga Project to support metallurgical testwork. The local block grades are not as well supported as the payable metal block grades of the current Mineral Resource Estimate.

1.11.4 Recovery Methods

The Loma Larga process plant flowsheet and design are robust and allow for the treatment of the various ore types that will be encountered over the Project LOM. It is also considered to be conventional and fit for purpose of producing separate saleable gold-copper and pyrite concentrates.

Plant flotation circuit will use Jameson flotation cell technology which provides the benefit of reduced Capex through the small footprint and does not require flotation blowers for air supply.

The design considers one stage of copper cleaner flotation and two stages of pyrite cleaner flotation. The design of the Jameson cells selected for the duty accommodates a production increase from 3,000 to 3,400 t/d without the addition of extra cleaning stage.

1.11.5 Environmental Studies, Permitting and Social or Community Impact

The following interpretations and conclusions are noted:

- No known environmental issues were identified by the QP from the information provided by DPM for review. In the QP's opinion, there are no environmental issues that could materially impact the ability to extract the Mineral Resources and Mineral Reserves based on the review of the available documentation.
- DPM maintains an up-to-date record of the legal permits obtained to date and future permits to be obtained according to the applicable legislation and regulatory requirements. The

environmental permits already approved and future environmental permits to be obtained are listed in Table 20.1.

- The Environmental License resolution for exploitation activities for the Project was issued by MAATE on June 23, 2025 and subsequently revoked by the Government of Ecuador on October 3, 2025. The viability of the Project is dependent on the granting of the Environmental license.
- DPM developed an environmental permitting schedule listing the main permitting processes to be completed ahead of the construction stage and the anticipated timeline.
- DPM is preparing to restart negotiations with the Ecuadorian Government for the Project regarding the Mining Exploitation Agreement between DPM and the Government of Ecuador.
- Geochemical characterisation studies have been developed for the Project and a water management system protective of water quality has been engineered according to the results of the geochemical studies and predictions from water quality numerical modelling.
- The expansion of the public access road from San Gerardo requires land acquisition by DPM, which may involve the need for involuntary resettlement of some people living along the road. Although it is anticipated that potential resettlement would involve few people, it must be properly managed by DPM with close support from the social management team through a process that is transparent, equitable, and effective.
- DPM has adopted several communication mechanisms to inform the public about the Project and community development projects. DPM has implemented a consultation and grievance mechanism for the communities to provide feedback or file requests, questions, concerns, and/or complaints regarding the Project.
- DPM is working on activities to promote development of local suppliers, grow relationships with current and potential local suppliers, generate a supplier database, and promote local ventures. The Company has been implementing its social action and development plan by supporting activities associated with economic production activities, education, health, infrastructure, culture, and recreation.
- According to information provided by DPM, the Escaleras community is recognised as Indigenous by the Ecuadorian government.
- A conceptual Mine Closure Plan was developed by DPM as part of the EIA. Two (2) years prior to the planned completion of the Project (i.e. the operating phase), DPM must submit the detailed Mine Closure Plan to the environmental authority for approval.

1.11.6 Economic Analysis

Based on the assumptions presented in this Report, the results of the economic analysis indicate a positive after-tax NPV of US$488 M at a discount rate of 5% and an after-tax IRR of 18.1%. The sensitivity analysis indicates that the Project economics are most sensitive to the price of gold. The Project maintains a positive NPV even at a gold price 20% below the base case of US$1,900/oz.

1.12 Recommendations

Specific recommendations for the Project are summarised below, and full details are provided in Section 26 of the Report.

1.12.1 GEOLOGY

Ensure that procedures for investigating, correcting, and documenting results of QA/QC noncompliance issues such as biases or failures are followed for future planned drillhole programs.

DPM is currently preparing to restart drilling operations once it receives regulatory permission to do so, with an initial 23 kilometres of drilling planned. Drilling will focus on infilling existing mineral resources and extending the mineralisation where the potential exists.

Additionally, as part of the project engineering requirements, geotechnical and hydrogeological drilling is planned to collect information to support the design of the underground and surface infrastructure. Furthermore, a series of dedicated metallurgical twin holes are planned to collect fresh samples for confirmatory metallurgical testwork as well as pilot scale testing and additional variability sampling. The proposed drilling program is presented in Table 1.7.

Table 1.7 – Proposed Drilling Program

Purpose	Drillholes	Meterage
Plant Site and Ancillary Facilities		
Geotechnical	43	1,245
Hydrogeological	4	150
Filtered Tailings Storage Facility		
Geotechnical	6	95
Hydrogeological	2	120
Dual Geotech / Hydro	5	85
Condemnation	5	2,000
Loma Larga Orebody		
Metallurgical	10	2,310
Infill / Extension	68	16,600
Hydrogeological	2	400
Camp Area		
Geotechnical	5	75
Total	**150**	**23,080**

DPM has prepared a budget to support the drilling, analytical and metallurgical testwork requirements, which is due to be released once regulatory permission to conduct surface drilling activities has been granted.

1.12.2 Mineral Processing and Metallurgical Testing

- Representative gold-copper and pyrite concentrates, and flotation tailings samples will need to be produced in sufficient quantities for filtration testwork to be conducted by potential vendors.
- Ramp up to 3,400 t/d as soon as practicable (de-bottleneck process sub-systems) and validate further increases in plant throughput through de-bottlenecking.
- Evaluate processing lower grade ores to extend the LOM.

1.12.3 Mineral Resource

- Develop an understanding of the work required to support upgrading areas of Indicated Mineral Resources in the High-Grade Main Zone to Measured Mineral Resources.
- Perform additional drilling in the High-Grade Upper Zone and Low-Grade Main and Lower Zones to upgrade Inferred to Indicated.

1.12.4 Mineral Reserve Estimate

- Perform a program of infill drilling once the orebody is reached by the ramp to better delineate the limits of the high-grade zones and refine the limits of the orebody.

1.12.5 Recovery Methods

- Update the process mass and water balances with new testwork data, mine plan data, and/or vendor data when available.
- Consider equipment consolidation and bundling from reputable equipment manufacturers.

1.12.6 Project Infrastructure

- Investigate alternative power generation technologies, including but not limited to wind and solar.
- Perform further geotechnical and survey information to progress and optimise designs.
- Identify specific effluent water treatment requirements based on the geochemical characterisation and the preliminary assimilative capacity study.
- Complete tailings filtration testing to support the FTSF design.
- Perform geotechnical investigation in borrow areas to assess quality and quantity of available sources to produce construction materials.
- Prepare plans to support construction, operation, maintenance and closure of FTSF.

- Prepare documentation and registers to assess and maintain, during the life cycle of the FTSF, compliance with GISTM and local Ecuadorian guidelines including the preparation of the FTSF construction permit per the guidelines.
- Update water balance model to incorporate any changes on the mine plan, detailed FTSF fill plan, including updates to geochemistry of leachates.
- Update domestic water distribution to include a second domestic water treatment plant on-site. Capex reductions can potentially be realised through elimination of pipeline between camp and site.

1.12.7 Environmental Studies, Permitting and Social or Community Impact

- Continue updating the record of current legal permits and future permits to be obtained.
- Review and update the environmental permitting schedule regularly to ensure adequate time frames for regulatory processes are captured and potential risks associated with environmental permitting approvals are flagged in a timely manner.
- Review and update regularly the community engagement plan as social risks requiring mitigations change and evolve.

1.12.8 Economic Analysis

- Update Economic Analysis based on the outcome of Exploitation Contract negotiations between DPM and the Government of Ecuador, including items such as Production Royalty calculations and tax exemptions.
- Maintain engagement with a tax consultant to ensure that the taxation calculations applied in the analysis are aligned with the latest regulations from the Government of Ecuador.

2 INTRODUCTION

DPM Metals Inc. (formerly Dundee Precious Metals Inc.) (DPM or the Company) is a Canadian-based international gold mining company with headquarters in Toronto, Ontario, Canada and operations and projects located in Bulgaria, Bosnia and Herzegovina, Serbia, and Ecuador. DPM is listed on the Toronto Stock Exchange (TSX) (TSX:DPM) and the Australian Stock Exchange as a Foreign Exempt Listing (ASX:DPM) (ARBN: 689370894), with headquarters at 150 King Street West, Suite 902, Toronto, Ontario M5H 1J9. DPM acquired the Loma Larga development project (the Project) through its acquisition of INV Metals Inc. (INV) in July 2021.

The Loma Larga Project is located in the country of Ecuador, about 30 km southwest of the city of Cuenca and 15 km north of the town of Girón. It comprises concessions covering an area of approximately 7,960 ha.

Loma Larga is a gold-copper-silver deposit classified as a high sulphidation epithermal system and alteration is characterised by multiphase injections of hydrothermal fluids strongly controlled by both structure and stratigraphy. The deposit is a flat lying to gently western dipping, north-south striking, cigar shaped body, which has a strike length of approximately 1,600 m north-south by 120 m to 400 m east-west and up to 60 m thick, beginning approximately 120 m below surface.

In 2022, DPM authorised the preparation of a Feasibility Study Update (2022 Update) for the Project coordinated by Canadian consulting firm G Mining Services (GMS), in an effort to improve project economics through various updates, including but not limited to updates to mine design, process plant and site infrastructure. The 2022 Report performed the mine planning and design, infrastructure, capital estimation and economic evaluation, and coordinated the overall preparation of the 2022 Update.

In 2023, DPM retained DRA to coordinate the optimisation of various project components of the 2022 Update, which led to the preparation of a Feasibility Study Update Optimisation (FSUO), including but not limited to advancing definition of engineering design and development, constructability and operability, completion of trade-off studies, and obtaining updated pricing for equipment and contractor packages.

In 2025, DPM again retained DRA to coordinate the preparation an updated Capex, Opex and financial model based on the latest market conditions, based on the design and scope from the FSUO, with some minor changes, leading to the preparation of a 2025 Feasibility Study Update Technical Report (Technical Report or Report).

2.1 Study Participants

DPM retained various Consultants to coordinate the preparation of this Report. More specifically, the Consultants listed below provided Qualified Persons (QPs) (as identified in Section 2.2) to participate in and to complete Project and Report as described below:

- **DRA Americas Inc.** (DRA): review metallurgical testwork, develop design criteria and recovery methods, process plant, and related plant infrastructure, as well as compile this Report.
- **Global Mine Design Ltd.** (GMD): develop Mineral Reserve Estimate, design the underground mine and related mine infrastructure.
- **NewFields Mining & Technical Services LLC** (NewFields): design filtered tailings management facility.
- **Paterson & Cooke**: develop paste backfill design.
- **SLR Consulting (Canada) Ltd.** (SLR): develop Mineral Resource Estimate (MRE) as well as environmental studies, permitting, and social impact.

2.2 List of Qualified Persons

As described above, the QPs from DRA and other specialised consultants prepared the work in a collaborative manner. The QPs are responsible for preparation of this Report, and their roles and responsibilities, by Section, are listed in Table 2.1.

Table 2.1 – Qualified Persons and their Respective Sections

Section	Title of Section	Qualified Person(s)	Company
1	Summary	All QPs	All
2	Introduction	Ryan Telford	DRA
3	Reliance on Other Experts	Ryan Telford	DRA
4	Property Description and Location	Ryan Telford	DRA
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Ryan Telford	DRA
6	History	Katharine Masun	SLR
7	Geological Setting and Mineralisation	Katharine Masun	SLR
8	Deposit Types	Katharine Masun	SLR
9	Exploration	Katharine Masun	SLR
10	Drilling	Katharine Masun	SLR
11	Sample Preparation, Analyses and Security	Katharine Masun	SLR
12	Data Verification	Katharine Masun	SLR

Section	Title of Section	Qualified Person(s)	Company
13	Mineral Processing and Metallurgical Testing	David Frost	DRA
14	Mineral Resource Estimates	Katharine Masun	SLR
15	Mineral Reserve Estimates	Dr. Lewis Meyer	GMD
16	Mining Methods (except for 16.1 to 16.4, 16.8, and 16.10)	Dr. Lewis Meyer	GMD
	16.1 General Description	Phil Earl	GMD
	16.2 Mining Methods	Phil Earl	GMD
	16.3 Geotechnical Considerations	Phil Earl	GMD
	16.4 Underground Mine Design	Phil Earl	GMD
	16.8 Mine Ventilation	Phil Earl	GMD
	16.10 Paste Backfill	Leslie Correia	Paterson & Cooke
17	Recovery Methods	David Frost	DRA
18	Project Infrastructure (except for 18.6 to 18.9)	Ryan Telford	DRA
	18.6 Tailings Dewatering and Paste Plant	Leslie Correia	Paterson & Cooke
	18.7 Filtered Tailings Storage Facility	Carl John Burkhalter	NewFields
	18.8 Waste Rock and Ore Stockpile	Carl John Burkhalter	NewFields
	18.9 Water Management	Carl John Burkhalter	NewFields
19	Market Studies and Contracts	Daniel Gagnon	DRA
20	Environmental Studies, Permitting and Social or Community Impact	Luis Vasquez	SLR
21	Capital and Operating Costs (except for 21.1.8 and 21.2.4)	Ryan Telford	DRA
	21.1.8 Capex – Underground Mine	Sean McCarthy	GMD
	21.2.4 Opex – Underground Mine Operating Costs	Sean McCarthy	GMD
22	Economic Analysis	Daniel Gagnon	DRA
23	Adjacent Properties	Katharine Masun	SLR
24	Other Relevant Data and Information	Ryan Telford	DRA
25	Interpretation and Conclusions	All QPs	All
26	Recommendations	All QPs	All
27	References	All QPs	All

2.3 Site Visits

Table 2.2 provides details of the personal inspections of the Property by the QPs.

Table 2.2 – Site Visit

Qualified Person	Company	Date of Site Visit
Ryan Telford	DRA	February 25, 2025
David Frost	DRA	June 27 to 28, 2017
Daniel Gagnon	DRA	August 1 to 2, 2017
Phil Earl	GMD	August 21 and 24, 2023
Sean McCarthy	GMD	August 21 and 24, 2023
Carl John Burkhalter	NewFields	April 11, 2022
Katharine Masun	SLR	February 17 to 20, 2014 and February 25. 2025

2.4 Effective Date

The effective date of the Report is May 28, 2025. Apart from the revocation of the Environmental License subsequent to the effective date, the QPs are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this report misleading.

2.5 Sources of Information

QP, Ryan Telford, P. Eng. from DRA was the lead author for the Technical Report and for supervision and compilation of this Report, in collaboration with the other QPs responsible for preparing various portions of this Report. This Technical Report is inclusive of the work and deliverables by and/or under the supervision of all QPs.

The Project assessments of the QPs were based on maps, published material, pre-existing reports, Project development work specifically performed by the Consultants and others, and data, professional opinions and published and unpublished material provided by DPM. The QPs reviewed all relevant data provided by DPM and/or its agents. The QPs reviewed and evaluated all information used to prepare this Report and believe that such information is valid and appropriate considering the status of the Project and the purpose for which this Report is prepared. A full listing of references is provided in Section 27. The QPs do not disclaim any responsibility for any of this information.

2.6 Units and Currency

In this Report, all currency amounts are in US Dollars ($ or US$), unless specifically stated otherwise. Quantities are generally stated in *Système international d'unités* (SI) units as per standard Canadian and international practice, including tonne (t) for mass, and kilometre (km) or metre (m) for distance. Abbreviations used in this Report are listed in Section 28.

2.7 Non-GAAP Financial Measures

Certain financial measures referred to in this Report are not measures recognised under IFRS Accounting Standards (IFRS) and are referred to as non-Generally Accepted Accounting Principles (non-GAAP) financial measures or ratios. These measures have no standardised meaning under IFRS and may not be comparable to similar measures presented by other companies. These measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

The non-GAAP financial measures used in this Technical Report and common to the gold mining industry are defined below:

- Cash cost per tonne of ore processed: Cash cost per tonne of ore processed consists of all production related expenses including mining, processing, services, filtered tailings and paste fill, royalties and general and administrative, divided by the tonnes of ore processed.
- Cash cost per ounce of gold sold: Cash cost per ounce of gold sold consists of all production related expenses including mining, processing, services, filtered tailings and paste fill, royalties and general and administrative, plus treatment charges, penalties, transportation and other selling costs, net of by-product credits, divided by payable gold ounces.
- All-in sustaining cost per ounce of gold sold: All-in sustaining cost consists of all production related expenses including mining, processing, services, filtered tailings and paste fill, royalties and general and administrative, plus treatment charges, penalties, transportation and other selling costs, cash outlays for sustaining capital expenditures and leases, and rehabilitation-related accretion and amortization expenses, net of by-product credits, divided by payable gold ounces.

Cash cost per tonne of ore processed, Cash cost per ounce of gold sold, and all-in sustaining cost per ounce of gold sold capture the important components of the Company's production and related costs and are used by the Company and investors to monitor cost performance at the Company's operations.

As the Project is not in production, the QPs do not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios presented may not be reconciled to the nearest comparable measure under IFRS.

3 RELIANCE ON OTHER EXPERTS

The QPs have reviewed and analysed data and reports provided by DPM, together with publicly available data, drawing its own conclusions augmented by direct field examination.

The QPs who prepared this Report relied on information provided by experts who are not QPs. The QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report. Except as noted below, the QPs do not disclaim any responsibility for any of this information.

For Section 4 QP Ryan Telford (DRA), has relied upon:

- DPM for information regarding the Project mining concessions and their current legal status as discussed in Section 4.4 of this Report.

QP Ryan Telford (DRA), has not independently verified legal ownership of surface title and exploration licenses comprising the Project beyond information that is publicly available or been provided by DPM. The Property description presented in this Report is not intended to represent a legal, or any other opinion as to title ownership.

For Section 19, QP Daniel Gagnon (DRA) fully relied upon:

- Commercial terms Excel Spreadsheet provided by DPM dated April 17, 2025.

For Section 22, QP Daniel Gagnon (DRA) fully relied upon:

- Royalty and Taxation memo provided by DPM (*DPM retained EY Addvalue Asesores Cia. Ltda., based in Quito, to review the tax calculations included in the financial model*) via email dated May 10, 2025

The QPs have assumed, and relied on the fact, that all the information and existing technical documents listed in the References Section 27 of this Report are accurate and complete in all material aspects, but have taken all appropriate steps in their professional judgement to confirm same and are not disclaiming any responsibility for this Report except as permitted under applicable securities laws. The QPs reserve the right, but will not be obligated, to revise the Report and conclusions, if additional information becomes known subsequent to the date of this Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Land Tenure

4.1.1 MINING CONCESSIONS

DPM acquired 100% of the company INV Metals Inc. in July 26, 2021 and obtained 100% title to the Loma Larga property (Property). The change of name from INVMETALS to DPM materialised through a public deed executed before the Twenty-Fifth Notary´s Office of Quito on November 10, 2021 and duly registered in the Mercantile Registry of Canton Cuenca in December 22, 2021. Previously, INVMETALS had acquired IAMGOLD's Ecuadorian subsidiary in November 2012, which was the former owner of Loma Larga's mining titles. The change of name from IAMGOLD to INVMETALS materialised through a public deed executed before the Sixth Notary´s Office of Cuenca on February 28, 2013 and duly registered in the Mercantile Registry of Canton Cuenca on December 12, 2013. Loma Larga's mining concessions, each with an approximate initial 30-year term, are presented in Table 4.1 and are depicted in Figure 4.1.

Table 4.1 – Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project)

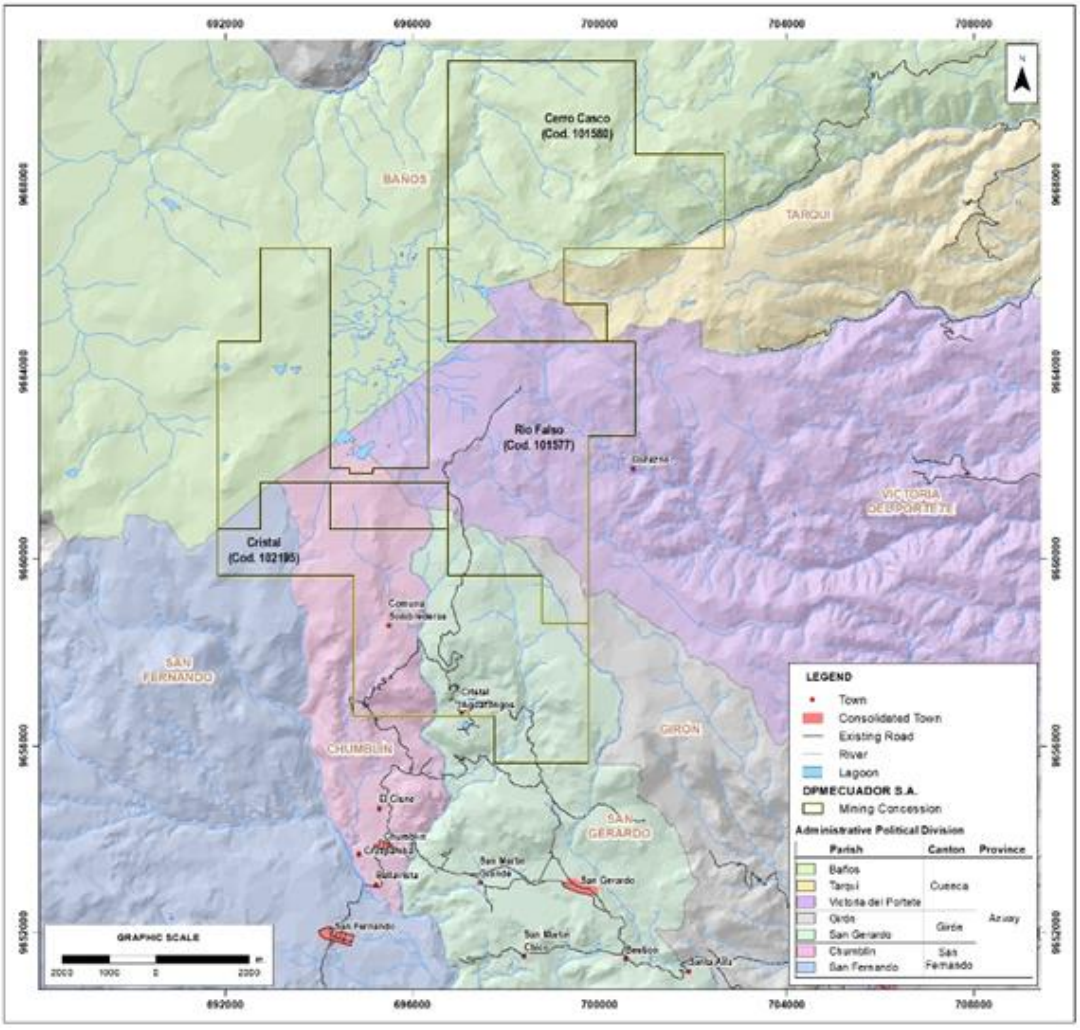
Name	Code	Area (ha)	Registered	Expiry
Cerro Casco	101580	2,572	November 23, 2001	July 1, 2030
Cristal	102195	2,250	June 5, 2003	July 12, 2032
Rio Falso	101577	3,208	November 23, 2001	July 1, 2030

The concessions Cerro Casco, Río Falso and Cristal are located in the province of Azuay, and its extension covers three (3) cantons and seven (7) parishes as listed in Table 4.2.

Table 4.2 – Political and Administrative Location Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project)

Province	Canton	Parish
Azuay	Cuenca	Baños Tarqui Victoria del Portete
	Girón	Girón San Gerardo
	San Fernando	Chumblín San Fernando

Figure 4.1 – Initial IAMGOLD Mining Concessions (INV Metals Inc. – Loma Larga Project)



Source: Mining Concessions Registered, 2001 and 2003

To meet the requirement of Article 38 of the Ecuadorian Mining Law and to maintain concessions in good standing, DPM has been submitting annual exploration reports for each claim, itemising expenditures, exploration activities and the investment plan for the following period. The concessions have been most recently validated in March 2025.

In May 2014, the Company had the right under the Mining Act to submit an application to advance the concessions from the initial exploration stage to a four-year advanced exploration stage. The company's minimum exploration commitment was US$500,000 per year for the period from 2014 to 2018, which has been met on an annual basis over this period. The property was reduced by 70 ha to a revised total of 7,960 ha as required by the application (Table 4.3).

Table 4.3 – Property Size for Economic Evaluation Stage

Mining Concession	Code	Current Area (ha)
Cerro Casco	101580	2,552
Rio Falso	101577	3,168
Cristal	102195	2,240
Total		**7,960**

The government approved the conversion of the Project's mining concessions to the advanced exploration stage on October 23, 2014, complying with the concession reduction process, as noted in the registry of minutes at the Mining Registry on May 4, 2016. In October 2018, the company advised the Ministry of Energy and Non-renewable Natural Resources that they were changing from Advanced Exploration to Economic Evaluation for the three (3) concessions of the Loma Larga Project.

Through resolutions of the Ministry of Energy and Non-Renewable Natural Resources, granted to INV MINERALES ECUADOR S.A. INVMINEC (now DPMECUADOR S.A.) the declaration of term extension of the economic evaluation period, of the Cerro Casco, Rio Falso and Cristal concessions, for a term of two (2) years, counted from the date of issuance of said resolutions and their notification. Said resolutions were duly notarised and registered in the Mining Registry of the Zonal Coordination 6 Azuay on July 27, 2021.

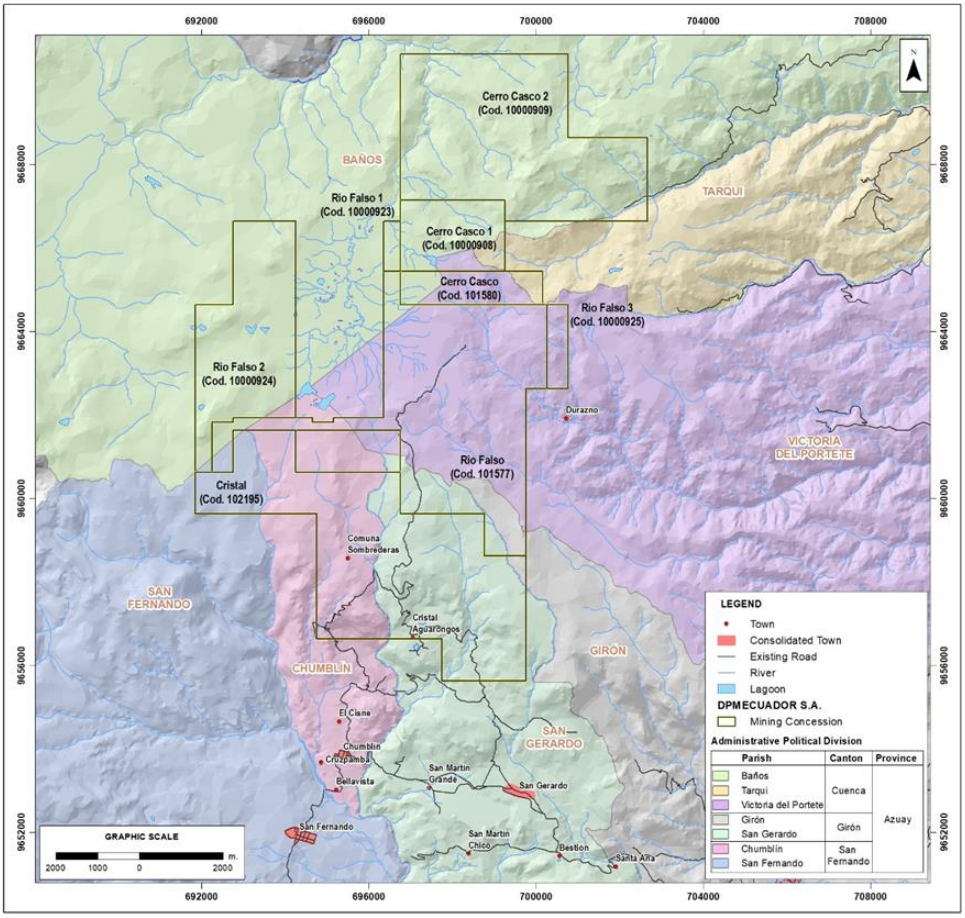
Article 32 of the Mining Law states that the mining title is subject to material division or accumulation, with a limit of minimum one (1) mining hectare and maximum 5,000 mining hectares per concession. Additionally, Article 39 of the Mining Law states that for the exploitation phase and the subsequent signing of the Mining Exploitation Contract, no concession holder may have one (1) or more mining titles that together total an area greater than 5,000 mining hectares.

Consequently, INV MINERALES ECUADOR S.A. INVMINEC (now DPMECUADOR S.A.), submitted two (2) requests for the material division of the mining concessions: Cerro Casco (code 101580) (August 31, 2020) and Río Falso (code 101577) (August 31, 2020). As a result of this division, the three (3) concessions (Cerro Casco and Río Falso) divided into seven (7) concessions: Cerro Casco (code 101580), Cerro Casco 1 (code 10000908), Cerro Casco 2 (code 10000909), Río Falso (code 101577), Río Falso 1 (code 10000923), Río Falso 2 (code 10000924), Río Falso 3 (code 10000925). The Cristal concession (code 102195) was not materially divided as listed in Table 4.4 and is illustrated in Figure 4.2.

Table 4.4 – Size of the Mining Concessions as Result of the Material Division

Mining Concession	Code	Current Area (ha)
Cerro Casco	101580	272
Cerro Casco 1	10000908	425
Cerro Casco 2	10000909	1,855
Río Falso	101577	2,015
Río Falso 1	10000923	48
Río Falso 2	10000924	1,005
Río Falso 3	10000925	100
Cristal	102195	2,240
Total		**7,960**

Figure 4.2 – Size of the Mining Concessions as Result of the Material Division



Source: Mining Concessions Registered, 2021 and 2022

Subsequently, on April 27, 2022, DPM submitted an accumulation request for the areas of the mining concessions Cerro Casco (code 101580), Río Falso (code 101577), and Cristal (code 102195), to create the Cristal concession (code 102195 - Table 4.5) for the exploitation phase of the Loma Larga Project. This request has been approved by resolution No. MEM-CZCS-2022-0169-RM issued by the Zonal Coordination No.6 of the Ministry of Energy and Mines dated October 21, 2022 as listed in Table 4.6 and depicted in Figure 4.3.

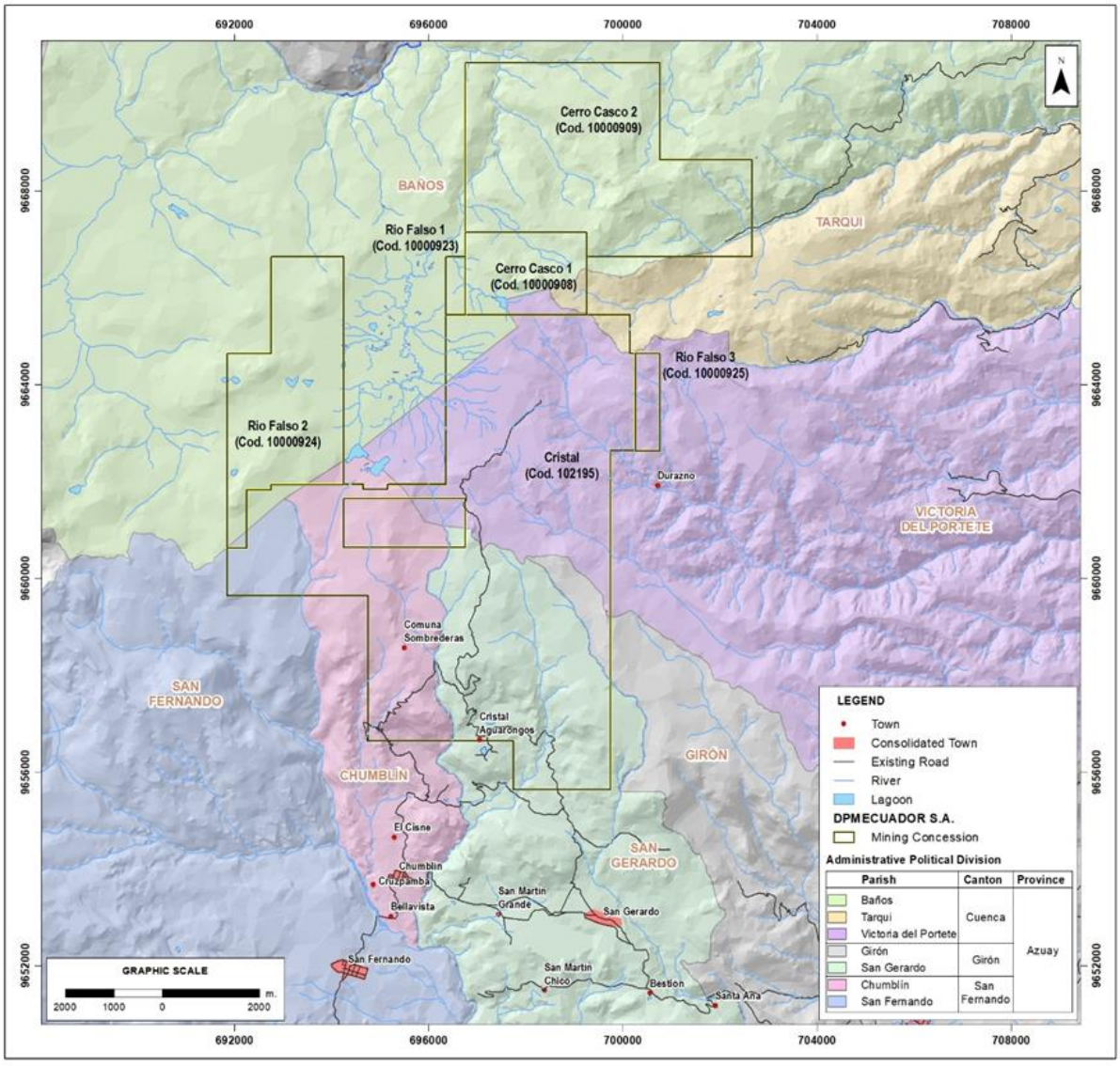
Table 4.5 – Cristal Concession (Exploitation Phase – Loma Larga Project)

Name	Code	Area (ha)	Registered	Term
Cristal	102195	4,527	November 14, 2022	10 years, 7 months, and 26 days

Table 4.6 – Size of the Mining Concessions as Result of Accumulation

Mining Concession	Code	Current Area (ha)
Cerro Casco 1	10000908	425
Cerro Casco 2	10000909	1,855
Río Falso 1	10000923	48
Río Falso 2	10000924	1,005
Río Falso 3	10000925	100
Cristal	102195	4,527
Total		**7,960**

Figure 4.3 – Size of the Mining Concessions as a Result of Accumulation



Source: Mining Concessions Registered, 2021 and 2022

DPM's mining concessions intersect with the National System of Protected Areas, Protected Forest and Vegetation Areas, State Forest Heritage, and Water Recharge Zone as depicted in Figures 4.4 and 4.5. These areas are:

- The *Área Nacional de Recreación Quimsacocha* (ANRQ) - created on January 25, 2012, by Ministerial Agreement No. 007 published in the Official Gazette No. 680 of April 11, 2012. This area includes a "biological corridor" that overlaps partially the Cerro Casco 1 (code 10000908) and Rio Falso 1 (code 10000923) concessions.

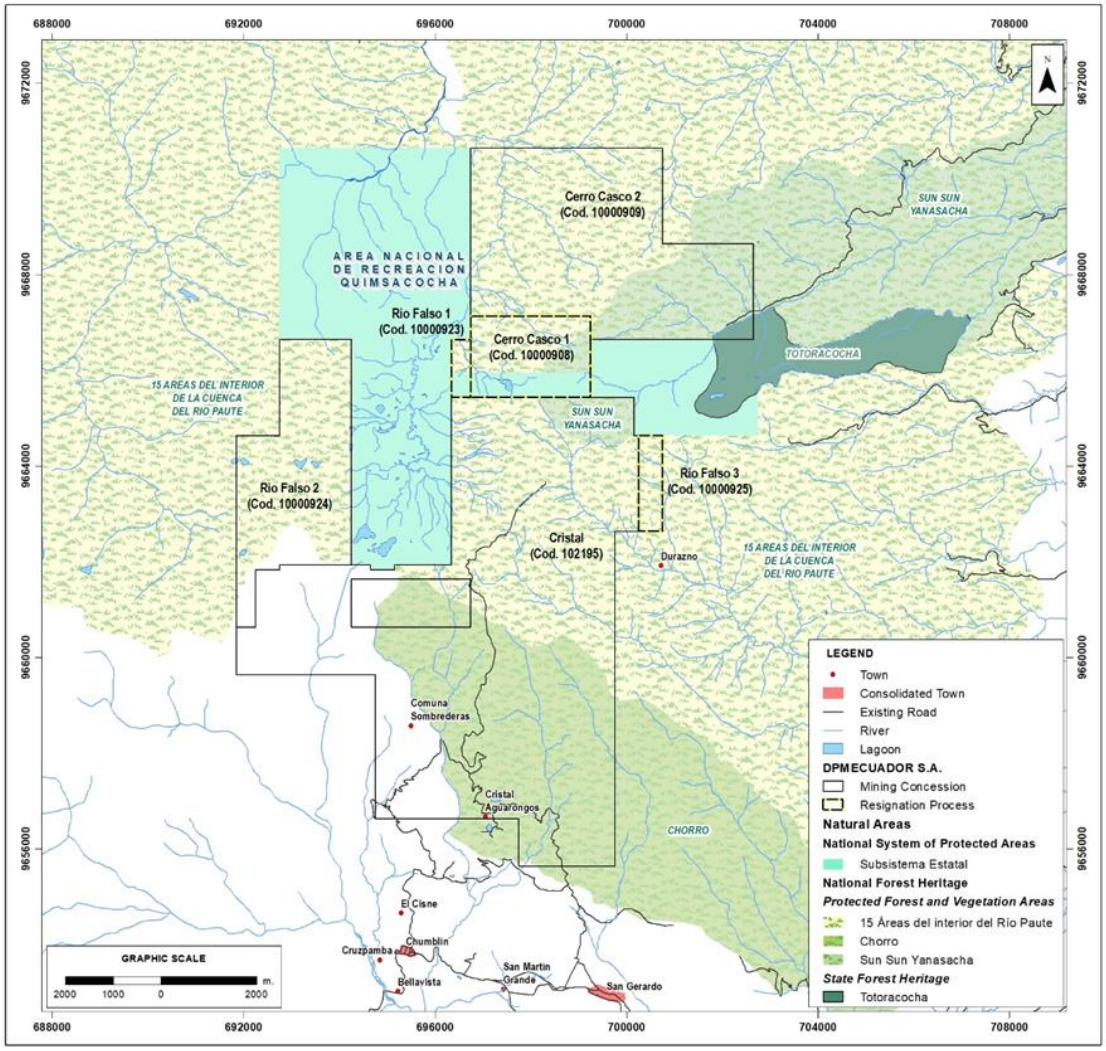
- The Protective Forest and Vegetation El Chorro, *15 Áreas al Interior de la Cuenca del Río Paute y Sunsun Yanasacha* – created through Ministerial Agreement No. 12, published in the Official Gazette No. 143 of March 4, 2010, with an update on August 3, 2010.State Forest Heritage Totoracocha.
- Water Recharge Zone of the Cuenca Canton.

DPM is currently in the process of relinquishment of Rio Falso 1 and Cerro Casco 1 mining concessions at the Zonal Coordination of the Ministry of Energy and Mines in Cuenca, based on the recommendations of the Ministry of Environment, Water and Ecological Transition.

Current requirements in these areas are focused on protection of the biodiversity, land reclamation, use and management of water resources, and research and strengthening of the organisations involved in the management of the protected area. It is expected that through the Environmental Impact Statement (EIS) review process, the Ministry of Environment and Water will establish specific requirements for the Project.

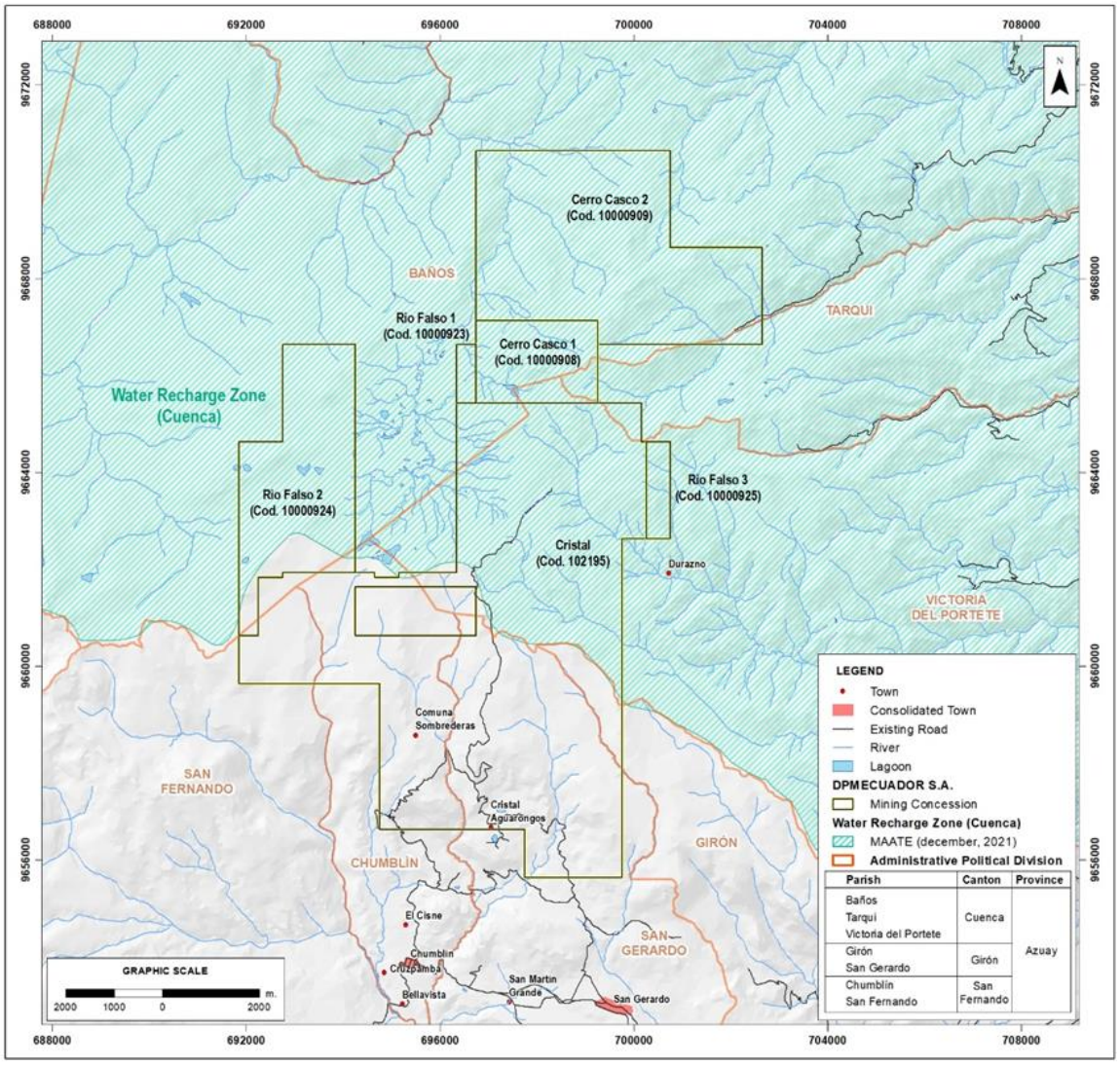
The Mining Act of Ecuador stipulates that holders of mining concessions must pay an annual “Patent Conservation Fee” for each mining hectare by March in each year. The conservation patent fee from the date of the concession is granted up until December 31 in the year the initial exploration period expires and is equivalent to 2.5% of a unified minimum wage (equivalent to US$470/month for 2025) for each mining hectare. This fee doubles to 5% of the minimum wage per hectare for the advanced exploration and economic evaluation periods. During the operational phase of the mining license, the fee doubles again to 10%, per hectare per year.

Figure 4.4 – National System of Protected Areas, Protected Forest and Vegetation Areas, State Forest Heritage



Source: DPM, 2023

Figure 4.5 – Water Recharge Zone



Source: MAATE, 2022

4.2 Surface Rights

In addition to the mining concessions, DPM had previously purchased two (2) areas of surface rights, approximately 300 ha, at the Los Pinos Camp area and a 200-ha parcel within the Forestry Reserve referred to as Chorro Tasqui. In addition, an easement agreement was entered into with a local landowner (Table 4.7). The two (2) land purchases as well as the easement agreements were transferred to INV (now DPMECUADOR S.A.) and have been maintained in good standing.

Table 4.7 – Current Surface Rights Agreements (INV Metals Inc. – Loma Larga Project)

Date	Document	Owner
March 17, 2006	Purchase-Sale Public Deed to Mr. Jorge Jarrin.	DPMECUADOR S.A. (previously INV Minerales Ecuador S.A.)
March 8, 2007	Purchase-Sale Public Deed to Mr. Leonardo Castro.	DPMECUADOR S.A. (previously INV Minerales Ecuador S.A., previously IAMGOLD ECUADOR S.A.).
June 27, 2007	Easement Agreement (to carry out exploration activities in the North Concession Cerro Casco)	Mr. Juan José Mogroveio
February 16, 2024	Easement Agreement update (to carry out exploration activities in the North Concession Cerro Casco, Río Falso and Cristal)	Mrs. Blanca Laura Mogrovejo Mogrovejo Mr. Carlos Hugo Mogrovejo Mogrovejo Mr. José Antonio Mogrovejo Mogrovejo Mrs. María del Cisne Mogrovejo Mogrovejo Mr. Víctor Bolívar Mogrovejo Mogrovejo Mrs. Cruz Mercedes Mogrovejo Illescas

4.3 Land Acquisition Strategy

The Project's planned mine infrastructure will be developed within the surface rights already owned by DPMECUADOR S.A. No landowner royalties, back-in rights, or other holds on the owned properties exists. As a result, there is no need to acquire additional land to develop mine infrastructure. However, for the linear components associated with the Project (access road from San Gerardo and the 69 kV transmission line), DPM will need to negotiate an easement or land purchase for certain areas. DPM has developed a land acquisition strategy and land access strategy that incorporates the relevant national legislation and corporate policies.

The land acquisition and access strategy prioritise achieving easement access agreements through a process that is transparent, equitable, and effective. Through this process, DPM will seek to reduce the effects caused by the Project's easement access requirements.

IFC's Performance Standard No. 5 (PS5) on Land Acquisition and Involuntary Resettlement will be a guiding principle for obtaining land access. This Performance Standard refers to the management of physical and economic involuntary displacement resulting from Project-related land acquisition. DPM does not expect any resettlement of houses for the development of the Project.

PS5 objectives include the following:

- Land rights and/or assets acquired through expropriation or other compulsory procedures.
- Land rights and/or assets acquired through negotiated settlements, if expropriation or other compulsory processes can be initiated in case negotiation fails.

- Restrictions that result in people experiencing loss of access to land, assets, natural resources or livelihoods, irrespective of whether such rights of restriction are acquired through negotiation, expropriation, compulsory purchase, or by means of government regulation.
- Relocation of people without formal, traditional, or recognisable usage rights under national laws, who are occupying or utilising land prior to the cut-off date.
- Displacement of people as a result of Project impacts that render their land or assets unusable or inaccessible.

The proposed land access strategy for the linear components of the Loma Larga Project is structured in three (3) phases:

1. Preparation:
    - Definition and prioritisation of land access requirements based on the design of linear components.
    - Identification of key risks and impacts.
    - Definition of the framework that specifies the structure within which individual affected landowners and users will be indemnified for the impacts of the right of way acquisition.
2. Collective Implementation:
    - Provide information collectively to those affected to make clear the terms and conditions that will guide the land access process.
    - Perform the baseline studies (socio-economic survey of affected households, and the land and assets inventory of the properties' area affected by the right of way) that are required to apply the entitlement framework to each individual landowner.
3. Individual Implementation:
    - Negotiation process at the individual household level to arrive at final indemnifying agreements.

4.4 Permitting

The Project requires certain permits to allow exploration, construction execution and operational phases. These permits are discussed in Section 20.5 of this Report.

4.5 Royalties

The Ecuadorian government levies a of 5% royalty on the NSR for production of metallic raw materials.

Refer to Section 22.6 for additional details on Mineral Royalties.

4.6 Other Significant Factors and Risks

The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Loma Larga Project is located 30 km southwest of the city of Cuenca and approximately 15 km north of the town of Girón. The current regional and national infrastructure is adequate to access the Project site through a well-established network of existing major ports and roadways. The road between San Gerardo and the Project site will be upgraded during the early stages of the Project (as part of a separate project) and will be ready to support the operations phase of the Project. Access roads and other Project Infrastructure are described in Section 18 of this Report.

5.2 Ports

Puerto Bolivar, in Machala, has infrastructure and facilities to serve most of the Project needs for importation, construction, and operations phase. Manta Port is the closest regional port which will support any oversize and roll-on / roll-off cargo. The port at Machala is sufficient to handle all other cargos.

The copper and pyrite flotation concentrates will be exported from Puerto Bolivar, Machala. The concentrate will be loaded in bulk in standard sea containers.

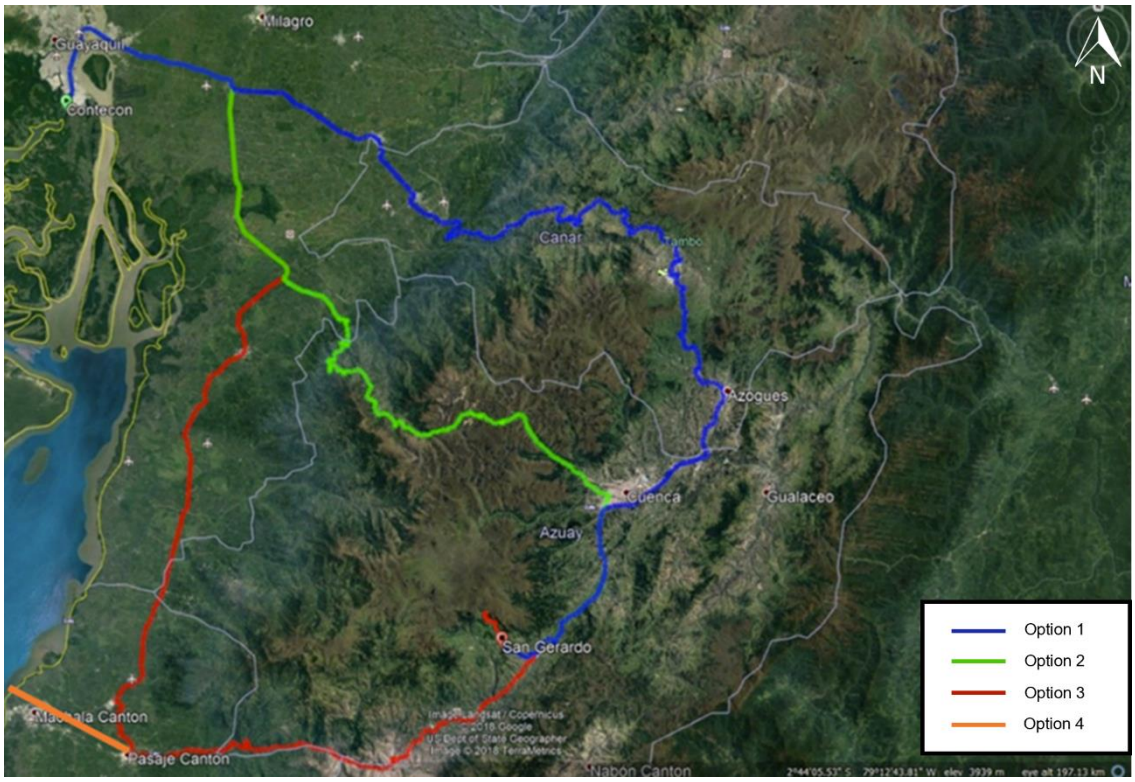
5.3 Road Transportation

The *Ministerio de Transporte y Obras Publicas* (MTOP) (in English, “Ministry of Transport and Public Works”) is responsible for issuing permits for the transport of overweight, oversize, high-volume or hazardous cargo. A permit will be required for the import of the major equipment for the Loma Larga Project.

There are three (3) main routes between Guayaquil and Project Site and one (1) between Machala and the Project Site as shown in Figure 5.1.

- Option 1 (Blue) – Guayaquil, La Troncal, Cuenca, Girón (approximately 310 km).
- Option 2 (Red) – Guayaquil, Machala, Girón (approximately 300 km).
- Option 3 (Green) – Guayaquil, El Cajas National Park, Cuenca, Girón (approximately 260 km).
- Option 4 (Orange) – Machala, Girón (approximately 180 km).

Figure 5.1 – Route Options Between Guayaquil / Machala and Girón



Source: Google Earth, 2022

DPM will engage with the MTOP during the evaluation and selection of the best route for overweight or oversize cargo imported for the Project; and for the export of concentrate from site to Machala – Puerto Bolivar. The road conditions between Guayaquil and San Gerardo are in good condition. A road study by Noroccidental indicates that the roads from the site to Machala can be improved to accommodate the largest/heaviest anticipated construction loads. However, further studies will be made during detailed engineering to ensure that the most economical route regarding infrastructure and road/bridge improvements costs, will be selected.

The road from San Gerardo to the Loma Larga Site is currently in a condition adequate for the work that has been performed to date. An upgrade planned for the Project to make it suitable for a mining operation is described in Section 18.3 of this Report.

A staging area will be required in the area of the incoming port. The preferred alternative would be to use any of several existing commercial options at secure port storage facilities, located in the secure port area, which have their own lifting equipment. Where required, mining equipment can be unloaded in San Gerardo and driven to site.

A staging area will not be required in the local area outside the port.

5.4 Climate

The analysis of the meteorological conditions of the area was based on site and regional climate data collected by PROMAS - Universidad del Azuay. The climate data analysis was completed by NewFields (2018) using data collected by the meteorological stations. Data from the Quimsacocha 1 meteorological station was used to provide an indication of the regional air temperature. The average annual temperature ranged from a high of 7.5ºC in 2010 to a low of 5.7ºC in 2012 (represent values for full years of data available between 2005-2013). The absolute minimum and absolute maximum daily air temperatures ranged from a low of -5.3ºC on November 28, 2005, to a high of 24.2ºC on October 3, 2008.

Average peak 24-hour precipitation values from three (3) of the meteorological stations were as follows:

- Quimsacocha 1 meteorological station: 31.3 mm.
- Chanlud meteorological station: 40.2 mm.
- Labrado meteorological station: 36.4 mm.

All three (3) sites were on the same order of magnitude with an average from the three (3) sites of approximately 36 mm (NewFields, 2018).

Based on an evaluation of the existing monthly and annual precipitation, there is some uncertainty in the long-term average precipitation at the site (NewFields, 2018). Therefore, a conservative approach (overestimation) has been used for the purposes of the FS.

Table 5.1 shows the average estimated monthly and annual site precipitation based on long-term monthly precipitation data from Labrado with values increased by 30 percent (%). This 30 %increase corresponds to the differences observed between Labrado and Zhuracay Pozas from 2011 to 2014. As presented in Table 5.1, the resultant mean precipitation rate is 1,625 mm per year. Actual mean annual precipitation could be less and closer to the uncorrected Labrado value of 1,256 mm per year; ongoing data collection will allow precipitation estimates to be refined in the future.

Table 5.1 – Estimated Mean Monthly and Annual Precipitation at Loma Larga Site

Month	Mean Precipitation (mm)
January	123
February	148
March	173
April	190
May	152

Month	Mean Precipitation (mm)
June	133
July	123
August	94
September	107
October	129
November	129
December	124
Annual	**1,625**

Source: NewFields, 2018

Review of the available data from the Quimsacocha 1 meteorological station (2005 – 2013) showed that wind direction is highly variable and there is no single predominant direction (NewFields, 2018). The mean daily wind speed over the entire measurement period was 4.2 m/s, with the maximum daily wind speed of 11.9 m/s recorded on October 31, 2011. A wind rose for 2012 data showed that winds were recorded as being predominantly from the east-northeast direction (31%) and from east (19%) and northeast (16%). The highest wind speeds were from the easterly direction. During the period of record, winds were higher than 8 m/s (28.8 km/h) 11.4% of the time and calm (less than 0.5 m/s) 8.2% of the time. The wind classes 2-4, 4-6 and 6-8 m/s held 68.1% of the wind frequency.

The mean relative humidity over the entire measurement period of record at the Quimsacocha 1 meteorological station was 91.7%.

Daily data of wind speed, temperature, relative humidity, and solar radiation between August 2005 to October 2015 was used to estimate evaporation (NewFields, 2018). The daily data was used to generate average monthly estimates from both the Penman-Monteith and the Hargreaves-Samani equations. Using an average of values determined by the two (2) methods; average annual evaporation is estimated to be approximately 844 mm with monthly distributions shown in Table 5.2.

Table 5.2 – Mean Monthly Potential Evaporation Estimates at Loma Larga Site

Month	Mean Evaporation (mm)
January	73.7
February	70.2
March	77.8
April	64.7
May	65.3

Month	Mean Evaporation (mm)
June	56.6
July	58.0
August	60.8
September	73.4
October	83.0
November	80.1
December	80.9
Annual	**844.2**

5.5 Local Resources

The Project can attract resources from the surrounding local communities, including San Gerardo, Chumblin, San Fernando, and Victoria del Portete. A baseline study has been completed to understand businesses and individuals that may be interested and/or able to support the Loma Larga (Propraxis, 2018). DPM is incorporating this information and engaging with the local communities as the Project advances.

5.6 Infrastructure

There is currently a minimal amount of infrastructure on the property mainly consisting of several manmade water ponds / reservoirs for storage and treatment of water, a nursery which is used to grow fauna and flora for present and future rehabilitation purposes, and buildings used for the temporary storage of cores. The north-south access road, which has been deemed a public road by the government, extends all the way past the future portal entrance and up to the mine concession above the orebody.

In addition, there is a small camp at Los Pinos that can house currently 30 people including office space and there is an existing 12.7 kV single-phase electrical line power source from the local grid. There is also limited cellular and Wi-Fi service that will be upgraded as the Project advances.

5.7 Physiography

The property of Loma Larga is located within the Western Cordillera of the Andes, which is made up of a series of narrow lands, oriented in a north-easterly direction. The Project is located in the southern part of the Chaucha continental terrain, in the physiographic province of the Western Cordillera. The terrain is composed of volcanic rocks from the Tertiary continental arc deposited on marine sedimentary rocks to fluvial from the Upper Cretaceous, which in turn were deposited on metamorphic rocks in the Paleozoic and Mesozoic bases.

The physiography at the Project consists of desert plains and rugged valleys, mainly formed by glaciers, with an altitude ranging from 3,500 m above msl to 3,960 m above msl.

Vegetation is sparse and typical of the Andean region above tree line. Much of the Property is covered by Andean “paramo”, a type of moorland vegetation consisting mainly of coarse grasses (*Calamagrostis sp.*), Pads (*Plantago sp.*), and upper montane forest. There are stands of pine on hillsides adjacent to the concessions, most of which were planted as part of a forestation project.

6 HISTORY

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

6.1 Prior Ownership, Exploration, and Development History

Exploration activity began in the area in the late 1970s when a United Nations survey identified the Tasqui and Jordanita base metal stream sediment geochemical anomalies 5 km south of the margin of the Quimsacocha caldera.

In 1991, the property was acquired by COGEMA (now Orano), which completed 2,944 m of diamond drilling in 17 holes on vein and disseminated targets. COGEMA entered into a joint venture with Newmont Mining Corporation (Newmont) and TVX Gold Inc. in 1993. Newmont, as field operator, drilled 82 holes totalling 7,581 m. With the average hole being less than 100 m deep, the drill program failed to reach the levels of the Loma Larga deposit. The flat-lying nature of the target mineralisation was not recognised. The best intersection reported was 83 g/t Au and 316 g/t Ag over 2 m and the joint venture was dissolved.

COGEMA entered into an option agreement with IAMGOLD in 1999. However, no work on the property was carried out for several years. In 2004, IAMGOLD resumed exploration activities, drilling a total of 65,117 m across 280 drill holes, leading to the discovery of the Loma Larga deposit in 2004 (IAMGOLD, 2009).

In April 2008, the Constituent Assembly of Ecuador passed a mandate to revise mining laws. The Ecuadorian Ministry of Mines and Petroleum issued a 180-day suspension of all mining and exploration activity while laws were being revised.

In October 2008, a new Ecuadorian Constitution was approved which established that non-renewable resources, such as mineral and petroleum deposits, could only be produced in strict compliance with the environmental principles set forth in the Constitution. Resumption of mining activities was granted by the Ministry of Non-Renewable Natural Resources of Ecuador on February 14, 2011.

Following a competitive bidding process in 2011, INV was selected by IAMGOLD to exclusively negotiate a purchase agreement. On June 20, 2012, INV entered into a share purchase agreement with IAMGOLD and its two (2) subsidiaries, AGEM Ltd. and Repadre Capital (BVI) Inc., to purchase a 100% interest in IAMGOLD Ecuador S.A. The transaction closed on November 14, 2012. The subsidiary was renamed INV Minerales Ecuador S.A in January 2014.

After the acquisition of the Project, INV initiated a 12-hole drill campaign which included seven (7) holes to test step-out targets to extend the deposit, three (3) deeper holes to test for stacked lenses

at depth, and two (2) holes drilled to obtain core samples for metallurgical testing. The program totalled 3,684.7 m.

Exploration activities to date at Loma Larga have been focused within the Rio Falso concession with the objective being drilling, core collection, logging and testing of drill samples to understand and evaluate the potential mineral resource and establish the technical, environmental, and economic conditions that must be considered in the evaluation of a potential mine.

A Technical Report on the Loma Larga Project, Azuay Province, Ecuador, dated August 29, 2016 (2016 Technical Report) was published supporting a pre-feasibility study in 2016 that envisaged operating the mine at a throughput of approximately 3,000 t/d.

Geotechnical, hydrological, and exploration drilling was performed by INV in 2016-2017. A total of 6,978.21 m were drilled in 32 drillholes.

As a result of this drill program, an indication of the presence of multiple feeder zones along the north south length of the deposit were identified.

In 2018, Western Mining Services (WMS) completed a project assessment and target study on Loma Larga (Western Mining Services, 2018). Following a review of the Project geology, geochemistry, and geophysics, WMS identified and prioritised nine (9) targets within INV concessions for future exploration programs.

A Technical Report on the Loma Larga Project, Azuay Province, Ecuador, dated January 11, 2019 (2019 Technical Report) supporting a feasibility study was published on January 14, 2019, that supported the 2016 Technical Report throughput of approximately 3,000 t/d.

A Technical Report on the Loma Larga Project, Azuay Province, Ecuador, dated April 8, 2020 (2020 Technical Report) was filed by INV, supporting its updated feasibility study. The feasibility study was based on an updated LOM plan and mineral reserves, updated costs, contained various layout changes and relocation of the filtered tailings storage facility compared to the 2019 Feasibility Study.

In July 2021, Dundee Precious Metals (DPM), DPM acquired ownership of all the shares of INV, and changed the name of its now wholly owned INV subsidiary to DPM Ecuador Holdings Inc. (DPME) in October 2021. The 2020 Technical Report was readdressed to DPM and published on November 29, 2021, to support its own disclosure. No further changes were made to the 2020 Technical Report beyond addressing it to DPM, inserting references to DPM and other necessary updates where appropriate.

Following its acquisition, DPM initiated a drilling program on the Loma Larga Project in early 2022. Drilling was paused at the end of February 2022 as a result of the Action filed against the Government Ministry of Environment (known as *Ministerio del Ambiente, Agua y Transición Ecológica de la República del Ecuador* (MAATE)) and Ministry of Energy and Mines by certain non-

government organisations and local agencies. Drilling activities remain paused, pending the resolution of the court process.

In 2022 and 2023, limited metallurgical testwork was completed on existing core samples (drilled prior to the pause indicated above) to help provide further process definition. Section 13 contains more details of this testwork.

DPME has completed its EIS by Ecuadorian standards and has obtained final approval in April 2024. The environmental license is subject to DPME's compliance with the court's appeal ruling; by December 2024, the biotic report, water report, and Environmental Consultation in direct influence area of the Loma Larga project were delivered to the court with a 78% acceptance rate.

Environmental License No. MAATE-SUIA-LA-SCA-2024-00001 for the 69 kV electrical transmission line was granted on April 23, 2024.

6.2 Historical Mineral Resource Estimates

INV Metals prepared previous Mineral Resource Estimates on the Project prior to acquisition by DPM in 2021. These estimates have been superseded and not considered significant for the purposes of this Technical Report.

7 GEOLOGICAL SETTING AND MINERALISATION

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

7.1 Regional Geology

Ecuador can be subdivided into a number of distinct physiographic provinces, which broadly coincide with the subdivision of the crust into several terranes (Figure 7.1) (MacDonald et al., 2013).

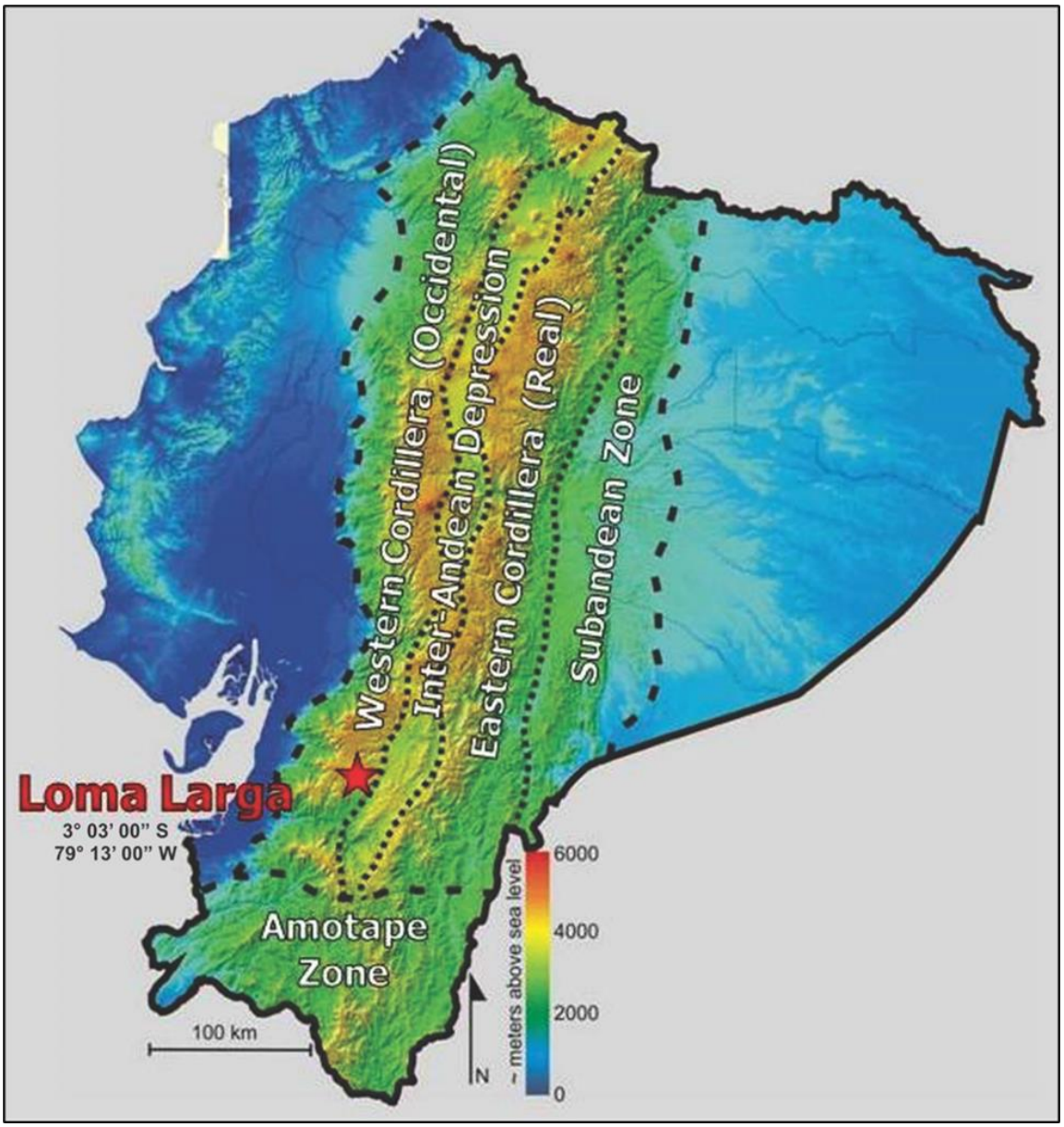
The Loma Larga property is located within the Ecuadorian cordillera, which consists of a number of narrow, north to northeast trending terranes which were formed during the separation of the Central and South American plates and accreted onto the Amazon Craton from the Late Jurassic to Eocene (Chiaradia, 2004). Most of the terranes extend for several hundreds of kilometres in a north-northeast direction and are only a few tens of kilometres wide. These terranes are separated by deep north-northeast trending faults and were created during the Tertiary and Quaternary periods by subduction related continental arc magmatism and reactivation of the terrane bounding faults.

The Project lies in the southern part of the Chaucha continental terrane, in the Western Cordilleran physiographic province, as shown in Figure 7.2.

These fault zones are interpreted to have been active during the entire evolution of the intervening basin. During each reactivation phase, fault movements influenced the location of some intrusive and subvolcanic bodies while some acted as channels for the mineralising hydrothermal fluids.

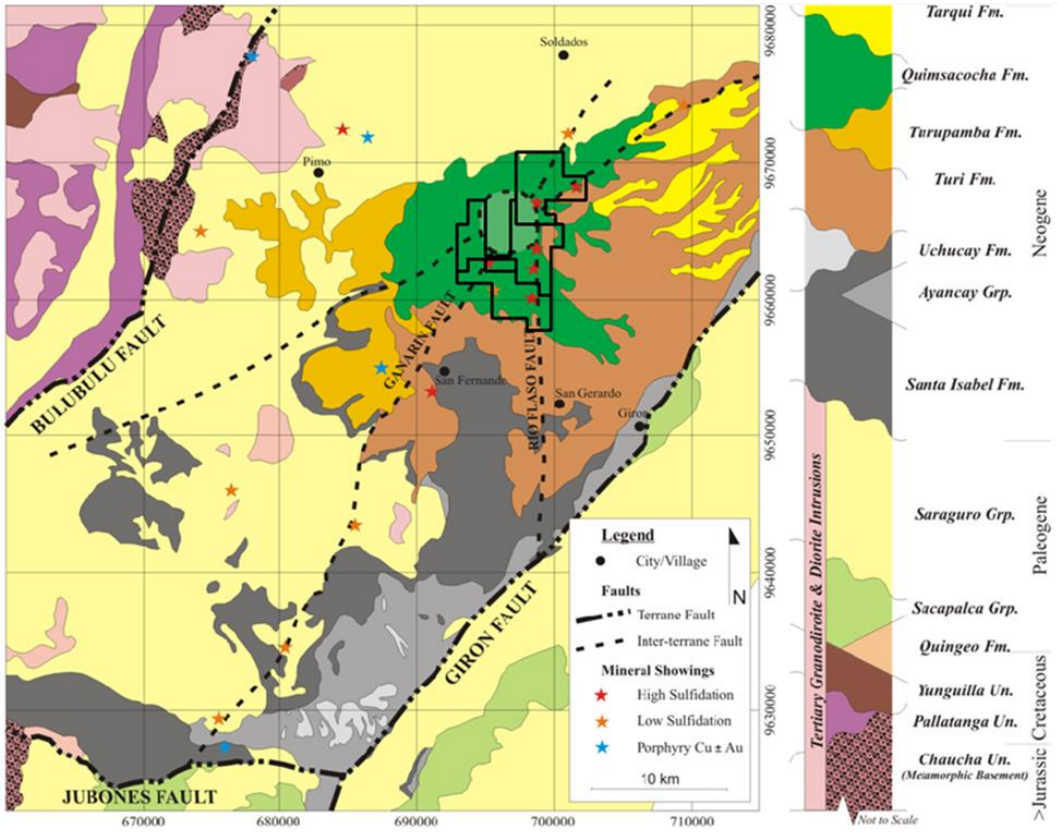
The Chaucha terrane consists of Tertiary continental arc volcanic rocks deposited upon Cretaceous marine to fluvial sedimentary rocks, which in turn were deposited on basement Paleozoic and Mesozoic metamorphic rocks (MacDonald et al., 2010).

Figure 7.1 – Major Terranes of Ecuador



Source: Modified from MacDonald et al., 2012

Figure 7.2 – Regional Geology



Source: Modified from MacDonald et al., 2012

7.2 Local and Property Geology

The Loma Larga property is located between the Gañarin fault to the northwest and the Girón fault to the southeast. A collapsed caldera structure, 4 km in diameter, the remnant of an eroded stratovolcano, lies along (and probably emplaced and controlled by) the Gañarin fault and 400 m west of the main Loma Larga mineralisation. The caldera is underlain by late felsic domes and is cut by a multiphase diatreme. The north-south trending Rio Falso fault, which appears to be a conjugate fault linking the Gañarin and Girón faults, is the locus for alteration and mineralising fluids (Figure 7.2 and Figure 7.3).

The property and immediate surrounding area are mostly underlain by Upper Miocene volcanic and volcaniclastic rocks, of the Turi, Turupamba, Quimsacocha, and Tarqui formations (Scott Wilson RPA, 2006) (Figure 7.2 and Figure 7.3). These formations are flat lying to gently dipping and usually do not outcrop on the property. The outcrops that are exposed form a radial pattern around the caldera and gently dip away from it to the south and east.

The property is largely underlain by the Quimsacocha Formation which hosts the Loma Larga deposit and consists of alternating andesitic banded lava flows with phenocrysts of fresh plagioclase and andesite tuffs and breccias, distributed radially only around the outside of the caldera. The Quimsacocha Formation overlies the Turi Formation which consists of tuffaceous breccias, conglomerates, and sandstones with a high content of andesitic clasts and occasional clasts of tuffaceous breccia.

The Turupamba Formation is composed of rhyolitic to dacitic tuffs with a lesser number of lapilli tuffs and appears to be the result of numerous minor ash falls with periods of fluvial and lacustrine sedimentation. In the immediate surrounding area and bordering the Quimsacocha Formation is the Saraguro Group to the north, the Tarqui Formation to the east, and the Turupamba Formation to west.

The Saraguro Group consists of andesitic to rhyolitic pyroclastics, lava flows and volcaniclastics as well as subordinate rocks.

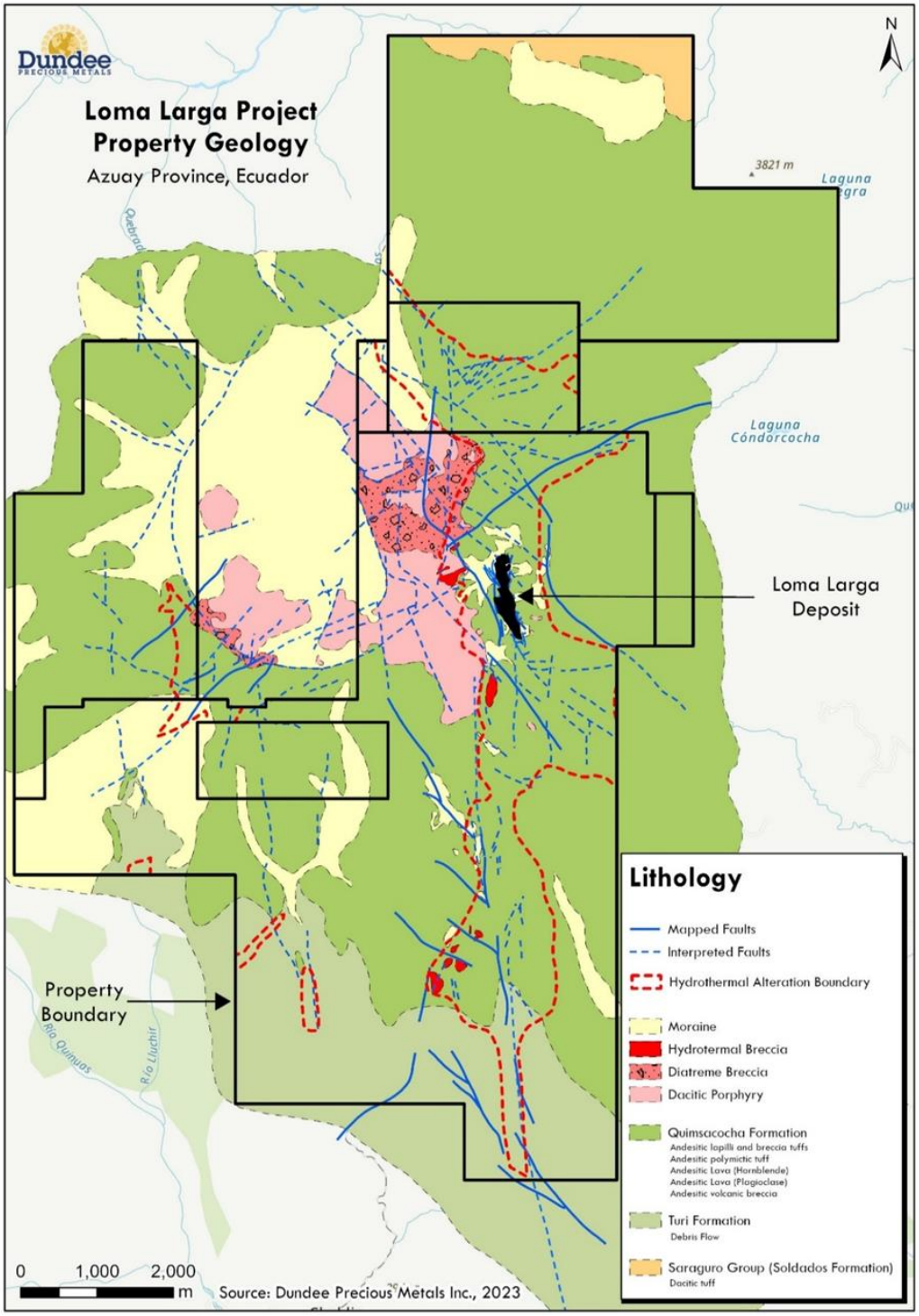
The Turupamba Formation is composed of rhyolitic to dacitic tuffs with a lesser amount of lapilli tuffs. The Turupamba Formation appears to be the result of numerous minor ashes falls with periods of fluvial and lacustrine sedimentation.

The Tarqui Formation overlies the Turi Formation. It unconformably overlies all older formations and has a maximum thickness of 400 m. Near the Loma Larga deposit, the Tarqui Formation mostly consists of strongly weathered quartz phyric rhyolite tuffs. In contrast, further north the unit is mainly composed of thinly banded tuffs, tuffaceous sandstones, and conglomerates. Plant remnants and coal are also common.

Following caldera collapse, a post-mineralisation volcanic intrusive event occurred, resulting in dacitic to rhyolitic domes and quartz-feldspar porphyritic dacite cryptodomes, emplaced into and around the caldera in the Pliocene. Accompanying these lithologies are caldera collapse related breccias and diatreme breccias, which locally contain mineralised clasts.

Much of the surface of the property is composed of Pliocene and Quaternary alluvial debris, glacial moraine, and lacustrine deposits.

Figure 7.3 – Property Geology



7.3 Mineralisation

At Loma Larga, like in most typical high sulphidation epithermal systems, alteration is characterised by multiphase injections of hydrothermal fluids strongly controlled by both structure and stratigraphy. The alteration-mineralising event is characterised by an early alteration phase caused by a strong inflow of volatile, acidic fluids which cooled progressively and were neutralised by their reaction with country rock, leading to the formation of silicified layers surrounded by alteration halos of clay minerals, while the sulphides and gangue minerals associated with the mineralisation were deposited by later fluids inside the silicified bodies (IAMGOLD, 2008).

The majority of the limited amount of outcrop exposed at Loma Larga exhibits silica alteration, due to its resistance to weathering. In epithermal environments the silica alteration displays evidence of hot acidic leaching. Multiple types of silica alteration occur at Loma Larga, including vuggy, sugary, banded, fracture fill, and hydraulic breccia (MacDonald, 2010).

Alteration can be seen to be structurally controlled as it typically occurs as silica ribs mimicking fault locations and orientations. The most significant alteration zone, host to the deposit, is coincident with the north-trending Rio Falso fault, extending for over 8 km north-south, along the eastern edge of the collapsed caldera. This long, linear zone contains multiple large pods of silica alteration ranging up to 2 km in east to west width. The location of the Rio Falso fault suggests that it was coeval with or postdates the caldera collapse (MacDonald, 2010).

The silica alteration is surrounded by varying widths of a halo of argillic alteration, grading from higher to lower temperature mineral assemblages including pyrophyllite, alunite, dickite, kaolinite, illite, and smectite.

The high sulphidation epithermal gold-copper-silver mineralisation in the Loma Larga deposit is also stratigraphically controlled as it occurs at lithological contacts between Quimsacocha Formation andesitic lavas and tuffs and reaches greater thickness in the more permeable tuffs. The deposit is a flat lying to gently western dipping (less than ten degrees), north-south striking, cigar shaped body, which has a strike length of approximately 1,600 m north-south by 120 m to 400 m east-west and up to 60 m thick, beginning approximately 120 m below surface (cross-section in Figure 7.4 and long section in Figure 7.5). It also dips slightly to the north, such that the mineralised zone is closer to surface at the south end. Resources are defined as a smaller, higher-grade subset within this mineralisation.

Mineralised zones are characterised by multiple brecciation and open-space filling events and sulphides such as pyrite, enargite, covellite, chalcopyrite, and luzonite or, at lower sulphidation states, tennantite and tetrahedrite. Higher grade intervals typically coincide with increased amounts of enargite, minor barite, and intense hydraulic brecciation that contains subrounded to rounded silicified fragments. Visible gold is rare. Gold mineralisation is found, for the most part, in one of the following mineralogical assemblages: (a) vuggy silica plus fine-grained pyrite and enargite; (b)

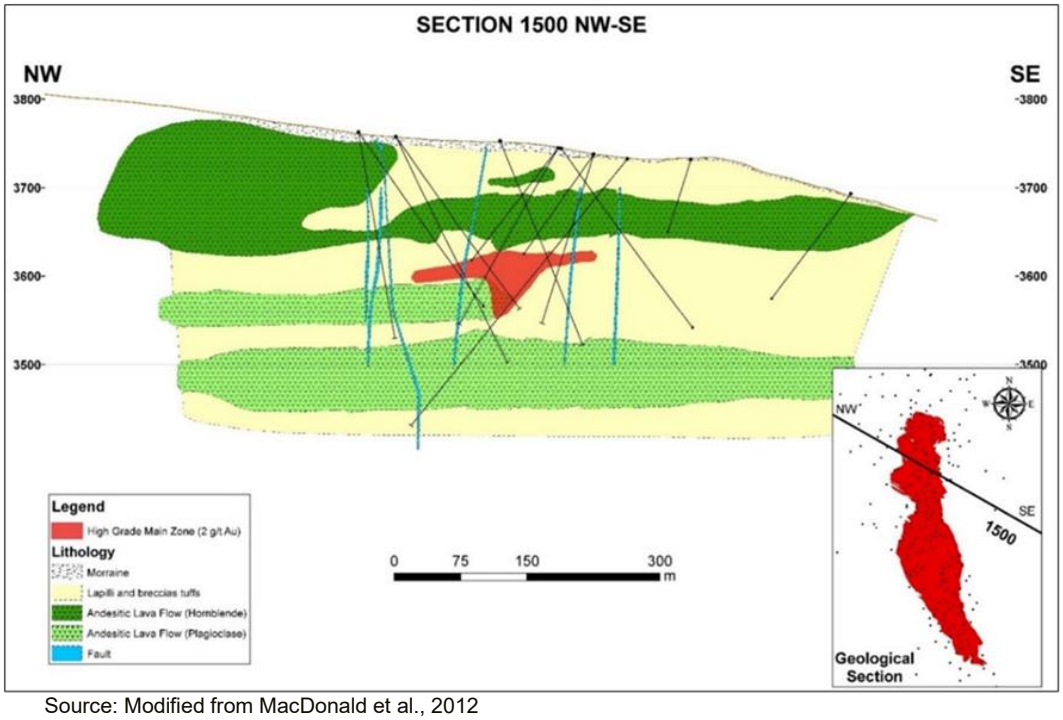
massive pyrite, including a brilliant arsenical pyrite; or (c) vuggy silica with grey silica banding, sulphide space-filling and banded pyrite. Very fine-grained pyrite is dominant in semi-massive to massive zones and is interpreted to have formed earlier than coarser fracture and vug-filling pyrite (MacDonald, 2010).

The focus of mineralisation occurs at approximately 3,610 m (± 30 m) elevation, where structural feeder zone(s) intersected a permeable tuff horizon that was acid leached. There is an upper barren silicic lithocap, locally indicative of steam heated alteration, which is typically barren of mineralisation, although there is an outcrop exposure of this zone that contains minor, fine visible gold (MacDonald, 2010). Silica textures within the upper zone range from sugary, to two-phase, massive, vuggy, and laminated, while the main body centred at 3,610 m exhibits either massive or vuggy silica with intense brecciation in the core and pervasive veinlet and vug infilling alunite alteration. A third lower silicified horizon described below is primarily vuggy in nature (MacDonald, 2010).

Drilling has led to the recognition of a third, deeper zone of residual quartz, below approximately 3,550 m elevation. This horizon is typically 20 m in thickness but is locally thicker. Because these deeper intersections can be correlated between several drillholes and sections, INV has interpreted the zone as a polylithic tuffaceous horizon (as opposed to a vertical feeder); mineralisation is typically 10 to 20 m thick with gold grades of 1.0 g/t to 3.0 g/t. IAMGOLD's drilling on many sections stopped within 10 to 20 m of passing through the main zone, leaving the third horizon and potentially deeper ones virtually untested.

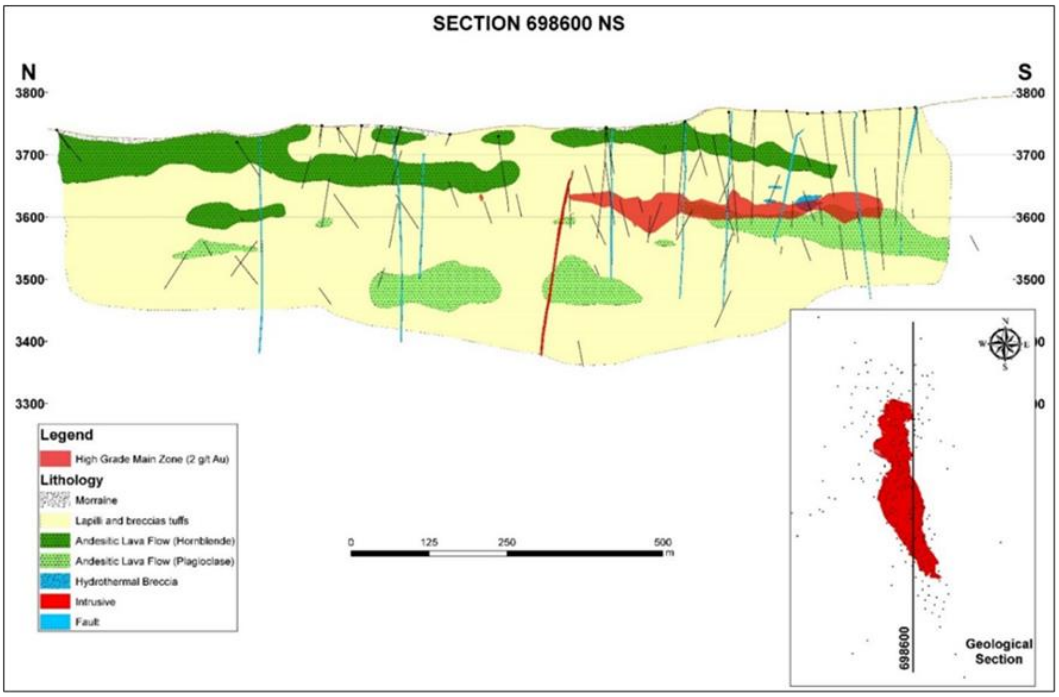
There may be multiple feeder structures along the north-south length of the deposit, possibly associated with north-northeast trending en-echelon structures (Hedenquist, 2013). It is interpreted that at least two (2) vertical to sub-vertical feeder zones occur in the central to eastern part of the deposit, in the vicinity of IAMGOLD's discovery hole IQD-122 (9.2 g/t Au over 102 m). Above this thick, high-grade zone, there is an upper lens of mineralisation that is included in the current resource as inferred based on the limited number of drillholes intersecting it. Significant drillhole intersections in the Loma Larga upper lens include 24.0 g/t Au over 9.0 m in drillhole IQD124 and 8.4 g/t Au over 30.7 m in drillhole IQD152501.

Figure 7.4 – Cross-Section of Loma Larga Deposit, Looking Northeast



Source: Modified from MacDonald et al., 2012

Figure 7.5 – Longitudinal Section of Loma Larga Deposit, Looking East



Source: Modified from MacDonald et al., 2012

The presence of copper mineralised porphyry fragments within the diatreme exposed approximately 1 km to the northwest of the current northern margin of the deposit, along with the zonation of decreasing copper in the deposit from north to south, suggests that the causative intrusion that drove the hydrothermal system is located to the north, and that fluid flow was from north to south. This is consistent with observation of chalcocite and chalcopyrite associated with stockworks of veinlets at depth in the north (Hedenquist, 2013). Evidence for a deeper porphyry copper-gold deposit is suggested by the following observations:

- Quartz veinlets with centre lines and margins of chalcocite to the north (IQD-109, approximately 317 m to 343 m at 2.02 g/t Au and 0.14% Cu as chalcocite; as values of less than 100 ppm).
- High Cu/As ratios to the north, i.e., enargite and tennantite poor despite Cu values greater than 0.1 wt%, suggesting the presence of chalcocite and/or chalcopyrite.
- Broad illite-pyrite alteration with disseminated magnetite plus pyrite ± magnetite veinlets associated with a fragmental intrusion.
- The diatreme to the northwest, on the eastern margin of the caldera, which contains quartz-feldspar porphyry fragments plus potassic and phyllic-altered clasts with anomalous copper (up to 0.1% Cu) and gold (up to 0.5 g/t Au), associated with quartz ± magnetite ± pyrite veinlets with disseminated magnetite, chalcopyrite, chalcocite, covellite, and molybdenite (Hedenquist, 2013).

There are multiple exploration targets. For example, INV's 2013 drillhole LLD-367 intersected 4.9 g/t Au, 48.7 g/t Ag, and 0.51% Cu over a core length of 25.1 m, including 11.9 g/t Au, 78.7 g/t Ag, and 0.33% Cu over 6.2 m. This intersection is located approximately 165 m north of the northern limits of the current Loma Larga Mineral Resource. Additional drilling completed in 2016 and 2017 investigated the extent of mineralisation north and west of the currently defined resource at Loma Larga.

In 2018, Western Mining Services (WMS) completed a project assessment and target study on Loma Larga (Western Mining Services, 2018). Following a review of the Project geology, geochemistry, and geophysics, WMS identified and prioritised nine (9) targets within INV concessions. The targets include eight (8) high sulphidation style Au-Cu targets to the north, east, west, and south of Loma Larga, and one (1) low sulphidation style Au-Ag target to the west of the orebody. WMS also identified the opportunity to discover a porphyry style Au-Cu deposit at depth to the east of the Quimsacocha crater, proximate to the Loma Larga deposit. Further to the study, WMS provided rough estimates of the size of the potential discoveries within each target area and provided recommendations to better define targets and improve exploration effectiveness.

The isolated gold- and coper-rich drillhole intersections between 3,550 m and 3,500 m elevation constitute high grade mineralisation that may be hosted within a structurally related feeder zone, as

at shallower depths, rather than being associated with the lower silicic horizon. Further north, similar high-grade intervals that have been intersected at approximately 3,600 m elevation have yet to be tested at depth for possible structural control.

In addition to the deeper high-grade zones that have yet to be tested for their potential depth extent, there are also shallower high-grade zones that have not yet been tested. It is anticipated that drilling scissor holes would define their potential depth extent and determine if the high-grades are hosted in structurally related feeder zones. This includes, for example, IQD-323 with 2.7 m of approximately 77 g/t Au, 89 g/t Ag, and 7.2% Cu at approximately 3,630 m elevation, at the base of the silicic zone.

In 2017, INV completed several drillholes along the western margin of the Loma Larga deposit. Further drilling is however recommended to define and constrain the margins of the mineralisation, which remains open. In addition, deeper drilling is needed to define a horizon at approximately 3,550 m elevation.

The indications of porphyry potential at depth in the north, including veinlet stockwork with copper sulphides, is consistent with the lithocap alteration and high sulphidation-state sulphides of Loma Larga; the location of the intrusive centre of the porphyry and its depth remain to be determined by further study, and eventual drill testing.

South of the southern extent of the current Loma Larga Mineral Resource there is a silver-rich zone with drillhole IQD-265 intersecting 2.4 m of 3,180 g/t Ag, 0.2 g/t Au, and 0.1% Cu, near the base of the silicic zone, under the main mineralised horizon. The silver-to-gold ratio for the resources is approximately 10 to 1, or less, whereas the silver-rich intervals in IQD-263 and 265 are two hundred-to-one, or higher. Sections to the north of the currently defined deposit also have high silver-to-gold ratio intersections. For example, drillhole IQD-269 intersects four (4) metres of 896 g/t Ag and 1.3 g/t Au, and drillhole IQD-269 intersects five (5) metres of 566 g/t Ag and 3.8 g/t Au. The two (2) drillholes intersect 0.3% Cu and 0.6% Cu, respectively. Given the uncertain mineralogy responsible for the high silver-to-gold ratio and resulting high-grades, INV commissioned Inspectorate Services Peru S.A. to carry out a mineralogical study using optical microscopy and electronic scanning. This study determined that the silver species present include silver sulphides (argentite - acanthite), sulfosalts (AgCuAs and CuAgAs) and sulphides (AgAs).

8 DEPOSIT TYPES

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

Epithermal deposits are defined as precious and base metal deposits that formed at depths of less than 2 km and at temperatures of less than 300°C in subaerial environments within volcanic arcs at convergent plate margins and in intra-arc and back-arc as well as post-collisional extensional settings (Robert et al., 2007). A hydrothermal system consisting of fluids derived at depth will undergo abrupt physical and chemical changes at the shallow depths where epithermal deposits form. These changes occur because of the change from lithostatic to hydrodynamic pressure, which results in boiling, interaction of the fluids with near-surface water, permeability changes, and reaction between the fluids and host rocks. These changes affect the capacity of the hydrothermal fluid to transport metals in solution. This decrease in the solubility of metals in the fluids results in the metals being deposited within a restricted space as a result of focussing of the fluid flow near the surface (White and Hedenquist, 1990).

Epithermal deposits are typically classified as either high- or low-sulphidation based upon the sulphidation state of the ore minerals; as enargite and luzonite for high sulphidation and chalcopyrite-galena-sphalerite for low sulphidation. The term high-sulphidation is frequently misused to suggest a high sulphide or high sulphur deposit, whereas it in fact refers to the chemical state of the metals (the sulphur to metal ratio).

A schematic section showing alteration zonation through a high sulphidation gold deposit environment is provided in Figure 8.1. High sulphidation epithermal copper-gold-silver deposits develop in settings where volatiles (dominantly gases such as SO_2, HF, and HCl) and metal bearing fluids vent from hot magma sources at considerable depth and travel rapidly to elevated crustal settings, without reaction with wall rocks, or mixing with groundwater. The volatile component, which rises more rapidly than the fluids, becomes progressively depressurised and SO_2 in particular comes out of solution and in turn oxidises to form H_2SO_4, such that the rising and cooling fluid becomes increasingly acidic (to pH of 1.0 to 2.0) as it ascends to epithermal levels, where it reacts with wall rocks to produce advanced argillic alteration. Because of the progressive cooling and neutralisation of the hot acid fluid by wall rock reaction, the advanced argillic alteration is zoned outwards from a central core of vuggy or residual silica, from which everything but silica has been leached by the strongly acidic waters, through alteration zones dominated by alunite, pyrophyllite, dickite, kaolin, and then illite (Figure 8.1) (Corbett, 2005). Use of a portable infrared mineral analyser (PIMA) has been critical in a number of discoveries, including Loma Larga, as it allows one to vector towards mineralisation based on the zonation of the various clay alteration minerals.

The shape and intensity of alteration vary according to crustal level and permeability of the host rocks. At Loma Larga the acidic fluids preferentially altered the more permeable andesitic tuffs,

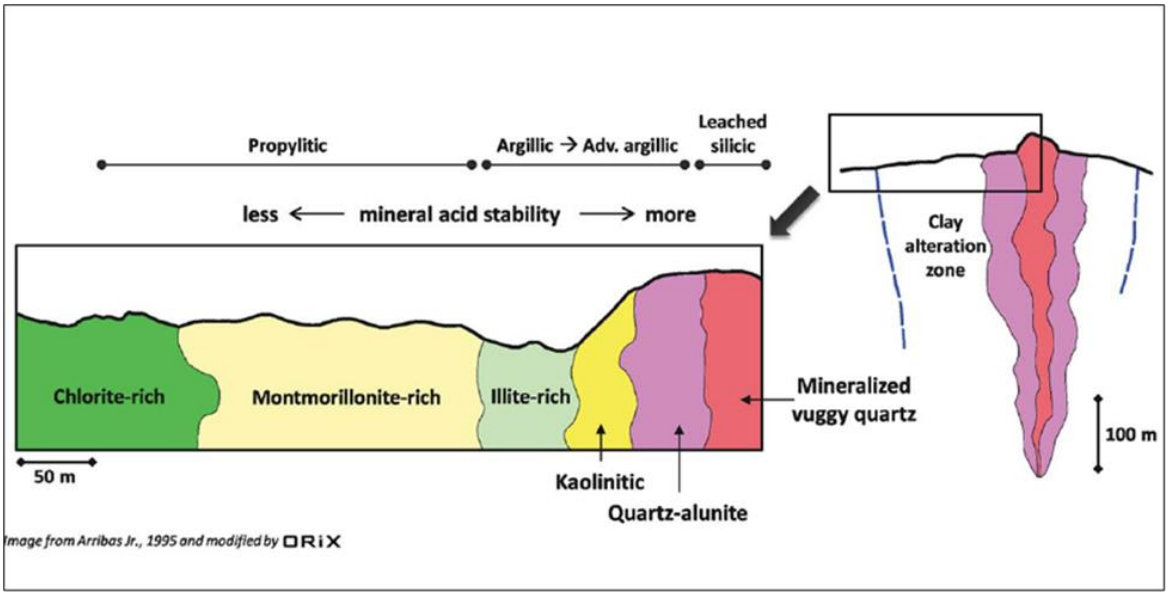
sandwiched between more resistant and less altered andesitic lavas. The liquid rich phase of the high sulphidation hydrothermal fluid generally follows the volatile rich portion. Most copper-gold mineralisation deposition, usually as sulphide breccia infill of competent vuggy silica and silica-alunite altered clasts, typically is associated with pyrite-enargite (and its low temperature polymorph luzonite) and lesser covellite (at deeper levels) and local, generally peripheral, tennantite-tetrahedrite, as well as barite and alunite, and post-dates the alteration.

The hot magma source for the hydrothermal system is frequently a copper-gold porphyry (Figure 8.2). Approximately half of the major high sulphidation ore deposits around the world have evidence for underlying porphyry mineralisation, based on observations of deeply eroded systems (e.g., Minas Conga, Peru; Halilaga, Turkey), deep drilling (e.g., Far Southeast, Philippines), and porphyry altered and mineralised clasts contained within diatremes (numerous systems, including Loma Larga) (Hedenquist, 2013).

The Far Southeast porphyry deposit located under and adjacent to the Lepanto high sulphidation deposit was discovered as a follow-up to the presence of copper-bearing porphyry fragments within a diatreme breccia exposed at surface. A similar situation occurs at Loma Larga, with a diatreme to the north and west of the gold deposit containing copper-mineralised, altered porphyry fragments, suggesting that at some unknown depth a copper ± gold porphyry may have driven the hydrothermal system responsible for forming Loma Larga.

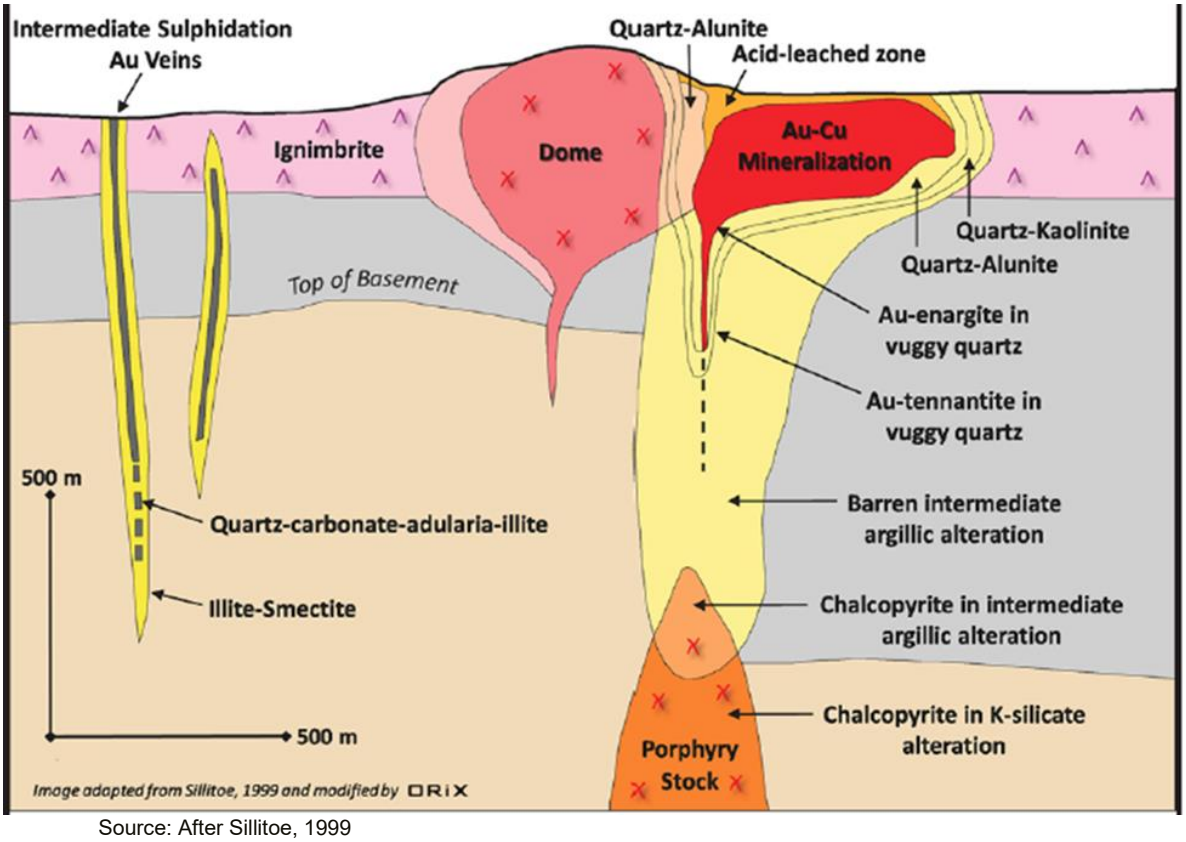
High sulphidation deposits are the major gold producers in the Andes (e.g., Yanacocha, Pierina, El Indio, La Coipa, Veladero), and represent some significant undeveloped resources (e.g., Pascua-Lama and Loma Larga). Other global examples include Lepanto, Philippines; Pueblo Viejo, Dominican Republic; Mulatos, Mexico; Paradise Peak, Nevada; and Chelopech, Bulgaria.

Figure 8.1 – Zonation of Alteration in High Sulphidation Deposit



Source: After Arribas Jr., 1995

Figure 8.2 – Schematic Section of a High Sulphidation Deposit



Source: After Sillitoe, 1999

9 EXPLORATION

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

Exploration activities by all previous owners are discussed in Section 6.

INV carried out exploration activities on the property in 2017 and 2018. These are summarised below.

- Contracted Geoscience North Ltd. to organise and analyse geophysical data within the Rio Falso concession. Data sets from Newmont and Val D'Or Sud America were evaluated and a report was produced outlining potential exploration targets (Geosciences North, 2018).
- WMS was contracted to complete a project assessment and target study on the Project. WMS analysed diamond drillhole data, geological maps, and geophysical data to develop a targeting matrix for the Loma Larga exploration program. A final report was provided in June 2018 (Western Mining Services, 2018).
- Together with Geomingold Cia. Ltda. of Ecuador, the regional geology map for the Loma Larga concessions was updated during June of 2018. The mapping targeted areas with no previous geological mapping along the concession boundaries and validated areas of previous mapping.

No exploration work has been conducted by or on behalf of DPM on the Project.

10 DRILLING

10.1 IAMGOLD Drilling

In 2002, IAMGOLD initiated diamond drilling programs based on the exploration information accumulated by previous operators as well as their own follow-up exploration programs. The Loma Larga deposit was discovered as a result of the 2004 drilling campaign, during which a total of 13,930 m was drilled in 45 holes. Four (4) of the holes were drilled in the Loma Larga Zone, and returned encouraging results, including one (1) sample of 214 g/t Au. From October 2005 to August 2006, IAMGOLD drilled an additional 24,542 m in 125 holes, of which approximately half was infill drilling, and the remainder was undertaken to test for a possible extension.

IAMGOLD contracted two (2) companies: Kluane and Paragon, to carry out drilling. All work was done by diamond core drilling and the core diameters used were HQ (63.5 mm) and NQ (47.6 mm) for Paragon and NT (56 mm) and BT (42 mm) for Kluane. All collars were surveyed. Most of the holes were drilled at 55° to -65° but ranged from -45° to -90°. All core obtained from the drilling was placed in wooden boxes and transferred to the logging room at the field camp for logging and sampling for assay. Core diameter depended on the drill type used by each company.

From 2004 to December 2007, IAMGOLD drilled a total of 65,117 m in 280 holes (IAMGOLD, 2009).

No drilling was carried out between 2008 and 2012.

10.2 INV Drilling

10.2.1 2013 Drilling Campaign

After acquiring the Project in 2012, INV contracted Orix Geosciences Inc. (Orix) to carry out a detailed 3D compilation and reinterpretation of the deposit, with the goals of furthering the understanding of the deposit and identifying drill targets. Geoscience North was engaged to review and compile all previously collected geophysical data, which was then integrated into the Orix compilation. INV's drill program in 2013 included 12 diamond drillholes totalling 3,684.7 m, including two (2) holes drilled for metallurgical testwork, three (3) holes to further define the High-Grade Main Zone, and seven (7) holes to test step-out targets to extend the deposit.

Holes LLD-371 and LLD-372 were drilled to obtain material for metallurgical testwork. The metallurgical drilling in the high-grade zone clearly demonstrates that the Loma Larga gold deposit contains a high-grade core surrounded by a lower grade halo. For example, the drillhole LLD-371 intersected 77 m grading 13.7 g/t Au, including 20 m grading 34 g/t Au, while LLD-372 intersected 36.5 m grading 11.8 g/t Au including 6.3 m grading 47.7 g/t Au. Holes LLD-373 to LLD-375 were drilled to better define the margins of the high-grade zone.

The results for these holes are provided in Table 10.1.

Table 10.1 – Metallurgical and High-Grade Zone Drill Results

Hole		From (m)	To (m)	Width (m)	True Width* (m)	Au (g/t)	Ag (g/t)	Cu (%)
LLD-371		105.00	182.00	77.00	77.00	13.65	40.15	0.52
	including	132.00	152.00	20.00	20.00	33.95	98.84	1.23
	and	158.00	163.00	5.00	5.00	12.21	24.96	0.36
LLD-372		142.50	179.00	36.50	36.50	11.80	79.57	0.93
	including	170.00	176.30	6.30	6.30	47.72	217.32	3.32
LLD-373		48.50	54.86	6.36	6.36	1.66	8.54	0.12
		166.50	171.00	4.50	4.50	10.36	57.30	2.90
	including	167.50	168.60	1.10	1.10	38.20	214.90	11.21
LLD-374		53.40	69.00	15.60	15.60	2.73	15.69	0.77
	including	56.40	60.40	4.00	4.00	4.65	30.48	1.29
	and	65.00	69.00	4.00	4.00	4.46	5.59	1.00
		133.50	146.00	12.50	12.50	2.81	15.58	0.24
	including	133.50	137.00	3.50	3.50	5.14	34.00	0.50
	and	143.80	146.00	2.20	2.20	5.01	14.44	0.43
LLD-375		124.00	162.00	38.00	37.90	3.27	55.65	0.23
	including	155.00	162.00	7.00	7.00	6.83	131.21	0.19
	and	128.00	131.00	3.00	3.00	6.60	48.10	0.14

* True Width is an estimation that considers both the dip of the borehole and the interpreted dip of the mineralised zone of 10 to the west.

10.2.2 2017 Drilling Campaign

INV completed nine (9) geotechnical and nine (9) hydrogeological drillholes on the Loma Larga deposit to obtain data for modelling (Table 10.2). Among the nine (9) hydrogeological drillholes, five (5) of them were for dual purposes: hydrogeological and geotechnical investigation. Various samples of the core from these holes were used for the metallurgical and geotechnical testwork programs as part of this Report. Geotechnical samples included strength testing (i.e., unconfined compressive strength, triaxial, and tensile) and unconfined swelling tests. The location of the drillholes is shown in Figures 10.1 and 10.2, and Table 10.3 summarises the drilling and assay sample results.

Table 10.2 – Geotechnical and Hydrogeological Drillholes Completed in 2016 – 2017

Drillhole	Hole Type	Easting (m)	Northing (m)	Elevation (m)	Azimuth (deg)	Dip (deg)	Length (m)	Nº of Samples
LLDGT-001	GT	697,787	9,663,472	3,804	180	-60	50.29	-
LLDGT-002	GT	698,032	9,663,701	3,781	75	-60	138.90	-
LLDGT-003	GT	698,012	9,663,522	3,781	360	-70	216.27	14
LLDGT-003B	HG/GT	697,956	9,663,489	3,784	60	-80	127.00	16
LLDGT-003C	HG/GT	697,953	9,663,487	3,785	240	-90	134.20	54
LLDGT-004	GT	698,111	9,663,590	3,777	90	-60	281.95	57
LLDGT-005	HG/GT	698,623	9,663,566	3,747	270	-70	220.70	74
LLDGT-006	HG/GT	698,455	9,663,937	3,750	45	-60	231.04	32
LLDGT-007	HG/GT	698,599	9,663,372	3,770	225	-70	213.30	88
LLDHG-008	HG	698,576	9,663,764	3,746	225	-60	231.30	135
LLDHG-009	HG	699,149	9,663,312	3,628	0	-90	22.86	-
LLDHG-009A	HG	699,151	9,663,315	3,627	0	-90	12.50	-
LLDHG-010	HG	698,854	9,663,921	3,666	0	-90	15.24	-
LLDHG-010A	HG	698,619	9,663,915	3,731	0	-90	2.20	-
LLDHG-011	HG	698,582	9,663,461	3,767	300	-74	208.79	120
LLDHG-012	HG	697,888	9,663,357	3,798	70	-80	109.73	-
LLDHG-013	HG	697,885	9,663,355	3,798	250	-75	156.98	-
LLDHG-014	HG	697,761	9,663,096	3,795	270	-90	60.85	-
*GT = Geotechnical; HG = Hydrogeological						**Total**	**2,410.8**	**590**

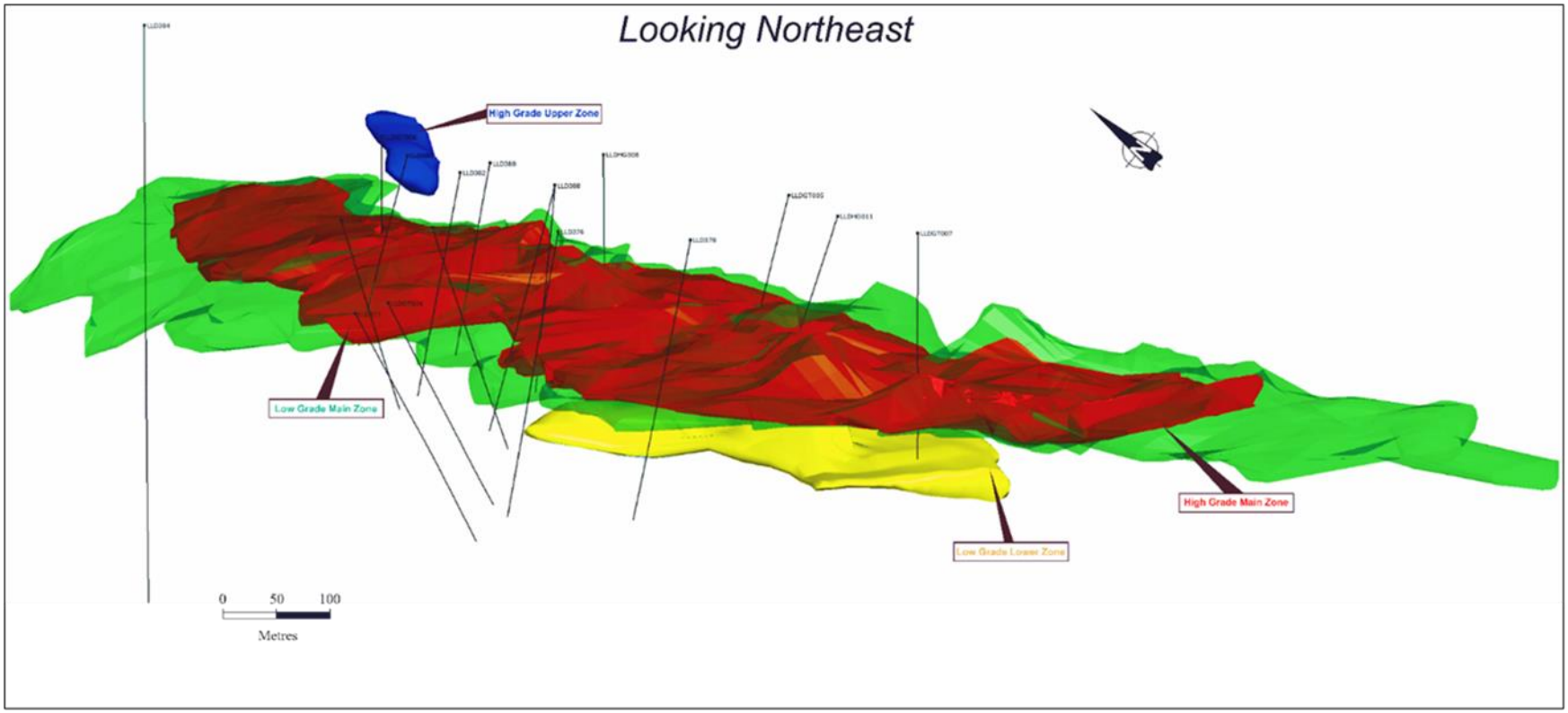
Table 10.3 – Selected Drill Results from the 2017 Geotechnical and Hydrogeological Drilling

Hole		From (m)	To (m)	Width (m)	True Width* (m)	Au (g/t)	Ag (g/t)
LLDGT-004		223.50	243.00	19.50	16.90	2.17	52.5
	including	223.50	238.00	14.50	12.60	2.63	51.5
LLDGT-007		71.45	77.50	6.05	5.70	1.24	1.5
		131.24	183.23	51.99	48.90	3.25	27.6
	including	131.24	153.00	21.76	20.40	4.20	11.8
	and	163.00	170.00	7.00	6.60	4.88	53.7
LLDGT-008		60.50	71.50	11.00	9.50	7.27	91.8
	including	64.50	67.50	3.00	2.60	12.97	155.8
		92.80	173.00	80.20	69.50	3.28	13.6
LLDGT-011		120.00	161.00	41.00	39.40	4.72	14.0
	including	141.00	149.00	8.00	7.70	8.68	21.9
		173.00	196.00	23.00	22.10	3.00	12.8
	including	189.00	196.00	7.00	6.70	4.82	18.1

* True Widths determinations are estimated at 84-96% of the reported core length intervals for most of the holes, estimated sectionally based on the current alteration zone interpretation.

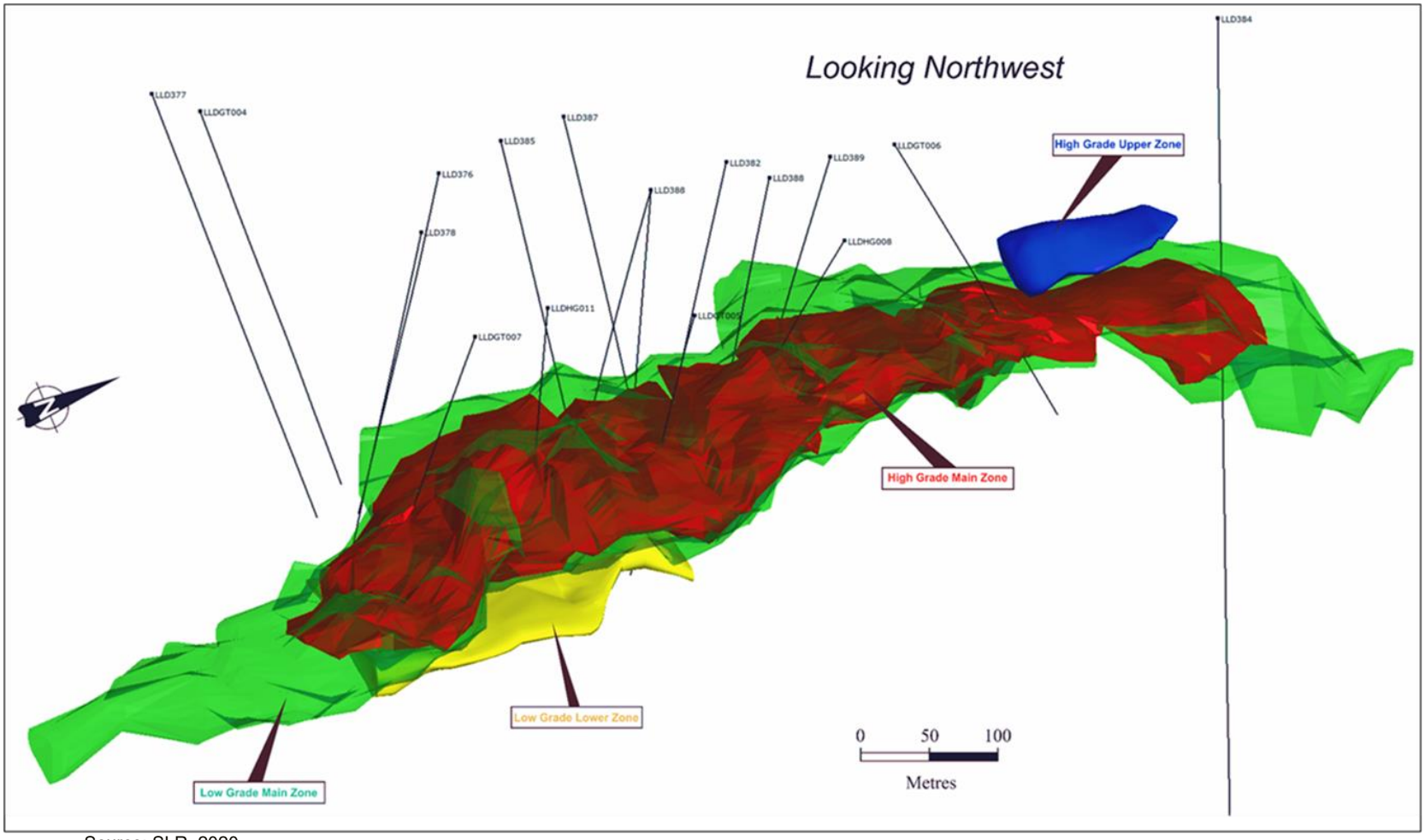
In 2017, INV initiated an exploration program at Loma Larga to evaluate the exploration potential of the property and further delineate gold mineralisation within silica alteration intersected in 2017, during geotechnical and hydrogeological drilling program. The 14-hole, 4,318 m drill program commenced in April and was completed in August of 2017. Mineralisation was intersected to the west of the delineated resource at Loma Larga. Tables 10.4 and 10.5 summarise the drillhole program and results, and the location of the drillholes are shown in Figures 10.1 and 10.2.

Figure 10.1 – Location of INV 2016-2017 Drillholes in 3D Isometric View



Source: SLR, 2020

Figure 10.2 – Location of 2016-2017 Loma Larga Drillholes in 3D Isometric View



Source: SLR, 2020

Table 10.4 – Summary of 2017 Loma Larga Drill Program

Drillhole	Easting (m)	Northing (m)	Elevation (m)	Azimuth (deg)	Dip (deg)	Length (m)	Nº of Samples
LLD-376	698343	9663594	3,782	275	-75	277.4	128
LLD-377	698065	9663588	3,779	90	-60	321.6	89
LLD-378	698422	9663497	3,779	270	-75	274.3	60
LLD-379	697838	9663499	3,794	90	-70	397.8	171
LLD-380	697772	9663497	3,799	90	-80	320.0	86
LLD-381	698027	9663510	3,783	90	-68	285.0	89
LLD-382	698422	9663802	3,761	270	-75	217.9	79
LLD-383	698447	9663701	3,769	270	-70	242.3	119
LLD-384	698460	9664256	3,772	0	-90	1,160.2	607
LLD-385	698291	9663709	3,769	90	-70	266.7	148
LLD-386	698447	9663701	3,769	270	-85	205.7	69
LLD-387	698262	9663801	3,758	90	-70	225.6	61
LLD-388	698464	9663803	3,758	270	-75	187.5	52
LLD-389	698439	9663889	3,749	270	-70	185.9	75
					Total	**4,567.9**	**1,833**

Highlights of the 2017 drill program include a deep hole (LLD-384) drilled to a depth of 1,160 m to follow up on historical drillholes which indicated the potential for porphyry style mineralisation outside of the mineral resource to the north of the known deposit. LLD-384 returned encouraging results including 0.2 g/t Au and 523 ppm Cu over 622.22 m (from a depth of 538 m), including 0.29 g/t Au and 416 ppm Cu over 214.5 m (Table 10.5).

Table 10.5 – 2017 Selected Loma Larga Drill Program Results

Hole		From (m)	To (m)	Length (m)	True Width (m)	Au (g/t)	Ag (g/t)
LLD-376		187.00	196.00	9.00	8.70	3.25	32.3
	including	192.00	196.00	4.00	3.90	5.27	64.6
LLD-377		226.50	254.50	28.00	24.20	2.35	29.6
	including	230.50	236.50	6.00	5.20	3.44	16.2
LLD-378		170.00	187.00	17.00	16.40	1.65	30.6
LLD-381		249.20	252.80	3.60	1.58	2.45	28.9
LLD-382		167.50	186.50	19.00	18.40	2.66	57.1
	including	171.50	180.50	9.00	8.70	3.25	91.0

Hole		From (m)	To (m)	Length (m)	True Width (m)	Au (g/t)	Ag (g/t)
LLD-383		170.00	221.25	51.25	48.15	3.22	48.7
	including	179.00	217.00	38.00	35.70	2.53	31.6
	and	213.00	217.00	4.00	3.80	3.36	65.8
		218.50	221.25	2.75	1.74	20.24	434.4
LLD-384		538.00	1,160.22	622.22	*	0.20	0.83
(Porphyry)	including	720.50	935.00	214.50	*	0.29	0.8
LLD-385		161.00	214.00	53.00	49.80	1.59	25.5
	including	162.00	177.00	15.00	14.10	1.83	11.9
	and	193.00	201.00	8.00	7.50	2.69	54.6
		244.00	262.00	18.00	16.90	2.28	38.8
	including	250.00	252.70	2.70	1.70	8.75	211.2
LLD-386		152.00	189.35	37.35	37.20	4.89	52.2
	including	162.00	189.35	27.35	27.20	5.89	67.4
	and	174.00	181.00	7.00	6.97	9.39	72.0
LLD-387		137.00	197.20	60.20	56.60	0.83	26.0
	including	149.00	153.00	4.00	3.80	1.44	4.5
	and	186.00	197.20	11.20	10.50	1.27	64.0
	and	192.00	197.20	5.20	4.90	1.65	94.5
LLD-388		126.00	161.00	35.00	33.80	2.32	20.3
	including	126.00	137.00	11.00	10.60	3.36	14.0
	and	133.00	137.00	4.00	3.90	5.31	24.9
LLD-389		153.00	172.00	19.00	17.9	1.81	29.7
	including	165.00	172.00	7.00	6.6	2.24	32.2

* Note: True widths determinations are estimated at 84-100% of the reported core length intervals for most of the holes, estimated sectionally based on the current alteration zone interpretation. The true width of Hole LLD-384 cannot be determined at this time.

10.3 DPM Drilling

10.3.1 2022 Drilling Program

During 2022, DPM initiated a diamond drilling program to support geotechnical, hydrogeological, and metallurgical investigations. The total planned amount of drilling was 18,970 m, distributed over planned site infrastructure, tailings management facilities and in the vicinity of the Loma Larga Mineral Resource footprint.

This drilling program was suspended on February 25, 2022. A total of 670.1 m of condemnation drilling was completed prior to DPM temporarily pausing drilling activities at the end of February as a result of the Action filed against the MAATE by certain non-government organisations and local agencies. Drilling activities remain paused, pending the resolution of the court process. From the 670.1 m of drilling completed during 2022, no discernible mineralisation was encountered.

Additional diamond drill holes have been planned to test the full extent of mineralisation and to upgrade resource classification.

10.4 QP's Opinion

In the QP's opinion, the drilling, logging, and drill core and channel sampling procedures meet industry standards. The QP is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

As part of the September 2018 Mineral Resource Estimate, RPA (now SLR) compiled and reviewed all of the Loma Larga Project Quality Control (QC) sample results for INV's 2016-2017 drilling campaigns. SLR reviewed the QC sample results from the Loma Larga 2013 drill campaign and also completed a procedural and statistical review of all historical QC data on the Project.

The current Mineral Resource estimate incorporates drilling completed up to the end of August 2017. No further drilling has been completed since this date. The QP compiled and reviewed the QC sample results for the 2017 exploration, geotechnical, and hydrogeological drilling programs.

In the QP's opinion, the results of the QC samples, together with the Quality Assurance/Quality Control (QA/QC) procedures implemented at Loma Larga, provide adequate confidence in the data collection and processing, and the assay data is suitable for Mineral Resource Estimate.

11.1 Pre-2012 Programs

No information is available on the procedures utilised for the drill campaigns conducted by Newmont (1992-1993) and COGEMA (now Orano) (1994-1996); and assay results from those programs are not included in the current Mineral Resource.

From 2002 to 2008, sample preparation, analysis, and security on the Loma Larga Project were conducted by IAMGOLD. This work is summarised in this section from IAMGOLD (2009) and RPA (2012).

No drilling or sampling occurred on the Loma Larga Project between 2008 and 2012.

11.1.1 Sample Collection

IAMGOLD collected samples for geochemical analyses primarily in mineralised zones. Geological criteria (feeders, hydrothermal breccias, mineralisation styles, etc.) guided sample length, to a maximum of 2 m. For intervals where core loss was recorded, sample length may be more than 2 m.

After the drill core was logged, boxes tagged, and samples identified, all selected samples were cut in half with a diamond saw. One half was sent for geochemical analysis; the other returned to the box and later transferred to a secure warehouse at the IAMGOLD exploration office in the city of Cuenca, Ecuador. Care was taken to prevent contamination, accidental swapping, or loss during crating and transportation of the core. IAMGOLD personnel transported the core from the Project to Cuenca and a shipping contractor was used to transport the core on to Quito.

Sample collection was completed at the Project camp using the following methodology:

- Core was marked every metre from the collar down to the bottom of the hole to check and correct driller's metre marks.
- For each hole, a digital file was created, and all core boxes were photographed and identified by the "FROM" and "TO" depths.
- Geotechnical logging followed by geological logging was carried out and then saved in a dedicated database.
- The core was sampled by cutting in half using a Boart diamond saw with an eight-inch blade.
- Samples were tagged for geochemical, metallurgical, and geotechnical tests.
- Samples were crated and shipped to the IAMGOLD Cuenca regional office where they were numbered, and QC samples (standards and blanks) were placed into the sample stream at predetermined frequencies.
- Four (4) to five (5) sample bags were grouped into rice bags that were sealed with tie-wraps and shipped to the assay laboratory. A copy of the sample checklist held inside each rice bag was also forwarded to the assay laboratory. A laboratory employee cross-checked the list upon reception and faxed the list to IAMGOLD's Quito office where it was reviewed for inconsistencies.
- All pulps and rejects were kept at the laboratory's yard. Only check assay pulps and rejects were discarded.

11.1.2 Density Sampling

To estimate rock densities at the Project, IAMGOLD took representative samples of typical lithologies, alteration, and mineralisation types. Samples were approximately 3 cm in length, and density was determined by the Archimedes principle.

IAMGOLD's density sampling procedure consisted of:

- Identifying a representative zone (lithology and grade) and selecting a 3 cm sample of core.
- Weighing the sample dry.
- Immersing the sample in melted paraffin.
- Weighing the sample and paraffin.
- Weighing the sample and paraffin in water.

IAMGOLD compiled a database of 10,793 density measurements (10,153 at Loma Larga), of which 7,767 were used for the current Mineral Resource estimation (see Section 14).

In the opinion of the QP, the density measurements are suitable for use in the Mineral Resource estimate.

11.1.3 Sample Preparation and Assay Protocols

All samples were sent to internationally recognised and independent laboratories for preparation and testing. Prior to September 2004, samples were prepared in Quito by ALS Chemex and analysed by ALS Chemex laboratory in North Vancouver, Canada. IAMGOLD's system of check samples identified inconsistent results for ALS Chemex and from October 2004 onward (to the end of drilling by IAMGOLD by 2008); samples were prepared by Inspectorate del Ecuador S.A. in Quito and analysed by BSI Laboratories in Lima, Peru (BSI). Both analytical laboratories are accredited to ISO/IEC 17025 for specific registered test and certified to ISO 9001 standards.

The sample preparation and assay procedures are summarised below.

- Samples were dried, if necessary.
- The entire sample was crushed to 95% passing 10 mesh.
- 1,000 g was riffle-split and pulverised to 90% passing 150 mesh.
- A 250 g portion of each sample was returned to IAMGOLD.
- Each sample was analysed for Au and Ag by fire assay and for a multi-element package, using an aqua regia digestion with an Inductively Coupled Plasma (ICP) finish.
- Assay results were sent by the laboratory to IAMGOLD by email, followed by a hard copy assay certificate.
- Results were imported into IAMGOLD's database.

11.2 INV 2013 and 2016-2017 Drill Program

11.2.1 Sample Collection

Collection of core samples was carried out in mineralised or altered zones. The sample length was determined according to geological criteria (feeder zones, hydrothermal breccias, mineralisation styles and percentages, etc.) with a maximum length of 2 m; samples do not cross geological or hydrothermal alteration contacts. One (1) metre sample length was the default size within homogenous mineralised zones. At intervals where there was core loss, the sample length may be more than 2 m. Periodic sampling of barren-looking wall rocks was done to ensure no cryptic mineralisation was missed. After logging and tagging, all selected samples were cut in half with a diamond saw. For the two (2) metallurgical drillholes (LLD371 and LLD372) in 2013, the core was split in half, with one half sent for routine analysis (see Sample Preparation and Assay Protocols). The remaining half was then quarter split, with one quarter retained as a permanent record, and one quarter shipped to Lakefield, Ontario, for metallurgical testwork.

Core logging was carried out at INV's base camp, Campamento Cristal, and consisted of the following activities:

- Core boxes, core blocks, and sample intervals were all well marked with marker blocks nailed into place and positions marked on sides of trays.
- Aluminum tags were used to provide permanent markings on core blocks.
- Core photos were taken using a custom photo stand and tripod to provide a uniform field of view and high-resolution photos.
- Core recovery and basic geotechnical data was collected at the same time that the driller's core run marker block position was checked.
- Core was labelled at one metre intervals and then logged by a project geologist, who marked sample intervals.

The samples were double bagged in high density, clean, unused, transparent plastic bags. Each sample was assigned a unique sample number and labelled with permanent marker. A pre-printed paper sample ticket was placed inside each sample. Once the samples were bagged, they were sealed. The sample bags, up to a maximum of six (6), were placed inside of jute sacks, and labelled very clearly with the name of the Project and the number of the samples in each sack. The sacks were then placed in secure, sturdy fabric bags, and locked with tamper proof seals for transport to the preparation laboratory.

In 2016, both the retained core and the samples to be dispatched for analysis were transported by INV personnel from the Loma Larga site to Cuenca. In Cuenca, the samples taken for analysis were trucked by Transportes Ortiz, a local trucking company, to the preparation laboratory in Quito. In 2017, the samples taken for analysis were trucked by INV personnel from the Loma Larga site to San Gerardo. In San Gerardo, the samples taken for analysis were trucked by Top Logistics Solutions to the preparation laboratory in Quito. All retained core is stored at INV's core storage facility located on the property.

Sample preparation was carried out by Inspectorate del Ecuador S.A. (Inspectorate), part of the Bureau Veritas Group, at their office located at Calle Garcia Moreno 886 y Calle 23 de Abril, Llano Grande-Quito, Ecuador. Following sample preparation, Inspectorate sent the prepared samples by air freight to their analytical laboratory at Av Elmer Faucett 444, Callao-Lima, Peru. Inspectorate holds an international certificate for ISO 9001:2008 and fulfills NTP-ISO 17025:2006.

11.2.2 Density Sampling

IAMGOLD had previously compiled an extensive database of density values and INV did not carry out additional density sampling on the Project.

11.2.3 Sample Preparation and Assay Protocols

Sample preparation and analytical procedures are as follows:

- Samples are dried, if necessary.
- The entire sample is crushed to 95% passing 10 mesh.
- 1,000 g is riffle-split and pulverised to 90% passing 150 mesh.
- 200 g samples are returned to INV's Quito office to recode in a simple random ID_INV to ID_LAB code.
- Reference material and blanks are inserted by INV personnel.
- Samples are assayed for gold by fire assay and a multi-element package using an aqua regia digestion with an ICP finish.
- Assay data is emailed simultaneously to INV's Quito office as both Excel and PDF files (the latter as the digital equivalent of an assay certificate).

In the QP's opinion, the sample preparation, security, and analytical procedures are adequate to support a Mineral Resource Estimate.

11.3 Results of QA/QC Programs

Quality Assurance (QA) consists of evidence to demonstrate that the assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) used in order to have confidence in future resource estimations. Quality Control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical) precision (repeatability) and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling-assaying variability of the sampling method itself.

11.3.1 Pre-2012 Programs

No QA/QC results are available for programs prior to 2002. As noted above, data from those drill campaigns was not used to estimate the current Mineral Resource. All results and discussion in this section relate to IAMGOLD's campaigns from 2002 to 2008. For additional detail, tables, and figures, the reader is referred to the 2016 Technical Report (RPA, 2016).

IAMGOLD developed an industry standard QA/QC program for the Project early on in the exploration work, which consisted of the regular submission of blanks, duplicates, and standards. Furthermore, IAMGOLD defined control limits and implemented procedures to follow up results that were at or exceeded these limits (see below). The reader is referred to IAMGOLD (2009) for a comprehensive

overview of the QA/QC program from 2002 to the completion of the latest drill campaign at the end of 2007.

IAMGOLD inserted Certified Reference Material (CRM or analytical standard) and blank samples, each at a rate of one in every 15 samples (6.7%). In addition, for every 20 samples, a pulp duplicate was prepared and submitted, and ten percent of the pulp samples were split and sent to a second laboratory (replicates). Pulp replicates were introduced in 2005. IAMGOLD randomly selected samples every three (3) months to be re-analysed at ALS Chemex (the secondary laboratory).

From 2002 to 2008, a total of 1,015 CRMs and 714 blank samples were inserted into the process stream. IAMGOLD also collected 1,046 pulp replicates, 456 pulp duplicates, and 263 triplicates (replicates) for comparative analysis.

Once results from the analytical laboratory were uploaded, IAMGOLD personnel plotted the control sample results to determine if the analyses for CRMs, blanks, and duplicates were within pre-defined limits. IAMGOLD reviewed all failures in the following way:

- Notified IAMGOLD Technical Management.
- Requested a new analysis on the failed sample. In the case of a failed CRM, the five (5) samples analysed prior to and after the CRM would also be re-analysed.
- When the new analyses were received, a decision was made on how to handle the results. This included rejection of the initial batch results, re-submittal of the samples, or averaging the original and re-analysed batch.
- Once final certificate was verified, results were submitted to the Project database.

In August 2005, Lynda Bloom of Analytical Solutions Ltd., Toronto, conducted a review of the QC program. In general, Ms. Bloom concluded there was no evidence of contamination in the analysis of the 2005 diamond drill samples. Gold assays were biased low by approximately five (5) percent based on reported values for CRMs and comparisons with ALS Chemex assays. A comparison of BSI and ALS Chemex silver and copper assays demonstrated there were biases depending on grade. Ms. Bloom recommended that analytical procedures at both laboratories should be investigated to determine the differences. Ms. Bloom also recommended that the IAMGOLD QC program should be augmented by studies of the laboratory pulp duplicate assays as well as duplicate assaying of preparation and core samples.

Scott Wilson RPA (2006) compiled and reviewed all Project QC sample results for the 2006 drilling program and concluded that the statistics and scatterplots of the CRMs and duplicate samples revealed a high degree of assay precision and accuracy, and that the results of the blank samples suggested that contamination was not an issue at the Project. SLR notes that only the gold results were reviewed, as gold represented approximately 90% of the unit value of the mineralisation.

IAMGOLD (2009) reviewed the QA/QC results for gold only, for all drilling completed on the Project, and RPA (2012) evaluated the results of IAMGOLD's final drilling program in 2007 (i.e., all drilling that occurred after the effective date of Scott Wilson RPA, 2006) for gold, silver, and copper.

During IAMGOLD's drilling campaigns, control samples included 714 blanks, 1,015 CRMs, 456 pulp duplicates, and 263 pulp triplicates.

The QP reviewed the results for the CRMs, and although there are no failures, SLR notes that there is a low bias to gold results for SG14 (15 of 18 samples returned values below the certified value) and silver results for SN16 and SI15 (all analyses are less than the certified mean minus one SD) and SG14 (16 of 18 are less than the certified mean minus one SD). This suggests that the silver assay results may be low, at least for silver grades less than 20 g/t.

Of the 42 blank samples analysed, a single sample returned a gold value (10 ppb Au) greater than detection limit (5 ppb Au), which is still within the control limits (three (3) times detection limit is equivalent to 15 ppb Au). These results, along with conclusions reached by Bloom and Scott Wilson RPA (2006), indicate that there is no evidence of gold or silver contamination at the Project.

SLR examined scatterplots of the pulp duplicate and triplicate (replicate) assays. The results showed excellent agreement between the primary laboratory duplicates and triplicate gold and silver assays. Both sets of data had R-squared values of 0.98 to greater than 0.99 (a near perfect correlation) and a coefficient of variation of 0.98.

In the QP's opinion, the results of the QC samples, together with the QA/QC procedures used during IAMGOLD's ownership of the property provide adequate confidence in the data collection and processing, and the assay data is suitable for Mineral Resource estimation.

11.3.2 2013 PROGRAM

In July 2013, INV completed a 12-hole drilling program at Loma Larga, collecting and assaying 1,561 samples. During the drilling campaign, INV maintained a rigorous QA/QC program that incorporated the regular submission of blanks, duplicates, and standards. Specifically, the program included:

- Preparation duplicates (a second pulp prepared from the coarse reject) were prepared and inserted every 20 samples according to the downhole sampling sequence.
- Assay duplicates (a second analysis of the same pulp) were prepared concurrently with the preparation duplicates, every 20 samples according to the same downhole sequence as the preparation duplicates.
- Field duplicates were inserted one in every 40 drill samples, with the duplicate pair consisting of a pair of quarter core samples obtained from the original half core that would have normally been sent for assay.

- Ten (10) percent of all pulps were sent for check assay (pulp replicates) to a secondary laboratory, ALS Chemex in Vancouver, British Columbia (ALS Chemex is independent from INV).
- CRMs for gold and silver, in the form of pulps, were inserted into the sample stream at a ratio of one (1) in every 15 samples, independent of the duplicates. CRMs were commercially prepared sourced from Rocklabs in New Zealand. Gold standards included three (3) different grades ranging from less than 1.0 g/t Au to greater than 8.0 g/t Au.
- Blank pulp samples were inserted into the sample after each high-grade standard (SN16), and at regular intervals throughout the sample stream. SLR notes that the analysis of a blank pulp does not provide information on possible cross contamination during the sample preparation process. INV initially used commercial sand as the blank, but subsequently prepared and certified an in-house rock blank standard. Commencing with the final drillhole of the 2013 program, INV inserted the in-house blank consisting of uncrushed rock into the sample stream in intervals thought to be mineralised. These blank samples provide information on possible cross contamination during the sample preparation process. Only three (3) in-house blanks have been analysed, and there is insufficient data to permit a statistical analysis.

The QP reviewed the QA/QC results of INV's 2013 drilling campaign at Loma Larga, which comprised 12 holes. As part of the drilling program, 74 CRMs and 68 blank samples were inserted into the process stream. Additionally, INV collected 24 field duplicates, 77 pulp duplicates, 77 reject duplicates, and 167 pulp replicates as listed in Table 11.1.

Table 11.1 – QA/QC Review Summary – 2013 Drilling Program

Metal	Blanks Count	Field Dup. Count	Pulp Dup. Count	Reject Dup. Count	CRM Count	Check Assays Count
Au	42	24	77	77	74	167
Ag	42	24	77	77	74	167
Cu	42	24	77	77	0	167

Results from the analytical laboratory are imported into the INV's exploration database and plotted to determine if the control sample results are within pre-defined limits. If the analysis for a CRM is outside a two (2) standard deviation range of the CRM certified gold content, INV requests a new analysis from the laboratory, including five (5) samples upstream and five (5) samples downstream of the suspected failed sample. When the new analyses are received, INV personnel decide how the repeated analyses are managed. This includes rejection of the original batch, averaging of the original and repeat results, or re submittal of the entire batch.

If duplicate mineralised sample results differ by more than 20% from the original assays, unless the values are insignificant and outside the resource, a new analysis is requested on the first, second, and third pulps. Once the problem is identified, a new analysis of the batch is requested by INV, if needed.

Dr. Matthew Gray of Resource Geosciences Inc., an independent consultant to INV, conducted a review of the drilling database of this campaign, including a statistical and procedural QA/QC review (Gray, 2013). SLR notes that Dr. Gray only reviewed the results for gold.

Dr. Gray compared analyses from the original laboratory certificates with INV digital database. Approximately 20% of the samples were verified in this manner and no discrepancies were observed. In addition, Dr. Gray conducted a statistical review of the drillhole database, analysing the assay results for mineralised standards, blanks, preparation and assay duplicates, and independent laboratory check assays, and determined that the gold assays provided by Inspectorate are reliable.

Dr. Gray concluded that field duplicates demonstrated variability indicative of primary heterogeneity of the gold distribution (i.e., “nugget effect”), and that field duplicate data overestimates the sampling error because field duplicates are quarter core samples. Nonetheless, there is no significant bias between the sample sets. Gray noted a mislabelled field bank and a significant control sample failure, which was followed up satisfactorily by INV. Coarse reject and pulp duplicate results, once mislabelling errors were accounted for, were acceptable. Check assays performed at a secondary laboratory were materially equivalent for gold, silver, and copper, without significant bias.

Dr. Gray recommended that INV conduct field duplicate analyses using half core field duplicates in order to determine if half core samples are adequately sized to provide sampling error within acceptable limits. Furthermore, Dr. Gray suggested that metallic sieve fire assays of representative intervals of high-grade zones be conducted using the existing core library. The QP has reviewed Dr. Gray’s report and the underlying data and concur with the recommendations.

11.3.2.1 Certified Reference Material

INV used three (3) CRMs obtained from Rocklabs, certified for gold and silver: SG14, SI15, and SN16. The CRMs were inserted as pulps into the sample stream after the laboratory had completed its sample preparation, thus testing the analytical accuracy of the laboratory, but not contamination during sample preparation. INV inserted a total of 74 CRMs for 12 drillholes in 2013.

Table 11.2 lists the CRM name, expected mean, standard deviation, and the number of CRMs inserted during the drilling program. No copper CRM was used during the 2013 drilling campaign.

Table 11.2 – CRMs Used in 2013 Drilling Program

Gold			
CRM	**Number**	**Mean** (g/t)	**St. Dev.**
SG14	25	0.989	0.044
SI15	21	1.805	0.067
SN16	28	8.367	0.217

Silver			
CRM	**Number**	**Mean** (g/t)	**St. Dev.**
SG14	25	11.12	1.03
SI15	21	19.68	1.02
SN16	28	17.64	0.96

Using INV's criterion, four (4) of the 74 results failed (5.4%) for gold, which is slightly higher than the three (3) percent that would be expected to fall outside the limits. Of these four (4) failures, two (2) returned assayed values greater than ten percent (10%) above the certified mean, and one (1) failure was greater than ten percent (10%) below the certified mean. The failures were deemed to be minor, and no re-analyses were requested by INV. The mean gold content reported by Inspectorate for all gold control samples was within three percent (3%) of the certified mean for all three (3) CRMs (Table 11.3).

Table 11.3 – Summary of CRM Results

Gold						
CRM	**Number**	**Observed Au** (g/t)		**Expected Au** (g/t)		**% Diff. Observed to Expected Mean**
		Mean	**Std. Dev.**	**Mean**	**Std. Dev.**	
SG14	25	0.978	0.036	0.989	0.044	-1.11
SI15	21	1.748	0.063	1.805	0.067	-3.17
SN16	28	8.631	0.199	8.367	0.217	+3.15
Silver						
CRM	**Number**	**Observed Ag** (g/t)		**Expected Ag** (g/t)		**% Diff. Observed to Expected Mean**
		Mean	**St. Dev.**	**Mean**	**St. Dev**	
SG14	25	11.168	0.532	11.12	1.03	+0.43
SI15	21	19.429	0.450	19.68	1.02	-1.28
SN16	28	18.182	0.805	17.64	0.96	-3.07

The QP evaluated the QA/QC results within a more conventional three (3) standard deviation range of the CRM certified gold content. Observed results that fell outside this range were deemed to be failures. Using this criterion, one (1) of 74 results failed (1.35%): a single control sample result fell greater than three (3) standard deviations above the mean (+16%).

The QP further notes that although observed results from CRM SG14 do not show an obvious bias, bias is observed in both SI15 and SN16 (Figure 11.1 to Figure 11.3). The observed results from SI15 are clearly biased low, and the observed results from SN16 are biased high. The QP recommends that INV follow up the high bias observed in SN16.

INV noted a significant control sample gold failure in Job 13-703-00313-01 and subsequent follow-up determined a laboratory error was the cause. A sequence of 20 samples was re-assayed (46464 to 46484), and the QP reviewed the corrected data.

The QP also evaluated the QA/QC results for silver. A description of the evaluation and supporting graphs are available on the RPA Technical Report on the Loma Larga Project (2016).

Figure 11.1 – Gold Control Chart for 2013 Drilling – CRM SG14

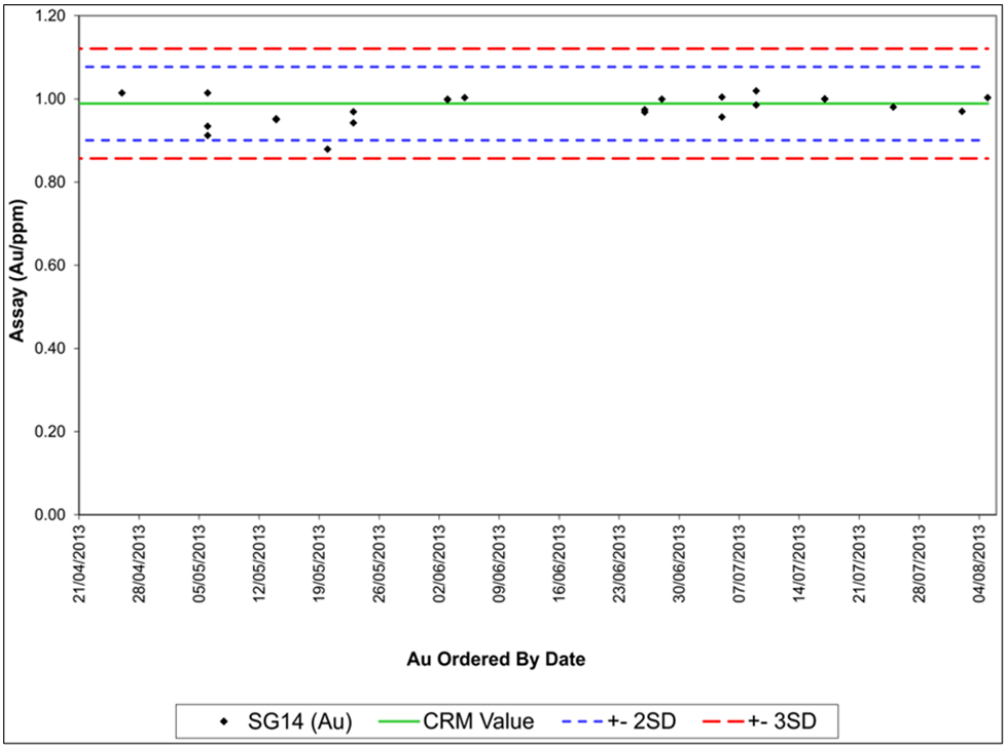


Figure 11.2 – Gold Control Chart for 2013 Drilling – CRM SI15

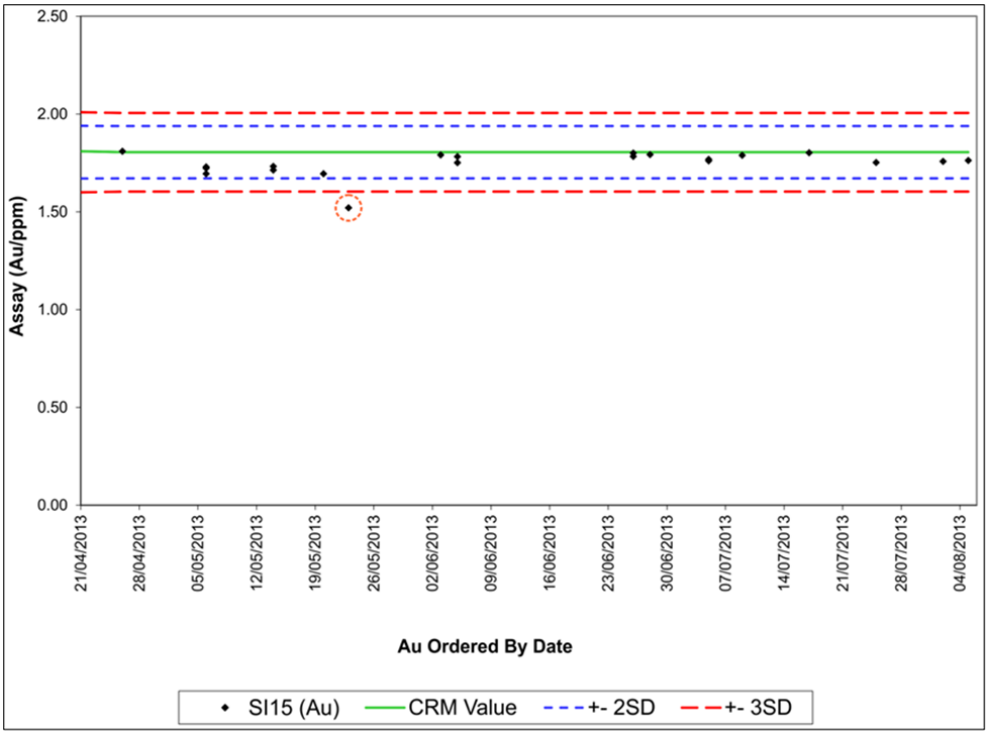
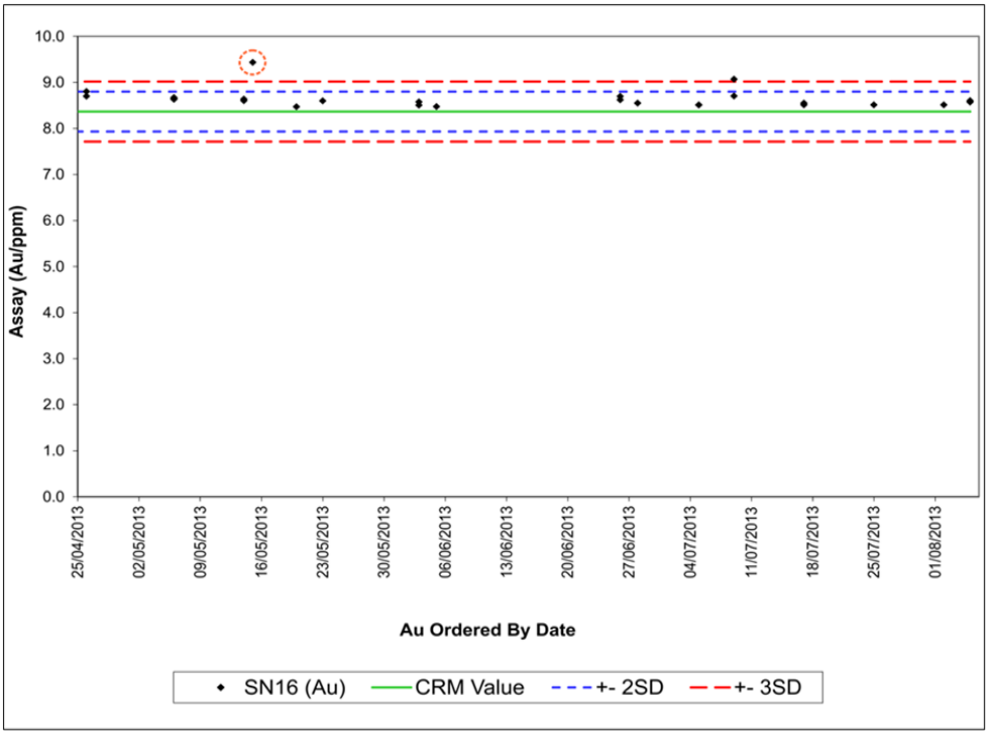


Figure 11.3 – Gold Control Chart for 2013 Drilling – CRM SN16



11.3.2.2 Blanks

Contamination and sample numbering errors are assessed through blank samples. A significant level of contamination is identified when the blank samples yield values exceeding ten (10) times the detection limit of the analytical method. For the 2013 drilling program at the Loma Larga Project, gold values should be below 0.05 g/t, silver below 2 g/t, and copper below 20 ppm. A total of 68 blank samples were inserted into the sample stream by INV during the 2013 drilling program. The results for these samples were plotted chronologically to determine if any trends had occurred over time. All blank assay results for gold and silver were at or below detection limit, and copper results varied between 7 ppm and 13 ppm, with no detectable systematic pattern.

It is the QP's opinion that these results demonstrate no evidence of contamination.

In the QA/QC report dated June 17, 2013, Dr. Gray called attention to the failure of blank sample 46321 in Job 13-703-00278-01. A review of original data indicated an inadvertent mislabelling of a core sample as a blank. The data used in this review by the QP is the corrected data set.

11.3.2.3 Duplicates

Field Duplicates

Field duplicates assess the variability introduced by sampling the same interval. The duplicate splits are bagged separately with separate sample numbers so as to be blind to the sample preparation laboratory. The duplicates contain all levels of sampling and analytical error and are used to calculate field, sample preparation, and analytical precision. To permit the most meaningful interpretations, field duplicates should be the same size as the regular samples sent for analysis.

A total of 24 field duplicate samples were submitted to Inspectorate during the 2013 drilling program. INV chose to use quarter core field duplicates to maintain a complete core record. That is, the interval selected for a field duplicate included two (2) samples of quarter core (a regular sample and a duplicate), whereas regular samples comprised a half core split. The variances, however, of the smaller sample size will be greater, thus the field duplicates overestimate the sampling error associated with the standard half core assay samples.

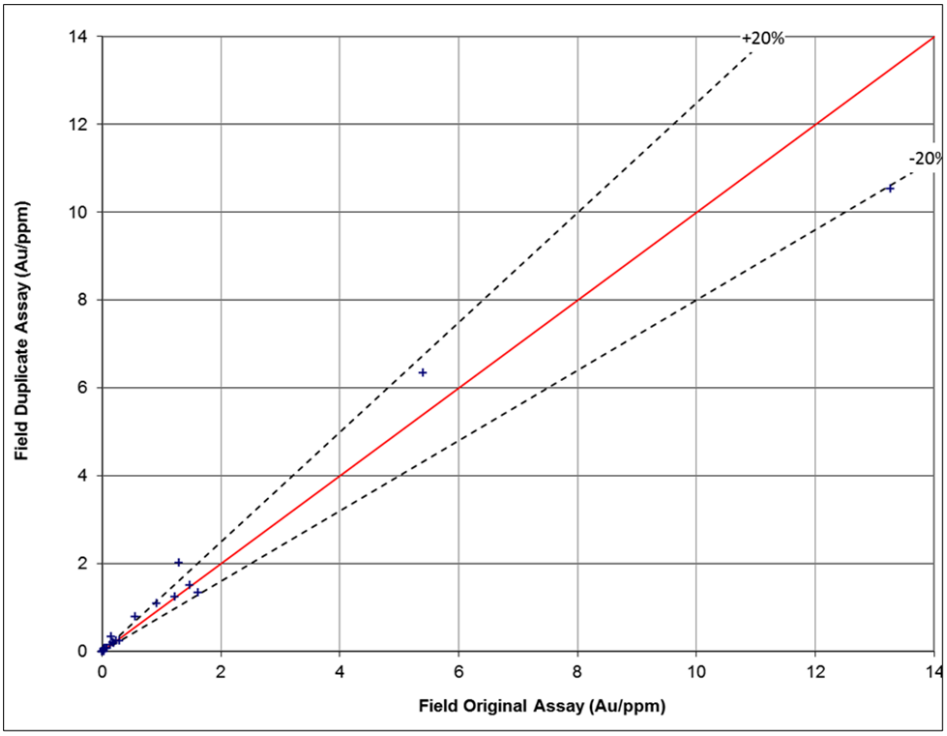
The QP reviewed the gold, silver, and copper field duplicate data. Summary statistics for gold are presented in Table 11.4 and scatterplots for gold in Figure 11.4. Summary statistics, scatterplots, and description of the reviews for silver and copper are available in the RPA Technical Report on the Loma Larga Project (2016).

The percent relative difference between mean gold content of field duplicates and originals is less than three percent (3%), and although observed variance is greatest at the lowest grade ranges, there is a paucity of data (one data point) for gold grades above the Loma Larga resource average.

Table 11.4 – Summary of Field Duplicate Results

Description	Original	Duplicate
Number of Samples	24	24
Mean	1.14	1.11
Maximum Value	13.27	10.53
Minimum Value	0.01	0.01
Median	0.16	0.20
Correlation Coefficient	0.984	
Percent Difference Between Means	2.4%	

Figure 11.4 – Gold Field Duplicate Scatterplot



Reject Duplicates

Reject duplicates (or coarse reject duplicates) are duplicate samples taken immediately after the first crushing and splitting step. This was done by Inspectorate at INV's request. The reject duplicate will inform about the subsampling precision, that is, the errors due to sample size reduction after crushing and the errors associated with weighing and analysis of the pulp. Pulverisation and assaying follow the same procedure, at the same laboratory, for each sample in the duplicate pair.

Table 11.5 summarises the basic statistics of the gold reject duplicate pairs and the gold scatterplot is illustrated in Figure 11.5. Summary statistics, scatterplots, and description of the reviews for silver and copper are available on the RPA Technical Report on the Loma Larga Project (2016). Gold, silver, and copper all show excellent correlation between means and very low percent difference between means (all less than one percent absolute difference). No bias is observed at either very low-grades, or near-average resource grades of gold, silver, and copper.

Table 11.5 – Summary of Reject Duplicate Results

Description	Original	Duplicate
Number of Samples	77	77
Mean	19.7	19.7
Maximum Value	456.0	455.0
Minimum Value	0.2	0.2
Median	1.5	1.5
Correlation Coefficient	1.00	
Percent Difference Between Means	-0.6%	

The duplicate data pairs taken in 2013 from Loma Larga follow the expected progression of decreasing variance and percent difference between means from field to preparation to assay duplicates. Although the precision is somewhat high for field duplicate pairs, it is not atypical for an epithermal gold deposit, and also must be interpreted in consideration of the relatively small quarter core sample size, which would overestimate the sampling error associated with the half core assay samples.

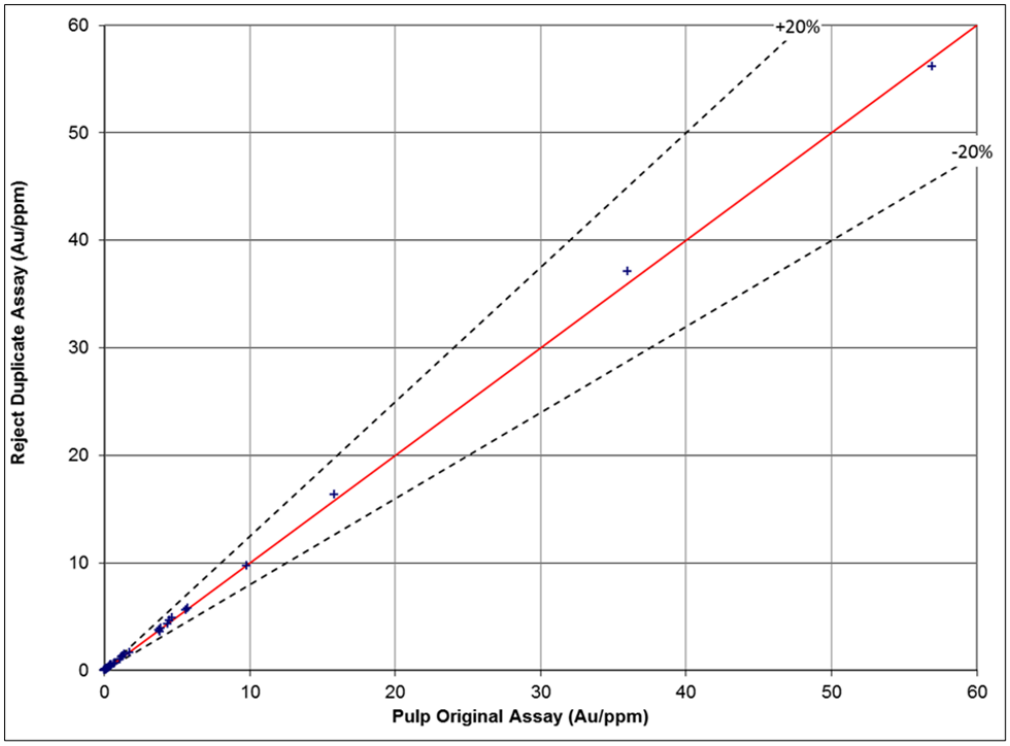
With development at Loma Larga focused on high-grade zones within the deposit, the QP recommended that duplicate samples test intersections that are representative of the expected average grade of the Mineral Resource.

The QP suggested that INV procure reference standards with grades that better reflect the range of gold grades within the Mineral Resource. The QP recommended the following gold grade ranges:

- A “low-grade” standard (5 g/t to 7 g/t).
- An “average grade” standard (9 g/t to 12 g/t).
- A “high-grade” standard (greater than 30 g/t).

The QP also suggested that INV obtain a copper standard with a grade range of 6,000 ppm to 8,000 ppm (approximately the average grade of the Loma Larga Mineral Resource).

Figure 11.5 – Gold Reject Duplicate Scatterplot



Pulp Duplicates

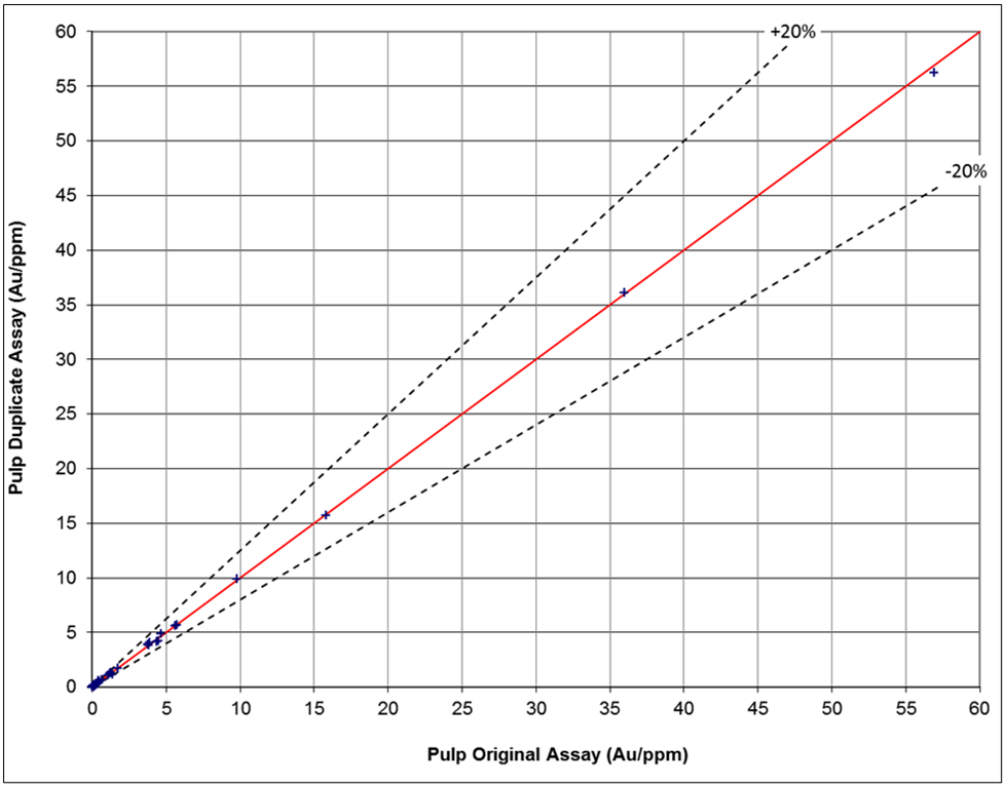
Pulp duplicates consist of second splits of final prepared pulverised samples, analysed by the same laboratory as the original samples under different sample numbers. The pulp duplicates are indicators of the analytical precision, which may also be affected by the quality of pulverisation and homogenisation. INV chose to analyse pulp duplicates on the same samples that were selected for reject duplicate analysis.

Table 11.6 summarises the basic statistics of the gold pulp duplicate pairs and the gold scatterplot is illustrated in Figure 11.6. Summary statistics, scatterplots, and description of the reviews for silver and copper are available on the RPA Technical Report on the Loma Larga Project (2016). Gold, silver, and copper all show excellent correlation between means and very low percent difference between means. No bias is observed either at very low grades (below 1 g/t Au, 10 g/t Ag, and 1,000 ppm Cu), or near average resource grades of gold, silver, and copper.

Table 11.6 – Summary of Pulp Duplicate Results

Description	Original	Duplicate
Number of Samples	77	77
Mean	2.21	2.20
Maximum Value	56.90	56.20
Minimum Value	0.01	0.01
Median	0.14	0.13
Correlation Coefficient	1.00	
Percent Difference Between Means	0.4%	

Figure 11.6 – Gold Pulp Duplicate Scatterplot



Pulp Replicates (Check Assays)

A total of 167 regular samples (a rate of greater than ten percent) were selected from the 2013 drilling program and duplicate splits of the pulps were sent to ALS Ltd., which served as the secondary (check) laboratory. Sample preparation was completed at ALS's Ecuador facility, ALS

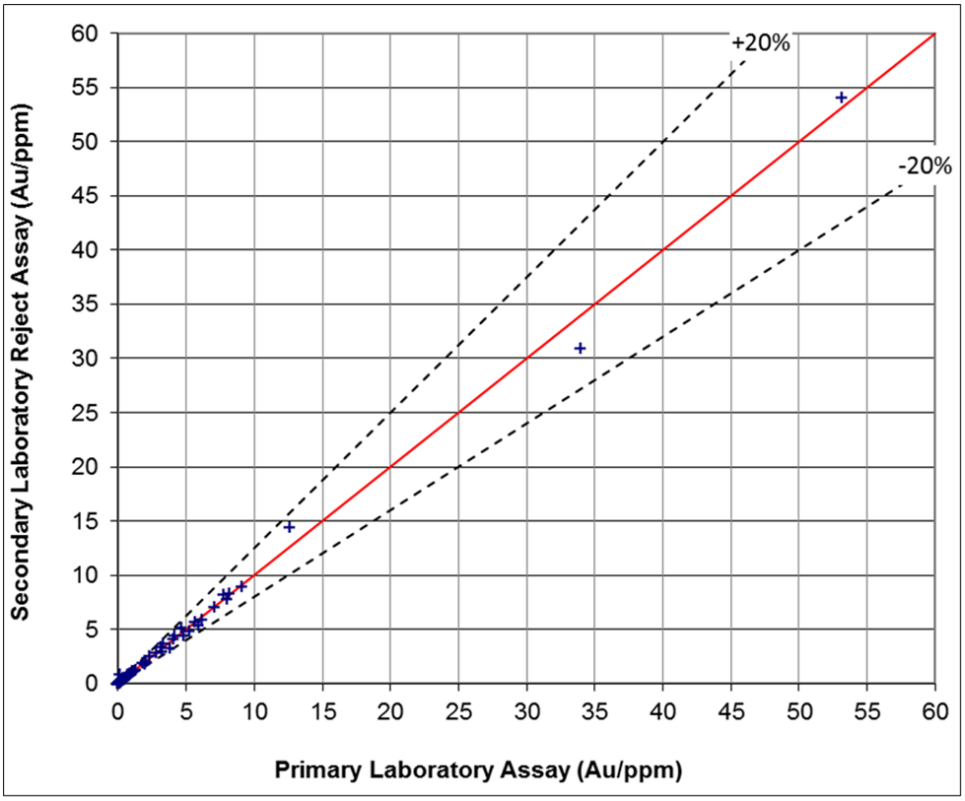
Laboratory Group Quito, and pulps were forwarded to ALS Peru S.A. in Lima for laboratory for analyses.

Table 11.7 summarises the basic gold statistics of the pulp replicate pairs and the gold scatterplot is illustrated in Figure 11.7. Summary statistics, scatterplots, and description of the reviews for silver and copper are available on the RPA Technical Report on the Loma Larga Project (2016). Gold, silver, and copper all show excellent correlation between means and very low percent difference between means. No bias is observed either at very low-grades, or near average resource grades of gold, silver, and copper.

Table 11.7 – Summary of Pulp Replicate Results

Description	Primary Lab	Secondary Lab
Number of Samples	167	167
Mean	1.41	1.40
Maximum Value	53.15	54.00
Minimum Value	0.01	0.01
Median	0.11	0.09
Correlation Coefficient	0.998	
Percent Difference Between Means	0.7%	

Figure 11.7 – Gold Pulp Replicate Scatterplot



11.3.3 2016-2017 PROGRAM

In August 2017, INV completed a 23-hole drilling program at Loma Larga, collecting and assaying 2,423 samples. During the drilling campaign, INV maintained a rigorous QA/QC program that incorporated the regular submission of blanks, duplicates, and standards. Specifically, the program included:

- Preparation duplicates (a second pulp prepared from the coarse reject) were prepared and inserted every 20 samples according to the downhole sampling sequence.
- Assay duplicates (a second analysis of the same pulp) were prepared concurrently with the preparation duplicates, every 20 samples according to the same downhole sequence as the preparation duplicates.
- Field duplicates were inserted one in every 40 drill samples, with the duplicate pair consisting of a pair of quarter core samples obtained from the original half core that would have normally been sent for assay.
- CRMs for gold, in the form of pulps, were inserted into the sample stream at a ratio of one (1) in every 15 samples, independent of the duplicates. CRMs were commercially prepared

sourced from Rocklabs in New Zealand. Gold standards included seven different grades ranging from less than 1.0 g/t Au to greater than 8.0 g/t Au.

- Blank pulp samples were inserted into the sample after each high-grade standard (SN16, SN60), and at regular intervals throughout the sample stream. SLR notes that the analysis of a blank pulp does not provide information on possible cross contamination during the sample preparation process. INV initially used commercial sand as the blank, but subsequently prepared and certified an in-house rock blank standard commencing with the final drillhole of the 2013 program. The blank consists of uncrushed rock and is inserted into the sample stream in intervals thought to be mineralised. These blank samples provide information on possible cross contamination during the sample preparation process.

For the current Mineral Resource estimate, the QP reviewed the QA/QC results of INV's 2016- 2017 drilling campaign at Loma Larga. Table 11.8 lists the drillholes, all of which were completed after the effective date of SLR's previous 2016 Mineral Resource estimate.

Table 11.8 – Summary of 2016-2017 Drilling Program

Hole	Hole ID	Hole	Hole ID
1	LLD-376	13	LLD-388
2	LLD-377	14	LLD-389
3	LLD-378	15	LLDGT-003
4	LLD-379	16	LLDGT-004
5	LLD-380	17	LLDGT-005
6	LLD-381	18	LLDGT-006
7	LLD-382	19	LLDGT-007
8	LLD-383	20	LLDGT-03B
9	LLD-384	21	LLDGT-03C
10	LLD-385	22	LLDHG-008
11	LLD-386	23	LLDHG-011
12	LLD-387		

During the 2016-2017 drill campaign, 84 CRMs and 94 blank samples were inserted into the process stream. INV also collected 127 pulp duplicates, and 127 reject duplicates, as listed in Table 11.9.

Table 11.9 – QA/QC Review Summary – 2017 Drilling Program

Metal	Blanks Count	Pulp Dup. Count	Reject Dup. Count	CRM Count
Au	94	127	127	84
Ag	94	127	127	0
Cu	94	127	127	0

Results from the analytical laboratory are imported into INV's exploration database and plotted to determine if the control sample results are within pre-defined limits. If the analysis for a CRM is outside a two (2) standard deviation range of the CRM certified gold content, INV's procedure is to request a new analysis from the laboratory, including five (5) samples upstream and five (5) samples downstream, of the suspected failed sample. When the new analyses are received, INV personnel decide how the repeated analyses are managed. This includes rejection of the original batch, averaging of the original and repeat results, or re-submittal of the entire batch. SLR notes that INV did not utilise these procedures for CRM results that were outside of the control limits.

If duplicate mineralised sample results within the resource area differ by more than 20% from the original assays, INV's procedure is to request a new analysis on the first, second, and third pulps. Once the problem is identified, a new analysis of the batch is requested by INV, if needed.

11.3.3.1 Certified Reference Material

INV used eight (8) gold CRMs obtained from Rocklabs: HISILK2, HISILP1, SG14, SG66, SI15, SJ63, SN16, and SN60. The CRMs were inserted as pulps into the sample stream after the laboratory had completed its sample preparation, thus testing the analytical accuracy of the analytical process, but not contamination during sample preparation. INV inserted a total of 84 CRMs for 23 drillholes in 2017. Three (3) of the eight (8) CRMs consisted of single samples and as a result, cannot be used for statistical purposes, therefore reducing the practical number of CRMs to 81.

Table 11.10 lists the CRM name, expected mean, standard deviation, and the number of CRMs inserted during the drilling program. No silver or copper CRM were used during the 2017 drilling campaign.

Table 11.10 – Gold CRMs Used in 2017 Drilling Program

CRM	Number	Mean (g/t)	St. Dev.
HISILK2	1	N/A	N/A
HISILP1	1	N/A	N/A
SG14	1	N/A	N/A
SG66	5	1.070	0.049
SI15	33	1.810	0.067
SJ63	8	2.490	0.155
SN16	27	8.367	0.217
SN60	9	8.600	0.223

INV chose to evaluate the QA/QC results within a conventional three (3) standard deviation range of the CRM certified gold content. Observed results that fell outside this range were deemed to be failures. Using this criterion, 10 of 81 results failed (8.1%) for gold. As a result of the unusually high failure rate of SG66 and SJ63 with respect to their certified mean values, INV evaluated these CRM results using the mean of their respective assay results as opposed to the expected values provided on the Certificate of Analysis. Using the recalculated mean and standard deviation parameters, two (2) of the 81 results failed (2.4%) for gold, which is lower than the three percent (3%) that would be expected to fall outside the limits. The mean gold content reported by Inspectorate for all gold control samples was within 3% of the certified mean for CRMs SG66, SI15, SN16 and SN60, but above it for CRM SJ63 (Table 11.11, Figure 11.8 to Figure 11.12).

Table 11.11 – Summary of CRM Results

CRM	Number	Observed Au (g/t)		Expected Au (g/t)		% Diff. Observed to Expected Mean
		Mean	Std. Dev.	Mean	Std. Dev.	
HISILK2	1	N/A	N/A	N/A	N/A	N/A
HISILP1	1	N/A	N/A	N/A	N/A	N/A
SG14	1	N/A	N/A	N/A	N/A	N/A
SG66	5	1.086	0.032	1.070	0.049	-1.50%
SI15	33	1.805	0.067	1.810	0.067	0.28%
SJ63	8	2.632	0.055	2.490	0.155	-5.70%
SN16	27	8.370	0.217	8.367	0.217	-0.04%
SN60	9	8.595	0.223	8.600	0.223	0.06%

The QP noted that there may be a slight overestimation bias on results from CRM SN16. The observed results from CRM SG66 appear to be biased low. The QP recommended that INV investigate the high and low bias observed in CRMs SN16 and SG66, respectively. The QP recommended that INV investigate the differences between the supplied expected mean and round-robin determined mean observed in CRMs SG66 and SJ63 by conducting an external check.

No assay batches determined to have failed QA/QC were re-assayed.

Due to the low NSR value of silver and copper (less than 1%) no CRM evaluation for silver or copper was performed.

Figure 11.8 – Gold Control Chart – CRM SG66

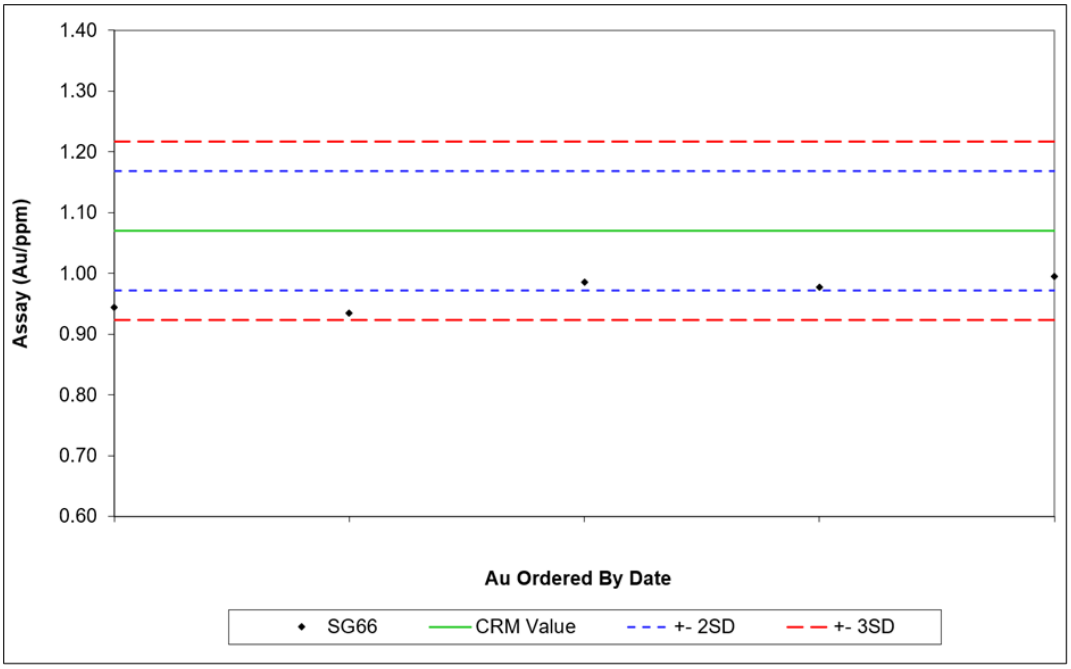


Figure 11.9 – Gold Control Chart – CRM SI15

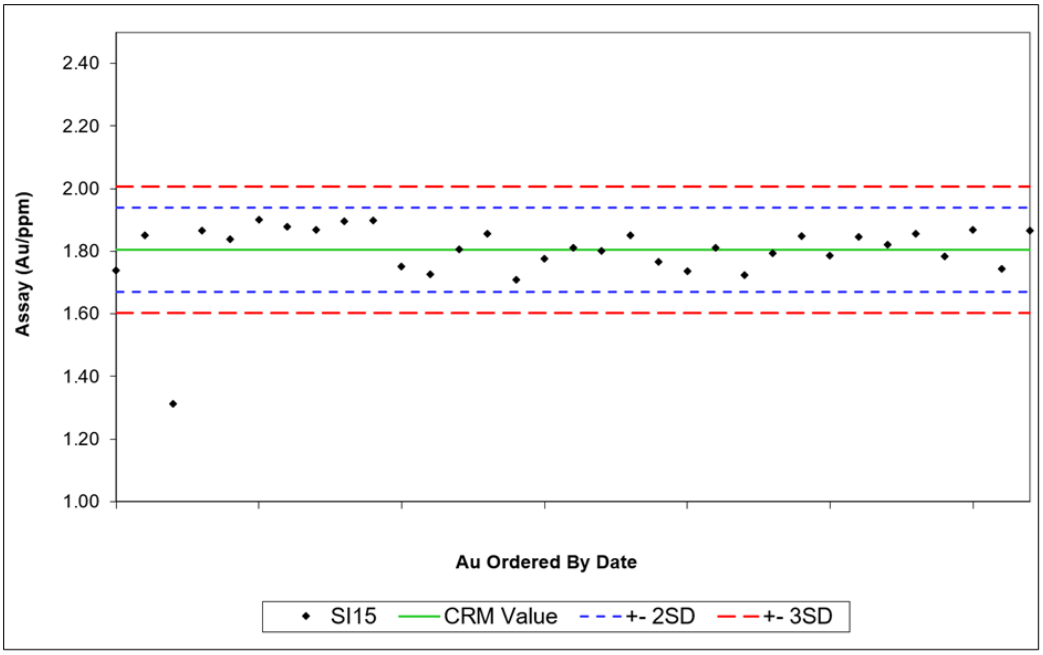


Figure 11.10 – Gold Control Chart – CRM SJ63

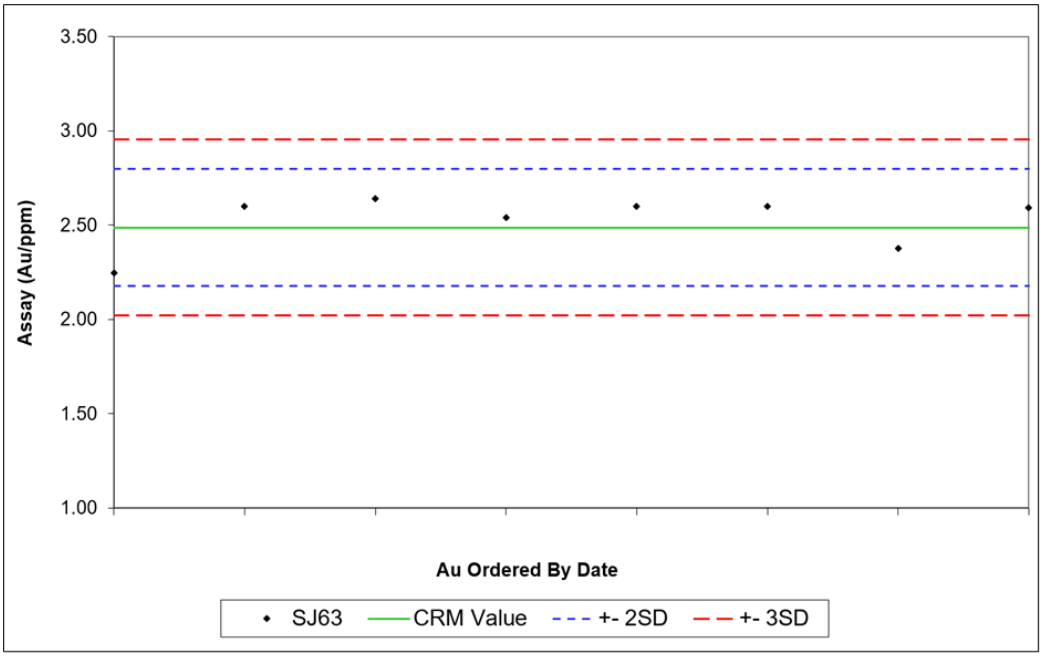


Figure 11.11 – Gold Control Chart – CRM SN16

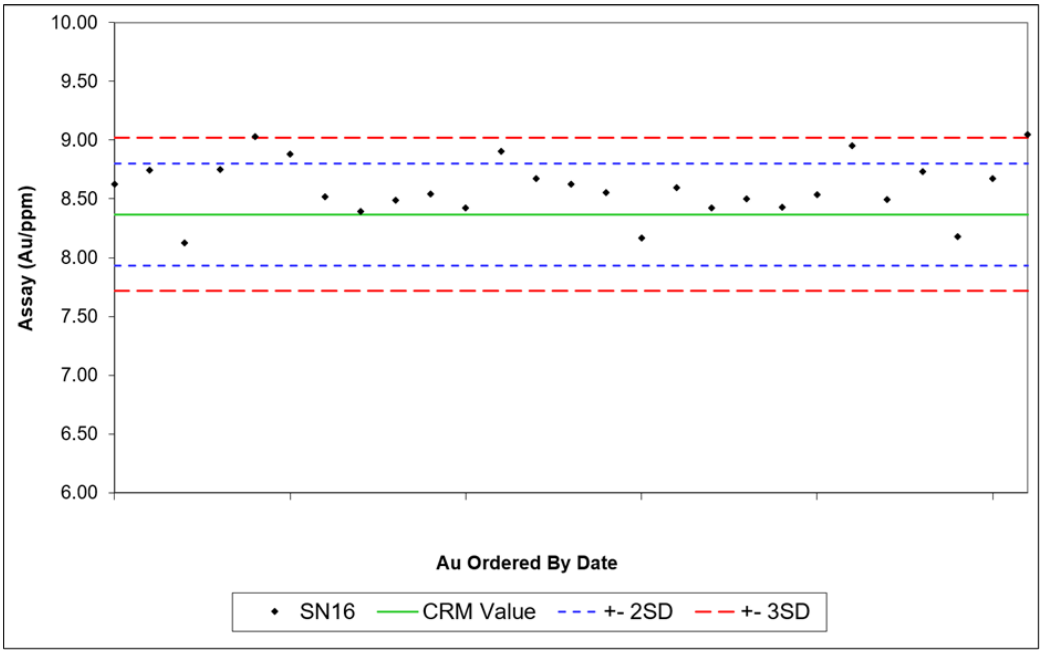
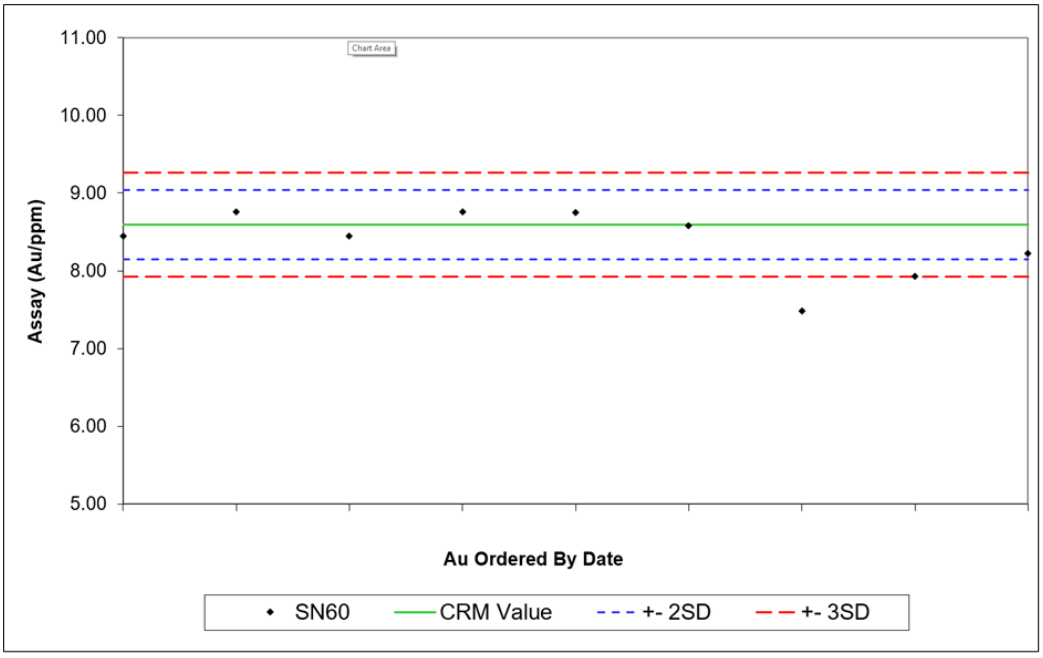


Figure 11.12 – Gold Control Chart – CRM SN60



11.3.3.2 Blanks

Contamination and sample numbering errors are assessed through blank samples. In general, contamination is identified when the blank samples yield values exceeding ten times the detection

limit of the analytical method. For the 2016-2017 drilling program, gold values should be below 0.05 g/t Au, silver below 2 g/t Ag, and copper below 20 ppm Cu. A total of 94 blank samples were inserted into the sample stream by INV during the 2016-2017 drilling program. The results for these samples were plotted chronologically to determine if any trends had occurred over time. All blank assay results for gold and silver were at or below detection limit, and copper results varied between 2 ppm and 24 ppm, with no detectable systematic pattern.

It is the QP's opinion that these results demonstrate no evidence of contamination.

11.3.3.3 Duplicates

Field Duplicates

No field duplicates were collected during the 2016-2017 drilling program. The QP recommends that the practice of collecting field duplicates resumes in future drilling campaigns.

Reject Duplicates

Table 11.12 summarises the basic statistics of the reject duplicate pairs and scatterplots of each data set are illustrated in Figure 11.13 to Figure 11.15. Gold, silver, and copper all show excellent correlation between means and very low percent difference between means (all less than one percent (1%) absolute difference). No bias is observed at either very low-grades, or near-average resource grades of gold, silver, and copper.

As in 2013, the duplicate data pairs taken in 2016-2017 from Loma Larga follow the expected progression of decreasing variance and percent difference between means from preparation to assay duplicates.

Table 11.12 – Summary of Reject Duplicate Results

Description – Gold (g/t)	**Original**	**Duplicate**
Number of Samples	127	127
Mean	1.11	1.11
Maximum Value	14.10	14.90
Minimum Value	0.01	0.01
Median	0.17	0.16
Correlation Coefficient	0.988	
Percent Difference Between Means	-0.1%	

Description – Silver (g/t)	Original	Duplicate
Number of Samples	127	127
Mean	8.02	8.07
Maximum Value	141.50	142.00
Minimum Value	0.20	0.20
Median	2.00	1.70
Correlation Coefficient	0.997	
Percent Difference Between Means	-0.6%	

Description – Copper (ppm)	Original	Duplicate
Number of Samples	127	127
Mean	710.62	704.76
Maximum Value	17,610	16,800
Minimum Value	2.00	2.00
Median	126.00	130.00
Correlation Coefficient	0.999	
Percent Difference Between Means	0.8%	

Figure 11.13 – Gold Reject Duplicate Scatterplot

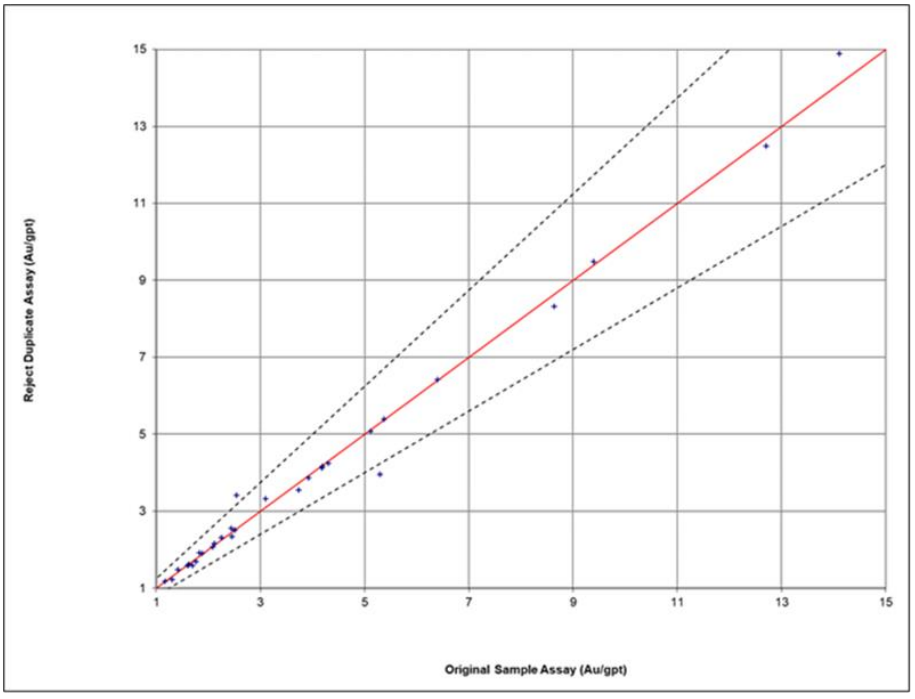


Figure 11.14 – Silver Reject Duplicate Scatterplot

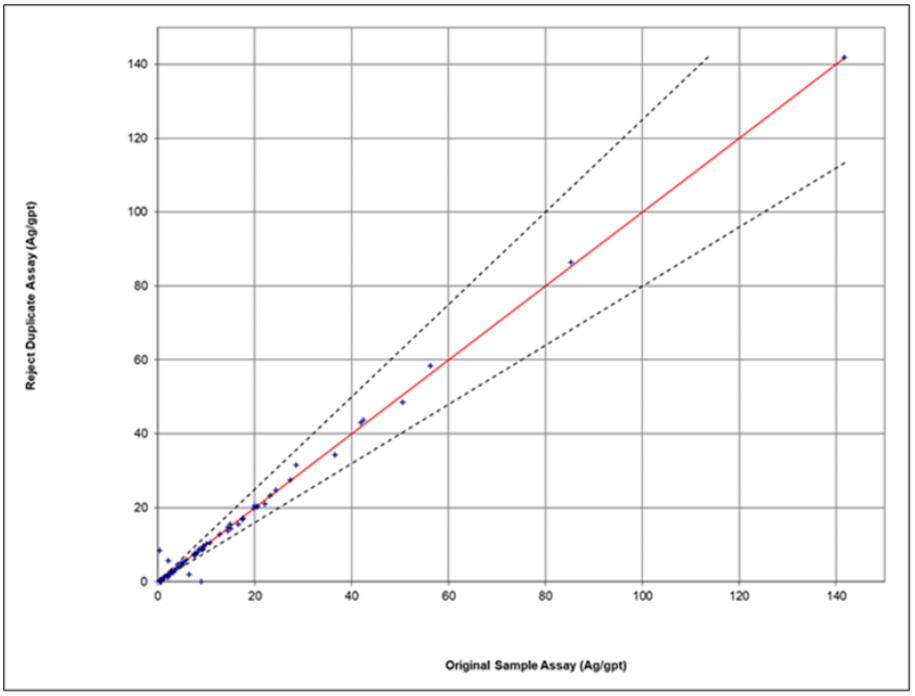
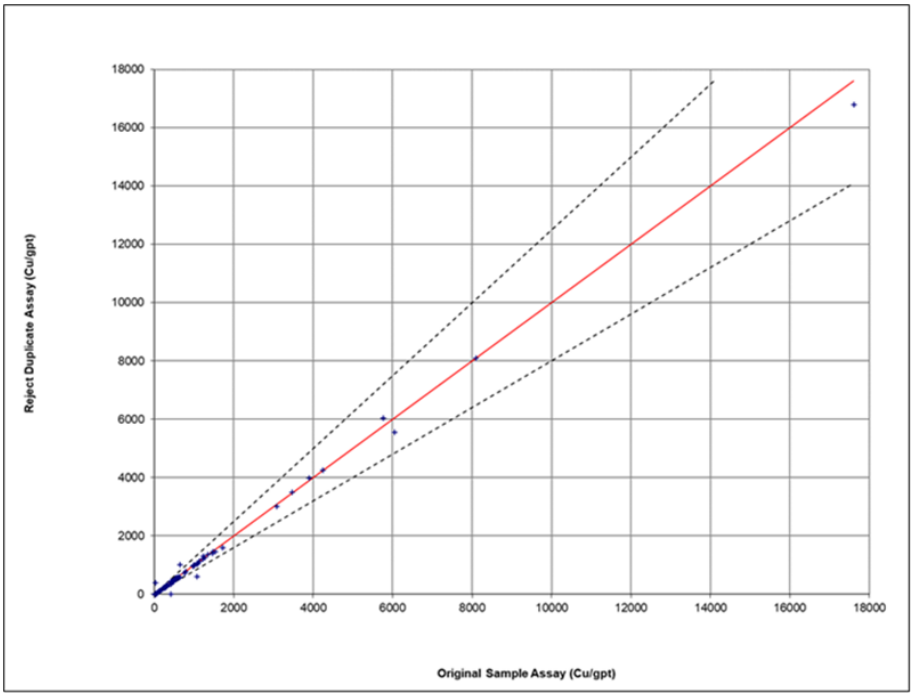


Figure 11.15 – Copper Reject Duplicate Scatterplot



Pulp Duplicates

Pulp duplicates consist of second splits of final prepared pulverised samples, analysed by the same laboratory as the original samples under different sample numbers. The pulp duplicates are indicators of the analytical precision, which may also be affected by the quality of pulverisation and homogenisation. INV chose to analyse pulp duplicates on the same samples that were selected for reject duplicate analysis.

Table 11.13 summarises the basic statistics of the pulp duplicate pairs and scatterplots of each data set are illustrated in Figure 11.16 to Figure 11.18. Gold, silver, and copper all show excellent correlation between means and very low percent difference between means. No bias is observed either at very low grades (below 1 g/t Au, 10 g/t Ag, and 1,000 ppm Cu), or near average resource grades of gold, silver, and copper.

Table 11.13 – Summary of Pulp Duplicate Results

Description – Gold (g/t)	**Original**	**Duplicate**
Number of Samples	127	127
Mean	1.11	1.12
Maximum Value	14.10	14.20
Minimum Value	0.01	0.01
Median	0.17	0.17
Correlation Coefficient	0.989	
Percent Difference Between Means	-0.7%	

Description – Silver (g/t)	**Original**	**Duplicate**
Number of Samples	127	127
Mean	8.02	8.02
Maximum Value	141.50	140.00
Minimum Value	0.20	0.20
Median	2.00	1.70
Correlation Coefficient	0.997	
Percent Difference Between Means	0.0%	

Description – Copper (ppm)	Original	Duplicate
Number of Samples	127	127
Mean	710.62	703.94
Maximum Value	17,610	17,380
Minimum Value	2.00	2.00
Median	126.00	129.00
Correlation Coefficient	0.999	
Percent Difference Between Means	0.9%	

Figure 11.16 – Gold Pulp Duplicate Scatterplot

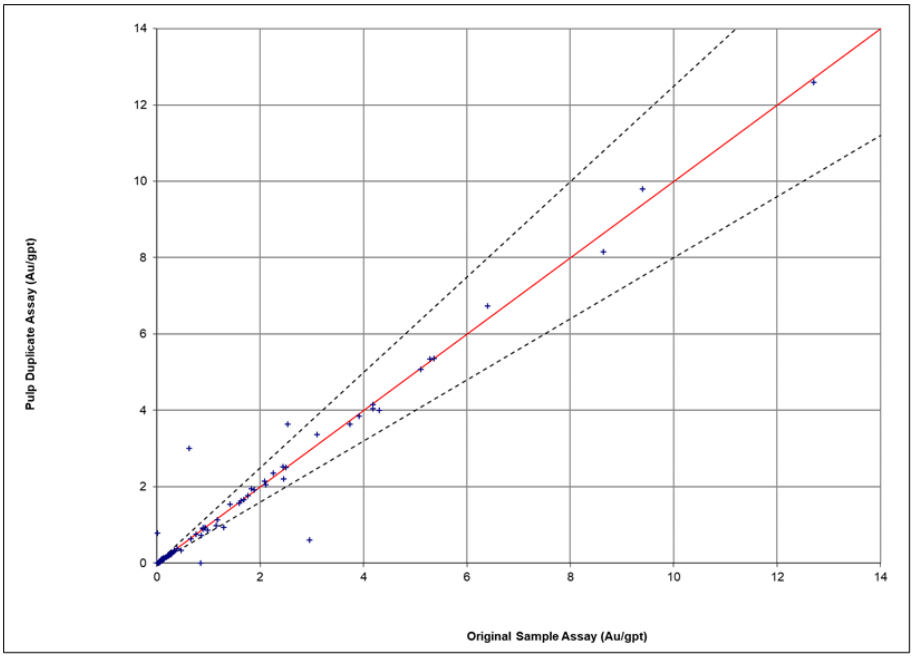


Figure 11.17 – Silver Pulp Duplicate Scatterplot

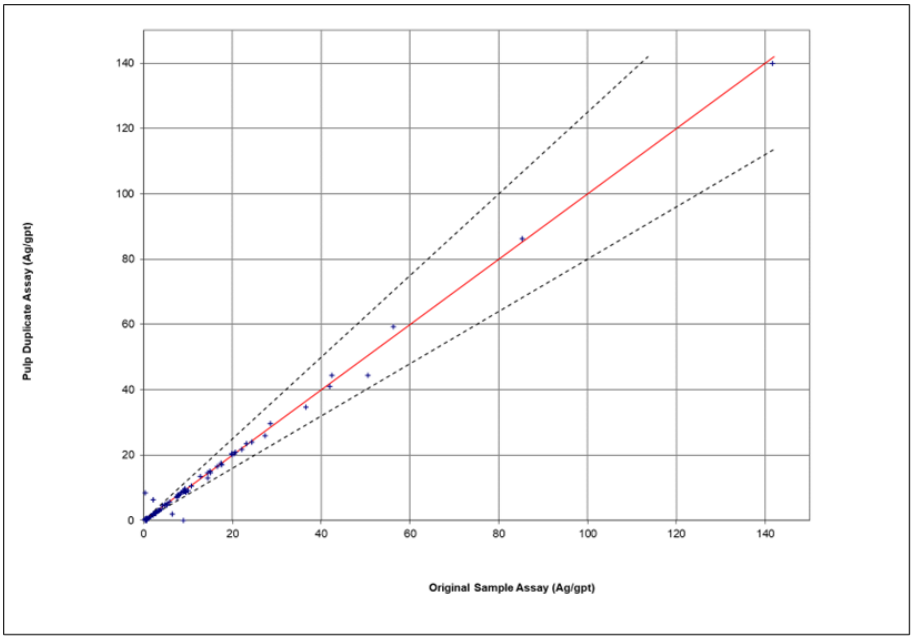
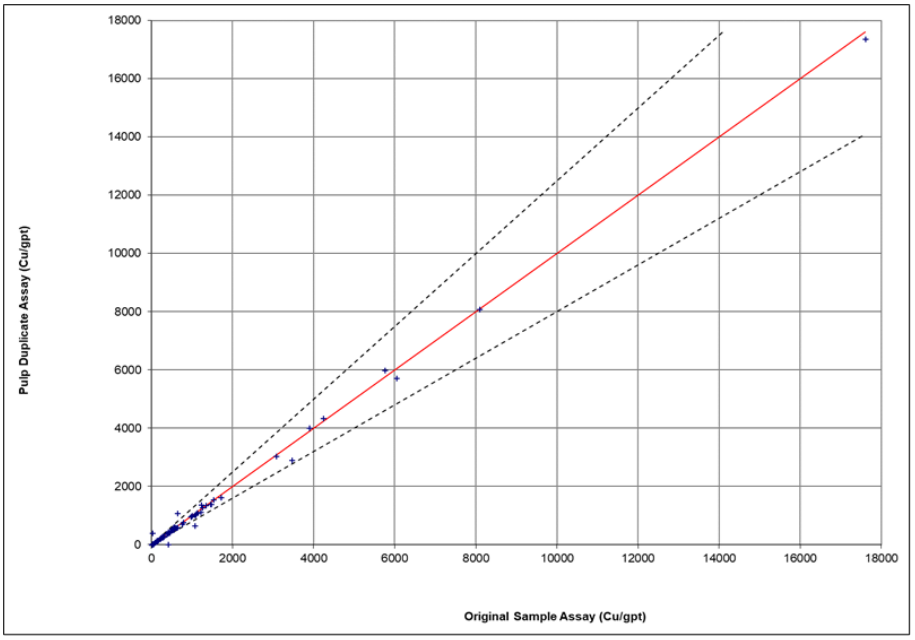


Figure 11.18 – Copper Pulp Duplicate Scatterplot



11.3.4 Recommendations and Enhancements to QA/QC Program

The QP strongly recommends that procedures for investigating, correcting, and documenting results of QA/QC non-compliance issues such as biases or failures are followed for all drilling programs. Non-compliance results include:

- 2013:
  - High bias observed in gold CRM SN16 since the expected grade of 8.37 g/t Au is very near the average gold grade of the Loma Larga Mineral Resource.
  - High bias observed in silver analyses after a brief hiatus in testing should be followed up with the laboratory.
- 2017:
  - High bias observed in CRM SN16.
  - Initial high bias of CRM SI15.
  - Calibration issues with CRM SN60.
  - Low bias in CRM SG66.
  - Potential high bias in CRM SJ63.

The biases in the CRM results have been observed throughout several drilling campaigns at Loma Larga and the QP recommends that INV completes an external check on the reference materials used on the Loma Larga Project.

The QP recommends that field duplicate analyses are carried out using half core duplicates. To ensure that the results are meaningful in the context of the Loma Larga resource, the QP recommends that duplicate field samples are collected within mineralised core intersections. Duplicate analyses can be achieved by using the existing core library.

Furthermore, the QP recommends the resumption of the regular submission of check assays (pulp replicates) to a secondary laboratory.

In 2017, INV introduced eight (8) gold analytical standards; however, only four (4) had sufficient data to be meaningfully analysed. Of the four (4) CRMs, two (2) had grades approximately 8 g/t Au (SN16 at 8.37 g/t Au and SN60 at 8.595 g/t Au), and two (2) CRMs were less than 3 g/t Au (SG66 at 1.156 g/t Au and SJ63 at 2.488 g/t Au). The QP recommends that the number of CRMs used are reduced and ensure that the grades are consistent with the ranges expected to be found at Loma Larga (up to and greater than 30 g/t Au), as recommended in 2016. Similarly, the QP reiterates the 2016 recommendation that analytical standards are obtained for silver and reintroduced a copper CRM.

11.4 QP's Opinion

The QP is of the opinion that the results of the QC samples, together with the QA/QC procedures implemented at Loma Larga, provide adequate confidence in the data collection and processing, and the assay data is suitable for Mineral Resource estimation.

12 DATA VERIFICATION

12.1 Site Visit

QP Katharine Masun, P.Geo., SLR Principal Resource Geologist visited the Loma Larga property site on February 19, 2014, and February 25. 2025. During the site visit in 2014, Ms. Masun inspected the Loma Larga property, including the location of drill collars LLD148, LLD149 and LLD370. In 2025, Ms. Masun verified the drill collar locations of LLD385, LLD389, and LLGT006.

In 2014, Ms. Masun reviewed the geological core, checked lithology, mineralisation, and sampling against drill logs of the following drillholes: IQD122, IQD183, IQ210, LLD367 and LLD372. In 2025, Ms. Masun reviewed the geological core, checked lithology, mineralisation, and sampling against drill logs of the following drillholes: LLD386, LLDHG011, and LLD389. During the core review, no notable discrepancies were found; metre tags were placed in the correct locations in the core boxes, samples were clearly and accurately marked, and core boxes were clearly labelled.

12.2 Manual Database Verification

Sampling details for the historic drilling program by IAMGOLD were verified by RPA (now SLR) in 2006. At that time, the RPA QP validated the drillhole database up to hole IQD354. In 2012, SLR's QP verified 30 drillholes completed by IAMGOLD in 2008, which included 28 resource delineation drillholes and two (2) drillholes for metallurgical testwork. Prior to accepting the resource database used to estimate the current Mineral Resources for the Loma Larga Project, the QP reviewed and verified 12 drillholes completed by INV in 2013, which included drillholes LLD364 to LLD375. The verification work included a review of the QA/QC methods and results, checking assay certificates against the database assay table, a site visit and review of drill core, and standard database validation tests.

In 2018, the QP verified the assay results from 23 drillholes that were completed subsequent to the 2016 Technical Report. Verification included checking assay certificates against the database assay table. The QP also completed standard database validation tests of the new drilling.

The QP considers the resource database reliable and appropriate to support a Mineral Resource Estimate.

In addition, the QP reviewed and verified the 248 drillholes with sulphur data, including 26 drillholes that were completed by INV during the 2017 drilling campaigns on the Project. The verification work consisted of reviewing the QA/QC results, checking assay certificates against the database assay tables and standard manual validation tests.

The review of the resource database included header, survey, lithology (major and minor), assay, and density tables. Database verification was performed using tools provided within the Dassault

Systèmes GEOVIA GEMS Version 6.6 software package (GEMS). As well, the assay and density tables were reviewed for outliers. Any inconsistencies that were identified were promptly corrected by INV.

A visual check of the drillhole collar elevations with respect to the topographic surface and drillhole traces was completed. The QP noted 19 drillhole collars within the mineralised wireframe domains that were greater than one (1) metre above the topographic surface, and four were greater than three (3) metres. The QP followed up these discrepancies and found that it was not a transcription error. The QP recommends that INV resurvey the drillhole collars listed in Table 12.1.

Table 12.1 – Drillhole Collar Errors

Drillhole	Collar Error (m)
IQD253	6.44
IQD145001	5.37
IQD330	4.61
IQD314	4.06
IQD142501	2.95
IQD133	2.26
IQD190	2.12
IQD181	2.09
IQD165	2.02
IQD176	1.76
IQD146	1.72
IQD179	1.37
IQD142	1.32
IQD344	1.24
LLD375	1.19
IQD138	1.14
IQD175	1.14
LLD370	1.10
IQD163	1.04

The QP verified all 1,722 assay records. This included a comparison of assay results in the resource database to 18 digital laboratory certificates of analysis (COA), which were received directly from Inspectorate. Any inconsistencies that were identified were promptly corrected by INV.

The QP verified 2,423 assay records to 30 digital COAs from 23 drillholes that were sampled during INV's 2017 drilling programs. A visual check of the drillhole GEMS collar elevations and drillhole traces was completed. No errors or inconsistencies were found.

12.3 Independent Assays of Drill Core

The QP did not collect samples from drill core for independent assay during the 2014 site visit.

In 2005, Wayne Valliant, P.Geo., Principal Geologist with Scott Wilson RPA, collected seven (7) samples of quartered core for independent analyses at SaGS Minerals Services, Toronto. Analyses were by fire assay for gold and ICP for silver and copper. Although seven (7) samples are not sufficient for statistical comparisons, Scott Wilson RPA (2006) found the agreement to be reasonable and confirmed the presence of gold in the samples.

12.4 QP's Opinion

The QP is of the opinion that database verification procedures for the Loma Larga Project comply with industry standards and are adequate for the purposes of Mineral Resource Estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This is an update of the Loma Larga FS dated April 8, 2020, issued on November 29, 2021, and filed on SEDAR+.

13.1 Introduction

- Metallurgical testing data for ore scheduled to be processed in the LOM plan indicates that the recovery models and methods are appropriate.
- Samples used to generate the metallurgical data have been representative, and support the estimates of LOM performance.
- The QP is not aware of any processing factors or deleterious elements that could have a significant impact on potential economic extraction.

13.2 Historical Metallurgical Testwork

A review of the available historical metallurgical data was carried out, with a focus on obtaining pertinent results to the current process design. The findings of the review are presented in the following sections.

13.2.1 IAMGOLD METALLURGICAL TESTWORK PROGRAMS – 2006 TO 2010

Numerous metallurgical testwork programs were carried out by IAMGOLD between 2006 and 2010. In total, a set of 26 samples with gold grades ranging between 5 and 42 g/t and copper grades between 0.30 to 5.40% were received at SGS Lakefield, Ontario, Canada, and tested.

The focus of these programs was to identify suitable conditions to maximise gold recovery via a hydrometallurgical processing route to generate gold doré on-site at Loma Larga.

The following results are relevant to the current version on the Loma Larga flowsheet:

- The ore was identified as being refractory in nature.
- Mineralogical testing indicated that the bulk of the gold (72%) in the ore is associated with pyrite, followed by enargite (12%) and the remainder is free and or locked within siliceous gangue.
- A total of nine (9) Bond Ball Mill Work Index tests were completed. Results ranged from 15.2 kWh/t to 18.0 kWh/t, with an average value of 16.8 kWh/t.
- Gravity preconcentration achieved less than 10% recovery from the contained gold in feed material.
- Whole ore cyanidation achieved low gold recoveries (16 to 30% of the gold in the feed) with high cyanide consumption levels.

- Flotation testing achieved high gold and copper recoveries of 90 to 93% and 94 to 96%, respectively, into a single bulk concentrate containing enargite and pyrite.

13.2.2 INV METALS PRE-FEASIBILITY METALLURGICAL TESTWORK PROGRAM – 2014

The focus of the 2014 metallurgical program was to investigate possible flotation configurations to produce saleable concentrate(s) from the Loma Larga deposit. Both bulk and sequential (copper, pyrite) flotation circuits were investigated using a series of open circuit tests. A locked cycle test was subsequently completed for the sequential flotation flowsheet. The results of the 2014 metallurgical program were used in the 2016 Pre-Feasibility Technical Report prepared by RPA.

The following tests were carried out:

- Physical characterisation and mineralogy.
- Bulk rougher flotation.
- Bulk rougher cleaner flotation.
- Sequential cleaner flotation.
- Locked cycle flotation.

The two (2) composite (59 kg and 30.5 kg) samples were prepared from 230 kg of quartered HQ drill core intervals by SGS Lakefield. Drillholes LLD 371 and LLD 372 contributed the sample material for both composites. Composites head assays were found to be 12.5 g/t and 6.38 g/t Au and 0.59% Cu and 0.51% Cu, respectively.

The 2014 testwork program demonstrated, at a preliminary level, the following:

- The ore can be considered moderately hard, as assessed by the bond work index.
- The ore is highly abrasive.
- Bulk flotation results in significant gold losses at the cleaner flotation stage.
- Sequential flotation resulted in the production of saleable concentrates. However, reagent consumption levels needed to be examined further.

The conclusions from the 2014 metallurgical testing program were that a sequential flotation process producing separate enargite and pyrite concentrates was identified as being the preferred process route. The sequential flotation consisted of producing a high pH pulp in a rougher copper stage to depress pyrite (and contained gold) followed by a low pH pyrite flotation stage via the addition of acid to the copper rougher tails stream.

13.2.3 TESTWORK FOR 2019 FEASIBILITY STUDY

The metallurgical testwork program conducted in 2017-2018 was designed to optimise the flotation flowsheets and reagent schemes from the 2014 program. Both bulk and sequential flotation flowsheets were investigated, as each option assessed in earlier works was not considered to be optimised (high reagent consumptions).

The metallurgical testwork program consisted of:

- Physical characterisation of three (3) composites and ten (10) variability samples.
- Ninety-one (91) x Rougher/Rougher Cleaner tests on 14 composites.
- Twelve (12) x Variability Rougher tests on 10 variability composites.
- Forty-six (46) x Cleaner Flotation tests.
- Four (4) x Locked Cycle tests.
- Pilot-scale Staged Flotation Reactor (SFR) tests.
- Tails samples collected for downstream characterisation testing.

13.2.3.1 Sample Selection

The source material for the metallurgical program was the remaining composite material from the 2014 metallurgical program and drill core from Loma Larga 2017 drill program. In total, 14 composites were prepared over the life of the program.

It should be noted that the samples used for the metallurgical testwork were at SGS Lakefield prior to the commencement of the feasibility study program. It is understood that all samples are from diamond drill cores, drilled between 2014 and 2017. The impact of sample age appears to be minimal with respect to metallurgical performance, however it was not checked in detail.

The samples were prepared by INV Metals on-site at Loma Larga and shipped to SGS Lakefield in Ontario, Canada.

The selection of drill core intervals for the testwork composites and variability samples was carried out by INV. Orix Geoscience Inc. performed a comprehensive analysis of the ore types within the deposit, as shown in Table 13.1, and concluded that the samples chosen by INV were representative of the overall deposit.

A Life-of-Mine (LOM) plan was generated by INV and an updated production plan was developed by BBA which generated a high-level process plant production schedule.

At the time of the sample selection for the 2017-2018 testwork program, the production schedule and LOM plan generated in the feasibility study was not available.

Table 13.1 – Ore Type Categories

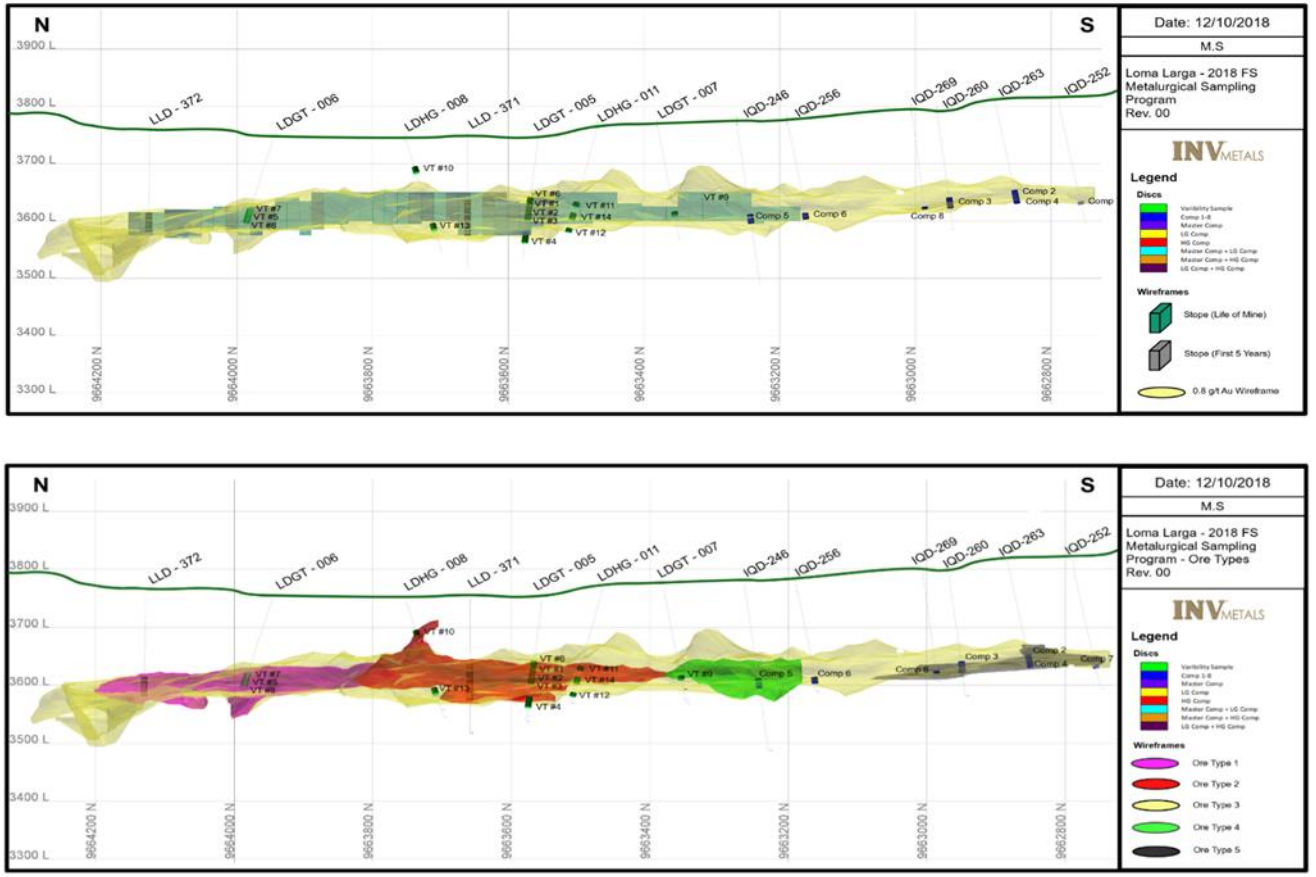
Geological Data Type		TYPE 1	TYPE 2	TYPE 3	TYPE 4	TYPE 5
PLNDDH		2,3	6	1,4,5,7,8,9,10,11	12, 13	14, 15, 16
Lithology		Tuff, BxHyd, And, CBx, MBx	Tuff, MBx, BxHyd, RBx	Tuff, And, CBx, BxFr, BxHyd	Tuff, And, MBx, CBx, RBx	Tuff, RBx, BxHyd, MBx, CBx
Alteration		Sil, Kaol, Dck, Alun	Sil, Kaol, Alun, Dck	Sil, Kaol, Alun, Dck	Sil, Kaol, Alun, Pyro	Sil, Alun
Mineralization		Py, En (Ba, Sph, Lz, Cpy, Cv)	Py, En (Ba, Sph, Lz, Tn)	Py, En (Ba, S, Sph, Tn, Lz)	Py, En (Ba, Sph, S, Tn, Gal)	Py, En (Ba, Gt, S, Tn)
Assays	Au	Moderate-High	Moderate-High	Moderate	Moderate	Moderate-Low
	Ag	Moderate-High	Moderate-High	Moderate-Low	Moderate	Moderate-High
	Cu	Moderate-High	Moderate-High	Moderate-Low	Moderate	Moderate-Low
	As	Moderate-High	Moderate-High	Moderate-Low	Moderate	Moderate-Low
	Sb	High	Moderate-High	Low	Moderate	Moderate
	Hg	High	Moderate-High	Low	Moderate	High
	Mo	Moderate-High	Moderate	Moderate-Low	Low	Moderate
	Bi	Moderate-High	Moderate	Low	Low	Low
	Fe	Moderate-High	Moderate-Low	Low	High	High
	Pb	Moderate-Low	Low	Low	Moderate-High	Moderate-Low
	Zn	Moderate-Low	Moderate-Low	Moderate-Low	Moderate	Low

Source: Ore type categories work performed by Orix Geoscience Inc., 2016

Figure 13.1 illustrates the location of each drillhole and the variability sample intervals, with respect to the ore body.

Drillholes LLDGT-005 and LLDGT-006 were used to form the majority of the composites and as the source of the variability samples for the Project. Both LLDGT-005 and LLDGT-006 are within the boundary of the deposit and intersect Types 1 and 2 ore types. Ore types are defined in accordance with Figure 13.1.

Figure 13.1 – Drillhole Locations and Ore Types



Source: INV, 2018

13.2.3.2 *Head and Mineralogy Characterisation*

Head assays and mineralogical analysis was carried out on subsamples on a number of composites and variability samples. Head assays for gold ranged from 1.06 to 15.4 g/t, averaging 5.6 g/t Au. Head assays for copper ranged from 0.033 to 4.12% Cu, averaging 0.50% Cu, while sulphur assays ranged from 3.95 to 15.3% S, averaging 8.93% S.

Mineralogy was consistent for all samples examined.

At a P_{80} of 75 µm, liberation (>80% of the cross-sectional particle area) averaged 85% for pyrite, with little variance in this value across the samples evaluated. The liberation characteristics for enargite were not as favourable, averaging 59%, with considerably higher variance. Non-liberated enargite was more commonly associated with pyrite than quartz, while non-liberated pyrite was associated largely with quartz.

13.2.3.3 *Physical Characterisation*

Physical characterisation testwork was conducted on three (3) different composite samples and ten (10) variability samples.

For the coarser particle grindability tests, i.e., Drop Weight Test (DWT), SAG Mill Comminution (SMC), and SAG Power Index (SPI), the indices indicate samples were medium to moderately hard or competent on average in comparison to the SGS database. For the mid-range particle grindability tests, i.e., RWI, samples averaged moderately hard, while for the fine particle grindability tests, i.e., BWI, samples were considered moderately hard on average.

13.2.3.4 *Flotation Testwork*

Flotation testing was completed to evaluate the performance of the bulk and sequential flowsheets. The flotation testwork was performed to provide sufficient information for selection of the optimal processing flowsheet and design and to allow for the estimation of recoveries, concentrate grades, Capex and Opex to a feasibility levels.

All flotation testing was carried out at the SGS facilities in Lakefield, Ontario.

The sections below summarise the results of the bulk and sequential flowsheet options investigated during the study. The outcomes of these investigations were as follows:

- The sequential flowsheet achieves higher grade copper and pyrite concentrates.
- The sequential flowsheet delivers higher overall metal recoveries to final concentrate.
- The sequential flowsheet achieved these recoveries utilising lower reagent dosages.

13.2.3.5 Bulk Flotation

The bulk flotation circuit had considered that enargite and pyrite were to be recovered together in a bulk concentrate, using a bulk rougher and three (3) stage cleaning separation process; followed by selective cleaning to produce both copper and pyrite concentrates. The concentrate of the copper cleaning circuit being the final copper concentrate and the combined tails streams the pyrite concentrate. Both concentrates contain both gold and silver.

Two (2) bulk circuit rougher tests and a further 33 combined bulk rougher-cleaner tests were completed.

The first test (F23) was carried out using the high-grade gold "Blend composite" at a primary grind of 80% passing 60 µm, 1.3 kg/t Lime, 30 g/t PAX, 15 g/t 3418A (copper collector) and 25 g/t MIBC. Recoveries of the major elements were 95.7% for copper, 95.1% for arsenic, 93.6% for gold and 88.8% for silver.

The second test (F32) rougher only bulk test was carried out to evaluate the staged use of 3418A versus PAX. The test conditions being a low pulp pH (7.0), 3418A (15.0 g/t), PAX (30.0 g/t) and MIBC (20.0 g/t). Overall recoveries were 94.7% for copper, 95.1% for arsenic, 92.9% for gold and 88.5% for silver.

The tests were carried out on Composite 1 and Blend composites.

The following observations were noted:

- Only F11 resulted in a reasonable copper concentrate recovery and grade of 76.6% and 30.3% Cu respectively.
- Gold recovery to copper concentrate was on average 20%.
- Gold losses to the bulk rougher tails and first bulk cleaner tails were on average 10.5%.

Bulk variability tests were completed, including a bulk rougher stage and then multiple stages of bulk concentrate cleaning.

All of the variability tests were run at the following conditions:

- Primary grind of 80% passing 60 µm. Add lime to target pulp pH 7.0.
- Bulk rougher for 20 minutes after 6 minutes conditioning time with 15 g/t 3418A, 25 g/t MIBC, 25 g/t PAX.
- Regrind bulk concentrate for 15 minutes with 7.5 g/t 3418A and 15.0 g/t PAX.
- Bulk 1st cleaner for 10 minutes after two (2) minutes conditioning time with up to 5 g/t 3418A, 8.0 g/t MIBC & 10 g/t PAX as required. Some tests did not use extra reagents at all.
- Bulk 2nd cleaner for 10 minutes after one (1) minute of conditioning time with 8 g/t MIBC.

No extra lime was added to the regrind and cleaner stages and pulp pH stabilised between 7.0 - 8.0 in cleaner stages. Total PAX addition was up to 70 g/t.

The following was noted from the variability test results:

- Copper head grades were highly variable from 0.05 to 4.0% Cu.
- Copper recoveries ranged from 70.0 to 95.3% despite some very low copper grades.
- Arsenic responds in a similar manner to copper as expected as the predominant copper bearing mineral is enargite.
- Sulphur recoveries are also good but on average 5% lower than required to maximise gold recovery.
- The sulphur grade recovery curves were not the same as gold recovery curves, implying that the residual gold not recovered with copper is not totally associated with the pyrite.
- Gold head grades cover a very wide range of assays from 2.62 to 15.5 ppm Au. Gold recovery performance was highly variable from 71.5 to 90.9%, at grades of 14 to 52 g/t Au.

Second stage cleaning results are presented in Table 13.2.

Table 13.2 – LCT #2 Recovery to Bulk Second Cleaner Concentrate

Product	Weight	Assays (%, g/t)					Distribution (%)				
	(%)	Cu	As	S	Au	Ag	Cu	As	S	Au	Ag
Bulk 2nd Cl Conc	**11.25**	**1.68**	**0.63**	**48.4**	**36.1**	**203**	**87.0**	**82.9**	**86.9**	**82.2**	**82.6**
Bulk 1st Cl Scav Tail	15.27	0.025	0.016	1.35	1.45	6.66	1.8	2.9	3.3	4.5	3.7
Bulk Ro Tail	73.48	0.033	0.017	0.84	0.90	5.16	11.2	14.2	9.8	13.3	13.7
Combined Tail	**88.75**	**0.032**	**0.017**	**0.93**	**0.99**	**5.41**	**13.0**	**17.1**	**13.1**	**17.8**	**17.4**
Head (calc.)	100.0	0.22	0.086	6.27	4.95	27.6	100.0	100.0	100.0	100.0	100.0

Source: SGS, 2018

The overall recovery of copper to final copper concentrate was relatively low (40-80%) and that the concentrate grades were generally understood to be less than marketable. The cause for the relatively low-grades can be linked to the high pyrite content in all of the samples, ranging from 60 to 80% by weight.

13.2.3.6 Sequential Flotation

The sequential flotation circuit considered that enargite and pyrite are recovered sequentially and separately via separate copper sulphides and pyrite roughing and cleaning circuits.

In the first stage of flotation, enargite and other copper sulphides are preferentially recovered via the use of a moderately high pulp pH and a collector that is more selective to the recovery of copper sulphides vs. iron sulphides.

The copper concentrate collected may also recover associated pyrite in composites and as such, this concentrate stream is sent to a regrind circuit for further liberation before being cleaned in three (3) sequential stages of copper cleaner flotation.

The tails stream from the copper rougher and cleaner circuits is then combined and fed to the pyrite rougher flotation stage, where the pulp pH is reduced and a strong collector (PAX) introduced to activate pyrite for optimum recovery.

The recovered pyrite rougher concentrate is then reground to achieve suitable liberation and cleaned in two (2) sequential stages of cleaning for gangue rejection.

A total of 58 rougher, rougher-cleaner, and locked cycle tests were completed as part of the sequential flotation testing program.

13.2.3.7 Locked Cycle Tests

A total of four (4) locked cycle tests (LCTs) were completed for the 2017-2018 metallurgical testing program as follows:

- LCT #1 – Bulk flotation circuit tested using Blend Composite.
- LCT #2 – Bulk rougher cleaner of SFR concentrates.
- LCT #3 – Sequential flotation circuit using Low-Grade Comp.
- LCT #4 – Sequential flotation circuit using “Five-year Composite”.

The test conditions used for each of these tests, including results obtained and comments on results achieved, are provided in the following sections.

13.2.3.8 Locked Cycle Test #1

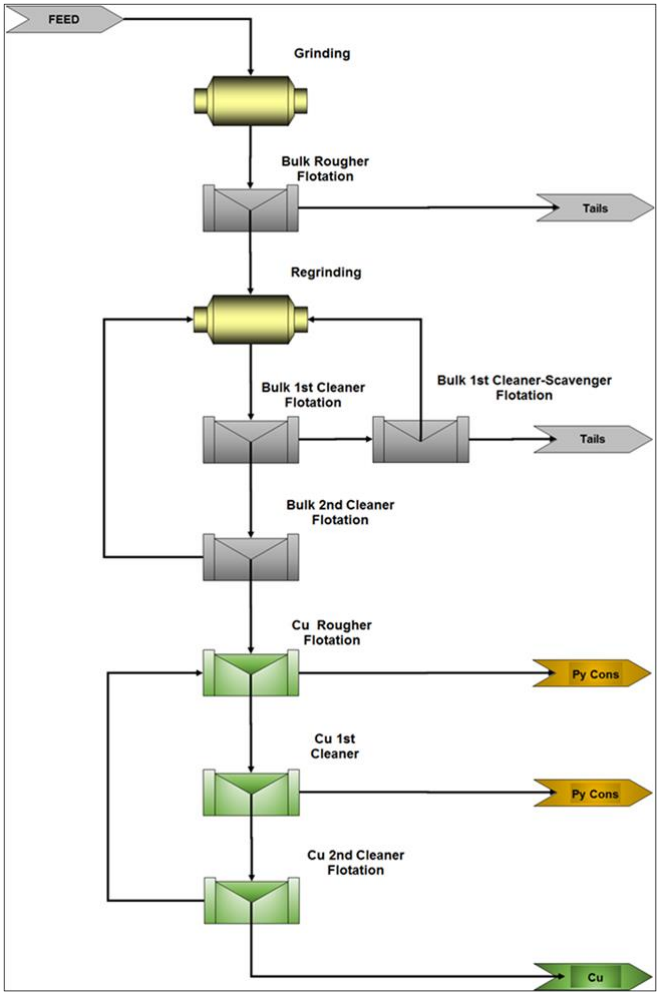
LCT #1 evaluated the performance of the bulk flotation flowsheet. Feed material for the test was the “Blend Composite”. The LCT #1 test conditions flowsheet is illustrated in Figure 13.2.

The target grind sizes, pulp pH and reagent additions and actual test conditions achieved were as follows:

- Primary grind target 80% passing 60 µm.
- Bulk concentrate regrind target 80% passing 30 µm - actual 29 µm.
- Each of the flotation stages included one minute of conditioning time.

- The bulk rougher stages included six (6) stages of flotation, the two (2) stages in the bulk 1st cleaner, two (2) stages in the bulk 2nd cleaner, and single stages thereafter.

Figure 13.2 – LCT #1 Flowsheet



Source: Promet 101, 2018

Table 13.3 outlines the results obtained during LCT #1. The results of the LCT showed a copper concentrate grade of 17.3%, with a mass recovery of 2.94% and overall metal recovery of 76.5%. Gold recovery to the pyrite concentrate was relatively poor at only 63.4%. The performance of this bulk circuit test indicates insufficient rejection of pyrite with associated gold from the copper circuit to the pyrite concentrates.

Table 13.3 – LCT #1 Results

Stage	Assays %, g/t (Au, Ag)						
		Cu	Au	Ag	As	S	Fe
Flotation Feed		0.66	17	70	0.23	12	10
Bulk rougher tail		0.04	1.4	5.4	0.01	0.8	1
Bulk rougher conc		1.97	49	207	0.68	35	29
Bulk 1st cl-scav tail		0.21	11	33	0.07	6	7.2
Bulk 2nd cleaner conc		2.68	64	277	0.92	47	38
Bulk circuit combined tails		0.06	2.5	8.8	0.02	1.4	1.7
Cu rougher conc		5.66	82	442	1.99	50	39
Cu rougher tail		0.44	52	146	0.11	46	39
Cu 1st cleaner tail		0.84	56	329	0.23	51	43
Cu 2nd cleaner conc		17.3	141	857	6.23	46	28
Pyrite conc		0.54	53	191	0.14	47	40
	Recoveries % of Feed						
	Masspull	Cu	Au	Ag	As	S	Fe
Bulk Rougher	32.3	95.7	94	95	96.1	96	94
Bulk 1st Cleaner	25.2	94.5	90	92	94.9	94	90
Bulk 1st Cl-scav	0.6	0.6	1.1	0.8	0.58	1.2	1.4
Bulk 2nd Cleaner	23	92.8	88	90	93.1	91	87
Cu Rougher	10.5	89.4	51	66	91.9	44	40
Cu 1st Cleaner	5.49	83.1	35	43	86.8	23	19
Cu 2nd Cleaner	2.94	76.5	25	36	80.8	12	8.1
Pyrite conc	20.03	16.3	63	55	12.4	80	79
	Stage Recoveries %						
	Masspull	Cu	Au	Ag	As	S	Fe
Bulk Rougher	32.3	95.7	94	95	96.1	96	94
Bulk 1st Cleaner	71.8	96.4	93	95	96.4	94	92
Bulk 1st Cl-scav	6.06	17	16	15	16.1	20	17
Bulk 2nd Cleaner	91.3	98.2	98	98	98.2	97	96
Cu Rougher	41.2	90	52	68	92.7	43	41
Cu 1st Cleaner	52.3	92.9	68	65	94.5	51	47
Cu 2nd Cleaner	53.6	92.1	72	84	93	51	43

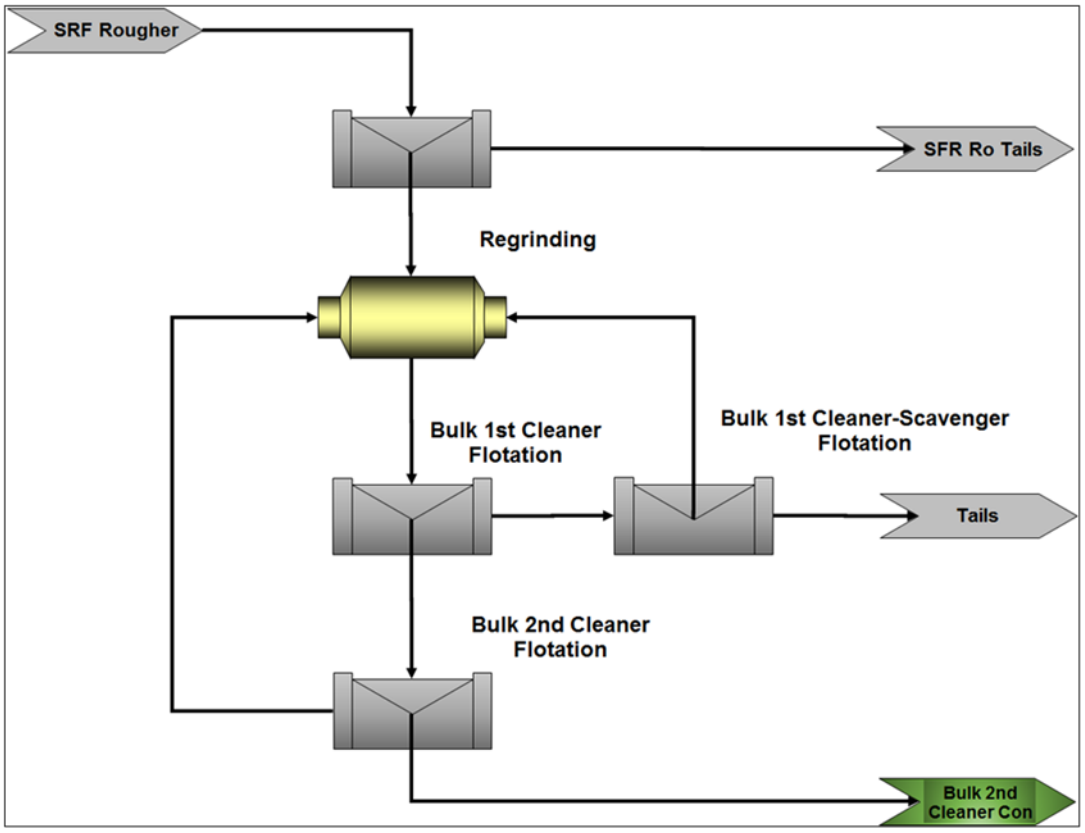
Source: Promet 101, 2018

13.2.3.9 Locked Cycle Test #2

LCT #2 was on the bulk concentrate generated as part of pilot testing of the Staged Flotation Reactor (SFR).

Figure 13.3 outlines the flowsheet used in LCT #2. It includes the SFR rougher to generate a bulk concentrate, concentrate regrind, two (2) stages of cleaning and a cleaner scavenger. The final concentrate generated would typically be re-cleaned at high pulp pH to depress pyrite and generate the two (2) final concentrates. As such, the second cleaner concentrate generated in this LCT represents the total recovery of copper and precious metals when using this circuit.

Figure 13.3 – Loma Larga 2017 Metallurgical Testing Program LCT #2 Flowsheet



Source: Promet 101, 2018

The feed material for this bulk cleaner locked cycle test was generated as part of the SFR pilot plant program using assay reject samples.

Table 13.4 outlines the results obtained during LCT #2.

Table 13.4 – LCT #2 Stream Assays

Stage	Assays %, g/t (Au, Ag)						
		Cu	Au	Ag	As	S	Fe
SFR Feed		0.21	5	28	6.3	0.09	5.7
SFR Rougher Con		0.71	16	89	21	0.27	18
Bulk 1st Cleaner Con		1.47	34	187	45	0.57	39
Bulk 1st Cl-Scav Con		0.23	14	57	15	0.12	15
Bulk 2nd Cleaner Con		1.66	36	203	49	0.63	41
SFR Rougher Tail		0.03	0.9	5.2	0.8	0.02	1.1
Bulk 1st Cl-Scav Tail		0.03	1.5	6.6	1.4	0.02	1.3
Combined Tail		0.03	1	5.4	0.9	0.02	1.1
	Recoveries % of Feed						
	Masspull	Cu	Au	Ag	As	S	Fe
SFR Rougher Conc	26.7	88.7	87	86	90	85.4	86
Bulk 1st Cleaner	12.7	87	87	86	91	84.8	87
Bulk 1st Cl-Scav	0.38	0.41	1.1	0.8	0.9	0.53	1
Bulk 2nd Cleaner	11.2	86.9	82	83	87	82.5	82
	Stage Recoveries %						
	Masspull	Cu	Au	Ag	As	S	Fe
SFR Rougher Conc	26.7	88.7	87	86	90	85.5	86
Bulk 1st Cleaner	44.5	97.5	94	95	96	96.1	95
Bulk 1st Cl-Scav	2.4	18.6	19	18	21	15.5	23
Bulk 2nd Cleaner	88.4	99.8	95	96	95	97.4	95

Source: Promet 101, 2018

The SFR gold and copper recoveries to the bulk concentrate were relatively low at 86.7 and 88.7%, respectively. Mass recovery was high at 26.7% with good sulphur recovery.

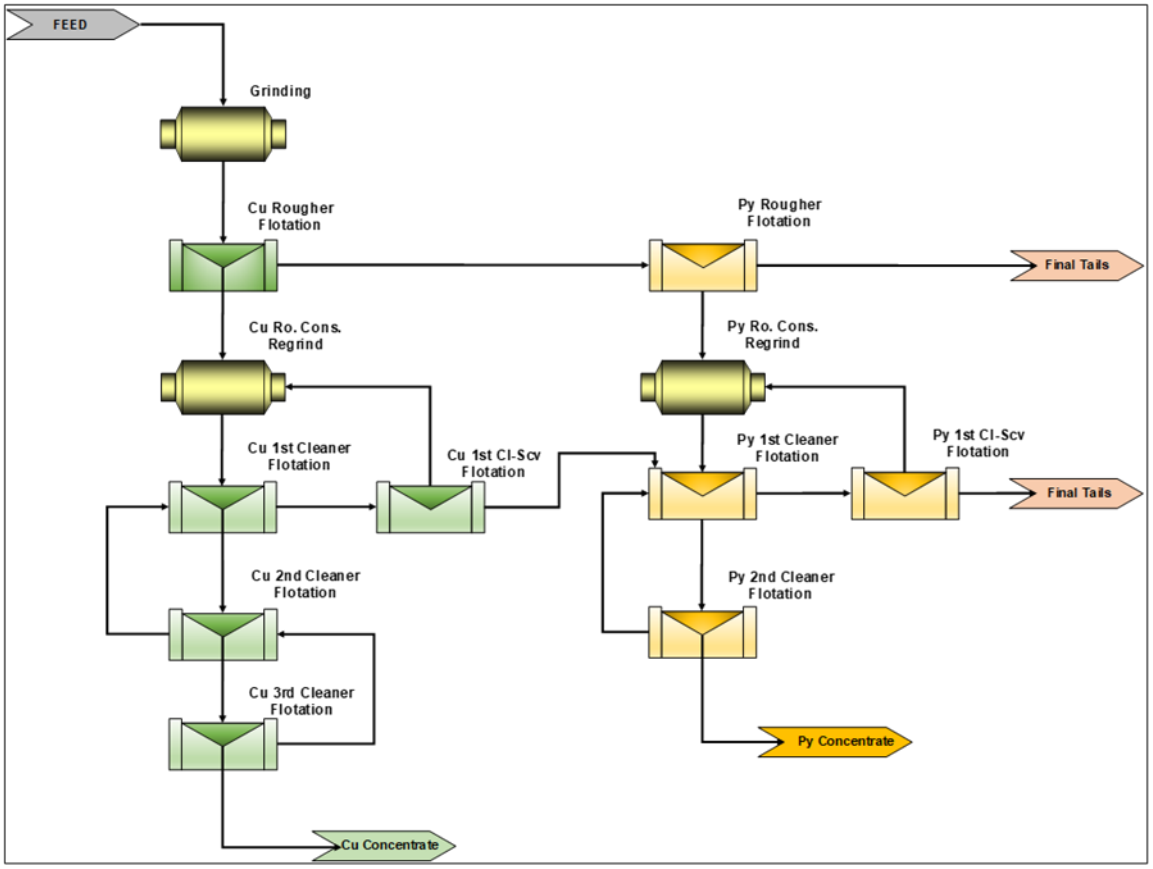
Subsequent regrind and cleaning resulted in a further recovery loss of 4.5 % and 1.8% for gold and copper, respectively. The maximum gold recovery possible from the bulk cleaned concentrate is 82.2%.

13.2.3.10 Locked Cycle Test #3

LCT #3 tested the sequential flowsheet at the conditions established during the previous sequential flotation optimisation phase of the program and approximate LOM head grade.

The flowsheet used for this LCT is presented in Figure 13.4.

Figure 13.4 – LCT #3 Flowsheet



Source: Promet 101, 2018

Table 13.5 – LCT #3 Sample Head Grade

Description	Head grades % or ppm							
	Copper	Gold	Arsenic	Iron	Sulphur	Zinc	Lead	Antimony
Composite	0.51	5.96	0.18	7.49	8.61	182	120	117

Source: SGS, 2018

For this test, four (4) stages of copper rougher collection, with a total of 12 minutes of residence time, followed by six (6) stage of pyrite rougher collection, with a total of 20 minutes of residence time, was used. The target grind sizes, pulp pH and reagent additions and actual test conditions achieved were as follows:

- Primary grind target 80% passing 60 µm.
- Copper regrind target 80% passing 30 µm.
- Pyrite regrind target 80% passing ~30 µm – Actual 29 µm.
- Pyrite rougher tails Actual 80% passing 58 µm.

- Conditioning time post grinding is five (5) minutes.

The results of LCT #3 were statistically balanced to obtain stage by stage recoveries and performance. This balance was the basis for process design and flotation circuit sizing. Table 13.6 to Table 13.8 present the key performance data obtained for the individual stages of the flotation circuit for LCT #3.

Table 13.6 – LCT #3 Stream Balance Data

Stage	Assays %, g/t (Au, Ag)					
	Cu	Au	Ag	As	S	Fe
Flotation Feed	0.51	5.96	33.5	0.180	8.61	7.50
Copper Rougher	11.5	46.1	493	4.23	20.2	8.29
Copper 1st Cleaner	30.2	96.3	1,270	10.9	36.0	14.1
Copper 2nd Cleaner	32.2	101	1,348	11.7	36.0	12.9
Copper 3rd Cleaner	33.5	104	1,397	12.1	35.9	12.2
Copper Cl-Scav	3.10	39.4	239	1.09	41.8	33.4
Pyrite Rougher	0.18	16.4	50.1	0.039	34.2	30.4
Pyrite 1st Cleaner	0.28	22.7	61.2	0.089	42.9	36.9
Pyrite 2nd Cleaner	0.30	24.1	66.9	0.092	46.4	39.6
Pyrite Cl-Scav	0.13	11.5	17.29	0.059	17.5	14.9
Final Tails	0.02	0.98	5.38	0.005	1.07	1.39

Source: Promet 101, 2018

Table 13.7 – LCT #3 Overall Recoveries Balanced Data

Stage	Recoveries % of Feed						
	Masspull	Cu	Au	Ag	As	S	Fe
Copper Rougher	4.01	90.2	31.1	59.0	94.2	9.42	4.44
Copper 1st Cleaner	1.49	88.3	24.1	56.5	90.5	6.24	2.80
Copper 2nd Cleaner	1.40	88.2	23.6	56.1	90.4	5.84	2.41
Copper 3rd Cleaner	1.33	87.4	23.1	55.5	89.7	5.55	2.16
Copper Cl-Scav	0.10	0.61	0.66	0.72	0.60	0.49	0.45
Pyrite Rougher	21.5	7.48	59.0	32.1	4.60	85.2	87.1
Pyrite 1st Cleaner	17.7	9.71	67.3	32.3	8.76	88.3	87.1
Pyrite 2nd Cleaner	15.6	9.10	63.2	31.2	8.01	84.2	82.5
Pyrite Cl-Scav	0.13	0.03	0.25	0.11	0.04	0.26	0.25

Source: Promet 101, 2018

Table 13.8 – LCT #3 Stage Recoveries Balanced Data

Stage	Stage Recoveries %						
	Masspull	Cu	Au	Ag	As	S	Fe
Copper Rougher	4.01	90.2	31.1	59.0	94.2	9.4	4.44
Copper 1st Cleaner	34.9	96.4	73.7	93.1	94.7	58.9	50.6
Copper 2nd Cleaner	90.3	99.0	96.2	98.3	99.1	89.4	79.0
Copper 3rd Cleaner	95.0	99.2	97.8	98.9	99.2	95.0	89.6
Copper Cl-Scav	3.60	18.3	7.67	17.1	11.7	11.2	16.5
Pyrite Rougher	22.4	75.9	85.5	78.2	78.6	94.1	91.1
Pyrite 1st Cleaner	67.2	89.5	94.5	87.8	88.6	94.4	92.4
Pyrite 2nd Cleaner	88.3	93.7	94.0	96.5	91.4	95.4	94.7
Pyrite Cl-Scav	1.50	2.63	6.33	2.44	3.54	5.00	3.51

Source: Promet 101, 2018

The feed composite used for LCT #3 was relatively close in grade to that of the expected feed gold grade of the process facilities for the first five (5) years of operation, but almost double that of the copper grade. A relatively high pulp pH was used to ensure low pyrite recovery in the copper rougher stage, with a relatively low mass recovery being achieved (~4.01%) and good copper (90.2%) and arsenic (94.2%) recoveries. Subsequent copper cleaner stages rejected 66.8% of the mass from the rougher concentrate, but only lost 2.8% of the copper. Gold and sulphur rejected from the copper rougher concentrate being 8.0 and 3.9% respectively. The copper concentrate grade achieved in the cleaning stages was 33.5% Cu, 12.1% As, 1,397 g/t Ag and 104 g/t Au. Gold recovery to the copper concentrate was 23.1% of total gold.

The pyrite rougher stage recovered 85.5% of the feed gold and 94.1% of the feed sulphur into 21.5% of the total feed. Subsequent cleaning of the pyrite rougher concentrate resulted in a total of 63.2% of feed gold being recovered into the final pyrite concentrate. Note that for this test the copper cleaner tails were sent to the pyrite first cleaner stage, not the pyrite rougher feed.

Overall gold recovery was 86.3% at a feed grade of 5.96 g/t gold, as compared to 88.3% for LCT #1 with a feed grade of 16.7 g/t gold and 86.9% for LCT #2 at a feed grade of 4.95 g/t gold.

The different recoveries of copper and arsenic in the copper cleaner-scavenger stage implies that there is some arsenic present in the form of arsenopyrite which is depressed, or non-arsenic associated copper minerals. Arsenopyrite was noted in some of the QEMSCAN analysis.

13.2.3.11 Locked Cycle Test #4

The objective of LCT #4 was to test a more representative sample of the initial 5-year average LOM plan and confirm the influence of collector addition and residence time on the gold recovery within the pyrite circuit.

Feed material for LCT #4 was composited from sub-samples of the "5-year composite" and individual variability samples. The sample head grade of the final composite is presented in Table 13.9.

Table 13.9 – LCT #4 Sample Head Grade – 5-Year Composite

Head grades % or ppm							
Cu	Au	As	Fe	S	Zn	Pb	Sb
0.25	5.10	0.092	5.94	6.47	177	198	101

Source: SGS, 2018

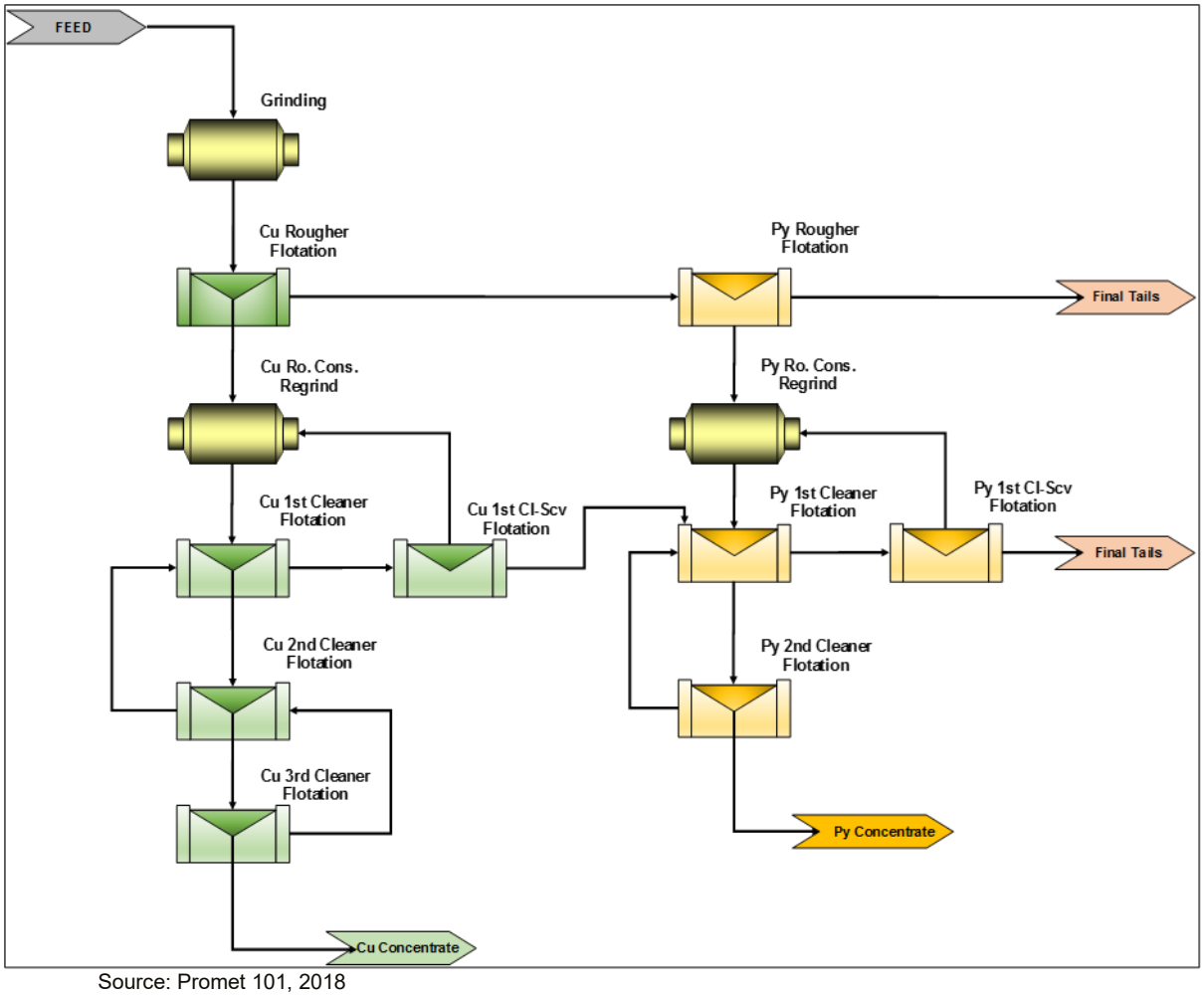
This test utilised four (4) stages of copper rougher collection with a total of 12 minutes of residence time, followed by six (6) stages of pyrite rougher collection, with a total of 20 minutes of residence time.

The target grind sizes, pulp pH and reagent additions and actual test conditions achieved were as follows:

- Primary grind target 80% passing 60 µm – Actual 67 µm.
- Copper regrind target 80% passing 30 µm – Actual 17 µm.
- Pyrite regrind target 80% passing 30 µm – Actual 31 µm.
- Pyrite rougher tails Actual 80% passing 55 µm.
- One (1) minute of conditioning time.
- Moderate copper rougher pulp pH was targeted to limit lime consumption and acid use in pyrite rougher feed.

The flowsheet used for this LCT is presented in Figure 13.5.

Figure 13.5 – LCT #4 Flowsheet



Source: Promet 101, 2018

The results of LCT #4 were balanced to obtain stage by stage recoveries and performance. This balance (in conjunction with the results of LCT #3) was the basis for the project process design and flotation circuit sizing stage recoveries. Table 13.10 to Table 13.12 present the key performance data obtained for the individual stages of the flotation circuit for LCT #4.

Table 13.10 – LCT #4 Stream Assays – Balanced Data

Stage	Assays %, g/t (Au, Ag)					
	Cu	Au	As	Ag	S	Fe
Flotation Feed	0.25	5.15	0.093	24.1	6.38	5.98
Copper Rougher	5.59	23.1	2.07	304	18.6	18.9
Copper 1st Cleaner	15.1	43.7	5.55	932	27.8	16.9
Copper 2nd Cleaner	22.4	54.1	8.12	1,340	31.1	15.3

Stage	Assays %, g/t (Au, Ag)					
	Cu	Au	As	Ag	S	Fe
Copper 3rd Cleaner	27.4	61.7	10.1	1,615	32.6	13.6
Copper Cl-Scav	1.67	24.8	0.59	226	24.0	20.9
Pyrite Rougher	0.094	19.9	0.028	55.6	27.5	23.9
Pyrite 1st Cleaner	0.27	27.6	0.092	76.7	37.7	33.4
Pyrite 2nd Cleaner	0.29	29.6	0.098	82.4	40.5	35.8
Pyrite Cl-Scav	0.090	6.04	0.044	0.98	6.51	7.15
Final Tails	0.013	0.73	0.006	1.24	0.67	1.12

Source: Promet 101, 2018

Table 13.11 – LCT #4 Overall Recoveries – Balanced Data

Stage	Recoveries % of Feed						
	Masspull	Cu	Au	Ag	As	S	Fe
Copper Rougher	4.08	89.8	18.3	50.8	90.8	11.9	12.8
Copper 1st Cleaner	1.40	83.2	11.9	53.8	83.5	6.09	3.96
Copper 2nd Cleaner	0.93	82.0	9.77	51.6	81.2	4.53	2.38
Copper 3rd Cleaner	0.74	79.8	8.85	49.4	80.2	3.77	1.68
Copper Cl-Scav	0.45	2.96	2.16	4.3	2.86	1.69	1.57
Pyrite Rougher	19.4	7.1	74.8	45.2	5.9	83.4	77.6
Pyrite 1st Cleaner	15.0	15.9	80.3	46.9	14.8	88.4	83.6
Pyrite 2nd Cleaner	13.8	15.7	79.1	46.2	14.4	87.2	82.3
Pyrite Cl-Scav	1.66	0.59	1.94	1.23	0.78	1.69	1.98

Source: Promet 101, 2018

Table 13.12 – LCT #4 Stage Recoveries – Balanced Data

Stage	Stage Recoveries %						
	Masspull	Cu	Au	Ag	As	S	Fe
Copper Rougher	4.08	89.8	18.3	50.8	90.8	11.9	12.8
Copper 1st Cleaner	27.0	86.5	50.6	90.4	86.1	38.4	23.7
Copper 2nd Cleaner	58.5	96.1	76.4	92.2	96.0	66.1	51.1
Copper 3rd Cleaner	79.6	97.3	90.6	95.8	98.7	83.2	70.6
Copper Cl-Scavenger	11.9	22.9	18.7	75.6	21.2	17.3	12.3
Pyrite Rougher	20.2	70.1	91.4	91.8	64.2	94.6	89.0
Pyrite 1st Cleaner	58.6	88.7	91.9	96.7	83.8	93.7	90.8
Pyrite 2nd Cleaner	91.8	98.5	98.5	98.5	97.4	98.6	98.4

Stage	Stage Recoveries %						
	Masspull	Cu	Au	Ag	As	S	Fe
Pyrite Cl-Scavenger	15.7	29.1	27.6	76.4	27.2	28.5	23.5

Source: Promet 101, 2018

For this LCT, the copper rougher stage recovered 89.8% of feed copper. Subsequent cleaning of rougher concentrate produced a cleaner concentrate, grading 27.4% copper and 1,615 ppm silver. This corresponds to an overall of recovery 79.8% copper. Final mass recovery to copper concentrate was low at only 0.74% of feed. Overall mass rejection in the copper cleaner circuit was 81.8%.

The extended pyrite residence time and increased collector dosage resulted in excellent sulphur (i.e., pyrite) and hence gold recovery to the pyrite rougher concentrate. Gold recovery was 91.5%, copper recovery 88.7% and sulphur recovery 94.6% of feed to that stage. It is noted that the copper cleaner tails stream with its associated gold was fed directly to the pyrite cleaner circuit.

The final gold recovery to the pyrite concentrate being 79.1%. Overall gold recovery achieved to the two (2) concentrates was 88.0%.

13.2.3.12 Staged Flotation Reactor

The potential application of Woodgrove Technologies' Staged Flotation Reactor (SFR) was investigated as part of the 2017 metallurgical testwork program. Seven (7) pilot plant scale SFR rougher tests were conducted at SGS Lakefield. Composite and variability assay reject material was used for all SFR testwork. The response to bench flotation of each pilot SFR test feed sample (after reagent addition) was determined by SGS, for comparison purposes.

Several findings were made during the SFR pilot plant preliminary program:

- The use of MIBC in conditioning and in the initial stages of the SFR tests increased metallurgical performance. MIBC performance increased when used in conjunction with X-133 frother.
- When directly compared to one another, X-133 frother outperformed W31 frother.
- Adding collector to the grind did not appear to positively affect overall metallurgical performance.
- PAX addition was effective when added in doses 10 g/t and larger, though adding significant amounts (30 g/t) did not result in improved metallurgical performance.
- Little additional recovery occurs in the final eight (8) minutes of bench flotation (from minutes 12 to 20). Although the SFR performance is not optimised, it is anticipated that the equivalent SFR performance for 12 minutes of bench-scale lab flotation time would be eight (8) stages of rougher SFRs.

13.3 2022 Metallurgical Testwork Program Update

In 2021, the Loma Larga Project was acquired by DPM who undertook a comprehensive review of the FS process design requirements. As a result of the review, the requirement to produce a high-grade copper concentrate was replaced by a lower grade gold-copper (Au-Cu) concentrate with gold and copper being payable metals. The arsenic grade needs to be limited to a maximum of 3.5% As per the smelter terms. This grade of arsenic corresponds to approximately 8.8% Cu grade in Au-Cu concentrate.

An ore composite and variability material from 2017-2018 testwork was available at SGS Lakefield, and a decision was made by DPM to complete a small scope program. Testing was completed from July to December 2022.

The 2022 metallurgical testwork program was designed to confirm results of the 2017 testwork and assess any opportunities with regards to:

- Impact of the coarsening primary grind size on the ore flotation response.
- Rougher concentrate regrind requirements.
- Evaluation of the number of cleaning stages required to achieve target grades for Au-Cu, and pyrite flotation circuits.
- The use of advanced flotation technologies including the Woodgrove SFR, Eriez Hydrofloat and GT Jameson cell technologies.
- Tailings settling, pressure filtration, and rheological properties for coarse material (125 µm target primary grind size).
- Generation of the large tailings sample for Paterson & Cooke tailings testing.

The drill core samples were selected from material that was used for the 2017-2018 program and stored at SGS facilities in Lakefield.

Two (2) composites were prepared for 2022 testwork program.

- LOM low-grade composite (COMP1) for metallurgical tests.
- Low-grade composite (COMP2) for Paterson & Cooke's tailings testwork.

Details on composites origin, weight, and expected key head grades can be found in Table 13.13.

Table 13.13 – 2022 Composite Selection Details

Composite	COMP1	COMP2
Number of Intervals	29	71
From, m	93	99
To, m	162	177
Drillholes	LLDGT-007, LLDGT-008, LLDGT-0011	LLDGT-007, LLDGT-008, LLDGT-0011
Ore Type	2, 3, 4	2, 3, 4
Lithology	Tb, BxHyd	Tb, BxHyd, Bx, LvP
Total Sample Interval Length, m	29.2	71.0
Total Sample Mass, kg	87.6	233.4
Expected Grades		
Au, g/t	4.91	2.32
Cu, %	0.26	0.07
S, %	6.30	5.87

Source: SGS, 2022

Two (2) rougher tests were completed to assess if there are any signs of sample aging and oxidation within the samples planned for COMP1, and results of the tests were compared to the rougher tests completed in 2017-2018. Based on this comparison, no signs of sample oxidation were found in the testing material for 2022 program.

Rougher concentrates produced from COMP2 were used for the Woodgrove's Benchtop Flotation Reactor (BFR) testing due to the small quantity of COMP1 ore (86 kg total).

Selection of drill core intervals for COMP1 testwork composites was carried out by DRA. Drillholes LLDGT-007, LLDGT-008, and LLDGT-011 were used to form the testwork composites. Intervals selected are within the boundary of the 2019 Feasibility Study mine production schedule and LOM plan and intersect Types 2, 3, and 4 ores types. Domaining was conducted with the Cancha geometallurgical software to ensure the composite representativity.

13.3.1 HEAD ASSAYS

Head assays for COMP1 and COMP2 can be seen in Table 13.14, and whole rock analysis for COMP1 is in Table 13.15.

Table 13.14 – COMP1 and COMP2 Head Assays

Item	Unit	COMP1	COMP2
Cu	%	0.3	0.071
Fe	%	5.3	5.35
As	%	0.12	0.032
Al	%	4,280	-
Ca	g/t	109	-
Co	g/t	24	-
Cr	g/t	177	-
Cu	g/t	3,030	-
Fe	g/t	53,700	-
K	g/t	784	-
Li	g/t	< 10	-
Mg	g/t	43	-
Mn	g/t	47	-
Na	g/t	322	-
Ni	g/t	< 20	-
P	g/t	57	-
Pb	g/t	273	-
Ti	g/t	759	-
Zn	g/t	393	-
Au	g/t	4.96	2.6
Ag	g/t	-	< 10
S	%	6.5	6.28
Cl (HNO_3 soluble)	g/t	< 10	-
F	%	0.007	-
Hg	g/t	2.5	-

Source: SGS, 2022

Table 13.15 – COMP1 Whole Rock Analysis

Item	Unit	Value
SiO_2	%	82.1
Al_2O_3	%	0.83
Fe_2O_3	%	7.4
MgO	%	0.03
CaO	%	0.01
Na_2O	%	0.06
K_2O	%	0.09
TiO_2	%	0.92
P_2O_5	%	0.02
MnO	%	< 0.01
Cr_2O_3	%	0.03
V_2O_5	%	< 0.01
LOI	%	5.34

Source: SGS, 2022

13.3.2 Mineral Content

QEMSCAN evaluation has been conducted for the COMP1 head ore and F8 open circuit cleaner test final tailings (pyrite rougher tailings). A summary of the results can be found in Table 13.16.

Table 13.16 – COMP1 Head Ore and Final Tails Mineral Content and Liberation

Sample	Head Ore	F8 Final Tails
Mineral	**% w/w**	
Pyrite	12.85	0.90
Enargite	0.68	0.10
Other Cu	0.03	0.03
Quartz	83.27	95.29
Remainder	3.17	3.68
% Liberation		
Pyrite	75.0	22.1
Enargite	56.9	1.0

Source: SGS, 2022

As seen from the table, mineral content and liberation generally corresponds to the samples tested in 2017-2018.

13.3.3 DIAGNOSTIC LEACH

A diagnostic leach tests was performed to determine the association of gold within various minerals within the sample tested including pyrite, arsenopyrite and other refractory minerals.

Diagnostic leach tests were performed for samples of COMP1 head ore, pyrite 1st cleaner tails (open circuit), and pyrite rougher tails (open circuit). Each test included an intensive cyanide leach followed by hot aqua regia acid leach to dissolve the residual minerals prior to cyanidation.

Just under half of the gold in tailings is considered sulphide locked in both the pyrite rougher and pyrite 1st cleaner tailings streams.

A large proportion of the cyanide leachable gold and silicate locked gold in tailings does not float readily and is therefore amenable for cyanidation. Silicate locked or gold associated with fine sulphides (a leach residue post cyanide and aqua regia leach) presents a minor fraction of gold losses to tailings. The quantity of this gold is higher for the pyrite rougher tailings due to the higher quartz content in that tails stream.

13.3.4 FRACTIONAL ANALYSIS

Fractional analysis of the head ore and flotation tails samples (same samples that were used for the diagnostic leach tests) was conducted in order to determine a distribution of major elements across the size fractions. A results summary can be found in Table 13.17.

Table 13.17– COMP1 Fractional Analysis

Sample	Size Fraction							
	+ 150 µm	+ 125 µm	+ 75 µm	+ 38 µm	+ 21 µm	+ 10 µm	- 10 µm	Total
Au Distribution, %								
Head Ore	6.0	6.5	19.2	17.7	26.0	10.7	13.9	100.0
Pyrite Rougher Tails		29.4	11.9	9.8	7.7	10.5	30.6	100.0
Pyrite 1st Cleaner Tails		2.1	8.7	13.1	5.9	7.0	63.2	100.0
Cu Distribution, %								
Head Ore	5.6	5.0	14.8	13.7	20.4	16.2	24.3	100.0
Pyrite Rougher Tails		34.9	16.4	13.5	3.5	4.8	26.8	100.0
Pyrite 1st Cleaner Tails		0.0	21.3	27.6	6.3	4.6	40.2	100.0
S Distribution, %								
Head Ore	5.7	5.9	18.1	19.9	27.8	11.2	11.4	100.0
Pyrite Rougher Tails		26.2	12.0	10.1	8.4	8.6	34.7	100.0
Pyrite 1st Cleaner Tails		0.0	12.8	15.0	4.6	3.6	64.1	100.0

Source: SGS, 2022

As shown in Table 13.17, the distribution of sulphides (elemental sulphur, copper, and gold) shows that the finer size fractions or ultra fines fraction contains higher sulphide contents compared to the head sample. For the pyrite rougher tails stream, it can be observed that the majority of sulphides present are locked in the plus 125 and minus 10 µm fines.

Pyrite first cleaner tailings is a product of the reground rougher concentrate flotation and have most of the sulphides concentrated in the minus 10 µm size fraction. These observations point to the necessity of the optimal regrind size for the rougher concentrates (both gold-copper, and pyrite) to avoid losses of the payable metals with extra fines.

13.3.5 Rougher Flotation

Rougher flotation tests were focused on evaluating the impact of ore primary grind size on gold and copper recoveries. The reagent scheme was used as per the rougher stage of the LCT3 locked cycle test form 2017-2018 program. A coarser primary grind withing the range indicated in the table has little impact on copper and gold recoveries, and weight recovery of the rougher concentrate. A pH of 11 at copper rougher flotation was still not high enough to prevent pyrite from activation resulted in additional gold recovery in copper circuit. The pH value was increased in open circuit cleaner tests starting from the test F6.

Rougher flotation kinetics shows little sensitivity to the primary grind size with a target grind size P_{80} between 65 and 125 µm.

13.3.6 Open Circuit Cleaners

The purpose of the open circuit cleaner tests was to evaluate any impact of the rougher concentrate regrind on recoveries and cleaner concentrate grade. Four (4) tests were conducted. Tests F5 and F6 had copper and pyrite rougher concentrate regrind target grind size P_{80} of 30 µm, and tests F7 and F_{80} had no rougher concentrate regrind.

Test F5 had a Cu rougher pH at 11, and this impacted gold recovery to a gold copper concentrate, further three (3) tests had copper rougher pH increased to 12, and gold distribution started reflecting the observed during 2017-2018 testing.

The cleaner tests conducted with a rougher concentrate regrind produce higher sulphur grades in both the gold copper, and pyrite cleaner concentrates due to the improved gangue rejection. Similar observations can be made for the cleaner concentrates Cu and Au grades.

Copper and gold recoveries to gold copper first cleaner concentrates are normally higher for tests with a regrind step. With respect to the second and third cleaner concentrates, copper recovery is marginally higher for tests without a regrind step. Gold recovery to the second and third cleaner concentrates is also marginally higher for tests F7 and F8 (no regrind). Test F5 completed with a

rougher concentrate regrind demonstrates a higher gold recovery to the second cleaner concentrate stream compared to the tests without a regrind.

Gold recovery to pyrite concentrates is marginally higher (excluding test F7) for tests without a regrind.

Results of the open circuit cleaner tests suggest that a decision on the introduction of the gold copper rougher concentrate regrind should be based on an evaluation of the added NSR value and capital and operating costs associated with a regrind circuit installation. Impact of the low head grade on the metallurgical response and concentrate grade should also be accounted for.

For the pyrite flotation circuit, a regrind is required to ensure target concentrate sulphur grades and metals upgrades are achieved.

13.3.7 Locked Cycle Tests

Two (2) LCTs were completed for COMP1. Each of the LCTs included a rougher and two (2) cleaning stages for both the copper, and pyrite circuits. LCT #1 had rougher concentrates regrind, and LCT #2 did not. Reagent dosages and flotation pH followed the F6-F8 open circuit cleaner test regimes.

A side-by-side comparison of LCT #1 and LCT #2 shows that the regrind of the rougher concentrates produces a higher gold grade, and substantially higher copper grade compared to “no regrind case”. The difference in overall copper recovery is marginal. Gold recovery is approximately 2.6% (24% relative difference) lower than for LCT #2.

For pyrite flotation, rougher concentrate regrind results in substantially higher gold, and sulphur grades for the concentrate with marginal difference in recoveries of gold and sulphur.

It is understood that results of the locked cycle testing support the regrind of rougher concentrates for both copper, and pyrite circuits due to the higher quality of the concentrates produced.

13.3.8 Woodgrove Tests

Woodgrove conducted a single-day flotation test program using their BFR to evaluate the applicability of Staged Flotation Reactor (SFR) technology for the Loma Larga ore. The testing was performed at the SGS laboratories in Lakefield, Ontario, Canada.

A BFR unit was used due to the insufficient quantity of both COMP1 and COMP2 material available for the SFR testing.

The study was performed using the low-grade composite (COMP2) with the purpose of obtaining a similar grade as the tails of the COMP1 bench test (F6) performed by SGS during their flotation testwork program. Overall, five (5) tests were performed: three (3) for the pyrite cleaner, and two (2)

for the copper cleaner. Pyrite tests were carried out with MIBC and PAX reagents while copper tests were carried out with MIBC and 3418A reagents.

During the BFR tests it was observed that formation of a stable froth is problematic. The issue was attributed to usage of MIBC, which is not a strong enough frother for the BFR operation. Insufficient quantities of test material did not allow for additional work, and therefore, more tests recommended to optimise the reagents scheme and process parameters will occur once sample are available in sufficient quantities.

13.3.9 JAMESON DILUTION TEST

A single bench scale Jameson dilution test was completed on COMP1 material.

The flowsheet of the test included rougher flotation, rougher concentrate regrind, and three (3) cleaning stages for each of the circuits (copper and pyrite).

Grinding, regrinding and pH followed the open circuit cleaner tests. Collectors were only added to the rougher circuit. MIBC was dosed as required to both rougher and cleaner flotation. Split water circuit scheme was applied where filtrates from copper, and pyrite flotation were used in the respective flotation circuits.

A summary of the test results can be found in Table 13.18.

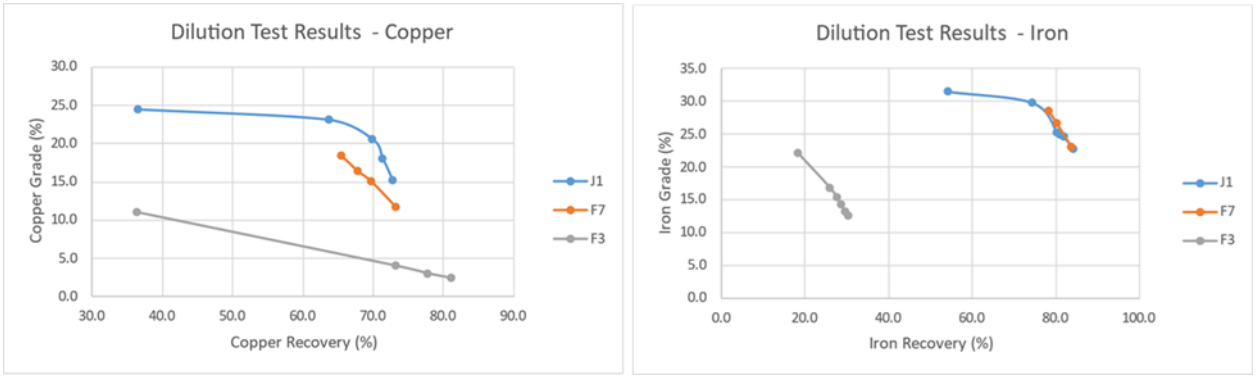
Table 13.18 – Jameson Dilution Test Results Summary

Product	Mass Pull, %	Assay, %, g/t						Distribution, %					
		Cu	Fe	As	S	Au	Ag	Cu	Fe	As	S	Au	Ag
Cu 3rd Clnr Conc	1.0	20.5	11.0	7.76	27.6	33.1	33.1	69.9	1.9	69.0	4.3	7.1	28.5
Cu 1st Clnr Conc	1.2	18.1	11.1	6.82	25.6	30.3	30.3	71.3	2.3	70.4	4.7	7.6	29.2
Cu Rougher Conc	1.4	15.2	11.2	5.74	23.4	27.5	27.5	72.7	2.8	71.8	5.2	8.3	30.2
Py 3rd Clnr Conc	18.2	0.23	25.0	0.09	28.6	19.0	19.0	14.5	80.8	14.4	82.4	74.8	48.2
Py 2nd Clnr Conc	18.4	0.23	24.9	0.09	28.4	19.0	19.0	14.8	81.2	14.7	82.8	75.4	48.2
Py 1st Clnr Conc	18.9	0.24	24.5	0.09	28.0	18.8	18.8	15.3	82.1	15.3	83.6	76.5	48.9
Py Rougher Conc	21.0	0.24	22.7	0.09	25.7	17.6	17.6	16.9	84.4	17.1	85.4	79.6	51.1
Ro Tail	77.6	0.04	0.93	0.02	0.77	0.7	0.7	10.3	12.8	11.1	9.5	12.0	18.7

Source: SGS, 2022

As seen from Table 13.18, copper concentrate grade is generally lower than produced during the open circuit cleaner tests. Copper recovery is generally in the same range with the respective open circuit bench scale open circuit cleaner tests. Pyrite circuit during the dilution test produced lower Au and S grades for the concentrate, and higher recoveries compared to the bench scale open circuit cleaner tests.

Figure 13.6 – Jameson Dilution Test Grade-Recovery Curves



Notes: J1 – Jameson dilution tests - F3 and F7 – conventional bench scale tests, rougher, and cleaner respectively
Source: SGS, 2022

Glencore Technology reviewed the test results and observed that entrainment losses have a significant impact on the copper grade vs recovery curve. While test J1 is a rougher Jameson cell simulation, it is compared to F7 which is a cleaner test. While this is not a “like for like or rougher vs rougher” comparison, it shows entrainment is a noticeable concern with the Loma Larga ore. Since the iron grade recovery curves are comparable, the gangue entrainment recovery is unlikely to be pyrite and must be another mineral. Theoretically, if a Jameson Cell was installed as a rougher scalper in this duty, it would be possible to recover 60% of the copper at 24% copper grade which would significantly debottleneck the rest of the circuit.

13.3.10 HYDROFLOAT BENCH SCALE TEST

Combined sample of pyrite rougher tailings from LCT #1 and LCT #2 was used for the bench scale hydrofloat test in attempt to scavenge coarse particles and the associated pyrite and gold from the tailings.

The hydrofloat unit feed was conditioned with 60 g/t PAX at pH of 7.5, and then flotation was performed. No frother addition has been done as per the test procedure.

The test was unsuccessful as the tailings material was too fine to settle down and build a proper bed within the hydrofloat unit. A limited quantity of the feed material did not allow for any further optimisation.

13.3.11 SOLID-LIQUID SEPARATION

The tailings dewatering testwork program included of flocculant scoping and static thickening, dynamic thickening, underflow rheology, and underflow pressure filtration. All tests were performed with the LCT pyrite rougher tailings combined sample.

Flocculant scoping tests consisted of evaluating the performance of various types of flocculants, where Magnafloc 10 shown the best results. The static settling tests were done to examine the settling rate, and the underflow clarity as well as the total suspended solids following solid-liquid separation. Thee (3) test results were used as the starting conditions for the dynamic thickening tests and are presented in Table 13.19.

Table 13.19 – Static and Dynamic Thickening Test Results

Test	Flocculant Dosage	Underflow Density	Thickener Unit Area	Total Suspended Solids
	(g/t)	(% w/w)	(m²/(t/d))	(mg/L)
Static Thickening	10	66.0	0.06	10
Dynamic Thickening	20	61.0	0.04	52
Dynamic Thickening	20	67.6	0.08	357

Source: SGS, 2022

The best results were achieved using a dosage of 10 g/t Magnafloc 10. This test produced an underflow with 66.0% solids by weight and an overflow clarity lower than 10 mg/L total suspended solids (TSS). The solids loadings and initial settling rate were 0.06 m²/(t/d) and 868 m³/m²/d, respectively.

Two-stage dynamic thickening was performed using 10% by weight pyrite rougher tailings samples dosed with 20 g/t Magnafloc 10. The analysed unit areas for the samples ranged from 0.04 m²/(t/d) to 0.08 m²/(t/d). At the lower bound, the underflow density was 61.0% w/w by solids while at the higher bound, it was 67.6% w/w by solids. The overflow TSS ranged from 52 mg/L to 357 mg/L. These results are shown in Table 13.19.

Extending the thickening duration proved useful to increase the underflow density from 67.6% w/w by solids to 75.5% w/w by solids. At such density, the yield stress was found to be 123 Pa. At 74.2% w/w underflow density, the yield stress value is 27 Pa (yield stress value at approximately 25 Pa is required for conventional pumping). The results summary is included in Table 13.20.

Four (4) concentric cylinder rheology tests were performed on samples ranging from 70.3% to 74.2% solids by weight and two (2) vane rheology tests were conducted on samples ranging from 75.2% to 76.3% w/w solids by weight. These last two (2) samples were not adequate for the cylinder rheology tests due to their high solids content. The test results are shown in Table 13.20.

Table 13.20 – Rheology Test Results

Test	Feed Density	Yield Stress
	(% w/w)	(Pa)
Extended dynamic thickening	75.5	123
Rheology – concentric cylinder rheology	70.3	5.4
Rheology – concentric cylinder rheology	71.2	8.1
Rheology – concentric cylinder rheology	72.1	14
Rheology – concentric cylinder rheology	74.2	27
Rheology – Vane rheology	75.2	40
Rheology – Vane rheology	76.3	104

Source: SGS, 2022

Based on these tests, the critical solids density (CSD) and yield stress under unsheared flow conditions for the tailings were found to be approximately 75% w/w solids and 35 Pa, respectively. All shear data was invalid because of the fast-settling nature of the samples.

Pressure filtration tests were performed on tailings sample having been thickened to 75% w/w solids using both a filtration pressure of 4.1 and 6.9 bar. The results from the testwork, shown in Table 13.21, indicate that residual cake moisture content ranging between 7.6-10.5% by weight with cake thickness of 30-45 mm can be achieved at filtration throughputs of 1,009 to 2,185 kg/m²h according to the filtration time. Considering the estimated full cycle time, the filtration throughput is 232 to 360 kg/m²h.

Table 13.21 – Pressure Filtration Test Results

Operating Conditions		Filter Outputs				
Pressure Level (bar)	**Filtration Time** (s)	**Cake Thickness** (mm)	**Filtration Time Cycle Throughput, dry solid** (kg/m²•h)	**Estimated Full Cycle Throughput, dry solid** (kg/m²•h)	**Cake Moisture** (% w/w)	**Filtrate TSS** (mg/L)
4.1	187	45	1,414	336	9.3	60
	186	40	1,269	301	10.5	66
	204	35	1,009	256	9.9	74
	160	30	1,103	232	10.2	109
6.9	141	45	1,890	360	8.4	74
	109	41	2,185	336	8.5	62
	117	35	1,757	287	8.1	30
	116	30	1,513	246	7.6	44

Source: SGS, 2022

13.3.12 QUALITY OF THE CONCENTRATES

Detailed assays for copper and pyrite concentrates produced during the LCTs were completed by SGS Lakefield assay laboratory upon DPM request. Summary of the results can be found in Table 13.22.

Table 13.22 – Concentrate Detailed Assays

Assay	Unit	LCT Cu Conc. (Cycles A/B/C/E/F combined)	LCT Cu Conc. (Cycle D)	LCT Py Conc. (Cycles A/B/C/E/F combined)
As	g/t	12,300	6,190	427
Ba	g/t	62	55	155
Be	g/t	< 0.05	< 0.05	< 0.05
Bi	g/t	48	46	27
Cd	g/t	123	< 80	< 80
Co	g/t	77	103	84
Cu	g/t	179,000	87,300	2,860
Li	g/t	< 20	< 20	< 20
Mo	g/t	48	35	9
Ni	g/t	71	52	40
Pb	g/t	1,630	998	387
Sb	g/t	2,450	1,180	43
Se	g/t	< 30	< 30	< 30
Sn	g/t	88	46	< 20
Sr	g/t	37	22	20
Tl	g/t	< 40	< 40	< 40
Y	g/t	< 0.5	< 0.5	< 0.5
Zn	g/t	14,800	6,860	435
Ga	g/t	-	-	-
Te	g/t	-	-	-
SiO_2	%	23.7	15.2	41.0
Al_2O_3	%	0.24	0.14	0.20
Fe_2O_3	%	25.1	42.1	36.3
MgO	%	0.016	0.008	< 0.005
CaO	%	0.3	0.2	< 0.1
K_2O	%	0.05	< 0.04	< 0.04

Assay	Unit	LCT Cu Conc. (Cycles A/B/C/E/F combined)	LCT Cu Conc. (Cycle D)	LCT Py Conc. (Cycles A/B/C/E/F combined)
TiO_2	%	0.58	0.44	0.65
MnO	%	0.012	0.008	0.005
Cr_2O_3	%	0.07	0.04	0.03
V_2O_5	%	< 0.008	< 0.008	< 0.008
Ge	g/t	51	26	2
Hg	g/t	54.9	31.8	7.6
Cl (HNO_3 soluble)	g/t	28	37	17
F	%	< 0.005	< 0.005	< 0.005
Pt	g/t	< 0.02	< 0.02	< 0.02
Pd	g/t	< 0.02	< 0.02	< 0.02
Au	g/t	31.3	28.3	20.2
Ag	g/t	494	277	48.4
S	%	32.0	40.8	29.7

Source: SGS, 2022

13.4 2023 Jameson Dilution Tests at Base Metallurgical Laboratories

Considering the attractive Capex for the Jameson cell, in July-August 2023 a 122 kg LOM composite sample was tested at BML. The sample was compiled from the historical drillcore stored on site. Samples were selected to be representative of Mineral Reserve average grades, equal representation of key lithologies and spatial representation across the deposit. Care was taken to ensure only fresh (un-oxidised) core was selected. Sample head assays can be found in Table 13.23.

Table 13.23 – Head Assays

Cu	S	Ag	As	Sb	Au	Fe
%	%	g/t	ppm	ppm	g/t	%
0.34	7.96	29	1319	75	5.02	6.44

Source: SGS, 2023

The mineral content of the sample was determined by Bulk Mineral Analysis (BMA) via QEMSCAN. The composite contained 14.4% by weight sulphides. The remaining mass was comprised mainly of quartz, clays, and oxides. All copper was found in the form of enargite. No other appreciable forms of copper sulphides were detected. Enargite made up about 3% of the total sulphur content.

The Jameson dilution test protocols were applied considering the available quantity of ore.

Rougher and cleaner conditions were investigated and compared to conventional flotation results.

The following key test parameters were used for all tests:

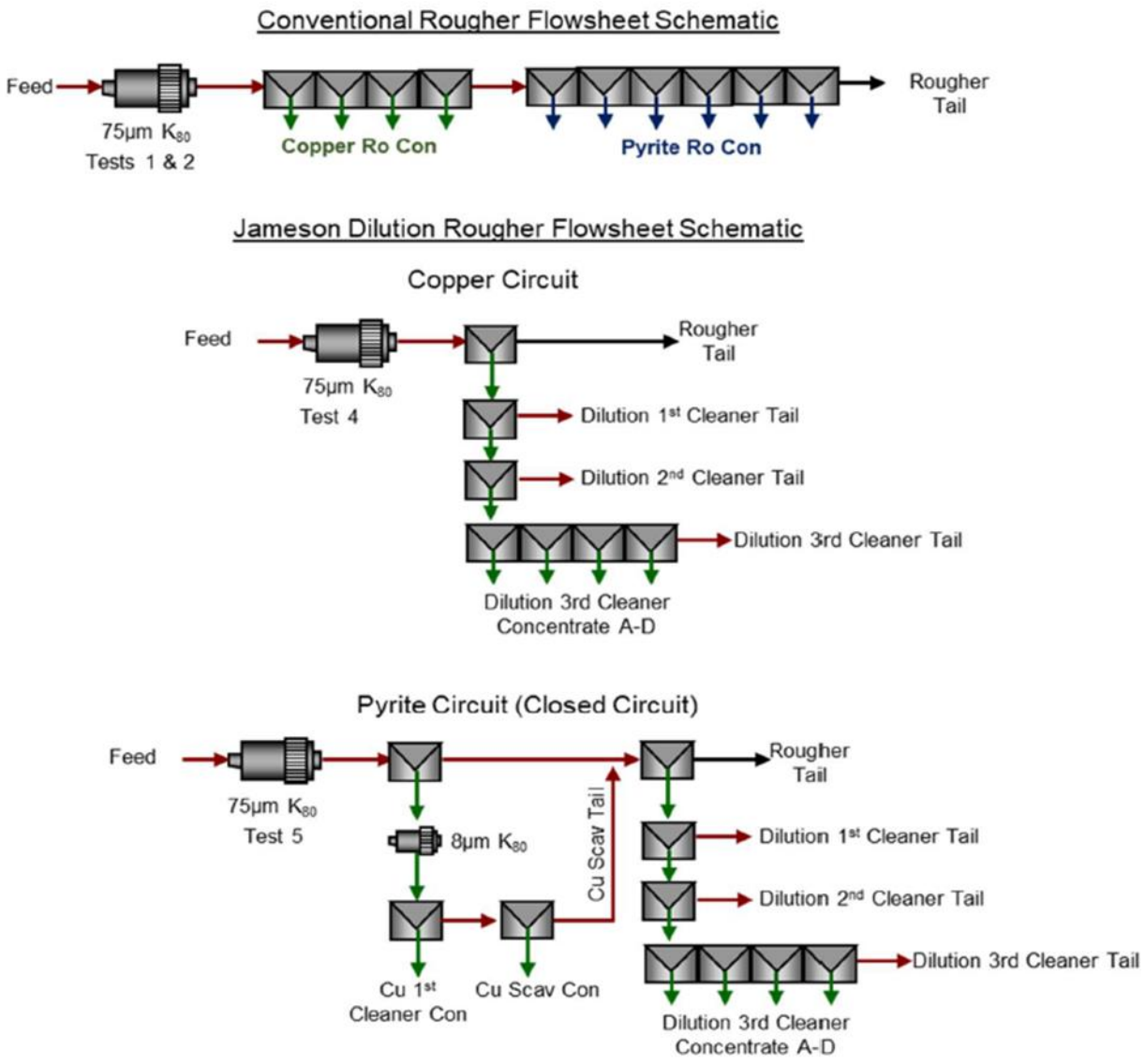
- Primary Grind Size: 75 µm K_{80}.
- Regrind Targets: 30 µm K_{80} for Copper and Pyrite.
- Pulp pH: 11.5 for Cu Ro (lime) and 7.0 & Py Ro (H_2SO_4).
- Collector: 3418A and Potassium Amyl Xanthate (PAX) as collectors.
- Frother: Methyl Isobutyl Carbinol (MIBC).

A copper regrind target of 30 µm K_{80} was requested; however, the copper rougher concentrate generated from a 75 µm K_{80} primary grind was sized at approximately 14 µm K_{80}. It is likely that some preferential grinding was occurring during the primary grind, resulting in finely ground copper particles. The copper regrind, however, was kept in the circuit to ensure polished surfaces for collector absorption.

13.4.1 ROUGHER TEST

A total of four (4) rougher flotation tests were completed on the sample; two (2) conventional rougher tests and two (2) following the Jameson Dilution protocol. Rougher flotation schematics can be found in Figure 13.7. Rougher tests summary is in Table 13.24 and Figure 13.8.

Figure 13.7 – Rougher Tests Schematics



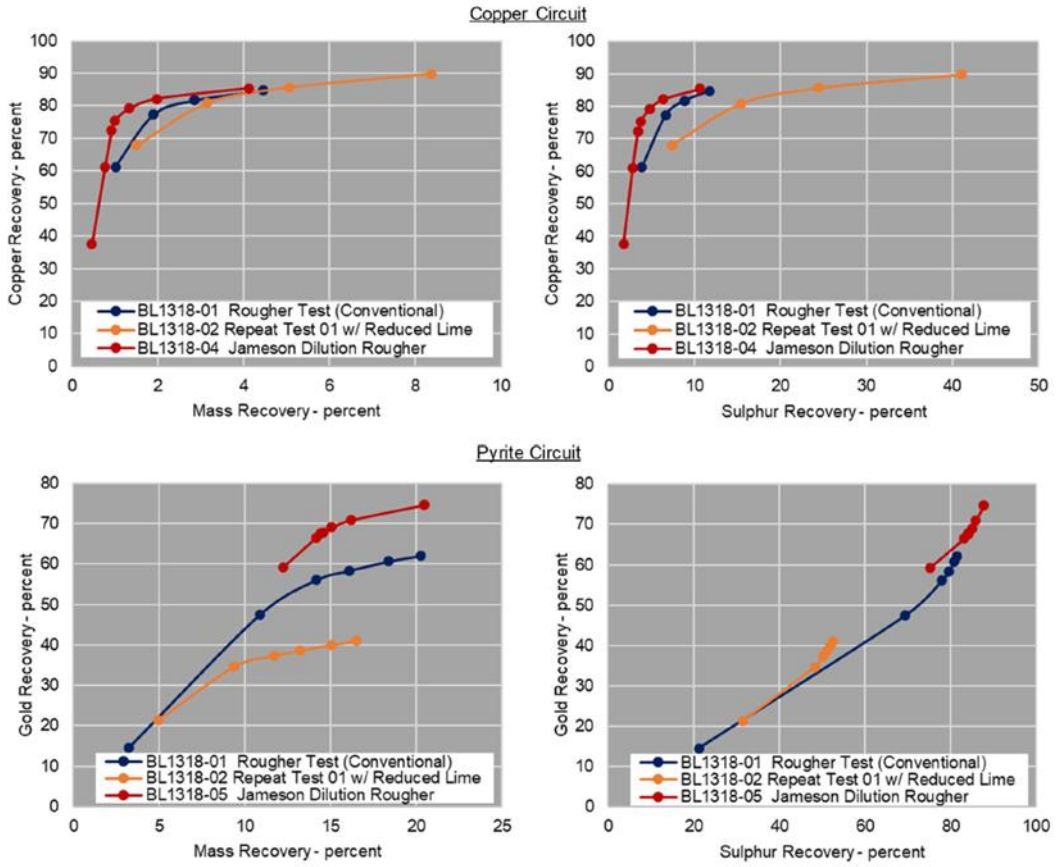
Source: BML, 2023

Table 13.24 – Rougher Results Summary

Test	Product	Weight	Assay – percent or g/t								Distribution – percent						
		%	grams	Cu	Fe	S	Ag	As	Sb	Au	Cu	Fe	S	Ag	As	Sb	Au
Conventional Flotation																	
1	Cu rougher conc.	4.4	89.2	6.4	13.2	22.0	339.3	29,728.8	1372.9	35.2	84.8	9.2	11.8	53.5	88.0	78.1	31.2
	Pyrite rougher conc.	20.2	406.0	0.2	26.3	33.4	55.5	651.5	56.5	15.4	12.1	84.2	81.5	39.8	8.8	14.6	62.1
2	Cu rougher conc.	8.4	167.6	3.6	29.9	40.3	225.9	15,930.5	745.6	31.3	89.8	39.7	41.1	68.4	90.5	83.0	52.7
	Pyrite rougher conc.	16.5	330.9	0.1	20.7	26.0	42.5	564.4	46.9	12.3	7.2	54.2	52.4	25.4	6.3	10.3	41.1
Jameson Dilution																	
4	Cu rougher conc.	4.1	82.5	7.1	12.0	21.4	447.3	29,439.3	1677.0	31.7	85.4	8.2	10.6	55.3	87.4	83.9	27.7
5	Pyrite rougher conc.	20.5	408.4	0.3	23.7	35.2	89.0	931.1	70.6	17.0	18.5	88.7	87.8	51.7	15.2	17.6	74.7

Source: BML, 2023

Figure 13.8 – Rougher Results



Source: BML, 2023

As seen from Table 13.24, the baseline rougher Test #1 resulted in copper rougher concentrate metal recoveries of 84.8% copper and 31.2% gold – at a mass recovery of 4.4%. Additionally, 62.1% of the gold was captured within the pyrite rougher concentrate.

An additional test was completed at a fixed lime dosage. Results show the decreased lime addition had a negative impact on copper-sulphur selectivity, resulting in a lower quality copper rougher concentrate.

Tests #4 and #5 were completed using the Jameson dilution protocol. Both circuits displayed performance benefits from the Jameson dilution protocol. For the copper circuit, the first stage of dilution cleaning was able to reject approximately 50% of the rougher concentrate mass, while maintaining a comparable copper rougher concentrate recovery of 82.2%.

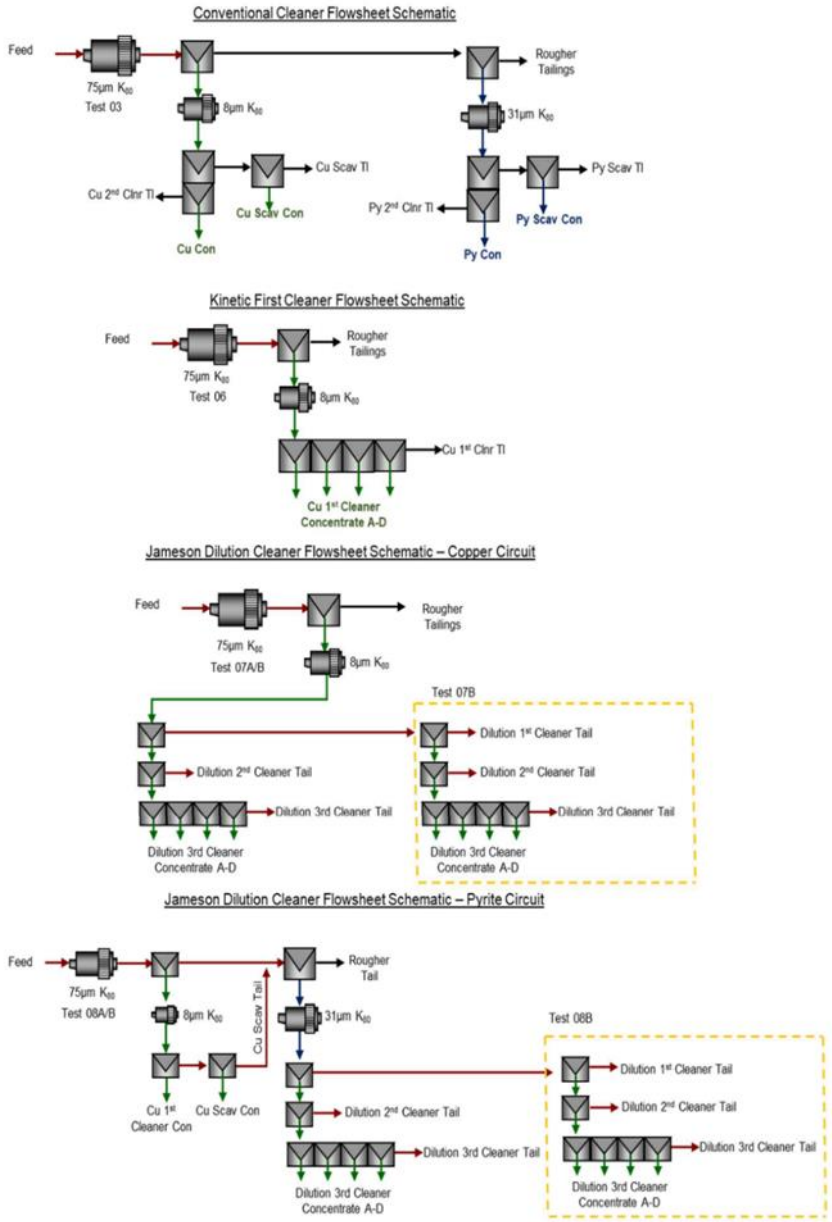
The pyrite rougher circuit displayed a similar response to the Jameson dilution protocol – showing comparable metal recovery at a reduced mass recovery. However, the Jameson dilution rougher

was conducted within a closed-circuit testing setup, and conventional testing was not. Comparative tests required for conclusive results.

13.4.2 CLEANER TESTS

Four (4) Cleaner flotation tests were conducted. The tests schematics can be found in Figure 13.9. Summary of results is provided in Table 13.25 and Figure 13.10.

Figure 13.9 – Cleaner Tests Schematics



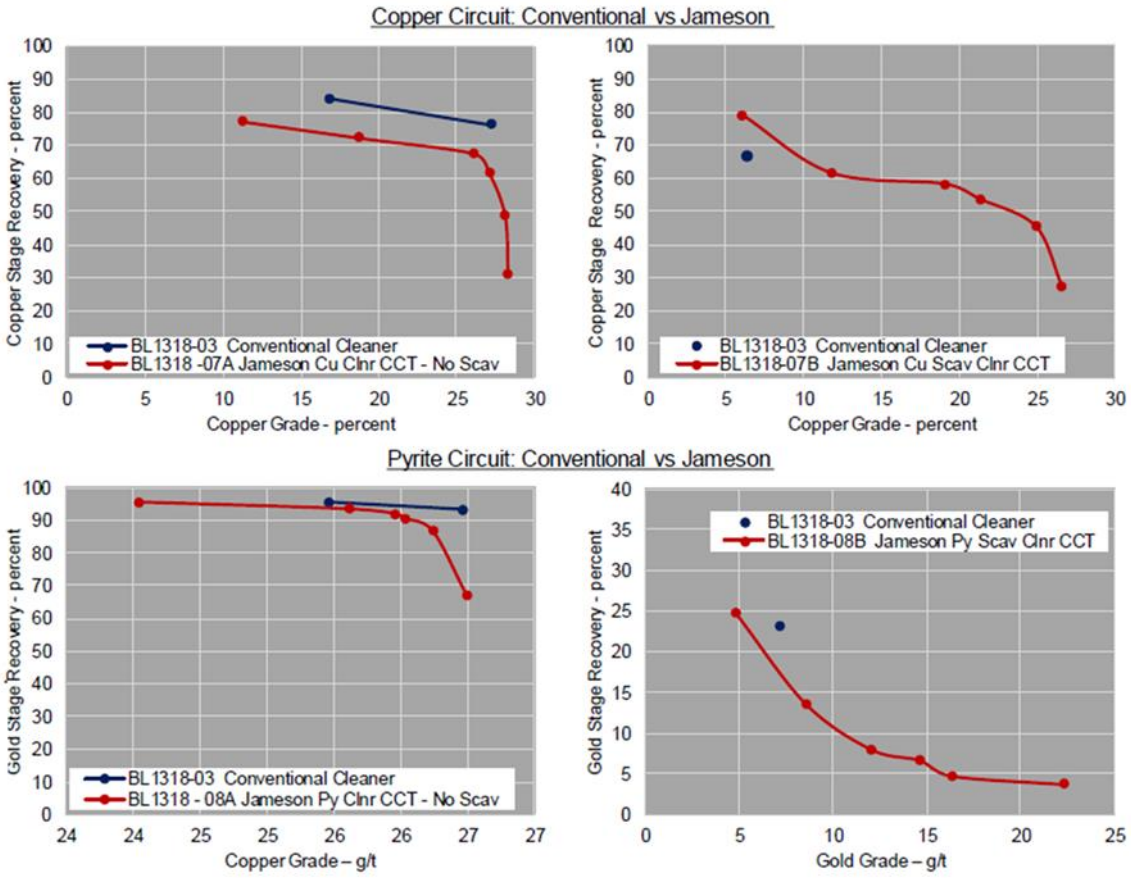
Source: BML, 2023

Table 13.25 – Cleaner Results Summary

Test		3				7A	7B	8A	8B
		Conventional Flotation				Jameson Dilution			
Product		Cu Clnr Conc.	Cu Clnr Scav Conc.	Py Clnr Conc.	Py Clnr Scav Conc.	Cu Clnr Conc.	Cu Clnr Scav Conc.	Py Clnr Conc.	Py Clnr Scav Conc.
Weight	%	0.8	0.5	11.6	0.5	0.7	0.2	9.7	0.1
Assay - % or g/t	Cu	27.2	6.4	0.33	0.14	26.1	19.1	0.35	0.28
	Fe	8.30	17.1	42.9	6.80	6.18	11.2	41.9	8.75
	S	30.5	28.2	54.8	7.22	26.1	23.1	50.8	7.91
	Ag	1,400	388	89	32	1205	265	99	36
	As	119,780	26,540	1,071	559	116,308	399	1,098	551
	Sb	7,100	1,401	89	37	6,148	411	155	35
	Au	63.9	63.9	26.5	7.16	74.2	88.3	26.0	12.1
Distribution – %	Cu	64.8	9.0	11.4	0.2	56.6	11.1	10.3	0.1
	Fe	1.1	1.3	80.1	0.5	0.7	0.3	68.0	0.1
	S	3.1	1.7	79.5	0.4	2.5	0.7	65.3	0.1
	Ag	37.5	6.1	34.5	0.5	35.7	8.2	31.4	0.1
	As	66.9	8.7	8.7	0.2	59.6	12.4	9.0	0.0
	Sb	65.3	7.6	11.8	0.2	56.8	12.8	17.7	0.0
	Au	10.1	5.9	60.5	0.7	10.0	3.2	50.5	0.2

Source: BML, 2023

Figure 13.10 – Cleaner Results



Source: BML,2023

As shown from Table 13.25 and Figure 13.10, for copper flotation, if high-grade concentrates are the requirements, the conventional flotation produce slightly higher recovery and concentrate grades; however, it is understood that with the lower copper concentrate grade target this difference will becomes less significant and is compensated with the volume of payable copper concentrate.

The results were more aligned in the pyrite circuit, yet the Jameson performance is still lower than the conventional circuit based on the results completed to date. To meet the test goals, the Jameson dilution tests used more feed material. Consequently, there is a possibility that the cleaner stage did not receive enough time to accommodate this higher mass, which could account for the result disparities, and therefore some residence time optimisation testing should be planned for future work.

13.5 Deleterious Elements

Both copper and pyrite concentrates will be subject to penalty conditions, should significant grades of zinc, lead, mercury, antimony, bismuth and arsenic be present in the concentrate.

In-situ assays of these elements within the block model are highly variable and recoveries are difficult to predict based on the limited testwork focusing on these elements.

Tables 13.26 and 13.27 present the recoveries of the main deleterious elements into the copper and pyrite concentrates, respectively. The results are based on the two (2) LCT's completed in 2017 metallurgical testwork program.

Table 13.26 – Deleterious Elements Recovery to Copper Concentrate

Element	2018 LCT #3 Assay ppm/Recoveries %			2018 LCT #4 Assays ppm/Recoveries %			Recommended Recovery for NSR
	Head	Cons	Rec	Head	Cons	Rec	
Zinc	151	3,100	28.4	177	8,815	38.5	35.0
Lead	103	760	9.97	198	3,380	0.29	5.0
Antimony	113	7,110	86.0	101	7,516	32.7	50.0
Mercury	3.49	85.5	33.6	-	79	-	30.0

Source: SGS, 2018

Table 13.27 – Deleterious Elements Recovery to Pyrite Concentrate

Element	2018 LCT #3 Assay ppm/Recoveries %			2018 LCT #4 Assays ppm/Recoveries %			Recommended Recovery for NSR
	Head	Cons	Rec	Head	Cons	Rec	
Zinc	151	582	65.1	177	509	40.6	55.0
Lead	103	484	80.4	198	727	50.5	65.0
Antimony	113	70	9.72	101	<60	34.9	25.0
Mercury	3.49	12.1	57.0	-	15.9	-	50.0

Source: SGS, 2018

The final assay value of bismuth in both the copper and pyrite concentrates indicated levels less than 300 ppm and 100 ppm, respectively.

13.6 Metallurgical Variability

The metallurgical testwork completed to date supports the FS and is based on samples which adequately represent the variability of the Loma Larga deposit and associated mine plan.

Additional variability work is required post-FS to optimise the process parameters and reduce Project risks. This variability work should include comminution, Jameson cell flotation tests including LCTs on variability samples, solid-liquid separation tests.

Mineralogical analysis of the various composites and variability samples has shown that the Loma Larga deposit can be considered reasonably homogenous.

No variability work was performed for the 2022 nor the 2023 programs due to a limited quantity of sample material available for testing and absence of new drill core availability.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

SLR estimated Mineral Resources for the Loma Larga Project using all drillhole data available as of October 31, 2018. This Mineral Resource Estimate was previously updated October 31, 2018, and reported in the 2019 Technical Report (DRA, 2019) and readdressed to DPM in 2021 (DRA, 2021). No additional Mineral Resource drilling has been completed on the Project since the previous estimate. Mineral Resources were reported inclusive of Mineral Reserves in the previous Technical Report (DRA, 2021) and were subsequently disclosed exclusive of Mineral Reserves in DPM's 2021 Annual Information Form.

The current Mineral Resource Estimate incorporates updated metal prices and mining costs, is based on an underground mining scenario, and is reported exclusive of Mineral Reserves. In order to ensure that the resources have sufficient spatial continuity, demonstrating Reasonable Prospects of Eventual Economic Extraction (RPEEE), the Mineral Resource Estimate was reported within underground resource reporting shapes generated in Deswik Stope Optimizer (DSO) software, satisfying the minimum mining size, continuity criteria, and using an NSR cut-off value of US$65/t. NSR cut-off values for the Mineral Resources are based on a gold price of US$1,700/oz, silver price of US$20.00/oz, and copper price of US$3.75/lb. Mineral Resources exclusive of Mineral Reserves, effective as of September 29, 2023, are summarised in Table 14.1. The Mineral Resources conform to CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) as incorporated by reference in NI 43-101.

SLR was provided with a drillhole database consisting of 365 holes, totalling 81,183 m, with 249 of the holes (58,990 m) located within the mineralisation domains. INV completed two (2) drilling programs on the Project since the 2016 Technical Report and the current Mineral Resource update incorporates the new drilling results.

Three-dimensional (3D) grade shell wireframes were constructed at 2.0 g/t Au (High-Grade Zone) and 0.8 g/t Au (Low-Grade Zone). The QP used cross sections, long sections, and plan views to interpret and validate the wireframes.

Table 14.1 – Mineral Resource Estimate Summary Effective as of September 29, 2023

Resource Classification	Tonnage (Mt)	Au (g/t)	Contained Au (Moz)	Ag (g/t)	Contained Ag (Moz)	Cu (%)	Contained Cu (Mlb)
Measured	0.3	4.80	0.05	20.8	0.2	0.23	1.7
Indicated	5.7	2.79	0.51	21.9	4.0	0.16	19.8
Measured + Indicated	6.0	2.90	0.56	21.8	4.2	0.16	21.5
Inferred	2.2	2.54	0.18	37.0	2.7	0.19	9.3

Notes:

- Mineral Resources have been estimated by Katharine Masun, M.Sc., MSA, P. Geo., of SLR, an independent Qualified Person as defined by NI 43-101.
- CIM (2014) definitions were followed for Mineral Resources.
- The Mineral Resources have been reported within underground reporting shapes generated with Deswik Stope Optimizer (DSO) using a net smelter return (NSR) cut-off value of US$65/t.
- Mineral Resources are estimated using a long-term gold price of US$1,700 per ounce, silver price of US$20 per ounce, and copper price of US$3.75 per pound.
- Mineral Resources are exclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Average bulk density is 2.7 t/m³.
- Numbers may not sum precisely due to rounding.

The Loma Larga High-Grade Zone comprises two (2) mineralised zones: High-Grade Main Zone and High-Grade Upper Zone. The Low-Grade Zone comprises two (2) domains: Low-Grade Main Zone wireframe domain that encompasses the High-Grade Main Zone, and Low-Grade Lower Zone, which lies below the Low-Grade Main Zone.

Variography was performed on the 2.0 m Au, Ag, Cu, S, and density composites from the High-Grade Main Zone and Low-Grade Main Zone. Block grade interpolation was carried out using Ordinary Kriging (OK) and the gold grade shell wireframe models were used to constrain the grade interpolations. A soft boundary was used between the Low- and High-Grade Main Zones for density block interpolation.

The polymetallic sulphide mineralisation at the Loma Larga deposit contains significant values of Au, Ag, and Cu. Therefore, original assays were converted into NSR values (US$/t). The NSR values account for parameters such as metal price, metallurgical recoveries, smelter terms and refining charges, and transportation costs. For the purposes of developing an NSR cut-off value for an underground operation, a total operating cost of US$65/t milled was assumed, which includes mining, processing, and general and administrative (G&A) expenses.

Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate. The QP understands that DPM is considering all its options to rectify the revocation of the environmental

license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project, including the Mineral Resource Estimate. See “Post-Effective Date Events” at the beginning of this Report.

14.2 Mineral Resource Database

SLR received new drillhole data in CSV format. Data were amalgamated and parsed as required and imported into GEMS for modelling. Table 14.2 summarises records directly related to the resource estimate.

Table 14.2 – Mineral Resource Database

Attribute	Number
Holes	249 (199 with sulphur assays)
Surveys	806
Assays (Payable Metals)	23,857 (10,816 within mineralisation domains)
Assays (Sulphur)	13,688 (6,890 within mineralisation domains)
Assay Composites (>0.5 m in length)	6,236
Sulphur Composites (>0.5 m in length)	3,993
Full Zone Width Composites	692
Density Measurements	10,153 (7,767 within mineralisation domains)
Density Composites (>0.5 m in length)	4,988

Section 12 describes the verification steps undertaken by the QP. In summary, all minor discrepancies identified were resolved and the QP is of the opinion that the GEMS drillhole database is valid and suitable to estimate Mineral Resources for the Project.

14.3 Geological Interpretation and 3D Solid

The wireframe model of the mineralised domains was used to constrain block model interpretation.

Prior to creating the mineralised wireframe domains for the Loma Larga Project, the QP validated the drillhole database by completing the following:

- Checking for location and elevation discrepancies by comparing collar coordinates with historical data.
- Checking for inconsistencies in drillhole dip directions
- Checking gaps, overlaps, and out-of-sequence intervals for both assay and lithology tables.

For the High-Grade Zone, the QP reinterpreted the mineralised wireframe domains, restricting the modelling cut-off grade to 2.0 g/t Au. The High-Grade Zone 3.0 g/t Au wireframe model from the previous resource estimate was used to assist in domaining the mineralisation. The modelling cut-off grade was reduced to align with the mine design and planning process.

For the Low-Grade Zone, the QP updated the wireframe domains at a modelling cut-off grade of 0.8 g/t Au with new drilling. The Low-Grade Lower Zone was combined into a single domain.

The Loma Larga High-Grade Zone comprises two (2) mineralised domains: Main and Upper (Figure 14.1, Figure 14.2, and Figure 14.3). The Loma Larga Low-Grade Zone also comprises two (2) mineralised domains: Main and Lower (Figure 14.1, Figure 14.2, and Figure 14.3). Mineralisation in the High-Grade Zone and Low-Grade Zone has been categorised into rock codes according to Table 14.3.

Table 14.3 – Rock Codes

Mineralisation	GEMS Solid Name	Rock Code	Volume (m³)
High-Grade Main Zone	130/final/2018	130	4,854,433
High-Grade Upper Zone	131/final/2018	131	64,605
Low-Grade Main Zone[1]	108/final/2018	108	13,420,552
Low-Grade Lower Zone	208/final/2018	208	922,193

1 Inclusive of the High-Grade Main Zone

A description of each modelled zone follows.

14.3.1 HIGH-GRADE ZONE

- The High-Grade Main Zone is a north-northwest trending, flattened, cigar-shaped zone. It is approximately 1,150 m in length and 120 m to 290 m in width, averages 40 m to 50 m in thickness, and lies about 120 m to 125 m below the surface. The Main Zone is intersected by 199 drillholes, i.e., 16 more than in the 2016 model (RPA, 2016).
- The High-Grade Upper Zone is a north-northeast trending, tabular body approximately 125 m in length by 55 m in width, averaging 10 m in thickness. It lies 55 m to 60 m below the surface and approximately 60 m above the Main Zone. The zone plunges steeply to the south (approximately 60º) and the width of mineralisation increases from approximately 5 m to 20 m. The Upper Zone is intersected by six (6) drillholes.

14.3.2 Low-Grade Zone

- The Low-Grade Main Zone is a north-trending elongated zone that encompasses the High-Grade Main Zone and is nearly 1,600 m in length and ranges from 100 m to 400 m in width. It averages approximately 50 m to 60 m thick and lies approximately 110 m to 120 m below the surface. It is intersected by 247 drillholes and dips at approximately 2° to the south.
- The Low-Grade Lower Zone is a northeast trending, generally flat-lying, tabular body approximately 200 m in length by 120 m to 340 m in width, averaging 10 m to 20 m in thickness. It lies 10 m to 20 m below the Low-Grade Main Zone and is intersected by 37 drillholes.

14.3.3 High-Grade Zone Interpretation

Mineralisation for the High-Grade Zone Main Zone was interpreted by the QP using GEOVIA GEMS 6.8.2 (GEMS). Sectional interpretations were performed on screen via strings snapped to drillhole intersections on northwest-southeast vertical cross sections spaced 12.5 m apart and plans spaced at a minimum of 25 m. Above the High-Grade Main Zone, where spatial continuity was evident base on drillhole spacing, the high-grade assays were incorporated into the High-Grade Upper Zone.

Strings were joined together using tie lines to honour the drillhole assay data between sections and triangulated to build 3D wireframe solids. At model extremities, strings were extrapolated along dip approximately 10-15 m or less, unless lithological interpretation suggested extending mineralisation further. The zones of interpreted mineralisation were generally contiguous; however, the wireframes were extended through drillholes with low-grade or narrow intersections to preserve continuity.

The re-interpretation at a 2.0 g/t Au cut-off grade resulted in a volume increase of approximately 28% (High-Grade Main Zone only), when compared to the High-Grade Zone, also modelled at a 3.0 g/t Au cut off, from the 2016 Mineral Resource Estimate (4.85 Mm^3 vs. 3.78 Mm^3).

14.3.4 Low-Grade Zone Interpretation

Mineralisation for the Low-Grade Zone was interpreted using northwest-southeast sections spaced at 25 m and plans spaced at a minimum of 25 m. Based on the visual continuity of assay results and using the 2016 Low-Grade Zone wireframe model as a guide, sections and plans were used to make an on-screen interpretation of grade shells at a cut-off of 0.8 g/t Au. Sectional interpretations were performed on screen via strings snapped to drillhole intersections on the northwest-southeast oriented cross. Polylines were joined together in 3D using tie lines. At the model extremities and along section lines, polylines were extrapolated less than 25 m beyond the last drillhole section. Solids were further reviewed in longitudinal sections in order to confirm continuity. The QP made minor modifications to the interpretation along the margins of the Low-Grade Main Zone and combined the two (2) domains that comprised the Low-Grade Lower Zone in the previous resource estimate into a single wireframe.

The incorporation of new drilling and the minor modifications to interpretation of the Low-Grade Zone wireframes has resulted in the following changes when compared to the 2016 Mineral Resource Estimate:

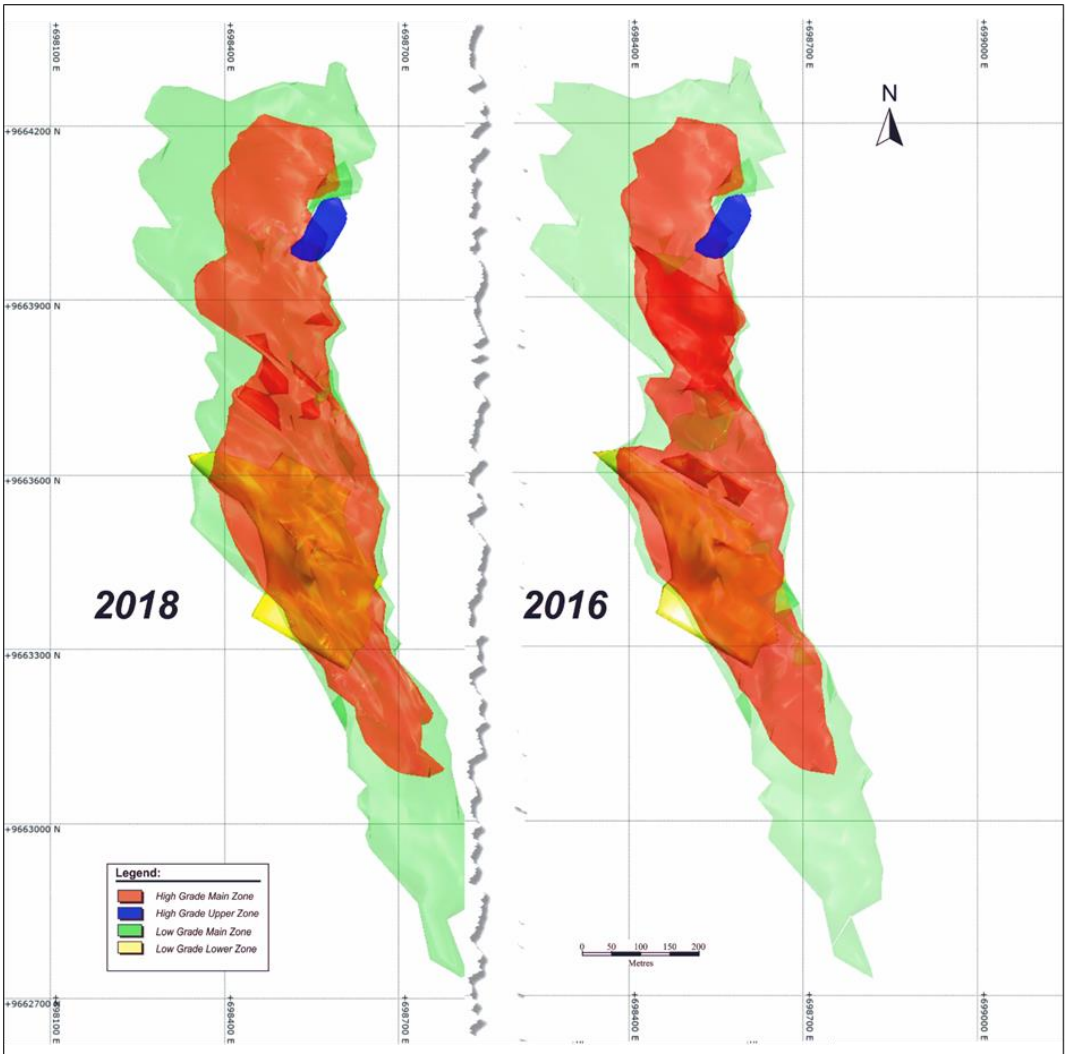
- A volume increase of less than 1.2% in the Low-Grade Main Zone inclusive of the nested High-Grade Main Zone (13.4 Mm^3 vs. 13.3 Mm^3).
- An increase of 4.8% in the Low-Grade Lower Zone (0.92 Mm^3 vs. 0.88 Mm^3), when compared to the Low-Grade Zone, also modelled at a 0.8 g/t Au cut-off grade.
- Exclusive of the High-Grade Main Zone, the volume of the Low-Grade Main Zone has decreased 11% (8.6 Mm^3 vs. 9.5 Mm^3).

Figure 14.1 illustrates the revisions to the Low-Grade Zone wireframes in plan view.

All Mineral Resources estimated on the Loma Larga Project are located within the mineralised zone wireframes.

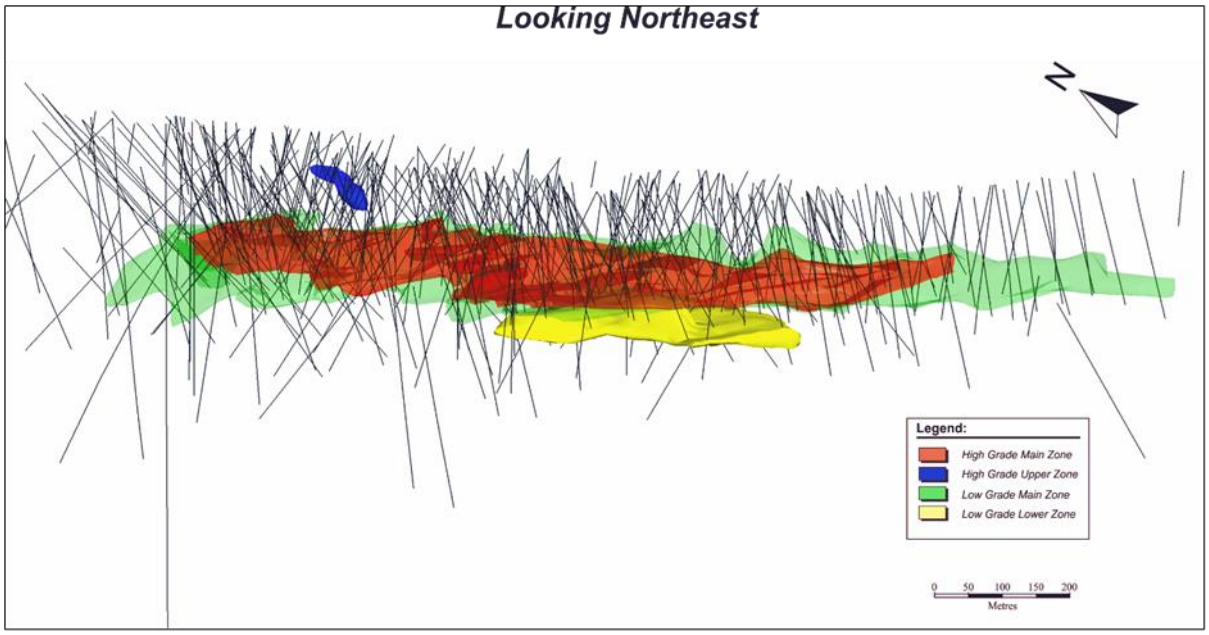
The QP note that there are additional drillhole intercepts outside the mineralised wireframe domains that are greater than 2.0 g/t Au and represent exploration potential. In the QP's opinion, the isolated location and narrow thickness of these intercepts together with the wide drillhole spacing, or substantial intervening material that is below the cut-off grade, precludes the inclusion of the intercepts as Mineral Resources at this time.

Figure 14.1 – 2018 versus 2016 Wireframe Domains in Plan View (3,600 m Level)



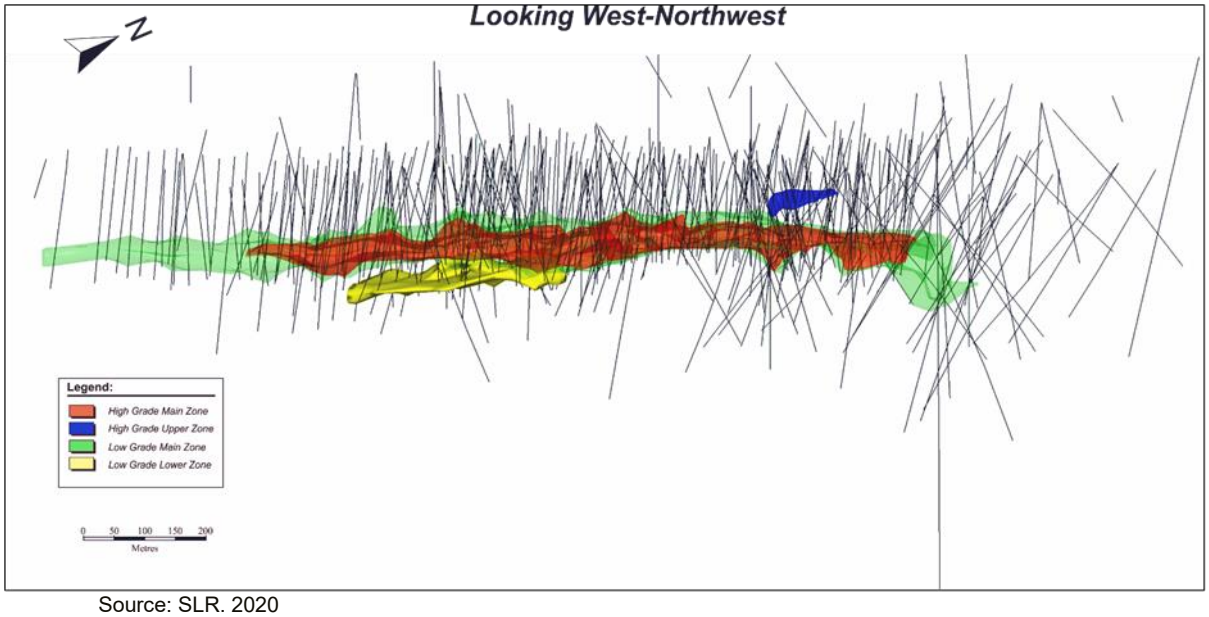
Source: SLR. 2020

Figure 14.2 – 3D View of 2018 High-Grade (2.0 g/t Au) and Low-Grade (0.8 g/t Au) Wireframe Domains, Looking Northeast



Source: SLR, 2020

Figure 14.3 – 3D View of 2018 High-Grade (2.0 g/t Au) and Low-Grade (0.8 g/t Au) Wireframe Domains, Looking West



Source: SLR. 2020

14.4 Statistical Analysis

Assay values located inside the wireframes, or resource assays, were tagged with mineralised zone domain identifiers (rock codes) and exported for statistical analysis. The QP compiled and reviewed the basic statistics for Au, Ag, and Cu assays, which are summarised in Table 14.4.

Table 14.4 – Descriptive Statistics of Resource Assay Values

Description	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)
High-Grade Main Zone				
Number of Cases	5,255	5,255	5,255	5,255
Minimum	0.07	0.00	0.50	1
Maximum	4.43	768.67	2,295.6	204,000
Median	1.00	3.46	16.60	1,169
Length Weighted Mean	-	6.40	33.21	3,762
Standard Deviation	0.33	18.24	65.78	10,242
Coefficient of Variation	0.31	2.85	1.98	2.72
High-Grade Upper Zone				
Number of Cases	67	67	67	67
Minimum	0.52	0.21	0.10	53
Maximum	2.00	105.50	250.1	67,000
Median	1.10	3.28	13.49	2,604
Length Weighted Mean	-	9.51	26.13	6,078
Standard Deviation	0.45	18.15	38.93	9,966
Coefficient of Variation	0.37	1.91	1.49	1.64
High-Grade Zone Total				
Number of Cases	5,322	5,322	5,322	5,322
Minimum	0.07	0.00	0.10	1
Maximum	4.43	768.67	2,295.6	204,000
Median	1.00	3.46	16.50	1,180
Length Weighted Mean	-	6.45	33.10	3,796
Standard Deviation	0.33	18.24	65.47	10,240
Coefficient of Variation	0.31	2.83	1.98	2.70

Description	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)
Low-Grade Main Zone[1]				
Number of Cases	4,839	4,839	4,839	4,839
Minimum	0.18	0.01	0.00	1
Maximum	5.00	101.80	1,743.0	216,000
Median	1.00	1.25	6.70	338
Length Weighted Mean	-	1.67	16.16	920
Standard Deviation	0.39	2.88	49.11	3,505
Coefficient of Variation	0.33	1.72	3.04	3.81
Low-Grade Lower Zone[1]				
Number of Cases	655	655	655	655
Minimum	0.24	0.02	0.20	8
Maximum	3.00	59.33	721.0	126,800
Median	1.00	1.62	10.20	990
Length Weighted Mean	-	2.19	15.89	1,969
Standard Deviation	0.32	2.88	37.44	6,977
Coefficient of Variation	0.30	1.31	2.36	3.54
Low-Grade Zone Total[1]				
Number of Cases	5,494	5,494	5,494	5,494
Minimum	0.18	0.01	0.00	1
Maximum	5.00	101.80	1,743.0	216,000
Median	1.00	1.28	7.20	383
Length Weighted Mean	-	1.73	16.13	1,036
Standard Deviation	0.38	2.88	47.96	4,052
Coefficient of Variation	0.33	1.67	2.97	3.91

[1] Unsampled intervals not included in statistical analysis.

14.5 Capping High-Grade Values

In order to highlight areas of higher sulphur content, outlier values were not capped, and the QP did not perform capping analysis on the sulphur assays.

Table 14.5 summarises capping grade values used in the current Mineral Resource Estimate and compares the values to the capping grade values used in the 2016 Technical Report.

Table 14.5 – Capped Grade Values of Resource Assays

Grade Element	Capped Value	No. of Samples Capped	% Metal Removed	Capped Value	No. of Samples Capped	% Metal Removed
	2018 High-Grade Zone			2016 High-Grade Zone		
Au	75 g/t	39	9	65 g/t	48	12
Ag	380 g/t	24	4	350 g/t	31	5
Cu	45,000 ppm	67	10	40,000 ppm	81	13
	2018 Low-Grade Zone			2016 Low-Grade Zone		
Au	25 g/t	12	9	35 g/t	14	12
Ag	220 g/t	36	10	200 g/t	44	10
Cu	20,000 ppm	25	10	20,000 ppm	36	20

Capping outliers in the High-Grade Zone to 75 g/t Au, 380 g/t Ag, and 45,000 ppm Cu results in the reduction of the coefficients of variation (COV) for Au, Ag, and Cu to 1.49, 1.46, and 2.00 respectively, and slightly decreases the average grades of resource assays (Table 14.6).

For the Low-Grade Zone, Au was capped at 25 g/t, high-grade Ag outliers were capped at 220 g/t Ag, and high-grade Cu outliers were capped to 20,000 ppm Cu. The COVs of capped Au, Ag, and Cu are 1.12, 1.73, and 2.09, respectively (Table 14.6).

Table 14.6 – Descriptive Statistics of Resource Capped Assay Values

Description	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)
High-Grade Main Zone				
Number of Cases	5,255	5,255	5,255	5,255
Minimum	0.07	0.00	0.50	1
Maximum	4.43	75.00	380.0	45,000
Median	1.00	3.46	16.60	1,169
Length Weighted Mean	-	5.79	31.92	3,368
Standard Deviation	0.33	8.53	46.46	6,767
Coefficient of Variation	0.31	1.47	1.46	2.01
High-Grade Upper Zone				
Number of Cases	67	67	67	67
Minimum	0.52	0.21	0.10	53
Maximum	2.00	75.00	250.1	45,000
Median	1.10	3.28	13.49	2,604
Length Weighted Mean	-	8.99	26.13	5,871
Standard Deviation	0.45	15.70	38.93	8,849

Description	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)
Coefficient of Variation	0.37	1.75	1.49	1.51
High-Grade Zone Total				
Number of Cases	5,322	5,322	5,322	5,322
Minimum	0.07	0.00	0.10	1
Maximum	4.43	75.00	380.0	45,000
Median	1.00	3.46	16.50	1,180
Length Weighted Mean	-	5.84	31.84	3,404
Standard Deviation	0.33	8.68	46.36	6,807
Coefficient of Variation	0.31	1.49	1.46	2.00
Low-Grade Main Zone[1]				
Number of Cases	4,839	4,839	4,839	4,839
Minimum	0.18	0.01	0.00	1
Maximum	5.00	25.00	220.0	22,000
Median	1.00	1.25	6.70	338
Length Weighted Mean	-	1.62	14.51	848
Standard Deviation	0.39	1.83	25.56	1,832
Coefficient of Variation	0.33	1.13	1.76	2.16
Low-Grade Lower Zone[1]				
Number of Cases	655	655	655	655
Minimum	0.24	0.02	0.20	8
Maximum	3.00	25.00	220.0	22,000
Median	1.00	1.62	10.20	990
Length Weighted Mean	-	2.15	14.66	1,629
Standard Deviation	0.32	2.23	20.87	2,654
Coefficient of Variation	0.30	1.04	1.42	1.63
Low-Grade Zone Total[1]				
Number of Cases	5,494	5,494	5,494	5,494
Minimum	0.18	0.01	0.00	1
Maximum	5.00	25.00	220.0	22,000
Median	1.00	1.28	7.20	383
Length Weighted Mean	-	1.68	14.52	934
Standard Deviation	0.38	1.88	25.08	1,956
Coefficient of Variation	0.33	1.12	1.73	2.09

[1] Unsampled intervals not included in statistical analysis.

14.6 Compositing

Assay sample lengths range from 0.07 m to 5.00 m within the wireframe domains. Slightly less than half the samples (47%) were taken at 1.0 m lengths. The median assay length is 1.0 m and mean assay length is 1.11 m. Approximately 0.7% have lengths greater than 2.0 m. Given these distributions and considering the width of mineralisation, the QP determined that a composite length of 2.0 m was appropriate.

A set of wireframe intersection intervals were identified for each drillhole within the mineralised wireframe domains and resource assays were composited starting at the first composite control interval from the collar and resetting at each new wireframe boundary. Within the Low-Grade Zone wireframe models, unsampled intervals were treated as null. Within the High-Grade Zone wireframe models, all unsampled intervals were metallurgical samples and treated as unsampled. A total of 151 Au, Ag, and Cu composites measuring less than 0.5 m were removed from the database prior to statistical analysis, variography, and resource estimation. The elimination of the small composites did not affect the overall integrity of the composited database.

Table 14.7 summarises statistics of the capped and uncapped composite resource assay values. When Au, Ag, and Cu values are compared to uncapped resource assays, the average grades have decreased slightly, while the COV values have been greatly reduced.

Table 14.7 – Descriptive Statistics of Resource Capped Composite Values

Description	Uncapped			Capped		
	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)
High-Grade Main Zone						
Number of Cases	2,846	2,846	2,846	2,846	2,846	2,846
Minimum	0.24	0.70	1	0.24	0.70	1
Maximum	341.68	1,483.8	143,469	75.00	373.6	45,000
Median	3.70	18.57	1,331	3.69	18.56	1,331
Arithmetic Mean	6.38	33.38	3,744	5.78	32.07	3,352
Standard Deviation	14.18	55.82	8,581	7.44	41.23	5,871
Coefficient of Variation	2.22	1.67	2.29	1.29	1.29	1.75
High-Grade Upper Zone						
Number of Cases	43	43	43	43	43	43
Minimum	0.27	1.20	65	0.27	1.20	65
Maximum	60.29	130.4	33,732	43.51	130.4	25,262
Median	3.86	15.00	3,183	3.86	15.00	3,183
Arithmetic Mean	9.38	26.06	6,154	8.89	26.06	5,956
Standard Deviation	12.12	26.91	7,563	10.38	26.91	6,910

Description	Uncapped			Capped		
	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)
Coefficient of Variation	1.29	1.03	1.23	1.17	1.03	1.16
High-Grade Zone Total						
Number of Cases	2,889	2,889	2,889	2,889	2,889	2,889
Minimum	0.24	0.70	1	0.24	0.70	1
Maximum	341.68	1,483.8	143,469	75.00	373.6	45,000
Median	3.71	18.50	1,346	3.71	18.49	1,345
Arithmetic Mean	6.42	33.27	3,780	5.83	31.98	3,391
Standard Deviation	14.16	55.50	8,570	7.50	41.06	5,895
Coefficient of Variation	2.20	1.67	2.27	1.29	1.28	1.74
Low-Grade Main Zone						
Number of Cases	2,968	2,968	2,968	2,968	2,968	2,968
Minimum	0.00	0.00	0	0.00	0.00	0
Maximum	71.04	1,385.4	78,334	24.46	220.0	22,000
Median	1.31	7.11	373	1.31	7.10	372
Arithmetic Mean	1.64	16.00	896	1.59	14.38	826
Standard Deviation	2.29	44.68	2,551	1.51	23.50	1,482
Coefficient of Variation	1.39	2.79	2.85	0.95	1.63	1.79
Low-Grade Lower Zone						
Number of Cases	368	368	368	368	368	368
Minimum	0.00	0.00	0	0.00	0.00	0
Maximum	24.00	434.9	75,711	15.77	207.1	20,917
Median	1.66	10.85	1,037	1.64	10.85	1,027
Arithmetic Mean	2.14	15.69	1,942	2.10	14.41	1,600
Standard Deviation	2.13	32.14	5,849	1.82	19.20	2,354
Coefficient of Variation	0.99	2.05	3.01	0.87	1.33	1.47
Low-Grade Zone Total						
Number of Cases	3,336	3,336	3,336	3,336	3,336	3,336
Minimum	0.00	0.00	0	0.00	0.00	0
Maximum	71.04	1,385.4	78,334	24.46	220.0	22,000
Median	1.34	7.61	419	1.34	7.60	419
Arithmetic Mean	1.70	15.97	1,011	1.65	14.38	912
Standard Deviation	2.27	43.47	3,109	1.55	23.06	1,619
Coefficient of Variation	1.34	2.72	3.07	0.94	1.60	1.78

14.7 Density

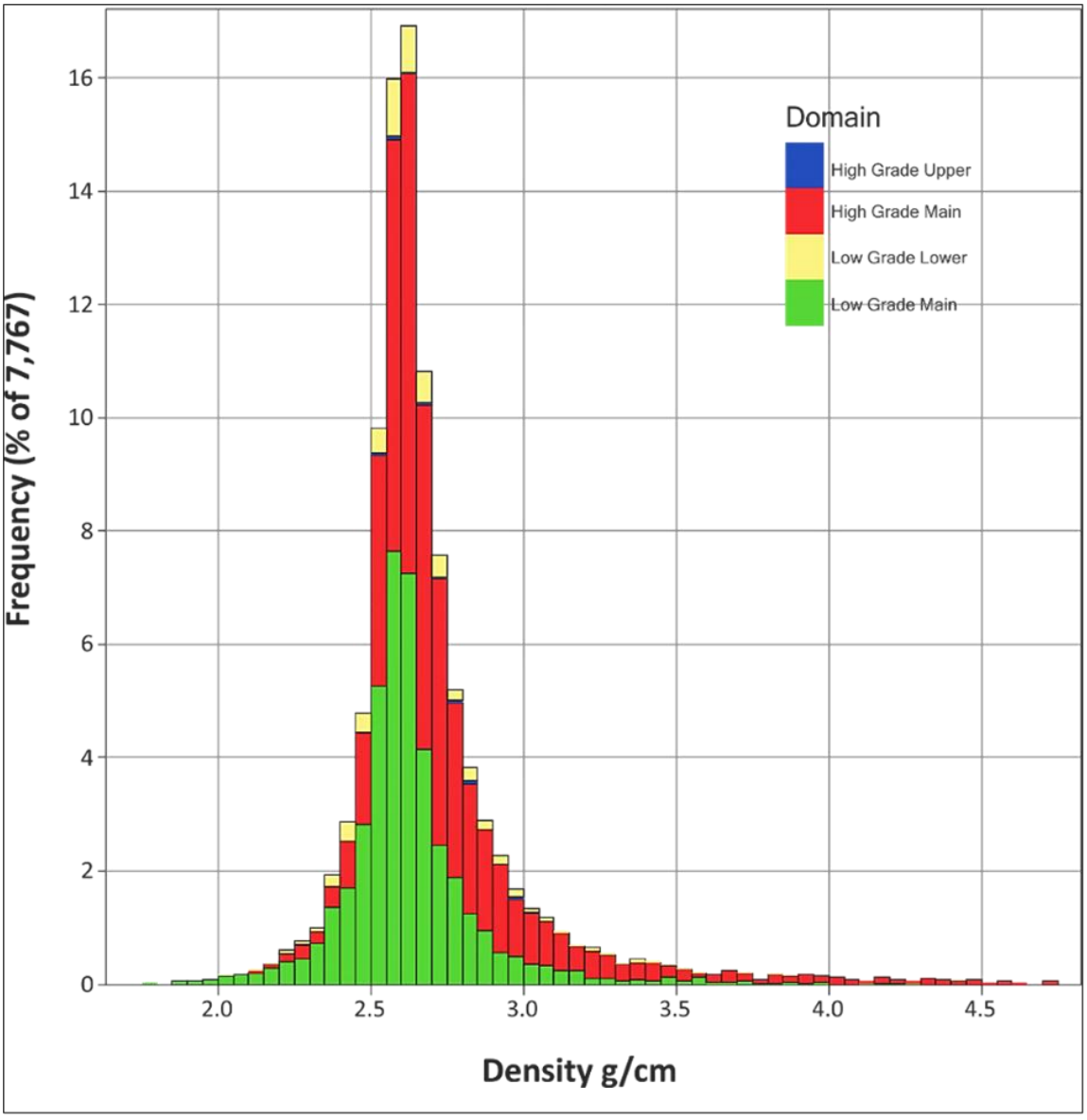
IAMGOLD began a systematic density measurement program at the Project in 2005 (IAMGOLD, 2009). To estimate densities, representative samples of typical lithology, alteration, and mineralisation styles were collected. Selected core was sealed in wax before estimating the density using the water immersion method.

The Loma Larga density database included 10,793 density measurements covering a diverse range of lithology, mineralisation, and alteration types. Approximately 72% (7,767) of the measurements were located within the mineralised wireframe domains. For this Report, the QP reviewed the descriptive statistics for density samples taken within the mineralisation wireframes (Table 14.8). Although the density data shows an approximately normal distribution, there is a long tail to the right of the mean (positively skewed). The QP reviewed the descriptive statistics (Table 14.8), histograms (Figure 14.4), and box plots (Figure 14.5) for density samples taken within the mineralisation wireframes and determined that there were no high density outliers that should be removed from the data set. The QP further tested whether density weighting should be applied to grade interpolation. The QP concluded that there was no strong correlation between grade and density.

Table 14.8 – Descriptive Statistics of Resource Density Values

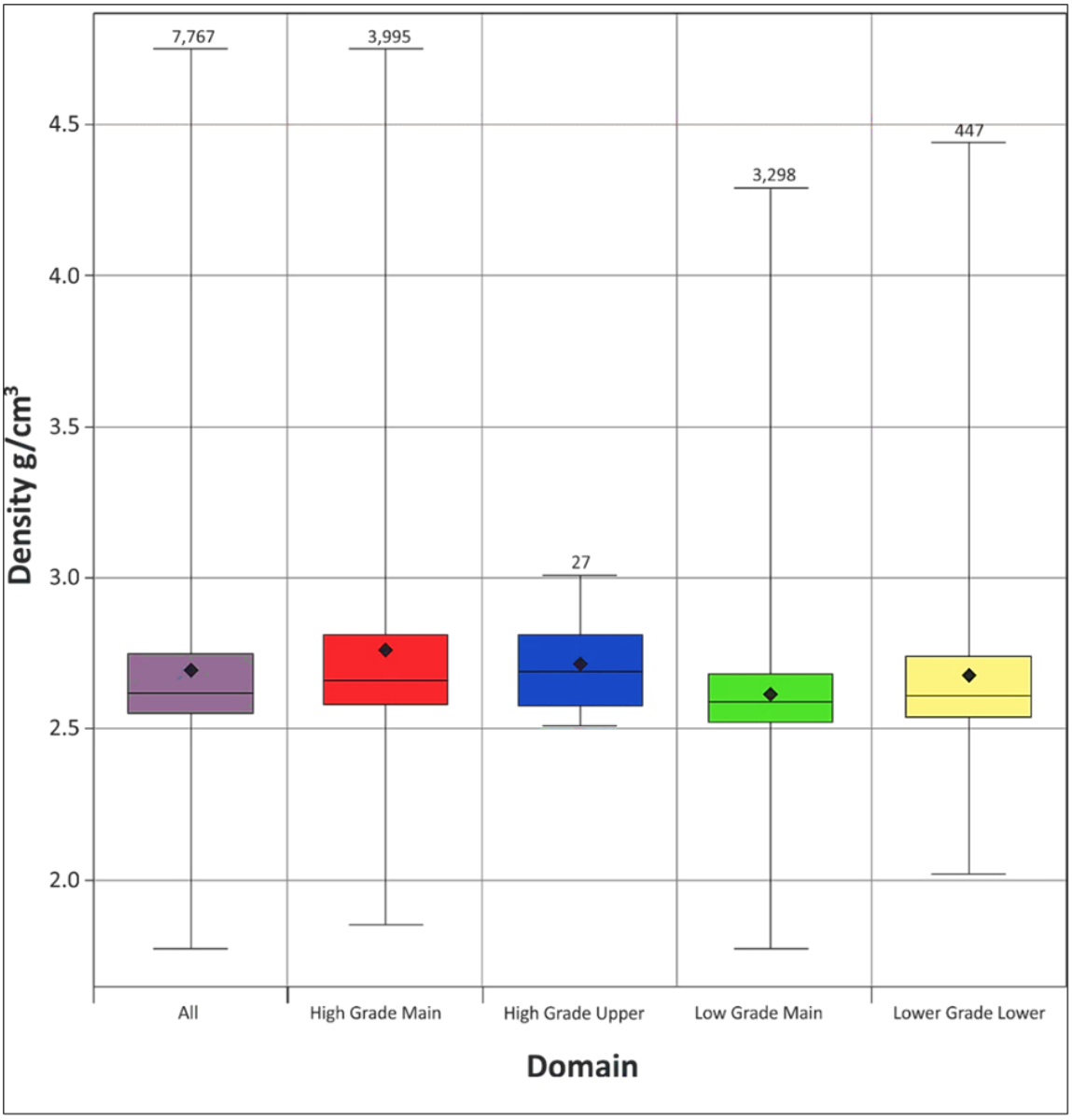
Domain	Rock Code	No of Cases	Min (g/cm³)	Max (g/cm³)	Median (g/cm³)	Arithmetic Mean	Std Dev.	COV
High-Grade Zone								
	130	3,995	1.85	4.75	2.66	2.76	0.34	0.12
	131	27	2.51	3.01	2.69	2.72	0.15	0.05
All High-Grade Zones		4,022	1.85	4.75	2.66	2.76	0.34	0.12
Low-Grade Zone								
	108	3,298	1.77	4.29	2.59	2.62	0.23	0.09
	208	447	2.02	4.44	2.61	2.68	0.29	0.11
All Low-Grade Zone		3,745	1.77	4.44	2.60	2.62	0.24	0.09
All Composites		**7,767**	**1.77**	**4.75**	**2.62**	**2.70**	**0.30**	**0.11**

Figure 14.4 – Histogram of Resource Density Samples



Source: DRA, 2021

Figure 14.5 – Box Plot of Resource Density Samples by Domain



Source: DRA, 2021

The average density of the High-Grade Main and Upper Zones is 2.76 g/cm³ and 2.72 g/cm³, respectively. The Low-Grade Main and Lower Zones have average density values of 2.62 g/cm³ and 2.68 g/cm³, respectively, and these values are consistent with those calculated by the QP in 2012 and 2016.

As with the resource assays, density samples were composited to 2.0 m, starting at the first mineralised wireframe boundary from the collar and resetting at each new wireframe boundary.

Composites less than 0.5 m were removed from the database for Mineral Resource estimation and unsampled intervals were treated as unsampled.

Table 14.9 summarises the descriptive statistics of the composited resource density samples.

Table 14.9 – Descriptive Statistics of Composited Density Samples

Domain	Rock Code	No. of Cases	Min (g/cm³)	Max (g/cm³)	Median (g/cm³)	Arithmetic Mean	Std. Dev.	COV
High-Grade Zone								
	130	2,201	2.09	4.46	2.68	2.74	0.25	0.09
	131	21	2.38	2.90	2.65	2.67	0.12	0.05
All High-Grade Zone		2,222	2.09	4.46	2.68	2.74	0.25	0.09
Low-Grade Zone								
	108	2,474	1.85	3.61	2.57	2.58	0.18	0.07
	208	292	2.20	3.90	2.59	2.65	0.23	0.09
All Low-Grade Zone		2,766	1.85	3.90	2.57	2.59	0.19	0.07
All Composites		**4,988**	**1.85**	**4.460**	**2.62**	**2.66**	**0.23**	**0.09**

14.8 NSR Cut-Off Value

A potential underground production scenario serves as the basis for estimating the cut-off grade for Mineral Resources at Loma Larga and NSR factors were developed by DPM for the purposes of Mineral Resource reporting.

NSR is the estimated value per tonne of mineralised material after allowance for metallurgical recovery and consideration of smelter terms, which may include payables, treatment charges, refining charges, price participation, penalties, smelter losses, transportation, and sales charges. These assumptions, summarised in Table 14.10, are based on the current processing scenario and results from metallurgical testwork.

Table 14.10 – Cut-Off Value Assumptions

Parameter	Unit	Value	
		Copper Concentrate	Pyrite Concentrate
Au Price	US$/oz	1,700	
Ag Price	US$/oz	20	
Cu Price	US$/lb	3.75	
Au Recovery	%	15	75.2
Ag Recovery	%	50	45.7

Parameter	Unit	Value	
		Copper Concentrate	Pyrite Concentrate
Cu Recovery	%	92[1]	0
Au Payable	%	96	94.3[2]
Ag Payable	%	90	88.9[3]
Cu Payable	%	88.6[4]	0
As Penalties	US$	calculated[5]	None
Base Treatment Charges	US$/dmt	100	150
Transport[6]	US$/wmt	50	50
Port Costs	US$/wmt	13	13
Ocean freight	US$/wmt	50	50
Weighing, Sampling, and Moisture Determination Charges	US$/wmt	5	5
Au Refining Charges	US$/oz	5	5
Ag Refining Charges	US$/oz	0.5	0.5
Cu Refining Charges	US$/lb	0.01	0

Mining	US$/t ore	26.54[7]
Process	US$/t ore	19.10
Tailings Management and Backfill Plant	US$/t ore	6.13
G & A Costs	US$/t ore	8.07
Sustaining Costs	US$/t ore	7.84
Total Operating Cost	US$/t ore	67.68

1 Formula used: If (Cu (%)>0.1, Cu (%)<1.0), value is 5 x LN [Cu (%) + 92] and if Cu (%)≥ 1.0, value is 92.
2 Formula used: 97/Au Grade x (Au Grade –1).
3 Formula used: 95/Ag Grade x (Ag Grade –30).
4 Value is the minimum between 0.965 and ((Cu Conc Grade-1) / (Cu Conc Grade)) x 100.
5 Formula used: ((As Grade-0.3)/0.1) x 5.0.

The NSR calculation have been modified from those used in the 2021 Mineral Resource Estimate, and the net effect on the NSR factors is material. Changes include lower metal prices, updates on copper recovery, concentrate grade and payables in both concentrates, addition of arsenic penalties and higher operating costs, including transport costs.

The NSR value (in units of US$/t) for each block in the block model, was compared directly to unit operating costs required to mine that block. For the purposes of developing an NSR cut-off value for an underground mining operation, a total operating cost of US$65/t (rounded) was assumed, which

is the total cost per tonne of ore including mining, processing, G&A, sustaining, and closure costs. The NSR cut-off value has increased from US$55/t used in the 2021 Technical Report.

In the QP's opinion, an NSR of US$65/t (rounded) is suitable for an underground mining scenario.

14.9 Variography and Interpolation Values

In 2018, the QP re-evaluated variography for Au, Ag, Cu, and density and also prepared variograms using composites located within the main mineralised domains. The nugget effect was established with the downhole linear semi-variogram and was well developed in all cases.

For both the High-Grade and Low-Grade Zones, grades for Au, Ag, and Cu were interpolated using Ordinary Kriging (OK) and density values were also interpolated using OK. Variography was used to determine the search ellipsoid orientation and dimensions. Parameters derived for the High-Grade Main Zone were applied to the High-Grade Upper Zone and the parameters derived for the Low-Grade Main Zone were applied the Low-Grade Lower Zone. Interpolation and search parameters used by RPA are summarised in Table 14.11.

Table 14.11 – Block Estimation Parameters

		Au	Ag	Cu	Density
Parameter	Interpolation Method	OK	OK	OK	OK
	Boundary Type	Hard	Hard	Hard	Soft
High-Grade Zone Sample Restrictions	Min. No. Comps.	4	4	4	3
	Max. No. Comps.	12	12	12	15
	Max. Comps. Per Drillhole	3	3	3	3
High-Grade Zone Search Ellipse	Range X (m)	200	180	190	300
	Range Y (m)	140	150	140	240
	Range Z (m)	40	60	65	55
High-Grade Zone Search Anisotropy1	Z	+0°	+165°	+165°	-160°
	Y	+0°	+15°	+10°	+15°
	Z	+75°	-90°	-90°	+90°
Variogram Model High-Grade Zone	Nugget (C0)	0.06	0.09	0.05	0.11
	Relative Nugget (C0)	6%	9%	5%	11%

		Au	Ag	Cu	Density
Structure High-Grade Zone	C1	0.61	0.11	0.28	0.32
	Range X (m)	43	29	20	12
	Range Y (m)	17	18	26	20
	Range Z (m)	2	6	8	3
	C2	0.34	0.81	0.68	0.58
	Range X (m)	98	89	93	41
	Range Y (m)	71	73	70	71
	Range Z (m)	20	27	33	22
	Total Sill	1.00	1.00	1.00	1.00
Low-Grade Zone Sample Restrictions	Min. No. Comps.	4	4	4	3
	Max. No. Comps.	12	12	12	15
	Max. Comps. Per Drillhole	3	3	3	3
Low-Grade Zone Search Ellipse	Range X (m)	240	390	390	300
	Range Y (m)	150	270	190	240
	Range Z (m)	60	45	60	55
Low-Grade Zone Search Anisotropy[1]	Z	-165°	+0°	+45°	-160°
	Y	+10°	+0°	+10°	+15°
	Z	+75°	+75°	+45°	+90°
Variogram Model Low-Grade Zone	Nugget (C0)	0.23	0.16	0.28	0.11
	Relative Nugget (C0)	23%	16%	28%	11%
Structure Low-Grade Zone	C1	0.73	0.29	0.33	0.32
	Range X (m)	53	26	57	12
	Range Y (m)	43	65	56	20
	Range Z (m)	12	17	6	3
	C2	0.04	0.56	0.39	0.58
	Range X (m)	118	268	258	41
	Range Y (m)	76	180	125	71
	Range Z (m)	29	29	41	22
	Total Sill	1.00	1.00	1.00	1.00

1 Rotation around each axis (positive is counterclockwise)

Block grade interpolation was carried out in a single pass using OK. Interpolation was restricted by the mineralised wireframe models for Au, Ag, and Cu, which were used as hard boundaries to prevent the use of composites outside of the zones. A minimum of four (4) 2.0 m composites and a

maximum of 12 composites were used to interpolate grades within each block for Au, Ag, and Cu, with a maximum of three (3) composites per drillhole.

A soft boundary was used to interpolate density in the High- and Low-Grade Main Zones. A soft boundary allowed composites from one (1) zone to be used for density interpolation of the other zone. A minimum of three (3) 2.0 m composites and a maximum of 15 composites were used to interpolate density within each block with a maximum of three (3) composites per drillhole. The wireframe model was used to restrict the density interpolation of the Low-Grade Lower Zone, using the same composite sample restriction as the main zones. The average value of domain density composites was coded directly into the block model for the High-Grade Upper Zone.

14.10 Block Model

A model of 2,992,500 blocks was built in GEMS with block sizes of 5 m by 10 m by 5 m with 270 columns, 235 rows, and 90 levels. The block model was subsequently converted to octree format for import and reporting in Leapfrog Edge software. The model is oriented N15ºW and fully encloses the modelled resource wireframes. The extents and dimensions of the Leapfrog block model are summarised in Table 14.12.

Table 14.12 – Block Model Dimensions

Description	**Easting** (X)	**Northing** (Y)	**Elevation** (Z)
Minimum (m)	698,366.43	9,662,637.824	3,400
Maximum (m)	699,316.43	9,664,387.824	3,850
Extents (m)	950	1,750	450
Description	**Column**	**Row**	**Level**
Block Size (m)	5	10	5
Number of Parent Blocks	190	175	90
Sub Block Count	4	4	32
Sub Block Minimum Size (m)	1.25	2.5	0.15625

Block model attributes include rock type, density, Au, Ag, Cu, S grades, and NSR value (Table 14.13).

Table 14.13 – Block Model Field Descriptions

Block Model	Description
Rock Type	Air=0, Waste=99, High-Grade Main Zone = 130, High-Grade Upper Zone = 131, Low-Grade Main Zone = 108, Low-Grade Lower Zone = 208/308
Density	High-Grade and Low-Grade Main Zones interpolated density composite samples High-Grade Upper Zone = 2.67 g/cm³
Percent	Percent of block ascribed to the assigned rock type
AU_KR	Interpolated Au capped grades using OK
AG_KR	Interpolated Ag capped grades using OK
CU_KR	Interpolated Cu capped grades using OK
S[1]	Interpolated S values using OK
As_ppm[2]	Interpolated As values using OK
CLASS	Classification of block (1 = Measured, 2 = Indicated, 3 = Inferred)
Min-Dist	Distance from the block centroid to the nearest sample used in block estimate
No-DHs	Number of drillholes used in block estimate
No-Samp	Number of sample points used in block estimate
NSR_23	2023 NSR factor calculated from OK metal grades

[1] See Section 14.15 - Sulphur Model.
[2] As was estimated into the block model to assist NSR calculations.

The density factor applied to the High-Grade Main Zone and Low-Grade Main and Lower Zones was interpolated into each block using OK and applied to convert block volume to tonnage. A tonnage factor of 2.67 g/cm³ for the High-Grade Upper Zone was coded directly into each block based on the rock type model.

The block model was converted to a sub-cell block model with 4 x 8 x variable sub-blocks per parent cell to calculate Mineral Reserves and generate underground reporting shapes with DSO software. The 2023 Loma Larga Mineral Resources were reported using Datamine software.

14.11 Block Model Validation

The QP carried out a number of block model validation procedures including:

- Visual comparisons of block gold, silver, and copper grades versus composite grades.
- Statistical comparisons.
- Comparison of the volumes of the wireframe models to the block model volume results.

- Trend plots of block and composite gold, silver, and copper grades by elevation and northings / eastings.
- Comparison of block and composite grades in blocks containing composites.

Block model grades were visually examined and compared with composite grades in vertical cross sections and on elevation level plans. The QP found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drillhole assay and composite grades and that there was no significant bias.

Grade statistics for assays, composites, and resource blocks were examined and compared for the High-Grade Main and Upper Zones and Low-Grade Main and Lower Zones (Table 14.14). In the Low-Grade Lower Zone, average block grades are slightly higher than average composite grades. This is attributed to a larger influence of higher grade drillholes in some parts of these zones due to their relative location and spacing locally, and accordingly the blocks are assigned an Inferred classification (Figure 14.6). Otherwise, the comparisons of average grades of capped assays, composites, and blocks are reasonable, in the QP's opinion.

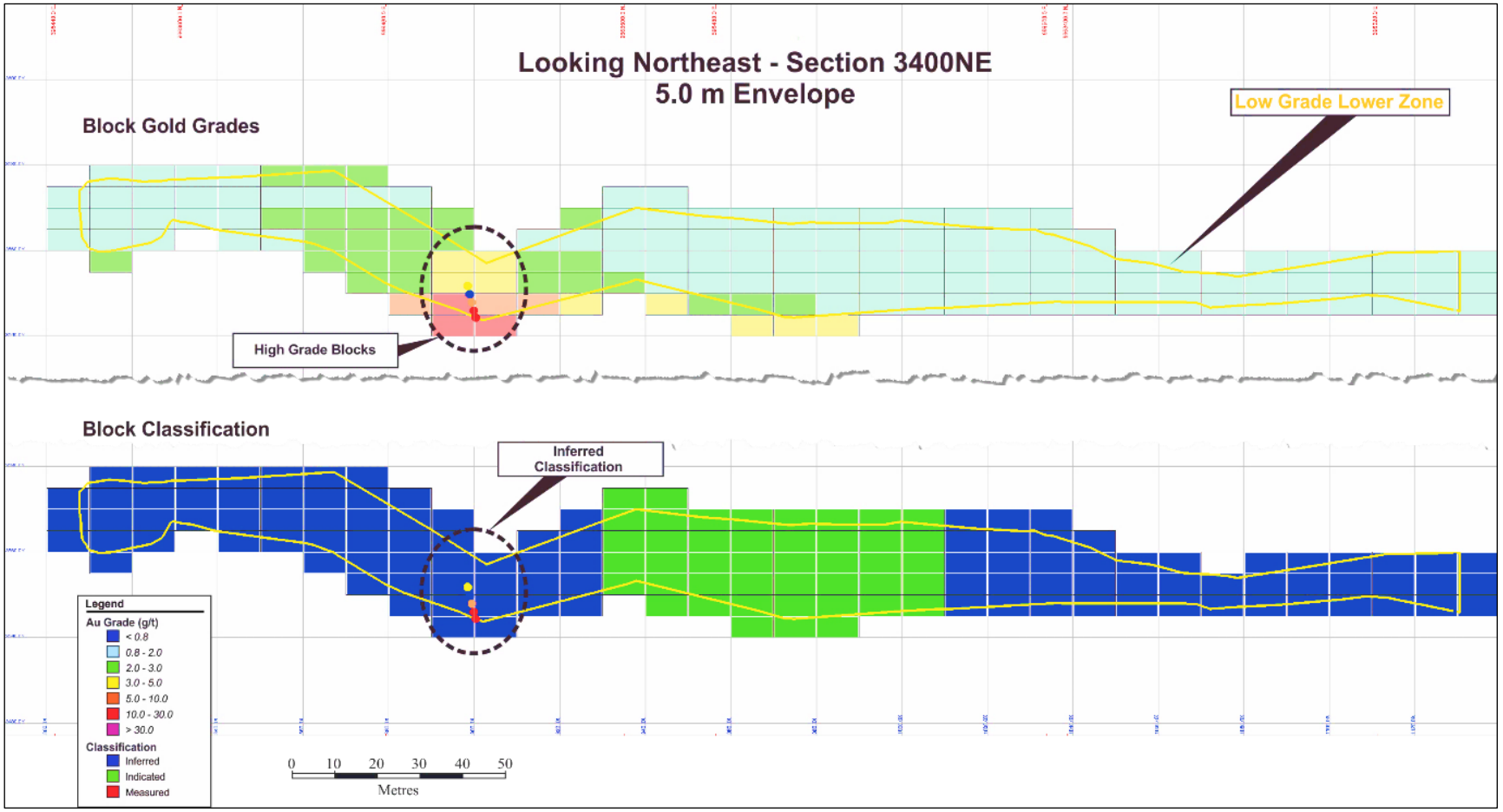
Table 14.14 – Comparison of Grade Statistics for Assays, Composites and Resource Blocks

Zone	Capped Assays			2.0 m Capped Composites			Block Model Grades		
	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)
High-Grade Main Zone									
Number of Cases	5,255	5,255	5,255	2,846	2,846	2,846	27,365	27,365	27,365
Minimum	0.00	0.50	1	0.24	0.70	1	1.49	4.22	153
Maximum	75.00	380.0	45,000	75.00	373.6	45,000	58.80	243.17	36,047
Median	3.46	16.60	1,169	3.69	18.56	1,331	4.09	23.41	1,913
Arithmetic Mean[1]	5.79	31.92	3,368	5.78	32.07	3,352	5.34	31.58	3,060
Standard Deviation	8.53	46.46	6,767	7.44	41.23	5,871	3.82	24.44	3,299
Coefficient of Variation	1.47	1.46	2.01	1.29	1.29	1.75	0.72	0.77	1.08
High-Grade Upper Zone									
Number of Cases	67	67	67	43	43	43	548	548	548
Minimum	0.21	0.10	53	0.27	1.20	65	2.48	6.28	1,751
Maximum	75.00	250.1	45,000	43.51	130.4	25,262	20.30	66.47	18,331
Median	3.28	13.49	2,604	3.86	15.00	3,183	6.57	19.65	5,358
Arithmetic Mean[1]	8.99	26.13	5,871	8.89	26.06	5,956	7.54	22.62	5,895
Standard Deviation	15.70	38.93	8,849	10.38	26.91	6,910	3.68	10.30	2,409
Coefficient of Variation	1.75	1.49	1.51	1.17	1.03	1.16	0.49	0.46	0.41

Zone	Capped Assays			2.0 m Capped Composites			Block Model Grades		
	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)	Au (g/t)	Ag (g/t)	Cu (ppm)
Low-Grade Main Zone									
Number of Cases	4,839	4,839	4,839	2,968	2,968	2,968	53,957	53,957	53,957
Minimum	0.01	0.00	1	0.00	0.00	0	0.11	0.78	23
Maximum	25.00	220.0	22,000	24.46	220.0	22,000	11.38	199.66	11,098
Median	1.25	6.70	338	1.31	7.10	372	1.45	10.47	673
Arithmetic Mean[1]	1.62	14.51	848	1.59	14.38	826	1.54	15.94	837
Standard Deviation	1.83	25.56	1,832	1.51	23.50	1,482	0.62	18.23	733
Coefficient of Variation	1.13	1.76	2.16	0.95	1.63	1.79	0.40	1.14	0.88
Low-Grade Lower Zone									
Number of Cases	655	655	655	368	368	368	5,781	5,781	5,781
Minimum	0.02	0.20	8	0.00	0.00	0	0.30	2.53	186
Maximum	25.00	220.0	22,000	15.77	207.1	20,917	11.24	153.12	16,444
Median	1.62	10.20	990	1.64	10.85	1,027	1.78	13.14	1,120
Arithmetic Mean[1]	2.15	14.66	1,629	2.10	14.41	1,600	1.96	16.06	1,548
Standard Deviation	2.23	20.87	2,654	1.82	19.20	2,354	0.91	13.43	1,587
Coefficient of Variation	1.04	1.42	1.63	0.87	1.33	1.47	0.47	0.84	1.03

[1] Assay mean values are length weighted.

Figure 14.6 – High Gold Grade and Classification of Selected Block in the Low-Grade Lower Zone



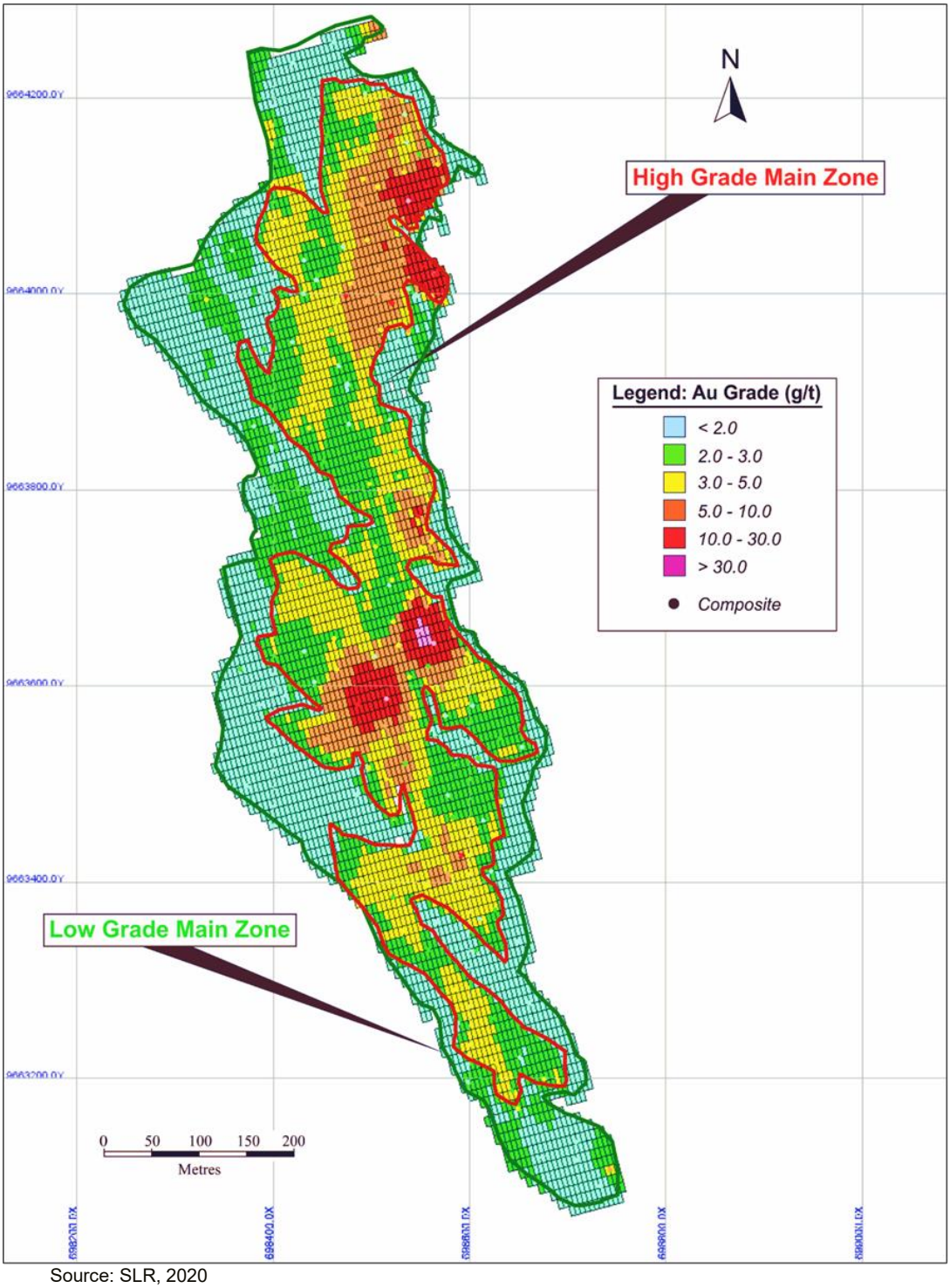
Source: SLR, 2020

Examples of the composite and block grades of Au, Ag, and Cu in plan and sections are provided in Figure 14.7, Figure 14.8, and Figure 14.9, respectively.

To check for conditional bias, trend plots were created which compared the Au, Ag, and Cu block model grade estimates of the High-Grade Zone and Low-Grade Zone to capped composite sample average grades. Figure 14.10 and Figure 14.11 illustrate the gold trend plots for the High-Grade Main Zone and Low-Grade Main Zone, respectively. In the QP's opinion, there is no significant bias between the resource block grades and the composited capped assay samples.

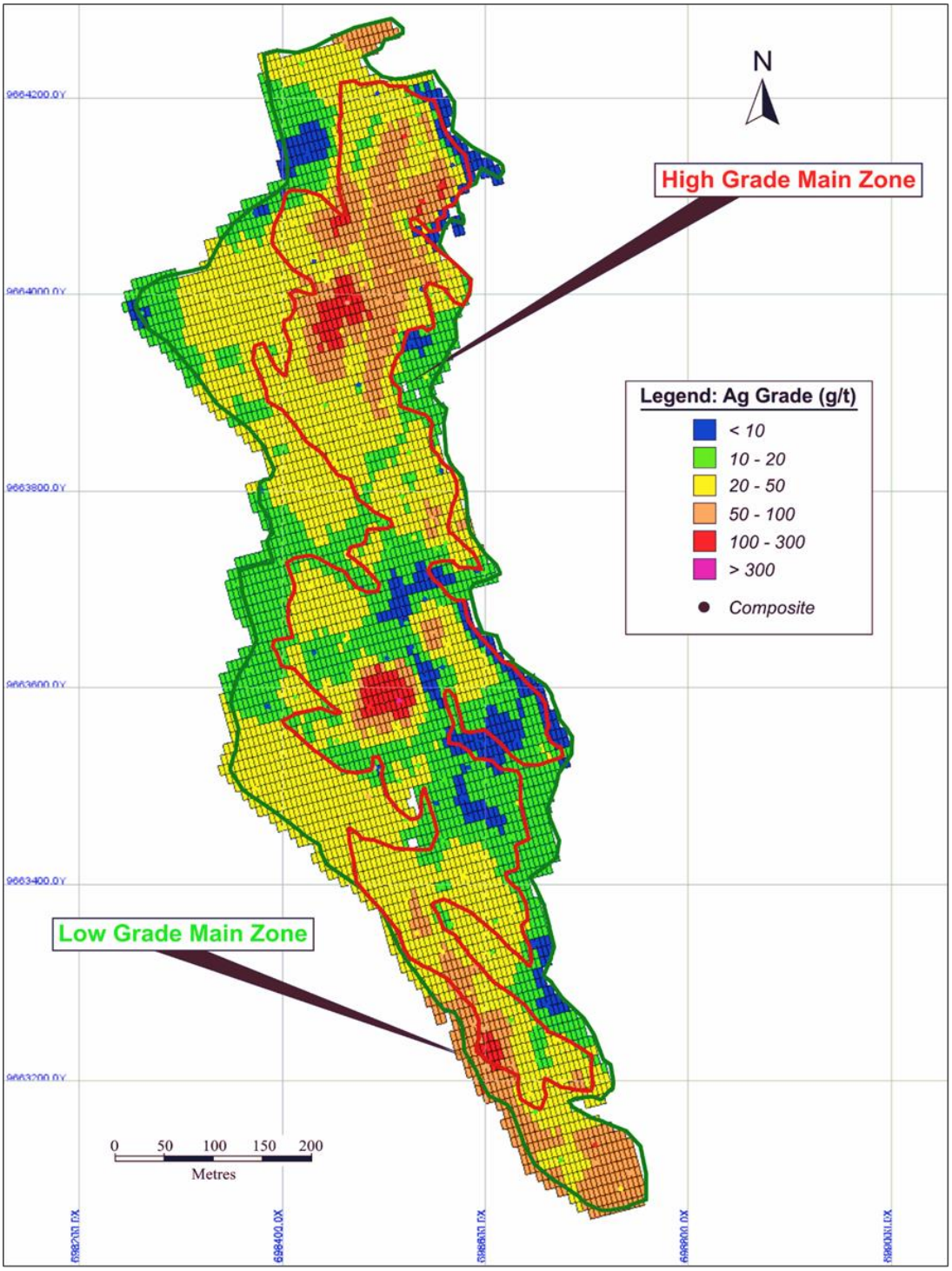
The QP compared the volume of the wireframe models to the block model volume results. The estimated total volume of the wireframe models is 14,407,350 m³ and the block model volume in GEMS is 14,395,927 m³. In addition, the wireframe volume was compared to the block model volume in Micromine, which is 14,396,544 m³. The volume difference in both cases is -0.08%, which the QP considers to be an acceptable result.

Figure 14.7 – Capped Gold Composites and Blocks on 3,600 m Level



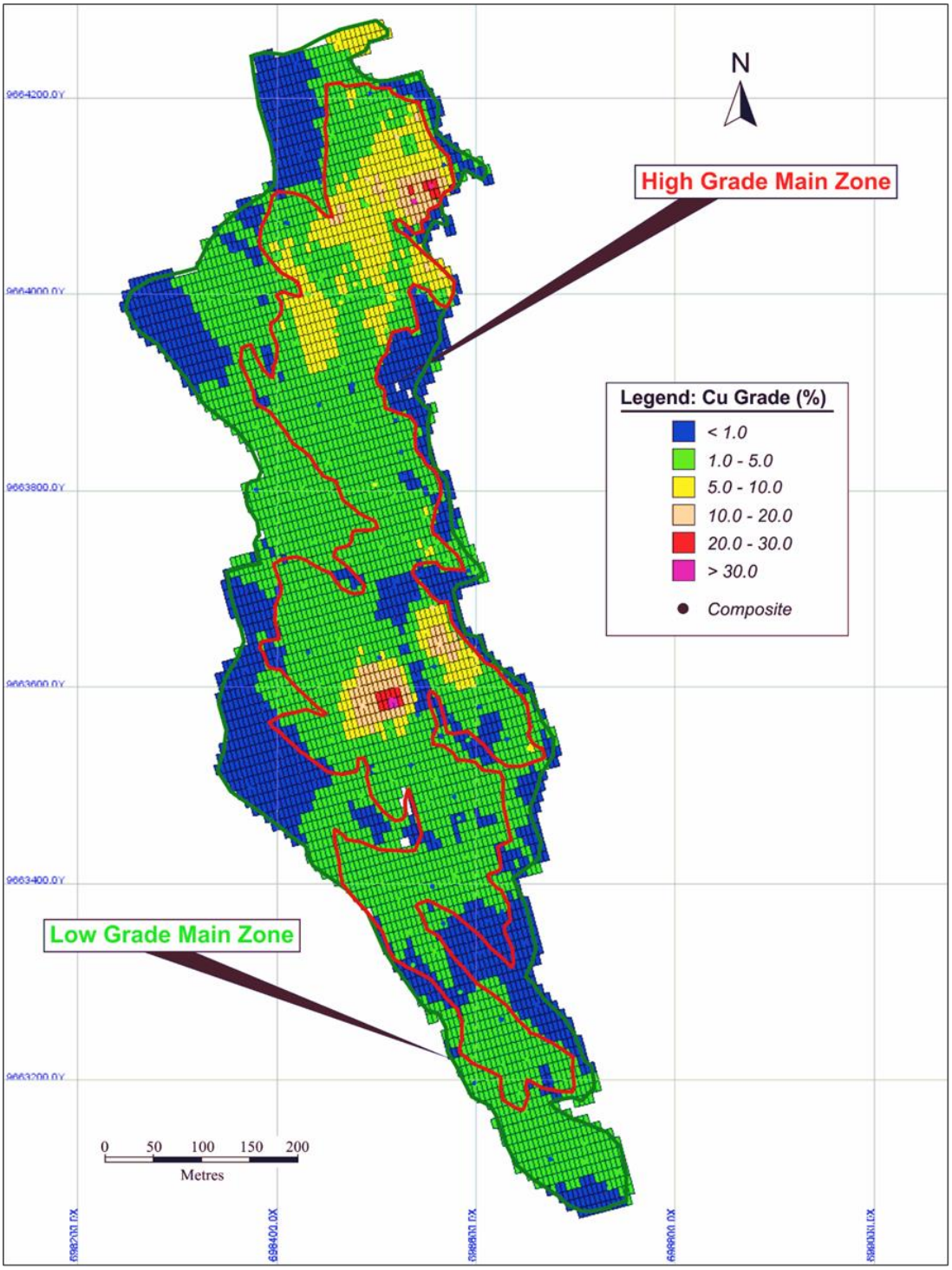
Source: SLR, 2020

Figure 14.8 – Capped Silver Composites and Blocks on 3,600 m Level



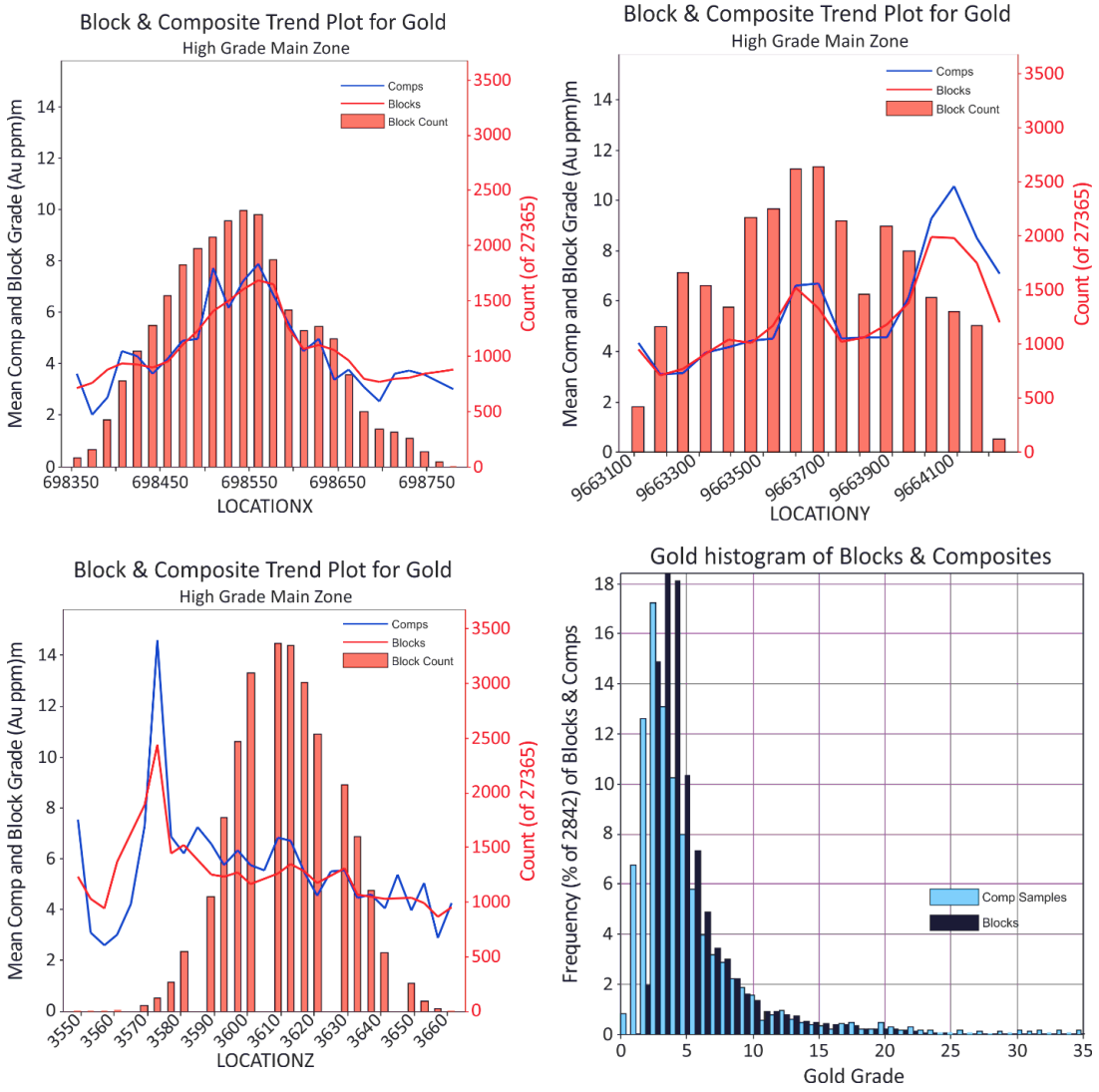
Source: SLR, 2020

Figure 14.9 – Capped Copper Composites and Blocks on 3,600 m Level



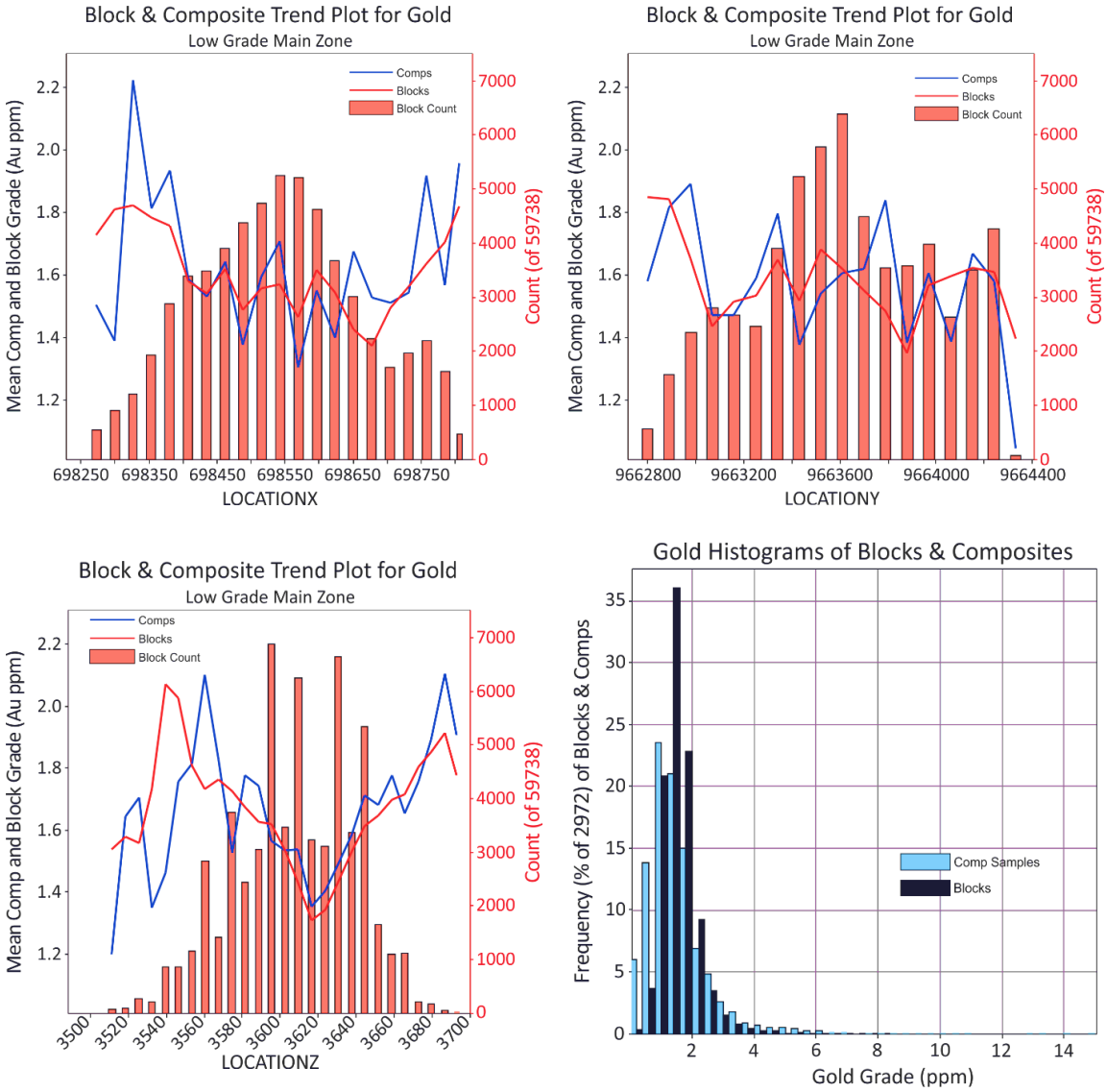
Source: SLR, 2020

Figure 14.10 – High-Grade Main Zone Trend Plots and Histograms of Capped Gold Assays vs. Block Grades



Source: SLR, 2020

Figure 14.11 – Low-Grade Main Zone Trend Plots and Histograms of Capped Gold Assays vs. Block Grades



Source: SLR, 2020

14.12 Classification

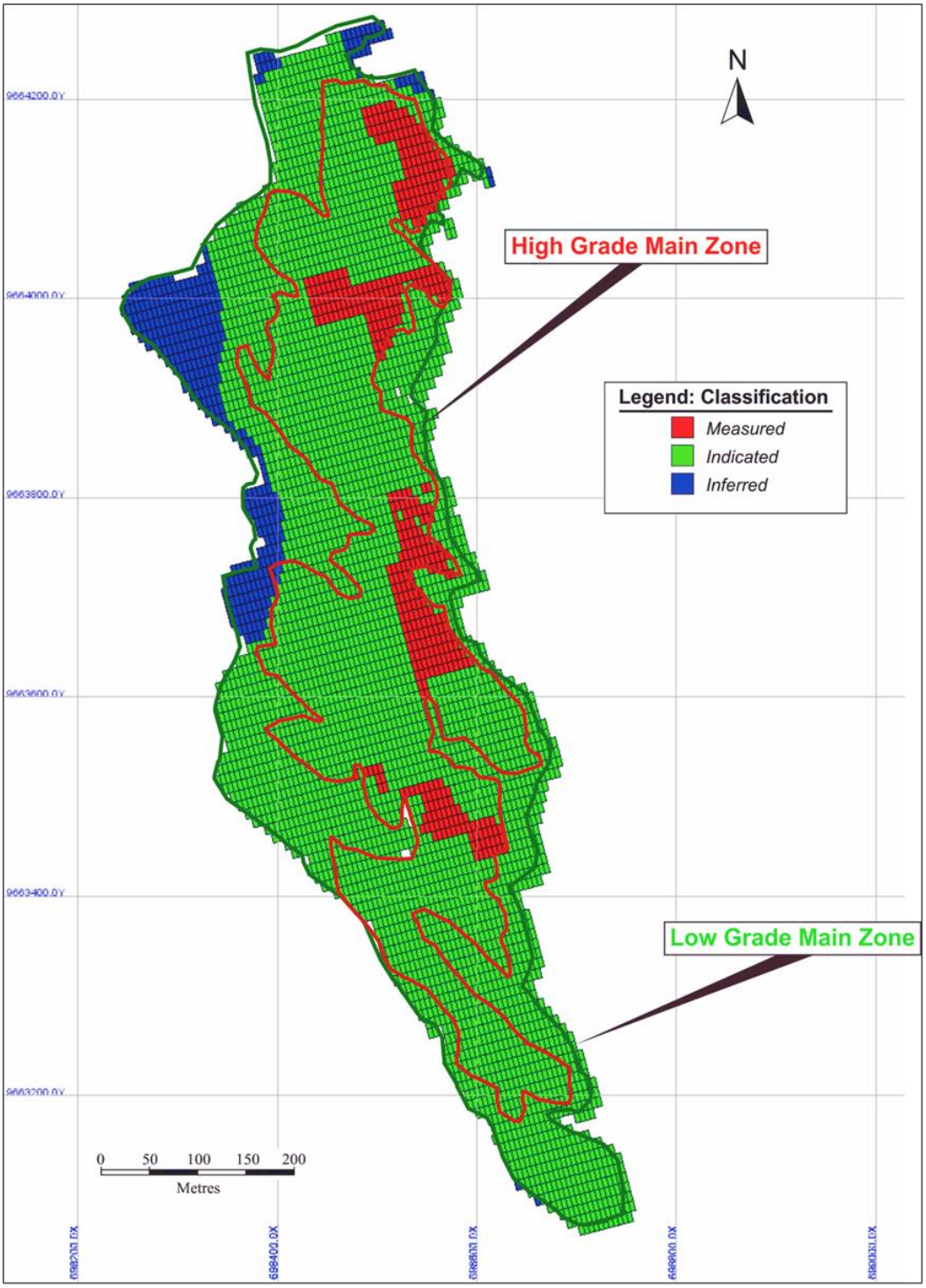
Definitions for resource categories used in this report are consistent with those defined by Canadian Institute of Mining, Metallurgy, Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) as incorporated by reference in NI 43-101. In the CIM classification, a Mineral Resource is defined as "*a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction*". Mineral Resources are classified into Measured, Indicated, and Inferred categories, according to the confidence level in the estimated blocks.

The QP classified the Mineral Resources on the Loma Larga Project as Measured, Indicated, and Inferred. All blocks within the High-Grade Main Zone were classified as Measured or Indicated Mineral Resources, and all blocks in the High-Grade Upper Zone were classified as Inferred Mineral Resources. Blocks within the Low-Grade Main Zone and Low-Grade Lower Zone were classified as Indicated and Inferred. The QP manually classified the blocks within these zones based on drillhole spacing and grade continuity. The general guidelines for drillhole spacing are as follows:

- Measured: 25 m x 25 m.
- Indicated: 50 m x 50 m.
- Inferred: > 100 m x 100 m.

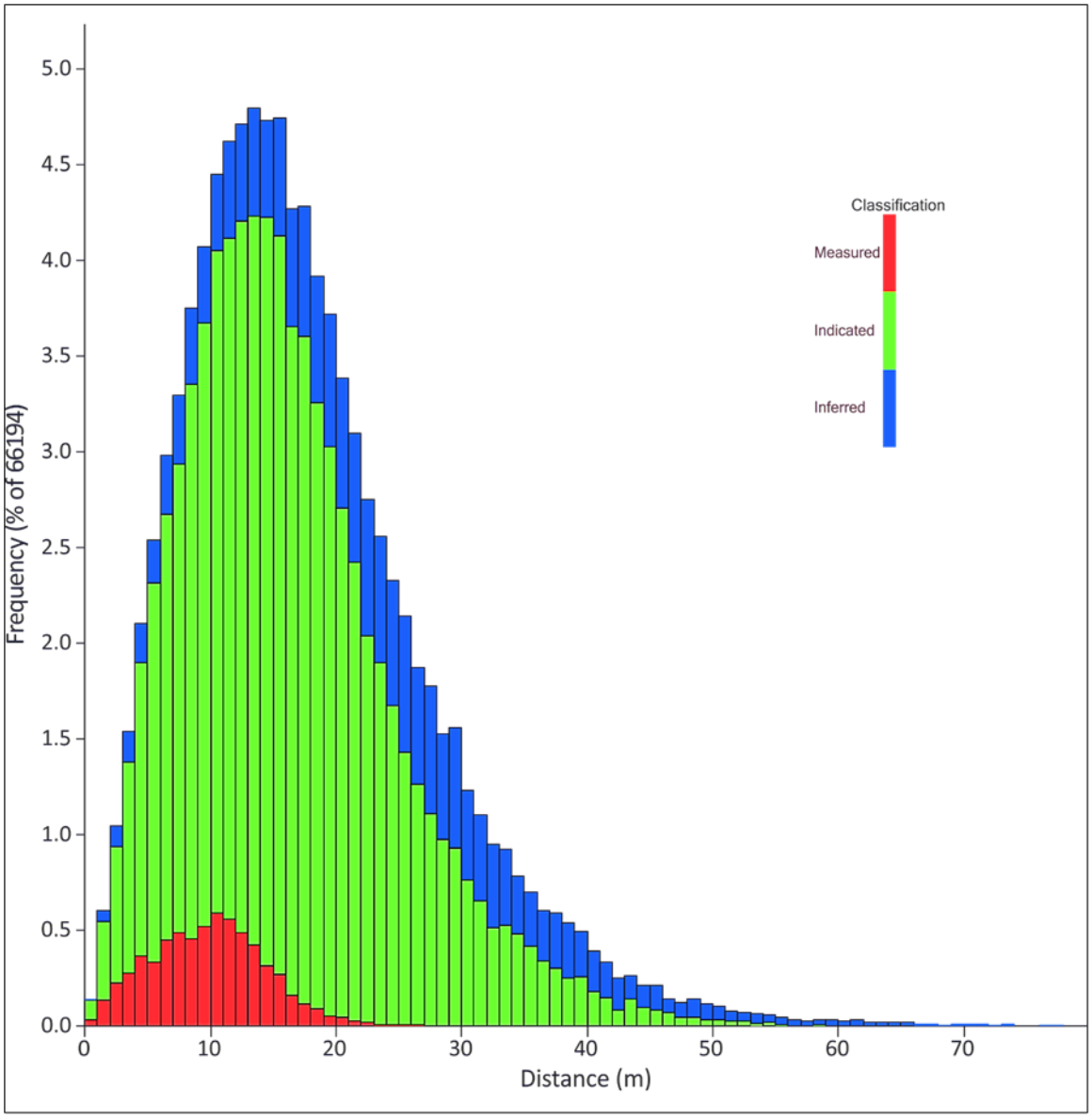
Figure 14.12 illustrates block classification on the 3600 m Level, and the distance to the nearest sample is shown for Measured, Indicated, and Inferred blocks as a histogram in Figure 14.13. Drillhole spacing for Measured blocks averages 20 m, and ranges from less than 10 m to 40 m. Nearly 75% of blocks classified as Measured have a drillhole spacing less than 25 m.

Figure 14.12 – Plan View of Block Classification on the 3,600 m Level



Source: SLR, 2020

Figure 14.13 – Cumulative Histogram of Distance to Nearest Sample Grouped by Class

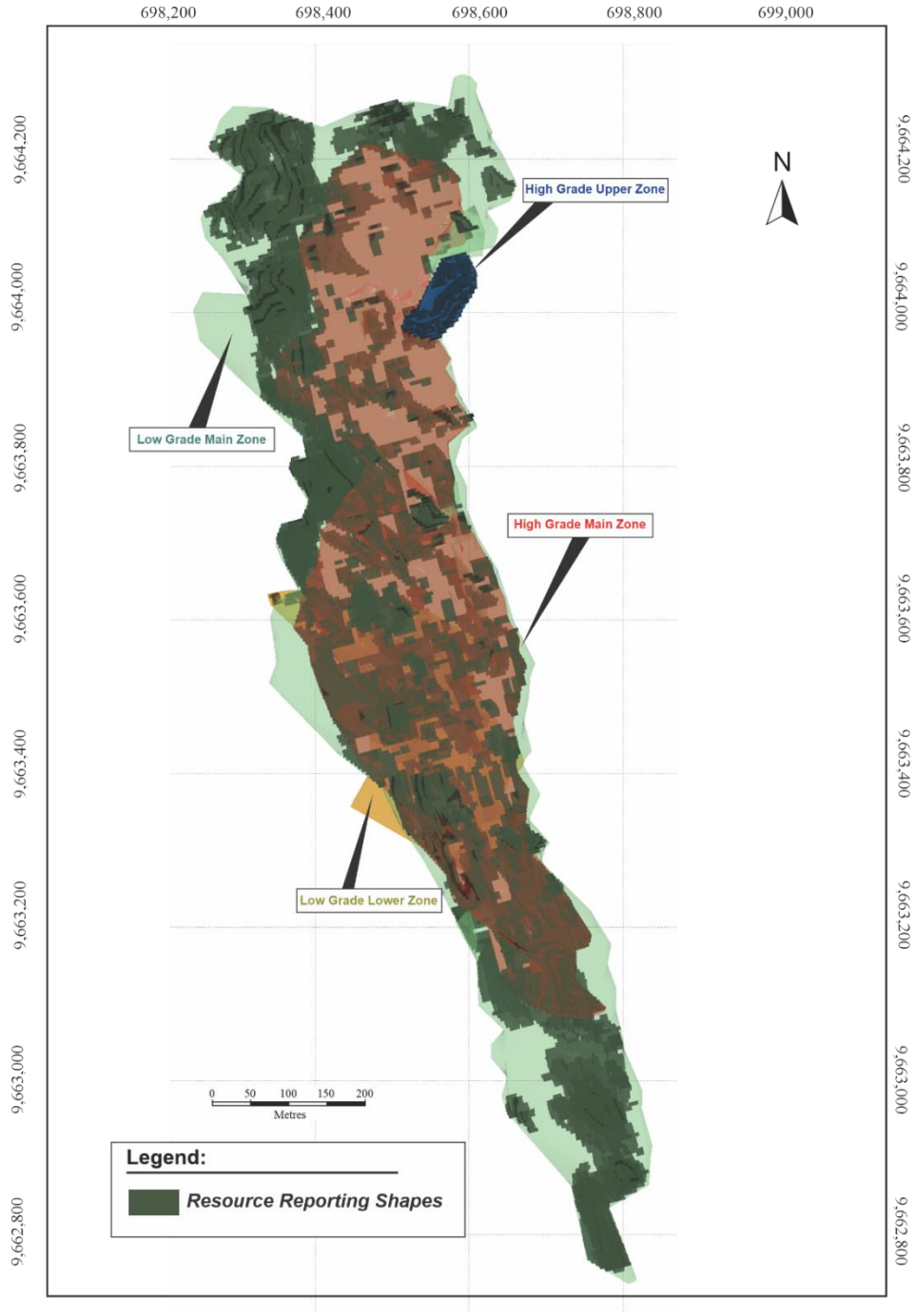


Source: SLR, 2020

Most of the blocks classified as Indicated Mineral Resources in the Low-Grade Main Zone have good continuity, but the QP cautions that there are a few isolated drillhole intercepts with grades above the cut-off that are classified as Indicated.

For the purposes of demonstrating RPEEE, underground reporting shapes were created using the DSO at an NSR cut-off value of US$65/t (Figure 14.14). DSO volumes constraints were set to a variable length of 10 m to 20 m, and a minimum width and height of 5 m. All blocks within the DSO volumes were reported to Mineral Resources.

Figure 14.14 – DSO Volumes Used for Underground Resource Reporting in Plan View



Source: SLR, 2023

14.13 Summary of Mineral Resource Estimate

Mineral Resources were estimated for the Loma Larga Project using drillhole data available as of October 31, 2018, no further drilling has been completed since that date.

In the QP's opinion, an NSR cut-off value of US$65/t is appropriate for constraining current Mineral Resources for the Project. The Mineral Resources exclusive of Mineral Reserves, effective September 29, 2023, are summarised in Table 14.15.

Table 14.15 – Mineral Resource Estimate Effective as of September 29, 2023

Resource Classification	Zone	Tonnage	Grade	Contained Gold	Grade	Contained Silver	Grade	Contained Copper
		(Mt)	(g/t Au)	(Moz Au)	(g/t Ag)	(Moz Ag)	(% Cu)	(Mlb Cu)
Measured	High-Grade Main Zone	0.3	4.80	0.05	20.8	0.2	0.23	1.7
Indicated	High-Grade Main Zone	2.7	3.70	0.32	22.7	1.9	0.2	9.8
	Low-Grade Main Zone	2.6	1.93	0.16	22.4	1.9	0.1	8.0
	Low-Grade Lower Zone	0.4	2.32	0.03	13.0	0.2	0.2	2.1
Total Indicated		**5.7**	**2.79**	**0.51**	**21.9**	**4.0**	**0.16**	**19.8**
Measured + Indicated	High-Grade Main Zone	3.0	3.83	0.37	22.5	2.2	0.17	11.5
	Low-Grade Main Zone	2.6	1.93	0.16	22.4	1.9	0.14	8.0
	Low-Grade Lower Zone	0.4	2.32	0.03	13.0	0.2	0.22	2.1
Total Measured + Indicated		**6.0**	**2.90**	**0.56**	**21.8**	**4.2**	**0.16**	**21.5**
Inferred	High-Grade Upper Zone	0.2	7.02	0.04	21.3	0.1	0.55	2.0
	Low-Grade Main Zone	1.7	2.06	0.11	41.2	2.3	0.15	5.6
	Low-Grade Lower Zone	0.3	2.74	0.03	23.7	0.3	0.23	1.7
Total Inferred		**2.2**	**2.54**	**0.18**	**37.0**	**2.7**	**0.19**	**9.3**

Notes:
- Mineral Resources have been estimated by Katharine Masun, M.Sc., MSA, P. Geo., of SLR, an independent Qualified Person as defined by NI 43-101.
- CIM (2014) definitions were followed for Mineral Resources.
- The Mineral Resources have been reported within underground reporting shapes generated with Deswik Stope Optimizer (DSO) using a net smelter return (NSR) cut-off value of US$65/t.
- Mineral Resources are estimated using a long-term gold price of US$1,700 per ounce, silver price of US$20 per ounce, and copper price of US$3.75 per pound.
- Mineral Resources are exclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Average bulk density is 2.7 t/m³.
- Numbers may not sum precisely due to rounding.

14.14 Comparison to Previous Mineral Resource Estimate

The current Loma Larga Mineral Resource Estimate supersedes the previous Mineral Resource Estimate dated March 31, 2020 (DRA, 2021). The current Mineral Resource estimate is substantially reduced in both tonnage and contained metal, with slightly lower average grades. This is primarily the result of reporting Mineral Resources exclusive of Mineral Reserves, along with the following:

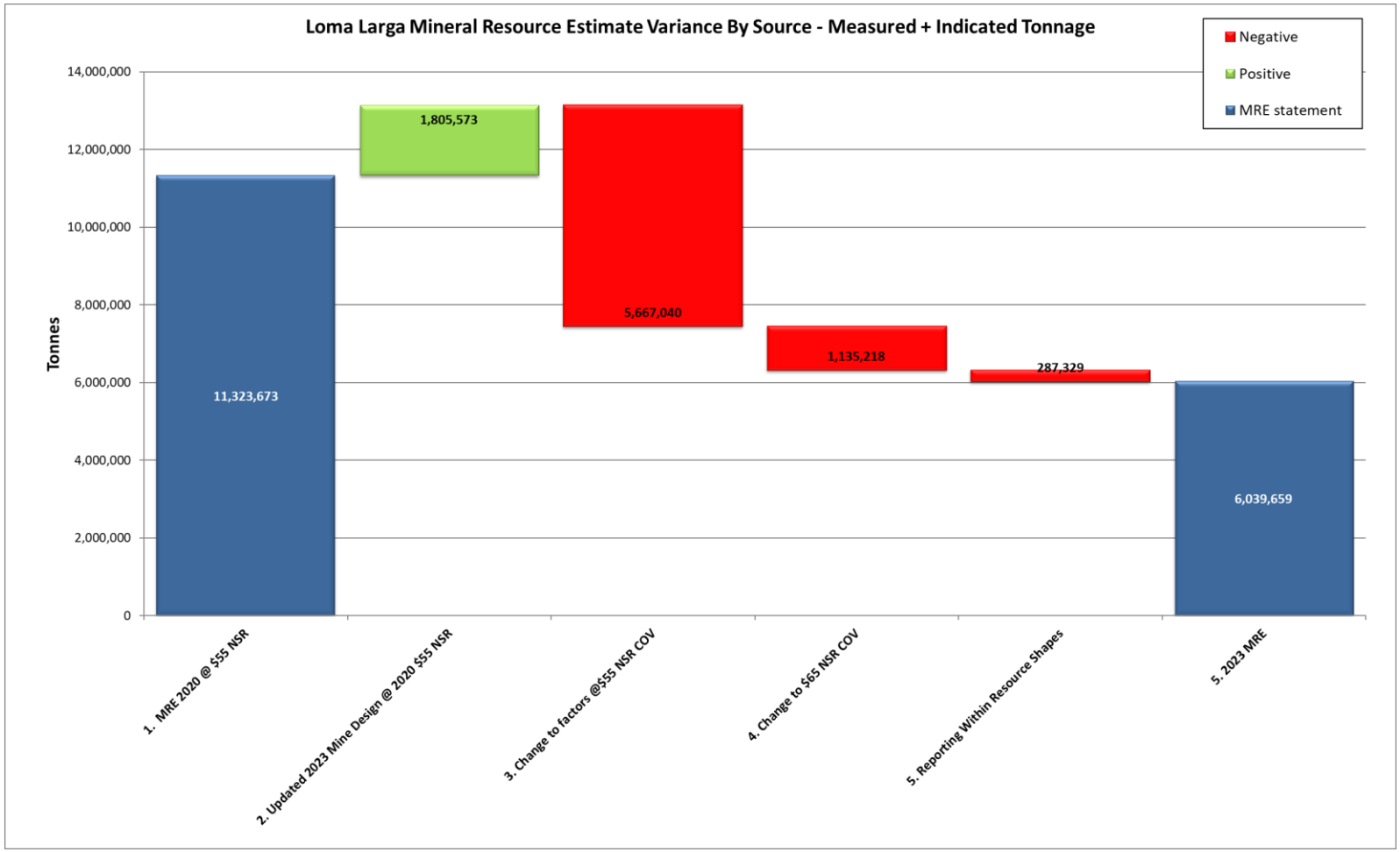
- Updated mine design.
- The economic assumptions incorporated into the current NSR calculation used lower metal prices, higher costs, and lower copper recoveries.
- Reporting resources within underground reporting shapes using a higher NSR cut-off value of US$65 vs. US$55.
- The decrease in grade is the result of reporting all blocks within the underground reporting shapes.
- The low-grades of the marginal blocks within the Low-Grade Main Zone and Low-Grade Lower Zone, which contributed a substantial tonnage to the previous Mineral Resource Estimate, did not meet the economic criteria to be incorporated into the underground reporting shapes at an NSR cut-off value of US$65 and were removed from the current Mineral Resource Estimate.

Comparison of the current Mineral Resource Estimate to the 2020 Mineral Resource Estimate, exclusive of Mineral Reserves, is presented in Table 14.16. Waterfall charts in Figure 14.15 to Figure 14.18 illustrate the key factors contributing to the change between the 2020 and 2023 Mineral Resource Estimates for Loma Larga.

Table 14.16 – Comparison Between the Previous Mineral Resource Estimate (March 31, 2020) and the Current Mineral Resource Estimate (September 29, 2023) Exclusive of Mineral Reserves

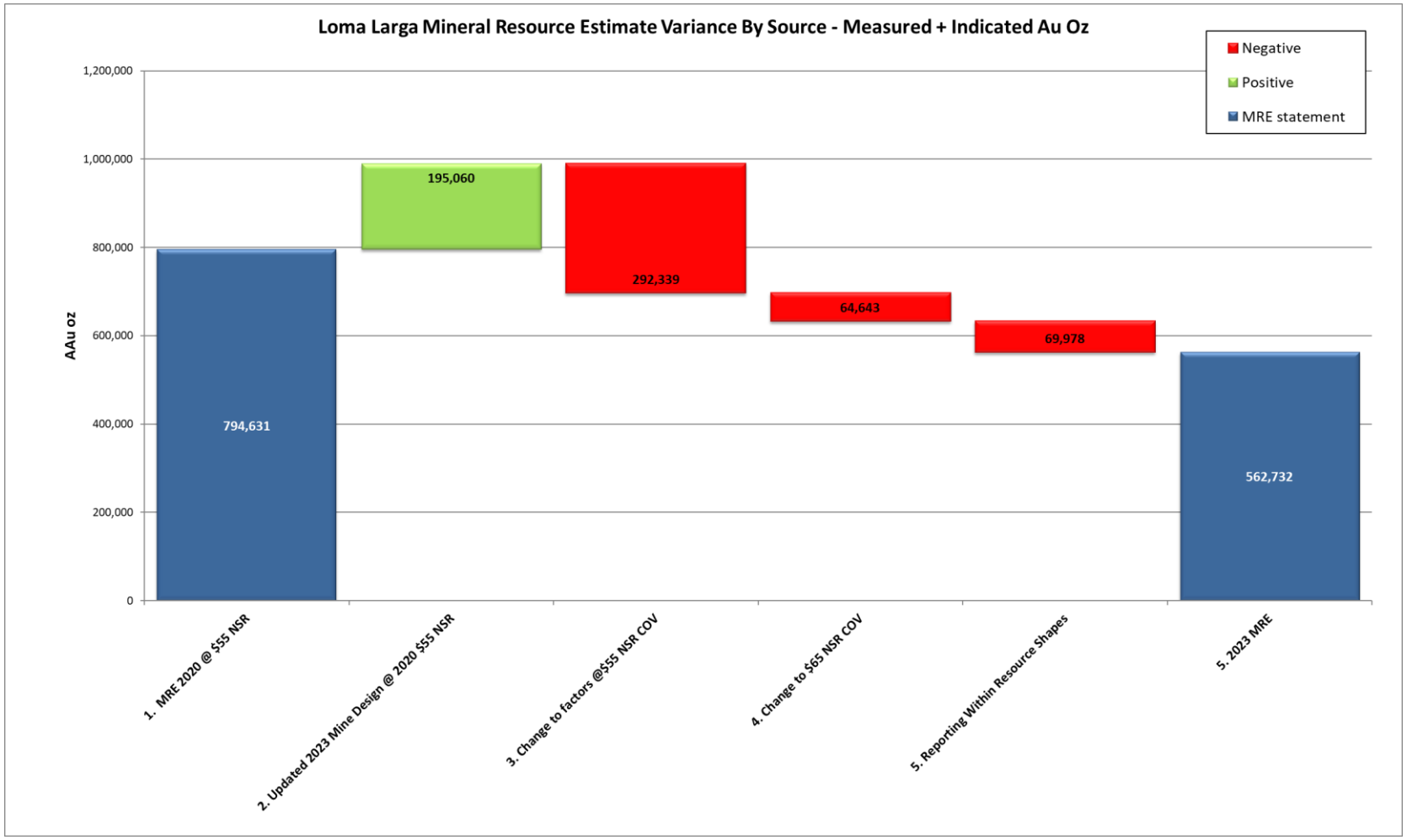
Classification	Tonnes (Mt)		Grade						Metal Content						% Difference						
			Au (g/t)		Ag (g/t)		Cu (%)		Au (Moz)		Ag (Moz)		Cu (Mlb)		Tonnes	Grade			Metal Content		
	2020	2023	2020	2023	2020	2023	2020	2023	2020	2023	2020	2023	2020	2023		Au	Ag	Cu	Au	Ag	Cu
Measured + Indicated	11.3	6.0	2.18	2.90	17.5	21.8	0.13	0.19	0.79	0.56	6.4	4.2	31.2	21.5	-47	33	24	52	-29	-34	-31
Inferred	6.2	2.2	2.03	2.54	25.6	37.0	0.12	0.17	0.40	0.18	5.1	2.7	16.9	9.3	-64	25	45	36	-55	-48	-45

Figure 14.15 – Waterfall Chart of Tonnage Variance between 2020 and 2023 Measured + Indicated Mineral Resources Exclusive of Mineral Reserves



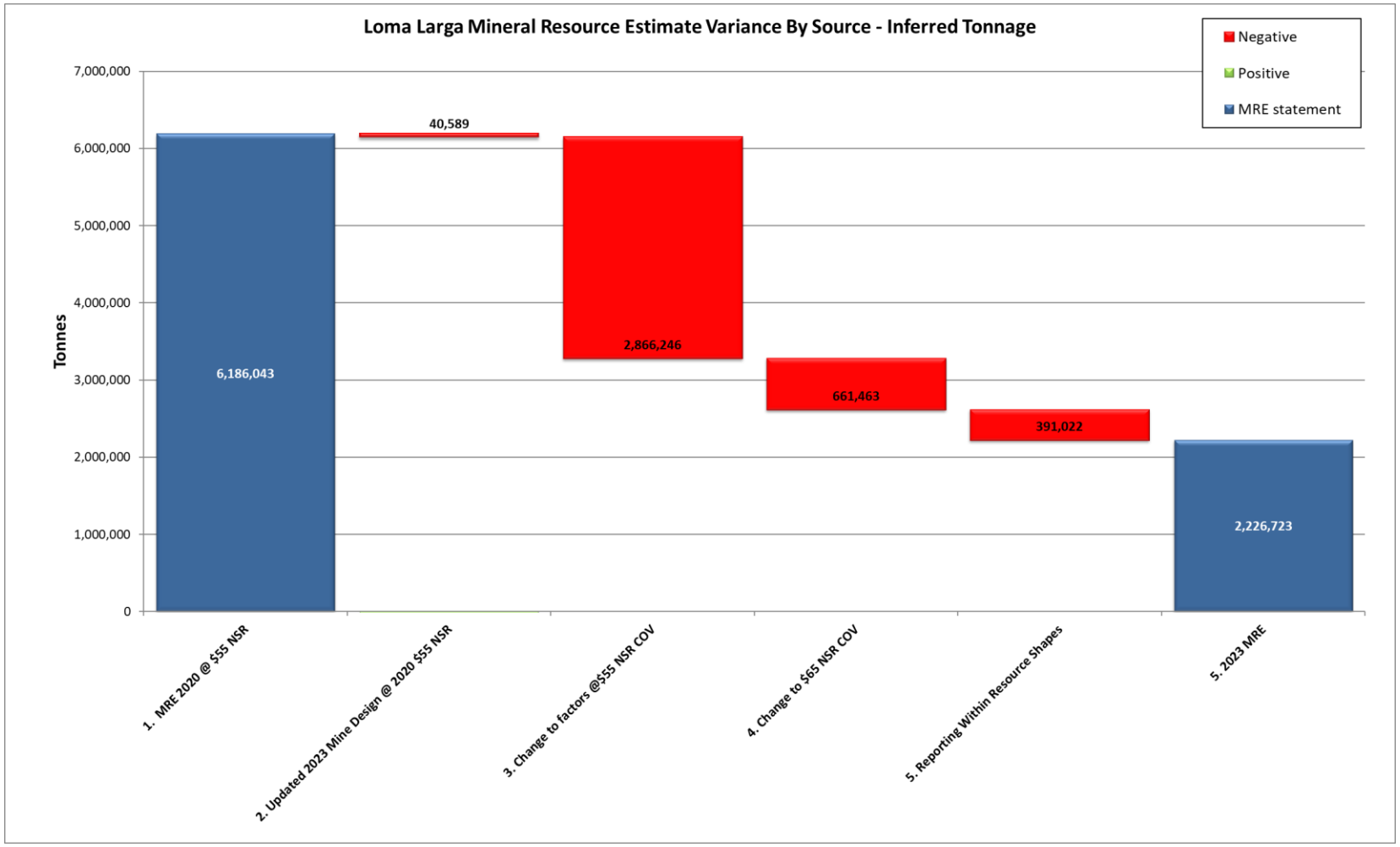
Source: SLR, 2023

Figure 14.16 – Waterfall Chart of Total Contained Gold Variance between 2020 and 2023 Measured + Indicated Mineral Resources Exclusive of Mineral Reserves



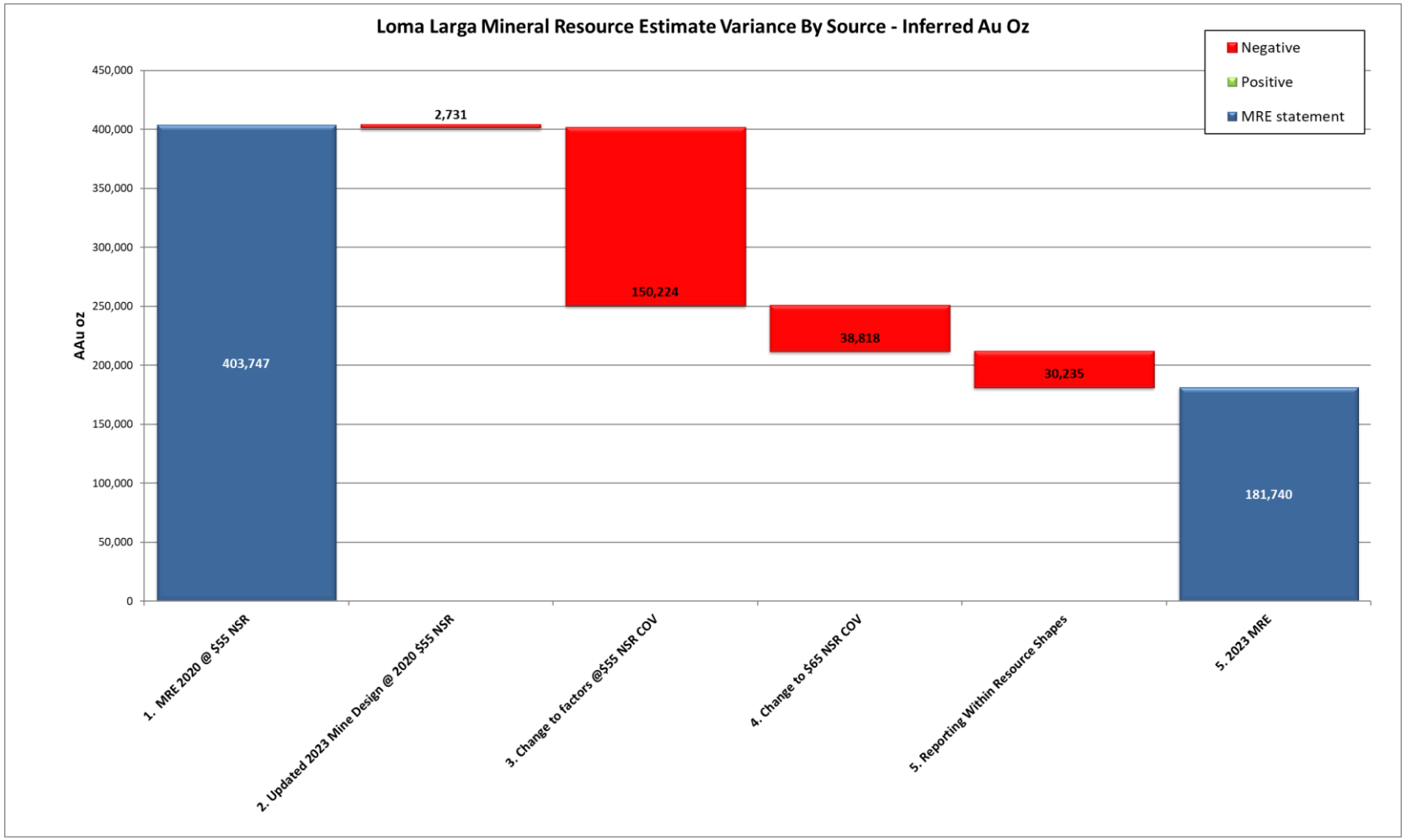
Source: SLR, 2023

Figure 14.17 – Waterfall Chart of Tonnage Variance between 2020 and 2023 Inferred Mineral Resources Exclusive of Mineral Reserves



Source: SLR, 2023

Figure 14.18 – Waterfall Chart of Total Contained Gold Variance between 2020 and 2023 Inferred Mineral Resources Exclusive of Mineral Reserves



Source: SLR, 2023

14.15 Sulphur Model

14.15.1 Sulphur Database

Using data available, the QP generated a block model for sulphur grades to support metallurgical testwork (RPA, 2018). INV provided sulphur assay data for drilling completed in 2017 in addition to sulphur assay data from drillholes completed prior to 2016 by IAMGOLD. Files were received in Excel format, amalgamated, and parsed as required, and imported into GEMS. Table 14.17 summarises data used for the sulphur block grade estimates.

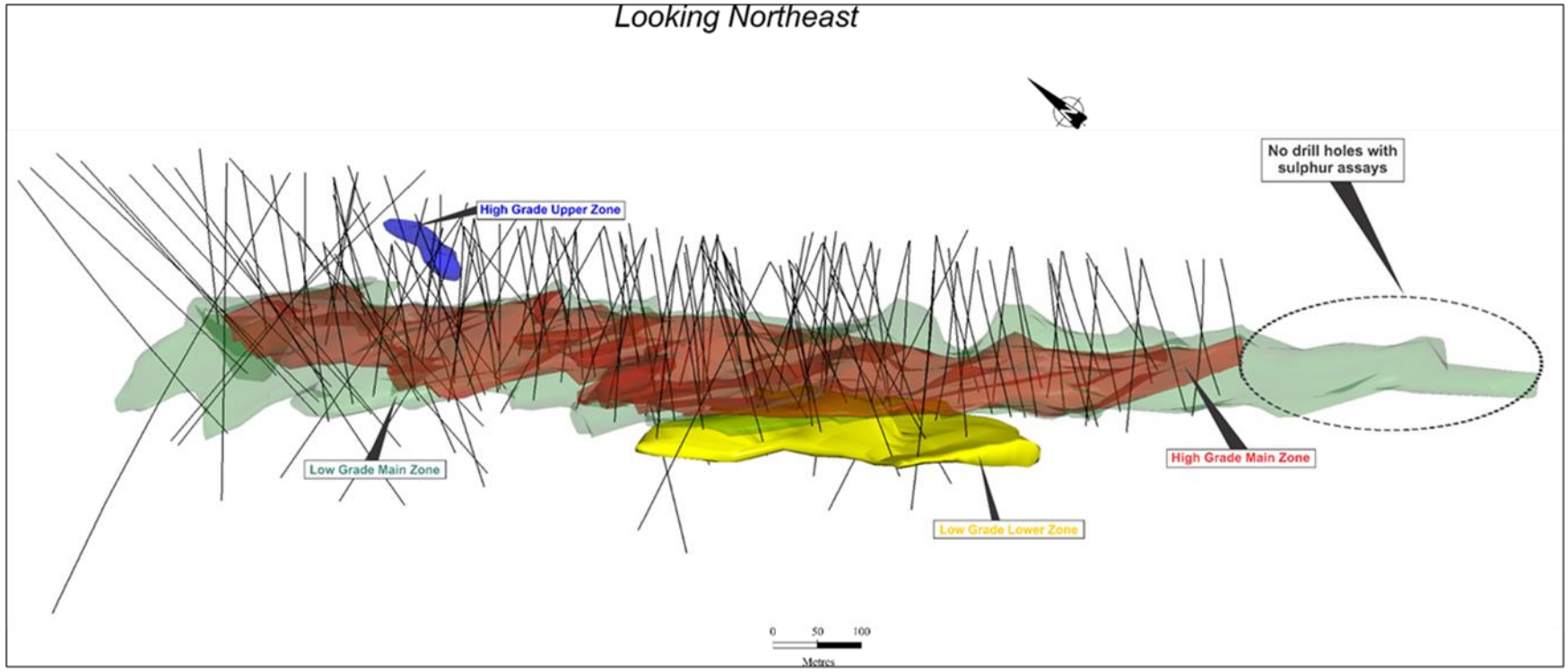
The QP notes that not all drillhole assay samples were analysed for sulphur (see Table 14.2), and there are no samples south of 9,663,100 m North. As a result, a small area of the southern portion of the Low-Grade Main Zone (located in Inferred Mineral Resources outside the current mine plan) does not have sulphur grade estimates and the QP did not assign default values to the blocks.

Table 14.17 – Sulphur Database Used for Block Grade Estimation

Attribute	Count
Holes	199
Total Length Drilled (m)	36,059
Assays	6,890
Composites (>0.5 m)	3,993

The sulphur block model estimate is based on assays from 199 drillholes that totalled 36,059 m in length. Drilling was completed by IAMGOLD and INV between 2005 and 2017. Figure 14.19 illustrates the location of drillholes with sulphur data within the Loma Larga Deposit.

Figure 14.19 – 3D Isometric View of Drillholes with Sulphur Data



Source: SLR, 2020

14.15.2 STATISTICAL ANALYSIS

Sulphur assays located inside the Loma Larga mineralisation wireframe domains were tagged within the GEMS project drillhole database rock codes. The QP exported the sulphur assays for statistical analysis and for later comparison with composite grades and block model grades.

Sulphur assay grades less than the detection limit were replaced with zero values, and sulphur assay grades above the analytical method detection limit were replaced by the upper detection limit (Figure 14.20). Intervals without sulphur assays were treated as unsampled.

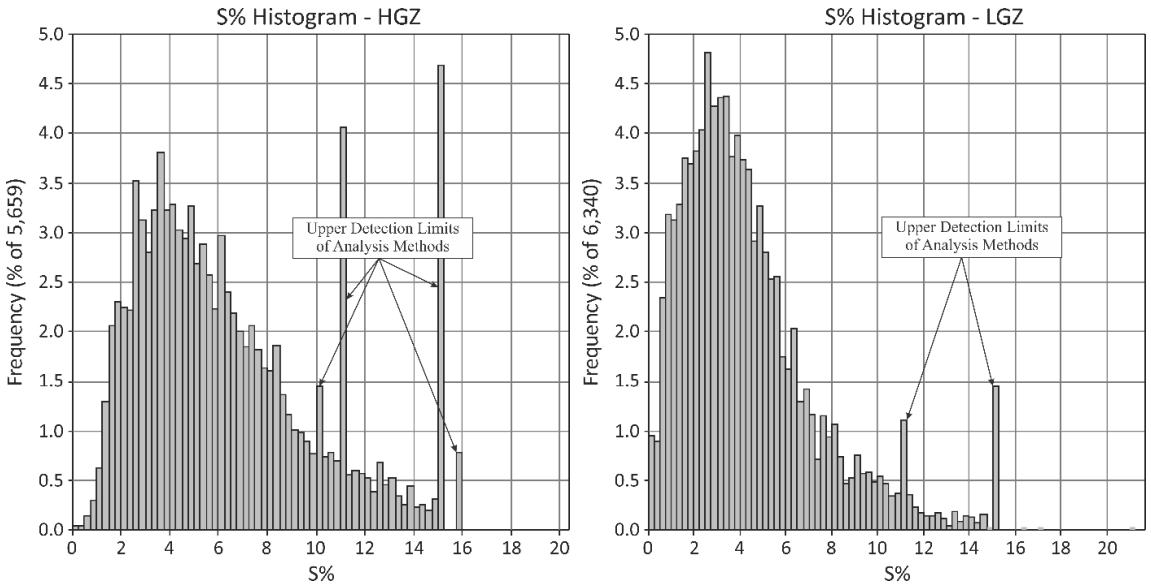
For statistical analysis, the sulphur assays have been subdivided into High-Grade Zone (Main Zone and Upper Zone) and Low-Grade Zone (Main Zone and Lower Zone). The basic statistics are summarised in Table 14.18, and the histograms are illustrated in Figure 14.20. The histogram shows distinct peaks where assays results were above the analytical upper detection limit and not reanalysed. The QP is unaware of the risk that may be associated with the uncertainty of high sulphur assays.

Table 14.18 – Descriptive Statistics of Geochemistry Resource Assay Values

High-Grade Zone	S %
Count	3,458
Minimum	0.18
Maximum	16.00
Median	5.58
Arithmetic Mean	6.47
Standard Deviation	3.74
Coefficient of Variation	0.58

Low-Grade Zone	S %
Count	3,432
Minimum	0.03
Maximum	21.18
Median	3.70
Arithmetic Mean	4.38
Standard Deviation	3.05
Coefficient of Variation	0.70

Figure 14.20 – Assay Histograms for Sulphur



Source: DRA, 2021

14.15.3 Capping of High Values

In order to highlight areas of higher sulphur content, outlier values were not capped and the QP did not perform capping analysis on the sulphur assays. However, many analyses were effectively capped by exceeding the upper detection limit of the analytical method with no subsequent reanalysis.

14.15.4 Compositing

Sulphur assay sample lengths range from 0.15 m to 4.20 m within the wireframe domains. Greater than 99% of the resource assays have lengths less than 2.0 m. The QP used composite lengths of 2.0 m.

A set of drillhole-wireframe intersection intervals were identified for each drillhole within the mineralised wireframe domains and sulphur assays were composited starting at the first composite control interval from the collar and resetting at each new wireframe boundary. Unsampled intervals were treated as unsampled and not used in the composite calculation. Composites measuring less than 0.5 m were removed from the database prior to statistical analysis and block model grade interpolation. The elimination of the small composites did not affect the overall integrity of the composited database.

Table 14.19 summarises statistics of the uncapped sulphur composite assay values.

Table 14.19 – Descriptive Statistics of Sulphur Composite Values

High-Grade Zone	S %
Count	1,908
Minimum	0.76
Maximum	16.00
Median	5.89
Arithmetic Mean	6.50
Standard Deviation	3.37
Coefficient of Variation	0.52

Low-Grade Zone	S %
Count	2,085
Minimum	0.03
Maximum	18.06
Median	3.87
Arithmetic Mean	4.41
Standard Deviation	2.79
Coefficient of Variation	0.63

14.15.5 VARIOGRAPHY AND INTERPOLATION VALUES

The QP reviewed the variography for sulphur and also prepared variograms using composites located within the main mineralised domains. The nugget effect was established with the downhole linear semi variogram and the longest semi-variogram range was oriented parallel to and across strike in the horizontal plane.

Interpolation and search parameters used for sulphur are summarised in Table 14.20.

Table 14.20 – Sulphur Block Estimation Parameters

Parameter	Method	OK
	Boundary Type	Hard
High-Grade Zone Sample Restrictions	Min. No. Comps.	3
	Max. No. Comps.	15
	Max. Comps. Per Drillhole	3
High-Grade Zone Search Ellipse	Range X (m)	160
	Range Y (m)	130
	Range Z (m)	130

Parameter	Method	OK
	Boundary Type	Hard
High-Grade Zone Search Anisotropy[1]	Z	0°
	Y	0°
	Z	+75°
Variogram Model High-Grade Zone	Nugget (C0)	0.05
	Relative Nugget (C0)	5%
Structure High-Grade Zone	C1	0.65
	Range X (m)	24
	Range Y (m)	8
	Range Z (m)	6
	C2	0.30
	Range X (m)	79
	Range Y (m)	63
	Range Z (m)	67
	Total Sill	1.00
Low-Grade Zone Sample Restrictions	Min. No. Comps.	3
	Max. No. Comps.	15
	Max. Comps. Per Drillhole	3
Low-Grade Zone Search Ellipse	Range X (m)	300
	Range Y (m)	215
	Range Z (m)	165
Low-Grade Zone Search Anisotropy[1]	Z	0°
	Y	0°
	Z	-40°
Variogram Model Low-Grade Zone	Nugget (C0)	0.05
	Relative Nugget (C0)	5%
Structure Low-Grade Zone	C1	0.52
	Range X (m)	72
	Range Y (m)	102
	Range Z (m)	13
	C2	0.43
	Range X (m)	203
	Range Y (m)	144
	Range Z (m)	111
	Total Sill	1.00

[1] Rotation around each axis (positive is counterclockwise).

Block grade interpolation was carried out in a single pass using OK. Interpolation was restricted by the mineralised wireframe models, which were used as hard boundaries to prevent the use of composites outside of the zones. A minimum of three (3) 2.0 m composites and a maximum of 15 composites were used to interpolate grades within each block for sulphur, with a maximum of three (3) samples per drillhole.

14.15.6 Block Model

A sulphur block model was added to the Loma Larga GEMS project. Table 14.21 summarises the statistics of the uncapped sulphur block grades.

Table 14.21 – Descriptive Statistics of Sulphur Block Values

High-Grade Zone	S %	Low-Grade Zone[1]	S %
Count	27,913	Count	58,784
Minimum	1.89	Minimum	0.50
Maximum	14.00	Maximum	14.43
Median	6.54	Median	4.38
Arithmetic Mean	6.71	Arithmetic Mean	4.71
Standard Deviation	1.72	Standard Deviation	1.91
Coefficient of Variation	0.26	Coefficient of Variation	0.41
% of Interpolated Blocks	100%	% of Interpolated Blocks	98.4%

14.15.7 Block Model Validation

The QP carried out a number of block model validation procedures including:

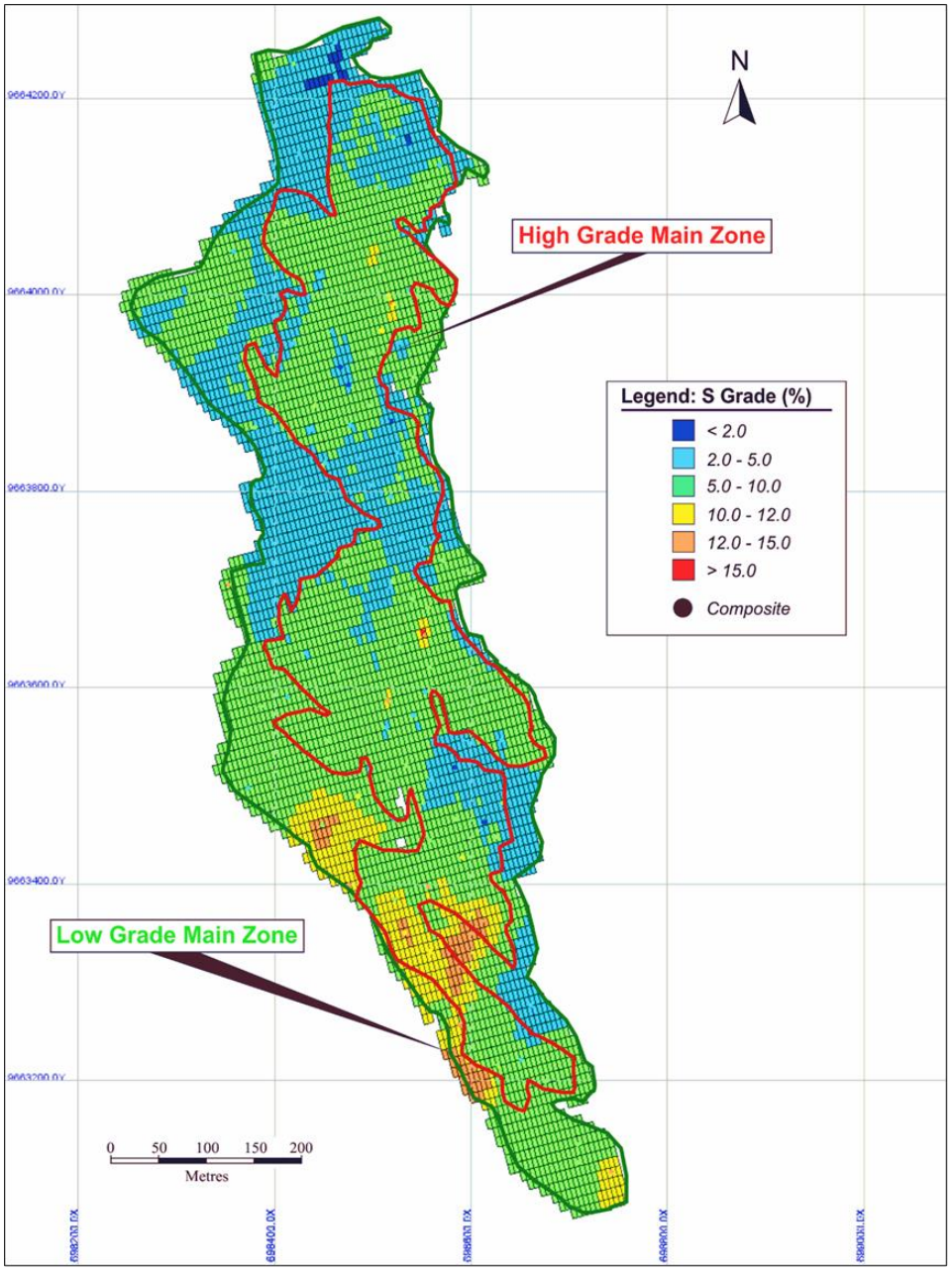
- Visual comparisons of block sulphur grades versus composite grades.
- Statistical comparisons of assays, composites, and blocks on plan and vertical section.
- Trend plots of block and composite sulphur grades by elevation and northings / eastings.

Block model sulphur grades were visually examined and compared with composite grades in cross section and on elevation plans. The QP found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drillhole assay and composite grades and that there was no significant bias. The QP notes that the mean block grades were slightly higher than mean composite grades. This result is expected since capping was not performed, and higher sulphur assays had a larger influence on block grades, especially in areas where drillhole spacing was wide.

Figure 14.21 shows sulphur composite and block grades in plan view.

In the QP's opinion, the sulphur block grade estimates are suitable to support metallurgical testwork for this Technical Report, but cautions that local block sulphur grades are not as well supported as the payable metal block grades in the Mineral Resource Estimate. This includes areas of the block model without data and incomplete or lacking information on the analytical methods used for sulphur assaying.

Figure 14.21 – Sulphur Composites and Blocks on 3,600 m Level



Source: SLR. 2020

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Mineral Reserve Estimate for Loma Larga Project consists of selected portions of the indicated and measured resource that are above a US $75.00/t Net Smelter Return (NSR) cut-off value. This value was applied when generating the stoping shapes. For the development in ore, a US $55.00/t Net Smelter Return (NSR) cut-off value has been used. The mineral reserves for Loma Larga are estimated at 12.59 Mt of recoverable and diluted ore grading 4.70 g/t Au, 28.56 g/t Ag, and 0.29% Cu. The Mineral Reserves comprise 22% in proven category (2.72 Mt grading 6.83 g/t Au, 33.23 g/t Ag and 0.43% Cu) and 78% in probable category (9.86 Mt grading 4.12 g/t Au, 27.35 g/t Ag and 0.25% Cu).

Mineral Reserves are inclusive of dilution and ore loss. GMD added the latest NSR value to the block model supplied by DPM using Datamine software. GMD conducted further work in Deswik software. GMD does not believe that the estimate of Mineral Reserves may be materially affected by metallurgical, permit application, legal, title, taxation, socio-economic, marketing, or political issues. However, GMD relies on information as presented in this Technical Report in relation to metallurgical, infrastructure, environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors.

The Mineral Reserves are based on the mining schedule presented in Section 16. The Mineral Reserves are the stope and development tonnage (including material above the cut-off and internal low-grade dilution) to which dilution and recovery factors (as explained below) are added.

Details of the mineral reserves are given in Table 15.1.

Table 15.1 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023 (GMD, 2023)

Ore Category	Tonne (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)	Cu Grade (%)	Contained Cu (Mlb)
Proven	2.72	6.83	0.60	33.23	2.91	0.43	25.58
Probable	9.86	4.12	1.31	27.35	8.67	0.25	54.39
Proven and Probable	**12.59**	**4.70**	**1.90**	**28.56**	**11.58**	**0.29**	**79.96**

1. Mineral Reserve is estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (Nov 29, 2019) and CIM Definition Standards for Mineral Resources & Mineral Reserves, May 19, 2014.
2. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the crusher.
3. Mineral Reserves account for unplanned mining dilution and ore loss that varies by orebody dimension and geotechnical properties per mining block area.
4. Mineral Reserves include long hole stopes and development in ore.
5. Mineral Reserves are reported at an NSR cut-off value of US$75/t for long hole stopes and US$55/t for development in ore.
6. Mineral Reserves are estimated using average gold price of US$1,500 per ounce, silver price of US$17.00 per ounce, and copper price of US$3.25 per pound.
7. Average bulk density is 2.71 t/m^3.
8. Numbers may not sum precisely due to rounding.
9. Metallurgical recoveries are not included in the contained values calculation.
10. The mine design and schedule for the Mineral Reserve Estimation was completed under the supervision of Dr. Lewis Meyer, CEng, QMR, FIMMM Associate Principal Engineer at Global Mine Design Ltd., who is a QP as defined under NI 43-101.
11. Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

15.2 Factors That May Affect the Mineral Reserve Estimate

The underground mine design and ore reserve estimate have been prepared by GMD to a level appropriate for a Feasibility Study. The Mineral Reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources with applicable modifying factors and therefore exclude any Inferred Mineral Resources. The latter are classified as waste for reporting purposes. The following items are factors that may affect the mineral reserves:

- Geological interpretation and mineral resource block modelling.
- Cut-off value estimations.
- Commodity prices and market conditions assumptions.
- Operating costs, sustainable costs for development assumptions.
- Geotechnical constraints, dilution, mining recovery factors and hydrogeological assumptions.
- Metallurgical process recoveries.
- Environmental permitting.

Net changes in tonnes and contained metals from the 2020 to the 2023 Mineral Reserves Estimate show a decrease of 1,341,4322 in tonnage (1.3 Mt), a reduction of 295,910 ounces of gold (0.3 Moz Au), a reduction of 1,690,402 ounces of silver (1.7 Moz Ag) and a reduction of 8.08 million pounds of copper (8.08 Mlb Cu). The corresponding percentage changes are a 9.6% decrease in tonnes, and 13.5% reduction in metal content gold, 12.7% in metal content silver and 9.2% in metal contained copper. The decrease is attributed to the changes in the economic assumptions incorporated into the current NSR calculation such as higher costs, and lower copper recoveries, as well as the higher NSR cut-off value of US$75/t vs. US$60/t.

Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. The QP understands that DPM is considering all its options to rectify the revocation of the environmental license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project, including the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

15.3 Stope Shape Design

The Deswik Stope Optimizer (DSO) software was used to produce the mineable portion of the Mineral Resource. The Vertical method was used within the DSO parameters to produce the transverse and longitudinal stope shapes with the applicable cut-off value.

Transverse stopes were generated first across the whole orebody. The DSO parameters used to prepare the stope shapes are provided in Table 15.2.

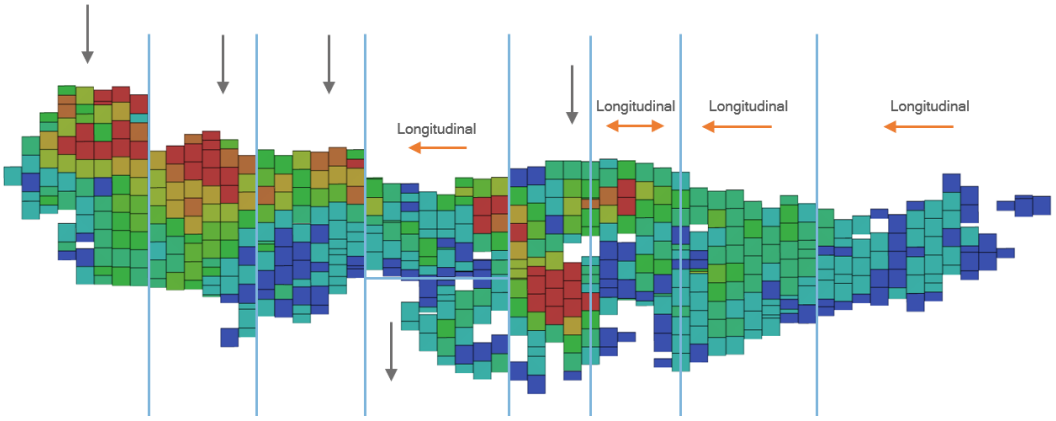
Table 15.2 – DSO Parameter

DSO Parameter	Value
Optimisation Method	Vertical
Orientation of DSO (Z axis rotation)	-15º
Stope Width Along Strike	20 m
Minimum Stope Width Hanging Wall to Foot wall	10 m
Maximum Stope Width Hanging Wall to Foot wall	20 m
Minimum Stope Dip Angle	70º
Minimum Stope Height	10 m
Maximum Stope Height	20 m
Minimum Stope Length	10 m
Maximum Stope Length	20 m

Stope 'runs' were all created with sub-shapes at 5 m increments between 20 m and 10 m. Additionally, a separate 10 m height and 10 m length run was produced. These two (2) DSO settings were re-run at an elevation 5 m higher, so that the optimal stope positions could be determined. The four (4) sets of DSO results were then observed for each panel with vertical slices of the block model to determine the best fit for the orebody. Stopes were combined where possible to make the largest mineable stopes. The maximum height was 40 m, and the maximum length was 30 m.

The transverse stopes enabled the revised mine layout to be observed. Combined with the updated information on structures in the geotechnical model, certain areas appeared better suited to longitudinal accesses. Therefore, additional DSO runs investigated options for including longitudinal stopes into the design. These areas are shown in Figure 15.1. The legend indicates grade, where red is the highest range and dark blue is the lowest range.

Figure 15.1 – Transverse Stopes with Areas Marked for Longitudinal Access Investigation



Source: Global Mine Design, 2023

Longitudinal stopes were generated in these marked areas using the same four (4) sets of DSO runs as the transverse stopes, with the x and y settings swapped to generate the longitudinal direction. The two (2) sets of stope shapes (transverse and longitudinal) were then combined to create the final stope layout.

Incorporating both transverse and longitudinal stopes provided an opportunity to reduce development metres in waste in the lower grade areas of the orebody. The haulage drifts along the footwall of the orebody and the ventilation drives in the hangingwall provide the ability to reinstate transverse accesses into longitudinal stope areas with minimal delay, should the need arise later in the mine life. With this contingency, the combined transverse and longitudinal layout is a robust and versatile mine design.

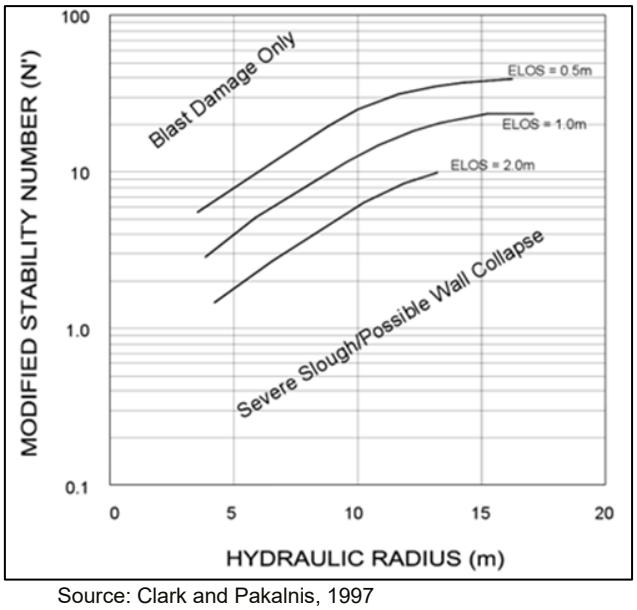
15.4 Dilution

15.4.1 Stability Graph Method

The original stability method for open stope design was developed by Mathews et al. (1981), later revised by Potvin (1988), Nickson (1992) and others, in order to provide guidance in the determination of stope sizes for use in mine design. This empirical stability graph method was used to determine preliminary stope sizing guidelines for stope faces exposed in each of the defined geotechnical domains. However, stability can only be qualitatively assessed as either being stable, potentially unstable or caved.

An alternative to the qualitative stability graph was devised by Clark and Pakalnis (1997), introducing the parameter termed the Equivalent Linear Overbreak/Slough (ELOS) to express volumetric measurements of overbreak an average depth over an entire stope surface, resulting in the design curve in Figure 15.2. The stability graph relates hydraulic radius of stope walls to empirical estimates of overbreak based on a Modified Stability Number, N, and is an indirect quantitative measure of dilution.

Figure 15.2 – ELOS Stability Chart



Source: Clark and Pakalnis, 1997

15.4.2 Loma Larga Dilution Evaluation

All available geotechnical data have been reviewed and re-characterised according to geotechnical domains comprising both alteration and lithology types. More information about rock quality characterisation and definition of geotechnical domains is provided in Section 16 of this Report. Stope overbreak analysis has been carried out corresponding to geotechnical domains. Stopes occur in five (5) domains; the number of stopes present in each domain summarised in Figure 15.3. The colour legend applied has been used throughout the Project to reference geotechnical domains (Refer to Section 16.3 for more details).

Figure 15.3 – Stopes by Geotechnical Domain



GMD Domain	Total number of stopes at $75/t
T (Si)	498
T (AA)	45
T (A-S)	8
L-P (Si)	64
L-P (AA)	5
Total	620

Source: Global Mine Design, 2023

The ELOS stability graph has been used to predict unplanned overbreak (dilution) which occurs because of instability within a stope. The dilution of each stope surface has been determined separately (e.g., east/west wall, north/south wall, or stope back), and the results from the ELOS graph provide the grade shell offset distance range of stope dimensions per domain, summarised in Table 15.3. It is important to note that the ELOS values assume that stope walls are unsupported; however, excessive overbreak would be dealt with during operations, either through engineering design and/or use of ground support.

Table 15.3 – Summary of Overbreak Results from ELOS Graphs

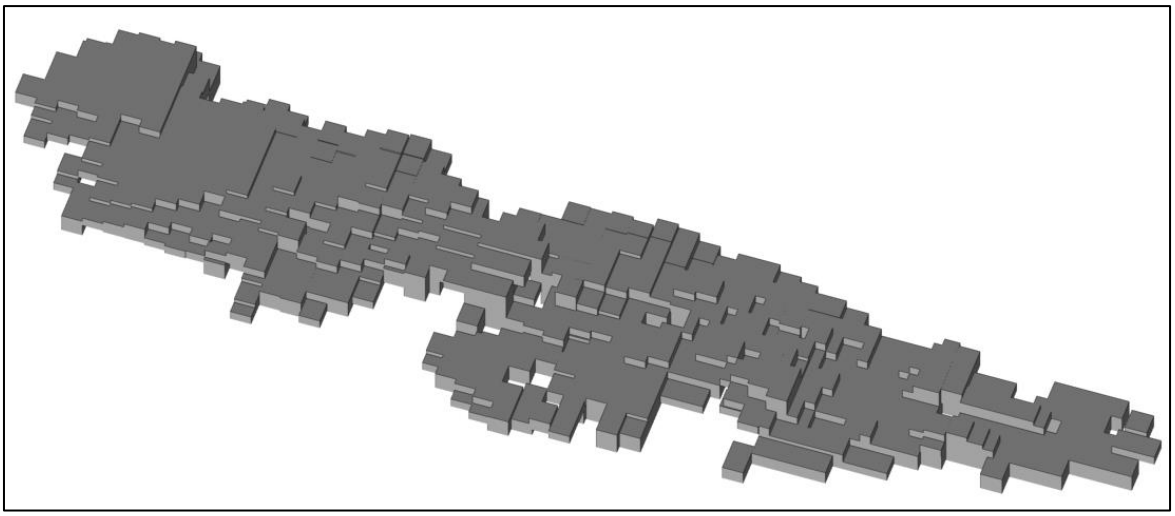
Domain	Overbreak from ELOS Graphs			
	West Orebody Contact	Inside Orebody	East Orebody Contact	Single Stopes with East & West Contacts
T(Si)	0.5 m in all walls except: 1 m overbreak in stope backs when HR = 6	0.5 m in all walls except: 1 m overbreak in stope backs	0.5 m	0.5 m in all walls except: 1 m overbreak in stope backs when HR = 6
T(AA)	0.5 m in all walls except: 1 m overbreak in stope backs	0.5 m in all walls except: 1 m overbreak in stope backs	0.5 m in all walls except: 1 m overbreak in stope backs when HR =5 2 m overbreak in stope backs when HR=6	0.5 m in all walls except: 1 m overbreak in stope backs
T(A-S)		East/west wall overbreak = 1 m North/south wall overbreak = 0.5 m Stope back overbreak = 2.5 m	2 m in all walls except: 1 m overbreak in stope walls when HR >3.3 2.5 m overbreak in stope backs	N/A
L-P(Si)	0.5 m	0.5 m	0.5 m	0.5 m
L-P(AA)	N/A	N/A	0.5 m in all walls except: 1 m overbreak in stope back	0.5 m in all walls except: 1 m overbreak in stope backs

Source: Global Mine Design, 2023

15.4.3 External Dilution

External dilution was estimated by building a grade shell around the final stopes. It was designed manually by offsetting polylines around each strip of stopes using ELOS values. The final 3D solid, as shown in Figure 15.4 was evaluated for tonnes and grades against the block model using Deswik software, the difference between stopes and the grade shell resulted in 738,718 tonnes or 6.1% of the Mineral Reserves.

Figure 15.4 – Grade Shell

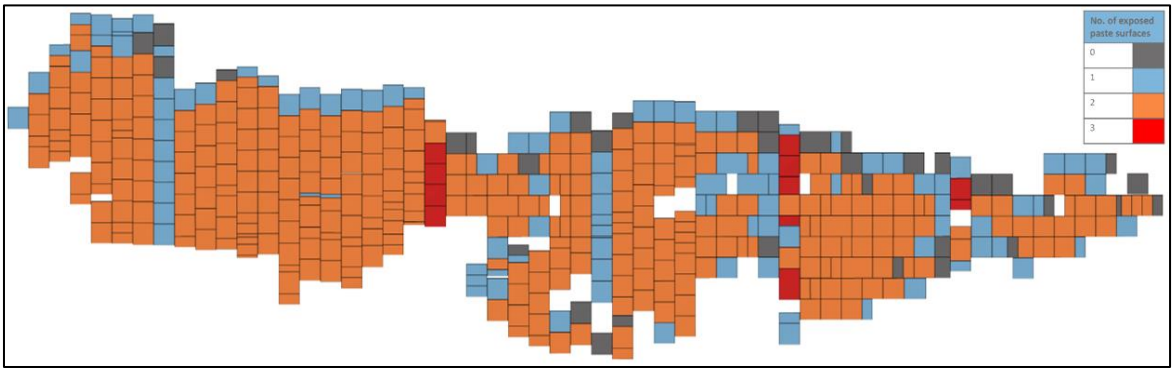


Source: Global Mine Design, 2023

15.4.4 PASTE DILUTION

Dilution from pastefill exposed during stope blasting was estimated by GMD using a nominal 0.5 m Depth of Failure distance per exposed pastefill surface. Exposed pastefill surfaces were determined from the mine sequence, as shown in Figure 15.5.

Figure 15.5 – Number of Exposed Pastefill Surfaces per Stope



Source: Global Mine Design, 2023

GMD calculated volume and tonnes of pastefill dilution for each exposed side wall, end wall and roof (back), as appropriate. The resulting 350,858 tonnes was a proportion of total stope rock tonnes to paste tonnes, equivalent to 2.9% of the Mineral Reserves.

15.5 Ore Recovery

Ore recovery (or mining loss) is based on the material in the model that is left behind for various reasons such as blasting, operational challenges or rock mechanics issues. The estimated mining recovery for Loma Larga long hole stopes is 95% whereas the development recovery is set to 100%. The overall ore loss is estimated at 4.8%.

15.6 Cut-off Value

The mineral reserves economic cut-off value (COV) for Loma Larga was estimated by DPM at US$75/t NSR for the stopes and US$55/t for the lateral development. Based on the data gathered until September 2023, it was decided to proceed with these operating cost assumptions, which became the NSR cut-off value for the mine design. The outputs from the latest economic model, as described in Section 22, varies from these initial assumptions.

An NSR value was estimated for each block of the block model allowing for metallurgical recovery and consideration of smelter terms, including payables, treatment charges, refining charges, price participation, penalties, smelter losses, transportation, and sales charges. The value selected as the cut-off for the stopes is the total cost per tonne of ore including mining, processing, G&A, sustaining and closure costs. The value for the lateral development includes only the processing, G&A, sustaining and closure costs.

Details of the NSR parameters used to calculate the value for each block are presented in Table 15.4.

Table 15.4 – Loma Larga NSR Cut-Off Parameters

Parameter	Unit	Value	
		Copper Concentrate	Pyrite Concentrate
Au Price	US$/oz	1,500	
Ag Price	US$/oz	17	
Cu Price	US$/lb	3.25	
Au Recovery	%	15	75.2
Ag Recovery	%	50	45.7
Cu Recovery	%	92[1]	0
Au Payable	%	96	94.3[2]
Ag Payable	%	90	88.9[3]
Cu Payable	%	88.6[4]	0
As Penalties	US$	calculated[5]	None
Base Treatment Charges	US$/dmt	100	200

Parameter	Unit	Value	
		Copper Concentrate	Pyrite Concentrate
Transport[6]	US$/wmt	50	30
Port Costs	US$/wmt	7	7
Ocean freight	US$/wmt	50	50
Weight & Sampling Department, Reps	US$/wmt	5	5
Au RC	US$/oz	5	5
Ag RC	US$/oz	0.5	0.5
Cu RC	US$/lb	0.01	0
Treatment and Refining Charges			
Mining	US$/t ore	25.86	
Process	US$/t ore	22.04	
Tailings management & Backfill plant	US$/t ore	7.68	
G & A Costs	US$/t ore	10.07	
Sustaining Costs	US$/t ore	7.36	

[1] Formula used: If (Cu (%) > 0.1, Cu (%) < 1.0), value is 5 x LN [Cu (%) + 92] and if Cu (%) ≥ 1.0, value is 92.
[2] Formula used: 97/Au Grade x (Au Grade –1).
[3] Formula used: 95/Ag Grade x (Ag Grade –30).
[4] Value is the minimum between 0.965 and ((Cu Conc. Grade-1) / (Cu Conc. Grade)) x 100.
[5] Formula used: ((As Grade-0.3)/0.1) x 5.0.
[6] These costs are not all inclusive as there was an oversight of US$6/wmt, on container rental. However, these are captured correctly in the financial model, described in Section 22.

15.7 Reconciliation

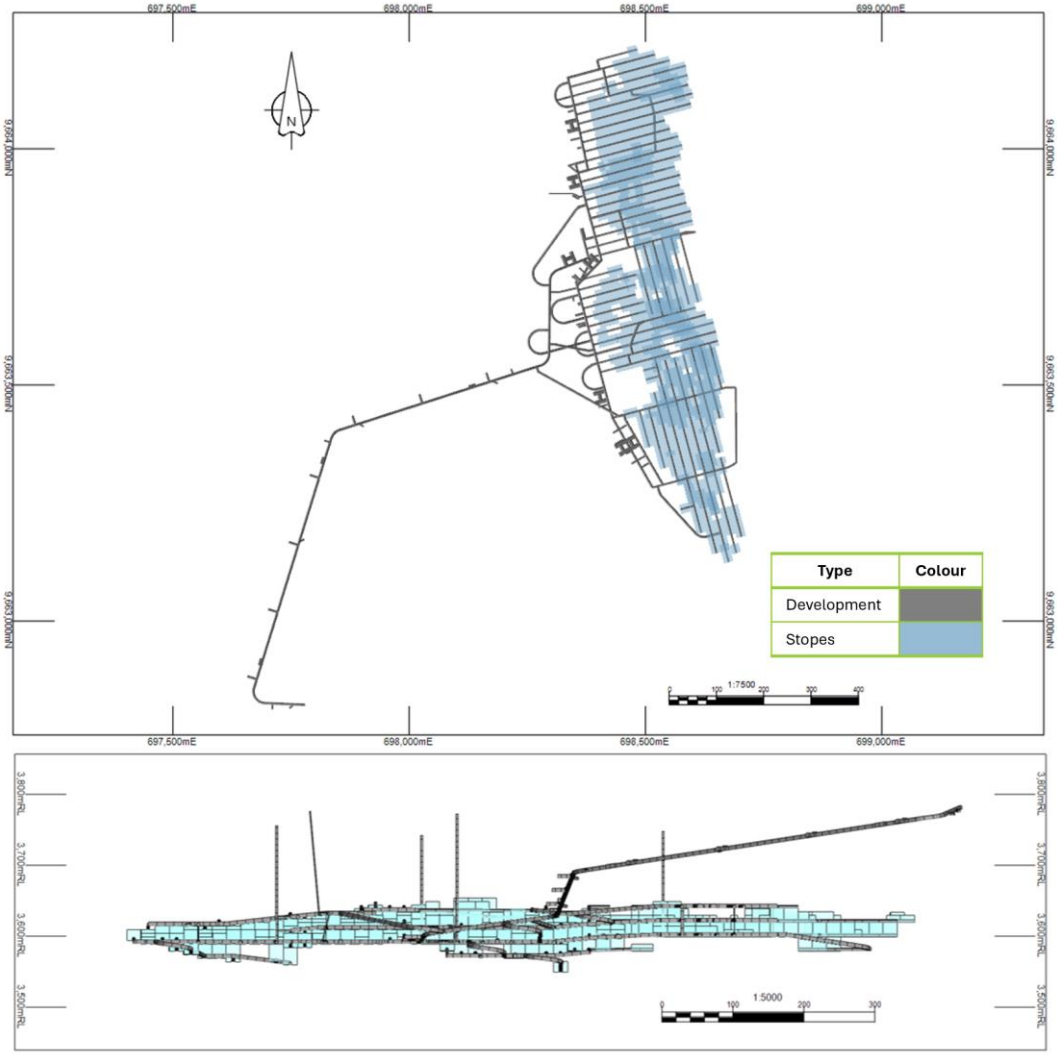
No reconciliation has been undertaken at this stage in the Project. The mine is a greenfield site and development has not yet been commenced.

16 MINING METHODS

16.1 General Description

The proposed Loma Larga underground operation will be accessed via a surface portal and main decline to the orebody. Access to production levels is provided through the main decline, which leads on to internal ramps to production levels. There are a total of four (4) ramps connected to the decline and five (5) inter-level ramps linking to five (5) production levels in total (two (2) main levels and three (3) smaller levels). The main decline is 1,460 m long and accesses the lower main level. The other main level is accessed via a ramp from the decline (120 m long). Typical vertical spacing between levels is 20 m. Figure 16.1 shows the general underground mine design.

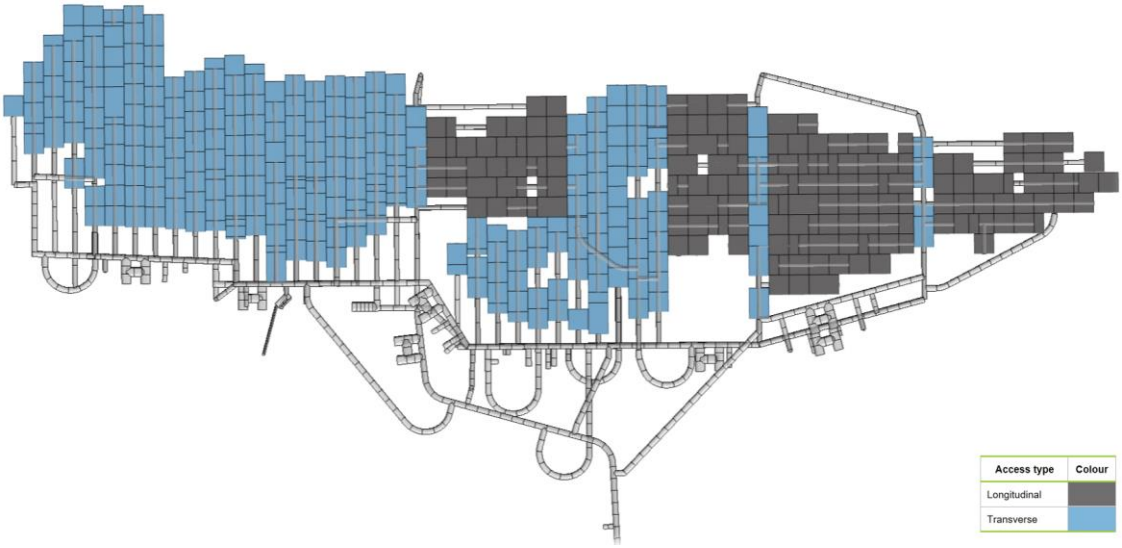
Figure 16.1 – Underground Mine Overview



Source: Global Mine Design, 2023

The mining method selected for the Loma Larga mine is fully mechanised, long-hole open stoping. Most of the stopes are mined overhand (bottom-up) in a continuous mining front; however, there are a small number of stopes to be mined under backfill for optimum scheduling. Based on the grade, geometry, and continuity of the orebody as well as development access efficiency, there is a combination of stope accesses: 60% of the stopes have transverse access and 40% of the stopes have longitudinal access. The orebody layout is shown in Figure 16.2.

Figure 16.2 – Stope Access Types



Source: Global Mine Design, 2023

Most of the recovered ore tonnage will be produced from stoping, with a smaller portion from lateral development (mostly ore drifts) required to access the production zones. Table 16.1 shows the ore tonnage by activity type.

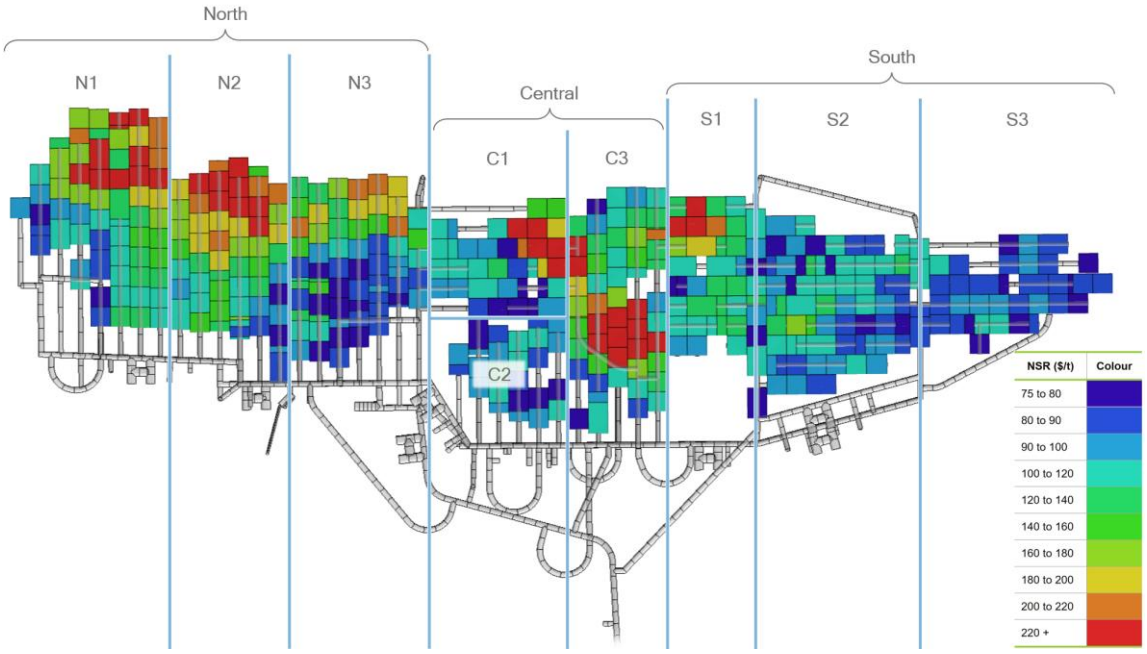
Table 16.1 – Ore Tonnage by Activity Type

Source of Ore tonnes	Ore tonnes (t)	% of Total
Stope Production	11,902,200	94.6
Horizontal Tonnes (Opex)	654,400	5.2
Horizontal Tonnes (Capex)	28,460	0.2
Total Recovered Ore Tonnes	**12,585,060**	**100.0**

The orebody was split into three (3) main zones: North, Central, and South. Each orebody zone was split further into three (3) mining zones as shown in Figure 16.3. Dividing the orebody in this manner enables scheduling to focus on mining high-grade material as soon as possible. The mine design

combines transverse ore development for maximum sequence flexibility with longitudinal access where geotechnically feasible to minimise development in waste. Footwall development has been placed as close as possible to the orebody, while maintaining geotechnical rules relating to sequencing and pillar offset.

Figure 16.3 – Mining Zones Used for Design



Source: Global Mine Design, 2023

16.2 Mining Method

16.2.1 DEVELOPMENT

All development will be formed by twin-boom jumbo drills. These drills are versatile and can be used for both development advance and ground support installation. Long Hole Stoping - Transverse and Longitudinal Accesses

The selected mining method is long-hole open stoping, which is a method well suited to the orebody geometry and is known for its flexibility and ability to minimise underground development. Once the decline reaches the orebody, five (5) 5.5 m wide by 6.0 m high main haulage drifts will be developed along the strike length of the orebody. The average vertical separation is 20 m. The mining zones are accessed from the haulage drifts by a series of 4.5 m wide by 4.5 m high ore drives. Most ore drives are driven at 20 m lateral intervals. The azimuth of the ore drives depends on the stope access type, with transverse accesses orientated perpendicular to the orebody strike and longitudinal accesses orientated parallel to the orebody strike. Development within the orebody will be sampled to update grade control block models.

The mining cycle within a single longitudinal or transverse mining zone generally starts with the furthest stope from the haulage drive, retreating to the access in a bottom to top sequence. However, there are a small number of stopes to be mined under backfill for optimum scheduling. The stope preparation sequence involves the installation of all services required to begin production, as well as marking all the survey reference lines on the walls for drilling, followed by raise boring an initial opening raise (slot) using a small raiseboring head. Production drilling is then carried out from the overcut, in the centre of the stope. Production drillholes are sampled to update grade control block models. The blasting sequence will generally involve multiple firings; the first one will increase the initial void according to the available volume for broken material. Production blast holes will be 102mm (4”) in diameter. Mucking will be performed using 17 tonne Load Haul Dump (LHD) machines remotely controlled. Ore material will be transported to the nearest stockpile (re-muck). The LHDs will transfer ore from the stockpile into 51-tonne trucks for transport to the surface ore stockpile.

Stopes were designed using a two-pass method. The first pass design was done using the Mining Stope Optimizer module in Deswik software (DSO) to automatically design longhole stopes based on the parameters described in Section 15. The second pass design consisted in manual refinement of these stopes, in Deswik to reduce dilution and increase ore recovery. The typical stope has a width of 20 m, a length of 20 m, a height of 20 m and contains circa 21,700 ore tonnes. The variation in stope wall dimensions in any direction ranges between 10 m and 40 m.

Where stopes are defined by an overcut access and an undercut access, the overcut access will serve as the drilling horizon and the undercut will serve as the mucking horizon (downhole stope). Where stopes are defined by only an undercut access, the undercut will serve as both drilling horizon and mucking horizon (uphole stope).

16.2.2 Drill and Blast Design

The long-hole transverse and longitudinal method chosen for the Loma Larga underground mine makes use of fan drilling to maximise ore recovery. Slot raise drilling will be performed with a V30 reaming head on an ITH drill. Most stopes are less than 30 m height therefore production drilling will be performed by a top hammer drill. Provision is allowed in the mine design for occasional stopes that exceed 30 m in height to be drilled with the ITH rig.

Table 16.2 summarises the drilling parameters used.

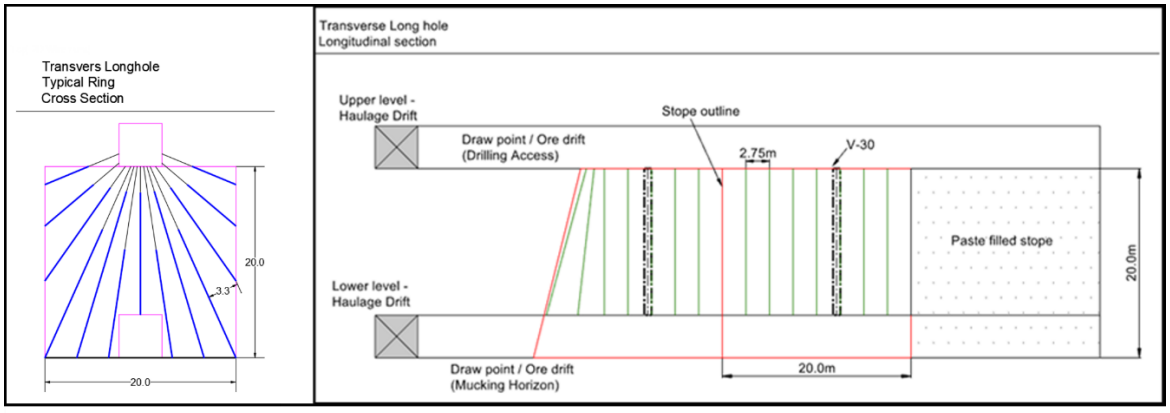
Table 16.2 – Drilling Parameters

Parameters	Unit	Stope Height	
		[10 m – 30 m]	[30 m – 40 m]
Model	Sandvik	DL422i	DU412i
Drill Type		Top Hammer	ITH
Spacing	m	3.3	3.3
Burden	m	2.75	2.75
Drilling Factor	T/m	11.7	11.6
Drill Hole Diameter	mm	101.6	101.6
Redrill Factor	%	10	
Penetration Rate	m/h	19	11
Opening Raise	mm	760	

Collars will have a minimum of 2.0 m of stemming. Electrical detonators will be used to optimise flexibility and fragmentation. A typical production hole will be loaded with emulsion and two (2) boosters and detonators. The powder factor varies from 0.53 for the shortest stopes to 0.47 for the tallest stopes.

Figure 16.4 illustrates a typical drilling and blasting pattern for downhole transverse access stoping.

Figure 16.4 – Schematic Sections of Transversal Long Hole Stope



Source: Global Mine Design, 2023

16.2.3 BACKFILL

The proposed mining method requires systematic filling of all stope voids for ground stability and extraction of adjacent ore. The principal backfilling method at Loma Larga will be cemented paste backfill (CPB) made of filtered tailings, water, and cement in varying proportions.

An additional provision has been allowed for in the mine design to utilise either cemented or uncemented rock fill, depending on operational constraints that may occur from time to time. The initial production requires cemented rock fill (CRF) to be used until the mill and the paste plant are both commissioned. 39 stopes were identified for rock fill in the mine schedule for disposal of waste rock during operations

16.3 Geotechnical Considerations

16.3.1 BACKGROUND TO GEOTECHNICAL EVALUATION

Geotechnical considerations for the underground development and production were fully revised by GMD in 2023 for Loma Larga Project. The geotechnical work package was commissioned directly by DPM and included a review and gap analysis of geotechnical work undertaken for the Feasibility Study that was completed in 2020 by DRA Americas Inc. (DRA, 2020), followed by a fully revised geotechnical evaluation based on the findings of the initial GMD review. A summary of the GMD geotechnical evaluation (Global Mine Design, 2023) is described in the subsections below.

16.3.2 GEOTECHNICAL DOMAINS

16.3.2.1 Initial Domain Creation from Drillhole Data

Loma Larga has an extensive drillhole database, which comprises lithology, alteration, and geotechnical data. The GMD Gap Analysis identified that all three (3) data types were required to fully characterise the geotechnical conditions; therefore, drillcore intervals were only selected for geotechnical evaluation if the three (3) datasets were present. A total of 61,993 m of drill core was used for geotechnical evaluation.

GMD created geotechnical domains from a combination of key lithology and alteration classifications that are present in the Loma Larga orebody. The main rock types inside the orebody comprise volcanic lavas and tuffs of varying qualities. The main rock type outside of the orebody is dacite that post-dates mineralisation and alteration; a diatreme unit is present close to the boundary between the dacite and the volcanic units but is only considered to be a minor unit.

The geotechnical domains within the orebody closely follow the alteration zoning shown in Figure 8.1 (Silica, Advanced Argillic, and Propylitic) for each lithology. Illite and Smectite have been specifically segmented from the Propylitic zone because both of these clay minerals can swell when

exposed and particularly when in contact with mine waters; therefore, they represent a unique potential risk that must be addressed in the mine design.

The updated geotechnical domains are summarised in Table 16.3, with the naming convention: Lithology (Alteration). The abbreviation code and colour scheme for each domain are used throughout the geotechnical evaluation. Overall, the Loma Larga geotechnical database is comprehensive and provides sufficient data for mine design.

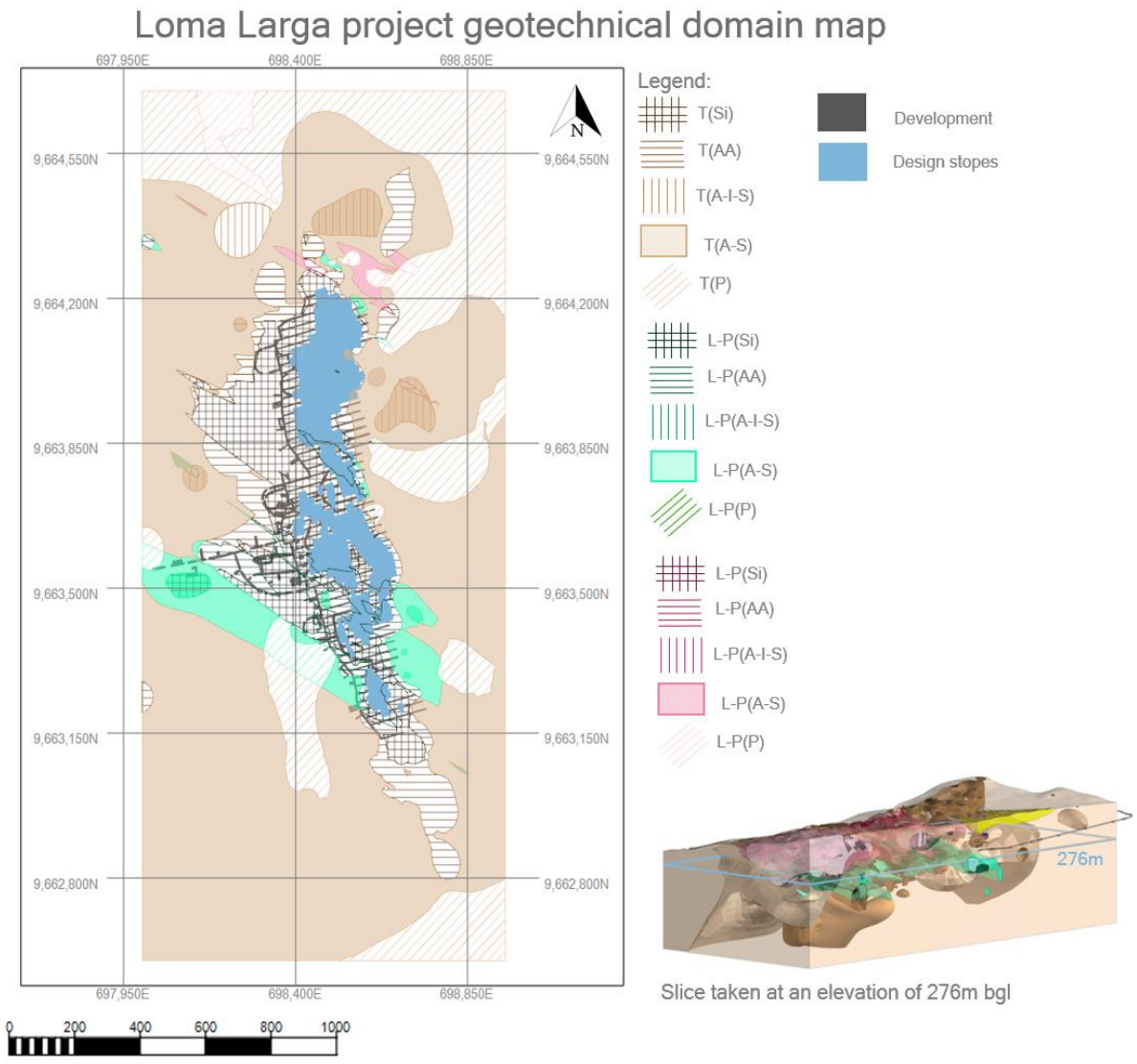
Table 16.3 – Summary of GMD Geotechnical Domains

Domain Name	Code
Dacite (outside of the orebody)	D(General)
Diatreme (general)	Dia(general)
Hornblende lava (silica)	L-H(Si)
Hornblende lava (advance argillic)	L-H(AA)
Hornblende lava (argillic-illite-smectite)	L-H(A-I-S)
Hornblende lava (argillic-smectite)	L-H(A-S)
Hornblende lava (propylitic)	L-H(P)

Domain Name	Code
Plagioclase lava (silica)	L-P(Si)
Plagioclase lava (advanced argillic)	L-P(AA)
Plagioclase lava (argillic-illite-smectite)	L-P(A-I-S)
Plagioclase lava (argillic-smectite)	L-P(A-S)
Plagioclase lava (propylitic)	L-P(P)
Tuff (silica)	T(Si)
Tuff (advanced argillic)	T(AA)
Tuff (argillic-illite-smectite)	T(A-I-S)
Tuff (argillic-smectite)	T(A-S)
Tuff (propylitic)	T(P)

The geotechnical domains were used in 3D form to add spatial context to the geotechnical data for mine design, as shown in Figure 16.5. All geotechnical data (rockmass classification, strength testing, ground support types, stope dimensions, etc.) have been categorised according to the domains.

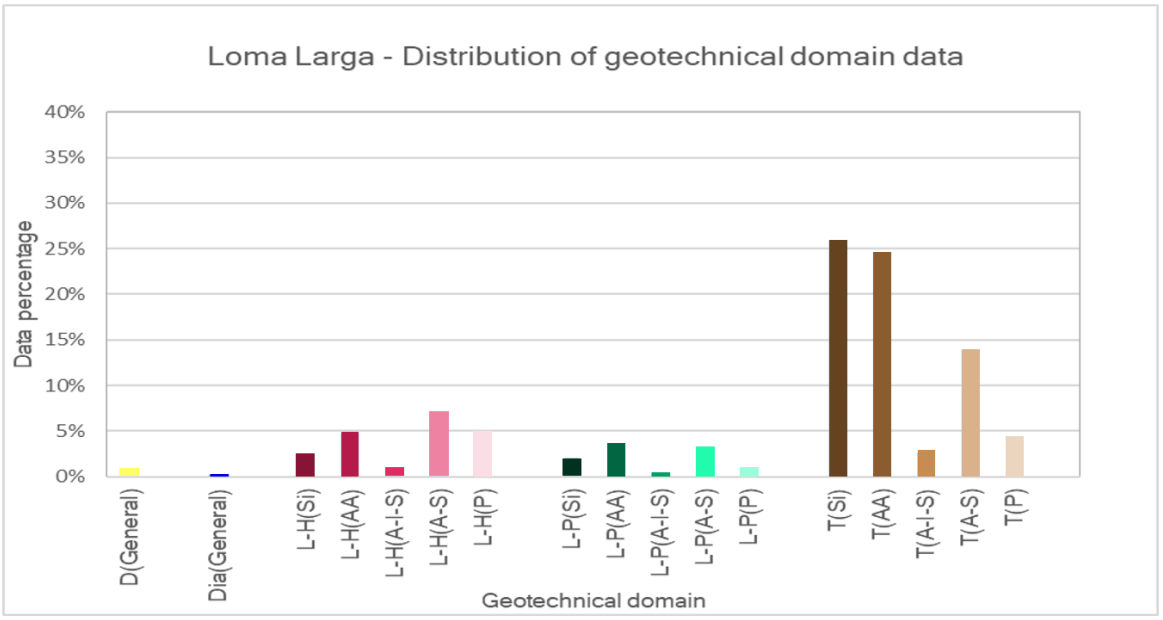
Figure 16.5 – GMD Geotechnical Domain Map



Source: Global Mine Design, 2023

Distribution of geotechnical domain data is shown in Figure 16.6. Tuffs are the most common lithology, followed by hornblende-rich lava (L-H), plagioclase-rich lava (L-P), dacite (D) and diatreme (Dia).

Figure 16.6 – Distribution of Geotechnical Data per Domain

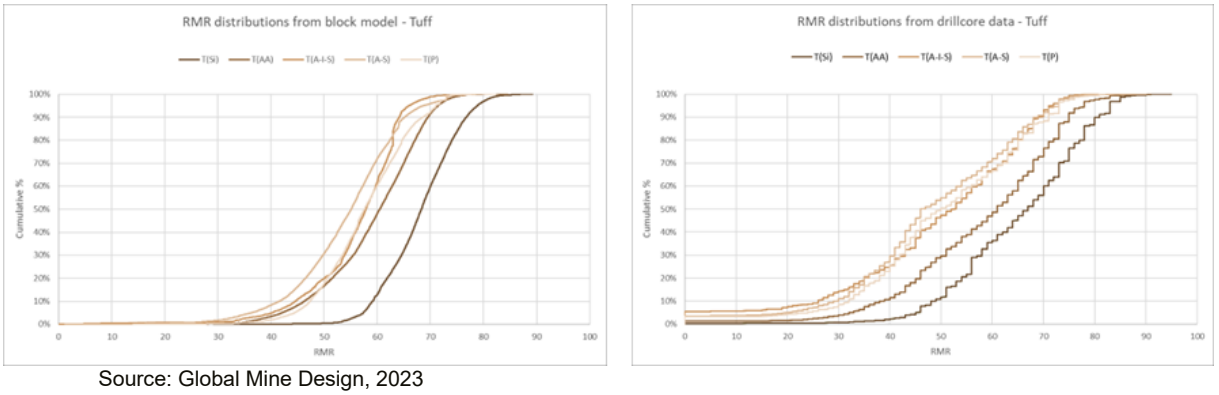


Source: Global Mine Design, 2023

16.3.2.2 Creation of a Geotechnical Block Model

Independently during the 2023 FSUO, DPM Technical Services personnel created a block model of rockmass quality data from the drillhole database. GMD validated the block model using the historical drillcore data and determined it suitable for use in design. An example comparison of the rockmass distributions from drillcore data and block model are shown in Figure 16.7.

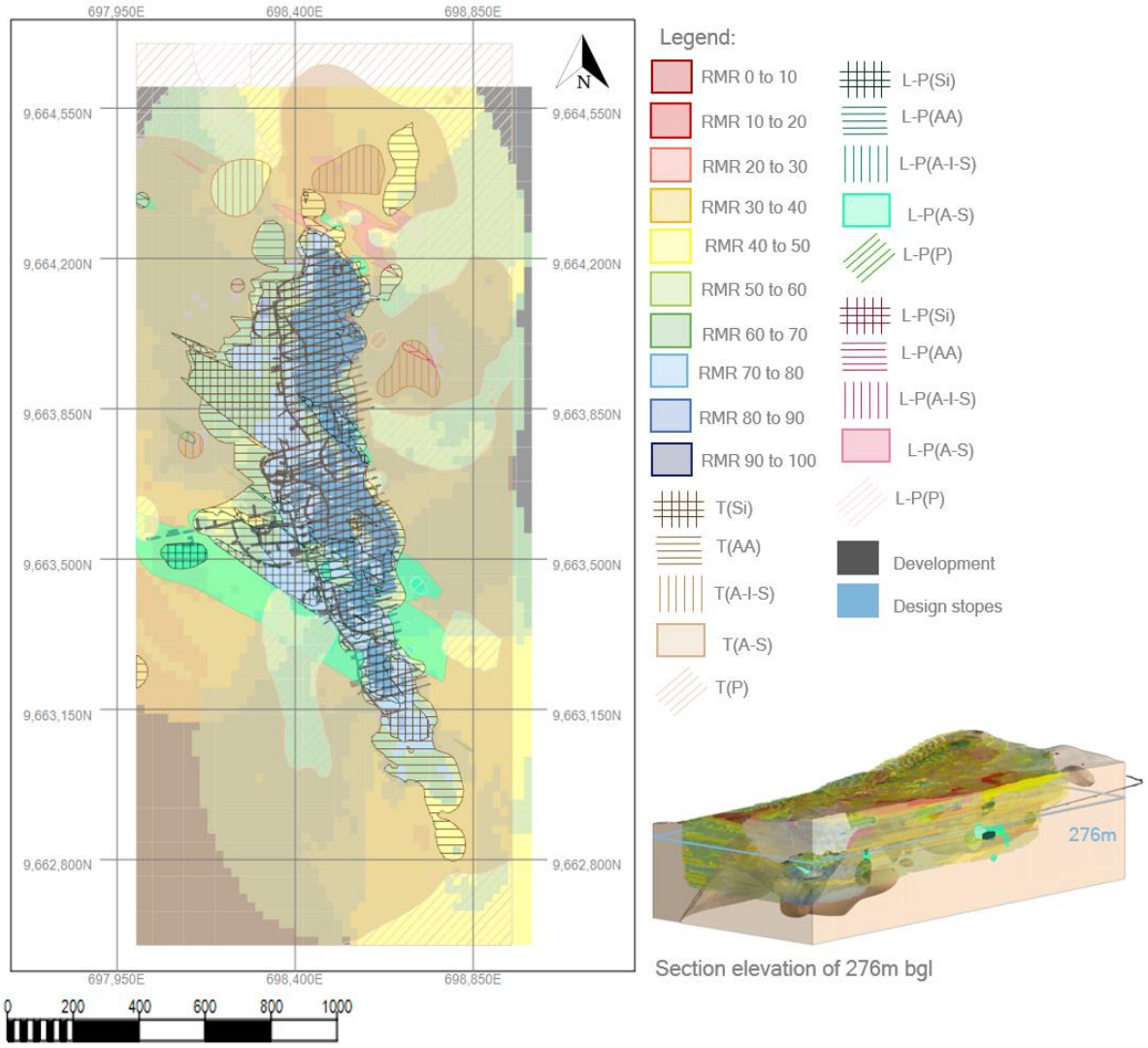
Figure 16.7 – Comparison of Rockmass Quality from Drillcore and Block Model Data



Source: Global Mine Design, 2023

Block models provide an additional means to add spatial context to geotechnical data; therefore, GMD subsequently combined the block model with the 3D domain shapes to enable continuous rockmass classification evaluation throughout the whole Loma Larga orebody, as shown in Figure 16.8.

Figure 16.8 – Combined Geotechnical Block Model and Domain Data



Source: Global Mine Design, 2023

16.3.3 ROCK QUALITY

Rockmass classification for each domain was determined according to both the Rock Mass Rating (RMR) system by Bieniawski (1989) and the Q system developed for the Norwegian Geotechnical Institute by Barton et al. (1974). The median values of each system are summarised for each domain in Table 16.4.

Table 16.4 – Summary of Rockmass Classification per Domain

Alteration	Median RMR_{89}	Median Q
D(General)	64	9.2
Dia(General)	50	2.0
L-H(Si)	63	8.3
L-H(AA)	59	5.3
L-H(A-I-S)	47	1.4
L-H(A-S)	46	1.2
L-H(P)	46	1.3

	Median RMR_{89}	Median Q
L-P(Si)	71	20.1
L-P(AA)	65	10.3
L-P(A-I-S)	46	1.2
L-P(A-S)	49	1.7
L-P(P)	50	1.9
T(Si)	67	12.9
T(AA)	61	6.6
T(A-I-S)	52	2.6
T(A-S)	46	1.3
T(P)	50	2.0

Rock strength was defined from a combination of the estimated Rock Strength Index (RSI) (ISRM, 2007) recorded during geotechnical core logging, and Uniaxial Compressive Strength (UCS) results from laboratory testing, if available. Data are summarised in Table 16.5.

Table 16.5 – Summary of Rock Strength Data

Alteration	**Lab' UCS** (MPa)	**RSI** (MPa)
D(General)	59	25-50
Dia(General)	-	5-25
L-H(Si)	-	100-250
L-H(AA)	45	25-50
L-H(A-I-S)	-	25-50
L-H(A-S)	15	5-25
L-H(P)	63	25-50

	Lab' UCS (MPa)	**RSI** (MPa)
L-P(Si)	104	100-250
L-P(AA)	75	25-50
L-P(A-I-S)	-	5 – 25
L-P(A-S)	25	25-50
L-P(P)	34	25-50
T(Si)	145	100-250
T(AA)	79	25-50
T(A-I-S)	-	25-50
T(A-S)	80	25-50
T(P)	61	25-50

Site characterisation led to the conclusion that rock quality is closely associated with alteration type. Specifically:

- Silica alteration generally maintains or improves the rockmass quality and intact rock strength of the original rock type, resulting in median RMR ratings of good and UCS around 100 MPa.
- Advanced Argillic alteration causes a mild degradation in rockmass quality and intact rock strength resulting in median RMR ratings between Fair to Good and UCS in the range of 45 to 79 MPa.
- Propylitic alteration causes a greater degradation in rockmass quality and intact rock strength resulting in median RMR ratings between Poor to Fair and UCS in the range of 15 to 80 MPa. The poorest conditions are where smectite and illite are present.

16.3.4 In-Situ Stress Field

Formal stress measurements have not been undertaken at Loma Larga mine; however, the tectonic environment means that the stress environment could be complex. Initial creation of the Andes was through subduction of tectonic plates, indicating thrust or reverse faulting (compression). However, alteration/mineralisation through fluid injection indicates later-stage normal faulting (separation) in the general region of Loma Larga, and potentially, localised strike-slip faulting if the Rio Falso fault is a conjugate fault that links the Gañarin and Girón faults.

The World Stress Map (2016) indicates mostly thrust data, with some strike-slip data in the vicinity of the Loma Larga site, as shown in Figure 16.9. Based on these data, a compressional stress tensor was assumed for the initial mine design, with the principal stress assumed to be horizontal with an azimuth ranging from 090° – 270° to 130° – 310°. The magnitude of the major principal stress was assumed to be twice the vertical stress, which was estimated from the expression:

$$\sigma_v = \rho \; x \; g \; x \; d$$

Where: $\rho = density\ (t/m^3),\ g = gravitational\ constant\ (9.81\ m/s^2) x\ d = depth\ (m)$

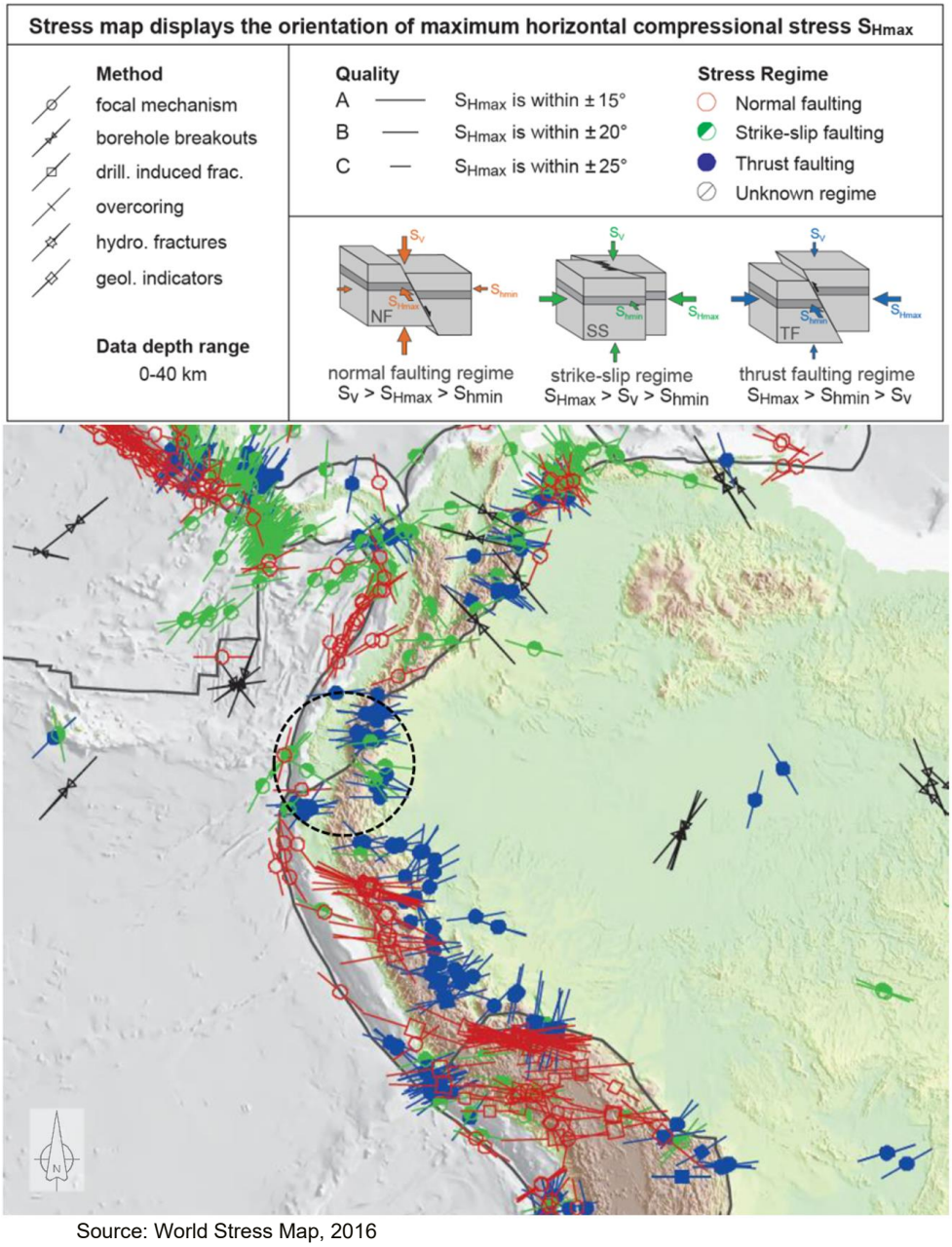
The height of Loma Larga above sea level may affect the applicability of this expression. Nevertheless, assuming validity of the above, an average rock density of 2.7 t/m³ and a maximum stoping depth of 250 m, the initial stress tensor for design is summarised in Table 16.6.

Table 16.6 – Initial Stress Tensor for Design

Principal Stress Component	**Magnitude** (MPa)	**Orientation** (Plunge/Trend)
σH	2 x σv	(0°/090°) to (0°/130°)
σh	1.5 x σv	(0°/000°) to (0°/040°)
σv	7.0	(90°/000°)

Further work would be required to determine potential localised variations in in-situ stress conditions, including the influence of large-scale structures. The general structural model is described in Section 16.3.5.

Figure 16.9 – General In-Situ Stress Conditions



Source: World Stress Map, 2016

16.3.5 STRUCTURAL DATA

16.3.5.1 Small-Scale Structures

Small-scale discontinuity (joint) orientations were analysed for each geotechnical domain separately and then compared together. This resulted in two (2) main sets observed for the majority of the tuff and lava domains and a further six (6) sets that were noted occasionally in various combinations across different domains.

The two (2) most common sets strike NE-SW and SE-NW respectively. These are sub-parallel to large-scale faults mapped within the orebody and therefore the orientations may be related to the faulting episodes.

Most domains recorded between two (2) and three (3) discontinuity sets, which indicates that wedges could form around mine openings. The joint set data per domain were subsequently used to validate ground support designs.

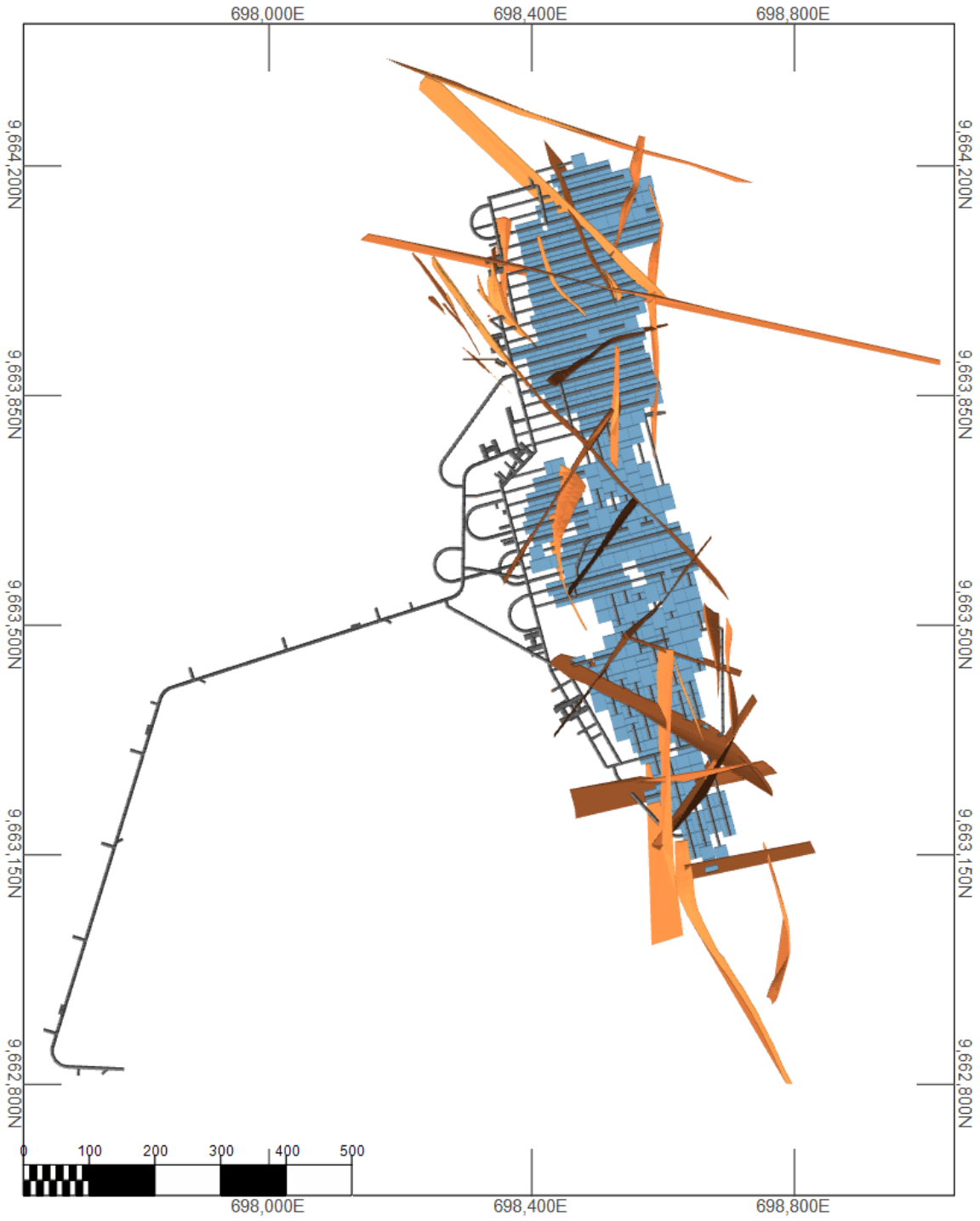
16.3.5.2 Large-Scale Structures

The site-scale structural model was created by Orix Geosciences Inc (Orix) in 2016 following their review of all available surface mapping and drillhole data and validation of previous structural interpretations. Faults were typically reported as sub-vertical and persistent (long).

Figure 16.10 replicates the Orix structural model relative to the latest mine design. As part of the geotechnical evaluation, GMD validated the structural model as far as practicable using drillholes and core photographs records. Very limited data on fault material or conditions (e.g., thickness, infill) were available.

It should be noted that the Orix model only focussed on the orebody region. Therefore, the area beyond the mineralised zone is probably not devoid of faults, only that any structures present are yet to be delineated.

Figure 16.10 – Loma Larga Fault Model



Source: Global Mine Design, 2023

A summary of fault conditions obtained during the GMD validation process is provided in Table 16.7. The median RMR values provide a reasonable approximation of fault qualities for the FSUO phase and do not indicate any immediate concerns. Local variability of fault properties can be studied in greater detail once faults become exposed in underground development.

Table 16.7 – Summary of Fault Information

Fault ID	Median RMR (Drillhole)	Development Domains
Boshier_1	50.5	-
Boshier_2	44.5	T(AA), T(A-S), L-P(AA), T(Si), T(A-I-S)
Boshier_4	40	L-P(Si), T(AA), T(A-S), L-P(AA), T(Si)
Boshier_5	46	L-P(A-S), L-P(AA), L-P(Si), T(Si)
INV_1	56	L-P(AA), L-P(A-S), L-P(P), L-P(Si), T(AA), T(A-S), T(Si)
INV_2	42	L-P(A-S), L-P(AA), L-P(Si), T(Si)
INV_3	51	L-P(A-S), L-P(Si), T(Si)
INV_4	44	L-P(AA), L-P(Si), T(AA), T(P), T(Si)
INV_5	45	L-P(AA), L-P(A-S), T(AA), T(A-I-S), T(A-S), T(Si)
INV_6	56	T(Si)
INV_8	33	-
INV_9	50	L-P(AA), L-P(A-S), L-P(Si), T(A-S), T(Si)
INV_10	40	L-P(A-S), T(A-S), T(P)
INV_11	44	-
INV_12	35	L-P(Si), T(AA), T(Si)
INV_13	46	L-P(Si), T(AA), T(Si)
INV_14	47.5	-
INV_15	-	-
INV_16	49	L-P(AA), T(AA), T(A-S), T(Si)
INV_17	45	L-H(AA), L-P(AA), T(AA), T(A-S), T(Si)
INV_18	43	L-P(A-S), L-P(Si), T(AA), T(A-S), T(Si)
INV_19	51	L-P(AA), L-P(A-S), T(AA), T(A-I-S), T(A-S), T(Si)
INV_20	44	T(AA), L-P(AA), L-P(Si), T(Si)
INV_21	49	T(AA), T(Si)
Or_1	55	L-P(AA), L-P(Si), T(AA), T(Si)
Or_2	45.5	-
Or_3	51	T(AA), T(A-S), T(Si)
Unknown	49.5	-

16.3.6 STOPE SIZING

Stope stability was assessed using the empirical stope stability chart method, which was first introduced by Mathews et al. (1981). The geotechnical parameters for each domain were used as the basis for the stope stability assessment. An additional factor was applied, based on the position of the domain relative to the orebody, where proposed stopes were determined to be located in one of three (3) positions:

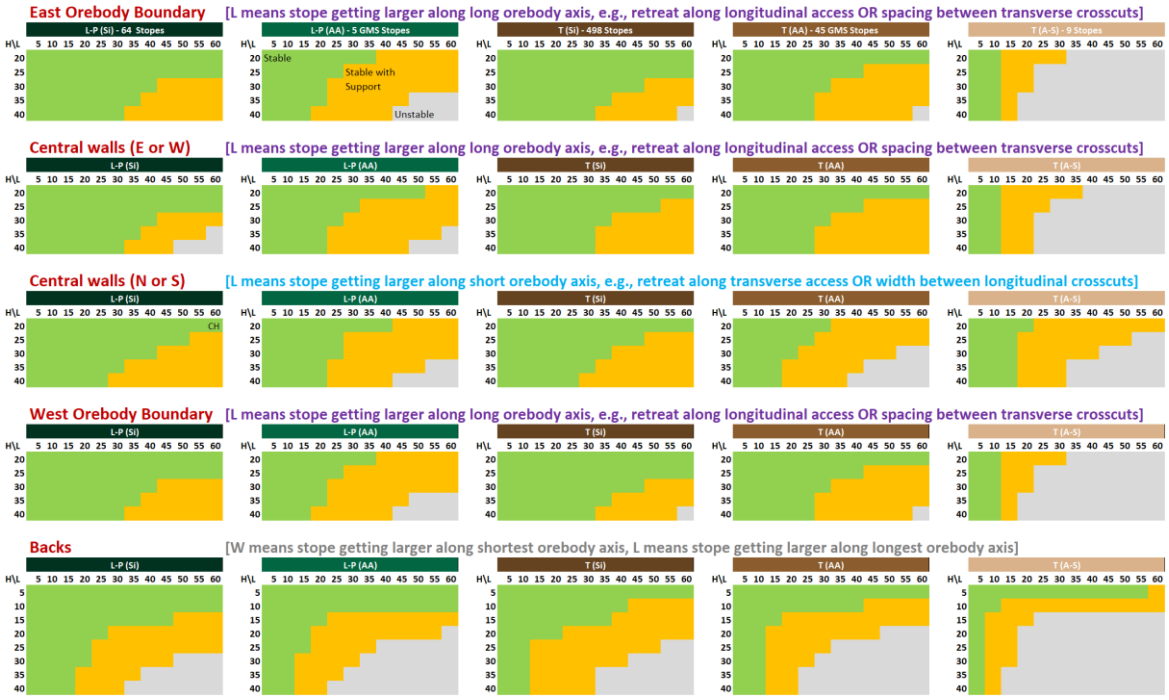
- On the East orebody boundary.
- In the Centre of the orebody.
- On the West orebody boundary.

The East orebody boundary represents the discrete change from host rock to orebody, whereas the West orebody boundary is determined from the NSR cut-off used in design. This means that the West boundary position could be subject to change in any future design iteration. The orientation of stope hangingwalls on the East orebody boundary and stope footwalls on the West orebody boundary vary according to the mineralisation/block model and therefore represent a slightly different case to stopes in the centre of the orebody. Stopes away from the orebody contacts (i.e., in the centre of the orebody) can use vertical walls, which are more straightforward to design.

The stability assessment results were presented in matrix form to enable GMD mining engineers to flex stope sizes and determine the most appropriate dimensions for each domain and stope position. Longitudinal and transverse accesses were also considered for increased flexibility.

The results matrix is shown in Figure 16.11, where green represents stable dimensions, yellow represents stable dimensions if support is used, and grey represents unstable dimensions.

Figure 16.11 – Matrix Summary of Stope Stability Assessment Results

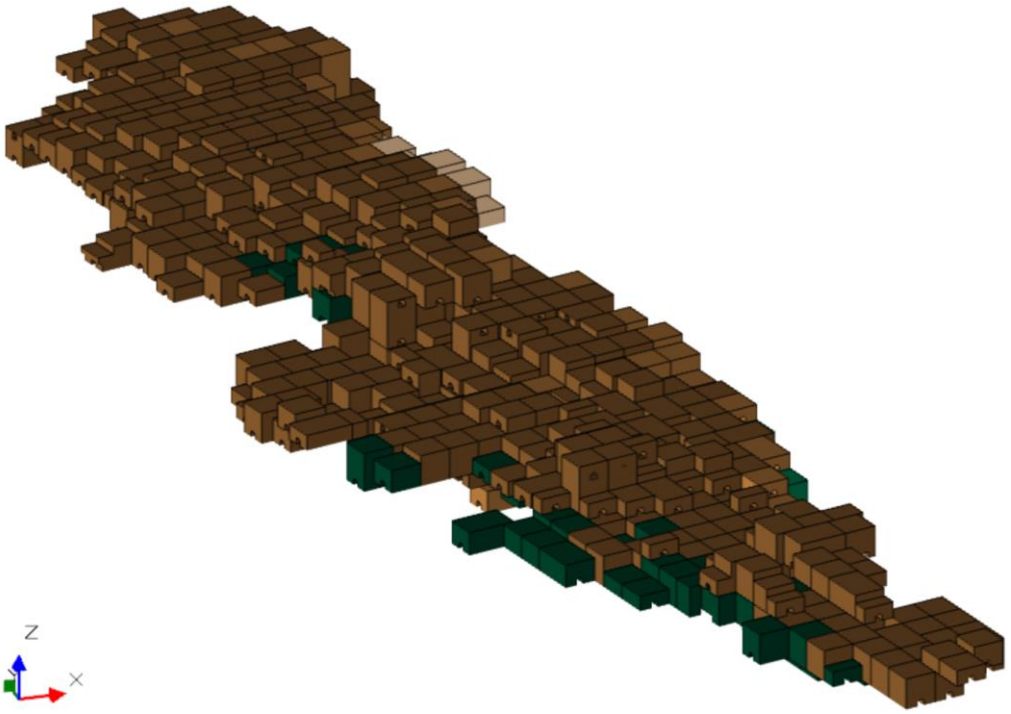


Source: Global Mine Design, 2023

As expected, stopes in lithologies with silica alteration can have the widest range of stable sizes, whereas stopes in lithologies with argillic and propylitic alteration (particularly where illite/smectite are present) require a more limited range of sizes. Although the backs represent a higher geotechnical risk than stope sidewalls (i.e., fewer stable sizes), they can be readily supported using deep-seated reinforcing elements such as cablebolts and therefore the impact to operations will not be excessive. Stope support details are provided in Section 16.3.7.

The resulting stopes are shown in Figure 16.12, coloured by domain. The number of stopes per domain are summarised in Table 16.8.

Figure 16.12 – View of Stopes by Geotechnical Domain



Source: Global Mine Design, 2023

Table 16.8 – Number of Stopes per Domain

GMD Domain	Number of stopes at NSR Cut-Off ($75/t)	Percentage
L-P(Si)	58	10.1%
L-P(AA)	5	0.9%
T(Si)	462	80.5%
T(AA)	42	7.3%
T(A-S)	7	1.2%

The range of stope dimensions per domain are provided in Table 16.9. The value in parentheses represents the modal average. The offset between crosscuts (regardless of transverse or longitudinal accesses) was fixed at 20 m. Other dimensions were varied to maximise the design economics.

Table 16.9 – Range of Stope Dimensions per Domain

GMD Domain	Transverse			Longitudinal		
	Length along orebody strike (m)	**Length across orebody strike** (m)	**Height** (m)	**Length along orebody strike** (m)	**Length across orebody strike** (m)	**Height** (m)
L-P(Si)	20 (20)	20-30 (20)	15-25 (15)	10-30 (20)	20 (20)	10-20 (10)
L-P(AA)		20 (20)	20 (20)	20-30 (20)		20 (20)
T(Si)		10-30 (15)	10-35 (15)	10-30 (20)		10-30 (20)
T(AA)		10-20 (10)	10-35 (10)	20-30 (20)		10-30 (10)
T(A-S)		10-20 (20)	10-35 (10)	- (-)		- (-)

The stope sizes are within a range commonly seen in the mining industry and are not considered excessive. Although geotechnical conditions indicated that larger stopes are possible, the ranges listed in Table 16.9 provided the most favourable economic return.

Stope designs are expected to incorporate repeating rings of blast holes, which enable production engineers to fire whole rings in slices into the void. Within this design method, the long axis of the stope can extend with each firing and technology such as Cavity Monitoring Surveys (CMS) enables wall stability to be monitored and individual stope panel lengths to be adjusted to local ground conditions. If excessive failure does occur, it will be possible to stop the stope, backfill, and re-slot to carry on. This production design method is therefore very efficient in adapting to poorer than expected ground conditions.

In addition, pastefill is the chosen backfill medium for the majority of stopes. Pastefill forms a tight contact with stope walls and can completely fill stope voids. These properties mean that there is a low anticipated risk of uncontrolled stope failures.

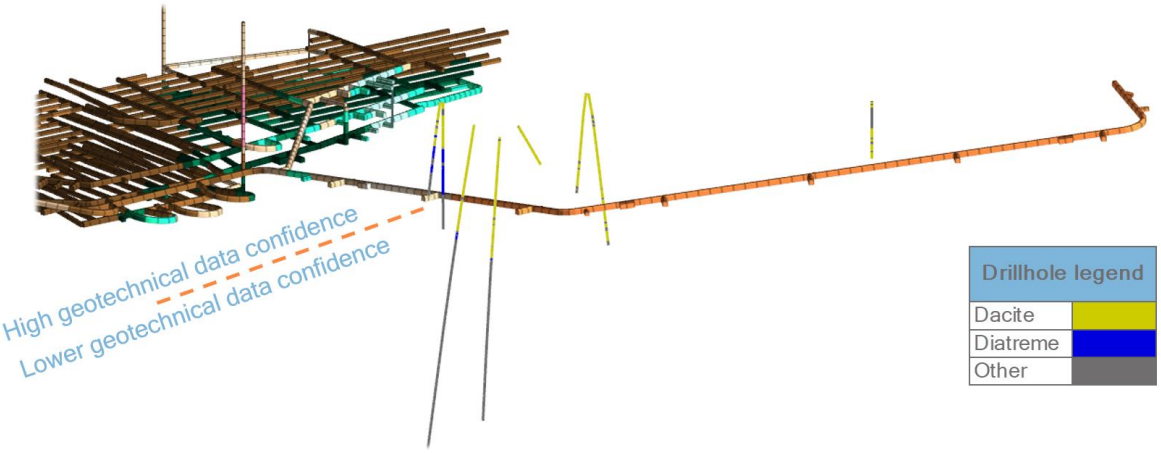
Overall, the geotechnical risk related to Long Hole Open Stoping is considered low. There is sufficient flexibility in the preliminary stope design and schedule sequence to enable optimisation of the initial parameters during the initial stages of orebody production.

16.3.7 GROUND SUPPORT

16.3.7.1 Portal (Decline Entrance)

The portal and decline are the most limited in terms of geotechnical data coverage. Figure 16.13 shows the drillholes presently available in the decline region. Although limited in number, these drillholes indicate that the decline should mostly be hosted in dacite (present outside of the orebody), which is relatively uniform and of good rockmass quality.

Figure 16.13 – Geotechnical Drillhole Data Available Near the Decline



Source: Global Mine Design, 2023

A geotechnical investigation is planned that will improve near-surface geotechnical information around the portal and surface infrastructure but is subject to environmental approval to begin the drilling campaign; therefore, geotechnical data at the portal is not yet available. The implied risk by the lack of geotechnical data has been addressed as far as possible at this Project stage through the proposed early-stage portal/decline entrance ground support design.

The portal box-cut will be excavated in stages, with surface support and reinforcing elements installed in-cycle.

The portal entrance will be supported with a combination of normal ground support (shotcrete surface support, mesh if required, rockbolts with plates, etc.) and steel sets. The proposed ground support installation cycle will include immediate installation of rock support to stabilise the ground, followed by installation of steel sets and backfilling any voids between steel sets and rock.

The portal support design incorporates a nominal 20m length of steel sets, but the sets can be extended as required, depending on rock quality during portal excavation. A dedicated geotechnical investigation during the portal box-cut excavation will provide the necessary information for the final portal support design.

16.3.7.2 Ground Support in Mine Development

Three (3) categories for primary ground support were defined according to rockmass quality. These are summarised for the relevant geotechnical domains in Table 16.10.

Table 16.10 – Ground Support Types per Domain

Ground Support Category	RMR Category	Geotechnical Domains	Surface Support	Rockbolt Pattern
1	Good	D(General) L-H(Si) L-P(Si) L-P(AA) T(Si) T(AA)	100 mm thick shotcrete, floor to floor	1.5 m between bolts in one ring 1.4 m between rings
2	Fair	L-H(AA) L-H(A-S) L-H(P) L-P(A-S) L-P(P) T(A-S) T(P)	100 mm thick shotcrete, floor to floor. Mesh if required	1.5 m between bolts in one ring 1.4 m between rings
3	Poor	Dia(General) T(A-I-S) L-H(A-I-S) L-P(A-I-S)	100 mm thick shotcrete, floor to floor. Mesh floor to floor over the top of shotcrete	1.5 m between bolts in one ring 1.2 m between rings

All geotechnical domains have between 35% and 70% of RQD data less than 50%, which indicates that relatively closely spaced fractures may form in the immediate skin around the excavation. Shotcrete is the most effective surface support for retaining small blocks; therefore, shotcrete has been chosen for all ground support categories. Once development is underway and the ground conditions have been validated in underground exposures, it may be possible to reduce the level of support in good quality ground.

The 100 mm thick layer, sprayed from floor to floor, is sufficient for the shotcrete to transmit tangential loads. Adding fibres to the shotcrete is not considered necessary at this stage.

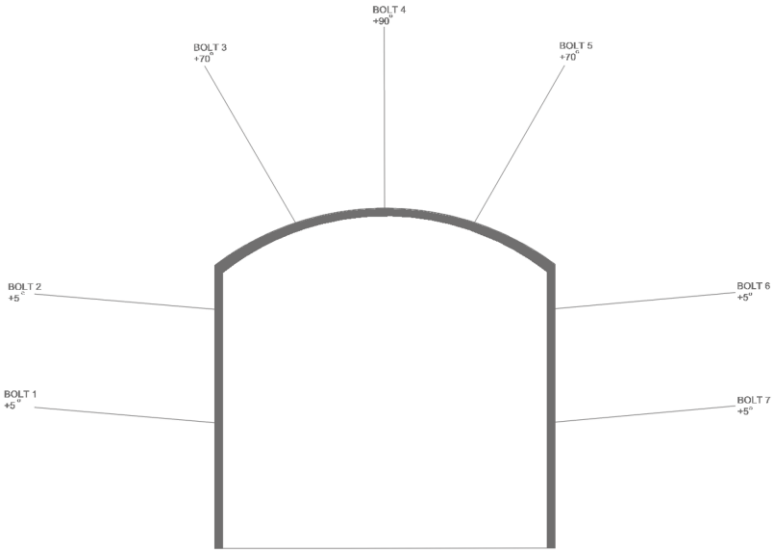
Mesh installed over the top of the shotcrete provides added deformation and retention capacity, should deformation in poorer quality rockmass occur. At this stage it is considered necessary in all

Category 3 support locations. In Category 2 locations, the need for mesh will be determined based on conditions during or after development. For budgeting purposes, an estimate of 50% of all Category 2 headings has mesh specified, which is considered conservative.

The ground support is currently designed to incorporate 2.4 m long fully resin-encapsulated rebar bolts, with a square plate of at least 150 mm x 150 mm and a screw nut surface fixture. The bolts should be at least 19 mm diameter. This is a standard bolt in the mining industry, easy to obtain and very versatile. However, resin-encapsulated rebar bolts can be difficult to install in poor ground conditions and it may be necessary to adjust to a friction bolt (e.g., Splitset) or an inflatable bolt (e.g., Swellex). Testing of the ground support patterns with friction bolts proved that they can be substituted, if necessary; however, if they are installed in long-term drives (e.g., decline), the friction bolts should be post-grouted with cement or resin to increase their support capacity and mitigate against corrosion.

The nominal rockbolt layout per ring is shown in Figure 16.14. The number of bolts per ring for each drive dimension are summarised in Table 16.11.

Figure 16.14 – Typical Rockbolt Pattern at Loma Larga



Source: Global Mine Design, 2023

Table 16.11 – Number of Bolts per Ring

Heading Type	Number of Bolts per Ring
4.5 mW x 4.5 mH	7
5.5 mW x 6.0 mH	9
9.0 mW x 6.0 mH	11

16.3.7.3 *Special Considerations for Swelling Clays*

Swelling clays (Smectite and illite) are located in propylitic alteration zones. Swelling clays will rarely affect stopes, primarily influencing mine access excavations (ramps and drifts). Development in the smectite and illite domains was reduced as much as possible in the FSUO mine design. Raises were re-located to avoid clay zones wherever possible.

Excavations in strongly altered smectite zones may incur very high swelling pressures, which is the reason for specifying shotcrete to be applied immediately on the rock surface, followed by mesh, where necessary. The mesh will provide additional deformation capacity in the ground support. At this stage, there is uncertainty around how the swelling clays will behave in underground conditions. In very severe cases, it may also be necessary to shorten development rounds in order to decrease the unsupported rock exposure during the development cycle and before shotcrete is applied.

It is also noted that the optimal bolt type and support design for poorer ground conditions will require some trial and optimisation as severe ground squeezing can be very difficult to control without proper mitigation measures. As previously noted, resin-encapsulated rebar bolts can be difficult to install in poor ground conditions and it may be necessary to adjust to a friction bolt (e.g., Splitset) or an inflatable bolt (e.g., Swellex).

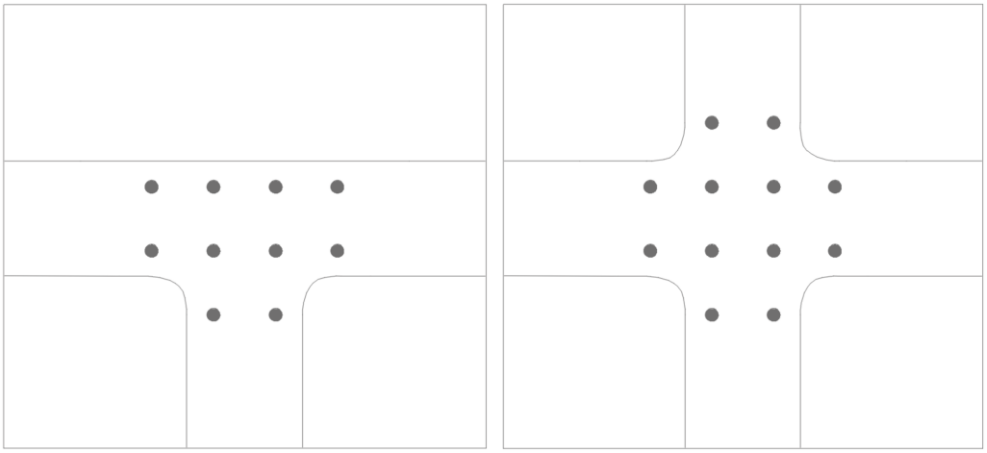
16.3.7.4 *Secondary Support for Larger Spans*

In addition to the primary support detailed above, larger spans such as intersections, or truck passing bays on the decline, will require secondary support in the drive backs. At this stage, 6.0 m double-stand fully cement-grouted cable bolts on a 2.5 m x 2.5 m pattern are used. These cable bolts are standard format in the mining industry. Double-strand cable bolts have a minimum design capacity of 45 tonnes.

Although the cable bolt pattern has been standardised for all routine drive sizes and wide spans, it is recommended that all large spans be structurally mapped to verify ground support. Cable bolts should be installed in the main drive, before the intersection is formed, with cable bolts installed in the secondary drive afterwards. It is also recommended that 4-way intersections be avoided to minimise ground control issues. Where they cannot be avoided, intersections should first be firmed as a 3-way, be fully supported with cable bolts, then the fourth drive be mined and supported.

The 3-way and 4-way intersections secondary support designs are shown in Figure 16.15.

Figure 16.15 – Ground Support for 3- and 4-way Intersections

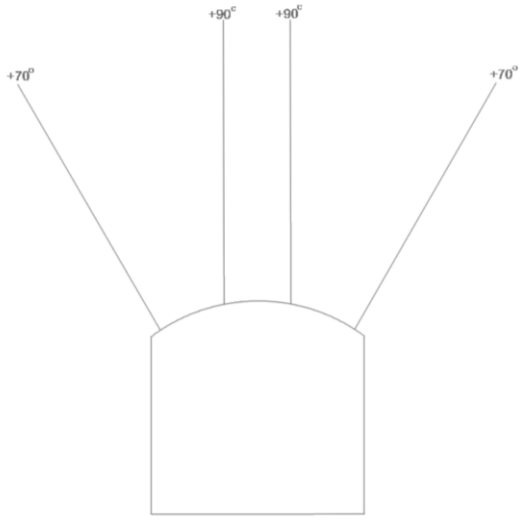


Source: Global Mine Design, 2023

16.3.7.5 Stope Support

Where support is required for stopes, the majority of cases require support only in the roof (back). For budgeting purposes, a standardised pattern was devised for a stope with nominal plan area of 20 m x 20 m, incorporating four (4) 6.0 m double-stand fully cement-grouted cablebolts in one (1) ring with 2.5 m between rings, as shown in Figure 16.16. The design was proven effective in theoretical ground conditions; however, it is recommended that support requirements for all stopes be reviewed on a case-by-case basis to ensure the cablebolt length and pattern are suitable for the encountered conditions.

Figure 16.16 – General Cablebolt Pattern for Stope Support



Source: Global Mine Design, 2023

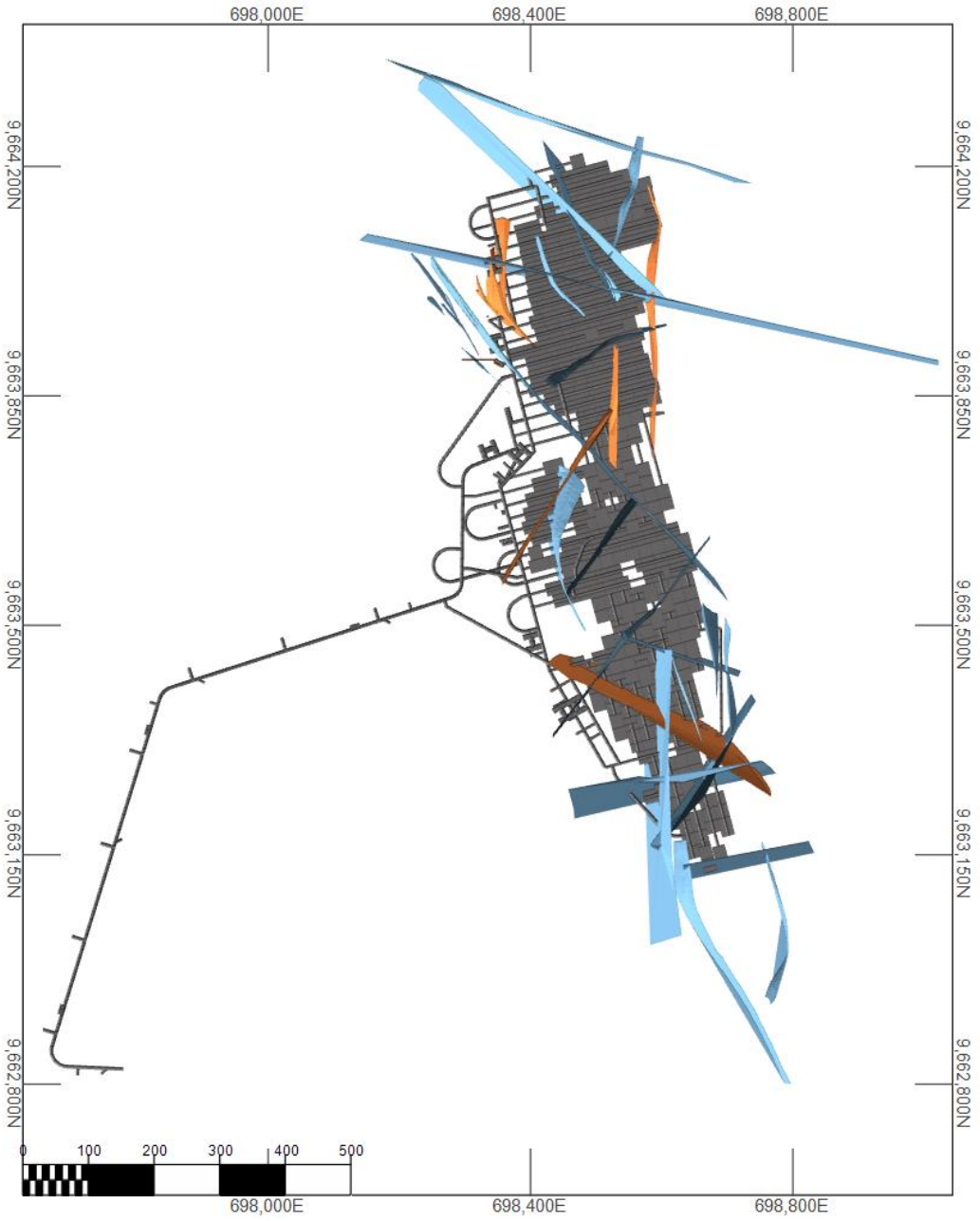
16.3.7.6 Special Consideration of Fault Conditions

All ground support designs were validated using both joint set (small-scale) structural data, and fault (large-scale) structural data. Faults were included with the domain data in accordance with Table 16.7. In most cases, cable bolts in addition to primary ground support decreased the likelihood of failure in openings to acceptable levels. The design Safety Factor (SF) of 1.4 or above could not be achieved for the faults listed below.

- Boshier_4
- INV_9
- INV_16
- INV_17
- OR_1

These faults are coloured orange in Figure 16.17. Particularly care must be retained when designing, planning, and working around these faults until the structural model can be fully validated and additional data on fault properties can be obtained.

Figure 16.17 – Faults to Be Observed Carefully During Mine Development



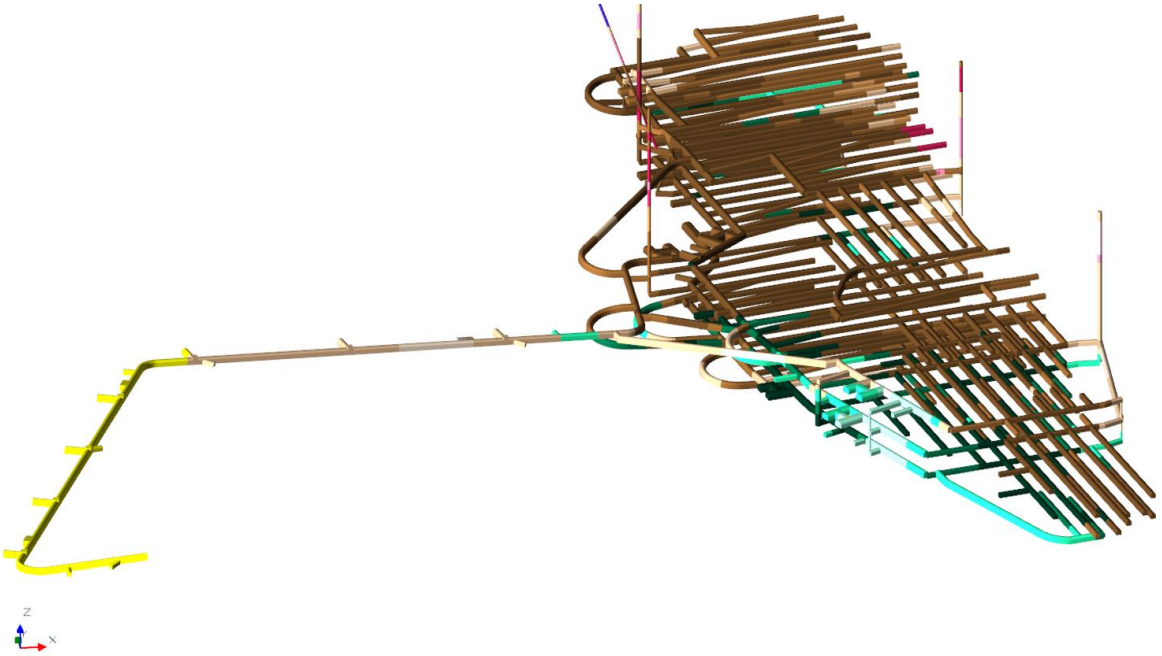
Source: Global Mine Design, 2023

16.3.8 DISTRIBUTION OF SUPPORT CATEGORIES

16.3.8.1 Mine Development

A preliminary estimation on the expected ground support requirements has been made for costing purposes. The three (3) primary support categories have been defined on the basis of the geotechnical domains. The geotechnical domains are shown superimposed on the development design in Figure 16.18 and Table 16.12 represents the distribution of support categories.

Figure 16.18 – Development According to Geotechnical Domains



Source: Global Mine Design, 2023

Table 16.12 – Support Category by Geotechnical Domains

Ground Support Category	Geotechnical Domains	4.5 m W x 4.5 m H		5.5 m W x 6.0 m H		9 m W x 6.0 m H	
		m	%	m	5	m	%
1	D(General)	36.7	0.1	816.7	2.5	18.7	0.1
	L-H(Si)	-	-	-	-	-	-
	L-P(Si)	2,595.4	8.0	295.4	0.9	57.7	0.2
	L-P(AA)	1,253.0	3.8	298.7	0.9	-	-
	T(Si)	15,602.1	47.5	1,852.0	5.7	197.7	0.6
	T(AA)	4,332.5	13.5	816.6	2.5	54.1	0.2

Ground Support Category	Geotechnical Domains	4.5 m W x 4.5 m H		5.5 m W x 6.0 m H		9 m W x 6.0 m H	
		m	%	m	5	m	%
2	L-H(AA)	73.4	0.2	-	-	-	0
	L-H(A-S)	8.0	0.1	-	-	-	-
	L-H(P)	-	-	-	-	-	-
	L-P(A-S)	564.5	1.8	559.7	1.7	65.8	0.2
	L-P(P)	82.7	0.3	192.0	0.6	35.8	0.1
	T(A-S)	1,112.1	3.5	754.4	2.3	17.8	0.1
	T(P)	76.4	0.2	156.0	0.5	5.0	0.1
3	Dia (General)	8.3	0.1	150.2	0.5	-	-
	T(A-I-S)	97.1	0.3	-	-	-	-

16.3.8.2 Stope Support

The estimated number of stopes potentially requiring cable bolt support is summarised in Table 16.13 for each domain. The estimated total number represents 16.7% of the total stopes.

Table 16.13 – Estimated Number of Stopes Potentially Requiring Cable Bolt Support

GMD Domain	Back Support	Hangingwall Support	Sidewall Support
L-P(Si)	0	0	0
L-P(AA)	4	0	0
T(Si)	35	0	0
T(AA)	42	0	2
T(A-S)	7	3	3
Total	**88 (15.3%)**	**3 (0.5%)**	**5 (0.9%)**

16.3.9 Ground Support Instrumentation

Instrumentation of ground support is recommended wherever the design assumptions of the support are likely to be tested. Typically, this is in intersections or large spans, in known poor ground conditions, near production zones, etc.

16.3.10 Raise Support

The current mine design has four ventilation raises and one escapeway raise. Each raise was evaluated using the techniques introduced by McCracken and Stacey (1989). The evaluation is preliminary at this stage, based on the available geotechnical domain data. Pilot drillholes are

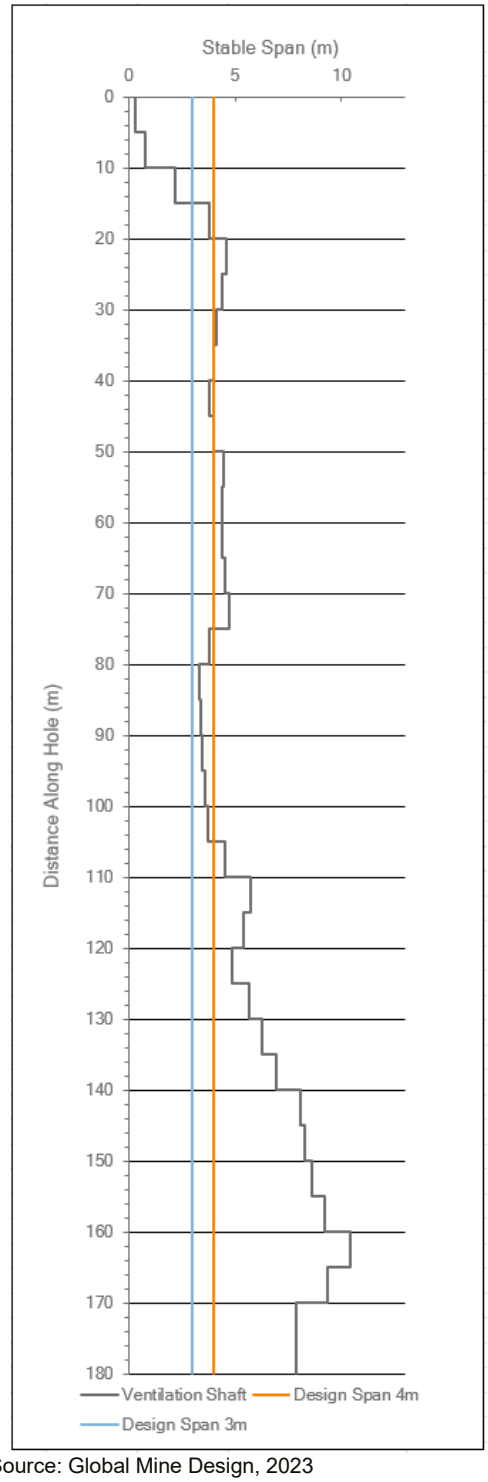
scheduled to be drilled at each raise location as part of the planned near-surface geotechnical investigation that is subject to permitting approval.

The most critical raise is the Central Western Raise, which is installed at the bottom of the decline as it is required before production stoping begins. Following the initial geotechnical investigation, the raise was moved to its current location and the diameter reduced to 3.0 m. All other raises were evaluated in the same manner and indicate stable conditions between 3 m and 4 m (the raise diameter varies based on ventilation requirements).

Based on the typical rockmass quality in the current geotechnical database, there is a possibility that the upper 15 m of each raise may experience stability issues, due to a combination of glacial soils over the site and the weathering affecting the transition from soil into rock. The geotechnical pilot hole planned for each raise location will determine if raise support is required in this section.

If the findings of any of the geotechnical pilot holes results in raises being moved, the new raise location will also require a new pilot hole to be drilled. Based on the extent of geotechnical data available for Loma Larga and the preliminary geotechnical assessments undertaken at the current design raise locations, the anticipated risk of needing to move the raises is low.

Figure 16.19 – Stability Evaluation of the Central Western Raise



Source: Global Mine Design, 2023

16.3.11 CROWN PILLAR

16.3.11.1 General Mine Conditions

A well-managed backfill program is of critical importance for maintaining global mine stability and to reduce the possibility of crown pillar failure. The mine plan has been designed to backfill each stope immediately after mucking out is completed. On this basis, the volume of void for any given period will be well-managed and issues relating crown pillar instability are not anticipated. If multiple stope panels are simultaneously left open beyond their allowable period, the probability of failure will increase. Overall, risk of short-term failure is mitigated by:

- Ground support installed in stope backs.
- Rapid cycle time to minimise the necessary stand-up time for critical crown pillar areas.
- Stope sequencing in the crown pillar area to ensure that production does not become concentrated in any one area.
- Continuous monitoring.
- Use of paste backfill with the intention of filling crown pillar stopes as tightly as possible.

With respect to monitoring, in most circumstances, visual monitoring of stope performance and/or void surveys will be sufficient. If critical stopes are identified during stoping operations (i.e., very poor local ground conditions encountered during development) conventional geotechnical instrumentation can also be implemented (i.e., slough meters or borehole extensometers installed in stope backs).

At the present time, the risk associated with fluid flow along fault structures is not quantified.

16.3.11.2 Portal and Decline Overburden

The footprint of some surface infrastructure (milling and processing facilities) lies directly over the decline alignment. The vertical distance ranges from 40 m near the portal to 140 m near the bottom of decline. Crown stability between the decline access and surface infrastructure is therefore critical to both underground production and milling/processing operations as interruption to either could have economic and/or safety consequences.

The decline ground support has been designed with this in mind, including steel set support for the portal entrance, which could be extended if required. In addition, monitoring instrumentation (extensometers, etc.) will be available for installation along the decline axis. An additional geotechnical investigation is also planned that will obtain additional information along the decline axis.

These data combined with the flexibility in the ground support scheme and the potential to increase the section length of steel sets in the portal entrance, provide assurance to the FSUO stage design.

It is recommended that the input parameters and assumptions are periodically reviewed as mine planning and design advance. Any changes required will be captured within Change Management procedures.

16.3.12 Provision for a Seismic Monitoring System

Although the Loma Larga is relatively shallow with respect to the ground surface, the property is situated around 3,800 m above sea level, in the Andes, which is a seismically active region. The Girón Fault is around 15 km northeast of the site and could generate a Magnitude 6 earthquake (NewFields, 2018c).

The silica alteration gives certain types of rock very brittle properties, whereas the argillic and propylitic alteration have more ductile properties. These opposing characteristics, plus the presence of large-scale faults within the production zones mean that the mine could induce seismic events up to Magnitude 2.

For these reasons, provision has been made for a seismic monitoring system to be installed at Loma Larga. Seismic monitoring systems are very versatile and can be used to ensure worker safety, mitigate engineering and production risks, optimise mine production and to engage with stakeholder communities. The preliminary system design will be installed in a staged approach commensurate with mining zones coming into production.

16.4 Underground Mine Design

16.4.1 Development Type Summary

The Loma Larga mine will be developed with underground headings to handle at least 3,400 t/d of ore plus related waste development. The complete list of development types with their associated dimensions is presented in Table 16.14.

Table 16.14 – Development Dimensions by Type

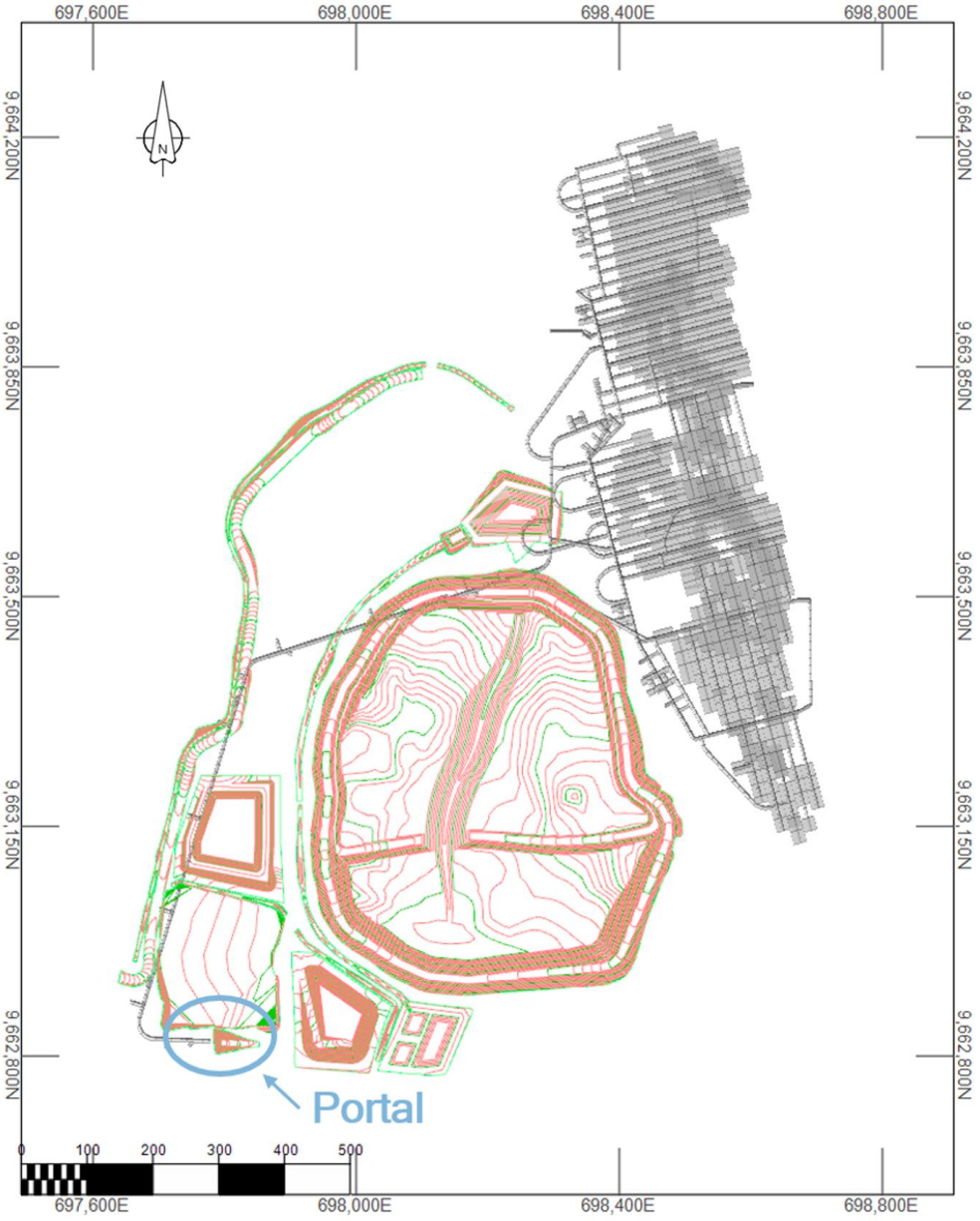
Development Type	Width (m)	Height (m)
Decline	5.5	6
Level Ramp	5.5	6
Haulage Drift	5.5	6
Stockpile	5.5	6
Re-muck	5.5	6
Electrical Bay	4.5	4.5
Electrical Main Substation	4.5	4.5
FAR Access	4.5	4.5

Development Type	Width (m)	Height (m)
Internal Ventilation Raise Access	4.5	4.5
Ore-Drift	4.5	4.5
Ore Pass Access	4.5	4.5
Paste Fill Bay	4.5	4.5
Pumping Station	4.5	4.5
RAR Access	4.5	4.5
Refuge	4.5	4.5
Safety Egress Access	4.5	4.5
Sump	4.5	4.5
Charging Bay	9	6
Maintenance Bay	9	6
Passing Bay	9	6

16.4.2 RAMP PORTAL

The ramp portal will be excavated in two (2) phases. The first phase will be to excavate the surface box cut. The soil will be excavated to the bed rock with an excavator. The bedrock will be benched down using a surface drill. The rock will then be trucked to the allocated waste stockpile. The portal area is presented in Figure 16.20.

Figure 16.20 – Portal Location



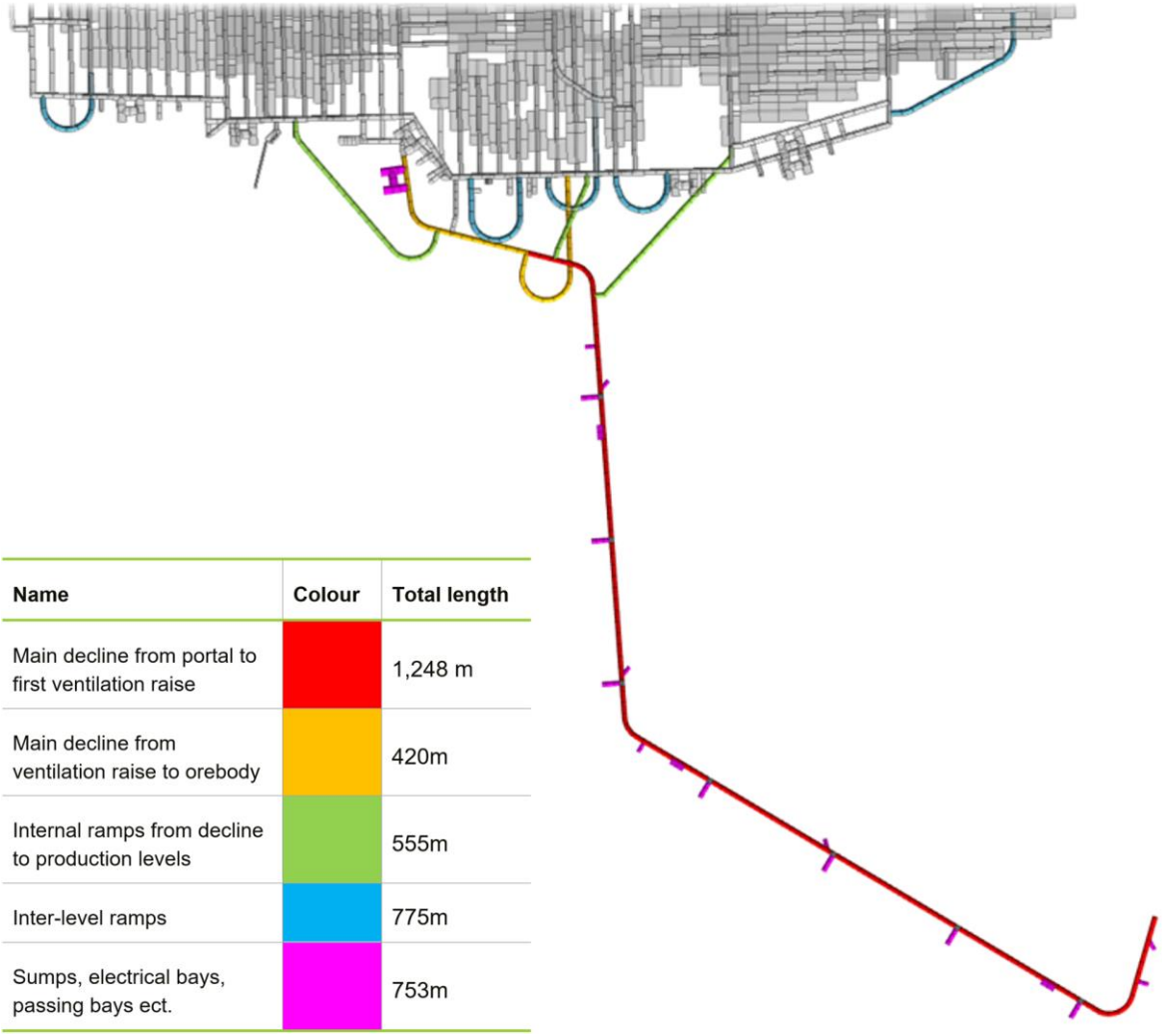
Source: Global Mine Design, 2023

16.4.3 Main Decline and Ramps

The ramp system is designed with a maximum slope of 1 in 7 to maximise haulage speeds. All ramps are sized 5.5m wide by 6.0m high to allow for fully-loaded 51-tonne underground trucks and 1.5 m ventilation ducts.

The ramp system is made up of the components illustrated in Figure 16.21. In addition, stockpiles (remuck bays) are located every 150 m, sumps every 300 m, and parallel passing bays every 450m. The general ramp section is shown in Figure 16.22.

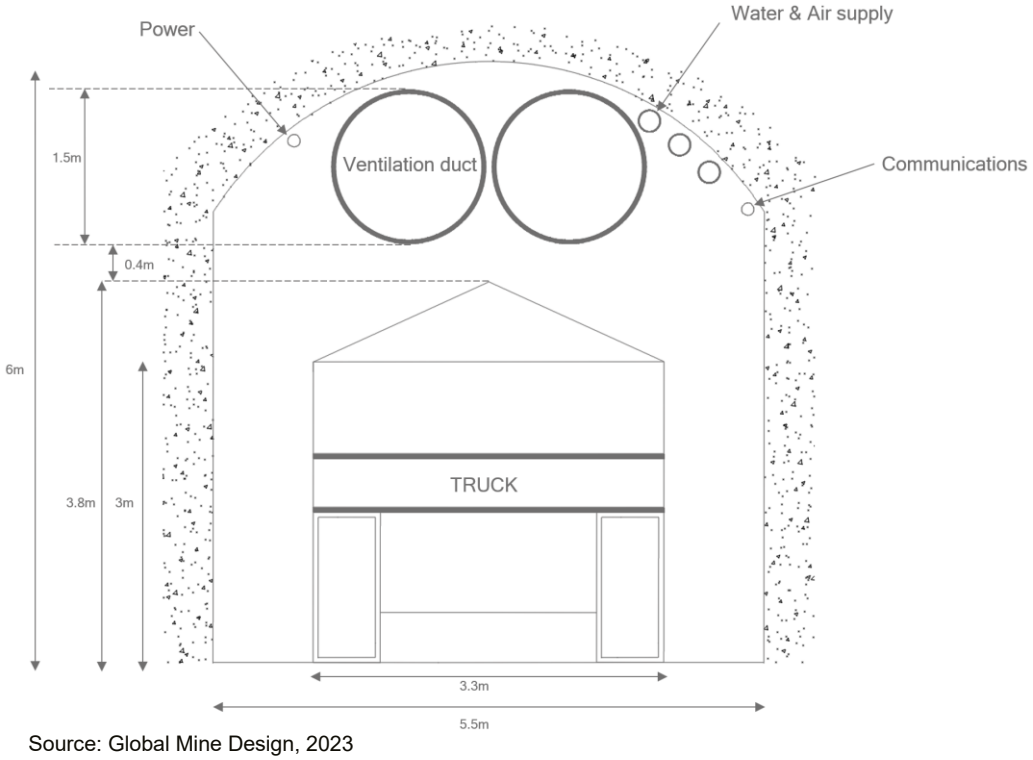
Figure 16.21 – Secondary Drive Types Associated with Decline Development



Name	Colour	Total length
Main decline from portal to first ventilation raise		1,248 m
Main decline from ventilation raise to orebody		420m
Internal ramps from decline to production levels		555m
Inter-level ramps		775m
Sumps, electrical bays, passing bays ect.		753m

Source: Global Mine Design, 2023

Figure 16.22 – Main Ramp General Arrangement



Source: Global Mine Design, 2023

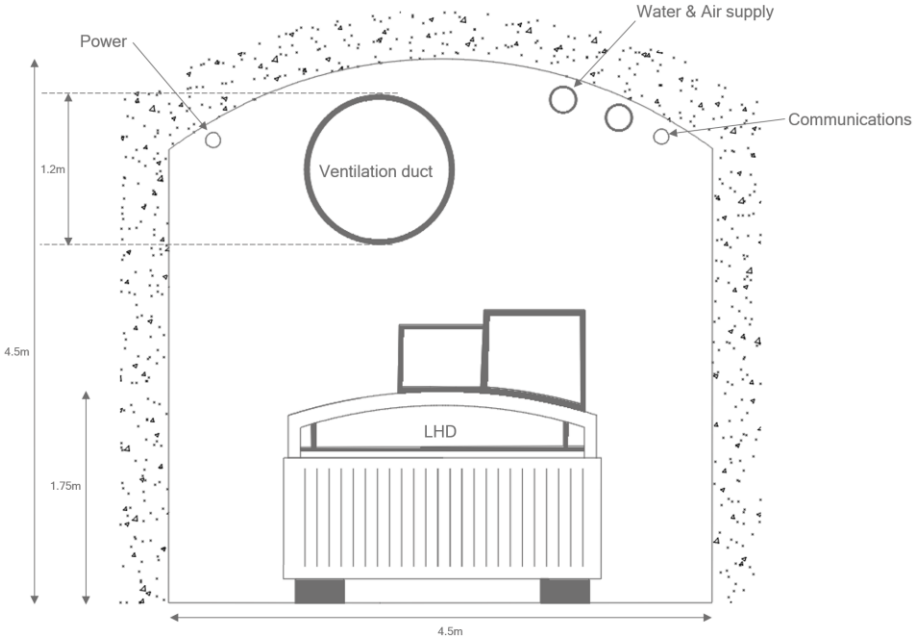
16.4.4 Main Haulage Drifts

The main haulage drifts were designed at 5.5 m wide by 6.0 m high to fit 51-tonne truck and 17-tonne LHD. They are laid out in the same manner as the decline but use a smaller ventilation duct (1.2 m). Haulage drifts provide the connection between the ore-drifts and the ramps for each level. During production, the main activity in the haulage drifts will be by LHDs carrying the ore from the stopes to the nearest stockpiles. Services installed in the main haulage drifts include compressed air, water, power, communication and blasting cables, as well as ventilation duct to the working areas. All haulage drifts will be driven at a 1 in 50 up-slope to drain ground water and process water towards the main pump station.

16.4.5 Ore-Drifts

The ore-drifts are designed at 4.5 m wide by 4.5 m high to fit 17-tonne LHDs used for the ore mucking. Temporary ventilation flexible ducting will be installed to ventilate the excavation and stope production operations. The general arrangement of the ore-drifts is shown in Figure 16.23.

Figure 16.23 – Ore-Drifts General Arrangement

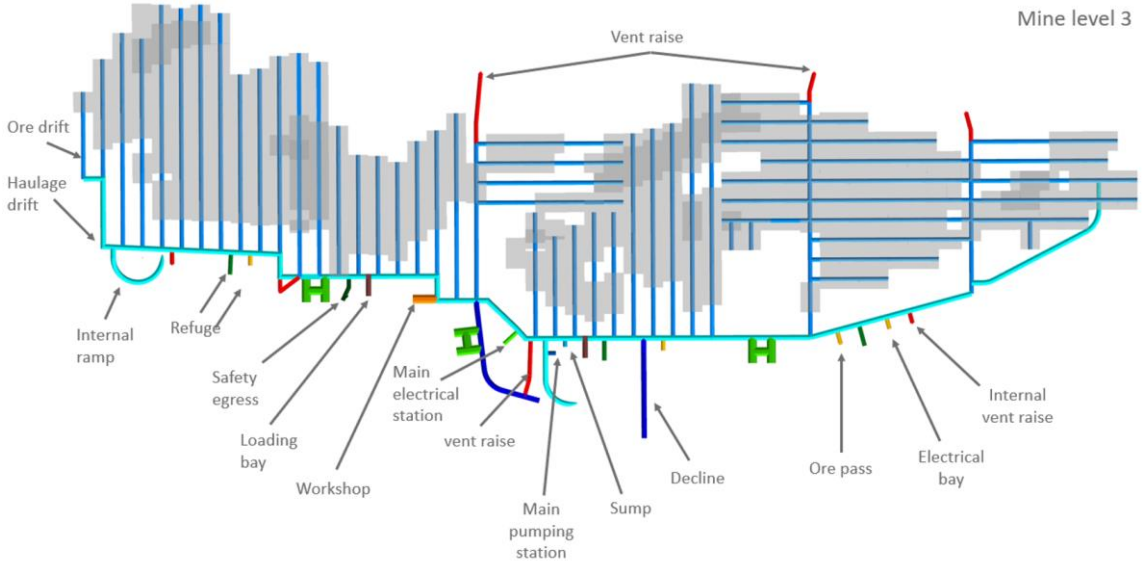


Source: Global Mine Design, 2023

16.4.6 OTHER DEVELOPMENT TYPES

As part of the mine development, other miscellaneous infrastructure will be excavated in the mine, such as the services or refuge bays. Figure 16.24 describes the other development types that are required in the mine.

Figure 16.24 – Plan View of Lower Level – Infrastructures



Source: Global Mine Design, 2025

16.4.7 Development Quantities

The development quantities by cost centre for the Loma Larga mine are presented in Table 16.15.

Table 16.15 – Development Quantities by Cost Centre

Development Type	Unit	Total
Lateral Capex	m	7,400
Vertical Capex	m	900
Lateral Opex	m	24,800
Total Development	**m**	**33,100**
Total Development	**kt**	**1,870**

16.5 Mine Schedule

The pre-production objective is to reach all major underground infrastructures as soon as possible and to prepare the first mining area for the production phase. Pre-production scheduling guidelines were established to:

- Provide a development ore stockpile for the plant commissioning.
- Establish the ventilation network and complete the second emergency egress route.
- Ensure sufficient development is completed to sustain ore production.
- Ensure that development is not progressed too far in advance of stopes.
- Prepare long hole stopes adequately to initiate production as soon as possible.

The underground LOM plan is based upon Proven and Probable Mineral Reserves and is scheduled using Deswik Sched software, a task-based dependency scheduler. Each task in the schedule is assigned a duration or a rate, depending on the type of activity. Resources and constraints are assigned to tasks and their capacity is profiled over the LOM to generate a balance mining plan.

16.5.1 Production Rates

To generate an accurate underground mine schedule, detailed cycle times were calculated for all major development and production activities. Production operating rates have been defined on each equipment cycle time. An overall mechanical availability of 85% is assumed for all major equipment, except LHDs at 82% and trucks at 89%, for which availability was set to the manufacturer's recommended values based on equipment studies. Table 16.16 summarises the rates and durations used for a typical stope production cycle.

Table 16.16 – Typical Stope Production Rate by Activity

Equipment	Task	Base Rate/ Duration	Unit
NA	Stope Preparation	3	days
Bolter	Cable Bolting	4	days
V30	Slot Raising	2	days
Production Drills (Top Hammer)	4” Down Hole Drilling	225	m/day
Production Drills (ITH)	4” Down Hole Drilling	120	m/day
NA	Charge and Blast	3	days
LHD	Stope Mucking	2,000	t/day
Paste Distribution	Barricade Preparation	2	days
Paste Distribution	Plug Fill	1	days
NA	Plug Curing Time	3	days
Paste Distribution	Stope Paste Fill	2,000	m³/day
NA	Long Time Curing	28	days

As for horizontal and vertical development, the advance rates vary mostly by cross section and whether the heading type is likely to be a single heading, or if multiple headings of the same type are mined. A summary of the average development rates is provided in Table 16.17.

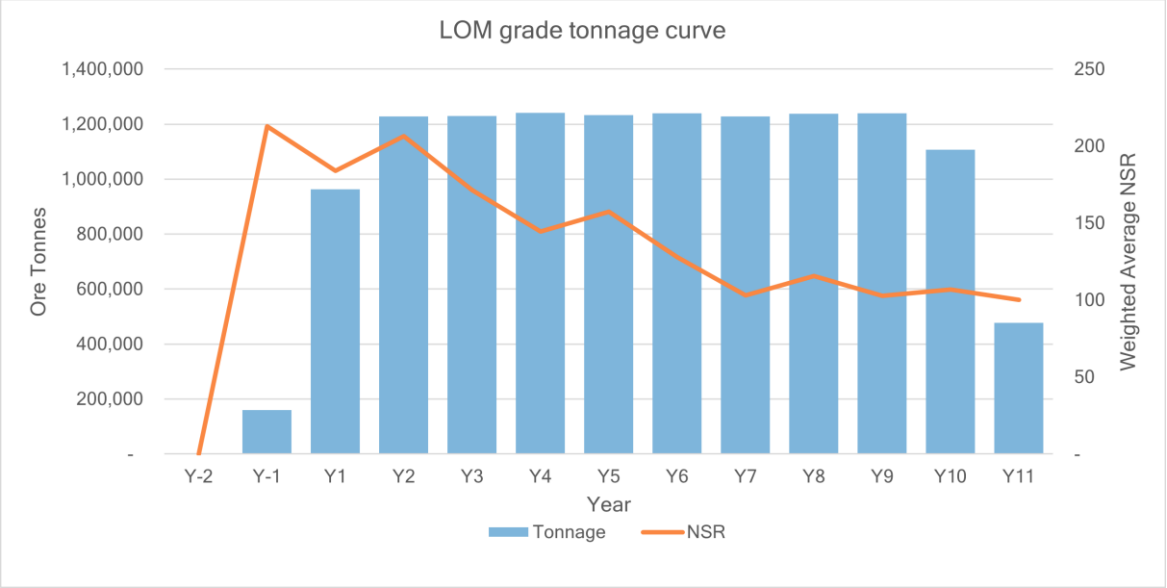
Table 16.17 – Advance Rates per Cross Section

Description	Advance Rate (m/d)
Lateral Development	
Ore Drive	6
Main Decline	6
Haulage Drifts	5
Raise Boring	5

16.5.2 PRODUCTION PLAN

The mine plan targets high-grade ore as early as possible in the production ramp-up phase. Material will be stockpiled during construction of the mill processing facility, which is expected to be fully commissioned at the start of Year 1. The production graph is shown in Figure 16.25 and is described below.

Figure 16.25 – Production Graph for FSUO Mine Plan



Source: Global Mine Design, 2023

Excavation of the portal and development of the main decline commences the mine plan in Year -2. Development will reach the ore during the second quarter of Year -1 and stope production will begin during the third quarter of Year -1. The commercial production period is planned to start from Q2 Year 1 when the mine reaches an ore production rate of 2,000 t/d (60% of nameplate capacity). During the ramp-up period, the higher-grade zones are targeted, and ore tonnes produced will increase to the steady-state production rate of 3,400 t/d by Year 2. The higher production rate will be maintained throughout the LOM until the very end of Year 10, when production begins ramping down due to a reduced number of production areas at the end of the mine life. The underground schedule summary is presented in Table 16.18.

16.5.3 Paste Schedule

Cemented Rock Fill (CRF) will be used in the early production period until the mill and the paste backfill plant is constructed and commissioned. Thereafter, all stopes are expected to be backfilled with Cemented Paste Backfill (CPB). Waste rock will be used to create berms to contain CPB as well as add a cap layer on top of the CPB (necessary for overhand mining sequences). Table 16.19 shows the planned volume of CRF and CPB over the LOM.

It may be advantageous to place loose rock fill into empty stopes from time to time, to dispose of development waste underground in an operationally efficient manner. Following a geotechnical assessment, 39 stopes were identified that could be used for uncemented rockfill, providing up to 227,000 m^3 of void. A total of 181,000 m^3 (79%) have been costed as waste rock fill and the remainder costed as pastefill. The exact material balance will be determined during operations but there is sufficient room to return all of the waste rock underground at the end of mine life, as well as provide flexibility to achieve short-term planning goals.

Table 16.18 – Underground Schedule Summary

Description	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Horizontal Development	m	32,201	1,242	5,572	6,918	6,516	5,399	2,556	2,149	1,201	640	-	-	9	-
Vertical Development	m	927	-	319	365	36	34	32	141	-	-	-	-	-	-
Total Development	**m**	**33,128**	**1,242**	**5,891**	**7,282**	**6,552**	**5,433**	**2,588**	**2,290**	**1,201**	**640**	**-**	**-**	**9**	**-**
Production Stoping	kt	11,902	-	67	844	1,084	1,143	1,158	1,155	1,186	1,202	1,238	1,240	1,107	477
Development Ore	kt	682	-	93	120	144	87	83	78	53	26	-	-	-	-
Total Ore	**kt**	**12,585**	**-**	**160**	**963**	**1,229**	**1,230**	**1,241**	**1,232**	**1,239**	**1,228**	**1,238**	**1,240**	**1,107**	**477**
Ore per Day, Average	kt/d		-	0.4	2.6	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.0	1.3
Contained Gold	Koz	1,904	-	38	190	268	220	181	202	170	142	163	145	130	56
Contained Silver	Koz	11580	-	129	947	1,486	1,403	1,483	1,390	1,128	784	797	785	884	364
Contained Copper	Mlb	80.0	-	1.6	8.9	13.1	10.4	9.8	8.9	6.3	4.8	5.1	4.8	4.6	1.6
Waste Produced	kt	1,192	101	267	286	224	216	48	35	8	6	-	-	0.4	-
Total dil & rec Material inc Waste	**kt**	**13,777**	**101**	**427**	**1,249**	**1,453**	**1,446**	**1,290**	**1,268**	**1,247**	**1,234**	**1,238**	**1,240**	**1,108**	**477**

Table 16.19 – Yearly Backfill Volume Summary

Method	Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
CRF	m³	-	58,590	65,000	65,000	16,250								
CPB	m³	-	-	88,506	354,479	425,875	428,855	439,858	418,864	431,565	428,701	442,702	375,482	193,507

16.6 Underground Mining Fleet

16.6.1 Mining Equipment List

The underground mining equipment fleet was chosen based on the evaluation of productivity needed to achieve the daily production rate of 3,400 t/d (achieved from Year 2), 362.5 days per year At this Project stage, all selected pieces of equipment are diesel powered.

The required operational quantity for all major equipment was estimated during the planning process. Yearly operation hours have been estimated for all auxiliary and transportation equipment. The required mobile equipment list by year is provided in Table 16.20.

Table 16.20 – Required Mobile Equipment by Equipment Type Year

Equipment Type	Supplier	Model	Electric / Diesel	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Production Equipment																
Jumbo 2 Boom	Sandvik	DD422iE	Diesel-Electric	-	4	4	4	4	4	4	4	4	4	4	4	4
Scoop 18T	Sandvik	SANDVIK LH517i	Diesel	-	3	5	5	5	5	5	5	5	5	5	5	3
Truck 50T	Sandvik	SANDVIK TH551i	Diesel	-	3	5	5	5	5	5	5	5	5	5	5	3
Long-hole Drill TOP HAMMER	Sandvik	DL422iE	Diesel-Electric	-	1	2	2	2	2	2	2	2	2	2	2	1
Explosive Truck	Maclean Engineering	MINE-MATE™ SERIES EC3-EMULSION	Diesel	-	2	2	2	2	2	2	2	2	2	2	2	2
Cable Bolter	Sandvik	DS421	Diesel-Electric	-	1	1	1	1	1	1	1	1	1	1	1	-
Raise Drill V30	Sandvik	DU412i	Diesel-Electric	-	1	1	1	1	1	1	1	1	1	1	1	1
Auxiliary Equipment																
Grader	Maclean Engineering	GR5 Motor Grader	Diesel	-	1	1	1	1	1	1	1	1	1	1	1	1
Flat Bed Trucks	Maclean Engineering	CS3-Crane Cassette	Diesel	-	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Lube Truck	Maclean Engineering	MINE-MATE FL3	Diesel			1	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer	Maclean Engineering	SS5 SHOTCRETE SPRAYER	Diesel	-	1	1	1	1	1	1	1	1	1	1	1	1
Trans-mixer	Maclean Engineering	TM3 TRANSMIXER	Diesel		1	1	1	1	1	2	2	2	2	2	2	1
ITC	IASA Global	CAT 930	Diesel	1	3	3	3	3	3	3	3	3	3	3	2	1
Transportation Equipment																
Light Vehicle	Toyota	Toyota Hilux	Diesel	2	15	15	15	15	15	15	15	15	11	11	10	4
Personal Carrier	Toyota	Toyota Troop carrier	Diesel	-	1	1	1	1	1	1	1	1	1	1	1	1
Ambulance	Toyota	Toyota Land Cruiser	Diesel	1	1	1	1	1	1	1	1	1	1	1	1	1

16.6.2 FLEET PRODUCTIVITY

Fleet productivity has been based on first principal calculations, benchmarking, supplier recommendations and practical experience. Equipment requirements were based on the estimated required operating hours in each period and the number of units of each piece of equipment needed to meet those hours. The mechanical availability of 82% was used for LHDs, 89% for trucks based on manufacturer's study-based recommendations. Availability of 85% was used for all other equipment.

The estimated annual net operating hours by equipment is presented in the Table 16.21.

Table 16.21 – Annual Net Operating Hours by Equipment Type

Equipment	Mechanical Availability (%)	NET Operating (h/y)
Jumbo	85	3,571
Scoop 17T	82	3,433
Truck 51T	89	4,238
Long-hole Drill Top Hammer	85	5,492
Explosive Truck	85	2,121
Cable Bolter	85	2,121
Long-hole Drill ITH	85	5,492
Grader	85	1,311
Flat Bed Trucks	85	2,121
Fuel Lube Truck	85	1,033
Shotcrete Sprayer	85	2,121
Trans-mixer	85	1,758
Land Cruiser	85	1,396
Personal Carrier	85	1,396
Ambulance	85	308

16.6.3 MAINTENANCE SCHEDULE

The maintenance for all equipment is done following the supplier recommendations. To maximise equipment lifespan, rebuilds are planned following supplier rebuild cycle recommendations. The main maintenance of the equipment will be done on surface, at the maintenance shop. Emergency repairs and equipment condition assessment on mobile equipment will be done underground at the maintenance bay.

16.7 Workforce Requirements

The underground mine development plan calls for all jumbo development and production to be completed by the owner, except the pre-production phase of the ramp that will be contractor-based. The ventilation raises, the emergency egress raise, decline development, and diamond drilling will also be done by a contractor.

Underground workforce requirements for the mine were estimated from first principles as well as supplier recommendations. They were then checked and finalised by ensuring all equipment was adequately staffed. The underground mine is scheduled to operate 362.5 days per year and in accordance with Ecuadorian mining regulations that allows no more than 6 hours per shift per day plus 1 hour of overtime work underground. Therefore, the mine plan is based on a provision of 3 x 7 hours shifts per day. No allowance has been allocated for weather delays.

Mine personnel are separated in five (5) categories: mine supervisions, production, mine services, maintenance, and technical services. Table 16.22 details the personnel required over LOM.

Table 16.22 – Mine Personnel List

Mine Personnel	MAX	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Mine Supervision														
Mine Manager	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Mine Superintendent	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Mine Clerk	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Trainer	12	-	12	12	8	2	2	2	2	2	2	2	1	1
Junior Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Mine Captain	4	-	4	4	4	4	4	4	4	4	4	4	4	1
Mine Supervisors - Production	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Mine Supervisors - Development	4	-	4	4	4	4	4	4	4	4	4	4	4	-
Mine Supervisors - Services	2	-	2	2	2	2	2	2	2	2	2	2	2	1
Production														
Long-Hole Driller	4	-	4	4	4	4	4	4	4	4	4	4	4	-
Long-Hole Driller-Raise Opening	4	-	4	4	4	4	4	4	4	4	4	4	4	4
Blasters	8	-	8	8	8	8	8	8	8	8	8	8	8	3
LHD Operators	20	-	8	20	20	20	20	20	20	20	20	20	16	6
Truck Operators	20	-	8	20	20	16	16	16	16	16	16	16	16	6
Jumbo Development and Bolting Operator	16	-	16	16	16	16	16	16	16	16	16	16	16	-
Miner Helper	12	-	12	12	12	12	12	12	12	8	8	8	8	3
Cable Bolter	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Mine Personnel	MAX	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Mine Services														
Service Operator - Utility Crew (full time)	8	-	4	8	8	8	8	8	8	8	8	8	8	0
Service Operator Utility Crew	12	-	12	12	12	12	12	12	12	12	12	12	8	8
Service Operator Lamp and Dry	-	-	4	4	4	4	4	4	4	4	4	4	4	4
Service Operator Labour Spare	5	-	5	5	5	5	5	5	5	5	5	5	5	0
Maintenance														
Maintenance Superintendent	1	-	1	1	1	1	-	-	-	-	-	-	-	-
Maintenance Superintendent	1	-	-	-	-	-	1	1	1	1	1	1	1	1
Maintenance Captain (Mechanics and Electrical)	1	-	1	1	1	1	-	-	-	-	-	-	-	-
Maintenance Captain (Mechanics and Electrical)	1	-	-	-	-	-	1	1	1	1	1	1	1	1
Maintenance Supervisor Mechanics	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Reliability Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Mechanical Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Trainer Maintenance	4	-	4	4	4	4	-	-	-	-	-	-	-	-
Trainer Maintenance	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Mechanics Mobile - Surface	40	-	16	32	40	40	40	40	40	40	40	40	40	24
Mechanics - UG	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Mechanics Mobile - Labour	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Mechanics Mobile - Welder	4	-	2	4	4	4	4	4	4	4	4	4	4	2
Mechanics Fixed - Surface	4	-	2	4	4	4	4	4	4	4	4	4	4	2
Maintenance Supervisor Electrical	2	-	2	2	2	2	2	2	2	2	2	2	2	1

Mine Personnel	MAX	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Electrical Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Electrical Mobile - Underground	4	-	4	4	4	4	4	4	4	4	4	4	4	2
Electrical Instrumentation - Surface	8	-	8	8	8	8	8	8	8	8	8	8	8	3
Electrical Fixed - Surface	2	-	2	2	2	2	2	2	2	2	2	2	2	1
Electrical Fixed - UG	8	-	4	8	8	8	8	8	8	8	8	8	8	3
Technical Services														
Technical Services Superintendent	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Chief Mine Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Mid Term Planner Engineer	1	-	1	1	1	1	1	1	1	1	1	1	-	-
Long Term Planner Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Ventilation/Construction Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Ventilation Technician	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Short Term Planner Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Chief Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geotechnical Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	-
Geotechnical Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	
Geotechnical Technician	2	-	2	2	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	4	2	4	4	4	4	4	4	4	4	4	4	4	2
Long Hole Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Long Hole Technician	2	-	2	2	2	2	2	2	2	2	2	2	2	1
Backfill Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Backfill Technician	2	-	2	2	2	2	2	2	2	2	2	2	2	1

Mine Personnel	MAX	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Chief Geologist	1	-	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist	1	-	1	1	1	1	1	1	1	1	1	1	1	-
Geologist	2	-	2	2	2	2	2	2	2	2	2	2	2	1
Geology Technician	4	2	4	4	4	4	4	4	4	4	4	4	4	2
TOTAL	**260**	**6**	**204**	**260**	**260**	**242**	**242**	**242**	**242**	**238**	**234**	**230**	**217**	**94**

16.8 Mine Ventilation

16.8.1 System Overview

The Loma Larga Project will use a variety of underground equipment during the life of mine. It is expected that development of the decline will be carried out by a contractor and once production begins, operations are to be led by DPM. During both stages diesel equipment will be used, as provided in Table 16.20. The ventilation system is designed to mitigate risks suggested through all stages of the mine life.

A variety of contaminants will be produced over the mine life, such as blasting fumes, heat, and DPM. Dust is another contaminant that will be produced. The presence of silica altered rockmass indicates the potential for silica dust that poses a long-term exposure risk to worker health. It is vital that a well-designed ventilation system and other management techniques are implemented to effectively manage dust and contaminants.

Ventilation analysis and network design were undertaken in Ventsim, which is a 3D mine ventilation simulation software that enables accurate calculation and simulation of ventilation scenarios. The Loma Larga mine plan was imported from Deswik into Ventsim allowing for ventilation system design at key stages of the mine life (ramp-up, full-production, etc.) as well as investigation of scenarios relating to heat sources, leakage, and shock-losses.

The key ventilation circuit components are listed in Table 16.23. The ventilation circuit is illustrated in Figure 16.26.

Table 16.23 – Drive Dimensions

Excavation Type	Dimensions
Decline and Haulage Routes	5.5 mW x 6.0 mH
Ore Drives	4.5 mW x 4.5 mH
Fresh Air and Exhaust Drives	4.5 mW x 4.5 mH
Charging Bay	9.0 mW x 6.0 mH
North West, Central East, and South East Surface Ventilation Raises	4.0 m diameter
Central Western Surface Ventilation Raise	3.0 m diameter
Escape Raise	2.4 m diameter
Internal Ventilation Raises	2.0 m diameter
Ore Pass	5.0 m diameter

Figure 16.26 – Overview Looking North of the Loma Larga Ventsim Model with Raise Names



Source: Global Mine Design, 2023

Utilising four (4) ventilation raises provides flexibility to adapt the ventilation circuit over the mine life. The first ventilation raise (3.0 m diameter) is scheduled to be available in the second half of Year -1, with additional raises developed in Year 1, Year 2 and Year 5. Main ventilation fans will always be installed in an exhaust (pull) configuration on top of the raises. In steady-state production, two

(2) sets of two (2) surface fans will work in parallel to generate negative pressures underground allowing air to be drawn down the decline and fresh air raises, to supply the required volumetric airflow requirements. The minimum required airflow is 356 m³/s. The design adheres to the air velocity limits at all stages of the mine.

Operating the mine as an exhaust ventilation circuit allows for fresh air to be drawn into the mine via the decline and escape raise when operational. In the unlikely event of an underground fire, this will allow access routes into the mine for emergency services (mines rescue) to be clear of contaminants, allowing for a quick response time.

The main ventilation circuit will be controlled via regulators underground. They are adjusted as necessary, to allow for the movement of air through internal ventilation drives and raises between mine production levels.

Where necessary, a secondary ventilation circuit will supply air to development and production headings away from the main circuit. Auxiliary fans situated within bulkheads in the fresh air drives and ducting will direct forced air into production headings. Auxiliary ventilation is managed daily by choking ventilation ducting to prevent airflow to unneeded work areas.

16.8.2 VENTILATION DESIGN CRITERIA

16.8.2.1 Air Temperatures and Humidity

Temperature and humidity were supplied by the University of Cuenca's research centre *Programa de Manejo de Agua y Suelos* (PROMAS) from a weather station "Quimsacocha 1", established on a property near the Loma Larga deposit. Data readings from the weather station have been supplied from 2005-2023 and are summarised in Table 16.24.

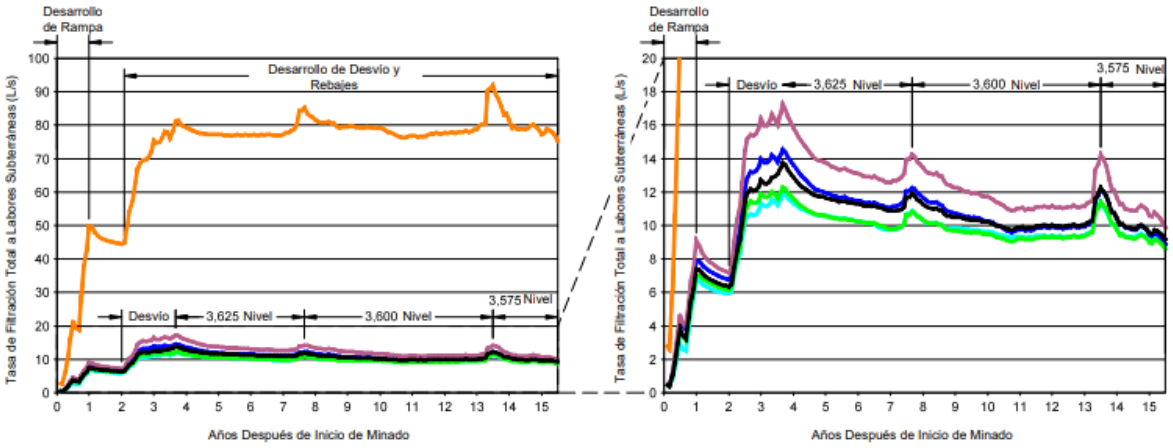
Table 16.24 – Temperatures and Humidity Recordings

	Temperature (°C)	**Humidity** (%)
Maximum	17.9	100.0
Minimum	0.0	26.5
Average	6.0	92.3

16.8.2.2 Groundwater and Rainfall

Groundwater and rainfall data were taken from a report developed by NewFields (2022). An overview of the findings is illustrated in Figure 16.27.

Figure 16.27 – Water Inflow Analysis



Source: NewFields, 2022

Although the groundwater analysis was generated on an older mine design from the 2020 Feasibility Stage, (pre-dating GMD involvement), the following assumptions were applied to the FSUO design:

- Average water ingress of 11 L/s was applied as a base value through the years.
- A maximum inflow of 7.5 L/s was applied during decline development.
- A maximum inflow of 13.8 L/s was applied in full production.

Groundwater inflows are modelled in Ventsim as heat presets, which occur where potential permeable unconformities, such as faults, intercept the mines workings.

16.8.2.3 Environmental Settings

Average daily environment settings used in the Ventsim model are summarised in Table 16.25.

Table 16.25 – Environmental Ventsim Settings

Setting	Value
Surface Datum Dry Bulb Temperature (Average)	6.0°C
Surface Datum Wet Bulb Temperature (Average)	5.5°C
Surface Datum Rock Temperature	4.0°C
Surface Datum Relative Humidity	92.3%
Geothermal Gradient	1.9°C/100 m
Surface Datum Barometric Pressure	60.0 kPa
Air Density	0.80 kg/m³

16.8.2.4 *Friction Factors*

Friction factors used in the Ventsim model are summarised in Table 16.26.

Table 16.26 – Friction Factors

Description	Value (kg/m³)
Decline	0.014
Blasted Airways (Horizontal Development)	0.010
Raise Bored Raises	0.005
Forced Flexible Ducting	0.0033
Escape Raise	0.04

16.8.2.5 *Leakage Factors*

Leakage factors were only applied to ventilation ducting. As leakage is only expected where the ducts are joined, and the drive dimensions provide a suitable clearance above mining equipment, a fixed rate of 50 mm²/m² was applied.

16.8.2.6 *Shock Losses*

A variety of shock losses will likely occur during vent operation, which were modelled as a mixture of fixed and variable values using a combination of calculations and shock loss estimation graphs. Table 16.27 shows the shock losses and their calculated equivalent length values.

Table 16.27 – Shock Losses

Shock Loss	Friction Factor (kg/m³)	Equivalent Length (m)
Entrance to Duct	0.0033	45
Exit of Duct	0.0033	45
Right Angle Bend in Duct	0.0033	18
Junction in Duct	0.0033	50-70
45o Corner in 1.5 m Duct	0.0033	15
45o Corner in 1.2 m Duct	0.0033	11
UG Raise Connections	0.005	200-240
Connections to Surface Raise	0.005	50-75
Surface Connection to Raise	0.005	30
Entrance to Decline	0.014	15

16.8.2.7 *Recommended Threshold Working Limits*

Threshold Limit Values (TLVs) were estimated from values used in international mining jurisdictions and are applicable for an eight (8) hour shift. TLVs are summarised in Table 16.28 and provided as an initial operational guide, which should be reviewed and adjusted accordingly in line with site regulations during operation.

Table 16.28 – Recommended Threshold Limit Values

Contaminant	Limit
CO_2	5000 ppm
CO	30 ppm
SO_2	5.2 mg/m³
NO_x	0.1 mg/m³
NH_3	17 mg/m³
Limestone Dust	0.05 mg/m³
Silica Dust	0.05 mg/m³
Diesel Particulate Matter	0.1 mg/m³
Radiation	<20 mSv/annum (Personnel Exposure)
H_2S	20 ppm

16.8.2.8 *Airflow Requirements Based on Equipment*

International airflow regulations ensure that risks posed by diesel equipment are mitigated in design, which is usually achieved by multiplying the rated diesel engine kW by a statutory airflow requirement. The statutory airflow requirement varies in different mining jurisdictions, with a typical range from 0.047 m³/s per kW to 0.092 m³/s per kW. The Australian standard of 0.05 m³/s-0.06 m³/s was used in this design.

Jumbos and production drills are diesel electric. It is assumed that all drills will be running off electrical power when drilling. Although toxic gases and contaminants are not generated, heat is still produced, which requires movement of air for cooling purposes. A statutory airflow requirement of 0.04 m³/s per kW was used for this design.

Values in Table 16.29 are based on the expected number of equipment to be underground at any one time, thus some equipment being a lower value than the total on site. A 10% adjustment factor has also been utilised to accommodate unforeseen leakage/other losses.

Table 16.29 – Equipment Airflow Requirements

Equipment	Engine Size (kW)	Electric/ Diesel	Equipment Utilisation	Maximum Equipment Number	Airflow Per Vehicle (m^3/s) Including Reqmt	Totals (m^3/s)
Jumbo 2 Boom	135	Diesel-Electric	85%	4	4.6	18.4
Scoop 17.2T	310	Diesel	100%	5	15.5	77.5
Truck 51T	515	Diesel	100%	5	25.8	128.8
Longhole Drill (ITH)	130	Diesel-Electric	85%	1	4.4	4.4
Longhole Drill (TOP HAMMER)	135	Diesel-Electric	85%	2	4.6	9.2
Explosive Truck	110	Diesel	85%	2	4.7	9.4
Cablebolter	75	Diesel-Electric	85%	1	2.6	2.6
Grader	108	Diesel	50%	1	2.7	2.7
Flatbed Trucks	150	Diesel	50%	1	3.8	3.8
Fuel / Lube Trucks	110	Diesel	50%	1	2.8	2.8
Shotcrete Machine	110	Diesel	85%	1	4.7	4.7
Shotcrete Mixer	150	Diesel	85%	2	6.4	12.8
IT Man Basket	105	Diesel	85%	3	4.5	13.4
LV	95	Diesel	50%	16	2.4	38.0
Personal Carrier	110	Diesel	50%	1	2.8	2.8
					Total Airflow	**331**
					Including 10%	**364**

16.8.2.9 Heat Settings for Equipment

Diesel heat outputs were estimated from the expected fuel consumption. BEV heat outputs were estimated from the engine kW rating. Water output was determined either from equipment specifications or where water outside of combustion is added to the environment (e.g., water used during drilling, or the higher than background moisture content of blasted ore).

As production equipment (trucks and LHDs) are generally moving, they were modelled as activity tracks in VentSim. Drills and shotcrete machines were modelled as spot heat sources. Note that heat and water presets are to be measured according to actual vehicle outputs during production and these values optimised within the VentSim model.

Table 16.30 – Heat Settings

Equipment	Engine Size (kW)	Engine Type	Water Output (L/s)	Sensible Heat Output (kW)	Latent Heat Output (kW)
Jumbo 2 Boom	135	Diesel-Electric	1.67	13.5	N/A
Scoop 17.2T	310	Diesel	0.07	225.9	103.4
Truck 51T	515	Diesel	0.09	324.7	148.7
Longhole Drill (ITH)	130	Diesel-Electric	1.50	13.0	N/A
Longhole Drill (TOP HAMMER)	135	Diesel-Electric	6.25	13.5	N/A
Explosive Truck	110	Diesel	N/A	82.5	46.5
Cablebolter	75	Diesel-Electric	1.67	7.5	N/A
Grader	108	Diesel	0.01	81.5	45.8
Flatbed Trucks	150	Diesel	0.01	100.4	56.6
Fuel / Lube Trucks	110	Diesel	0.01	82.5	46.5
Shotcrete Machine	110	Diesel	1.00	82.5	46.5
Shotcrete Mixer	150	Diesel	0.02	98.8	45.3
IT Man Basket	105	Diesel	0.01	70.6	32.3
LV	95	Diesel	0.01	56.5	25.9
Personal Carrier	110	Diesel	0.01	56.5	25.9

16.8.2.10 Dust Management

The ventilation circuit manages suspended dust particles by complying with air velocity limits per drive (Table 16.31). Air velocity management will be conducted by ventilation personnel during production, with adjustment of fans or resistances within drives (regulators, doors, or flaps).

Table 16.31 – Air Velocity Limits

Airway	Maximum Velocity (m/s)
Upcast Shafts	24
Downcast Shafts	24
Haulage Routes	6
Dedicated Escape Raise	12
Working Faces	4
Ducting	N/A

Dust management will also include dust suppression, such as watering down of faces prior to development drilling, regular watering down of the sidewalls and floor with a water truck, or by the installation of a sprinkler system in the backs of drives.

The task that generates the most amount of fine dust is production and development drilling; therefore, mitigation of sulphur dust explosion risks requires fines generated during these tasks to be bogged by an LHD prior to firing. Strict firing windows where no personnel are to be underground and re-entry plans will remove personnel from possible exposure to explosion hazards.

Further management protocols include the supply and use of respirators in mining areas that pose an elevated risk of personnel exposure, and by workforce education of the risks that dust exposure can cause.

16.8.3 VENTILATION REQUIREMENTS FOR DECLINE DEVELOPMENT

16.8.3.1 General Conditions

Excavation of the portal and development of the main decline commences the mine plan in Year -2. A forced air ventilation circuit will be required until the second half of Year -1 when the Central Western Raise is mined. The distance of the decline to the first raise is 1,430 m. Whilst the raise is being installed, development for ore drives and haulage drives will continue, bringing the total distance of ventilation ducting required to 1,620 m until the first main ventilation circuit is established. To cater for this distance of forced ventilation, two (2) 1.5 m diameter ducts were specified for decline development. Fan requirements are discussed in Sections 16.8.3.2 and 16.8.3.3.

The decline will be developed by a contractor work force; it is expected that the contractor company will use diesel trucks and LHDs. However, the diesel equipment to be used are not yet specified; therefore, the ventilation requirements during decline development were estimated from the equipment list at full mine production (Table 16.20). The selected fans for the decline development stage have contingency in speed capacity to adjust airflow requirements and allow for larger vehicles, if required.

16.8.3.2 Ventilation and Fan Requirements in First Year of Operations

By the end of the first year, the decline is expected to reach around 800 m in length. The required airflow in the decline in this period is summarised in Table 16.32. The values are based on estimations of utilisation and airflow requirements of the largest likely equipment.

Table 16.32 – Airflow Requirements in the First Year of Operations

Equipment	Engine Size	Electric/ Diesel	Equipment Requirement	Max Equipment Number	Airflow Per Vehicle Including Requirement	Totals
Scoop 17T	310	Diesel	100%	1	15.5	15.5
Truck 51T	515	Diesel	60%	2	15.5	30.9
					Total Airflow	**46.4**
					Including 10%	**51.0**

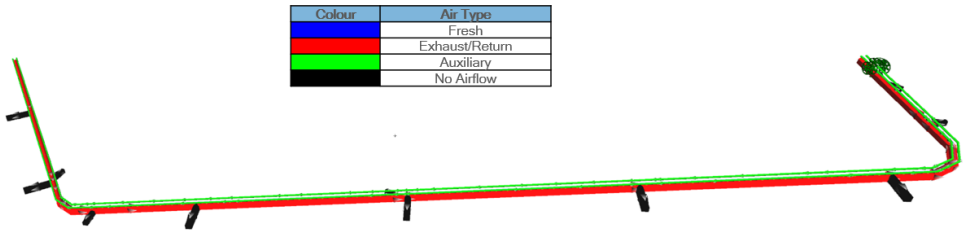
The fans will be installed at the portal for maximum fresh air intake. With the leakage factors modelled for the ventilation ducting, the fans will need to supply a significantly larger volume of air than to ensure the minimum volumetric airflow face requirement of 51.0 m³/s are upheld.

It is estimated with the selected fan type, only two (2) auxiliary fans will need to be required in the first year. The fans will be placed in parallel (i.e., side by side), with each fan providing:

- Airflow – 30 m³/s.
- Pressure - 1,717 Pa.

Figure 16.28 shows the layout of development completed in the first year, with the fans modelled in Ventsim.

Figure 16.28 – Simulated Ventilation During Decline Development in the First Year



Source: Global Mine Design, 2023

16.8.3.3 Ventilation and Fan Requirements in the First Half of the Second Year of Operations

In the first half (H1) of the Year -1, the decline is expected to reach the undercut location of the Central Western Raise. Development of haulage routes and ore drives will continue during raiseboring of the raise. The required airflow in the decline in this period is summarised in Table 16.33. The values are based on estimations of utilisation and airflow requirements of the largest likely equipment.

Table 16.33 – Airflow Requirements in H1 of the First Year

Equipment	Engine Size	Electric/ Diesel	Equipment Requirement	Max Equipment Number	Airflow Per Vehicle Including Requirement	Totals
Scoop 17T	310	Diesel	100%	1	15.5	15.5
Truck 51T	515	Diesel	60%	2	15.5	30.9
Jumbo 2 Boom	135	Diesel - Electric	85%	4	4.6	18.4
Cable Bolter	75	Diesel - Electric	40%	1	1.2	1.2
					Total Airflow	**66.0**
					Including 10%	**72.6**

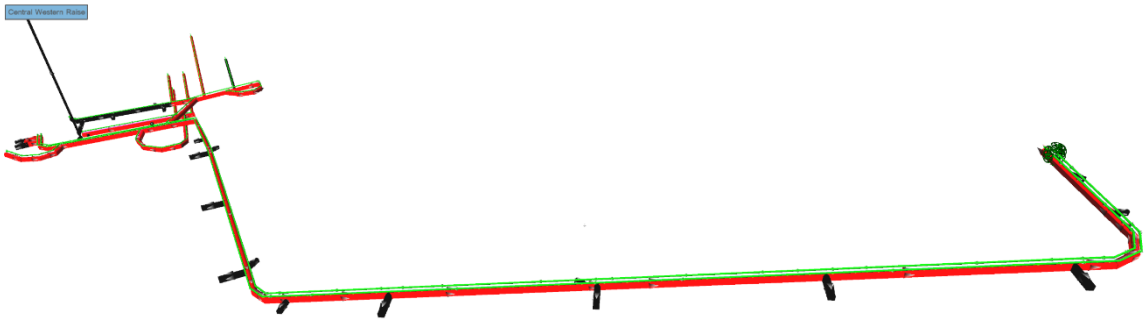
The same fan arrangement used in the first year of operations will be continued. Depending on the fan selected, a second fan in series may need to be installed on each fan to overcome the expected resistances and supply the required airflows.

Considering leakage and shock factors, modelling suggests that each fan should have a maximum supply of:

- Airflow – 40 m³/s.
- Pressure – 2,950-3,560 Pa.

Figure 16.29 illustrates the expected development in H1 of the second year. The Central Western Raises is being bored but is not broken-through.

Figure 16.29 – Simulated Ventilation in H1 of Year -1



Source: Global Mine Design, 2023

With development continuing past the ventilation raise location, ventilation duct management is required to ensure that the correct air quantities are supplied to working faces. The airflow

requirement delivered to the working faces is to be depicted by the equipment operating at the face, i.e., if a LHD is mucking a heading, at least 17.1 m³/s (including 10% contingency) is to be supplied.

16.8.4 Main Circuit Changes over the Life of Mine

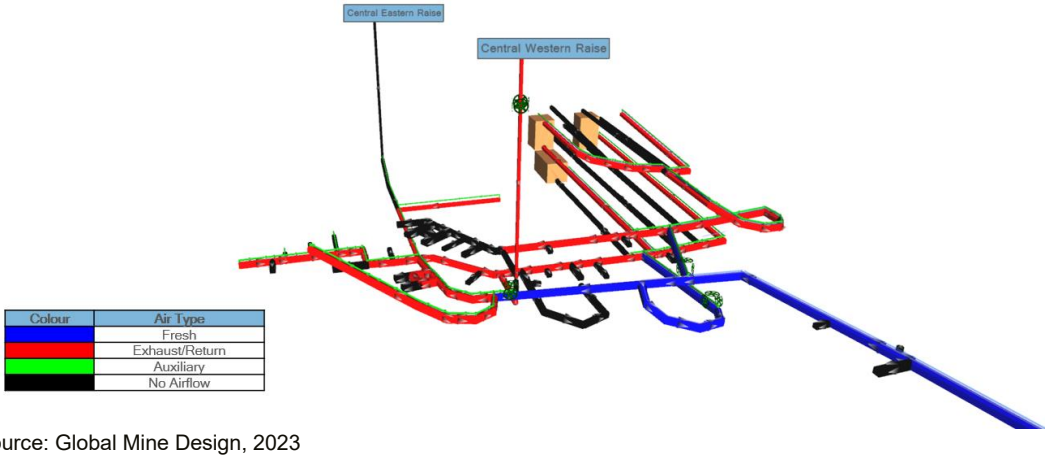
16.8.4.1 General Conditions

The main ventilation circuit comprises exhaust ventilation via surface fans that draw ("pull") fresh air down through the decline and fresh air raises. The full circuit uses four (4) ventilation shafts, which change between exhaust and fresh air raises over the life of mine as production continues and mining zones open and close. The following sections summarise the key changes in the main ventilation circuit over the mine life.

16.8.4.2 Central Western Raise - second half of Year -1

Following completion of the Central Western raise, exhaust fans will be placed on top of the raise to commence the first iteration of the main ventilation circuit. The circuit in this period is illustrated in Figure 16.30 (the Central Eastern Raise is being developed but is not broken through).

Figure 16.30 – Simulated Ventilation in the Second Half of Year -1



Source: Global Mine Design, 2023

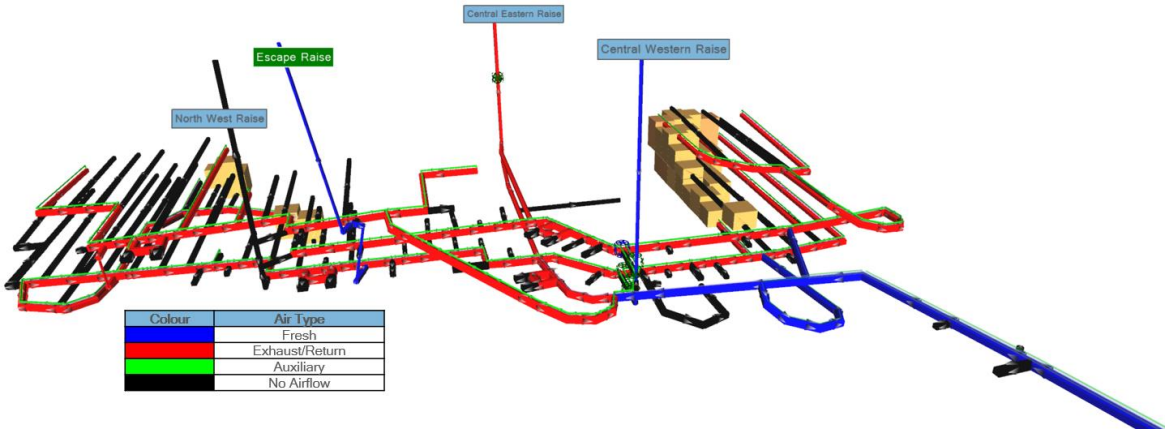
The Central Western Raise is expected to be operational prior to the start of stoping operations. The auxiliary fans used during decline development will be re-distributed around the mine to ensure correct airflows are supplied to development and production headings. The total exhaust for this period is 170 m³/s, otherwise the air velocity limit of the raise (24 m/s) would be exceeded.

16.8.4.3 Central Eastern Raise – Year 1

The Central Eastern Raise will breakthrough in Year 1, after which the surface fans from the Central Western Raise will be installed. The fan location will move the exhaust circuit to the Central Eastern

Raise and will convert the Central Western Raise to a Fresh Air Intake. The circuit in this period is illustrated in Figure 16.31 (the North West Raise is being developed but is not broken through).

Figure 16.31 – Simulated Ventilation in Year 1



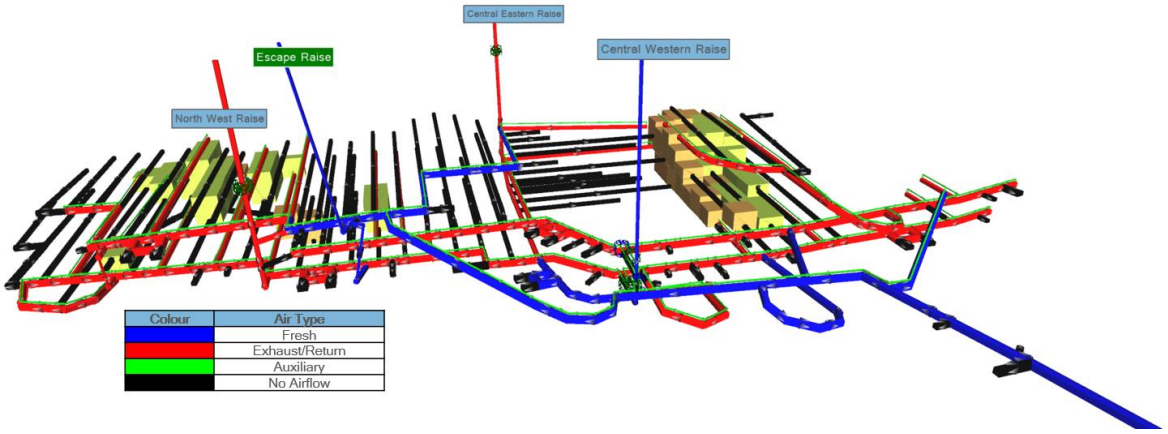
Source: Global mine Design, 2023

Auxiliary fans are to be positioned within bulkheads in the Central Western Raise ventilation drive, ensuring that no recirculation occurs and to supply fresh air to development and operational headings. Due to air velocity constraints within the raise, a maximum of four (4) auxiliary fans will be able to operate at the same time. The total exhaust airflow for this stage is to increase to 205 m³/s.

16.8.4.4 North West Raise – Year 2

After the North West Raise is established, the second set of surface fans will be installed on top of it, comprising two (2) fans in parallel in the same configuration used for the previous raises. The ventilation circuit now comprises two (2) exhaust raises (North-West and Central Eastern raises) and two (2) intakes (Central Western raise and the decline). This set-up is to remain the same until Year 5 when the South-East Raise is established. The circuit in this period is illustrated in Figure 16.32 (the Escape Raise is considered a separate ventilation system).

Figure 16.32 – Simulated Ventilation in Year 2



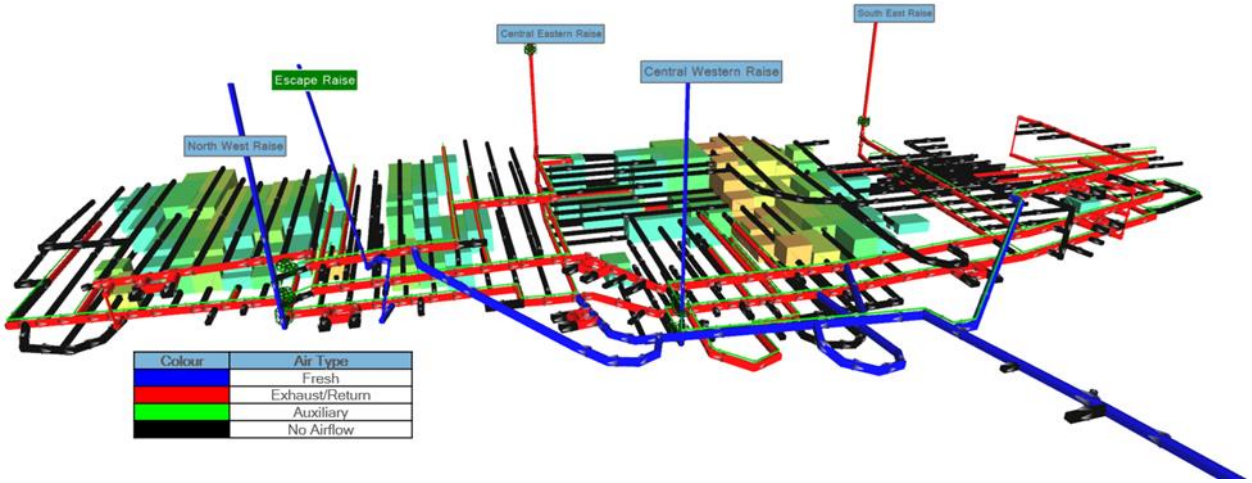
Source: Global Mine Design, 2023

Auxiliary fans placed in bulkheads in the Central Western Raise ventilation drives will remain, but the number of auxiliary fans will increase to two (2) auxiliary fans within each bulkhead, increasing the total number of auxiliary fans to six (6). The air velocity constraints remain, therefore only four (4) fans can operate at the same time. The total exhaust airflow increases to 372 m³/s as production ramps up to maximum rates.

16.8.4.5 South East Raise – Year 5

After the South-East Raise is established, the surface fans are relocated from the North-West raise, positioning both exhaust raises on the eastern side of the orebody. The three (3) main intakes situated on the western side of the orebody comprise the Decline, Central Western and North-West raises. The circuit in this period is illustrated in Figure 16.33.

Figure 16.33 – Simulated Ventilation in Year 5



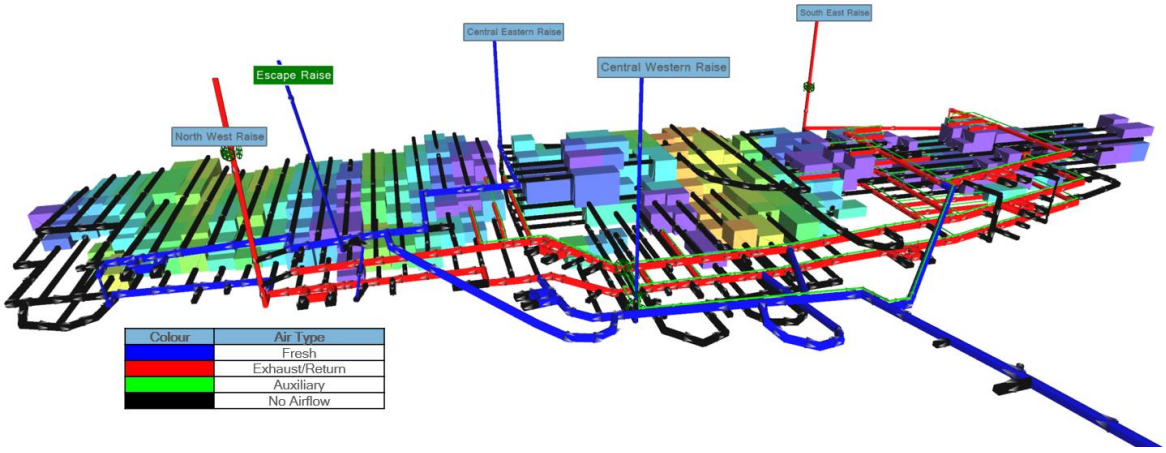
Source: Global Mine Design, 2023

Three (3) auxiliary fans are to be removed from each ventilation access on the Central Western Raise and installed within bulkheads in the North-West raise ventilation drives. This will allow for fresh air to be separately supplied to the North and South areas of the orebody. The total exhaust airflow for this period will be 375 m³/s.

16.8.4.6 *Late Production Stages – Year 9*

Mining of stopes on the access of the Central Eastern ventilation drives is to have stopes extracted and backfilled on the 1st and 2nd levels, increasing resistance significantly stalling the fans. It is proposed that the fans on this raise are to be repositioned to the North-West Raise allowing for the Central Eastern Raise to be used as a fresh air raise. The circuit in this period is illustrated in Figure 16.34.

Figure 16.34 – Simulated Ventilation in Year 9



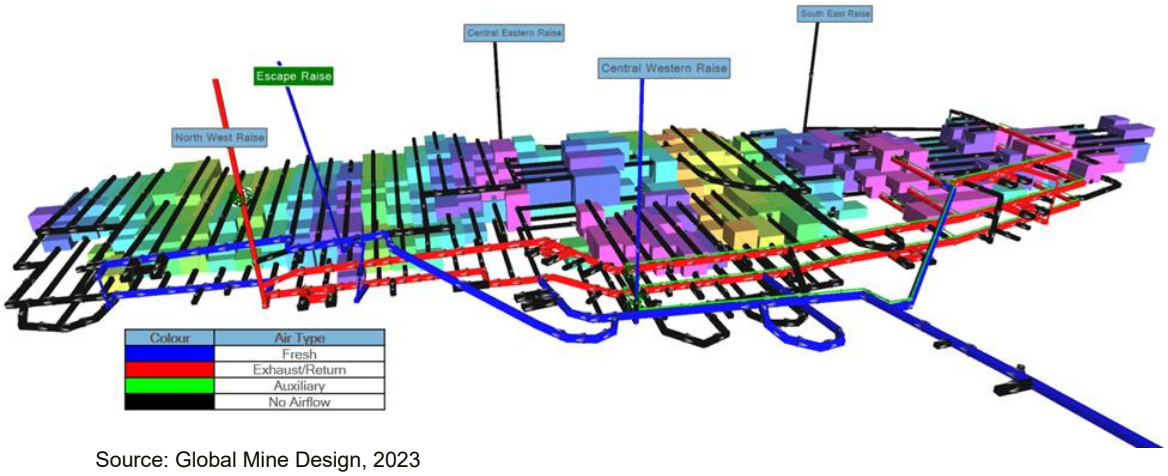
Source: Global Mine Design, 2023

Auxiliary fans situated in bulkheads in the North-West ventilation drives are to be removed, with one of the fans being repositioned to the bulkhead in the western ventilation drive of the Central Western ventilation raise. The other two (2) fans are to be utilised as contingency. This arrangement will the remain the same for the final two (2) years of the mines production. The total exhaust of the mine is to be 370 m³/s.

16.8.4.7 *Late Production Stages – Year 10*

Late-stage stoping will result in the Central Eastern and South East ventilation drives being mined as stopes and subsequently backfilled. For this period, exhaust surface fans remaining on the South East raise will be removed, and be used as contingency for the remaining life of the Project. Fresh air is to be only supplied by the decline and Central Western Raise. The circuit in this period is illustrated in Figure 16.35.

Figure 16.35 – Simulated Ventilation at End of Mine Life



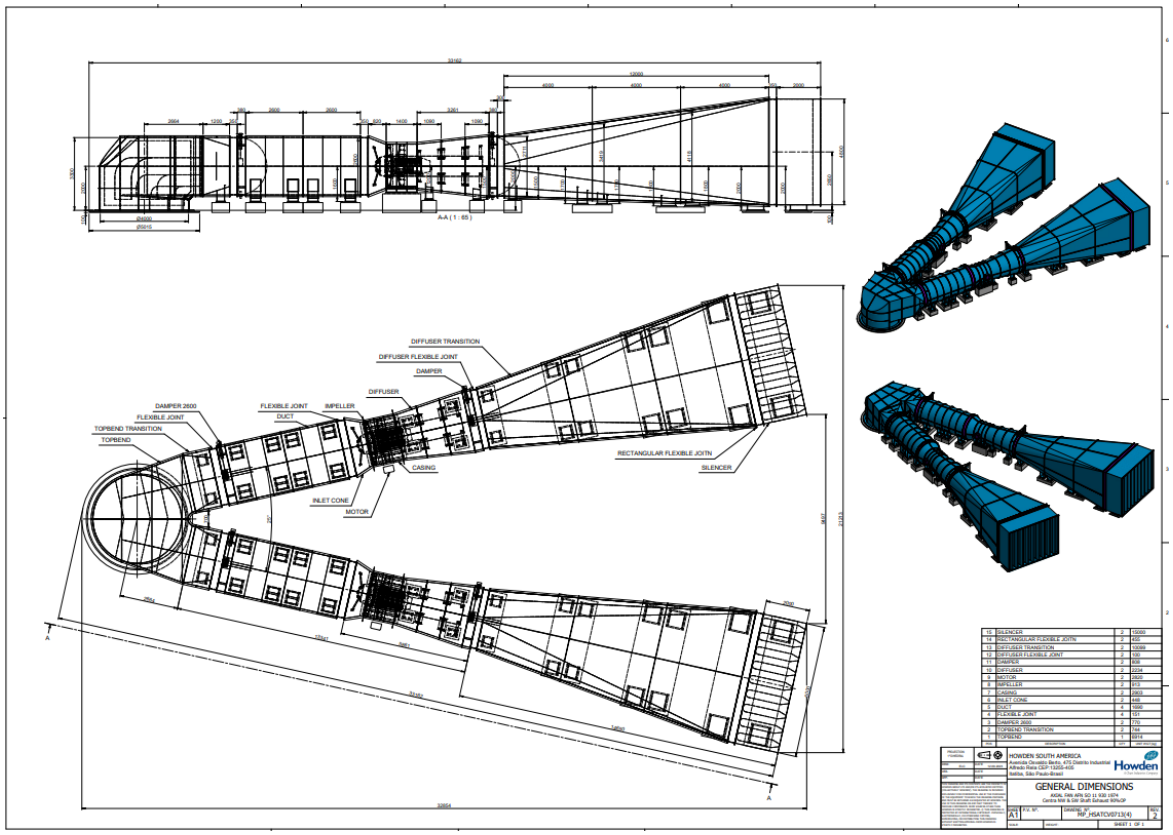
Source: Global Mine Design, 2023

Utilisation of surface fans on the North West Raise produces a total airflow of 230 m³/s.

16.8.5 Main Fan Specification

Each installation of surface exhaust fans will comprise two (2) fans in parallel, allowing for contingency during maintenance and any unexpected period of downtime. The General Arrangement of the surface fan layout is shown in Figure 16.36.

Figure 16.36 – General Arrangement Drawing for Surface Exhaust Fans



Source: Global Mine Design, 2023

Axial fans were recommended for Loma Larga, due to the following reasons:

- Axial fans generally have lower capital costs, as well as tend to be easier to conduct maintenance on.
- Duty of the fans can be varied easier by the adjustment of the fans blade angles.
- Architecture of axial fans allows for easier silencing if required.

From the variations modelled in Ventsim, the chosen axial fans are Howden AFN SO 11 930 1974. Each fan set will be installed with a variable speed drive allowing for the fans exhaust to be easily adjusted with fan speed. Table 16.34 illustrates the fans designed parameters.

Table 16.34 – Proposed Main Fan Design Parameters

Parameter	Design Value
Density	0.80 kg/m³
Flow Rate per Fan	110 m³/s
Flow Rate in Shaft Collar	220 m³/s
Fan Total Pressure with Loss	1,407 Pa
Total Pressure in Shaft Collar	-926 Pa
Rotation	1,185 rpm
Power Consumption per Fan	235 hp
Efficiency	89.3%
Work Regime	Continuous
Noise Level	<39 dB, at 1 km with silencer
Electric Motor	250 hp / 6 phase / 60 Hz

Table 16.35 summarises the surface fan requirements of the mine life.

Table 16.35 – Main Surface Fan Movements and Installations

Year	Number of Main Exhaust Fans	Information
Y-1 (H1)	0	Forced ventilation via auxiliary fans during decline development. Installation of fans and fan housing on Central Western Raise. This is required so that a main circuit is established prior to extraction of stopes.
Y-1 (H2 onwards)	2	Fans in parallel above the Central Western Raise Movement of surface fans from Central Western Raise (3.0 m dia) to Central Eastern Raise (4.0 m dia) once the raise has been bored. The housing of the fans will need to be adjusted to the diameter change. To ensure there is no delays to production during this movement, purchase of two (2) more surface fans as well as surface infrastructure may be required instead of a movement.
Y1	2	Movement of fans to Central Eastern Raise Installation of fans and fan housing on the North-West Raise once the raise has been bored. If additional fans were purchased in Y2 then the fans and infrastructure removed from the Central Western raise can be then reused here.

Year	Number of Main Exhaust Fans	Information
Y2	4	Addition of fans in parallel above North-West Raise Movement of fans from the North-West raise to the South-East raise, once the South-East raise has been bored. This ensures that contaminants produced from stopes in the southern part of the orebody are exhausted straight to surface rather than through haulage routes.
Y5	4	Movement of fans from North-West Raise to South-East Raise Movement of fans from the Central Western Raise to the North-East Raise. This is required due to stopes being taken on the ore drives attached to the Central Western Raise.
Y9 (H2 onwards)	2	South-East Raise blocked by production stoping. Fans moved back to North-West Raise Fans are to be removed from the South-East raise due to stopes being taken on ore drives connected to the raise.

Movements of surface fans over the mine life is imperative to ensure that a main ventilation circuit is always established, as well as contaminants are exhausted away from working areas (particularly where mucking and loading are occurring). Logistics of these moves are recommended to be well planned to ensure that there are no delays to production.

16.8.6 Auxiliary Fan Specifications

The airflow requirements are used to select the fan properties necessary to deliver the required airflow to the working faces at each design stage. This is done by an iterative process of calculating the expected pressures through ducting, choosing a suitable fan, modelling it in Ventsim and adjusting either fan or ducting to suit until the minimum requirements are met.

The final design used Minetek 280 kW Raptor auxiliary fans.

Two (2) auxiliary fans will be used to provide forced ventilation during development of the decline. These fans will then be re-used as auxiliary fans during mine production, with another four (4) purchased for the same purpose. In full production, the auxiliary fans will be placed in bulkheads in the fresh air ventilation drives to ensure no recirculation of contaminants occurs and that fresh air is supplied to working faces.

Each fan will be installed with a variable speed drive to ensure that the fan can be adapted to varying resistances with increasing lengths of ducting.

16.9 Underground Service Facilities

16.9.1 Maintenance

Most of the repair services on mobile underground equipment will be done on the surface. The maintenance shop will be built near the plant and office facilities. The underground mobile equipment will be on a maintenance schedule to minimise equipment breakdowns during work. Only emergency repairs will be done underground on mobile equipment. A small underground maintenance bay will carry minimal parts inventory to quickly respond to equipment breakdowns. Also, the fuel and lube truck will be able to fill up the hydraulic systems underground when needed.

The underground shop will carry spare pumps, pump motors, auxiliary fans, and motors, in case of a breakdown. The pump and fans will be replaced underground and serviced in the surface shop. The pumps will also be on a maintenance schedule and brought to the surface shop on a regular basis. The same principle will apply for the ventilation fans. Spare fans will be kept in inventory for all motor fans.

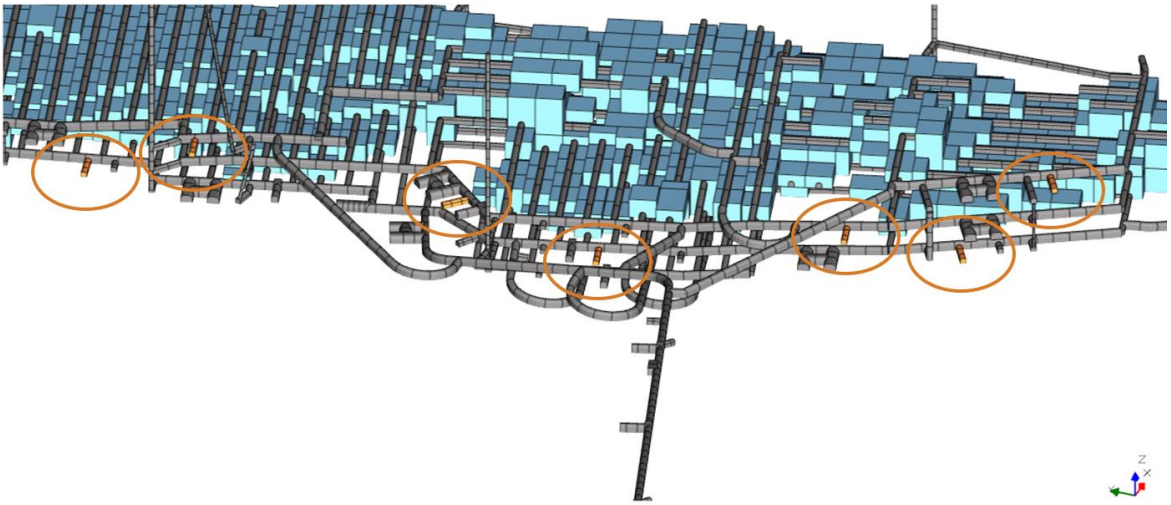
16.9.2 Explosive Magazines

The explosive storage magazines will be located on surface 1 km from any infrastructure. One storage magazine will be for the detonators and the other for packaged emulsion, boosters, and detonating cord. The magazines will be purchased from the explosive supplier. The mobile explosives loader will take the explosives directly from the magazines. The blasting crew will be responsible for recording the explosives taken in the inventory logbooks located in each magazine. Any and all unused explosives will be returned to the surface magazines and recorded in the logbooks. The site superintendent will be responsible for arranging regular powder magazine inventory as required by regulations.

16.9.3 Refuge Stations and Sanitary Facilities

Portable self-contained refuge stations will be purchased and installed to follow the lateral development. The portable refuge will primarily be installed near a fresh air intake and connected to the compressed air system coming from the surface. Seven (7) main refuge stations at 20 people per unit are planned for the LOM operation, as shown in orange in Figure 16.37. The refuge stations are designed to accommodate the necessary number of workers at any given time. Portable toilets will be placed at strategic locations close to advancing lateral development.

Figure 16.37 – Refuge Stations Location Overview



Source: Global Mine Design, 2023

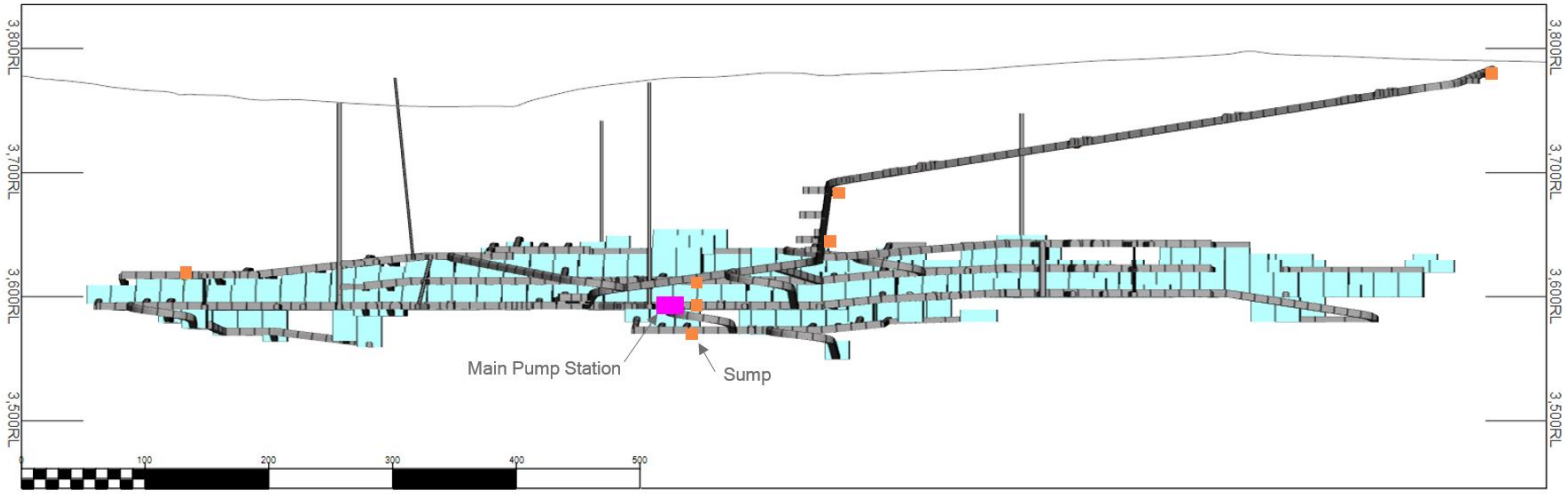
16.9.4 MINE DEWATERING

The dewatering capacity is designed at 22.0 L/s and was estimated based on the water usage of the planned equipment and dust control activities when the mine production reaches its peak. The groundwater inflow study for the mine life has been conducted by Itasca. The study indicates that during the ramp excavation the water inflow will be 7.5 L/s (Itasca, 2018a and 2018b). As the Project reaches full production, a peak water inflow of 13.8 L/s is anticipated due to the presence of the two (2) main ventilation raises and several open stopes, as well as a significant amount of level development (Itasca, 2018b).

During the ramp development phase, a sump will be excavated every 400 m to collect ground water running down the ramp. Also, a face pump will collect drilling water to convey it to the nearest sump. These sumps will settle the biggest solid particles and the wastewater will get pumped from one sump to the other all the way to the surface settling pound using a series of 15 HP submersible pumps through a schedule 10, 100 mm diameter discharge line.

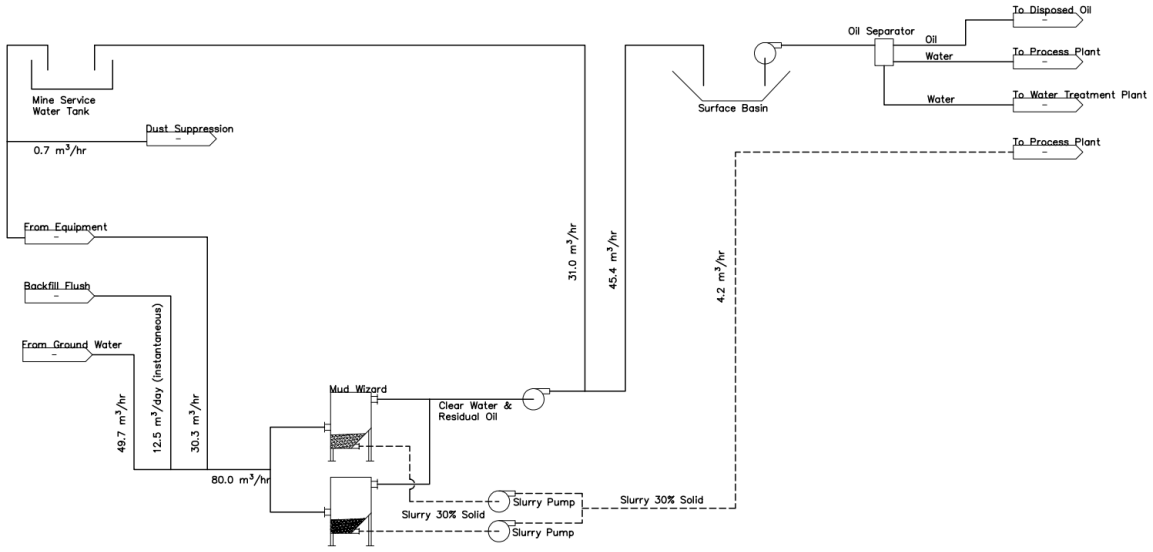
Permanent dewatering is to be conducted with a system of drain holes and submersible pumps which direct mine contact water to the main dewatering station. The main dewatering station will be located at the bottom of the main ramp, on the south side. Solid-liquid separation will be accomplished by feeding the collected mine contact water through two (2), parallel 8-foot Mudwizard SLS banks. The resulting clear water will be used for the underground service water, the excess will be pumped to the surface contact water pond using a 300 HP centrifugal pump through a schedule 10, 150 mm discharge line installed in the ramp. Figure 16.38 illustrates the dewatering strategy for the Project. Figure 16.39 shows the mine dewatering Process Flow Diagram.

Figure 16.38 – Dewatering Network



Source: Global Mine Design, 2023

Figure 16.39 – Mine Dewatering Process Flow Diagram

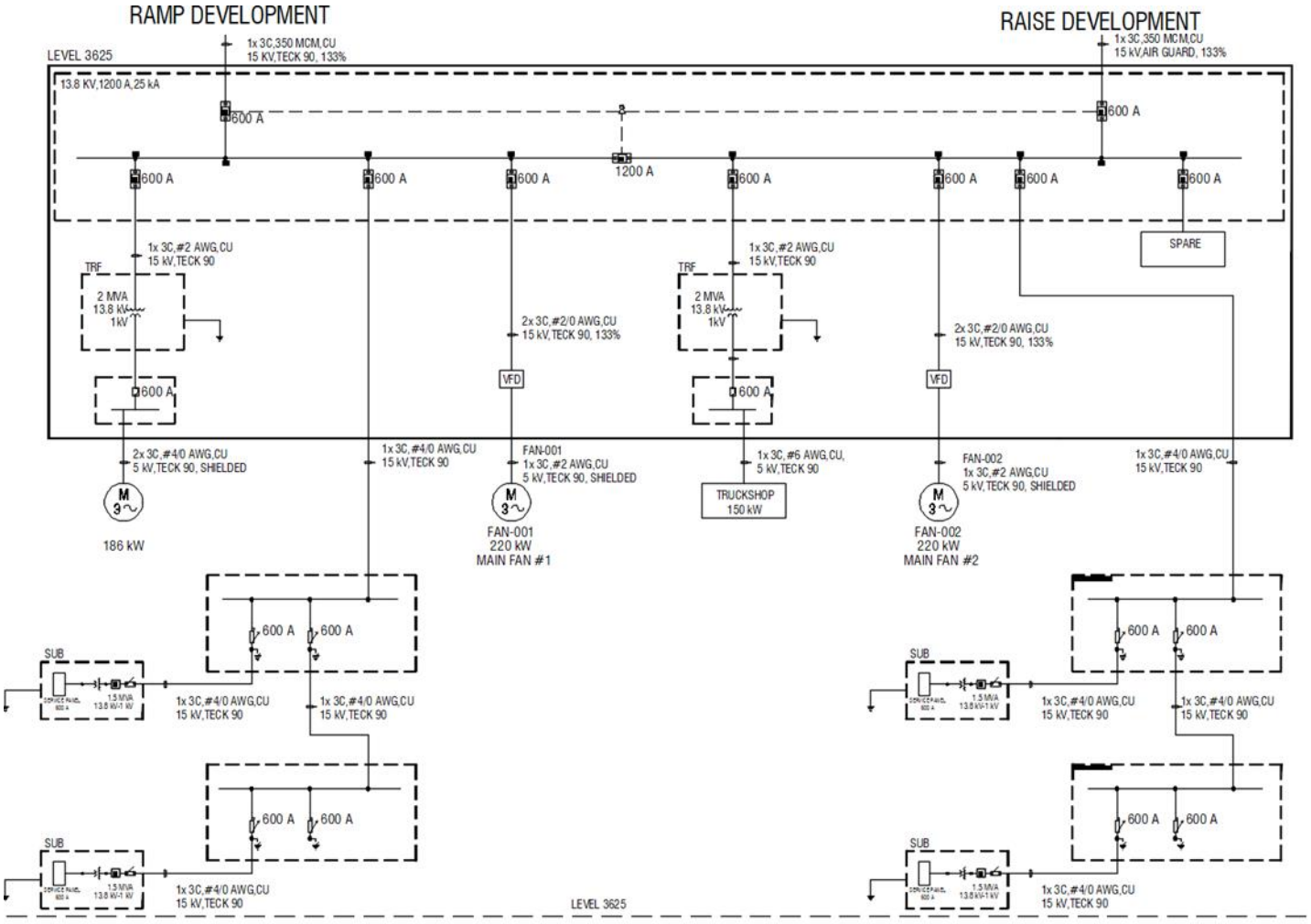


Source: Global Mine Design, 2023

16.9.5 UNDERGROUND MINE ELECTRICAL DISTRIBUTION

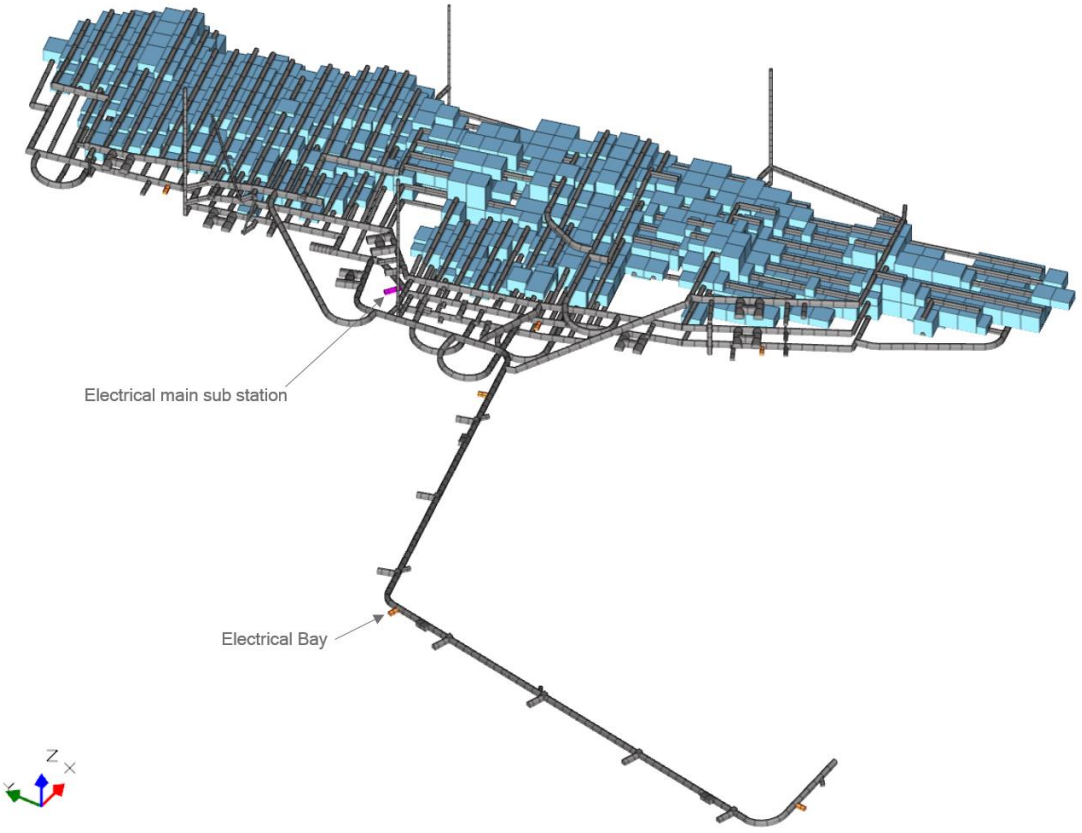
The objective of having sufficient electrical distribution is to provide the safest possible environment for the people working at Loma Larga, which will also create the best conditions for mobile equipment operators trying to perform their daily tasks. To satisfy the electrical requirements of Loma Larga, an electrical substation is required. The underground power requirement averages 2.1 MW with a total connected load of 4.4 MW. Figure 16.40 shows the Electrical Single Line Diagram. To provide power to the mine, a mining substation will be established near the mine entry from which, two (2) 13.8 kV feeds (one from the ramp, one from the raise) will be routed to the main underground switchroom/substation. Mine load centres will step the voltage down from 13.8 kV to 1000 V, as shown on Figure 16.41.

Figure 16.40 – Electrical Single Line Diagram



Source: Global Mine Design, 2025

Figure 16.41 – Underground Electrical Infrastructure Location



Source: Global Mine Design, 2023

16.9.6 UNDERGROUND COMMUNICATION

Underground communications will be enabled by a fibre optic backbone. Based on the reliability and performance requirements, an LTE network is recommended for the underground communication system. LTE antennas, repeater and leaky feeder systems will be deployed as the mine progresses. A private LTE network will be used to transmit data, video, and voice communication from underground to the mine surface as basic data, but also do fleet management and supervision. It can also be used to remotely operate fans, pumps, and the blasting system. It can connect to cameras and can be used for people and machine tracking and voice communication. Where appropriate, some data collection systems will employ more robust, portable network.

16.10 Paste Backfill

Cemented Paste Backfill (CPB), Cemented Rock Fill (CRF) and Rock Fill (RF) will be placed within the mine voids to manage closure, stabilise stope spans, and minimise waste haulage to surface. Approximately 93% of the void volume will be filled with either paste or cemented fill and the remaining 7% will be filled with waste rock.

Backfill stability is calculated using Mitchell formula (Mitchell, 1991), with a factor of safety of 1.5. The strengths and cure times for the entire paste columns are described in Table 16.36 according to the mine design (longitudinal, transverse or sill stopes) and mining sequence (retreat within same panel or retreat from neighbour panels of primary and secondary). The cure times for stopes mined through the same panel is 28 days while it is 180 days for secondary stopes mined next to primary stopes.

Table 16.36 – Strengths and Cure Times Requirements

Dimension Type	Mine Design	Stope Length (m)	Stope Width (m)	Stope Height (m)	Strength Requirement for Same Panel Stopes (kPa)	Strength Requirement for Secondary / Primary Stopes (kPa)
Average	Longitudinal	20	22	19	281	268
	Transverse	20	20	20	279	280
Minimum	Longitudinal	10	20	10	186	140
	Longitudinal	20	10	10	140	186
	Transverse	10	20	10	140	186
Maximum	Longitudinal	20	20	30	335	335
	Longitudinal	20	30	20	335	279
	Transverse	20	20	35	356	356

Dimension Type	Mine Design	Stope Length (m)	Stope Width (m)	Stope Height (m)	Strength Requirement for Same Panel Stopes (kPa)	Strength Requirement for Secondary / Primary Stopes (kPa)
	Transverse	30	20	25	381	310
Plug Only	Sill	NA	NA	NA	350	350

A summary of the paste recipes to achieve the target strength for the various strength requirements are summarised in Table 16.37. Tight filling of backfill is important for long term stability. Stopes should be backfilled as soon as reasonably practicable and should be tight filled to minimise potential for instability.

Table 16.37 – Target Strengths and Estimated Binder Requirements

Stope Type	Strength Requirement (kPa)	Cure Time	W:C Ratio	% Binder @ 77.5%m Solids Concentration
Stope of Average Dimensions	280	28 days	9.2	3.2
Stope of Minimum Dimensions	140	28 days	14.3	2.0
Stope of Maximum Dimensions	380	28 days	7.6	3.8
Stope under Paste (Sill)	350	28 days	8.0	3.6
Plug only	350	3 days	5.0	5.8

The rate of pour will be balanced against the rate of cure to limit the risk of liquefaction. Additionally, a non-continuous filling strategy will be used, where an initial pastefill plug will be poured, then a cure period of three (3) days will occur before the main pour to fill the remaining stope void. The plug corresponds to a volume of paste poured to a height of 2 m above the ore drift level. Waste berms are found to be sufficient to hold the paste as barricades, with non-continuous paste filling.

Backfill comprises between 2% to 3.8% of cement, depending on the above exposure requirements. At this stage, only paste strengths for the entire paste columns have been calculated, and the average strength has been applied to estimate the operational costs of paste filling.

The production of paste from the surface Tailings Dewatering and Paste Plant is described in Section 18. After the paste is produced in the plant it is pumped overland and delivered underground via two (2) surface boreholes located directly above the orebody (Figure 16.42 and Figure 16.43).

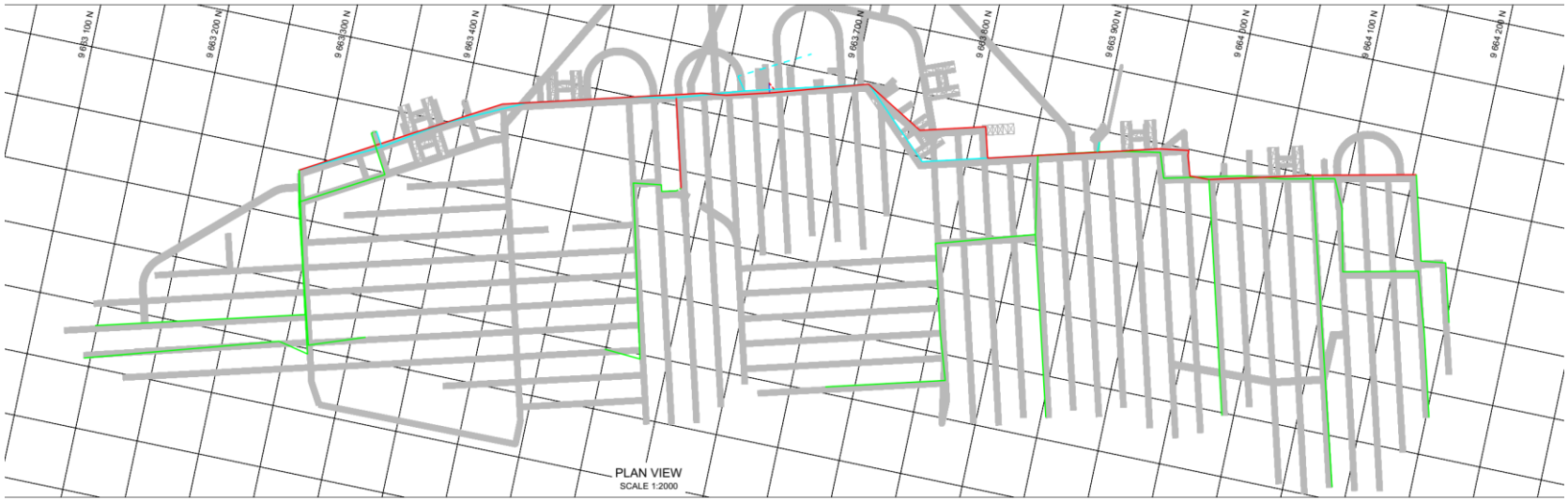
The surface boreholes were designed with a dip between 65° and 70°. The second borehole will be constructed as backup, which is considered good practice.

Once underground, distribution of the paste to the various working areas will be accomplished by switch-overs from the main trunk line(s) to the level piping and eventually to the stope piping, by manual switch-overs. A six-inch pipe size has been specified for the underground distribution system. The system will operate at a nominal pumping pressure of 85 bar to the various deposition points.

Instrumentation will be installed in key locations to report pressure data back to the plant operator. Manual valves will be installed just before the stopes to allow for diversion of flush water away from the stope to a containment area provisioned for by the mine.

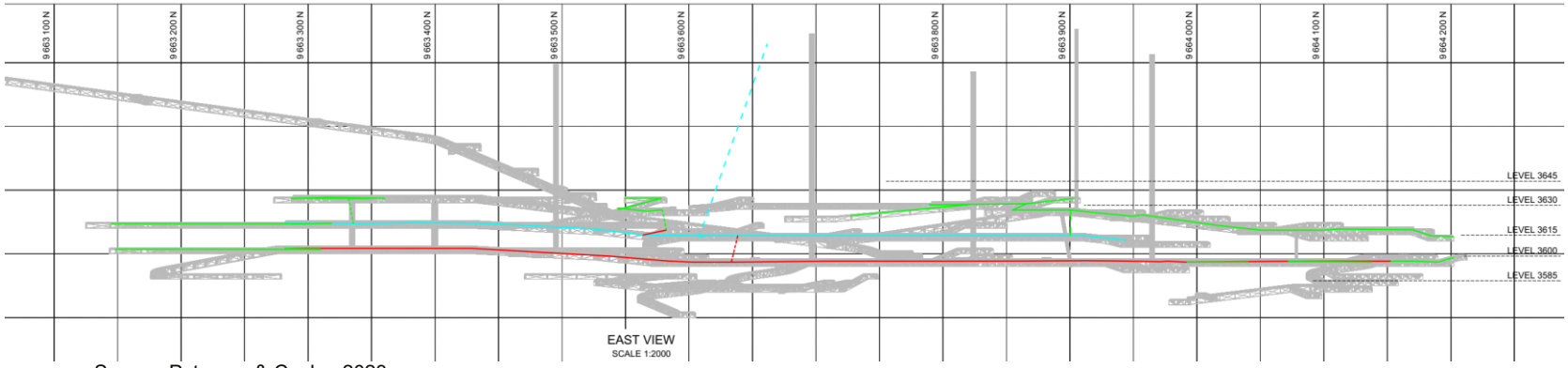
This underground distribution system will be flushed with water before and after each pour, namely pre-flush and post-flush. Pre-flush is used to check for blockage in the system, to confirm the routing is correct and to wet the lines so the paste maintains its moisture content as it travels down the system. This flush water will be diverted away from the stope and will report to the mine dewatering sumps.

Figure 16.42 – Underground Paste Distribution System (Plan View)



Source: Paterson & Cooke, 2023

Figure 16.43 – Underground Paste Distribution System (Elevation View)



Source: Paterson & Cooke, 2023

17 RECOVERY METHODS

17.1 Introduction

The process plant design is derived from data and design criteria provided by DPM, DRA, vendor data, testwork and regulatory permitting requirements. These data and criteria provide the basis for the calculations and design criteria derived from the mass balance.

The crushing and grinding circuit design is based upon the design throughput requirements and ore competency and hardness. The SAG mill sizing is based on achieving the grind size required to obtain the required flotation performance as well as the outcomes of the metallurgical testwork and mill simulations.

The design of the flotation circuit is based upon the testwork results of the 2017-2018 SGS testwork program. The 2022 and 2023 testwork programs were designed to confirm the results from the 2017-2018 testwork and update the flowsheet to one incorporating Jameson cell technology. The results of the 2017-2018 locked cycle testwork and 2022 and 2023 programs provided the basis for recovery and grade calculations, and enabled the sizing of the Jameson cell flotation circuit equipment.

The concentrates are dewatered using conventional plate and frame pressure filtration. The filtration circuit design is based on common design practices for the concentrate as no testwork data are yet available. The tailings dewatering is outside of the DRA scope and is addressed by Paterson & Cooke and NewFields for the tailings facility design.

For other equipment items, the peak production rates were used for sizing. For equipment influenced by volumetric flow, the sizing requirement is based off the requirement for 120% of the instantaneous flow.

There are three (3) types of water defined for the process: process water, fresh water, and gland seal water. The process water is recovered as an overflow from the concentrate thickeners, and final tailings thickener. In addition, process water circuit receives water from the mine, and fresh water for make-up.

Fresh water is sourced from the local environment and gland seal is produced by filtration from the freshwater tank. Plant fire protection system uses fresh water from the plant freshwater tank. Potable water for the plant is sourced from the plant fresh water source and is treated within the plant envelope as per the local regulations requirement.

17.2 Process Design Criteria

Table 17.1 summarises the Process Design Criteria (PDC) to process a nominal 3,000 t/d of ROM ore from an underground mine (increasing to 3,400 t/d in Year 2).

The plant will produce separate gold-copper (Au-Cu) and pyrite gold concentrates using a sequential conventional sulphide flotation flowsheet. Plant tailings will be pumped to Paterson & Cooke's battery limit for thickening, filtration, and production of the paste backfill.

Table 17.1 – Process Design Criteria Summary

Description	Unit	Nominal	Design
Feed Grades		LOM	85th percentile
- Cu	%	0.29	0.46
- Au	g/t	4.70	6.50
- Ag	g/t	28.57	36.92
- S_{total}	%	5.55	6.08
Run Time			
- Crushing Circuit	%	75	
- Concentrator	%	92	
Ore Hardness			
- Crushing Work Index (CWi)	kWh/t	11.5	
- Abrasion Index (Ai)	-	1.08	
- Bond Ball Mill Work Index (BWi)	kWh/t	18.5	
Crushing			
- Installed Power	kW	132	
- Feed Size F_{80}	mm	360	
- Product Size P_{80}	mm	127	
- Crushed Ore Stockpile (Live)	t	3,400	
Grinding			
- Mill Type	-	Single SAG mill, VFD drive	
- Mill Dimensions	-	6.9 m L x 6.9 m EGL	
- Installed Power	kW	5500	
- Circulating Load	%	330	
- Primary Grinding Product P_{80}	µm	75	

Description	Unit	Nominal	Design
Gold-Copper Concentrate Production			
- Rougher Conditioner Time	min	5.0	
- Rougher Cell Type/Model	-	Jameson Cell / E3432/8	
- Rougher Cell – Fresh Feed Design Flow	m³/h	498.9	
- # Rougher Cells	-	1	
- Rougher Scavenger Cell Type/Model	-	Jameson Cell / E3432/8	
- Rougher Scavenger Cell – Fresh Feed Design Flow	m³/h	497.1	
- # Rougher Scavenger Cells	-	1	
- Rougher Concentrate Regrind P_{80}	µm	30	
- # of Cleaning Stages	-	1	
-Cleaner Cell Type/Model	-	Jameson Cell / E1714/2	
- Cleaner Cell – Fresh Feed Design Flow	m³/h	49.2	
- # of Cells per Stage	-	1	
- Cleaner-Scavenger Cell Type/Model	-	Jameson Cell / E1714/2	
- Cleaner-Scavenger Cell – Fresh Feed Design Flow	m³/h	30.5	
- # of Cells per Stage	m³/h	1	
- Copper Concentrate Thickener Settling Rate	t/h/m²	0.14	
Pyrite Concentrate Production			
- Rougher Conditioner Time	min	5.0	
- Rougher Cell Type/Model	-	Jameson Cell / E3432/8	
- Rougher Cell – Fresh Feed Design Flow	m³/h	510.7	
- # Rougher Cells	-	1	
- Rougher Scavenger Cell Type/Model	-	Jameson Cell / E3432/8	
- Rougher Scavenger Cell – Fresh Feed Design Flow	m³/h	504.1	
- # Rougher Scavenger Cells	-	1	
- Rougher Concentrate Regrind P_{80}	µm	30	
- # of Cleaning Stages	-	2	
- 1st Cleaner Cell Type/Model	-	Jameson Cell / E4232/10	
- 1st Cleaner Cell – Fresh Feed Design Flow	m³/h	208.5	
- # of Cells per Stage	-	1	
- 1st Cleaner Scavenger Cell Type/Model	-	Jameson Cell / E1714/2	

Description	Unit	Nominal	Design
- 1st Cleaner Scavenger Cell – Fresh Feed Design Flow	m³/h	116.9	
- # of Cells per Stage	-	1	
- 2nd Cleaner Cell Type/Model	-	Jameson Cell / E4232/10	
- 2nd Cleaner Cell – Fresh Feed Design Flow	m³/h	108.6	
- # of Cells per Stage	-	1	
- Pyrite Concentrate Thickener Settling Rate	t/h/m²	0.11	

17.3 Production Summary

An integrated mine and process plant concentrator plan was developed with the following considerations:

- The process plant nominal throughput rate is 3,000 t/d and will increase to 3,400 t/d in Year 2, after the commencement of mining.
- During Years 10 and 11, the throughput reduces to 3,000 t/d as the quantity of the available processable ore is reduced.
- A pre-production stockpile is created during the mining development and ramp up period such that once the process plant commences operations, there will be no periods where plant operations will have to cease due to insufficient ore supply from the mine.
- The process plant will be operational in Month 1 of Year 2 LOM after the commencement of mining, this is defined as Year 1 of the concentrator production refer to the Table 17.2.
- The process plant has a throughput ramp up period of six (6) months and a metal recovery ramp up period of 11 months.

Table 17.2 outlines the anticipated feed and production results over the LOM.

The basis for the development of the concentrate recoveries and grades were the locked cycle tests carried out by SGS during the 2017-2018 metallurgical program. The LOM gold recovery is estimated at 89.6% with 15.8% of the gold recovery to gold-copper concentrate and 73.8% to pyrite concentrate.

Table 17.2 – Concentrate Production Schedule Summary

Year of Mine Production[1]	Units	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	LOM
Year of Concentrator Production[1]		Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	
Ore Tonnages and Grades														
Ore Processing (includes throughput ramp up)	t	-	987,500	1,241,000	1,241,000	1,241,000	1,241,000	1,241,000	1,241,000	1,241,000	1,241,000	1,095,000	574,543	12,585,043
Au Grade	g/t	-	6.24	6.75	5.74	4.55	5.05	4.32	3.66	4.06	3.67	3.64	3.65	4.70
Cu Grade	%	-	0.41	0.48	0.40	0.36	0.33	0.24	0.18	0.19	0.18	0.19	0.15	0.29
Ag Grade	g/t	-	28.88	36.83	35.73	37.10	35.29	28.84	20.54	20.04	19.70	24.49	23.85	28.57
As Grade	ppm	-	1,157	1,312	1,115	1,026	971	744	597	592	568	622	484	850
S Grade	%	-	6.09	5.71	5.26	5.42	5.52	5.43	5.53	5.36	5.34	5.74	6.11	5.55
Gold-Copper Concentrate														
Au Recovery	%	-	15.15	16.21	16.05	15.86	15.94	15.82	15.72	15.78	15.72	15.71	15.72	15.81
Ag Recovery	%	-	47.11	54.13	53.58	54.27	53.36	50.14	45.98	45.74	45.57	47.96	47.64	49.76
Cu Recovery	%	-	82.63	88.58	87.84	87.52	87.27	86.48	85.98	86.02	85.92	86.01	85.74	86.48
Au Grade	g/t	-	24.7	23.0	23.6	20.3	24.6	28.9	31.9	34.6	33.3	31.1	37.5	28.1
Ag Grade	g/t	-	355.5	419.0	489.5	566.8	575.9	610.6	523.5	495.1	517.6	638.5	744.6	531.4
Cu Grade	%	-	8.91	8.97	8.90	8.86	8.84	8.76	8.71	8.71	8.70	8.71	8.68	8.80
Gold-Copper Concentrate Tonnage														
Dry	dmt	-	37,795	59,050	48,534	44,082	40,584	29,391	22,386	22,976	21,526	20,147	8,767	355,239
Wet	wmt	-	41,995	65,611	53,927	48,980	45,094	32,656	24,873	25,529	23,918	22,386	9,742	394,710
Mass recovery (dry basis)	%	-	3.8	4.8	3.9	3.6	3.3	2.4	1.8	1.9	1.7	1.8	1.5	2.8
Pyrite Concentrate														
Au Recovery	%	-	70.68	75.64	74.89	74.01	74.38	73.84	73.35	73.64	73.36	73.34	73.34	73.76
Ag Recovery	%	-	44.98	40.87	41.42	40.73	41.64	44.86	49.02	49.26	49.43	47.04	47.36	45.01
S Recovery	%	-	80.01	82.12	78.51	79.81	80.60	79.88	80.66	79.27	79.17	82.34	85.33	80.45
Cu Recovery	%	-	8.18	8.77	8.70	8.66	8.64	8.56	8.51	8.52	8.51	8.51	8.49	8.56
Au Grade	g/t	-	37.1	43.1	39.4	29.9	32.8	28.3	23.4	26.9	24.3	22.4	21.1	30.3
Ag Grade	g/t	-	109.4	127.0	135.5	134.3	128.2	114.8	87.7	88.8	87.8	96.8	89.1	110.2
S Grade	%	-	41.04	39.56	37.82	38.44	38.83	38.48	38.86	38.19	38.14	39.66	41.10	38.95
Cu Grade	%	-	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15
Pyrite Concentrate Tonnage														
Dry	dmt	-	117,221	147,054	135,575	139,699	142,234	139,931	142,414	138,000	137,659	130,379	72,819	1,442,984
Wet	wmt	-	130,246	163,394	150,638	155,221	158,037	155,479	158,238	153,333	152,954	144,865	80,910	1,603,316
Mass Recovery (Dry Basis)	%	-	11.9	11.8	10.9	11.3	11.5	11.3	11.5	11.1	11.1	11.9	12.7	11.5

Note:
1. Mine start production in Year 1 of LOM, Concentrator starts in Year 2

17.4 Process Description

Figure 17.1 illustrates the simplified process plant flowsheet for the Project. The process plant includes the following principal process circuits:

- Jaw crushing and ore storage.
- Single SAG grinding, classification and grinding product buffer tank.
- Copper flotation and copper rougher concentrate regrind.
- Pyrite flotation and pyrite rougher concentrate regrind.
- Copper concentrate thickening, filtration, and loadout.
- Pyrite concentrate thickening, filtration, and loadout.
- Water circuits – fresh water, process water, gland seal water, fire protection water.
- Reagents and consumables.
- Compressed air facilities.

The paste backfill plant is designed by Paterson & Cooke and includes tailings thickening, filtration, backfill and transfer to the filtered tailings storage facility (FTSF). The FTSF is designed by NewFields and includes the interface with surface water management. These areas are covered in Sections 18.7 and 18.8 of this Report.

The process plant is designed to produce saleable separate gold-copper and pyrite concentrates. Testwork examined both bulk and sequential sulphide flotation circuit flowsheets; with sequential flotation being shown as the optimal flowsheet. The sequential flowsheet was shown to reduce capital and operating costs whilst producing high value concentrates.

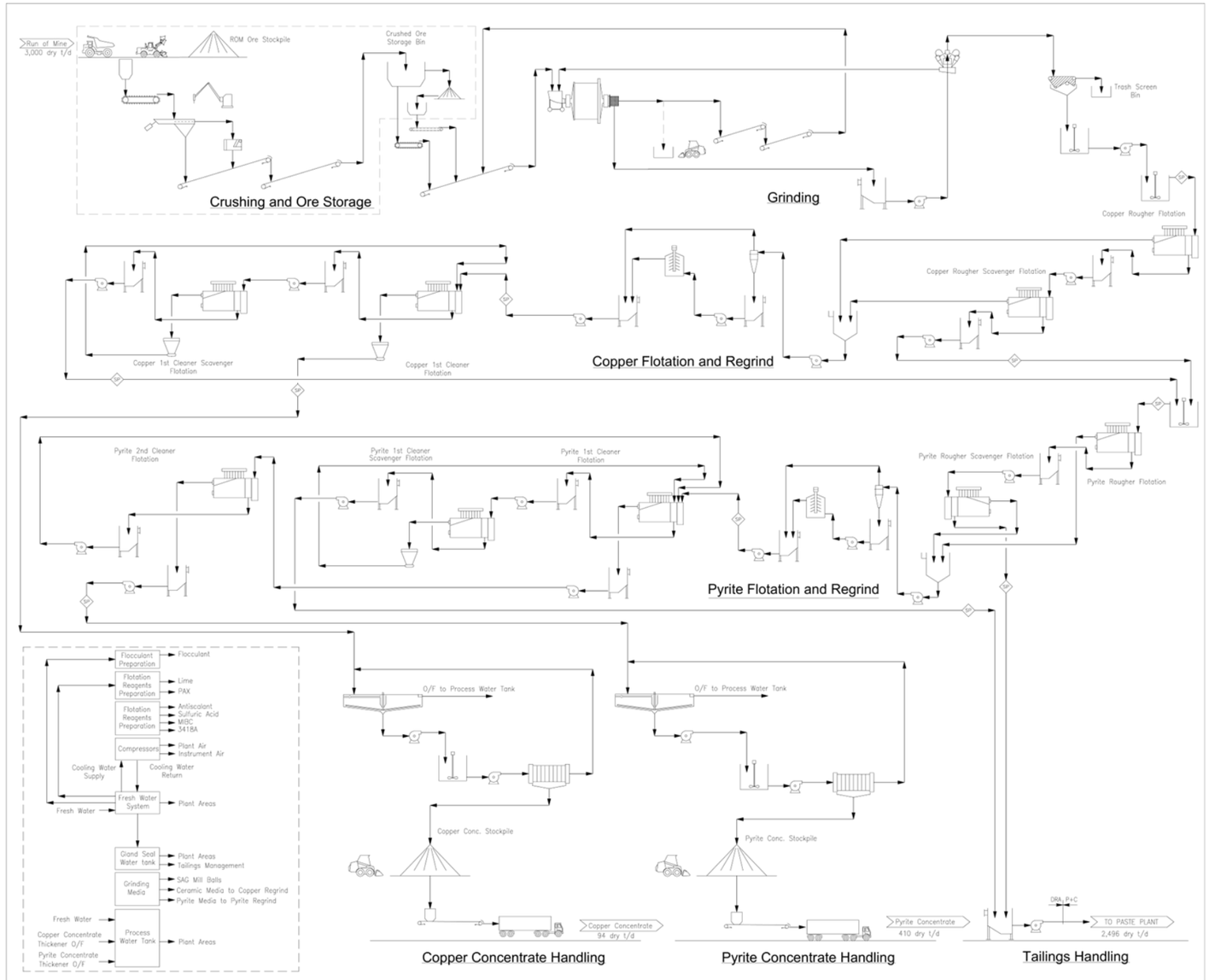
ROM ore from the underground mine is crushed on surface in a one-stage crushing open circuit primary jaw crusher. The material is then suitable for grinding and is fed to a 1,000-t live capacity crushed ore storage bin. This set-up provides an operating buffer between the crushing and milling circuits during operation downtime.

Material is withdrawn via the apron feeder at the bottom of the bin and conveyed to the SAG mill. Any overflow from the bin is re-fed to the SAG mill feed conveyor through the skid built hopper and belt feeder.

The SAG mill operates in a single-stage closed milling circuit with a bank of hydro-cyclones. The pebbles separated from the mill discharge by means of the trommel with the oversize conveyed back to the SAG mill, and undersize gravitating to the mill discharge pumpbox.

No pebble crushing is required as per the mill circuit simulation results. The process water is added to the SAG mill feed chute and milk of lime is added to the mill discharge pumpbox to maintain a target mill pulp pH. Hydrocyclone underflow material is returned to the SAG mill.

Figure 17.1 – Simplified Overall Process Flow Diagram



Notes:
1. SP – Blue Cube's on stream sampler and on stream analyser.
2. Flotation cells are Jameson technology.

Hydrocyclone overflow pulp reports to a grinding product buffer tank through the linear trash screen. The buffer tank provides the 15-minute slurry residence time required to smooth the SAG mill product particle size. The buffer tank discharge constitutes a feed for the copper flotation circuit.

Copper flotation feed is conditioned with milk of lime, MIBC, 3418A, and process water in the copper rougher flotation conditioning tank. The conditioned slurry flows by gravity to the feed box and of the copper rougher Jameson cell. Copper rougher tails are pumped to the copper rougher scavenger Jameson cell. The scavenger recovers the slow floating copper to the concentrate. Copper rougher, and scavenger concentrates both sent to the regrind. Copper rougher scavenger tailings are pumped to the pyrite rougher conditioning tank.

Gold-copper rougher, and rougher scavenger concentrates combined as a copper regrind feed and reground in a Variable Frequency Drive (VFD) driven ceramic media vertical stirred mill operating in open circuit with the scalping cyclone cluster which removes the fines of 30 µm d_{80} to the overflow with underflow being pumped to the regrind mill. The regrind mill discharge is combined with the cyclone overflow and is pumped to the copper cleaning flotation.

Copper cleaning flotation uses a single-stage cleaning process. Design of the Jameson cells selected for the duty accommodates a production increase from 3,000 to 3,400 t/d without the addition of an extra cleaning stage.

The gold-copper cleaning circuit consists of one (1) cleaner, and one (1) cleaner scavenger Jameson flotation cell of same size. Tails from the cleaner cell are pumped to the cleaner scavenger cell.

Tails from the cleaner scavenger cells are directed to the pyrite rougher conditioning tank, and the concentrate from the cleaner scavenger is returned to the cleaner cell feed box. Gold-copper cleaner concentrate reports to final concentrate thickening, filtration, and loadout.

Final gold-copper concentrate is thickened and dewatered with pressure filters and stored in bulk in the concentrate storage area prior to being loaded into containers and shipped.

In the pyrite rougher conditioning tank, the copper rougher scavenger tails slurry pH is reduced from 11-11.5 to 7 with the addition of sulphuric acid. The slurry is conditioned with the PAX collector, and process water is added for slurry density control. The MIBC frother is added to the pyrite rougher cell feed box.

The conditioning tank overflows to the pyrite rougher feed box and discharges into the pyrite rougher Jameson flotation cell.

Pyrite rougher tails are pumped to the pyrite rougher scavenger Jameson cell. The scavenger recovers the slow floating pyrite to the concentrate. Pyrite rougher, and scavenger concentrates

both sent to the regrind. Pyrite rougher scavenger tailings gravitate to the tails pumpbox and then pumped to the tailings thickening, filtration, and paste backfill plant withing the Paterson & Cooke battery limit.

Pyrite rougher, and rougher scavenger concentrates combined as a pyrite regrind feed and reground in a VFD driven ceramic media vertical stirred mill operating in open circuit with the scalping cyclone cluster which removes the fines of 30 µm d_{80} to the overflow with underflow being pumped to the regrind mill. The regrind mill discharge combined with the cyclone overflow is pumped to the pyrite cleaning flotation.

Pyrite cleaning circuit comprised of two (2) sequential stages of cleaning. The first stage of pyrite cleaning consists of one (1) first cleaner and one (1) first cleaner scavenger Jameson flotation tank cells. Tails from the first cleaner cell are pumped to the cleaner scavenger tank cell.

Tails from the first cleaner scavenger are combined with the pyrite rougher scavenger tails and pumped to the tailings thickening, filtration, and paste backfill plant withing the Paterson & Cooke battery limit. The first cleaner concentrate is pumped to the second cleaner Jameson cell, while concentrate from the first cleaner scavenger is returned to the first cleaner cell feed box.

The second cleaner bank consists of a single Jameson flotation cell. Second cleaner concentrate reports to the pyrite concentrate thickener, and the second cleaner tails report to the first cleaner cell feed box.

Pyrite concentrate is thickened and dewatered with pressure filters and stored in bulk in the concentrate storage area prior to being loaded into containers and shipped.

Water recovered from both the gold-copper concentrate and pyrite concentrate thickeners overflow is the source of the process water. This water is combined in the process water tank with the final tails thickener overflow, water from the mine and some fresh water make-up. The process water is then pumped to the concentrator areas as per the plant flowsheet.

17.5 Reagents and Consumables

17.5.1 Reagents

The following reagents are used throughout the process plant:

- Methyl Isobutyl Carbinol (MIBC) – Flotation frother for both gold-copper, and pyrite circuits.
- Aerophine 3418A – Copper collector.
- Potassium Amyl Xanthate (PAX) – Pyrite collector.
- Hydrated Lime – pH control within the grinding, and copper flotation.
- 93% Sulphuric Acid – pH control within the pyrite circuit.

- Antiscalant – To prevent scaling within the plant pipelines.
- Magnafloc 333 – Flocculant for gold-copper, and pyrite concentrates thickeners.

Reagent mixing will be completed in a designated area within the plant. The design of this area includes features such as bunding, with dedicated sump pumps. Separate onsite long-term reagent supply storage is provided a safe distance away from the process plant.

17.5.2 CONSUMABLES

Steel balls grinding media for the grinding mill is supplied in 200 L steel drums or 500 kg super sacks.

Ceramic grinding media for the regrind mills will be delivered in 1,000 kg super sacks.

Replacement filter clothes are also provided for the concentrate pressure filtration systems.

A summary of the reagents and consumables usage can be found in Table 17.3.

Table 17.3 – Reagents and Consumables

Description	Annual Usage		
	Unit	3,000 t/d	3,400 t/d
Reagents			
Hydrated Lime (Plant – Total)	t/y	3,415	3,871
Sulphuric Acid	t/y	876	993
Antiscalant	t/y	11	12
Flocculant	t/y	6.6	7.4
Collector (3418-A)	t/y	25	28
Collector (PAX)	t/y	113	128
Frother (MIBC)	t/y	139	158
Grinding Media			
Grinding Balls	t/y	1,910	2,164
Ceramic Media – Copper Regrind	t/y	6.6	7.5
Ceramic Media – Pyrite Regrind	t/y	39.0	44.2
Liners			
SAG Mill Liners	set/y	1	1
Copper Regrind Mill	set/y	2	2
Pyrite Regrind Mill	set/y	2	2
Filter Cloth			
Copper Filter Press	set/y	1	1
Pyrite Filter Press	set/y	1	1

17.6 Utilities and Services

17.6.1 Water Services

The use of external make-up water has been minimised as part of the process plant design. Process water is recovered within the circuit using thickeners and filtration unit operations. The process water will be recovered from pyrite concentrate thickener, copper concentrate and tails thickener, underground mine. Process water tank as well as distribution system have been incorporated into the design.

Fresh water will be withdrawn from local collection ponds or supplied from the water treatment plant. A combined freshwater tank will hold sufficient water to meet instantaneous process demands of the plant.

Gland seal water will be withdrawn from the gland seal water tank, filtered through the water strainers, and stored in a tank prior to distribution. The gland seal water make-up will be delivered from the freshwater tank.

Water used for the plant's fire extinguishing systems will be sourced from the fresh water tank. The system is considered to include an electrically driven pump, diesel pump, a jockey pump, and required auxiliaries and water supply pipelines.

Plant potable water system receives a fresh water from external water source separately permitted for the domestic water usage. The water will be treated to meet the local sanitary regulations requirements and distributed to the users within the process plant and the infrastructure.

Site runoff water will be collected to the respective contact and drainage pools, and treated to suit the discharge requirements, and then distributed as per the runoff water PFD.

Concentrator water balance reconciliation for 3,000 t/d operation can be found in Table 17.4. For 3,400 t/d plant, the freshwater intake will stay the same with the concentrator slurry densities adjusted, and increased return flow rates.

Table 17.4 – Concentrator Water Balance Reconciliation

Water IN	(m^3/h)	**Water OUT**	(m^3/h)
Fresh Water	28.8	Water in Tailings Management Discharge	292.6
Water in ROM Ore (adjusted for availability)	4.2	Water in Copper Concentrate	0.5
FTSF and Paste Backfill Plant Return Water	258.5	Water in Pyrite Concentrate	2.1
Mine Dewatering Water	9.7	Gland Seal Water to FTSF and Paste Backfill Plant	6.1
Total IN	**301**	**Total OUT**	**301**

17.6.2 Air Services

Compressed air for plant distribution will be provided by the centralised plant compressor plant. Air receivers will be positioned throughout the process plant to buffer and control fluctuations within the system.

Instrument air for the process plant will be provided by drying an off-take stream from the centralised compressed air plant. Air receivers will be positioned throughout the plant to buffer and control fluctuations within the system.

Compressed air for the crusher area will be provided by independent systems due to their distance away from the main plant.

17.6.3 Electrical Services

Power from the grid, and from the on-site diesel power plant will supply power to the electrical rooms (E-rooms) which will then feed to various processing units and related infrastructure.

Diesel generators will be used to provide emergency power, ensuring operational continuity during any power outages.

17.7 Sampling and On-Stream Analysers

The plant will use a Blue Cube's slurry analysers which diffuse reflectance spectroscopy to measure the mineral grades of slurries directly in the process. The system will sample the process streams for Cu, As, Fe, Au, Ag grades as well as particle size distribution. The technology used for the sampling and on-stream analysis does not require a sample delivery and dispatch system. The locations of the on-stream analysers are indicated as “SP” in Figure 17.1.

A summary of the samples and sampling requirements can be found in Table 17.5.

Table 17.5 – Sampling Requirements Summary

Sample	Analysis	**Metallurgical Accounting Required** (Yes/No)
SAG Mill Product	PSD, Metal Grades	Yes
Copper Rougher Tailings	Metal Grades	Yes
Copper 1st Cleaner Feed	PSD, Metal Grades	No
Copper 1st Cleaner Concentrate	Metal Grades	Yes
Copper 1st Cleaner Tailings	Metal Grades	No
Copper 1st Cleaner Scavenger Tailings	Metal Grades	Yes
Pyrite Rougher Tailings	Metal Grades	Yes

Sample	Analysis	Metallurgical Accounting Required (Yes/No)
Pyrite 1st Cleaner Feed	PSD, Metal Grades	Yes
Pyrite 1st Cleaner Tailings	Metal Grades	No
Pyrite 1st Cl-Scav Tailings	Metal Grades	Yes
Pyrite 2nd Cleaner Tailings	Metal Grades	No
Pyrite 2nd Cleaner Concentrate	Metal Grades	Yes

17.8 Process Control

The Project will have an adequate level of automation and remote-control facilities. Instrumentation will be provided within the plant to measure and control key process parameters and to provide a safe work environment.

All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS). The PCS that will be used for this plant is a PLC-based SCADA 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 operator interface terminals (OIT).

There will be two (2) modes for loop-controlled Control Variables available in the PCS: “Auto-Mode” and “Manual Mode”. Auto Mode will control the variables via the applicable PID loop based on the Process Variable set-point. Manual mode will allow the user to set the control variable directly on the SCADA OIT.

In general, the plant process drives will, as a minimum, report their Ready, Run and Fault status to the PCS system and will be displayed on the OIT.

The OIT will allow drives/valves to be selected to any of the following modes; Auto, Remote, Local, Maintenance or Out-of-Service modes via the drive/valve control popup on the OIT. Statutory interlocks such as emergency stops and overload protection are hardwired and will apply in all modes of operation.

The control of most of the vendor packages should be controlled remotely from PCS OIT and ethernet connection to control system network. 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.

17.9 Recovery Estimates

Recovery estimates were derived from the sequential flotation locked cycle test completed during the 2014 metallurgical program and LCTs #3 and #4, completed during the 2017 metallurgical program.

Using the results, recovery relationships for gold and copper concentrates were developed for the 2018-2019 Feasibility Study that considered production of high-grade copper concentrate.

The 2022 Update considers production of the gold-copper concentrate where, as per smelter terms, an arsenic grade is limited to 3.5% As by weight which corresponds to about a Cu grade of 8.8%. Providing for this requirement, the copper recovery algorithms generated from the 2017-2018 testwork were re-evaluated.

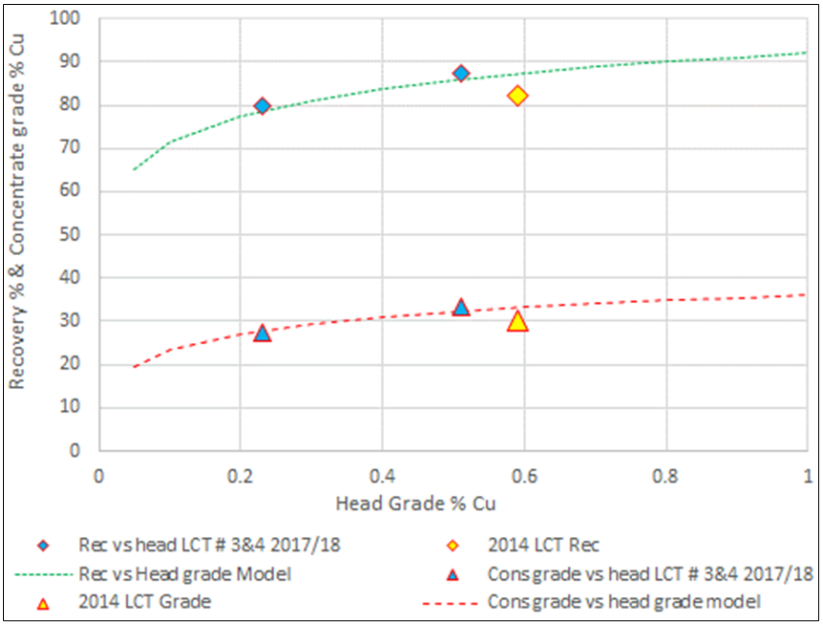
Pyrite concentrate production flowsheet stayed unchanged for the 2022 Update, and approach to the Au recovery projection to the pyrite concentrate remained unchanged.

The summary of the recovery projections can be found below.

17.9.1 Copper Recovery Copper Concentrate

Figure 17.2 illustrates the copper recovery and concentrate grade figures obtained during the locked cycle testwork conducted in 2017-2018. It is important to note that the relationship has been developed for the copper concentrate grades range from 20% through 35% Cu.

Figure 17.2 – Metallurgical Data and Model for Copper Recovery and Concentrate Grade



Source: Promet 101, 2018

A copper concentrate recovery and concentrate grade model can be constructed from fitting the data presented in Figure 17.2. The model is considered accurate for all copper head grades (HCu) greater than 0.1% and less than 1.0% and describes copper concentrate production within two (2) or three (3) cleaners circuit. Below lower head grade limit, all copper is assumed to report to the pyrite concentrate. Above a head grade of 1.0%, recovery is capped at 92% and concentrate grade at 36%.

$$Recovery_{Cu} = 9.0 \ln H_{Cu} + 92$$

$$Concentrate_{Cu} = 5.5 \ln H_{Cu} + 36$$

During the 2022 Update, the copper flotation circuit was changed to produce gold-copper (Au-Cu) concentrate which has a limit of 3.5% As, as per the smelter terms. Evaluation of the 2017-2018 locked cycles data demonstrates that copper first cleaner concentrate with 8.8% Cu grade has a corresponding As grade of 3.46% at 0.32% LOM Cu head grade.

It is understood that Au-Cu concentrate production circuit will operate based on “not to exceed” 3.5% As boundary condition and, therefore the Cu grade will be proportional to the As grade. Au-Cu concentrate grade and recovery are expected to fluctuate depending on the copper head grade in the ore.

The following formulas were developed to estimate the Cu recovery and gold-copper concentrate grade for the specified period of time (a year, month, quarter). The formulas were used for the concentrate production schedule and are based on fixed values for recovery (86.90%), and fixed grade (8.80%) over the life of mine:

$$Recovery_{Cu} = 86.90\% \cdot (1 + (A - B) \cdot 10)$$

$$Grade_{Cu} = 8.80\% \cdot (1 + (A - B) \cdot 10)$$

Where:

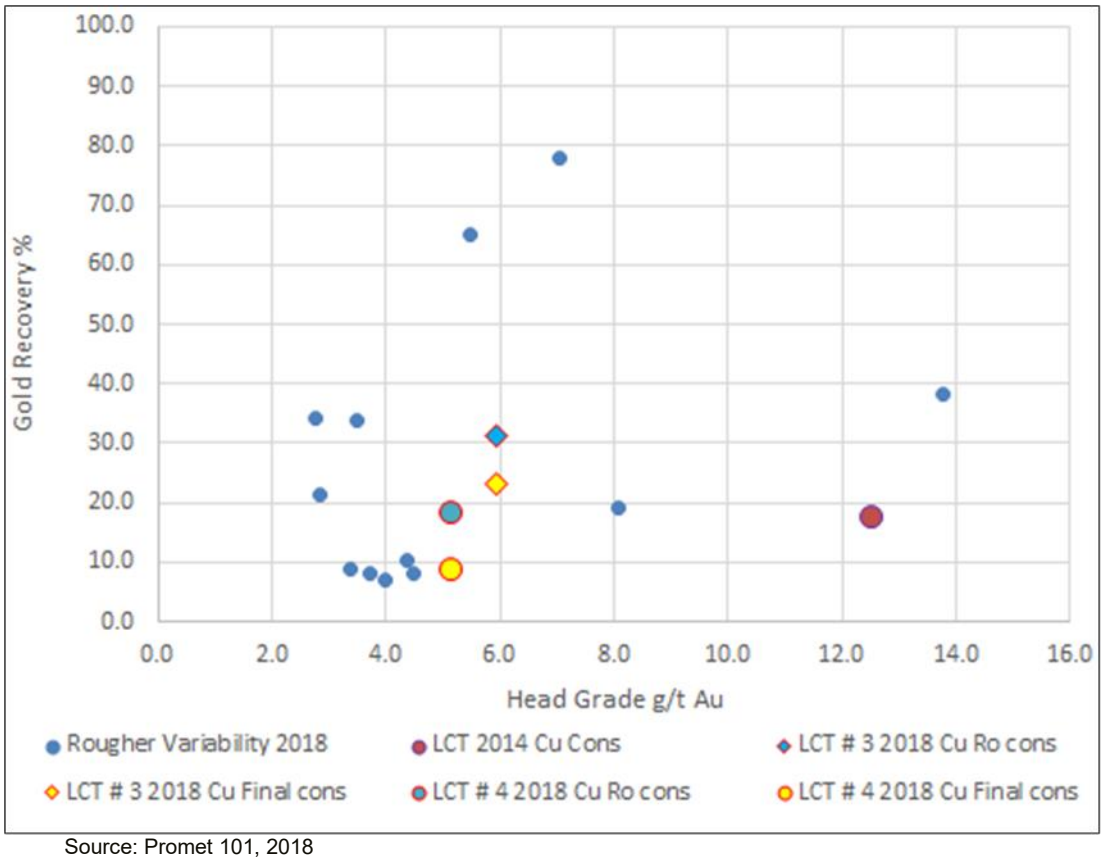
- 86.90% - LOM Cu recovery to gold copper concentrate, %
- 8.80% - LOM Cu grade of the gold copper concentrate, %
- A – Cu head grade for specific time period of life of mine, %
- B – Life-of-Mine (LOM) weighted average Cu head grade, %

These equations are considered accurate for all copper head grades greater than 0.17% and less than 0.47% (as per FSUO mine schedule) and describes Au-Cu concentrate production within cleaner circuit with a single cleaner and cleaner scavenger. Below lower head grade limit, all copper is assumed to report to the pyrite concentrate. Above a head grade of 0.47%, recovery is capped at 88.22% and concentrate grade at 8.93%.

17.9.2 GOLD RECOVERY TO COPPER CONCENTRATE

Development of a gold recovery model to the copper concentrate is not as robust as that of the copper model due to the significant variability in the association of gold with copper or iron sulphide minerals. Figure 17.3 presents the gold deportment in the 2018 rougher variability testing using the sequential flowsheet and the rougher and final cleaner stage recoveries for the 2014 and 2018 LCTs.

Figure 17.3 – Gold Deportment to Copper Concentrate



Source: Promet 101, 2018

Figure 17.3 clearly shows that rougher and cleaner recoveries are highly variable. For the 2014 LCT the recovery at a high feed grade (12.5 Au g/t) was 17.5%.

As a result of the observed variability, a fixed gold recovery to the copper final concentrate of 15% was assumed. This is considered the average of the LCTS #3 and #4 from 2017 program and the median of the variability samples from the 2014 testwork program.

During the 2022 Update, the following formula was developed for concentrate production schedule to accommodate an assumption on variation in gold recovery to gold-copper concentrate at variable head grade over the specified period of time (a year, month, quarter). The formula is based on the

fixed value of 15.88% LOM Au recovery to gold copper concentrate at LOM Au head grade of 5.11 g/t:

$$Recovery_{Au} = 15.88\% \cdot \left(1 + \left(\frac{A - B}{100}\right)\right)$$

Where:

- 15.88% - LOM Au recovery to gold-copper concentrate, %
- A – Au head grade for specific time period of life of mine, g/t
- B – LOM weighted average Au head grade, g/t

These equations are considered accurate for all Au head grades greater than 3.71 and less than 7.17 g/t (as per the mine schedule) and describes Au-Cu concentrate production within cleaner circuit with a single cleaner and cleaner scavenger.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

This section describes the main Project elements related to process, followed by supporting infrastructure. Three (3) different scale plans were produced for the Project. Figure 18.1 illustrates the overall on-site and off-site facilities. Figure 18.2 depicts the on-site facilities which include the mine and supporting facilities, the process area and the filtered tailings and water management systems. Figure 18.3 depicts the existing and new camp facilities.

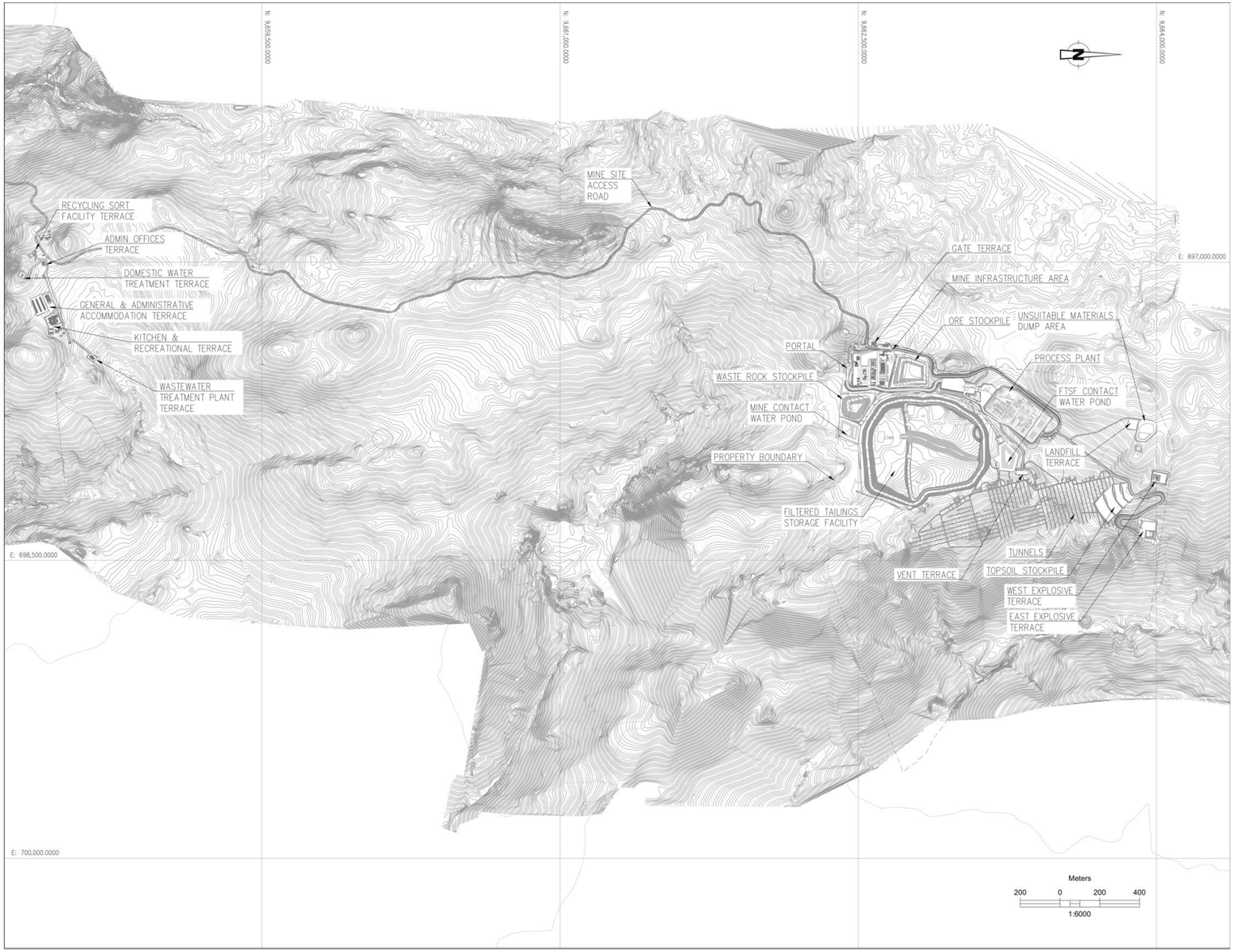
The site is accessible via the existing road from San Gerardo to Loma Larga. This road will require upgrading, as part of a separate municipal upgrade project. An existing camp will be upgraded for early works construction and then expanded for the main construction activities. The existing camp is located 5 km from the Project site.

A new 17 km long, 69 kV power transmission line (power provided by the national grid) will be required to cover the Project power requirements. A new 16.5 MW power plant will be constructed to cover any power outages.

In general, the Project facilities fall into two (2) areas as follows:

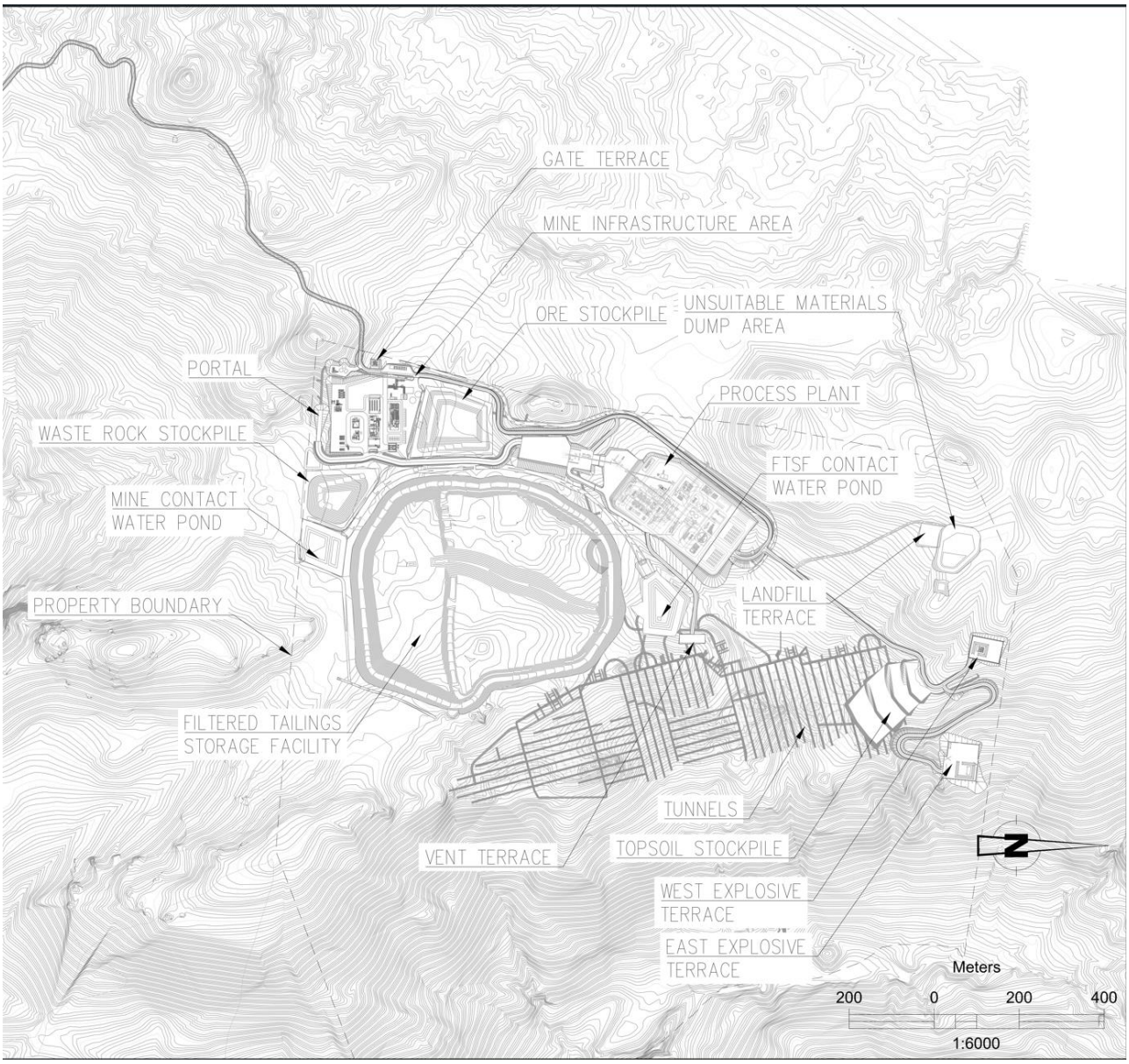
- Main Project area:
  - Mine and Mine Facilities.
  - Crushing and Processing Facilities.
  - Filtered Tailings and Water Management Facilities.
  - Site Roads.
  - Product Storage Facilities.
- Off-Site Project area:
  - Main Access Road (existing road, excluded from Project scope, upgrades, if necessary, will be pursued separately).
  - Power Plant.
  - Incoming 69 kV Power Line.
  - Construction and Permanent Camp.
  - Administration Offices, Assay Labs, and Canteen.

Figure 18.1 – Overall Mine Site Area (right) and Los Pinos Camp Area (left) Layout



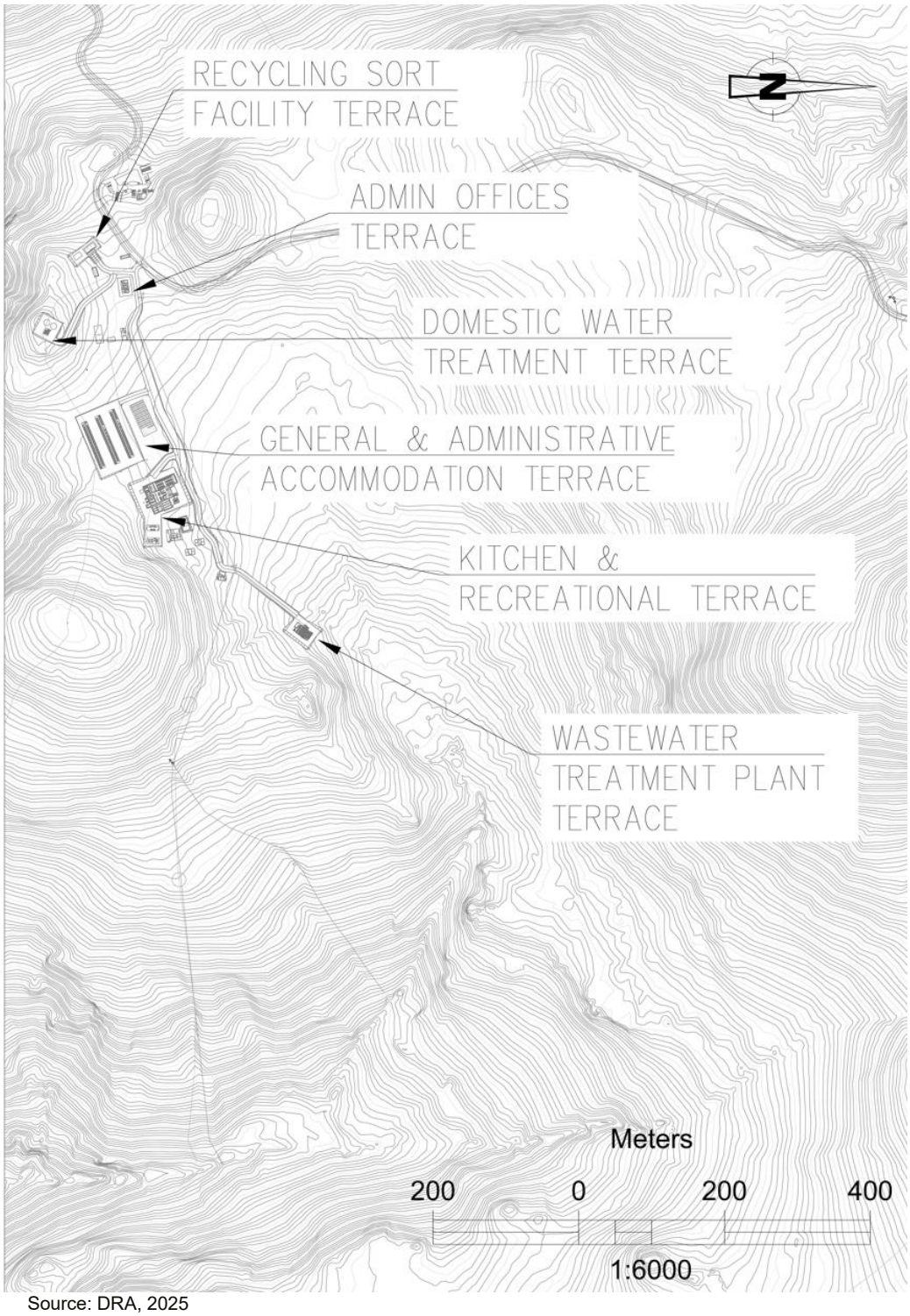
Source: DRA, 2025

Figure 18.2 – Overall Site Layout Terrace



Source: DRA, 2025

Figure 18.3 – Overall Off-Site (Los Pinos Camp) Layout



Source: DRA, 2025

18.2 Site Geotechnical Investigations

The geotechnical report prepared by NewFields dated May 22, 2020, indicates that although preliminary investigations have been carried out, and preliminary recommendations have been made, further geotechnical investigations will be required to finalise the detailed design of the Project infrastructure and roads.

NewFields recommends additional geotechnical bore holes to better define the rock quality near the surface in the vicinity of the Project facilities. Bore hole depths should also be selected in consideration of the cut and fill necessary for the development of the facility pads. Further recommendations by NewFields include the identification of borrow sources for construction materials. These materials must be characterised and tested to ensure they meet the recommended Project standards. Moreover, a typical road section should be developed for the site that defines recommended sub-base, base, and wearing course, and the durability of road construction materials needs to be assessed.

In summary, although the geotechnical report by NewFields provides preliminary Feasibility Study results and recommendations for roads and infrastructure, more detailed investigations and data collection will be required to finalise the design and construction plans for detailed design.

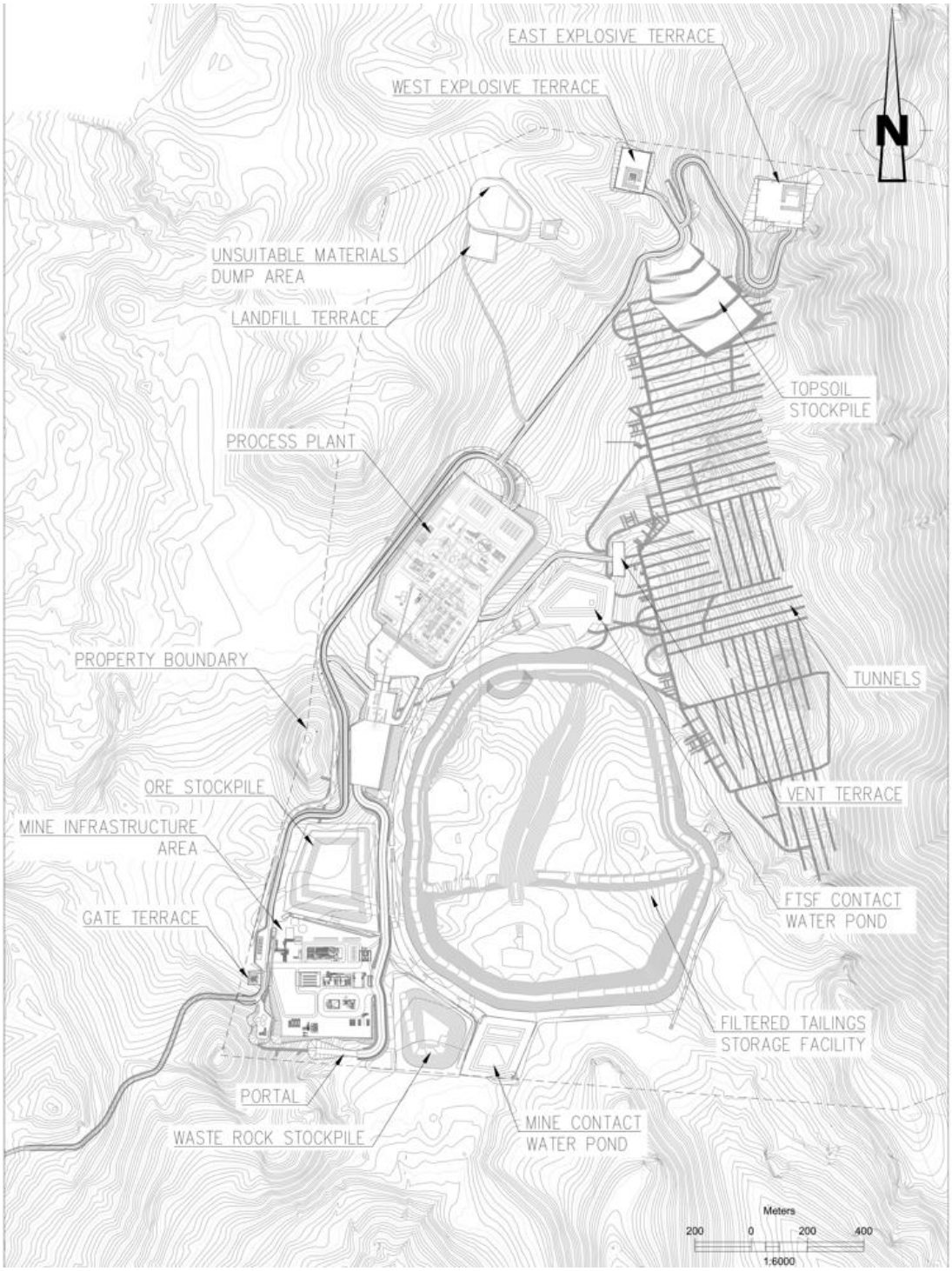
18.3 Access Roads

This section provides an overview of the main and site-specific roads, including their locations, dimensions, and intended functions. Figure 18.4 illustrates the layout of the on-site roads.

18.3.1 Main Access Road

The existing road from San Gerardo to Loma Larga is outside the Project's scope. Upgrades may be required and will be pursued separately. The road is owned by the local municipality, with DPM providing support for upgrades as needed.

Figure 18.4 – On-Site Roads



Source: DRA, 2025

18.3.2 PLANT ROADS

Site roads are required to access the various areas of the site, as follows (shown in Figure 18.4):

- **Gate to Process Plant** – A 1.5 km long road between the main access gate and the process plant entrance will be constructed. The overall road will be a 10.2 m wide gravel-wearing coarse road consisting of 7.2 m road (shoulder to shoulder) and a 3 m widened shoulder on the west side to accommodate pipes.
- **Access Topsoil Stockpile / West and East Explosive Magazines** – Three (3) roads, 1 km long in total x 7.2 m wide (shoulder to shoulder) will provide access to the explosive storages and topsoil stockpile areas.
- **Haul Road** – A 540 m long x 13.2 m wide haul road will be constructed between the portal and the process facility. This road will have a gravel-wearing coarse layer with safety berms on both sides and will accommodate mine trucks that deliver ore from the mine to the ore stockpile, and to the primary crusher stockpile area as well as to deliver waste to the waste stockpile.
- **Internal Process Plant Roads** – will access the crusher, milling, flotation and concentrate filtration areas.
- **Internal Roads for the Mine Infrastructure Terrace** – will provide access to the wash bay, truck shop, warehouse, and batch plant.

18.4 On-Site Infrastructure and Buildings – Mine Facilities

On-site buildings, platforms, and roads are designed for supporting Project activities. Infrastructure buildings are designed as modular prefabricated structures combined with steel and cladding where required. The purpose is to reduce as much as possible the amount of work on site. 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. Fire detection and suppression system will be in place and will be compliant with NFPA 30 and NFPA 13.

Various facilities erected for construction will continue to be used for mine operations.

Mine facility buildings include:

- Truck shop and maintenance facility, 55 m x 20 m – six (6) bays for heavy equipment maintenance and two (2) bays for light vehicle maintenance.
- Wash bay – adjacent to truck shop.
- Warehouse and Laydown, 25 m x 36 m.
- Explosives magazine – cartridge magazine, emulsion building and accessory buildings, with 670 m security radius.

- Fuel storage and distribution – 8,000 L diesel storage + 5,000 gasoline for light vehicles – sufficient for the all-diesel mining fleet.
- Mining facilities and services - offices, canteen, first aid and dry/change house.
- Mine guard house.

18.5 On-Site Infrastructure and Buildings – Process Plant Area

In addition to the main process plant facilities (crusher, ore stockpile, process plant, tailings facilities, etc.), the process area infrastructure includes various non-process buildings, including:

- Process offices (including met lab, change room and control room).
- Process shop.
- Reagents storage.
- Concrete shotcrete plant.

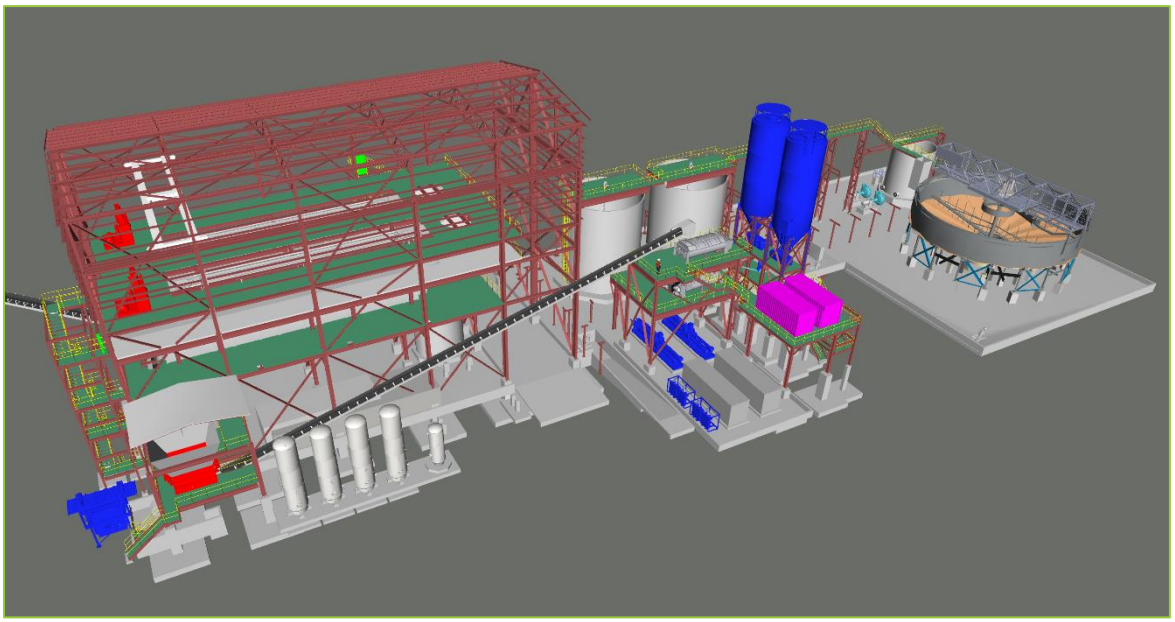
18.6 Tailings Dewatering and Paste Backfill Plant

All tailings produced by the process plant will be sent to the tailings dewatering and paste plant (Figure 18.5). Tailings will be dewatered and either stacked on the Filtered Tailings Storage Facility (FTSF) or used to supply the Loma Larga underground mine with paste backfill. The nominal capacity of the plant is 128 dry t/h (tailings).

The surface tailings dewatering and paste plant infrastructure consists of the following three areas:

- **Filtration Area**- Tailings are thickened and underflow is pumped to the filter feed tanks. Slurry is then pumped from the filter feed tanks to two pressure filters. The filter cake is either used for paste production or diverted to the FTSF. A maximum target filter cake moisture of ~15% (metallurgical) is assumed based on experience at other mines.
- **Mixing Area** – Filter cake is mixed with water in the conditioning mixer and flows to the continuous mixer where binder is added. A hopper buffers paste to the paste pump which pumps paste to the underground distribution system (UDS). The nominal design flow rate of the paste plant is set at 88 m³/h based on the mine plan completed in March 2025. The paste system is sized based on an all or nothing approach, i.e., all the tailings are sent underground as paste backfill or all the tailings are sent to the FTSF as filter cake.
- **Services Area** – Water is provided for paste mixing, flushing, and pump glands, as well as to the fire system and safety showers. Compressed air is provided for UDS purging, instruments and valves.

Figure 18.5 – Tailings Dewatering and Paste Backfill 3D Model

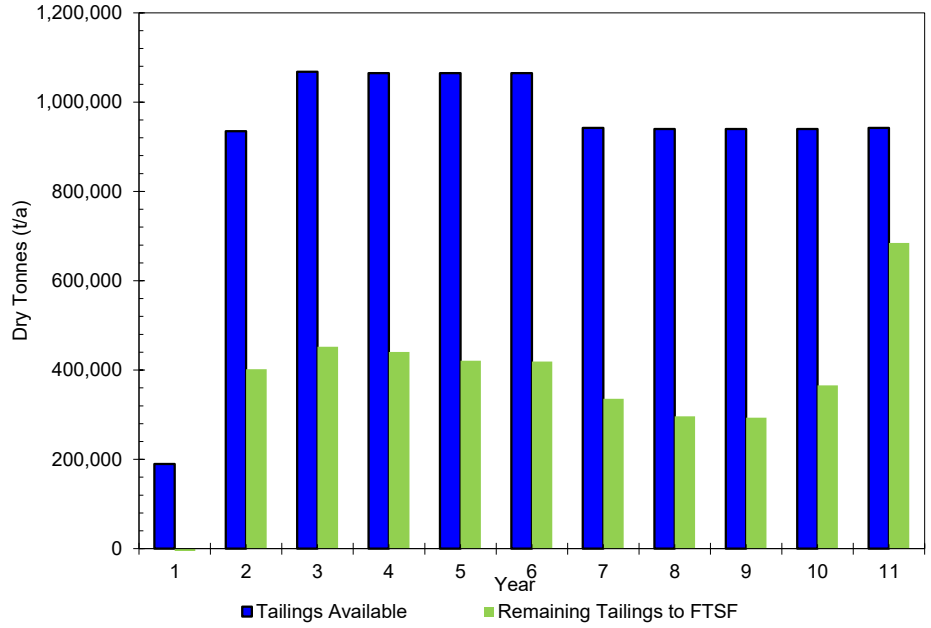


Source: Paterson & Cooke, 2023

18.6.1 TAILINGS SCHEDULE

Total tailings required for paste and total tailings reporting to the dry stack (tailings surplus) is presented in Figure 18.6.

Figure 18.6 – Tailings Required for Paste and Reporting to FTSF



Source: Paterson & Cooke, 2025

18.6.2 Tailings Dewatering and Paste Plant Process Overview

Figure 18.7 is a simplified process flow diagram of the dewatering and paste backfill plant and illustrates the basic system components.

The thickener feed box will receive tailings from the process plant. The tailings will be sent to the thickener, where diluted flocculant is added. The flocculant mixing plant will send flocculant to the flocculant dilution box where it is diluted with process water.

Thickener underflow is then pumped into two (2) filter feed tanks, which maintain underflow solids in suspension by agitators. The filter feed pumps provide slurry to the filter presses which separate feed into filter cake and filtrate. Bypass pumps also pump filter feed slurry directly to the mixing area.

Filter cake from the pressure filters falls onto the filter cake collectors. The cake is transferred onto a reversible transfer conveyor. The transfer conveyor either feeds forward directly into the surge bin integrated with a live bottom feeder, or feeds backward onto the stacker conveyor. The stacker conveyor and radial stacker transport and stockpile the filter cake at the FTSF.

Filtrate, cloth wash and core wash drains into the filtrate tank where the solids are kept in suspension by an agitator. The sump pumps also provide spillage to the filtrate tank. The filtrate from the filtrate tank is recycled to the thickener feed box by filtrate pumps.

The live bottom feeder controls the addition of filter cake from the surge bin to the filter cake conveyor, which transports the cake to the conditioning mixer. Trim water and/or trim slurry is added to the cake in the conditioning mixer to prepare the filter cake.

The filter cake mixture is then gravity fed into the continuous mixer, where it is mixed with the binder to create paste backfill. Depending on the mixture, it is possible to also add trim water and/or trim slurry to the continuous mixer.

As required, binder delivery trucks add binder to two (2) binder storage silos. Each silo is equipped with a dust collector. Rotary feeders, at the base of each silo, load the binder onto weigh feeders which feed the binder onto the screw conveyor. The screw conveyor adds binder directly into the continuous mixer.

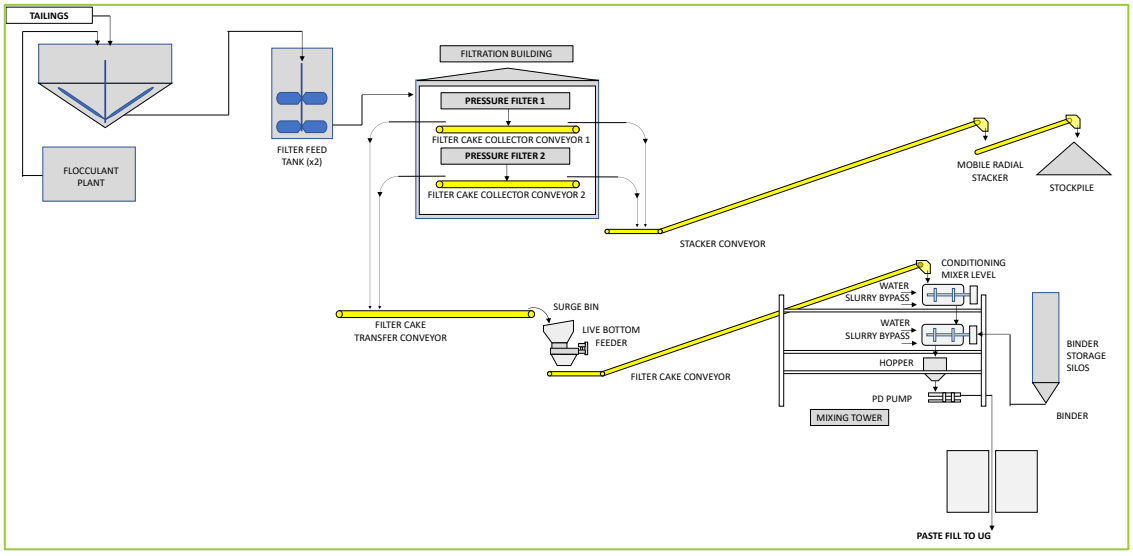
The paste produced for the continuous mixer is then gravity fed through the paste hopper to the paste pump which pumps the paste overland to a surface borehole to access the underground workings. The underground workings will be accessed via one of two (2) surface boreholes located above the orebody. The underground distribution system is further discussed in Section 16.9.

Three (3) sets of pumps draw water from the process water tank. The wash pumps provide cloth wash water and core wash water to the pressure filters and flush water to the paste hopper and

slurry pumps. The process water pumps provide trim water to the mixers, and the flush pump provides flush water to the UDS.

The drying air compressors provide unfiltered air to the drying air receivers. The air is then sent to the pressure filters and paste borehole, as required. Drying air is also diverted through an air filter and sent to the air dryer. The dry air passes through another air filter on the way to the instrument air receiver. The pressing air compressors provide air filtered by air filters to the pressing air receivers. The pressing air is then sent to the pressure filters.

Figure 18.7 – Tailings Dewatering and Paste Simplified Process Flow

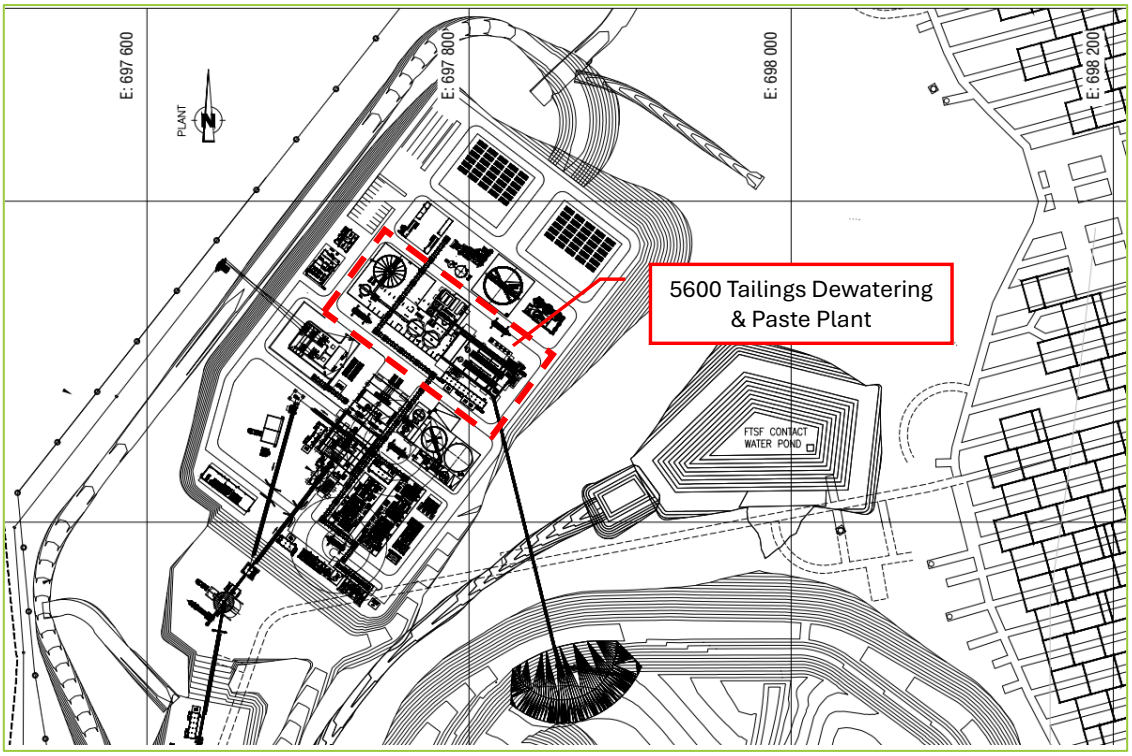


Source: Paterson & Cooke, 2023

18.6.3 SITE LOCATION

The Tailings Dewatering and Paste Plant is integrated into the overall process plant to allow common use of services and to reduce the overall plant footprint.

Figure 18.8 – Tailings Dewatering and Paste Plant Location

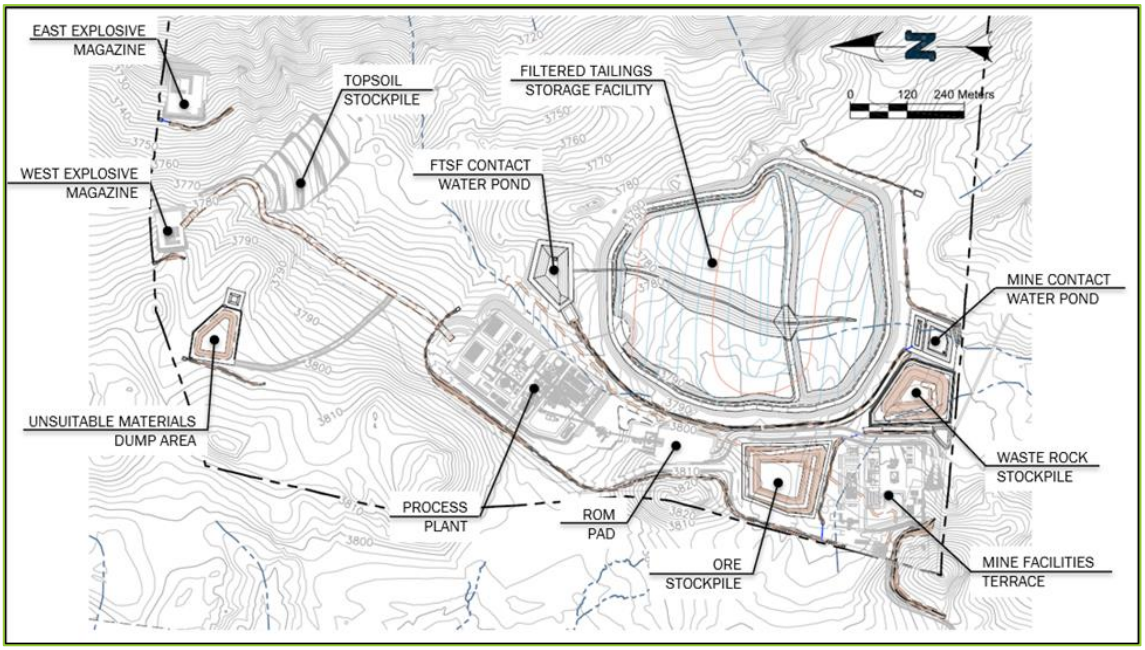


Source: Paterson & Cooke, 2023

18.7 Filtered Tailings Storage Facility

The FTSF will be located on the Atlantic side of the continental divide within the Victoria del Portete parish in the Cuenca Canton, Rio Falso Concession. The FTSF is located adjacent to the underground mine portal and the process facilities, at the head of a small drainage basin, at an approximate elevation 3,800 m above mean sea level, as illustrated in Figure 18.9.

Figure 18.9 – General Arrangement of Facilities around FTSF



Source: NewFields, 2023

Approximately 60% of the filtered tailings produced will be sent back to the underground mine as cemented paste backfill, and the remaining 40% will be stored on surface in the FTSF. It is currently estimated that about 4.3 million dry tonnes (Mt) will be placed in the FTSF over the LOM.

The estimated average production rate to the FTSF varies between 22,000 and 38,000 tonnes per month, which equates to average daily production rates between 760 and 1,250 t/d.

The selection of the technology and location of the storage facility for the surface tailings was based on a series of trade-off analyses (NewFields, 2017; NewFields, 2022; DPM, 2022) that studied different alternatives of tailings management technologies, including conventional slurry tailings, paste and thickened tailings, filtered tailings and co-disposal.

18.7.1 FTSF Surface Water Management

The FTSF design includes a stormwater management system to minimise run-on to the FTSF by diverting non-contact flows around the facility. This system consists of stormwater diversion channels located along the FTSF and process plant area perimeter. The design also includes an internal drainage system for surface contact stormwater within the facility, an underdrain collection system to collect fluids draining from the tailings all reporting to an external underdrain collection pond. A climate analysis was completed by NewFields (2018a, 2018b) to establish design criteria and assumptions for site-wide hydrologic modelling. The study includes hydrological criteria for stormwater management design and a site-wide water balance in support of the Feasibility Study

results to evaluate the proposed mine facilities for the Loma Larga Project as discussed in Section 18.9 and the FTSF forms a portion of it as discussed below.

The stormwater management system components include the following items: (Note: items 1, 8 and 9 are discussed further in Sections 18.9 and 18.10):

1. **Permanent Stormwater Diversion Channels**

   Designed to divert and route (non-contact) surface water flows around all facilities. Channels and outlet points will be armoured with riprap to minimise erosion and sediment transport.

2. **Temporary Stormwater Diversion Channels**

   Designed to support construction activities for each phase of the FTSF.

3. **Sedimentation Basins**

   Will be constructed at the discharge point of each Stormwater Diversion Channel, to collect and retain solids prior to the discharge back into the environment.

4. **Best Management Practices (BMPs)**

   Will also be employed during construction to collect runoff and minimise sediment during construction.

5. **FTSF Internal Stormwater Drainage System**

   Stormwater falling on the tailings surface will gravity flow to internal drainage systems to convey contact water to the underdrain collection system. Flow in the underdrain collection system will be routed by gravity to the FTSF Contact Water Pond located north of the FTSF.

6. **Freeboard Above the Tailings**

   A minimum of 1.5 m from the top of the tailings to the top of the embankments is maintained at all times in each phase to provide containment of all contact water.

7. **FTSF Contact Water Pond**

   All contact water collected in the FTSF, and plant site will be directed to the FTSF Contact Water Pond and pumped to the water treatment plant for treatment prior to reuse on the process or discharge into the Alumbre creek. The pond is double-lined with a geomembrane and an intermediate leak collection recovery system (LCRS). The pond capacity is sized to hold the run-off produced by the 24-h/200-year storm event occurring over the operating Phase 1 area plus bare geomembrane lined Phase 2 area (worst case scenario) in addition to the run-off from the process plant area, with a 1 m minimum freeboard.

8. **Waste Rock and Ore Stockpile Facilities Internal Stormwater Drainage Systems**

   Stormwater falling on the surface of the geomembrane-lined Ore Stockpile and Waste Rock Stockpile pads will flow to internal drainage pipes that will convey contact water to the

respective underdrain collection outlets. Flow in the underdrain collection systems will be routed by gravity to the Mine Contact Water Pond located south of the FTSF.

9. **Mine Contact Water Pond**

   Water collected in the Mine Contact Water Pond will be pumped to the FTSF Contact Water Pond. The pond is double lined with a geomembrane and an intermediate LCRS. The pond capacity is sized to hold the run-off produced by the 24-h/200-year storm event occurring over the geomembrane-lined Waste Rock and Ore Stockpile pads and the mine ancillary facilities area, with a 1 m minimum freeboard.

The feasibility design requires all embankments to be founded on competent bedrock. The overlying Paramo (organic soil), andosol, overburden and/or highly weathered rock will be removed prior to placement of embankment fill.

The primary embankment fill will be composed of competent rockfill excavated mainly from the FTSF basin areas. For purposes of this Report, a commercial quarry located 40 km from site is considered in the cost estimates for a percentage of the materials that cannot be made from onsite materials. For final design, studies are being currently undertaken in parallel to evaluate the feasibility for developing five (5) potential nearby quarries, as well as evaluate other off-site commercial borrow sources and/or mine waste for specific construction materials.

Aligned with the requirements stipulated in the Ecuadorian Tailings Guidelines MERNNR-MERNNR-2020-0043-AM (MERNNR, 2020), the Global Industry Standard for Tailings Management (GISTM, 2020) and the Canadian Dam Association (CDA, 2007; 2013; 2014; 2016), the FTSF is preliminarily classified as “Significant” in terms of risk, according to Table 2 of the Ecuadorian Tailings Guidelines (MERRNR, 2022), which is consistent with the CDA classifications (CDA, 2014) for risk.

For the slope stability analyses, minimum acceptable factors of safety for static and pseudostatic conditions were established as 1.5 and 1.0, respectively. These acceptance criteria are consistent with the Canadian Dam Association (CDA, 2007; 2013; 2014; 2016), recommendations and other international standards. The FTSF was designed considering the peak ground acceleration (PGA) associated with a 2,475-year return period (Operational Basis Earthquake) and the 10,000-year return period (Maximum Credible Earthquake), as determined by a site-specific seismic hazard analysis developed for the Project (NewFields, 2018c).

18.7.2 Geotechnical Site Characterisation

A preliminary geotechnical investigation was completed in early 2020 and consisted of 20 test pits excavated within the FTSF basin and an additional 15 test pits in areas adjacent to the FTSF. Throughout the area of the FTSF, numerous shallow and fully submerged low areas and depressions exist. Near surface water is associated with the Paramo; a shallow, perched hydrogeologic unit that comprises saturated fluvial materials. Subsurface conditions across the

footprint of the proposed FTSF and underdrain collection pond are variable and somewhat based on the existing topography. Natural depressions were characterised with thicker deposits of organics, and higher elevations within the FTSF footprint exhibited significantly less organics with rock outcrops in some areas.

Additional geotechnical and hydrogeological investigation programs are currently planned prior to the detailed engineering design phase, which will include a series of geotechnical boreholes, test pits and seismic geophysical line surveys in and around the FTSF area, process plant area, and along the mine decline and vents. The groundwater investigation objectives are to extend the existing groundwater monitoring network, monitor groundwater inflows and water levels during the advance to confirm the baseline characterisation study (Itasca, 2018), and further assess the hydraulic significance of geologic structure, and update groundwater predictions of underground mine inflows, as necessary.

18.7.3 Filtered Tailings Characterisation for Design

The following sections include a brief description of the geotechnical and geochemical characterisation of the tailings based on laboratory data.

18.7.3.1 Geotechnical Characterisation of Filtered Tailings

In 2020, a synthetic sample of tailings (identified as LL Comp 2) was produced and provided to NewFields to evaluate the geotechnical index properties. The results of the tests completed in December 2022 (NewFields, 2022) indicate the following:

- Particle size distribution was approximately 19% fine sand-sized particles, 77% silt sized particles and 4% clay-sized particles.
- The tailings sample was non plastic.
- Estimated maximum dry density of 1,682 kg/m³ and 16.7% optimum water content for compaction per the Standard Proctor Method (ASTM D 698).
- The shear strength data from the consolidated undrained triaxial compression tests indicates initially contractive behaviour that goes through phase transformation and dilates to the critical state. The critical state friction angle is estimated at 32°. The data also suggests the undrained shear strength ratio at critical state ranges from approximately 0.34 to 0.45.

Other tailings characteristics that were assumed for the FTSF design include:

- Moisture content (geotechnical) from the filter plant is 18%.
- In-place dry density of tailings is 1.55 t/m³.
- Specific Gravity of tailings solids is 2.59.

18.7.3.2 *Tailings and Quarry Materials Geochemistry*

Geochemical characterisation of rougher and cleaner tailings was conducted to support feasibility-level design of the FTSF (NewFields, 2018e). Samples were tested to determine the potential for tailings to generate acid and leach metals from the FTSF. To facilitate testing, approximately 10 kg of rougher and cleaner tailings were synthesised from a composite ore sample that was considered to be representative of the orebody that will be mined. The laboratory program included static tests [acid base accounting (ABA), paste pH, net acid generation pH (NAGpH) and elemental inductively coupled plasma (ICP) analysis], specialised tests (decant analysis, whole rock analysis, mineralogy and shake flask extraction) and kinetic tests [20-week humidity cell test (HCT)].

Results of the geochemical testing program for the rougher tailings indicate the following:

- ABA results classify both the rougher and cleaner tailings samples as potentially acid generating (PAG) based on low net neutralisation potential, low neutralising potential ratios, elevated sulphide, low paste pH and low final NAGpH.
- Leach test results suggest that rougher tailings could leach constituents of potential concern (COPC) at concentrations that exceed relevant water standards, including aluminium, arsenic, cadmium, chromium, cobalt, copper, iron, manganese, mercury, molybdenum, nickel, lead, selenium, sulphate, thallium, and zinc. COPCs in leachate from cleaner tailings could include aluminium, cadmium, chromium, cobalt, copper, iron, manganese, mercury, nickel, lead, selenium, sulphate, thallium, and zinc.
- HCT results classify the rougher and cleaner tailings as PAG, in agreement with ABA results. The HCTs exhibited acidic drainage (pH <6) for the entire test duration. COPCs identified in HCT leachate are consistent with other leach tests.

Results confirm previous conclusions from the Pre-Feasibility Study (RPA, 2016) that it will be necessary to collect and treat contact water within the FTSF and underdrain pond / line the facility.

Samples from five (5) prospective quarry sites were tested for geochemical weathering characteristics to determine suitability as an aggregate source (NewFields, 2018f). Three (3) sites (Prospecto Cantagallo, Prospecto Cantagallo Sur, Prospecto Cerro Cabina) were recommended as suitable borrow sources, as ABA and field leach test results suggest that use of the material for construction purposes was unlikely to release acid or COPCs above water quality limits.

18.7.4 TAILINGS MANAGEMENT

Within the FTSF, a front-end loader will reclaim filtered tailings from the radial stacker stockpile and load the material onto articulated 40-t haul trucks. The haul trucks will rear dump the tailings in the FTSF area, and with the use of bulldozers, the material will be spread to form thin layers of tailings that can then be compacted. The layers of tailings will be graded keeping a minimum grade (2% to 4%) to promote natural runoff of rainwater to the northern and the southern ends of the facility and

avoid water ponding on the FTSF that could increase the moisture of the tailings. Runoff water will flow via inclined risers to the underdrain collection system and to the external collection pond.

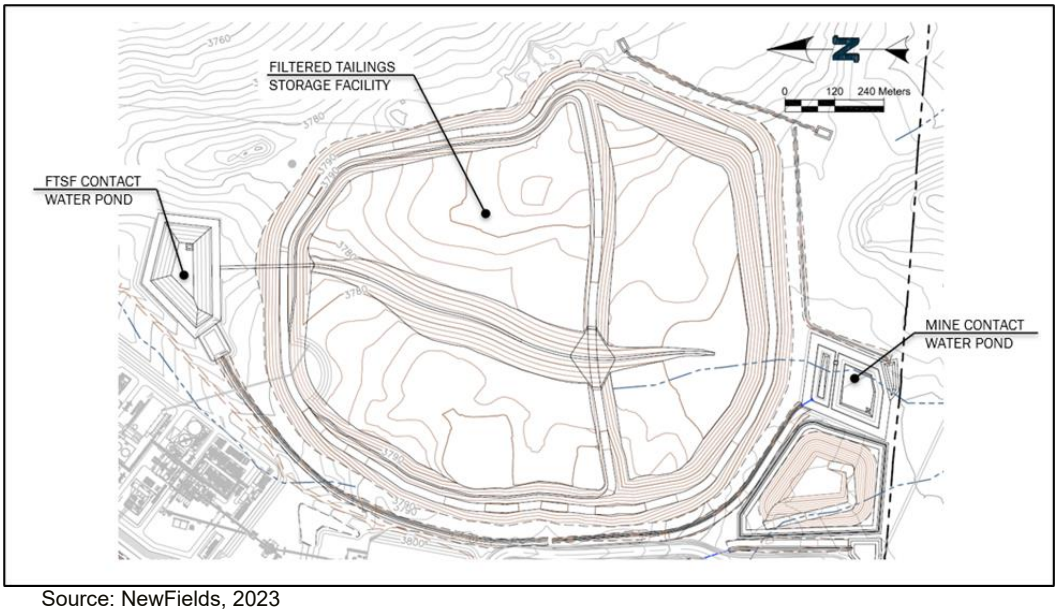
The filtered tailings will always be placed in an upgradient direction from the lowest areas to the highest areas, with the objective of providing physical stability of the tailings during operation, so that the areas with compacted tailings can provide support to the areas with fresh tailings. The continuous compaction of the placed tailings will provide adequate traffic conditions for the mobile equipment, increasing the efficiency of the tailings placement operations. Temporary internal service roads made from local soils or mine waste will be constructed above the geomembrane liner and impoundment drainage system to provide protection to the lining and collection systems. The temporary service roads can be raised and altered as needed if tailings become wet for traffic.

18.7.5 Filtered Tailings Storage Facility Design

The FTSF for the Loma Larga Project is a fully lined facility enclosed within a ring-shaped rockfill embankment, that will provide full containment of the filtered tailings, as shown in Figure 18.10. The FTSF comprises the following main elements:

- Groundwater management system.
- Perimeter rockfill embankment.
- Liner System,
- Seepage underdrain and stormwater management system.
- Liner protection layer and internal service roadways.

Figure 18.10 – FTSF General Layout



Source: NewFields, 2023

18.7.6 GROUNDWATER MANAGEMENT SYSTEM

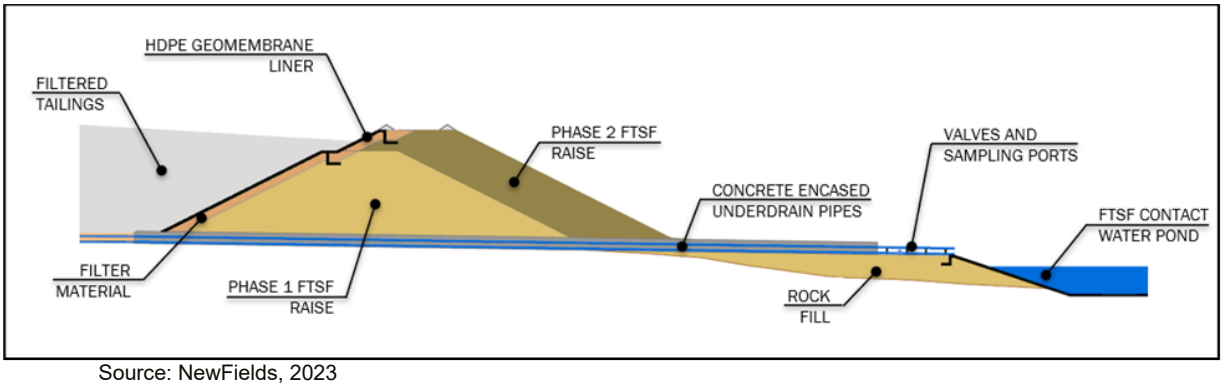
Fully saturated low areas and depressions exist within the FTSF basin area. To develop the FTSF, non-contact groundwater needs to be intercepted beneath the impoundment lining system and embankments and routed downstream of the facility. A groundwater collection drain system will be installed within the natural valleys and topographic low drainage areas created during construction. The system will consist of two (2) 200 mm, perforated, corrugated polyethylene pipes placed in excavated trenches. Trench sides will be sloped at 1.5H:1V and will be lined with nonwoven geotextile. The pipes will be covered and surrounded by drainage and filter geotextile. Finger drains will be constructed as needed to collect topographic low areas.

18.7.7 PERIMETER ROCKFILL EMBANKMENT

An embankment will be constructed utilising compacted rockfill for the entire perimeter of the facility, acting as the structural containment for the tailings. Prior to the placement of the rockfill, the entire footprint area below the embankment will be cleared of any organic soils, andosol, superficial soils, and weathered rock, to ensure that the entirety of the embankments is founded directly on competent bedrock.

The perimeter embankment of the FTSF will be primarily formed by competent rockfill sourced from the excavation of the FTSF basin and main trench, and other suitable materials produced by the excavations in other areas, such as the process plant terrace construction or road construction. In addition, mineralogical and geochemical data obtained from previous studies suggest that during the development of the mine ramp, a portion of the waste rock can be classified as non-PAG and can be used as rockfill for the construction of the FTSF embankment. The final configuration of the embankment will have upstream and downstream slopes of 2:1 (horizontal:vertical), with a minimum crest width of 10 m to provide light vehicle access all around the facility.

Figure 18.11 – FTSF Embankment and Seepage Collection Outlet Cross Section

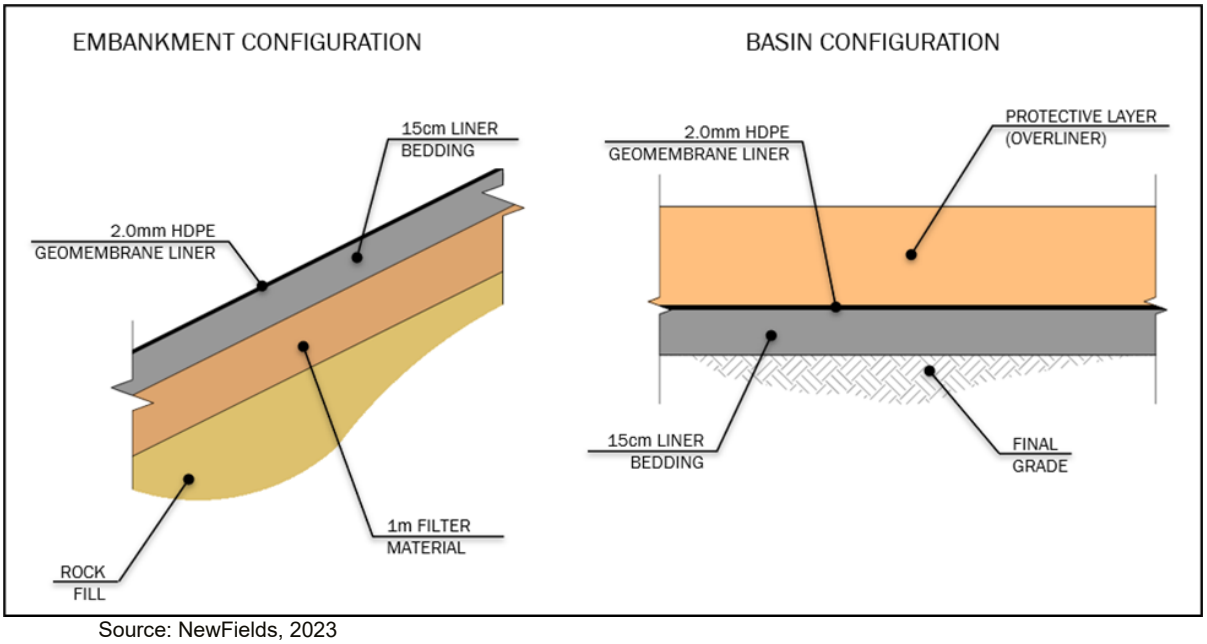


Source: NewFields, 2023

18.7.8 LINER SYSTEM

As described in Section 18.7.3.1, the tailings are classified as PAG and have the potential to leach COPCs. Therefore, the FTSF will include a geomembrane lining system to minimise the potential for leakage of contact water from the facility. The basin liner system will consist of a 150 mm compacted fine grained liner bedding layer, overlain by a geomembrane liner. The geomembrane liner component will consist of double-sided, textured, 2.0-mm thick high-density polyethylene (HDPE). The embankment liner system will be placed on the inside slopes of the perimeter embankments. The embankment liner system will consist of 1 m of filter material, 150 mm of liner bedding and 2.0 mm-thick double-sided textured HDPE. A cross-section through the FTSF typical embankment and details are shown in Figure 18.12. The final basin area of the FTSF liner placement is about 160,000 m² and the total lined area including the embankments is about 250,000 m². The basin liner will include a protective layer of sand overliner in areas that will receive traffic.

Figure 18.12 – Typical Liner System for FTSF Basin and Embankment inside Slopes



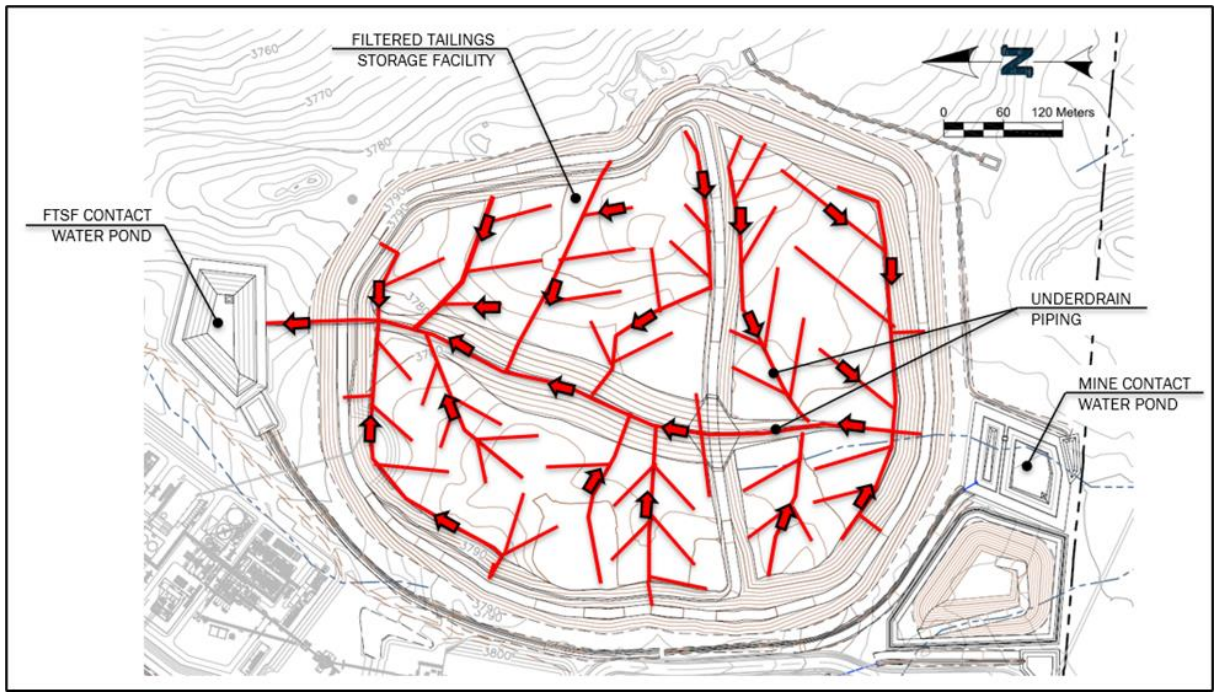
Source: NewFields, 2023

18.7.9 SEEPAGE UNDERDRAIN AND STORMWATER MANAGEMENT SYSTEM

The FTSF underdrain collection system will consist of a series of perforated, corrugated polyethylene pipes surrounded by drainage and filter materials installed above the geomembrane liner along topographic low points (Figure 18.13). The system is designed to 1) collect underdrainage from the tailings due to consolidation and gravity drainage; 2) collect stormwater runoff from the surface of the tailings via inclined riser piping; 3) decrease the overall hydraulic head on the geomembrane liner, and 4) allow for long-term drainage of the tailings mass throughout the closure

period. Flows will be conveyed from the impoundment to the FTSF Contact Water Pond via reinforced concrete-encased underdrain outlet pipes.

Figure 18.13 – FTSF Underdrain Collection System



Source: NewFields, 2023

18.7.10 FTSF Storage Capacity

The FTSF will be constructed in two (2) phases to limit the footprint of disturbed area as the mine develops and to facilitate effective management of Project capital resources. The two (2) phases (Phase 1 and Phase 2) are designed to store approximately 1.6 and 4.5 cumulative million dry tonnes of filtered tailings respectively, with 4- and 7-years capacity. The current design has the potential for additional future expansions by raising the embankment, which could provide up to 1.5 Mt of additional capacity of filtered tailings for potential future reserves.

18.7.11 FTSF Stability Evaluation

The stability of the embankments was evaluated for two (2) general cross sections through the FTSF embankments, including a north-south section and an east-west section. An interim loading scenario at the end of Phase 1 and maximum facility loading at the end of Phase 2 and at the end of a future Phase 3 were considered with the tailings at the freeboard elevation. Since the FTSF is fully lined with HDPE geomembrane, it was assumed that the embankments will remain unsaturated throughout the life cycle of the facility.

The stability evaluation indicates that acceptable factors of safety were achieved for static loading scenarios for the Phase 1 embankment as well as for the ultimate Phase 2, and future Phase 3 embankment with long-term, maximum tailings scenarios.

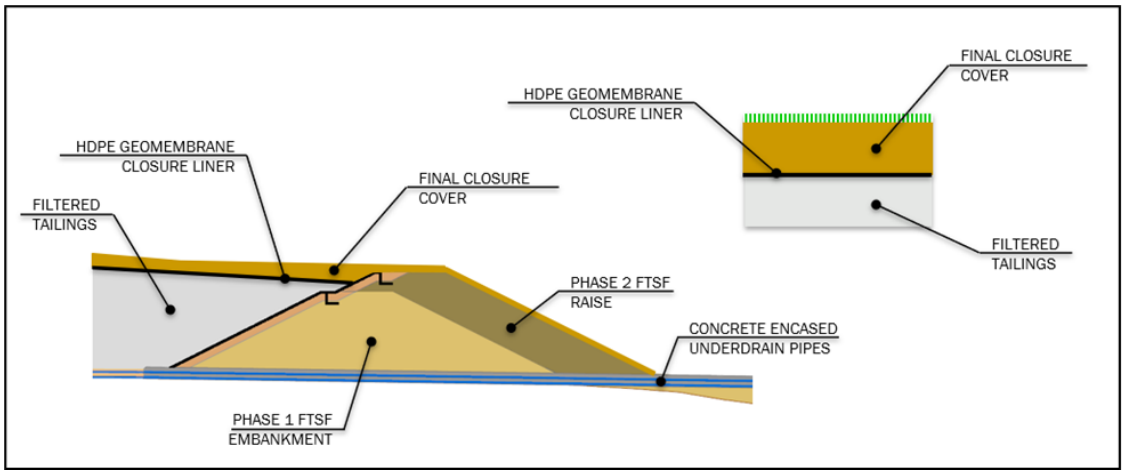
Since the facility is located within a seismically active area, the pseudostatic evaluations indicate that some deformations could take place during the design seismic events, and a subsequent displacement analysis was conducted to estimate the amount of slope movement resulting from these events. Results of the deformation evaluation indicate that up to 3 m of deformation could occur along the southern aspect of the Phase 1 tailings surface (temporary 5:1 slope) from the OBE event, but this deformation would be confined by the temporary Phase 1 berm and not result in an excursion outside of containment. Long-term deformations of the ultimate facility from the MCE event would be less than 1 m, primarily in the external slope of the perimeter rockfill embankments, and similarly would not result in an excursion of tailings outside containment.

18.7.12 Conceptual FTSF Closure

The FTSF closure strategy includes managing stormwater runoff, providing a cover system to limit infiltration, and providing post-closure maintenance. During placement, and in anticipation of the closure strategy, tailings will be deposited with a domed shape, mimicking the natural topography.

The tailings surface will be covered with a 2.0 mm HDPE geomembrane overlain by approximately 1 m of growth media material and then revegetated, as shown on Figure 18.14. Once the cover system is in place and infiltration of surface water is largely eliminated, post-closure underdrain flows will decrease over time. Underdrain flow collection will continue and transmitted to the water treatment plant (WTP) until flow ceases or water quality supports discharge. The underdrain collection pond will then be reclaimed by folding the liner system in on itself and covering the area with backfill for positive drainage. The area will then be covered with growth media and reseeded.

Figure 18.14 – Conceptual Closure of FTSF



Source: NewFields, 2023

Post closure will require maintenance of the closure cap until a vegetated surface can be established. The closure surface will be monitored, inspected, and repaired (if needed) to address water and wind erosion. In addition to the closure cap, the perimeter road, stormwater diversion channels and underdrain collection pond areas will also be monitored, inspected, and repaired, if needed. Instrumentation will be installed during construction and operation of the facility to monitor and record geotechnical parameters of the embankment such as settlements, and pore pressure. Preliminarily, the geotechnical monitoring system will consist of piezometers and settlement monuments.

18.8 Waste Rock and Ore Stockpiles

18.8.1 Waste Rock Stockpile

Waste rock geochemistry assessments consisted of identifying loading rates from the major lithological and alteration categories for the rock expected to be removed as waste material (e.g., non-ore-bearing) from the underground mine and ramp developments. Additionally, these materials will be exposed as wall rock in the underground workings and have the potential to generate acidity and release metals once exposed to oxygen.

Testing programs were carried out as follows:

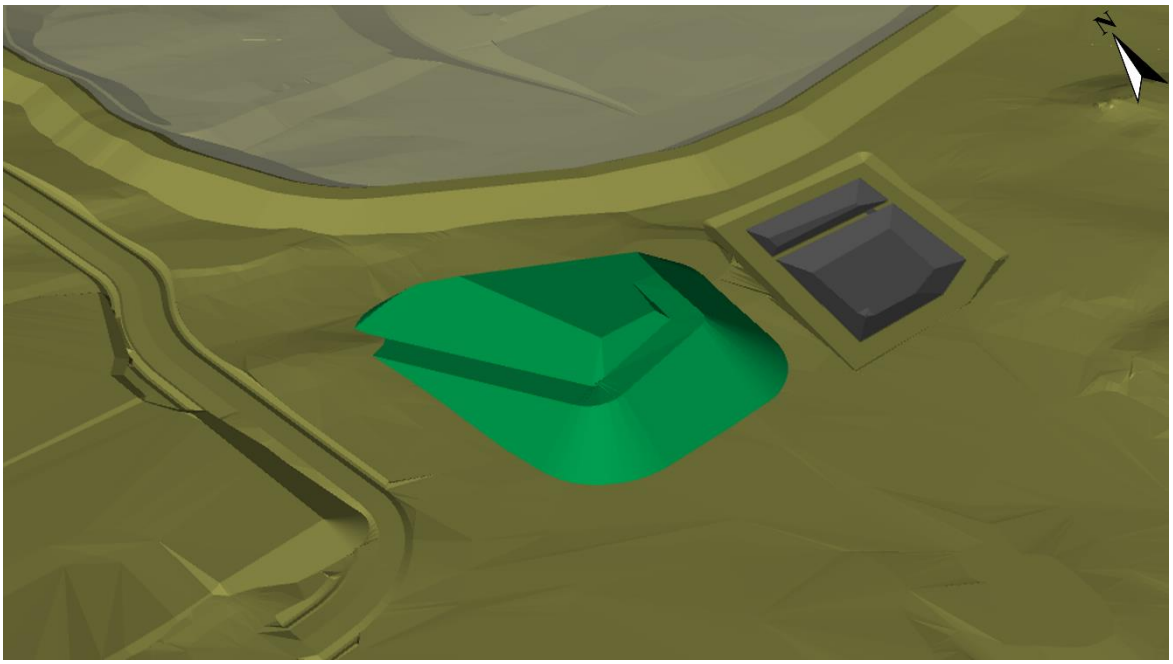
- **Phase 1**: Included static geochemical testing, such as acid-base accounting (ABA) and static leaching tests.
- **Phase 2**: Included kinetic geochemical testing, including humidity cell tests (HCTs), specific surface area measurements using gas adsorption and sieve analysis techniques, mineralogical determinations (X-ray diffraction (XRD)), and whole rock digestion methods.

The results of the Phase 1 and Phase 2 testing indicate that all of the altered andesite rock types present at the site are PAG and have the potential to leach a variety of constituents at concentrations that may exceed applicable water quality standards. The dacite rock type is essentially inert and is classified as non-PAG, but the dacite testing did indicate the potential to leach some metals at concentrations that exceed applicable water quality standards.

Due to the presence of PAG material, the Waste Rock Stockpile pad will be lined with an HDPE geomembrane to prevent migration of seepage into the native materials and groundwater regime. A Mine Contact Water Pond has been located downstream the Waste Rock and Ore Stockpile pads to receive any contact water from both areas where will be mixed with contact water from the mine dewatering systems. The collected contact water will be pumped from the Mine Contact Water Pond to the FTSF Contact Water Pond for subsequent treatment. The pad has a capacity to store approximately 250,000 tonnes of waste rock.

Figure 18.15 illustrates the waste rock stockpile.

Figure 18.15 – Waste Rock Stockpile



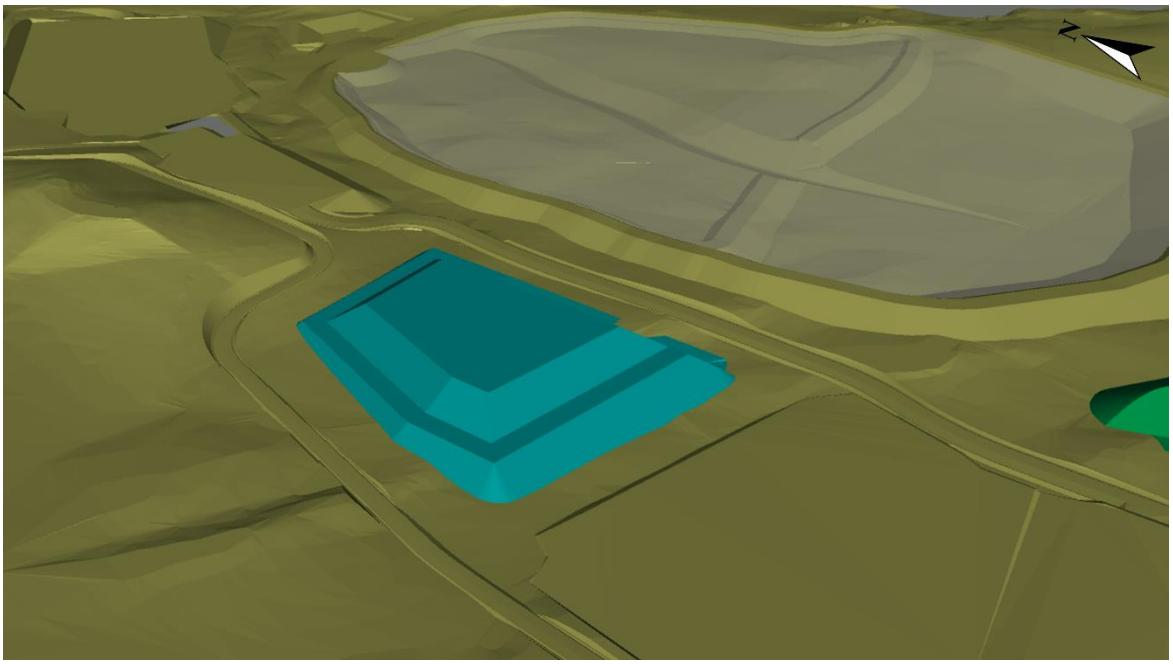
Source: NewFields, 2023

18.8.2 ORE STOCKPILE

The ore storage facility will comprise a prepared pad lined with an HDPE geomembrane to prevent migration of seepage into the native materials and groundwater regime. A set of perforated drain pipes will be installed above the liner and will be covered with a protection layer of coarse sand or gravel, with the objective of collecting the contact water received within the lined pad area. Similar to the Waste Rock Stockpile, the contact water will be directed to the Mine Contact Water Pond, and from there will be pumped to the FTSF Contact Water Pond for subsequent treatment. The pad has a capacity to store approximately 268,000 tonnes of ore.

Figure 18.16 illustrates the ore stockpile.

Figure 18.16 – Ore Stockpile



Source: NewFields, 2023

18.9 Water Management

The FTSF design includes a stormwater management system to minimise run-on to the FTSF by diverting non-contact flows around the facility. This system consists of stormwater diversion channels located along the FTSF and process plant area perimeter. The design also includes an internal drainage system for contact stormwater within the facility, an underdrain collection system to collect fluids draining from the tailings and an external underdrain collection pond. A climate analysis was completed by NewFields (2018a, 2018b) to establish a design criteria and later assumptions that were used for site-wide hydrologic modelling. Based on this study, the surface water management design was sized to manage the flows produced by a 24-hour/200-year storm, which is slightly more stringent than a 24-hour/100-year storm plus a 10% due to climate change. The stormwater management system components include permanent and temporary stormwater diversion channels and basins following best management practices (BMPs), internal drainage and freeboard in the FTSF, along with lined ponds for the FTSF, stockpiles and mine contact water.

18.9.1 Groundwater Management

Groundwater within the Project site occurs in two (2) zones or systems: Paramo System, and Bedrock System. The Paramo System is characterised by shallow groundwater in surficial soils and organic material of the high-altitude wetland system, while the Bedrock System is characterised by deep groundwater found in the fractures, cracks, and joints within the deeper bedrock. The

hydrogeologic studies completed for the Project have concluded that the Bedrock System does not interact with the Paramo System.

As the mine and associated ramp are constructed, groundwater will flow into these spaces and will be removed via pumping to maintain safe and efficient mining. The dewatering flows will be routed through a Mudwizard mud treatment system or alternatively through the surface dewatering pond, then through the Mine Contact Water Pond, the FTSF Contact Water Pond, and lastly the water treatment plant. Treated water from the water treatment plant will be recycled to the process and paste plants or discharged to Alumbre Creek. The overall effect of this pumping will be that a zone of dewatered rock, or a cone of depression, will form in the vicinity of the mine.

A numerical groundwater model was constructed to simulate mine dewatering and its effects on groundwater levels in the bedrock system and predict how much water will flow into the mine openings during the life of the mine. The model results indicate that the cone of depression in the bedrock system will be the largest at the end of mining but will only have minor effects on bedrock system in the immediate vicinity of the mine footprint, with no effects on more distant areas. The model predicts that there will be no effects to the water levels in the shallow Paramo system as the result of mine dewatering activity. Since groundwater in the Paramo system is perched above the mine workings, there is no need to remove water from the Paramo to safely complete the mining.

As described in Section 18.7.8, the engineering designs of the FTSF and contact water ponds were developed including an impervious barrier / liner to prevent seepage from the facilities to the underlying groundwater.

After mine closure, any remaining workings in the underground mine that have not been used to dispose of waste rock from surface will be filled with cemented tailings while the paste plant is in operation. Thereafter, mine dewatering will cease, and the mine will be allowed to fill with water. Adjacent groundwater levels will recover. This recovery of groundwater levels is predicted to occur over approximately 5 years, during which time the groundwater in the vicinity of the mine will flow toward these workings.

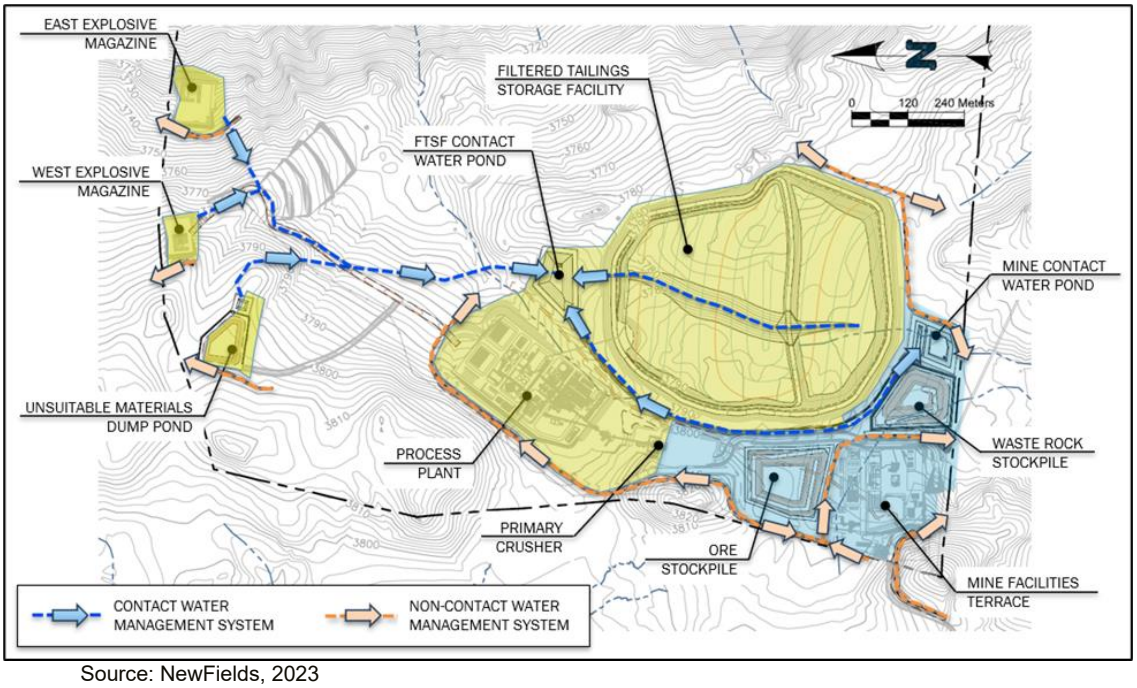
18.9.2 Non-Contact Water Management

The term non-contact water refers to any water flow that occurs outside the mining activity areas, including the process plant and FTSF. A perimeter storm water management system will capture non-contact water upstream of the mine and process areas and will divert the water around these areas and convey the water to the natural environment.

18.9.3 CONTACT WATER MANAGEMENT

The term contact water refers to any water flow that occurs inside the mining activity areas, including the process plant and FTSF. The surface contact water management system comprises two (2) contact water ponds, as illustrated in Figure 18.17, which receive mine dewatering and stormwater run-off flows.

Figure 18.17 – General Contact Water Management Schematic

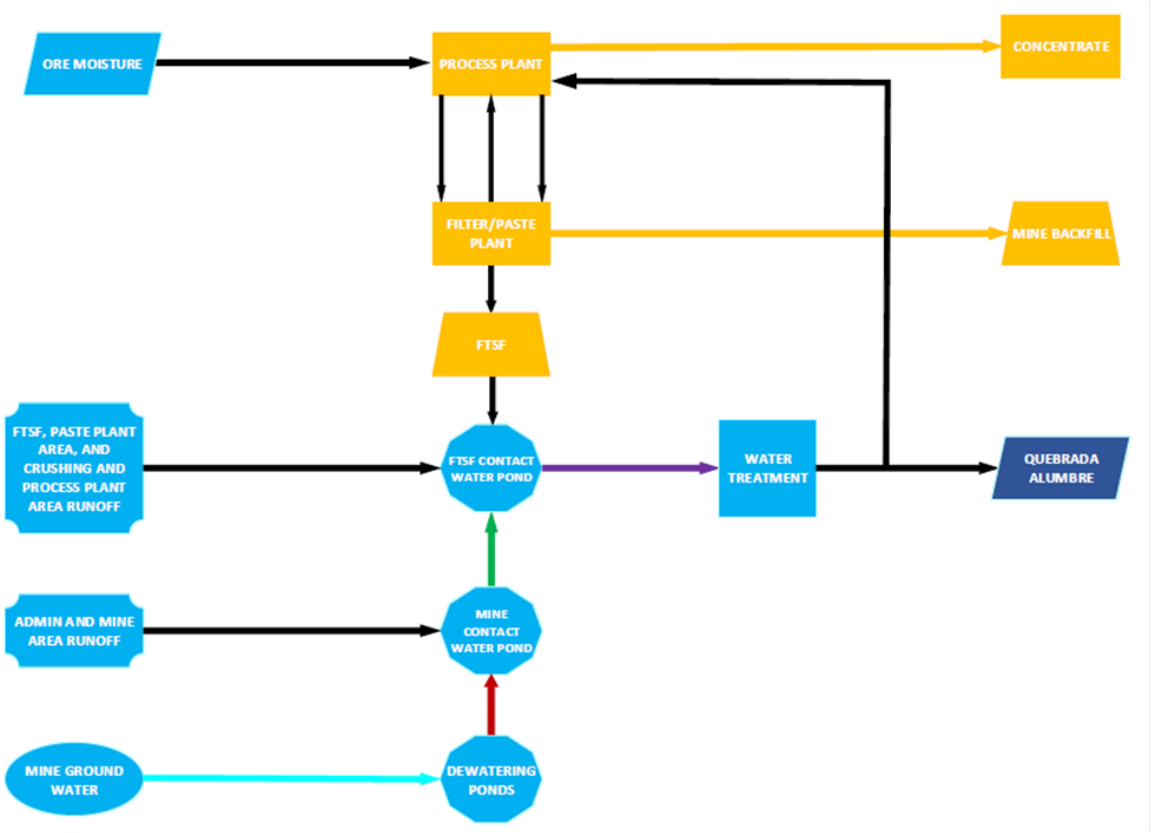


Source: NewFields, 2023

The mine is predicted to have a positive water balance; inflows exceed losses so contact water will be routed through the contact water ponds to the Main Water Treatment Plant (MWTP). Following treatment, use of the water to meet process and milling demands will be prioritised, and the excess flow will be discharged to Alumbre Creek in compliance with the Project environmental design criteria (Entrix, 2022). As a result of the process plant design, the requirement of fresh-water makeup during the operation phase of the mine will be minimum, as a large portion of the water within the system will be recovered through the thickening and filtration processes. Minor makeup water will be sourced from the Alumbre Creek.

A daily time-step water balance was completed to evaluate the amount of water expected to flow into, be released from and be stored within the mine. The water balance components and plans for water routing are illustrated in Figure 18.18.

Figure 18.18 – Water Balance Diagram



Source: NewFields, 2023

18.10 Water Treatment and Distribution

18.10.1 Main Water Treatment Plant

All water processed by the MWTP will be treated and discharged at a level that is in compliance with the maximum permissible limits (*límites máximos permitidos* or LMP) established by Ecuadorian regulations. The water will meet the quality standards for aquatic life and irrigation, or to match with the existing downstream conditions in the receptor environment.

The MWTP consists of two (2) modules:

- **Module 1**, which serves to remove solids and adjust pH, will be needed at the very beginning for the early works (Phase 1). This module will be designed to treat a flow rate of 150 m^3/h to account for any uncertainties that may arise during the early works. This module is to be installed at the earliest possible moment. Will serve to remove solids and adjust pH during the early works.
  - Coarse solids removal.

- pH adjustment.
- Sedimentation.
- Nitrification/Denitrification.
- Removal of suspended solids.
- Sludge dehydration.

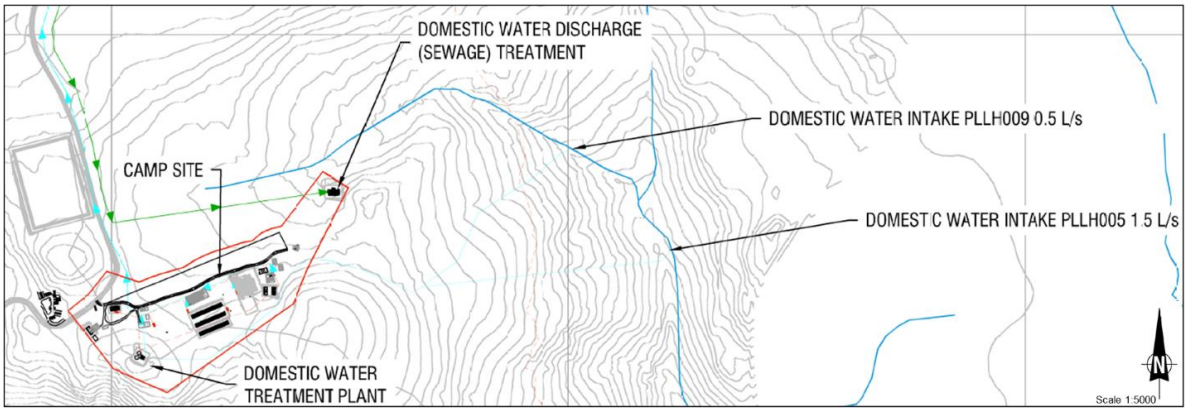
- **Module 2**, which consists of a mesh filtration stage followed by ultrafiltration, will be installed during preconstruction (Phase 2). Module 2 will be designed to treat a flow rate of 126 m^3/h, this was determined by the water balance model. During operations, the contact water will be pumped from the FTSF pond at a rate of 126 m^3/h and treated by both modules of the MWTP at that same flow rate. This module is installed during pre-construction and will serve to adequately treat contact water and ensure the discharge standards are met.
  - Mesh-Filtration.
  - Ultrafiltration.

18.10.2 DOMESTIC WATER

18.10.2.1 Domestic Water Treatment

Raw water will be captured at the points *Captacion Campamento Pinos* which will be supplied to Camp Los Pinos situated 1.16 km away and with an elevated height of 100 m. At Camp Los Pinos, the raw water will be treated at the Domestic Water Treatment Plant in order to remove suspended solids, TOCs, perform pH adjustment and disinfection. The domestic water will meet all the quality standard for human consumption according to Ecuadorian regulation.

Figure 18.19 – Off-Site Infrastructure Domestic Water Intake



Source: GMining, 2022

18.10.2.2 Domestic Water Distribution

After treatment at Camp Los Pinos, the domestic water is retained in domestic water tank storage and will be distributed at the following places:

- Administrative accommodation (one (1) building) – Water heaters will be installed.
- General accommodation (two (2) buildings) – Water heaters will be installed.
- Kitchen and lunchroom.
- Administrative office.
- Medical centre.

From the Camp domestic water tank, a portion of the water is fed via pipelines to the on-site domestic water tank located 5.1 km to the north and at an elevated location next to the processing plant. This tank is used to serve the following places:

- Industrial area at the mine entrance for domestic water hook-up and safety showers.
- Industrial area for domestic water hook-up and safety showers at processing and paste plants.
- Infrastructure buildings at the mine entrance and processing area.
- Infrastructure buildings at the processing area.

Note: Domestic water is required at explosive magazine for safety showers. The water will be an overflow from the Fire Water Tank at the Explosives area.

18.10.3 POTABLE WATER

Domestic water will be further treated into standard potable water by a reverse osmosis process located at the main camp kitchen. Potable water for facilities outside the main camp will be bottled and transported for distribution.

18.10.4 FIRE PROTECTION

Fire protection will be compliant with the requirements of NFPA statements and/or FM Global recommendations. The main equipment for fire protection is:

- Fire pumping station: pump controller, diesel pump, jockey pump, foam bladder tank.
- Supply and installation of portable extinguishers.
- Supply and installation of fire protection tank.
- Construction of fire pump enclosure.
- Supply and installation of fire pumps.

- Construction of water network and suppression systems, consisting of:
  - Buried HDPE pipes.
  - Surface pipes.
  - Sprinklers systems.
  - Cabinets systems.
  - Hydrants.

18.10.4.1 On-Site

The on-site fire water tank and pumping station is located at an elevated location, next to the Process terrace, where it will be installed as an overflow from the on-site domestic water tank. The on-site fire pump station will supply the building sprinkler systems, cabinet systems, hydrants at:

- Industrial area at the mine entrance.
- Infrastructure buildings at the mine entrance.
- Industrial area at the processing plant and the paste plant.
- Infrastructure buildings at the processing area.

The second on-site fire water tank and pumping station is located at Explosive magazine and is used to serve the industrial area at the explosive magazine.

18.10.4.2 Los Pinos Camp

A Camp fire water tank and Camp fire pump room will be installed adjacent to the Camp domestic water treatment plant. The fire water tank will be installed as an overflow from the Camp domestic water tank.

The Camp fire water tank will supply the building sprinkler systems, cabinet systems, hydrants at:

- Administrative accommodations (one (1) building).
- General accommodations (two (2) buildings).
- Kitchen and lunchroom.
- Administrative office.
- Medical centre.
- Recreation room.
- Camp guard house.

18.11 Waste Management

This section outlines the waste management materials generated during mine construction and operation, covering unsuitable materials, non-hazardous waste, recyclable non-hazardous waste, hazardous waste, and medical waste.

Unsuitable Materials:

Earthworks and non-acid-generating sterile waste will be stored in a designated dump with perimeter berms and underdrains to contain runoff. Contact water will be collected in a lined settling pond (~310 m³ capacity) and treated before discharge.

Non-Hazardous Waste:

Non-hazardous waste (debris, plastics, food, etc.) will be temporarily stored on-site for recycling or disposal by authorised contractors.

Recyclable Non-Hazardous Waste:

Recyclables will be collected and transported off-site by contractors. Any chemical or petroleum spills will be contained, collected, and disposed of at licensed facilities.

Hazardous Waste:

Stored in a dedicated on-site facility, hazardous waste will be transported by licensed handlers to approved disposal sites, ensuring compliance with Ecuadorian regulations.

Medical Waste:

Clinic and first aid medical waste will be collected and transported off-site to authorised disposal facilities.

18.12 Power Supply and Distribution

18.12.1 Transmission Lines and Power Distribution

The primary electrical power supply will be provided by the local utility company CENTROSUR, which serves the Azuay Province. CENTROSUR is connected to the national transmission grid at 230/138 kV and distributes power through its 69 kV and 22 kV networks. The main 69 kV transmission line to the mine site will interconnect with an existing CENTROSUR transmission line running between Lentag and Cuenca. A new interconnection switchyard will be constructed near Victoria del Portete to facilitate this connection.

Approximately 17 km of new 69 kV transmission line will be installed up to the plant site.

The incoming 69 kV to site will terminate into the main substation with two (2) fully redundant step-down transformers of 16/20 MVA, with a secondary voltage of 13.8 kV. 13.8 kV switchgear inside an e-room located within the main substation boundaries will provide multiple feeders to connect all site facilities.

It is not unexpected for the grid power to be down periodically. However, to minimise potential disruption to the operations, the Project will proceed with the design, construction, and operation of a diesel-based power generation plant, configured with N+2 redundancy and a maximum generation capacity of 16.5 MW. The plant will be located near electrical substation (S/E for *subestación eléctrica*) #3, and must comply with the following technical specifications:

- N+2 configuration to ensure operational redundancy.
- A 20/25 MVA KNAN/KNAF step-up transformer with an output voltage of 69 kV.
- A 69 kV switching bay that allows for connection to both CENTROSUR (for connection to the grid), and the diesel generation plant.

The existing 69 kV transmission line, approximately 17 km in length, between S/E #13 and the Project's substation, will be utilised for power generated by the diesel-powered plant.

The Project has planned to have grid power available 90% of the time, with the power plant providing power 10% of the time.

The Project will rely on mobile generators installed at the construction site. These generators will provide temporary power until the diesel generation plant, transmission line, and processing plant substation are completed and operational.

The 22 kV distribution line from Los Pinos will provide energy to the camp facilities, with a maximum connection capacity of 1 MVA. The existing single-phase line will be upgraded to a three-phase line to meet this requirement. It is worth noting that this line is also connected to local electrical loads.

18.12.2 Site Power Demand

Electrical power demand was determined from the underground mine, process design and equipment requirements, typical support services and infrastructure power draws, dewatering pumping, and treatment plants, and is summarised in Table 18.1.

Table 18.1 – Site Electrical Power Demand

WBS	Description	Power Installed (kW)	Power Installed (kVA)
1000	Mine General	0	0
2000	Underground Mine	1656	1852
3000	Ore Handling	220	254
4000	Processing Plant	5975	6490
5000	Filtered Tailings / Water Treatment Facilities	1991	2328
6000	On-Site Infrastructure, Site-Wide General	279	325
7000	Off-Site Infrastructure	61	70
	Total	**10182**	**11319**

18.12.3 EMERGENCY POWER SUPPLY

The mine site has four (4) emergency diesel motor generators (two (2) in the process plant and two (2) in the portal). Each generator has a capacity of 3125 kVA at 13.8 kV. Additional future provision has been made for one (1) generator in the process plant.

The quantity and size of emergency generators to be validated in the next phase of the project, when the emergency load list is confirmed.

The facilities for the camp will not depend on the generators installed in the processing plant or the mine. Therefore, an additional generator of 1400 kVA is considered, with a transformer from 13.8 kV to 22 kV at its output for the secondary medium-voltage distribution network.

18.13 Communications

The communication network on site will be a modular ethernet network for ease of future upgrade and functionality, with connection to internet and telecommunications providers dictated by their network speeds. The architecture will be a single physical layer network, segmented into several virtual networks to enable segregation and isolation of different types of network traffic.

The network backbone will consist of a multicore single mode fibre optic cable using one (1) dedicated network sharing automation, telecommunication, corporate network, fire detection system, CCTV system and UG communication. The use of a fibre network will ensure that communications will not be affected by interference from any electrical or magnetic sources along the various cabling runs. Fibre network is also the most efficient transmission mechanism with the least performance loss over long distances.

The multicore fibre optic cable network will provide redundancy in the form of a multi route configuration and the flexibility to create a mesh network design. A mesh network is a local network topology in which the infrastructure nodes (i.e., bridges, switches, and other infrastructure devices) connect directly, dynamically, and non-hierarchically to as many other nodes as possible and cooperate with one another to efficiently route data from / to clients.

The corporate network communication services are:

- Wired/Wireless Phones and System Server.
- Process and Security Camera System.
- Access Control System (gate, door).
- Fire Detection.

LTE signal for above ground activity will be provided through a local provider. A private LTE network will be deployed for the underground mine to communicate with all workers and equipment. Wi-Fi access point will be installed over the site for complete coverage at process plant and portal area.

18.14 Los Pinos Camp

18.14.1 Existing Infrastructure

Los Pinos Camp is located roughly 5 km South of the mine site. There are some existing installations, such as camp accommodation, kitchen and lunchroom, core shack, fab shop, a recycling facility, a fuel station, a basic guard house for the camp and a welcome centre (*centro de difusión*). It is planned to refurbish the current camp installations to lodge approximately 85 people for the early works. The *centro de difusión* will be refurbished and serve to control people coming to site. Figure 18.20 illustrates the general arrangement of existing installations.

Figure 18.20 – Existing Los Pinos Infrastructure

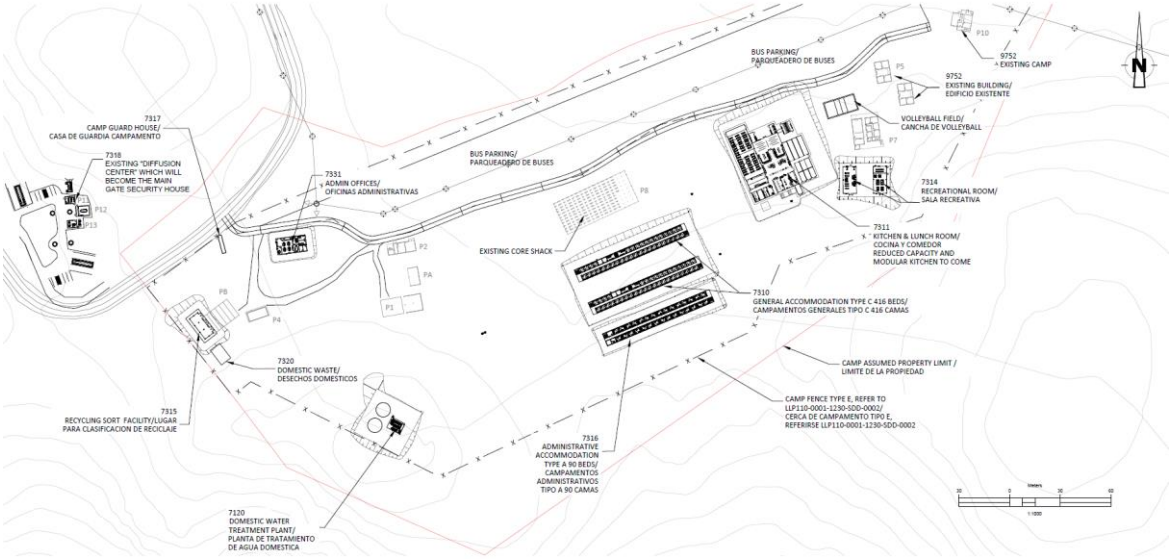


Source: DPM, 2022

18.14.2 PLANNED NEW INFRASTRUCTURE

It is planned to add accommodation to lodge 500 people at the same time on site (Figure 18.21). This includes the camps (three (3) new buildings), the kitchen and lunchroom, a recreation centre, and a camp admin office. A barbed wire fence will be installed around the camp to contain and secure the Property. A camp guard house will be located at the entrance to the camp area.

Figure 18.21 – Los Pinos New Installations



Source: DPM, 2022

18.15 Off-Site Infrastructure

DPM plans to construct off-site buildings, consisting of a single storey assay lab as a stand-alone structure, and a second administration building.

Location of the buildings are expected to be within 100 km of the site. Final location will be selected in the next phase of engineering.

18.15.1 Assay Lab

The sample preparation and assay laboratory facility is a single storey building. The facility will include a complete equipment package to comply with international standards and best practices. It will serve as an off-site testing and analytical facility for the geology department.

18.15.2 General Offices / Main Administration Building

This is a single storey building with general offices, and administration offices along with personnel from various departments Several office spaces related to the process plant support and management are also included.

18.16 Transportation and Logistics

The current regional and national infrastructure is adequate to access the Project site through a well-established network of existing major ports and roadways. The road between San Gerardo and the Project site will be upgraded during the early stages of the Project and will be ready to support the operations phase of the Project. Regional infrastructure (ports and roads) is described in Section 5 of this Report.

18.16.1 Personnel Transportation

On-site vehicles will transport employees from the bus unloading area adjacent to the change house to their place of work. Pickups will be provided to transport supervisors and maintenance personnel around the site. No private vehicles will be allowed on site.

18.16.2 Concentrate Storage, Handling and Transport

The copper concentrate will be directly loaded into standard 20-foot shipping lined containers prior to trucking to the port for export over the life of the operations. Handling equipment will be provided on-site for loading and handling the containers.

The pyrite concentrate filtration and storage area is located inside an enclosed building at the Project site. The pyrite concentrate will be loaded into standard 20-foot shipping lined containers prior to trucking to the port for export over the life of the operation. Handling equipment will be provided on-site for loading and handling the containers.

If facilities for bulk concentrate handling become available at the ports of export, bulk export of concentrate will be considered.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

Loma Larga will produce two (2) different concentrates, specifically a pyrite concentrate (also referred to as gold-pyrite), and a high-grade gold-copper concentrate. The gold-pyrite concentrate will represent the majority of the life-of-mine concentrate production (80% by weight), and it is expected to contain 30.3 g/t of gold and 110.2 g/t of silver on average over the LOM, with no significant deleterious elements. The gold-copper concentrate is expected to grade 28.1 g/t of gold, 8.80% copper, and 531.7 g/t of silver on average over the LOM. Deleterious elements in the gold-copper concentrate include 3.47% arsenic, 60 ppm of mercury, and 0.20% antimony.

Loma Larga's pyrite concentrate can be considered clean from an impurity perspective and relatively low in gold content. The market for these types of materials is generally gold smelters/roasters and refineries, copper smelters, especially where sulphur fuel may be required and, to a lesser extent, blending facilities where trading companies would mix such clean material with other complex gold and copper concentrates.

Traditionally Western smelters look for pyrite concentrates with a higher gold content (i.e., minimum 3 oz/t or 93 g/t), and as a result it is the Chinese market that consumes most of the lower grade pyrite concentrates exported globally. These concentrates are mainly processed by gold smelters and roasters located in Shangdong and Henan provinces. However, Chinese copper smelters have increasingly become more interested in pyrite concentrates, as their precious metal refining capacity has increased significantly. In addition, Chinese copper smelters use clean pyrite concentrates to balance and regulate their average feed intake.

The potential market for Loma Larga gold-copper concentrate is more focused, due to the expected arsenic level. The specification of Loma Larga's gold-copper concentrate, with >15 g/dmt gold and <3.5% arsenic, qualifies it to be imported to China as a gold concentrate, and there are several copper smelters in China which are able to process this quality of concentrate. In addition, there are a small number of Western custom copper smelters, including Glencore's Horne copper smelter in Canada and Sinomine's Tsumeb copper smelter in Namibia, who would be positioned to treat concentrates with such arsenic levels.

Blending facilities also represent a potential market for Loma Larga's gold-copper concentrate. Blending operations have proliferated in recent years as a solution to both the increasing level of impurities, especially arsenic, and the introduction of regulatory restrictions of harmful elements in destination markets. Originally, the largest blending facilities were in Mexico, Peru, and other countries in South America, as the majority of complex concentrates are found there. Several years ago, blending operations emerged closer to destination markets, in places such as Taiwan, South Korea, Spain and the Black Sea. More recently, China has allowed the development of "on shore" blending facilities, where local smelters may bring complex concentrates into bonded warehouses

where they are blended with clean concentrates before being imported into China within established impurity limits.

DPM has extensive experience in selling both pyrite concentrates and complex gold-copper concentrates from its Chelopech mine in Bulgaria. Chelopech produces a larger quantity of both types of concentrates, of a similar or lower-grade quality. Discussions about Loma Larga concentrates with DPM's current buyers of Chelopech concentrates have provided DPM with confidence that there is an available market willing to enter into long-term off-take agreements for the Loma Larga concentrates.

19.2 Treatment Charges and Other Selling Costs

Treatment charges and other selling costs for copper concentrate and pyrite concentrate produced at Loma Larga are based on market terms provided by DPM.

19.2.1 Metal Payability

The payable rates for each concentrate are outlined below:

- Copper concentrate:
  - Gold: 96.0% fixed.
  - Copper: Basis 96.5% payable subject to a -1% minimum deduction.
  - Silver: 90.0% fixed.
- Pyrite concentrate:
  - Gold: Basis 97.0% payable subject to a -1 g/dmt minimum deduction.
  - Copper: not payable.
  - Silver: Basis 95.0% payable subject to a -30 g/dmt minimum deduction.

Table 19.1 presents the average payable rates of the different metals in each concentrate over the LOM.

Table 19.1 – Average LOM Payable Rates per Metal per Concentrate

Payable Rates	Units	Copper Concentrate	Pyrite Concentrate
Gold	%	96.0	96.5
Copper	%	88.7	0.0
Silver	%	90.0	72.7

19.2.2 Concentrate Freight Costs

Transportation costs consider that the two (2) types of concentrates will each be transported in containers from the mine to their destination in China, via the port of Machala. The estimated freight costs are US$145/wmt for each of copper concentrate and pyrite concentrate.

Transport losses of 0.2% on a dmt basis were applied to each concentrate.

19.2.3 Treatment Charges and Refining Charges

Treatment Charges (TCs) of US$125/dmt were applied to each concentrate. The Refining Charges (RCs) applied for each metal in each concentrate are summarised in Table 19.2.

Table 19.2 – RCs for the Different Metals in Each Concentrate

Refining Charges	Units	Copper Concentrate	Pyrite Concentrate
Gold	$/oz	5.00	5.00
Copper	$/dmt conc.	See Note[1]	0.0
Silver	$/oz	0.50	0.50

1. Formula for Copper RC for Copper Concentrate: TC/1000*Cu Conc Grade*Cu Payable Rate*2204.62262

19.2.4 Penalties

No penalties were considered for the pyrite concentrate. However, penalties for arsenic, mercury, and antimony content were considered for the copper concentrate based on the following:

- Arsenic: US$3/dmt for each 0.1% As above 0.3% As.
- Mercury: US$0.2/dmt for each 1 ppm Hg above 20 ppm Hg.
- Antimony: US$5/dmt for each 0.1% Sb above 0.1% Sb.

19.3 Commodity Prices

The following price projections have been adopted for the Study base case as presented in Table 19.3. The net revenue of the Project is dominated by gold, which accounts for 87% of the revenue over the LOM. The selected gold price is conservative compared to the three-year trailing average of US$ 2,195/oz or the Q1 2025 average spot price of US$ 2,860/oz.

Table 19.3 – Commodity Prices Selected

Payable Rates	Units	Financial Model
Gold	US$/oz	1,900.00
Silver	US$/oz	27.50
Copper	US$/lb	4.00

19.4 Contracts and Off- Takes

DPM has no current contracts for project development, mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts or arrangements related to the Loma Larga Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section should be read in conjunction with, and is subject to, the section entitled "Post-Effective Date Events" at the beginning of this Technical Report.

20.1 Introduction

This Section presents available and relevant information regarding environmental studies, environmental permitting, and social or community aspects related to the Loma Larga Project.

The Loma Larga Project is located in the province of Azuay, in the western Andes Mountains, 30 km southwest of the city of Cuenca and approximately 15 km north of the town of Girón. The Project is one of five mining projects in the country declared as "National Strategic Projects" by the Ecuadorian Government (Ministry of Mines of Ecuador, 2016 and 2018).

The Project is located within the Cristal concession mineral right granted to DPMECUADOR S.A. This is a concession resulting from the division of the Cerro Casco (Code 101580) and Río Falso (Code 101577), and the subsequent amalgamation with Cristal (Code 102195). The existing road between San Gerardo and the Project site will be upgraded during the early stages of the Project construction.

20.2 Environmental Setting

The concession area of the Project is located in the cantons of Cuenca, Girón and San Fernando. The population in the Project's area of influence is mainly rural (excluding the city of Cuenca). The results of the social baseline study show that the number of people living below the poverty line is above the national average. Most of the people work in agriculture and cattle raising.

The Project is close to the continental divide between the Atlantic Ocean watershed and the Pacific Ocean watershed. The three main watercourses of the northern local watersheds (Bermejos river, Irquis [Quinuahuaycu] river, and Portete river) drain to the Atlantic Ocean. The three main watercourses of the southern local watersheds (Falso river, Cristal river, and Cristal- Alumbre creek) are tributaries of the Zhurucay river, which flows to the Pacific Ocean.

The Project site consists of desert plains and rugged valleys, mainly formed by glaciers, with an altitude ranging from 3,500 m ASL to 3,960 m ASL. Vegetation is sparse and typical of the Andean region above tree line. Much of the property is covered by Andean *páramo*, a type of moorland vegetation. There are stands of pine on hillsides adjacent to the concessions, most of which were planted as part of a forestation project. Terrestrial fauna groups for which baseline data have been collected include mammals, amphibians and reptiles, birds, and insects. Aquatic faunal groups for which baseline data have been collected include fish and macroinvertebrates.

An environmental baseline study was carried out using data records compiled from approximately 15 years of historical data pertaining to climate, air quality, noise, hydrology, hydrogeology, and the biological environment. This baseline study was used to evaluate potential impacts of the Project and to inform the development of the Environmental Management Plan. Key results obtained from the environmental baseline study, and reported in the Environmental Impact Assessment, are described below.

20.2.1 PHYSICAL ENVIRONMENT

20.2.1.1 Air Quality

Air quality sampling was conducted for a period of five (5) years at the mine site (sampling locations included the Pinos camp and the future site of the underground mine ramp), as well as in Chumblín and San Gerardo. Sampling was conducted for 24 hours, and the results were compared with the maximum permissible values provided in Annex 4 of Ministerial Agreement 097-A, and levels were below the limits established for each type of contaminant. The sampling locations correspond to areas where there were no sources generating pollution at the time of the study. Once mining begins, air quality monitoring will resume according to the Environmental Management Plan for the Project (see Section 20.3.2).

20.2.1.2 Hydrology and Groundwater

According to the Pfafstetter method, which is used to code river basins, two (2) watersheds located within the Project area of influence were considered in the baseline characterisation study. The hydrological analysis was conducted with available data from the existing hydrological stations.

The study results showed a marked seasonality in both watersheds where the maximum discharge rates were observed from January to June, and the minimum from August to September. The area is dominated by moderate to low streamflow, showing homogeneous hydrological behavior throughout the Project area. The annual streamflow varies between 40% and 70% of annual precipitation.

For groundwater evaluation, piezometers were installed in the Project area with a total of 21 sampling points. Hydrogeological field data collection included measurement of water table elevations and hydraulic conductivities. The collected data were used to develop a hydrogeological model. The main finding was that groundwater is produced in two (2) separate aquifers in the Project area: a deep aquifer and a shallow Moorland aquifer. The deep aquifer is controlled by fracture densities and alteration types, where silica rock presented the highest hydraulic conductivities and argillic altered rock showed the lowest hydraulic conductivities. Permeability in the deep aquifer is considered to be low to very low compared with general hydraulic conductivity values. The hydraulic conductivity values obtained from the field work were the basis of the groundwater conceptual model. The deep groundwater table is approximately 50 m below surface with descending gradients.

Water level gradient of the deep aquifer is generally towards the east, indicating that water flows from west to east in the Project area. Results of the assessment showed that the shallow and deep aquifers are not connected.

20.2.1.3 Water Quality

Water quality analysis was performed using data from 30 surface water sampling points covering all the streams inside the Project area of influence. Groundwater samples were also analysed for water quality. Water quality at the Project site has been monitored for 13 years, in accordance with the parameters and specifications recommended by the Ecuadorian Environmental and Water Authority in the guide for development of Environmental Impact Assessments for mining activities. Samples were taken in low, medium, and high streamflow conditions at each sampling point.

Spatial analysis showed that most of the sampling points presented similar data values for the parameters tested, except for two points where pH was significantly lower and the values for sulphates, electrical conductivity, and most of the cations were relatively high in comparison to the other sampling points. Several parameters had concentrations above the Water Quality Criteria presented in Annex 1 of Ministerial Agreement 097-A. These high concentrations may be attributed to the native soil at the site, which is derived from volcanic ash and contains organo-metallic compounds. During very wet conditions, these organo-metallic compounds can be transported into the streams. Dissolved and total cation analysis showed that for each monitored point, every parameter concentration was similar over time in all cases. This uniformity is due to the presence of relatively low suspended sediment at the sampling points. Concentrations found in an adjacent catchment showed a similar behaviour to the catchments at the Project. Once mining commences, the adjacent catchment, which will not be impacted by mining activities, could be used as a benchmark for monitoring programs included in the Environmental Management Plan for the Project, as indicated in the Environmental Impact Assessment.

20.2.1.4 Soil Quality

Two (2) types of soil are predominant in the Project area: Andosols and Histosols. Thirteen sampling points were selected, in which, in accordance with local legislation, a grid of 20 subsamples was made for each point. Following the national regulations, Soil Quality Criteria (SQC) is categorised in three groups: a) general parameters, b) inorganic parameters, and c) organic parameters.

General parameters measured include pH, electrical conductivity, and sodium adsorption ratio. Of these parameters, only pH was below the SQC (6-8) with an average of 4.8 +/- 0.3. Although, these values are outside of the SQC, they are considered typical for an Andean *páramo* ecosystem in southern Ecuador, where soils have high organic matter content resulting in low pH values.

Results for inorganic element parameters indicated values lower than the SQC, except for S, As, Cu, and Pb. In terms of S, all the samples were very elevated (1,508 ± 539 mg/kg, on average)

compared to the maximum permissible limit (250 mg/kg). High S values can be correlated with the high organic matter content in the soil. Many of the samples had elevated concentrations of As, Cu, and Pb relative to the SQC. These elevated concentrations may be caused by hydrolysis resulting from chemical rock weathering or by transportation to the soil surface horizons by vegetation, water flow, or exposed by rock outcrops. Based on the baseline characterisation study, a recommendation was made in the Environmental Impact Assessment to conduct more detailed studies to assess the reason(s) for higher concentrations in As, Cu, and Pb.

20.2.1.5 Landscape

The Canter Methodology was used to assess the natural landscape. The categories or factors assessed included: natural state, scarcity, aesthetics, and importance for conservation.

The geomorphology of the Project area is characterised by six (6) types of well defined landscapes: ancient lava flows, dacitic flows, swampy hollows of glacial-periglacial origin, glacial valley bottom, glacial lagoons, and wetlands.

The Project is located in the western Andes Mountains in Ecuador, defined by high peaks, and has been slightly modified by human activity. Agriculture is very limited and is concentrated in the lower sectors of the studied area.

20.2.2 BIODIVERSITY ENVIRONMENT

Ecuador is one of 17 megadiverse countries in the world. According to the Ecuadorian government, the country has 18,198 species of vascular plants, of which 25% are endemic. This floristic diversity is distributed in 91 plant ecosystems. The moorlands (high mountain ecosystems located in the Neotropics) comprise 5% of the land area in Ecuador. These areas are of biological importance due to high levels of endemism.

20.2.2.1 Flora

Quantitative and qualitative sampling methods were used to gather information on flora at the Project. During the sampling program carried out in December 2021 (the last monitoring event used to inform the EIA for the Project), 191 different species grouped into 125 genera and 60 families were recorded.

The composition of plants in the evaluated area is defined by dominant species that have previously been found in similar habitats in the moorlands of southern Ecuador. Among the most representative species are *Calamagrostis intermedia* and *Paspalum bonplandianum*, both of the Poaceae family.

20.2.2.2 Terrestrial Fauna

a. Mammals

Information was collected at three (3) quantitative and eight (8) qualitative monitoring points. The methods used to collect information were capture of micromammals, trap cameras, and observations made during walks.

In the Project area, a total of 12 mammalian species were reported, grouped into eight (8) families and seven (7) orders. Based on the Red List for Ecuador (Tirira, 2021), the baseline characterisation study identified two (2) species, *Caenolestes caniventer* and *Lycalopex culpaeus,* that are reported in the Vulnerable (VU) category; three (3) species, *Cryptotis montivagus*, *Odocoileus virginianus* and *Sylvilagus andinus*, in the Near Threatened (NT) category, one (1) species, *Sigmodon inopinatus* in the Endangered (EN) category; and the remainder of the species identified were in the Least Concern (LC) category. In addition, four (4) endemic species were registered, representing 7.7% of the endemic species reported for Ecuador.

b. Avifauna (Birds)

Both quantitative and qualitative sampling for birds was conducted. For quantitative testing, mist nets and counting points were established at sites that represented a diversity of microhabitats, as well as sites with Polylepis forests, grassland areas, and conserved shrub vegetation. Essential ecological characteristics such as food resources, habitats, shelter, and nesting sites were also considered during site selection. The same sites were also used for qualitative analyses.

In total, 58 bird species have been recorded at the Project site, grouped into 22 families and 11 orders, representing 10.6% of the total 562 bird species identified in the zoogeographic region in Ecuador where the Project site is located. The family with the highest number of recorded species is Trochilidae (hummingbirds), with ten (10) species; followed by Thraupidae (tanagers), with eight (8) species; and Furnaridae (spintails) and Tyrannidae (flytrap), with six (6) species each. The remaining families are represented by less than six (6) species.

c. Herpetofauna (Reptiles and Amphibians)

Information was collected at three (3) quantitative and eight (8) qualitative points within the Project area, and a total of 307 individuals (286 amphibians and 21 reptiles) were recorded. In the class Amphibia, seven (7) species belonging to three (3) families, Strabomantidae, Hemiphractidae, and Centrolenidae, were recorded. Within the class Reptilia, a species belonging to the family Gymnophthalmidae of the order Squamata and Iguanidae was recorded.

Four (4) endemic species were recorded: *Gastrotheca pseustes*, *Pristimantis cryophilius*, *Pristimantis lutzae,* and *Stenocercus festae*. These species are considered to have high sensitivity due to their endemic conservation status, defined by the International Union for Conservation of Nature (IUCN) and the Red List of Ecuador (https://bioweb.bio/faunaweb/reptiliaweb, 2021).

d. Entomofauna (Insects)

Quantitative and qualitative methods were used to collect representative information on the entomofauna populations in the Project area. Quantitative sampling consisted of pitfall traps for beetles, including Van Someren Rydon aerial traps (VSR), light traps, and dish traps.

A total of 112 beetles were recorded in the Project area, which were categorised into nine (9) species, eight (8) genera, eight (8) tribes, and three (3) families. For the study of Lepidoptera (butterflies and moths), two (2) individuals of different species were recorded and distributed in two (2) genera, two (2) tribes, and two (2) families.

e. Aquatic Fauna

Ichthyofauna (Fish)

For the assessment of the fish community, active and passive techniques appropriate for the sampling sites were used, including manual nets (triangular net).

Based on the field campaigns carried out during 2018, 2019, 2020, and 2021, a total of one (1) species of rainbow trout (*Oncorhynchus mykiss*) has been registered in the Project area. In 2018 and 2019, 21 and 22 individuals were recorded, respectively, and during 2020 and 2021, 16 and 9 individuals were recorded, respectively.

Aquatic Macroinvertebrates

Data collection on the benthic fauna populations at the site was carried out quantitatively using a D Net at eight (8) sampling points chosen to represent the various microhabitats in the water bodies, the influence of the riparian vegetation, and the substrate type. Manual collection of benthic fauna was also carried out on stones and leaf litter, which were found at the bottom, on the surface, and on the banks of the rivers/estuaries.

A total of 865 individuals were registered within the water bodies of the Project. The dominant families of macroinvertebrates include Simuliidae, Hyalellidae and Baetidae, with the presence varying depending on the specific sampling location.

20.2.3 SOCIO-ECONOMIC

Communities within the direct area of influence include Duraznos, Cristal Aguarongos, and Comuna Sombrederas de Chumblín. Communities in the indirect area of influence include the parishes of Victoria del Portete, San Gerardo, and Chumblín.

The Organic Code of the Environment (*Código Orgánico del Ambiente* - COA for its acronym in Spanish) includes issues such as climate change, protected areas, wildlife, forest heritage, environmental quality, waste management, environmental incentives, marine coastal zone, mangroves, access to energy resources, biosafety, and bio-trade, among others.

Following specifications outlined in COA, the cantons of Cuenca, Girón, and San Fernando are considered indirect areas of influence for the Project. Given that Cuenca is one of the largest cities in Ecuador, its role at the regional level is particularly valued. DPM has included the city of Cuenca in the Project's area of influence related to socio-economic matters as it is an economic hub in the Province of Azuay. DPM expects that the city will provide services and materials required for construction and operation of the Project that cannot be sourced from closer towns. Due to the size and complexity of the Cuenca economy, the impacts of the Project are expected to be positive.

According to the National Institute of Statistics and Census (Instituto Nacional de Estadistica y Censos – INEC for its acronym in Spanish), in 2022 the province of Azuay had a population of 801,609 inhabitants, the canton of Cuenca 596,101 inhabitants, the canton of Girón 12,182 inhabitants, and the canton of San Fernando 3,961 inhabitants. In total, the female population accounted for 52.7%.

20.2.4 ARCHAEOLOGY

An archaeological study was conducted from May to June 2020 to determine if pre-Hispanic artifacts exist in the Project area. The results were presented in a report issued to the National Institute of Cultural Heritage, Urban Planning Zone 6 (Cuenca). The report indicates that no pre-Hispanic artifacts were found in situ and that the environmental conditions are not favourable for the preservation of archaeological artifacts.

20.3 Environmental Studies and Management Plans

20.3.1 ENVIRONMENTAL IMPACT ASSESSMENT

An Environmental Impact Assessment (EIA) was prepared for the Project and submitted to the Ecuadorian Ministry of Environment, Water and Ecological Transition (*Ministerio del Ambiente, Agua y Transición Ecológica* - MAATE for its acronym in Spanish) in April 2021 to initiate the review and approval process. The EIA describes the physical, biological, and socio-economic baseline conditions of the Project area. The EIA specified the area of influence of the Project (i.e., the

geographical area potentially affected), and how various Project activities could affect the physical, biological, and socio-economic aspects within the area of influence.

The Intersection Certificate is a document issued by MAATE, to specify if a project intersects with an Environmental Protected Area belonging to the National System of Protected Areas, Protective Forests, or State Forest Heritage. According to the Intersection Certificate issued on April 9, 2021 by MAATE, the area of exploitation defined in the EIA overlaps with natural forests and vegetation areas comprised of Chorro and 15 areas within the Paute River Basin. In addition, the Project overlaps with the Macizo de Cajas Biosphere Reserve, which is categorised as an area for biodiversity conservation.

Given that the Project intersects an Environmental Protected Area, the process of EIA review involved the following steps:

- Determination of potential environmental impacts and mitigation measures and preparation of report.
- Registration as a hazardous waste generating project.
- Filing of the EIA for technical review and approval by MAATE.
- Environmental consultation with communities for inclusion in the EIA.

The EIA classifies the potential impacts as critical, severe, moderate or irrelevant. The key findings of the EIA are as follows:

- From a total of 100 potential impacts identified for the construction phase, 10% were classified as severe and 90% as moderate.
- From a total of 29 potential impacts identified for the operations phase, 3% were classified as severe and 97% as moderate.
- From a total of 13 potential impacts identified for the closure phase, none were classified as severe and 100% as moderate.
- The potential impacts classified as severe during the construction phase are mainly related to loss of vegetation cover, habitats fragmentation, soils compaction where infrastructure will be built and reduction of surface water infiltration due to changes to land cover from construction of infrastructure.
- The potential impacts classified as severe during the operations phase are mainly related to changes to water quantity and quality resulting from mine dewatering.

The technical pronouncement from MAATE favourable to the EIA and EMP of the Project was granted on April 19, 2022. Final approval of the EIA from a technical perspective was granted in April 2024.

20.3.2 Environmental Management Plan

The EMP for the Project was developed as part of the EIA. The EMP outlines actions required to prevent, mitigate, control, compensate, correct, and repair the potential adverse impacts resulting from the Project activities, and presents information regarding potential positive impacts of the Project activities. Additionally, the EMP describes the Project's Monitoring Plan, which was developed to track the effectiveness of the mitigation measures proposed in the EIA.

20.3.2.1 Environmental Management Sub-Plans

The following nine sub-plans are included in the EMP:

Impact Prevention and Mitigation Plan. This plan outlines specific measures identified in the EIA for the prevention of adverse impacts on air quality, ambient noise and vibration, soil quality, water resources, flora and fauna, archaeological resources, public services, handling, storage, and transport of chemical products, general maintenance, and occupational health and safety.

Contingency Plan. This plan, also called the Emergency Response Plan, describes the steps to follow and actions to implement in the event of an emergency to ensure a rapid, timely, and organised response.

Environmental Training and Education Plan. The objective of this plan is to develop environmental and socio-economic management procedures for the prevention and mitigation of potential impacts generated by the Project. Additionally, there is the expectation that the plan will create a culture of industrial health and safety inside and outside DPM for workers, contractors, temporary employees, visitors, and communities in the area of influence of the Project.

Waste Management Plan. This plan determines the management and final disposal of all waste (hazardous and non-hazardous solid and liquid) generated by the Project to avoid effects on the environment and human health during the construction, operational, and closure stages.

Community Relations Program. The main objective of this plan is to responsibly manage the potential impacts that the Project could generate in the area, and to maintain good relationships with the communities in the area of influence. The program includes, but is not limited to, communication, identification of opportunities for development, local labour recruitment, and environmental education.

Rehabilitation Plan for Affected Areas. The plan considers specific measures for progressive closure (closure of areas at the end of the construction stage and during the operational stage that will no longer be used), including rehabilitation and revegetation.

Mine Closure Plan. The closure and abandonment plan includes specific measures to identify the facilities and activities that must be closed at the end of the mine life. The Mine Closure Plan is discussed in Section 20.7.2.

Monitoring and Follow-up Plan. The objective of this plan is to monitor the compliance and effectiveness of the proposed measures to mitigate the impacts on the environment during the construction, operation, maintenance, and closure stages of the Project. This plan is described in more detail in Section 20.3.2.2.

Occupational Health and Safety Plan. In accordance with the provisions of Article 68 of the Mining Law of Ecuador, DPM has the obligation to preserve the life and mental and physical health of its workers. To meet this objective, the standards of safety and mining-industrial hygiene provided for in the relevant legal provisions must be applied, providing DPM's workers with health services and permanent care.

20.3.2.2 Environmental Monitoring Programs

The Monitoring and Follow-up Plan of the EMP includes the following environmental monitoring programs:

Air Quality and Air Emissions Monitoring Program. Measurements will be made at specific sites to monitor air quality. These results will be compared with the Ecuadorian and international regulations set to be adopted by the Loma Larga Project. Corrective actions will be determined for non-compliances identified during air quality monitoring.

For significant sources of emissions into the atmosphere, emission rates for the identified pollutants must be monitored to demonstrate compliance with the maximum permitted limits outlined in the Ecuadorian and international regulations. If the concentrations exceed the maximum permitted emission values, new methods or additional control equipment will be implemented.

Noise Monitoring Program. Industrial noise measurements will be recorded at specific sites in the construction and operation areas of the Loma Larga Project. Environmental noise measurements (day and night) will be recorded at locations identified in the socio-environmental baseline. The results will be compared with the Ecuadorian and international regulations set to be adopted by the Loma Larga Project. Corrective measures will be implemented in cases of non-compliance.

Vibration Monitoring Program. Vibration measurements (particle velocity) will be made at specific sites in the construction and operation areas of the Loma Larga Project, as well as in areas identified in the socio-environmental baseline. The results will be compared with the Ecuadorian and International regulations set to be adopted by the Loma Larga Project. Corrective measures will be established in cases of non-compliance.

Soil Monitoring Program. Monitoring of physicochemical parameters in soil will be carried out at sites impacted by chemical or fuel spills, as well as in the areas surrounding the vents. In addition, the agrological quality of the topsoil stored in the soil storage area will be managed and monitored. The results will be compared with the Ecuadorian and international regulations set to be adopted by the Loma Larga Project and corrective measures will be established in cases of non-compliance.

Surface Water Monitoring Program. Monitoring of potable water and surface water within the area of influence will be carried out at Loma Larga Project facilities. The results will be compared with the Ecuadorian and international regulations set to be adopted by the Loma Larga Project and corrective measures will be established in cases of non-compliance.

In addition, records of the water balance of the Project will be maintained, river flow measurements will be carried out in hydrometric stations placed in the bodies of water receiving effluents, and a water management program will be established in the micro-basins of the Irquis, Portete and Shurucay rivers.

Groundwater Monitoring Program. Monitoring of the physicochemical quality and quantity of groundwater, including springs, will be carried out in the implementation area of the Loma Larga Project. The results will be compared with the international regulations set to be adopted by the Loma Larga Project (there is no national regulation) and corrective measures will be established in cases of non-compliance.

Effluent Monitoring Program. Monitoring of the flow and physicochemical quality of effluents from industrial water treatment plants (blackwater and greywater), as well as monitoring of the physicochemical quality of surface water in the receiving bodies of the Loma Larga Project area will be carried out (emission and control). The results will be compared with the national and international regulations set to be adopted by the Loma Larga Project and corrective measures will be established in cases of non-compliance.

In addition, effluents from sedimentation pools during the construction phase will be monitored daily for temperature, pH, total dissolved solids (TDS), and conductivity to establish short term corrective measures.

Monitoring Program for Flora, Fauna, and Progressive Closure Rehabilitated Areas. The monitoring of plant diversity will be carried out in the plots used for permanent control and monitoring of terrestrial and aquatic fauna diversity (which will be correlated with the quality and quantity of water), and will include monitoring of bio-indicator species. Monitoring results will be compared to the baseline study results.

In addition, a pass/fail assessment will be carried out on the rehabilitation and revegetation initiatives during the progressive closure of the construction and operation stages, when necessary.

Tailings Monitoring Program. In order to monitor the stability of the Filtered Tailings Storage Facility (FTSF), settlement monuments will be placed in different parts of the dam and periodically monitored through topography measurements. The measurements will be recorded and analysed to detect any differential movement over time, trends, anomalies, etc. In addition, piezometers will be placed to monitor the presence of water and pore pressure inside the dam in order to verify that the design drainage conditions are met and detect any variation, trend over time, and potential anomalies. Corrective measures will be established in cases of non-compliance.

Waste Rock Monitoring Program. Monitoring of physical stability and water quality.

20.3.3 Environmental Geochemistry and Water Management

Various geochemical characterisation studies for the Project have been conducted in the past. The studies cover tailings, waste rock (material from the decline), ore, and materials from quarries. There are two main types of rock considered in the studies, namely, andesite and dacite. According to the characterisation studies, the ore, the tailings, and the andesite are classified as potentially acid generating (PAG) whereas the dacite as non-potentially acid generating (non-PAG).

The most recent geochemical testing was conducted on tailings backfill samples. According to the mine plan, a portion of the tailings production will be disposed on surface in the FTSF, and a portion will go to the underground mine as backfill.

In 2022 NewFields reviewed all characterisation work completed to date to summarise it and conduct a gap analysis. NewFields informed that, overall, they found the characterisation work adequate. Two gaps were identified by NewFields as follows:

- The characterisation available for the decline (i.e. rock intersected by the underground mine) reflects an old alignment of the decline.
- No characterisation of overburden material has been conducted.

NewFields made recommendations to DPM involving collection of additional samples, carrying out laboratory testing and conducting geochemical characterisation to address these gaps.

According to NewFields, numerical modelling and characterisation were conducted by Itasca before 2020. The numerical model captures the plan for the underground mine development and available field data previously collected. Additional geotechnical investigations planned in support of detailed engineering will provide additional data to refine the modelling work. Additional monitoring wells will be installed to collect groundwater data.

According to NewFields, water balance and water quality modelling has been conducted for the Project. The water quality modelling uses source terms defined from the geochemistry characterisation. Water quality predictions include the water treatment facility influent and the requirements of the receiving watercourse (also considering the background water quality). Mass

balance has been used to determine the treated effluent water quality that would meet the discharge criteria.

The engineering design for the Project considered that the water flowing into the underground mine has the potential to produce acid and leach metals. Likewise occurs with the tailings. As a result, the proposed water management strategy involves pumping the underground mine dewatering and the FTSF discharge to surface ponds, and then to a water treatment facility prior to releasing any contact water to the receiving environment.

The water management plan involves separation and management of contact water and non-contact water. Contact water will be managed with two ponds, one located north of the Project site (the Contact Water Pond), and one located south (the Mine Water Pond). Non-contact water will be managed with diversion channels lined with riprap to control erosion.

The Contact Water Pond would receive surface runoff from the process plant area and water discharged from the FTSF. The FTSF has been designed as a fully lined facility. The Contact Water Pond has been designed with double liner consisting of HDPE geomembrane.

The material to be deposited in the ore stockpile and the waste rock stockpile is anticipated to be PAG. The design of both stockpiles includes placement of a liner at the bottom (HDPE geomembrane). Surface runoff and seepage from these facilities would be conveyed to the Mine Water Pond together with the underground mine dewatering. The Mine Water Pond has been designed with double liner consisting of HDPE geomembrane.

Water collected in the Mine Water Pond would be pumped to the Contact Water Pond, Water collected in the Contact Water Pond would be pumped to either the process plant for ore processing activities, or to the water treatment facility prior to discharge to the environment.

The design concept for the liners includes underdrains underneath the liner to collect underground water. This water is non-contact water that could be diverted for direct discharge to the environment. A sampling point has been included in the water management plan to monitor the water quality of this system. If the water quality does not meet applicable standards for direct discharge to the environment, it would be intercepted and pumped to the Contact Water Pond.

Water quality monitoring during operations will confirm the adequacy of the proposed water management plan and engineering design to protect the environment.

20.4 Environmental Issues

No known environmental issues were identified by the QP from the information on environmental studies provided by DPM for review. In the QP's opinion, there are no environmental issues that could materially impact the ability to extract the Mineral Resources and Mineral Reserves based on the review of the available documentation.

20.5 Environmental Permitting

20.5.1 Legal Framework

The legal framework presented in this section was provided by DPM's legal counsel. Multiple aspects, including the mining regime, permitting environmental protection, occupational health, social dynamics, conservation, investments, taxation, and other factors related to the Loma Larga Project activities are subject to an array of Ecuadorian and regulatory standards and international best management practices. DPM is committed to adhere and comply with the Ecuadorian legislation and international standards applicable for the Loma Larga Project, such as the Constitution, the Mining Law and its regulations, the Organic Environmental Code, the Environmental Regulation for Mining Activities, the Water Law, as well as other regulations related to the mining sector.

The main mining related provisions established in the Constitution, applicable to the Loma Larga Project, are the following:

- Non-renewable natural resources belong to the State; therefore, the State will participate in the benefits from the exploitation of such resources, in an amount that will not be less than those of the company that exploits them. Due to their economic, social, political, and environmental significance and influence, non-renewable natural resources are considered as strategic sectors, whereby the State has the right to regulate, control, administer, monitor, and manage them and the State has the obligation to create government-owned companies for the management of strategic sectors. Mining activities are explicitly prohibited to be carried out in protected areas, intangible zones, and urban centres.
- The 2008 Constitution incorporates the principle that nature is a subject of rights and has the right to a full respect of its existence and the maintenance and regeneration of its vital cycles, structure, functions, and evolutionary processes. Likewise, nature has the right to restoration and any person, community, people, or nationality may demand from the public authority the fulfilment of the rights of nature. As mentioned above, mining activities are prohibited from being carried out in protected areas, aiming to ensure the conservation of biodiversity and the maintenance of ecological functions in such areas.

The Constitution stipulates that environmental consultations must be carried out with affected communities prior to the issuance of state decisions or authorisations that may affect the environment, regardless of whether these communities are Indigenous or not. This environmental consultation is carried out through dialogues that seek to reach an agreement, and if the consultation process results in a majority opposition from the respective community, the decision to implement or not the project will be adopted by a duly motivated resolution by the State. Subsequently, the environmental authority of Ecuador will proceed to issue the administrative resolution granting the Environmental License.

The Mining Law reiterates that non-renewable natural resources including underground resources are the property of the Ecuadorian State and are independent from the property rights over the surface land that covers the mines and deposits. Mining rights are understood as those rights arising from mining concession titles, mining exploitation contracts, licenses, and permits. The mining title, which confers a personal right, grants its holder the exclusive right to prospect, explore, exploit, process, smelt, refine, commercialise, and dispose of all mineral substances that may exist and be obtained in the area of such concession, becoming beneficiary of the economic revenues obtained from such processes.

Mining activity has eight (8) phases: prospecting, exploration, exploitation, processing, smelting, refining, sales, and mine closure. Mining concessions have a term of up to 25 years and may be renewed for equal periods. The concession is divided into two stages: exploration and exploitation. The exploration stage is further divided into the initial exploration period (for a term of four (4) years), advanced exploration (for a term of four (4) years) and the economic evaluation period of the deposit (a term of two (2) years renewable for two (2) additional years). Within the period of economic evaluation of the deposit, the mining concessionaire may request to move to the exploitation stage and execute the Mining Exploitation Contract with the Ministry of Energy and Mines (*Ministerio de Energía y Minas* – MEM for its acronym in Spanish). The execution of a Mining Exploitation Contract is always required for large-scale mining activities.

Prior to initiating mining activities, all titleholders must obtain an environmental license, a certificate of no impact on water sources, and the order of priority of access to water. All titleholders must also make an affidavit swearing that there will be no damage to infrastructure or cultural vestiges, among other damages. One (1) year after obtaining the environmental license, the mining companies must carry out an environmental compliance audit that allows the State to monitor, supervise, and verify compliance with the environmental management plans and applicable environmental regulations. Subsequently, environmental compliance audits will be carried out every three (3) years. Regarding water, all mining titleholders who obtain an authorisation and use water for their mining works and activities, must return the water to the original water source free of contamination, or in compliance with the permitted limits. Reuse of water for Project activities is a permanent obligation.

It is noted that during the operations phase mining companies must submit quarterly environmental monitoring and follow-up reports for the exploitation phase, and annual environmental management reports that will contain information supporting compliance with the EMP, as well as the applicable legislation.

The General Regulations to the Mining Law provide more detailed requirements to carry out several procedures provided in the Mining Law, such as the procedure to obtain mining rights through auctions and biddings, the requirements for the assignment or transfer and the assignment in guarantee of mining rights, the requirements for waiving or reducing mining rights, among others.

The Environmental Regulations for Mining Activities (*Reglamento Ambiental para Actividades Mineras* – RAAM for its acronym in Spanish) include environmental standards and technical guidelines for the mining national sector, including the registration and environmental licensing processes for mining activities. Large-scale mining projects in the initial exploration phase require an environmental registration, while those in the advanced exploration, exploitation and subsequent phases require an environmental license. It is mandatory that in addition to the environmental registry or license, they must also hold a certificate of intersection of the mining rights with the National System of Protected Areas, Protected Forests and Vegetation, State Forest Heritage, or other conservation areas. It is prohibited to carry out mining activities within areas of the National System of Protected Areas, however, if the mining concession intersects with Protected Forests and/or Forest Heritage, the concessionaire must obtain the certificate of environmental viability qualified with the report of feasibility of the mining right.

The mining sector in Ecuador is highly regulated, in order to guarantee sustainable development, mitigate and control the environmental activities, and to maximise the positive impacts caused by the development of the mining activity.

20.5.2 Environmental Approval

The main government institutions that regulate and approve mining projects in Ecuador are MAATE and MEM, including the Agency for Regulation and Control (*Agencia de Regulación y Control* Minero– ARCOM for its acronym in Spanish). Each of these institutions has representation in every Province for administrative and permitting processes. However, given that the Project belongs to the mining sector and it is a strategic Project for Ecuador, the processing of the main permits is conducted in Quito.

The Mining Exploitation Agreement between DPM and the Government of Ecuador will allow the Company to develop the mine. This contract will set forth the terms, conditions, and timeline for the development and operation of the mine. In addition, the contract will define the Company's obligations regarding environmental management, bonds, community participation, royalties, and closing. INV MINERALES ECUADOR S.A. initiated negotiations with the Ecuadorian Government on the terms and form of the Mining Exploitation Contract, and a preliminary agreement was reached

in May 2017. DPM is preparing to restart negotiations with the Ecuadorian Government for the Project, and it is anticipated that the Exploitation Contract will be executed after obtaining all relevant permits and financing, as well as other key milestones of the Project.

MAATE is the national environmental authority that governs, coordinates, and regulates the National Decentralised Environmental Management System (*Sistema Nacional Descentralizado de Gestión Ambiental*). This ministry is responsible for issuing the environmental license for the construction, operation, and closure of the Project for the exploitation and benefit phases, upon payment of administrative fees, and the issuance of the policy of full compliance with the Environmental Management Plan by DPM. MAATE issues environmental licenses through the office of the National Director of Prevention and Environmental Pollution. The technical pronouncement from MAATE favourable to the EIA and EMP of the Project under the Large-scale Mining regime for the phases of exploitation and beneficiation was granted on April 19, 2022. Final approval of the EIA from a technical perspective was granted in April 2024.

Due to concerns related to contamination that could be caused by the Project, a legal action was filed in the court of Azuay by members of the communities in 2022 resulting in a court ruling on August 28, 2023 requesting four reparative measures to complement the EIA. The ruling indicated that approval of the Environmental License is dependent upon completion of those activities, to be led and conducted by government agencies. The four reparative measures from the ruling, as well as their status, are summarised as follows:

- Biotic study in the Quimsacocha national recreation area to evaluate the potential impacts of the Project in this ecologically sensitive area included in the national system of protected areas, which is the highest category of protection that Ecuador currently has. The study was finalised in 2024, and the final report completed by MAATE was submitted to the court of Azuay in October 2024.
- Water study to evaluate the potential impacts of the Project given that the Project area corresponds to a location characterised as moorland (*páramo*) ecosystem. The moorland ecosystem is considered a water recharge zone and concerns exist in relation to the possible contamination of the water supply to the city of Cuenca. The study was finalised in 2024, and the final report completed by MAATE was submitted to the court of Azuay in October 2024.
- Environmental Consultation, which started in February 2024 and was completed and approved in November 2024. The final report completed by MAATE was submitted to the court of Azuay in December 2024.
- Indigenous Consultation in the community of Escaleras, which began in July 2024 and was completed and approved in May 2025. The final report completed by MEM was submitted to the court of Azuay in May 2025.

The Environmental License resolution for exploitation activities for the Project was issued by MAATE on June 23, 2025. As indicated under the section "Post-Effective Date Events" at the beginning of this Report, the Environmental License was revoked by the Government of Ecuador on October 3, 2025. The license was revoked by the Ministry of Environment and Energy (*Ministerio de Ambiente y Energía* – MAE for its acronym in Spanish). The granting of the Environmental License in Q3 2025 (or before) was a key milestone in the Project Schedule to meet the target of initiating the Project's construction in 2026.

20.5.3 Current Future Permits, Approvals and Authorisations

DPM maintains an up-to-date record of the legal permits obtained to date and future permits to be obtained according to the applicable legislation and regulatory requirements. The environmental permits already approved and future environmental permits to be obtained are listed in Table 20.1.

Table 20.1 – List of Approved and Pending Environmental Permits

Authority	Obligation/License	Status	Date of Issue (MM/DD/YEAR)	Expiration Date (MM/DD/YEAR)
Secretaría Nacional del Agua (SENAGUA)	Favourable Administrative Act of non-impact on water bodies and compliance with the order of priority of the right to water	Granted	1/18/2011	No expiration date
Secretaría Nacional del Agua (SENAGUA)	Authorisation for industrial use of surface water for productive activities	Granted	10/11/2016	10/10/2026
Ministerio de Energía y Minas (MEM)	Technical feasibility certificate for the construction of the FTSF and technical approval for the construction of new waste rock dumps	Granted	6/25/2022	No expiration date
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Planta Central Quito*	Environmental License for 69 KV transmission line	Granted	4/23/2024	Valid for the duration of the Project
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE)	Environmental License	Pending	-	-

Authority	Obligation/License	Status	Date of Issue (MM/DD/YEAR)	Expiration Date (MM/DD/YEAR)
Planta Central Quito				
Instituto Nacional de Patrimonio Cultural (INPC)	Affidavit of non-impact on archaeological remains or cultural heritage submitted by the titleholder	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Planta Central Quito*	Environmental License - Complementary Environmental Impact Assessment	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Planta Central Quito*	Environmental License - Environmental Impact Assessment for transport of concentrate	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Dirección Zonal 6 - Cuenca*	Environmental License for road construction	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Planta Central Quito*	Effluent discharge authorisation (included in the Environmental License)	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Dirección Zonal 7 - Cuenca*	Authorisation for industrial use of mine dewatering for productive activities	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE)	Authorisation for domestic use of water (Pinos camp)	Pending	-	-

Authority	Obligation/License	Status	Date of Issue (MM/DD/YEAR)	Expiration Date (MM/DD/YEAR)
Dirección Zonal 6 - Machala				
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Planta Central Quito*	Registry for generation of hazardous waste (*Registro de Generador de Desechos* – RGDP for its acronym in Spanish)	Pending	-	-
Ministerio del Ambiente, Agua y Transición Ecológica (MAATE) *Oficina Técnica Cuenca*	Hazardous Waste Reduction Program	Pending	-	-

20.5.4 Permitting Schedule

DPM developed an environmental permitting schedule listing the main permitting processes to be completed ahead of the construction stage and the anticipated timeline. Notwithstanding the revocation of the Environmental License, the main components of the environmental permitting milestones schedule are as follows:

- Restoring the Environmental License from MAATE.
- Signing the Mining Exploitation Agreement MEM.
- Completing the Environmental Registration of the expansion of the public road from San Gerardo to initiate construction of road improvements.
- Completing the power supply feasibility study and permitting.
- Completing the permitting of the FTSF.

20.6 Social or Community Requirements

20.6.1 Social License to Operate

The Social License to Operate a mining project encompasses “the perceptions of the community and other stakeholders about the legitimacy and credibility of the mine or project, and the presence or absence of true trust” (Thomson and Boutilier, 2011). This concept, which was initially used as a metaphor during the 1990s, became an increasingly strong planning and management tool for the mining and energy sectors. The aim is maintaining continuous level of approval by communities and other stakeholders in relation to the development of mining projects. According to Thomson and

Boutilier (2011) the Social License to Operate can be understood as resulting from four factors: economic legitimacy, socio-political legitimacy, interactional trust, and institutionalised trust.

An important component to obtain the Social License to Operate is establishing a social baseline for the areas surrounding the Project, which allows DPM to understand the socio-economic and socio-political context of the Project's area of influence in order to develop the community relations plan. To this end, DPM has established five lines of social involvement: economic production activities, education, health, infrastructure, and culture and recreation.

20.6.2 Social Issues

According to data obtained in the 2024 social baseline survey, the level of poverty due to unmet basic needs in the communities within the project's area of influence averages 60%. In this socioeconomic context, the population has expectations regarding opportunities associated to local development in areas such as direct employment, supply, and, in general, improvement in quality of life during the construction and operation phases of the project. In addition, it should be noted that migration indicators in the area of influence are, on average, above 33%, which affects community dynamics.

The expansion of the public access road from San Gerardo requires land acquisition by DPM, which may involve the need for involuntary resettlement of people living along the road. Although it is anticipated that potential resettlement would involve few people, it must be properly managed by DPM with close support from the social management team through a process that is transparent, equitable, and effective.

20.6.3 Community Engagement and Agreements

Communication is a key component to inform local, regional, and national interested parties about the status of the Project, address the local inhabitants' concerns regarding the potential environmental impacts of the Project, and share the progress of the social investment plan undertaken in local communities.

DPM has adopted several communication mechanisms to inform the public about the Project and community development projects, including the following:

- Public assemblies.
- Guided tours of the concession area.
- Itinerant information spaces (information tents, participation at conferences and exhibitions, etc.).
- Door-to-door information campaigns.
- Communication and information campaigns (traditional and digital media).

- Spaces for dialogue in Cuenca, Azuay, and nationally through public events.
- Printed materials for internal and external communication.
- Community development project planning sessions.
- Training sessions for community members.

Four (4) community information offices have been established by DPM in Victoria del Portete, San Gerardo, Chumblín, and Girón. Project information is available at these offices, and the offices provide space to hold meetings and other activities with local authorities and members of the communities.

DPM has implemented a consultation and grievance mechanism for the communities to provide feedback or file requests, questions, concerns, and/or complaints regarding the Project. According to a teleconference held with DPM on March 5, 2025, there is an accessible grievance mechanism to handle the requests, questions, concerns, and complaints in a systematic way and ensure that responses and resolutions take place and can be tracked by the social management team.

20.6.3.1 Community Relations Activities

DPM has indicated that it maintains a good relationship with the communities located within the social area of influence. The Company has been implementing its social action and development plan by supporting activities associated with economic production activities, education, health, infrastructure, culture, and recreation.

The activities and projects to be implemented have been identified by DPM based on the socio-economic baseline characterisation, the ongoing communication between the DPM social team and the communities, and information gathered from the Decentralised Autonomous Governments (*Gobiernos Autónomos Descentralizados* – GAD for its acronym in Spanish), which are small territorial political units. The GADs carry out development and territorial planning for their jurisdictions and organise community meetings attended by local leaders and authorities to plan and coordinate projects focused on improving the quality of life of residents. The criteria used by DPM to identify and implement social projects and activities within the five lines of social involvement (core areas) include alignment with the community's objectives and vision, generating a positive impact on the community, and sustainability.

DPM is working on activities to promote development of local suppliers, grow relationships with current and potential local suppliers, generate a supplier database, and promote local ventures. The following initiatives have been conducted in support of economic production activities:

- Strengthening the businesses of seven productive groups from the parishes of San Gerardo, Chumblin, and Victoria del Portete in alliance with actors linked to industry and commerce.

- Trainings and certifications for local suppliers in catering services, cleaning services, transportation services, manufacture of uniforms and work clothing, among others related to the future operations. Updating the database of local suppliers in order to prioritise the acquisition of goods and services from Cuenca, Girón, and San Fernando over other cities located further away from the Project area.
- Coordinating work with the Ministry of Agriculture and Livestock (*Ministerio de Agricultura y Ganadería*) in order to promote traditional productive activities typical of the area of influence of the Project.

Education support has involved the following initiatives:

- The “I am a DPM Scholarship holder” program aims to provide scholarships to local low-income residents so they can pursue their basic education (elementary school), high school, technical degrees, and certifications in arts and crafts.
- The “DPM Connects You” program consists of providing free internet service and equipment to educational centres and community public spaces in the area of influence of the Project, in agreement with the Ministry of Telecommunications (*Ministerio de Telecomunicaciones* – MINTEL for its acronym in Spanish).
- Training workshops to strengthen local talent and capabilities such as cooking, painting, baking, and hairdressing, among others.

Health support has involved the following initiatives:

- Medical brigades covering general medicine, psychology, and dentistry to provide care in mobile units in alliance with public institutions and foundations. Home medical care and vaccination campaigns, with a focus on supporting the "Zero Malnutrition" program.
- Gerontological care through home care, implementation of physiotherapy centres, and the creation of recreational spaces to share with seniors.
- Preventive health campaigns in educational centres and community spaces in alliance with public and private organisations.

Support on infrastructure has been focused on maintenance of community spaces such as schools, community houses, parks, sports courts, health centres and public roads, always in coordination with local governments and institutions.

The following initiatives have been conducted in support of culture and recreation:

- The “SPORT TEAM” program supports young athletes from the Azuay province.
- Participation in and support to activities organised by the communities such as religious festivals, sporting events, and other cultural activities.

20.6.3.2 Agreements

DPM has developed the following agreements:

- Inter-institutional cooperation agreements with the Decentralised Autonomous Governments of Girón, San Gerardo, and Chumblin.
- Inter-institutional cooperation agreements with three communities in the area of direct influence, namely Cristal Aguarongos, Comuna Sombreredas, and El Durazno.
- An agreement with the Ministry of Telecommunications.

20.6.4 INDIGENOUS PEOPLES

According to information provided by DPM, the Escaleras community is recognised as Indigenous by the Ecuadorian government, however, it is not in the direct area of influence of the Project.

20.7 Mine Closure Requirements

20.7.1 REGULATORY REQUIREMENTS

Article 27 of the Ecuadorian Mining Law (Law 45) and the Environmental Regulation of Mining Activities consider the closure of a mine as a regulated phase of mining activities. The closure phase consists of the termination of mining activities and the subsequent dismantling of the facilities.

According to Article No. 85 of the Ecuadorian Mining Law, owners of mining operations and beneficiation, smelting, and refinery plants must include planning for the shut-down of their activities in their EIA and provide the relevant guarantee. The closure plan must cover the mining, beneficiation, smelting, and refining activities included in the EMP. The planning must begin at the pre-feasibility stage of the project, and continue throughout the operational life of the mine, until the final closure and shutdown.

According to the *Ecuador Mining Act,* the shut-down plan for mining operations shall be reviewed and updated periodically in the Environmental and Budgeting Annual Programs and in the Environmental Compliance Audits, and should include information on sums invested or an estimate of the closure cost. In addition, no later than two years before the expected shut-down of the mining, beneficiation, smelting, or refinery project, the mining license holder must present the final Mine Closure Plan to the environmental authority of Ecuador for its approval. This plan must include activities for remediation, the method for verifying compliance with the proposed plan, an assessment of the social impacts and compensation plan, and updated guarantees, as indicated in the applicable environmental regulations. It must also include a plan for incorporating new forms of sustainable development.

To ensure that a mine is properly closed, most jurisdictions in Ecuador require a closure plan and financial assurance. Financial assurance is a financial guarantee that ensures the rehabilitation work

is performed. In Ecuador the mining license holder must provide a bank guarantee or take out an insurance policy, which must remain in force until the total closure of mining operations, to guarantee compliance with the environmental management plan.

20.7.2 Mine Closure Plan

A conceptual Mine Closure Plan was developed by DPM as part of the EIA. The specific objectives of the mine closure and reclamation activities include:

- Leave a site with physically and chemically stable landforms where ecosystem development can proceed, such that the site meets discharge surface water and groundwater quality standards of Ecuador.
- Incorporate stakeholder land closure concepts aimed to provide a desired next land use that is compliant with regulations and sustains socio-economic benefits.
- Implement the mine closure and reclamation program as cost effectively as possible with commercially available skills, equipment, and materials, taking into consideration prevailing local factors such as climate, seasonality, geography, demography, infrastructure, security, governance, capacity, and operational reliability.
- Include contingencies for temporary suspension of activities and permanent early closure.
- Achieve discharge surface water and groundwater quality, in alignment with the standards of Ecuador.

The conceptual Mine Closure Plan is applicable for the Life-of-Mine (LOM) extending over all Project areas, including the mine access road and power / waterline corridors. The closure of the offsite access road from San Gerardo to the mine site from the south is not included in the Mine Closure Plan as this is an existing public road to the Project gate.

Two (2) years prior to the planned completion of the Project, DPM must submit the detailed Mine Closure Plan to the environmental authority for approval. A post-closure management team will be established to facilitate mine closure activities a minimum of two (2) years prior to closure. Throughout the closure phase, an external qualified consultant will be contracted to review, approve, and sign off that the work has been completed.

The successful reclamation program will partially rely on saving topsoil as a future growth medium on disturbed sites, as well as the continued implementation of native plant re-establishment that has been developed on the site at the existing plant nursery. As part of the Project Execution Plan, certain areas will be designated for a topsoil stockpile and another area designated for the material removed from the sublayer of organic material. This material will be used for backfill operations during mine closure and reclamation.

The Mining Exploitation Contract will have specific closure considerations including the payment of all environmental liabilities corresponding to a period equivalent to that of the mining concession. An insurance policy or bank guarantee, unconditional, irrevocable, and of immediate collection in favor of the Ministry of the Environment will be required during closure.

The concepts for closure of the Project facilities and the proposed post-closure monitoring program developed by DPM during the preparation of the EIA are described below.

20.7.2.1 Underground Activities

For the underground mine, all stopes and all areas of development where there could be long term potential for ground failures resulting in a failure to surface, will be backfilled. The stope backfill will be either with paste fill, CRF, or waste as follows:

- The first stopes of the LOM will be backfilled with CRF until the paste plant is 100% operational.
- Some of the last stope of the secondary panels, when possible (access on top and bottom), will be backfilled with waste, if approved by the geotechnical consultant advising DPM.
- All other stopes will be backfilled mostly with paste fill. When possible, the first portion of the stope plugs (the first five metres of a seven-metre plug) will be filled with waste rock from the bottom access before continuing the backfilling with paste fill. The last metres of the stopes backfilled with paste fill will be filled with waste rock so that the equipment does not skid on the paste floor.
- Any excess waste material on surface (that is chemically stable) can likely be returned underground to fill the drives, at the discretion of the geotechnical consultant.

In the portal area, all structures and electrical equipment will be removed, including the powerline, from the main on-site substation. The surface areas will be stripped, covered with soil, and revegetated.

The portal area water and sediment ponds will be decommissioned, material and liners removed, and the area will then be backfilled and revegetated.

The ramp will be closed to access. A steel fence will be installed 6 m inside the portal with stainless steel 100 mm x 100 mm welded wire fabric to limit entry. The complete portal area will be backfilled with clean material to conform to the surrounding landform, covered with topsoil, and revegetated.

Ventilation raise areas and site access are to be decommissioned and reclaimed. All ventilation and escape raise equipment will be removed from the top of the raises and the fan foundations will be left in place.

The backfill paste plant area buildings, process facilities, utilities, underground piping, duct banks, and all other equipment will be removed from site, to be sold or otherwise disposed of. The concrete foundations will then be broken, left in place, and covered by overburden and 0.3 m soil to provide for successful revegetation growth media. Site drainage will be re-established to natural patterns.

20.7.2.2 Surface Infrastructure

At the end of the LOM, any material left on the mine ore storage area will be processed. The stockpile base seepage collection liner will be removed and disposed of. The site will be aerated, regraded to conform to the pre-operation or closure landform, and covered with overburden (to complete landscaping) and soil (for revegetation). Any hazardous or unsuitable material will be delivered to the FTSF area for proper disposal. Contaminated material, not permitted to be placed in the FTSF will be placed in lined 20-foot containers and delivered to a hazardous waste storage area approved and licensed by the authorities.

If the waste rock stockpile remains in place at closure, an adequate cover to handle the potential for PAG and metal leaching of deposited waste rock must be engineered prior to closure for long-term protection of water quality during post-closure.

The electrical substation and specific powerlines will be left in place as power is required to run the water treatment plant. DPM will hold discussions with CENTROSUR prior to closure to determine the ownership structure of the transmission lines in the future. Any on-site lines will be decommissioned, conductors removed from site, sold, or scrapped, and the power poles removed.

All underground piping not required for the water treatment facility will be removed along with all concrete duct banks. The water treatment facility could be transferred to the local landowners or community. If the negotiations do not come to fruition, the facility and all piping will be removed from the site.

The site perimeter access roads and the roads to the ventilation raises will be decommissioned, landscaped, scarified, covered with soil, and revegetated with grasses and plants. Ditches will be filled, culverts removed, and natural drainage patterns restored. All roads and tracks, except for pre-existing publicly owned roads, will be reclaimed in accordance with the process described above. Roads and access required for the water treatment facility and subsequent monitoring will remain until authorised by the authorities.

20.7.2.3 Filtered Tailings Storage Facility

The FTSF area will require ongoing maintenance as the water from within the process facility drains. The Operation, Maintenance and Surveillance manual for the FTSF will specify the inspection, monitoring, and reporting requirements of the facility until permanent closure is achieved.

The FTSF is estimated to have 4.3 Mt of dry tailings at closure. After a design review by the geotechnical engineer of the facility final conditions, regrading may be necessary, and the FTSF will then be covered with a geomembrane liner, overburden and topsoil to form a growth medium for the *páramo* vegetation. An adequate cover to handle the potential for PAG and metal leaching of deposited tailings must be engineered prior to closure for long-term protection of water quality during post-closure.

After reclamation is complete, the tailings will drain over a period of time. Monitoring of the underdrain system will continue, and water will be conveyed to the Contact Water Pond if necessary. The water treatment facility will remain operational to treat effluent to meet receiving environment water quality standards for a period of up to five (5) years. The FTSF will be permanently closed once surface and seepage water from the facility is demonstrated to meet discharge criteria using passive treatment.

The Contact Water Pond, when no longer required, will be reclaimed by removing the liner, demolition of all concrete, and removal of piping. The area will be covered with growth media and reseeded. Post closure will require maintenance of the closure cover until a vegetated surface can be established. The closure surface will be monitored, inspected, and repaired, if needed, to address water and wind erosion.

20.7.2.4 Off-Site Infrastructure

All roads and tracks, except for pre-existing publicly owned roads, will be reclaimed in accordance with the process described above. Roads and access required for the water treatment facility and subsequent monitoring will remain.

20.7.2.5 Water Treatment Facility

Post-closure water treatment has not yet been modelled and for the purposes of this Feasibility Study, has been assumed for a period of five (5) years with an additional five (5) years of monitoring.

Decommissioning of the water treatment facility will involve disconnecting electrical infrastructure, removal of all chemicals and equipment from buildings and on site, dismantling buildings, covering broken up cement foundations with overburden, soil, and revegetation.

20.7.2.6 Off-Site Structures

The structures in Pinos camp, core shack, dining area, and laundry will be demolished during the initial closure stages. The areas will be graded and revegetated. DPM owns several small parcels of land in the area on which a new core shack will be constructed, and the cores stored until they are no longer required.

The existing plant nursery will be maintained and operated for additional plant and seed stock into the post-closure period as some revegetation and reforestation is anticipated. The nursery will be closed, transferred to the local community, or transferred to adjacent landowners.

20.7.2.7 Post-Closure Monitoring

An adequate period of monitoring will be implemented to demonstrate compliance with proposed criteria and conditions that ensure the mine site is safe, stable, and has reached the planned closure objectives. Such conditions must be demonstrated for a period of five (5) years after the cessation of mining and closure of the mine or until the time that the Ministry of the Environment indicates. A progress and effectiveness report of the environmental measures implemented for the closure of the mine must be submitted to the environmental authority on a bi-annual basis for its approval.

Once the obligations of the closure and monitoring activities have been fulfilled after the operations have been completed, DPM will present a closure environmental audit, which will verify compliance with closure activities and will allow the nonrenewal of the environmental license. All environmental records will be presented in an environmental closure report. Communication of the results of the environmental audit will be defined in the respective Terms of Reference of the Audit and will be subject to the approval of the environmental authority. In the event that closure cannot be fulfilled for all of the aspects of the Project, such as water quality compliance with the criteria selected, DPM will present a closure plan that includes long-term management of that particular aspect until compliance can be achieved.

The closure environmental audit approval will determine the rehabilitation of the mining area and the requirement for any financial guarantee.

If the Project ceases operations due to expiration of the mining rights, the environmental authority will perform an inspection in order to determine the Project status. For operations that have an approved EMP, the company must apply the closure and abandonment plan specified therein. If the environmental authority determines that an area has not been intervened at all, a closure plan will not be applied, and only the report issued by MAATE and the inactivity certificate from MEM will be the enabling documents.

21 CAPITAL AND OPERATING COSTS

The capital cost estimate (Capex) and operating cost estimate (Opex) outlined in this Section are based on the detailed information presented in this Report.

The Capex and Opex estimates were compiled by DRA. Specifically, the following parties participated in the development of the Capex:

- DRA developed the Capex for the process plant and plant infrastructure for the Project scope as described in this Report.
- GMD prepared the underground mining costs.
- NewFields prepared construction quantities to support the Capex for the FTSF; DRA applied rates to complete the FTSF Capex.
- Paterson and Cooke prepared the Capex for the paste backfill system; DRA estimated bulk earthworks, concrete, and structural steel for the area.
- DPM provided Owner's general and administration costs, taxes and duties, and closure costs.

All costs are expressed in United States Dollars (US$) and are based on Q2 2025 pricing (or Q3 2023 pricing with escalation added where noted).

The Capex is deemed to have an accuracy of ±15% and was prepared in accordance with the AACE Class 3 estimating guidelines.

21.1 Capital Cost Estimate

Capital Cost Summary

Table 21.1 presents a summary of the initial and sustaining Capex over the LOM. All costs are segregated by Project WBS, and Owner's costs, contingencies and risk amounts are included.

The Capex was developed based on an EPCM project implementation model. Major equipment was specified, and priced quotations obtained from reputable equipment manufacturers. Construction contract packages were prepared, issued, and priced in-country by capable construction companies.

Table 21.1 – Capital Cost Summary (US$ '000)

Description	Initial Capex	Sustaining Capex	Total
Direct			
1000-Mine General	9,449	-	9,449
2000-Underground Mine	110,692	41,544	152,236
3000-Ore Handling	15,663	-	15,663
4000-Processing Plant	94,252	1,300	95,552
5000-Filtered Tailings / Water Treatment Facilities	61,224	18,977	80,201
6000-On-Site Infrastructure, Site-Wide General	48,945	-	48,945
7000-Off-Site Infrastructure	63,133	4,558	67,691
Subtotal	*403,358*	*66,379*	*469,737*
Indirect			
8000-Operational Readiness and Owner's Costs	66,383	-	66,383
9000-Indirect Costs	75,501	2,014	77,515
8700- Closure and Rehabilitation[2]	-	25,211	25,211
Subtotal	*141,884*	*27,225*	*169,109*
Contingency			
9900-Project Contingency	47,678	-	47,678
9930-Contingency - Sustaining Capital	-	6,638	6,638
9940-Contingency - Closure and Rehabilitation	-	2,521	2,521
Subtotal	*47,676*	*9,159*	*56,837*
Total Major Area Capex	**592,920**	**102,763**	**695,683**

Notes:
1. Initial capital and sustaining capital estimates include import duties.
2. Closure costs do not include non-refundable VAT of $4.2 million.
3. Numbers may not sum precisely due to rounding.

21.1.1 SUSTAINING CAPEX

For the preparation of the sustaining Capex, work which will be completed in future years without impacting the production, was identified and placed on a timeline. The work was then estimated using present day factors to arrive at an estimated cost. The estimate was prepared in as much detail as possible, to attain the accuracy ranges required for the Project. The sustaining Capex was then incorporated on a yearly basis in the Capex cash flow model, which was provided for economic modelling. Sustaining Capex components are listed in summarised in Table 21.1.

21.1.2 Closure and Rehabilitation Costs

Based on site layouts, a provision of $25.2 M was estimated for the closure and rehabilitation of the mine site. Requirements were established and cost estimation was based on material take-off and unit rates from recent database.

No provision is required for the dismantling and disposal of the industrial facilities as it is assumed that the costs will be compensated by the salvage value.

21.1.3 Base Date

The base date for the Capex is Q2 2025.

21.1.4 Base Currency

The estimate is presented in United States Dollars (US$). Prices obtained in other currencies are converted to US$ using the applicable exchange rates. Equipment and services quoted in other currencies are converted using exchange rates specified in Table 21.2. and are used and fixed in the estimate data sheet. All costs quoted in these currencies are linked to this exchange rate to enable adjustment at a later stage, if required. Ecuador utilises the US$ as its currency.

Table 21.2 – Foreign Exchange Rates

Currency	Name	Currency Equivalent to $1.00 USD	US Dollar Equivalent to 1.00 Currency
USD, US$	United States Dollar	1.000	1.000
CAD	Canadian Dollar	1.3514	0.7402
AUD	Australian Dollar	1.316	0.7599
EUR	European Euro	0.935	1.0695
GBP	British Pound	0.800	1.2500

21.1.5 Estimate Methodology

The approach taken in estimating the Project Capex was to define Project areas in accordance with the WBS and quantify or measure each cost element from engineering drawings, PFDs, mechanical equipment list, motor lists, cable schedules, and instrument lists.

Budget quotations from vendors were obtained for all major items of mechanical and electrical equipment whereas minor items of mechanical or electrical equipment, in general, were either single sourced, or taken from a DRA database.

The mine and plant Capex was developed assuming continuous engineering, procurement, and construction with no interruption of the execution programme after funding approval is obtained.

The estimate is based on an EPCM Project execution strategy whereby execution will be managed by an EPCM Contractor working in conjunction with the Owner's Team who provides overall direction and oversight. The EPCM Contractor would place contracts for and on behalf of the Owner. Contracts for major construction work packages would be tendered to multiple regional pre-selected construction contractors, covering general disciplines of earthworks, concrete, structural steel, platework, piping, mechanical installation, electrical and instrumentation installation. Major mechanical and electrical equipment will be supplied by equipment suppliers and free issued to the appointed contractors for installation or installed by the OEM. Bulk materials were supplied and installed by contractors.

The total Project Capex and duration are estimated on a single shift basis and therefore no secondary shifts, nor overtime work is allowed for in the Capex.

The productivity rates for the project are provided by local subcontractors or DRA experience where not provided.

21.1.6 Pre-Construction Decision Works

In order to fast track construction of the Project before formal approvals, some of the scope required for the Project (and described in the 2021 NI 43-101 Technical Report) will be built before the construction decision date. These "pre-construction decision" works would be completed at risk, including, but not limited to:

- 60% of engineering work.
- Land acquisition and permitting costs.
- Owner related costs during pre-construction decision period.
- Construction of access road (18 km).

As these costs are spent before the "construction decision" date, they are not included in the Project Capex.

21.1.7 UNDERGROUND MINE

21.1.7.1 Mine Capex

The mine Capex includes the facilities described in this Report and is summarised as follows:

- **Mining Surface Infrastructure**: ventilation, paste backfill plant, fuel storage and distribution, surface ore and waste storage.
- **Underground Mining**: mobile equipment, mine development, mine infrastructure, underground services and stationary equipment.

The underground mine Capex is summarised in Table 21.1.

21.1.7.2 Basis of the Mine Capex

The underground mine plan is based on three shifts of eight hours, with one hour of idle time within the shift. The seven (7) hours of working time per shift comprises six (6) hours of normal time and one (1) hour of overtime. From this basis, each task in the mine schedule is assigned a duration or rate, depending on the type of activity. Resources and constraints are assigned to tasks and their capacity profiled accordingly. The required number of each equipment is calculated from the total number of hours. The labour work force required to complete the work is calculated based on equipment fleet. Support crew and equipment are calculated from the direct equipment fleet and primary work force.

The Capex is based on the initial portal box-cut, decline development, and the first ventilation raise being mined by a contractor, with all subsequent mine development and production being undertaken by the Owner. Further Capex allowance is provided for an underground support plant and surface mine infrastructure. Underground support plant includes the main and auxiliary fans, the main mine dewatering pump system and additional submersible pumps, and safety gear that includes five (5) self-contained refuge chambers. Surface mine infrastructure includes a temporary laydown area, warehousing, a shotcrete/concrete batch plant, the fuelling system, electrical substations and switchgear, explosives and detonator magazines, surface settling pond and piping systems.

Development rates for the decline and ventilation shaft are obtained in the market from regional and international contractors in a formal tendering process. A bill of quantities was produced to quantify all mine development work items in accordance with a detailed preamble describing each item. Direct and indirect costs for the Capex were selected following adjudication of tenders.

The database of unit cost inputs for the estimate were derived from the following sources:

- Quotations from suppliers for new equipment.
- Cost data from other mines and projects.

- Estimates based on experience from other projects.
- Capital Acquisition costs primarily comprise mobile and fixed equipment purchases, including associated expenses such as shipping to the site and commissioning.

21.1.7.3 Assumptions

The following assumptions are made in preparing the cost estimate for the underground mine:

- Quotations obtained from vendors for equipment and materials are provisional and are intended solely for budgetary purposes. GMD has reviewed and deemed this reasonable for the current update.
- Tailings will be suitable for preparing paste backfill with the required quality and assumed cost.
- Waste rock generated from development will be suitable for preparing cemented rock backfill with the required quality and assumed cost.
- Personnel with the required experience and skills will be available starting from the initiation of the development of the twin declines. An Operational Readiness Plan (ORP) will ensure that there are sufficient costs and list the required skills to start and operate the facility.
- Mining, mobile and other equipment reaching the end of their useful life will be replaced with new units; consequently, the cost estimate does not include overhauls.
- Decommissioned equipment has no salvage value. Except for activities related to raise boring, DPM employees carry out all tasks within the underground mine, including service hole drilling and casing, horizontal development, construction of underground infrastructure, and installation of permanent services.

21.1.7.4 Exclusions and Exceptions

The mine capital estimate is a direct cost estimate and as such the following items are not included in the Capex for the underground mine:

- Indirect Costs.
- Operational Readiness.
- Any work outside the defined battery limits for the underground mine.

21.1.8 Process Plant and Associated Infrastructure

The process plant and infrastructure Capex includes the facilities described in this Report and is summarised as follows:

- **Process Plant**: Ore handling, crushing, reclaim and conveying; grinding; concentration and product handling; reagents; buildings, equipment and facilities; and utilities.

- **Waste Management**: Filtered tailings storage facility, waste rock, surface water management and effluent water treatment.
- **Plant Infrastructure**: Site development and roads, power generation and distribution and buildings.
- **Off-site Infrastructure**: offsite facilities in San Gerardo and Cuenca.

21.1.8.1 Quantity Development Basis

Quantities were developed based on the Mechanical Equipment List and supplemented by estimates or allowances, wherever applicable. The quantities were developed based on the following documents:

- Site Plot Plans.
- Mechanical Equipment List (MEL).
- Electrical Equipment List (EEL).
- Process Flow Diagrams (PFDs).
- Process Control Diagrams (PCDs).
- Layout Drawings.
- Basic Electrical Single Line Diagram.
- Sketches.

21.1.8.2 Earthworks

Earthworks are quantified by DRA from the site plan, general arrangement drawings, terrace model and access road design. A bill of quantities was produced by DRA, quantifying all earthworks items in accordance with a detailed preamble describing each item to allow for clear and consistent pricing.

Earthworks rates were obtained in the market from regional Contractors during a formal tendering process. Earthworks rates were selected following adjudication of tenders and clarifications with the Contractors. These rates were populated in the estimate to determine the Capex.

The earthworks Contractor's indirect costs were priced and applied as a percentage in the estimate.

21.1.8.3 Civil (Concrete) Works

Concrete quantities are quantified by DRA from the site plan and general arrangement drawings developed for this Project. A bill of quantities was produced quantifying all concrete items in accordance with a detailed preamble describing each item to allow for clear and consistent pricing.

Concrete rates were obtained in the market from regional contractors during a formal tendering process. Concrete rates were selected following adjudication of tenders and clarifications with the contractors. These rates were populated in the estimate to determine the Capex.

The concrete contractor's indirect costs were priced and applied as a percentage in the estimate.

21.1.8.4 Building Works

Building works were quantified from preliminary layout drawings and a schedule of square meterage produced by DRA. Buildings will be largely prefabricated modules, for ease of assembly on site.

Building rates were obtained from local vendors and contractors, benchmarked against DRA internal database information and used to populate the estimate.

The building works contractor's indirect costs were estimated and applied as a percentage in the estimate.

21.1.8.5 Structural Steelwork Supply and Erection

Steelwork quantities for plant structures were estimated by DRA from general arrangement and layout drawings specifically prepared for this Project referencing similar concepts from DRA's database. The material quantity estimates cover steelwork and ancillary equipment, i.e., sheeting, grating, handrailing, stair treads, etc. These quantities were put directly into the Capex.

Structural steel fabrication and erection rates were obtained from the market from regional contractors during a formal process of obtaining budgetary quotes. The steel supply and erection rates were selected following adjudication of tenders and clarifications with the contractors. These rates were populated in the estimate to determine the Capex.

Unit rates were priced per tonne of steel, with separate rates provided for shop fabrications, delivery to site and erection on site. Rates for steel fabrication, delivery and erection are categorised as either, light, medium or heavy steelwork. Rates for erection on site are for all heights and regardless of position in the structure.

The Contractor's indirect costs were priced and applied as a percentage in the estimate.

21.1.8.6 Platework and Lining

All platework and liner quantities are estimated and quantified by DRA from general arrangement and layout drawings. The material take-offs were inserted directly into the Capex.

Platework rates were obtained in the market from regional contractors during a formal process of obtaining budgetary quotes, and were selected following adjudication of tenders and clarifications with the contractors. These rates were populated in the estimate to determine the Capex.

The Contractor's indirect costs were priced and applied as a percentage in the estimate.

21.1.8.7 Mechanical Equipment

DRA issued enquires to prominent suppliers of mechanical equipment and received quotations. Up to three (3) quotations were received for major equipment (where available) while minor equipment was generally single sourced. Quotations for major equipment were technically and commercially adjudicated to determine selection and inclusion in the estimate.

The erection cost for mechanical equipment is based on rates from the equipment suppliers or from quotations from qualified contractors. Erection costs for minor and ancillary items are taken from previous quotations or DRA's database.

The equipment installation contractor's indirect costs were priced and applied as a percentage in the estimate.

21.1.8.8 Piping

Piping quantities for the plant were estimated by DRA from general arrangement and layout drawings prepared for this Project referencing similar concepts from DRA's database. The quantity take-off covered piping above 50 mm diameter with allowances for service piping below 50 mm.

Overland piping was quantified with preliminary material take-offs and priced using rates from suppliers and fabricators of piping systems. These quantities are inserted directly into the estimate.

Piping fabrication and erection rates were obtained in the market from regional Contractors during a formal tendering process. The piping supply and erection rates were selected following adjudication of tenders and clarifications with the Contractors. These rates were populated in the estimate to determine the capital cost.

Unit rates are priced per metre of pipe, with separate rates being provided for shop fabrications, delivery to site and erection on site. Allowances for valves was added to the piping costs.

The Contractor's indirect costs were priced and applied as per received proposals in the estimate.

21.1.8.9 Electrical and Instrumentation

Electrical and Instrumentation (E&I) equipment was priced using a combination of quotations, supplier pricing, DRA database and recent project information. Budget quotations were obtained for major equipment (such as MV switchgear, power and distribution transformers, electrical rooms, and MCCs) from reputable E&I suppliers represented in Ecuador.

Installation costs were developed using estimated quantities and priced by local installation contractors. Where additional information and rates were required for items or equipment not covered by quotations, this was obtained from DRA's E&I database prices.

Instrumentation quantities were based on a set of Process Control Diagrams (PCDs) and the MEL combined with the Instrumentation and Control components typical to concentrator plants similar to this Project. Instrument pricing was obtained from reputable international manufacturers.

The E&I Contractor's indirect costs are priced and applied as a percentage in the estimate.

21.1.8.10 Power Plant

Due to the uncertainty of receiving uninterrupted power from existing sources, DPM decided to pursue optional power sources and decided on a new power plant located adjacent to the Project site. Quotations for the complete power plant were obtained from suppliers and included in the Capex.

21.1.8.11 Transport and Logistics

Transport costs for major mechanical and electrical equipment are factored as a percentage of mechanical equipment costs based on DRA historical data and experience.

Transport costs for mechanical and electrical bulk materials, steelwork, platework, piping are quoted by the respective Contractors. Transport costs for modular buildings and camps are included.

21.1.8.12 Contractor Indirect Costs

Contractor indirect costs include all contractor overheads such as contractual requirements (safety, sureties, insurance, etc.), site establishment and removal, and company overheads. They also cover supervision, travel to and from site, contractor-supplied temporary facilities, offices and laydown areas, tools, and contractor equipment.

Indirect costs for each discipline are outlined in the sections above.

21.1.8.13 First Fills and Consumables

The extent of first fills and consumables is determined by DRA.

21.1.8.14 Spare Parts

Commissioning and one-year operating spares for mechanical equipment are estimated by a combination of priced spares and percentage of mechanical equipment supply prices.

Strategic spares are identified and individually priced by the relevant vendors and included in an itemised Strategic Spares schedule in the Capex.

21.1.8.15 Consultants (External)

Engineering and Procurement (EP) costs for mining, processing and associated infrastructure were developed from first principles by DRA and other consultants considering the team organisation structure, in-country detail design and Project schedule. EP costs were benchmarked to similar projects and are considered appropriate for the type, nature, size and location of the Project, and includes reimbursable travel and other sundry costs.

Section 21.1.6 addresses “Pre-Construction Decision Costs” as they relate to external consultant costs.

Construction management (CM) costs for the Project were developed by DPM.

Other Consultants include the following:

- Mining, Paste Plant, FTSF.
- Quality Control.
- Geotechnical Services.
- Fire Protection Consultant.
- FTSF Site Surveying Services.

21.1.8.16 Assumptions and Exclusions

All estimates are developed within a frame of reference defined by assumptions and exclusions. Specific assumptions on which the Capex is based are noted as follows:

- Quotes from vendors for equipment and materials are valid for budget purposes.
- Suitable backfill material will be available locally. Soil conditions will be adequate for foundation bearing pressures.
- Engineering and construction activities will be carried out in a continuous program with full funding available including contingency.
- Labour productivities are established with input from experienced local contractors and checked against DRA’s in-house database of current projects.
- Bulk materials such as concrete, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.
- Estimate scope is restricted to the battery limits as shown on the site plan, PFDs, equipment list and more specifically as described in the Scope of Work.

The following are not included in the Capex, and are specifically noted as exclusions:

- Escalation (other than escalating selects Q3 2023 costs to Q2 2025).

- Cost of schedule delays such as those caused by:
  - Scope changes.
  - Events that would be considered Force Majure.
- Environmental permitting activities.
- Financing, acquisition, and sunk costs.
- “Pre-Construction decision” costs.
- Additional studies prior to EPCM.
- All royalties, commissions, lease payments, rentals and other payments to landowners, title holders, mineral and surface rights holders, and other third parties not mentioned herein.
- Forward cover for any foreign content (assumed to be accommodated by DPM, if necessary).
- All operating costs.
- Any work outside the defined battery limits.
- Provision for project risks outside of those related to design and estimating confidence levels.
- Fuel storage, other than the provision of any fuel bunkers in the scope of work.
- Interest on capital loans.
- Costs expended after completion of this Report and before approval for Project execution.
- Mineral rights and the purchase or use of land.

21.1.8.17 Design and Growth Allowances

All quantities provided by Engineering and/or contractors were taken off as neat quantities. During the preparation of Capex, it is a rare occasion when everything is known, specific and measurable.

Design growth allowances are:

- Considered as a “known unknown” requirement, when preparing the direct cost and more accurately reflect what the final design will require.
- Applied at the direct cost level, to compensate for the degree of engineering that is incomplete and subject to minor modifications.

Design growth allowances will be added to compensate for known items not identified in the estimate line items (“known unknowns”).

This is distinct from contingency that is included below the line, which allows for “unknown unknowns.” Contingency is a separate account. Design allowances are built around currently identified scope and do not make provision for scope changes. Scope changes are excluded and should be treated as variations to the capital cost of the project.

Allowances for bulk materials will be applied to quantities and include growth on both labour and material.

21.1.9 CAPEX PRICING BASIS

DRA established bulk commodity quantity estimates through preliminary engineering for the layouts prepared for the Project, which are estimated with necessary adjustments for the level of accuracy. Pricing Basis.

Estimated pricing was determined from a combination of budget quotations, historical data, benchmarked industry factors and allowances. Table 21.3 summarises the source of pricing used by major commodity expressed as a percentage by value of the project direct cost.

A Capex for the Project was developed in Q3 2023, with most costs obtained from the market.

For the current Capex, updated pricing was obtained for most major mechanical and electrical equipment for Q2 2025. The construction contract pricing was based on the following:

- Bulk Earthworks – Q2 2025 budgetary pricing.
- Concrete – Q2 2025 budgetary pricing.
- Structural Steel – Q2 2025 budgetary pricing.
- Platework – Q3 2023 budgetary pricing escalated to 2025.
- Mechanical Installation – Q3 2023 budgetary pricing escalated to 2025.
- Piping – Q3 2023 budgetary pricing escalated to 2025.
- Electrical and Instrumentation – Q3 2023 budgetary pricing escalated to 2025.
- Buildings and Camp – Q2 2025 budgetary pricing.
- Mine decline – Q2 2025 budgetary pricing.

Table 21.3 – Pricing Source Breakdown

Description	Budget Quotation		Database (%)	Estimated (%)	Factored/ Allowance (%)
	%	Basis			
Bulk Earthworks	100	Q2 2025	0	0	0
Detail Earthworks	100	Q2 2025	0	0	0
Concrete	100	Q2 2025	0	0	0
Structural Steel	100	Q2 2025	0	0	0
Architectural Building	90	Q2 2025	0	0	10
Mechanical Equipment	90	Q2 2025	10		

Description	Budget Quotation		Database (%)	Estimated (%)	Factored/ Allowance (%)
	%	Basis			
Platework	100	Q3 2023 + escalation	0	0	0
Piping	100	Q3 2023 + escalation			
Electrical	80	Q3 2023 + escalation	20	0	0
Instrumentation	80	Q3 2023 + escalation	20	0	0

21.1.10 INDIRECT CAPEX

Indirect costs are those that specifically pertain to the design and construction of the Project, but are not considered direct costs (e.g., equipment, materials) used in the construction of the facilities. Indirect costs include design, procurement and construction management activities, vendor representatives, spare parts and first fills, and contractor's indirect costs.

21.1.10.1 Contractor Indirect Costs

Contractor indirect costs cover only mobilisation and demobilisation. All other indirect costs are included in the contractor rates provided for the direct cost.

21.1.10.2 Capital Spares and Inventory

Capital Spares and Inventory include:

- Capital Spares.
- Operational Spares.
- Commissioning Spares.
- Initial Fills.

21.1.10.3 Project External Consultants

Project External Consultants include:

- Consultant for Engineering Procurement.
- Consultant for Geology.
- Consultant for Geotechnical work.
- Consultant for Project Management.
- Consultant for EP support.

- Consultant Expenses.
- Vendor Representatives.

The requirements for vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment is deemed necessary for equipment performance warranties; and was calculated and included in the Capex.

21.1.10.4 Commissioning

Commissioning includes the cost for a commissioning team required for preparing the Process plant for the commissioning and ramping up of the plant operation.

Commissioning (up to hot commissioning) costs in the indirect estimate include:

- Supervision.
- List of packages.
- Preparation of turnover documentation.
- Quality revision.
- Adjustment of equipment, erection, and process specifications.
- Corrections.

The estimate includes the cost of support crews for commissioning among others, including:

- Tools and equipment.
- Safety equipment and services.
- Radios.
- Consumables.
- Transportation, housing and meals.

Pre-Commissioning Teams include:

- Start up and Commissioning-Owner's Team.
- Start up and Commissioning-EPCM Team.
- Start up and Commissioning-Contractor Support.
- Start up and Commissioning-Vendor Support.

Commissioning costs are included in the estimate.

21.1.10.5 Freight and Logistics

Freight and Logistics costs include allowances for the following items. The allowances are estimated on a percentage of equipment and material costs.

- Ocean freight.
- Land freight.
- Air freight.
- Brokerage and Agent Fees.

21.1.10.6 Owner's Costs

Owner's Costs were compiled by DRA with assistance from DPM. The following represents a list of items that DRA considers included in the Owner's costs:

- Definition drilling, assaying and related reports and models.
- Owner's Project administration team (allowance).
- Health safety and security.
- Closure and rehabilitation.
- Sunk costs (covered in the economic evaluation as supplied).
- Process rights, royalties, license fees, technology fees (addressed in financial model).
- Project financing and interest charges (covered in the economic evaluation).
- Training and recruiting of plant operating personnel (Operational Readiness).
- Working capital (covered in the economic evaluation) (Operational Readiness).
- Start-up and commissioning costs (DPM commissioning team).
- All costs associated with the permitting process.
- Builder's risk insurance.
- Cost of this or any other study.
- Emergency response.
- Environmental / permitting / government relations / community relations.
- Finance and general administration.
- Human resources.
- Information technology.
- Legal.

21.1.10.7 Escalation

Escalation costs past the base date are not included in the Capex.

21.1.11 CONTINGENCY

Contingency is a percentage applied to the direct and indirect costs; it is a provision of funds for unforeseeable capital costs within the defined project scope. Contingency is meant to cover normal inadequacies inherent in any project, execution definition and estimating deficiencies, but does not cover scope changes. These are due to the dynamic nature of project engineering and construction.

A contingency allowance was developed for the Capex based on the degree of definition of the scope and budgetary quotes received from vendors.

The Project capital estimate was developed as described in this section, supported by preliminary engineering which vary in level of development in specific discipline areas. The estimate is considered comprehensive and adequate for an FS. Contingency was assessed per discipline considering the level of engineering development, level and confidence of market pricing for supply and erection costs received.

A schedule and project duration have been considered in the development of this estimate detailing the design development assessment and contingency value assigned to each discipline.

The design development allowance, reflected as contingency, is assessed as a measure of the confidence in the design and estimating processes / outputs.

For this estimate, a total contingency of 9% on project direct and indirect costs has been provided.

21.2 Operating Cost Estimate

21.2.1 Introduction

This Section describes the basis of estimate and approach taken for the FS Opex.

The Opex estimate is presented in US$ and uses prices obtained in Q2 2025 (or escalated from prices obtained in Q3 2023, where noted). The estimate is deemed to have an accuracy of ±15%. DRA developed the Opex in conjunction with DPM, with specific costs provided by external consultants for mining (GMD), backfill (Paterson & Cooke) and tailings disposal (NewFields).

All labour, materials and consumables required for the operation are included in this Opex. The process inputs were generated by DRA, based on data obtained from metallurgical testwork results from SGS and others, using quotations from reputable suppliers for plant consumables, as well as current design and DRA database information.

For economic modelling, the Opex is divided into the following major areas:

- Mine operations.
- Process plant.
- Backfill plant and tailings disposal.
- G&A.

Operating costs, when reported as a LOM average annual value, are stated as a weighted average cost basis in accordance with the LOM production plan.

The following are examples of cost items specifically excluded from the Opex:

- VAT (included in the financial model), import duties and / or any other statutory taxation, levies by national and / or local institutions.
- Project financing and interest charges.

21.2.2 Operating Cost Summary

A summary of the overall LOM Opex, by major Project area is presented in Table 21.4. The costs presented exclude pre-production operating. All pre-production costs are covered in the Capex presented in Section 21.1 of this Technical Report.

Table 21.4 – Project LOM Operating Costs by Major Area

Major Project Area	LOM Total (US$ M)	LOM Unit Cost (US$/t)
Mining	385	30.57
Processing	251	19.98
Filtered Tailings Management	68	5.42
G&A	135	10.70
Total Cost	**839**	**66.67**

Numbers may not sum precisely due to rounding.

A summary of the unit operating costs, by major Project area, and for the total Project over LOM is presented in Figure 21.1.

Figure 21.1 – Project LOM Unit Operating Costs



Source: DRA, 2025

This section of the Technical Report shows the operational costs based of unit cost per tonne milled. The throughput used to express this value is 3,400 t/d which represents the nominal daily throughput in the concentrator for Years 2 through 9. For the first year of operation and last two years of operation (Years 10 and 11), the plant will operate at a throughput of 3,000 t/d. As such, the Opex decreases proportional to the ore tonnage.

21.2.3 Summary of Personnel Requirements

Table 21.5 presents the estimated personnel requirements for the Project. This workforce comprises staff as well as hourly employees. Supervisory personnel as well as the administration employees will work on a 5 day per week basis.

The hourly workforce at the plant will work on a rotation to provide coverage 24 hours per day, 7 days per week. It is assumed that all personnel will be housed at the site.

Table 21.5 – Total Personnel Requirement (Year 5)

Discipline	Number
Mining	242
Processing	85
Maintenance	30
Administration	127
Total Personnel	**484**

21.2.4 Underground Mining Operating Costs

The mining Opex consists of personnel, equipment operation, and materials and services. The mining Opex was estimated based on Owner furnished equipment, materials and mine operators and based on the annual mine plans. Table 21.6 presents the unit rates that were applied to the tonnages for each period of the mine plan to arrive at the total expenditures for the mine operations.

Table 21.6 – Summary of Mining LOM Average Opex

Description	LOM Total Opex (US$ M)	Average Opex (US$/t mined)
Labour	163	12.92
Equipment Operation	42	3.33
Materials and Services	150	11.93
Power	16	1.28
Fuel	14	1.11
Total	**385**	**30.57**

Numbers may not sum precisely due to rounding.

The mining Opex estimate generally excludes:

- Pre-production operating costs.
- Costs for surface activities such as ore processing, tailings, paste plant operation, and administrative works.
- Surface labour, including health and safety personnel.
- Capitalised underground development and construction activities.
- Any factors for contingency or escalation.

21.2.5 PROCESS OPERATING COSTS

The estimated mineral processing LOM operating costs for the plant production are summarised in Table 21.7. The unit cost is calculated based on total LOM ore processed through the concentrator. These costs are derived from testwork, supplier information, DPM inputs and / or DRA experience.

Table 21.7 – Summary of Processing LOM Average Opex

Description	LOM Total (US$ M)	LOM Annual Average (US$ M)	Unit Cost (US$/t)
Labour	83	7.79	6.62
Power	59	5.70	4.73
Consumables	54	5.35	4.33
Reagents	33	3.23	2.61
Maintenance	17	1.80	1.32
Miscellaneous	5	0.44	0.37
Total Process	**251**	**24.31**	**19.98**

Numbers may not sum precisely due to rounding.

21.2.5.1 Labour Costs

Process plant labour rates used for the Report were provided by DPM. The structure and number of team members were collectively determined by DPM and DRA.

The processing plant labour costs exclude any cost for administration and security. These costs are accounted for under G&A costs. Laboratory labour costs are included in the overall plant labour as well as FTSF and paste and backfill plant personnel.

During the first year of operation, the working crew will consist of 18 expat and international labourers. These will be gradually replaced by local workers to decrease annual costs. This change will adjust the total labour costs.

21.2.5.2 Electrical Power Costs

The electrical load requirements for the process plant were prepared in the overall process plant load list. The quantity of power absorbed includes all equipment categorised under WBS 2000, 3000, 4000, 5000 and 7000. The estimated average running load is calculated using expected power draw as determined for individual items, and after applying utilisation and electrical correction factors.

Power for the Project will be supplied by the grid, with back-up power provided by a fully redundant diesel-powered power plant. It is assumed that grid power will be available 11 months of the year, with unplanned downtime throughout the year, totalling one (1) month of downtime (during this time

the power plant will provide power). The overall estimated cost of power is $0.105/kWh, based on both grid power costs, and diesel consumption and unit costs.

21.2.5.3 *Process Plant Reagents and Grinding Media*

Numerous reagents are consumed in the plant, all of which are detailed in the Process Design Criteria (PDC). The consumption rates for the various reagents are based primarily on results obtained from laboratory scale testwork conducted by SGS Lakefield. In the absence of metallurgical testwork, reagent consumption estimates are based upon either vendor recommendations or determined from first principles, as detailed in the PDC.

The supply prices for all reagents are based on quotations obtained from reputable suppliers. The costs include a quoted overland transportation allowance to site and include all clearance charges and taxes that may be incurred.

Grinding media consumption is based on results obtained from testwork, namely abrasion indices, and established models. Consumption rates for the SAG mill and regrind mills are verified with the vendor. Supply prices for grinding and regrind media, are based on quotations obtained from reputable suppliers.

All reagents and consumables that are consumed in the laboratory, as well as all minor reagents, are excluded as the contributions are regarded as insignificant.

21.2.5.4 *Liners and Consumables*

The cost of liners and wear parts, also referred to as the front-end section of the plant, is based on the abrasion indices obtained from testwork conducted by SGS Lakefield. Wear rates for liner replacement frequencies are estimated based on vendor supplied data. Quoted prices for liner sets were obtained from the respective crusher original equipment manufacturer (OEM).

Unit costs (for full sets) were provided by various mill suppliers in their tenders for the Project.

The cost of the replaced filter cloths was referenced from the commissioning spares cost provided from an equipment vendor quote. It is estimated that each filter press will consume one (1) set of filter cloths per year.

21.2.5.5 *Maintenance Cost*

The maintenance costs for the plant were calculated based on estimates derived from the mechanical, platework and piping capital estimates for the Project. The maintenance cost is approximated based on a mechanical replacement for the crushing and grinding circuit equipment, flotation, concentrate handling, reagents, utilities, samplers, process buildings, mobile equipment, and facilities.

21.2.5.6 Miscellaneous Costs

Provision is made for a fully equipped assay and metallurgical laboratory. The labour costs for laboratory operation are covered under the process plant labour costs, while an additional allowance of US$240,000 per annum is provided for consumables. These consumables include mobile equipment, fuel, and all equipment necessary to conduct the mineral analysis for the plant. An allowance for external assaying is provided under “Miscellaneous” costs.

Minor operating costs (US$200,000 per annum) for the continued maintenance and operation of the water treatment plant and sewage treatment plant are provided under “Miscellaneous” costs.

21.2.6 Filtered Tailings Management

LOM operating cost of the filtered tailings management is estimated at US$68 M or US$5.42/t.

21.2.7 General and Administration Costs

Project-wide G&A costs include the following categories and allowances:

- Corporate and overhead labour.
- Corporate costs.
- Site costs.
- Mine support costs.

The G&A labour costs for the Project encompass all management, support, and overhead labour related to the mine. This includes both corporate and in-country personnel costs. However, the G&A allowance does not cover any costs related to corporate staff members in Toronto or Quito.

21.2.7.1 Corporate Costs

Corporate cost allowances include costs associated with insurances, legal services, accounting services and systems, property leases, corporate overhead costs, leases and local taxes.

21.2.7.2 Site Office Costs

Site office costs provide allowance for the following items:

- Transport.
- Fire protection.
- Site security.
- Information technology, software and communications.
- Health, safety and environment.
- Training and human resources.

- Miscellaneous office costs.

21.2.7.3 G&A Mine Support Costs

Mine support costs include all costs associated with employee travel, maintenance of light vehicles and external assaying allowances.

22 ECONOMIC ANALYSIS

22.1 General

The economic analysis presented in this Section contains forward-looking information as defined by Canadian securities legislation. The results of the analysis rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here.

The economic analysis is based on the Discounted Cash Flow (DCF) method on a pre-tax and after-tax basis. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis, although DPM may decide in the future to finance a portion of the Project with debt. The reference date of the analysis is aligned with the Final Investment Decision (FID) on the Project. Current Ecuadorian tax regulations were used to assess corporate tax liabilities. DPM retained EY Addvalue Asesores Cia. Ltda.to review tax calculations for the after-tax economic evaluation of the Project.

The cash flow model uses inputs from all elements of the Project to provide a comprehensive financial projection for the entire Project on an annual basis. The key metrics determined in the analysis are the Net Present Value (NPV) at a discount rate of 5%, the Internal Rate of Return (IRR), and the Payback Period. A sensitivity analysis was carried out to assess the impact of variations in metal prices, Capex, and Opex on the financial metrics.

The production schedule used in this analysis is based on the Life of Mine (LOM) Plan and the Concentrate Production Schedule outlined in Sections 16 and 17, respectively. The Capex and Opex are taken from the estimates detailed in Section 21.

All costs and pricing are in Q2 2025 United States dollars. No provision is made for the effects of inflation in this analysis.

22.2 Forward Looking Information

The results of the economic analyses discussed in this section represent forward-looking information as defined by Canadian securities legislation. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes assumptions and estimations of:

- Price of gold, silver, and copper.
- Amount of mineralised material and material grade.
- Proposed mine production plan.
- Mining dilution and mining recovery.
- Geotechnical or hydrogeological considerations during mining.

- Process plant production plan.
- Recovery rates of metals in the processing plant.
- Ability of plant, equipment, processes to operate as anticipated.
- Development capital costs.
- Sustaining and operating costs.
- Closure costs and unforeseen reclamation expenses.
- Environmental, social, and licensing risks.
- Taxation policy and tax rate.
- Royalty agreements.
- Cost inflation.
- Timing of the Project in terms of permitting approval, project development, construction, operation, and closure.
- General business and economic conditions.
- Ability to maintain social license to operate.
- Unrecognised environmental risks.

22.3 Assumptions

The following assumptions were made in the development of the economic analysis:

- Reference date for the analysis aligned with a FID on the Project.
- Revenue from the Project is derived from the sale of copper concentrate and pyrite concentrate.
- All concentrate is sold in the same year that it is produced.
- A flat 5% Net Smelter Return (NSR) Production Royalty was applied. The terms of the royalty are subject to ongoing negotiations between DPM and the Government of Ecuador.
- Corporate tax liabilities were assessed based on current Ecuadorian tax regulations.
- Corporate income tax rate of 20%, based on the Investor Protection Agreement (IPA) between DPM and the Government of Ecuador signed in 2022.
- State and employment tax is deductible for income tax purposes.
- The quantum of import duty exemptions is considered the same as that specified in the IPA between DPM and the Government of Ecuador signed in 2022. This Agreement is subject to renegotiation.
- The carrying value (sunk cost) of the Project provided by DPM.
- Discount rate of 5%.

- Long-term metal prices of US$1,900.00/oz gold, US$27.50/oz silver and US$4.00/lb copper were selected based on DPM market analysis.
- All costs and pricing are in Q2 2025 US dollars.
- No provision is made for the effects of inflation in this analysis.
- All environmental approvals, required permits, licenses and authorisations will be obtained from the relevant governments and other relevant stakeholders.
- The analysis is conducted on an unlevered, all-equity basis.

22.4 Economic Analysis Parameters

The key parameters used in the financial analysis are summarised in Table 22.1.

Table 22.1 – Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Commodity Price		
Gold	US$ per oz	1,900.00
Silver	US$ per oz	27.50
Copper	US$ per lb	4.00
Project Parameters		
Discount Rate	%	5.0
Mine Life	years	11
Mineral Reserve (LOM)	Mtpa	12.6
Grade Mined (LOM average)		
Gold	g/t Au	4.7
Silver	g/t Ag	28.6
Copper	% Cu	0.29
Annual Mill Throughput	Mtpa	1.241
Recovery (LOM average)		
Gold	%	89.7
Silver	%	94.8
Copper	%	95.1
Payability (LOM average)		
Gold	%	96.2
Silver	%	81.8
Copper	%	80.5
Total Payable Metal Production (LOM)		

Description	Unit	Value
Gold	Moz	1.6
Silver	Moz	9.0
Copper	kt	27.8
Gold Equivalent[5]	Moz Au Eq.	1.9
Average Annual Payable Metal Production (LOM)		
Gold	oz	149,170
Silver	oz	814,340
Copper	tonnes	2,527
Gold Equivalent[5]	oz Au Eq.	172,685
Government Royalty (NSR) [1]	%	5.0
Capital Cost Estimate [2]		
Initial Capital	US$ M	593
Sustaining Capital (LOM)	US$ M	75
Closure	US$ M	28
LOM Unit Operating Costs (LOM Average) [3]		
Mining	US$/t ore processed	30.6
Processing	US$/t ore processed	20.0
Paste Backfill	US$/t ore processed	5.4
General & Administrative	US$/t ore processed	10.7
Total Opex	US$/t ore processed	***66.7***
Cost Metrics (LOM Average) [4]		
Cash Cost per Tonne of Ore Processed	US$/t ore processed	99
Cash Cost per Ounce of Gold Sold	US$/oz payable gold	809
All-in Sustaining Cost per Ounce of Gold Sold	US$/oz payable gold	873

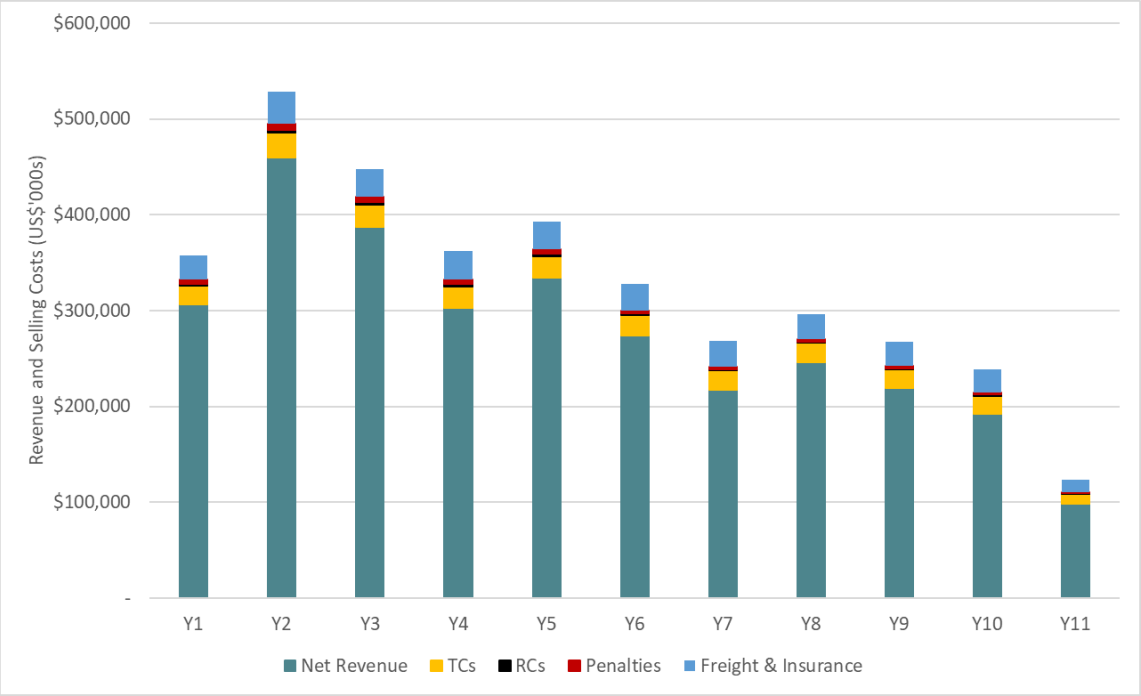
Notes:

1. The Government Royalty is currently under negotiation between DPM and the Ecuadorian Government. A flat NSR royalty rate has been used in this analysis.
2. Capex are inclusive of applicable import duties and taxes.
3. Unit Opex numbers are exclusive of applicable import duties, taxes and VAT.
4. Cash Cost per Tonne Processed, Cash Cost per Ounce Sold, and All-in Sustaining Cost per Ounce Sold are non-GAAP financial measures or ratios and have no standardised meaning under IFRS and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS. Refer to the "Non-GAAP Financial Measures" in Section 2.7 of this Report for more information, including a detailed description of each of these measures.
5. Gold equivalent calculations are based on the metal prices selected for the project: gold price of $1,900/oz, silver price of $27.50/oz, and copper price of $4.00/lb.
6. Numbers may not sum precisely due to rounding.

22.5 Revenue

Over the LOM, the Project generates US$3.6 Billion in gross revenue. After consideration of selling costs, the net revenue over the LOM is US$ 3.0 Billion. Figure 22.1 presents the profile of revenue and the various selling costs over the life of the Project.

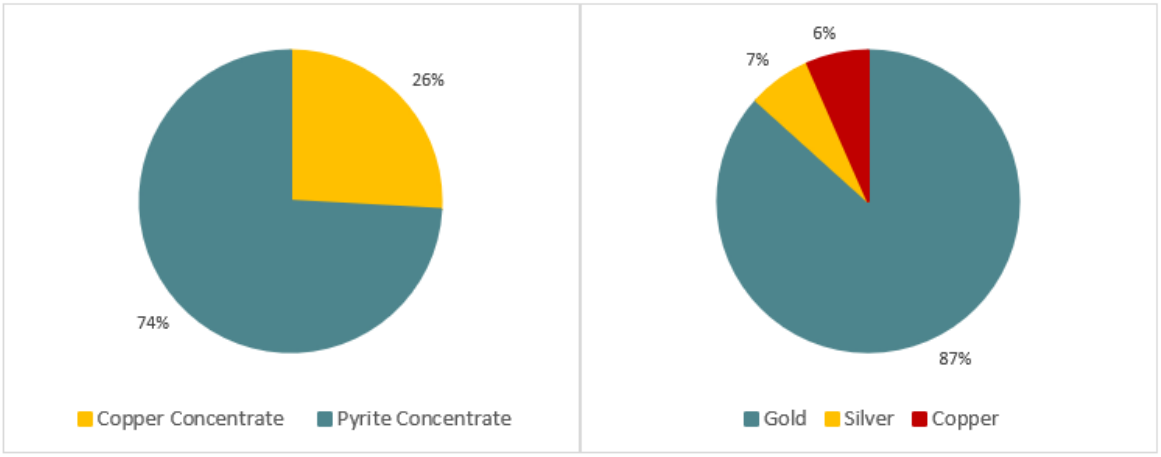
Figure 22.1 – Revenue Profile over the LOM



Source: DRA, 2025

Figure 22.2 shows that the majority (87%) of the net revenue over the LOM is attributable to gold. The remainder is split evenly between silver and copper. Approximately three-quarters of the LOM net revenue is accounted for by the pyrite concentrate. However, the copper concentrate contains more value per tonne of concentrate: $2,216/tonne of copper concentrate and $1,563/tonne of pyrite concentrate.

Figure 22.2 – Proportion of Net Revenue Attributable to Concentrate (Left) and Metal (Right)



Source: DRA, 2025

22.6 Mineral Royalties and Agreements

22.6.1 Production Royalty

The Project is subject to a fixed 5% royalty on net revenue from sales of precious metals and related by-products, payable to the Government of Ecuador. The rate and terms of this royalty are subject to renegotiation between DPM and the Government of Ecuador at a future date as part of the Exploitation Contract.

Under the existing agreement, US$15 M of royalties will be paid in advance of production ("Advanced Royalty"). The Advanced Royalty is deductible from Production Royalty obligations during the production period at the lesser of:

- 50% of royalties payable in a six (6) month period; or
- 10% of Advanced Royalty balance (Advance Royalty paid less deductions taken).

Based on the mine plan, the second method will apply, and the Advanced Royalty will be fully recovered over the LOM.

22.6.2 COGEMA Agreement

Under an agreement signed on March 5, 1999, between IAMGOLD and COGEMA (now Orano), COGEMA is entitled to payments following a production decision on the Project. In the pre-production phase, the payable amount is calculated based on two US dollars (US$2.00) per ounce of gold contained in proven and probable reserves plus the indicated and measured resources defined in the Feasibility Study. This payment is made in three (3) equal annual instalments from the date on which a formal production decision is made.

Following the start of production, COGEMA will be entitled to a 5% Net Profit Interest (NPI) on any mining operation undertaken.

22.6.3 IAMGOLD AGREEMENT

The Project is subject to a payment to be paid to IAMGOLD.

22.7 Taxation

The Project is subject to Income Tax, State and Employment Tax (or Profit Sharing Tax), and other non-income taxes such as Value Added Tax (VAT), import duties, and Sovereign Adjustment. To support the after-tax economic evaluation of the Project, DPM retained EY Addvalue Asesores Cia. Ltda., based in Quito, to review the tax calculations included in the financial model.

22.7.1 INCOME TAX

Corporate income tax is levied at a rate of 25% on companies domiciled in Ecuador and on foreign companies. For companies incorporated in Ecuador, income tax is payable on worldwide income. Under the terms of the existing IPA between DPM and the Government of Ecuador, a reduced income tax rate of 20% is applicable to the Project.

The income tax base is determined by the total taxable income less deductions available to taxpayers as per the tax legislation. The most notable of these deductions are depreciation and the State and Employment Tax. All deductions and rates are based on currently enacted legislation, and are subject to change in the future, or until the Company has a final IPA agreed with the Ecuadorian Government with tax stability benefits.

22.7.2 STATE AND EMPLOYMENT TAX

A State and Employment Tax (or Profit Sharing Tax) is applicable to taxable income of the Project at a fixed rate of 15%. Based on the Project's scale of mining, the State portion of the tax is 12% and the worker's portion is 3%. State and Employment Tax is deductible for income tax purposes.

The State and Employment Tax base is determined by total taxable income less deductions available to taxpayers as per tax legislation. The most notable of these deductions is depreciation.

22.7.3 VALUE-ADDED TAX AND IMPORT DUTIES

The following taxes and import duties are applicable on Capex and Opex items for the Project:

- Value-Added Tax (VAT): levied on the transfer of goods, imports, rendering of services and intellectual property rights at a general rate of 15%. VAT is applicable to all goods and services from domestic and foreign sources, excluding select items such as labour, electrical

power, and transport. It is understood that certain imported goods are taxed at a VAT rate of 0% depending on the customs code used for importation.

- Ad Valorem Tax: Levied on the imports of raw materials, supplies and capital goods contained on a list issued by the Tax Policy Committee, generally at a rate of 0% to 40% depending on item type (custom code) and country of origin.
- Currency Outflow Tax: A tax of 5% levied on all cash remitted abroad. The IPA between DPM and the Government of Ecuador stabilises this tax at a reduced rate of 3.5%.
- *Fondo de Desarrollo para la Infancia* (FODINFA): A custom duty applied at a rate of 0.5% as a contribution to the Development Fund for Children.

In recognition of the level of engineering definition in the Feasibility Study, a detailed line-by-line review of the Capex and Opex has not been conducted at this stage to precisely determine the taxes and import duties payable for each item, particularly the VAT and the Ad Valorem tax rate applicable. In this analysis, a flat 15% VAT rate has been applied to all Capex and Opex (excluding labour). An estimate of the import duties payable was prepared based on an assessment of the proportion of imported goods and services in the Capex and Opex, and the determination of a weighted average Ad Valorem tax rate from the list of capital items included in the existing IPA.

The existing IPA allows for VAT exemptions of US$13.9 M on US$110 M worth of imported capital (CIF basis) (subject to renegotiation), which have been applied. Tax legislation established that mining companies have the right to obtain a VAT refund related to the production of exported goods. The refund limit is 12% of the customs value of the exported goods; any VAT exceeding this amount is carried forward for recovery on the basis of exports in subsequent months. VAT paid is recoverable for a period of up to five (5) years following the filing of the VAT return once mine production begins, after which time these amounts expire and are considered a non-deductible expense. VAT payable on certain G&A expenditures not directly related to mine production, on Closure Capital and on payments under the COGEMA Agreement is considered non-refundable.

The weighted average Ad Valorem tax rate determined was 8.2%. An import duties and Currency Outflow Tax exemption of US$9.4 M was agreed in the existing IPA. This Agreement is pending renegotiation and these exemptions may vary. In the absence of any additional information, an equivalent import duties exemption of US$9.4 M was used in the current analysis.

22.7.4 Sovereign Adjustment

The Ecuadorian constitution requires that the government receives at least 50% of the benefits from non-renewable resource projects. Sovereign Adjustment is only payable when the present value of cumulative benefits from the Project to the Mining Concessionaire exceeds the present value of cumulative benefits to the government. Therefore, Sovereign Adjustment is triggered when the government's share of the cumulative benefits is below 50%.

The benefits to DPM are calculated annually as the net present value of the cumulative free cash flows of the Project since the concession was granted. The benefits to the Government of Ecuador are calculated annually as the net present value of the cumulative sum of corporate income tax, Production Royalty, State portion of the Profit-Sharing Tax, non-recoverable VAT, and prior year Sovereign Adjustment payments, if any. The discount rate used in the Sovereign Adjustment calculation is 20.9% on an unlevered basis, as calculated by DPM and determined in accordance with the formula established under Ecuadorian mining law.

At the base case metal prices presented in this Report, the Project will not trigger the Sovereign Adjustment payment.

22.7.5 Other Taxes

Other taxes considered in the analysis include:

- **Conservation Patent:** An annual fee paid by the mining concessionaire, calculated based on the hectares per concession, the Unified Basic Renumeration (UBR) and the stage of the concession.
- **Municipal Patent:** An annual tax levied by municipalities on all companies that carry out economic activities within municipal jurisdictions.
- **1.5x1000 Assets Tax and Contribution to the Superintendency of Companies:** Annual taxes calculated based on assets less current liabilities.

22.7.6 Depreciation and Amortisation

Calculations of depreciation and amortisation of capital are performed based on the current regulations under the Ecuadorian tax regime to determine eligible deductions from taxable income for Income Tax and State and Employment Tax. Two (2) main depreciation/amortisation methods are applied: Straight Line and Units of Production (UOP).

22.7.6.1 Straight Line Depreciation/Amortisation

Investments made during the prospecting, initial exploration, and economic evaluation phases and complementary exploration investments during the exploitation phase must be amortised in a straight line over five (5) years, counted from the start of production. The carrying value of the Project was assumed to cover the categories outlined above and has been amortised by this method.

Mobile mining equipment is depreciated on a straight-line basis over the useful life of each piece of equipment. For tax purposes, a minimum useful equipment life of ten (10) years is applicable.

22.7.6.2 UOP Depreciation

Construction and development investments must be depreciated by the UOP method from the start of production. The ounces of payable gold produced by the Project were defined as the units of production for this exercise. The capital depreciated by this method includes the Development Capex, Sustaining Capex and capitalised items such as capitalised patents/taxes.

22.8 Financial Analysis Results

Based on the assumptions outlined in this Section, the results of the analysis indicate a positive after-tax NPV of US$488 M at a discount rate of 5% and an after-tax IRR of 18.1%. A summary of the key financial metrics is presented in Table 22.2.

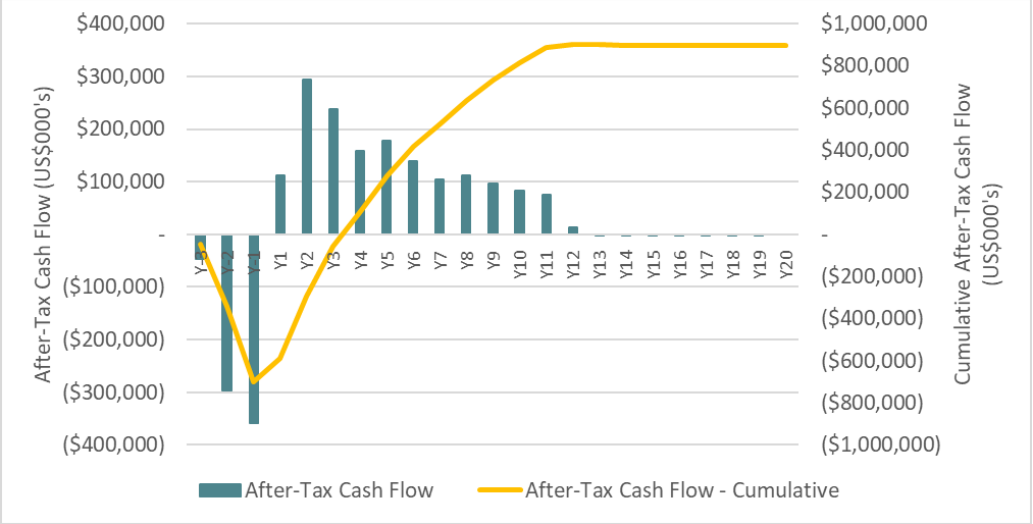
Table 22.2 – Financial Results

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	US$ M	627	488
IRR	%	21.3	18.1
Payback Period[1]	Years	2.9	3.4

Notes:
1. The Payback Period provided here is from the start of mill operation.
2. Numbers may not sum precisely due to rounding.

Figure 22.3 illustrates the annual and cumulative free cash flows of the Project on an after-tax basis. A summary of the financial model is presented in Table 22.3.

Figure 22.3 – Annual and Cumulative After-Tax Cash Flows over the Project LOM



Source: DRA, 2025

Table 22.3 – Financial Model Summary
(tail of Closure Capex payments not shown)

Period # / Period			1 Y-3	2 Y-2	3 Y-1	4 Y1	5 Y2	6 Y3	7 Y4	8 Y5	9 Y6	10 Y7	11 Y8	12 Y9	13 Y10	14 Y11	15 Y12	16 Y13	17 Y14
ECONOMIC DATA		*(Tot. / Avg.)*																	
Metal Price																			
Gold	*(US$/oz)*	**$1,900**	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900	$1,900
Silver	*(US$/oz)*	**$27.50**	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5	$27.5
Copper	*(US$/lb)*	**$4.00**	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00	$4.00
PRODUCTION DATA		*(Tot. / Avg.)*																	
Ore Mined																			
Ore Mined	*(ktonnes)*	**12,585**	-	-	160	963	1,229	1,230	1,241	1,232	1,239	1,228	1,238	1,240	1,107	477	-	-	-
Mine Life	*(years)*	**12**																	
Ore Grade																			
Gold	*(g/t Au)*	**4.7**	-	-	7.3	6.1	6.8	5.6	4.5	5.1	4.3	3.6	4.1	3.6	3.6	3.6	-	-	-
Silver	*(g/t Ag)*	**28.6**	-	-	24.9	30.2	37.4	35.5	37.2	35.1	28.3	19.9	20.0	19.7	24.8	23.7	-	-	-
Copper	*(% Cu)*	**0.29**	-	-	0.45	0.42	0.48	0.38	0.36	0.33	0.23	0.18	0.19	0.17	0.19	0.15	-	-	-
Arsenic	*(ppm As)*	**850**	-	-	1,179	1,199	1,296	1,088	1,023	966	726	586	591	566	625	458	-	-	-
Sulphur	*(% S)*	**5.55**	-	-	6.37	6.06	5.60	5.26	5.44	5.53	5.42	5.54	5.35	5.34	5.76	6.18	-	-	-
Ore Milled																			
Ore Milled	*(ktonnes)*	**12,585**	-	-	-	988	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,095	575	-	-	-
Plant Life	*(years)*	**11**																	
Ore Grade																			
Gold	*(g/t Au)*	**4.7**	-	-	-	6.2	6.8	5.7	4.6	5.1	4.3	3.7	4.1	3.7	3.6	3.6	-	-	-
Silver	*(g/t Ag)*	**28.6**	-	-	-	28.9	36.8	35.7	37.1	35.3	28.8	20.5	20.0	19.7	24.5	23.9	-	-	-
Copper	*(% Cu)*	**0.29**	-	-	-	0.41	0.48	0.40	0.36	0.33	0.24	0.18	0.19	0.18	0.19	0.15	-	-	-
Arsenic	*(ppm As)*	**850**	-	-	-	1,157	1,312	1,115	1,026	971	744	597	592	568	622	484	-	-	-
Sulphur	*(% S)*	**5.55**	-	-	-	6.09	5.71	5.26	5.42	5.52	5.43	5.53	5.36	5.34	5.74	6.11	-	-	-
Copper Concentrate Produced																			
Concentrate Sold	*(ktonnes)*	**355**	-	-	-	38	59	48	44	41	29	22	23	21	20	9	-	-	-
Concentrate Grade																			
Gold	*(g/t Au)*	**26.3**	-	-	-	24.6	23.0	23.6	20.3	24.6	28.9	31.9	34.6	33.3	31.1	37.5	-	-	-
Silver	*(g/t Ag)*	**512**	-	-	-	357	419	490	567	576	611	523	495	518	638	745	-	-	-
Copper	*(% Cu)*	**8.84**	-	-	-	8.92	8.97	8.90	8.86	8.84	8.76	8.71	8.71	8.70	8.71	8.68	-	-	-
Arsenic	*(ppm As)*	**3.5**	-	-	-	3.5	3.5	3.5	3.5	3.5	3.4	3.4	3.4	3.4	3.4	3.4	-	-	-
Sulphur	*(% S)*	**60.3**	-	-	-	60.8	61.2	60.7	60.4	60.3	59.7	59.4	59.4	59.3	59.4	59.2	-	-	-
Payable Metal																			
Gold	*(koz)*	**288**	-	-	-	29	42	35	28	31	26	22	24	22	19	10	-	-	-
Silver	*(koz)*	**5,257**	-	-	-	391	715	687	722	675	518	338	329	322	371	189	-	-	-
Copper	*(ktonnes)*	**27.8**	-	-	-	3.0	4.7	3.8	3.5	3.2	2.3	1.7	1.8	1.7	1.6	0.7	-	-	-
Pyrite Concentrate Produced																			
Concentrate Sold	*(ktonnes)*	**1,440**	-	-	-	117	147	135	139	142	140	142	138	137	130	73	-	-	-
Concentrate Grade																			
Gold	*(g/t Au)*	**30.3**	-	-	-	37.1	43.1	39.4	29.9	32.8	28.3	23.4	26.9	24.3	22.4	21.1	-	-	-
Silver	*(g/t Ag)*	**110**	-	-	-	108	127	135	134	128	115	88	89	88	97	89	-	-	-
Copper	*(% Cu)*	**0.15**	-	-	-	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	0.15	-	-	-
Sulphur	*(% S)*	**39.0**	-	-	-	41.1	39.7	37.8	38.4	38.8	38.5	38.9	38.2	38.1	39.7	41.1	-	-	-
Payable Metal																			
Gold	*(koz)*	**1,353**	-	-	-	135	197	166	130	145	123	102	115	103	90	47	-	-	-
Silver	*(koz)*	**3,701**	-	-	-	293	457	458	467	448	381	264	260	255	279	138	-	-	-
Copper	*(ktonnes)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Source: DRA, 2025

Table 22.4 – Financial Model Summary
(tail of Closure Capex payments not shown)

Period # / Period			1 Y-3	2 Y-2	3 Y-1	4 Y1	5 Y2	6 Y3	7 Y4	8 Y5	9 Y6	10 Y7	11 Y8	12 Y9	13 Y10	14 Y11	15 Y12	16 Y13	17 Y14
OPERATING & CAPITAL CASH FLOWS		*(Tot. / Avg.)*																	
Gross Revenue	*(US$'000s)*	**$3,609,118**	-	-	-	$357,135	$528,078	$447,975	$362,009	$392,916	$327,303	$267,932	$295,931	$267,837	$238,605	$123,397	-	-	-
Treatment Charges & Other Selling Costs	*(US$'000s)*	**$572,312**	-	-	-	$50,478	$68,425	$60,493	$59,593	$58,906	$53,312	$50,833	$49,885	$49,108	$46,420	$24,859	-	-	-
Net Revenue	*(US$'000s)*	**$3,036,806**	-	-	-	$306,656	$459,653	$387,481	$302,416	$334,011	$273,991	$217,099	$246,046	$218,729	$192,185	$98,538	-	-	-
Operating Costs	*(US$'000s)*	**$839,105**	-	-	-	$85,969	$90,144	$88,027	$80,459	$79,388	$77,632	$76,735	$74,682	$75,548	$67,802	$42,718	-	-	-
Mining	*(US$'000s)*	**$384,721**	-	-	-	$47,739	$47,003	$44,130	$36,812	$35,794	$34,039	$33,055	$31,035	$31,901	$27,948	$15,265	-	-	-
Process	*(US$'000s)*	**$251,496**	-	-	-	$20,634	$24,172	$24,928	$24,678	$24,624	$24,624	$24,711	$24,678	$24,678	$21,675	$12,094	-	-	-
Paste Backfill	*(US$'000s)*	**$68,179**	-	-	-	$5,350	$6,723	$6,723	$6,723	$6,723	$6,723	$6,723	$6,723	$6,723	$5,932	$3,113	-	-	-
G&A	*(US$'000s)*	**$134,708**	-	-	-	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	$12,246	-	-	-
Operating Costs per Tonne Ore Milled	*(US$/t milled)*	**$66.7**	-	-	-	$87.1	$72.6	$70.9	$64.8	$64.0	$62.6	$61.8	$60.2	$60.9	$61.9	$74.4	-	-	-
Operating Costs per Tonne Concentrate Sold	*(US$/t conc)*	**$467.5**	-	-	-	$554.6	$438.2	$479.1	$438.7	$435.1	$459.4	$466.6	$464.9	$475.5	$451.3	$524.6	-	-	-
Capital Expenditures*	*(US$'000s)*	**$657,947**	$32,582	$236,991	$302,563	$21,694	$12,003	$10,654	$9,112	$6,779	$4,357	$1,587	$4,023	$7,758	$10,658	$4,965	($13,160)	$1,777	$1,777
Development Capital	*(US$'000s)*	**$572,136**	$32,582	$236,991	$302,563	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*(US$'000s)*	**$73,017**	-	-	-	$21,694	$12,003	$10,654	$9,112	$6,779	$4,357	$1,587	$2,492	$3,790	$550	-	-	-	-
Closure Capital	*(US$'000s)*	**$27,732**	-	-	-	-	-	-	-	-	-	-	$1,531	$3,969	$10,108	$4,965	$1,779	$1,777	$1,777
Salvage Value	*(US$'000s)*	**$14,939**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$14,939	-	-
Change in Working Capital	*(US$'000s)*	**$0**	-	($5,850)	-	($59,492)	($4,691)	$18,681	$1,649	($4,040)	$7,498	$7,048	($3,583)	$3,347	$2,809	$36,623	-	-	-
ROYALTIES & OTHER PAYMENTS		*(Tot. / Avg.)*																	
Production Royalty	*(US$'000s)*	**$151,840**	$5,000	$5,000	$5,000	$13,833	$21,633	$18,159	$14,027	$15,716	$12,814	$10,058	$11,585	$10,291	$9,028	($303)	-	-	-
Pre-Production	*(US$'000s)*	**$15,000**	$5,000	$5,000	$5,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$136,840**	-	-	-	$13,833	$21,633	$18,159	$14,027	$15,716	$12,814	$10,058	$11,585	$10,291	$9,028	($303)	-	-	-
COGEMA Agreement Payments	*(US$'000s)*	**$53,624**	$1,492	$1,492	$1,492	$713	$8,607	$7,575	$4,349	$5,091	$5,452	$3,568	$4,077	$3,530	$3,091	$3,096	-	-	-
Pre-Production	*(US$'000s)*	**$4,475**	$1,492	$1,492	$1,492	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$49,149**	-	-	-	$713	$8,607	$7,575	$4,349	$5,091	$5,452	$3,568	$4,077	$3,530	$3,091	$3,096	-	-	-
IAMGOLD Agreement Payments	*(US$'000s)*	**$5,100**	-	$3,400	-	$1,700	-	-	-	-	-	-	-	-	-	-	-	-	-
TAXES & CREDITS		*(Tot. / Avg.)*																	
Income Tax	*(US$'000s)*	**$194,465**	-	-	-	$16,172	$35,723	$26,872	$17,320	$21,450	$20,527	$13,053	$16,886	$12,751	$9,668	$4,043	-	-	-
State & Employment Tax	*(US$'000s)*	**$171,901**	-	-	-	$14,584	$31,521	$23,711	$15,282	$18,927	$18,112	$11,517	$14,900	$11,251	$8,530	$3,567	-	-	-
Conservation Patent	*(US$'000s)*	**$3,352**	$110	$113	$117	$240	$247	$254	$260	$267	$274	$281	$287	$294	$301	$308	-	-	-
Pre-Production	*(US$'000s)*	**$340**	$110	$113	$117	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$3,013**	-	-	-	$240	$247	$254	$260	$267	$274	$281	$287	$294	$301	$308	-	-	-
Municipal Patent	*(US$'000s)*	**$1,650**	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75
Pre-Production	*(US$'000s)*	**$225**	$75	$75	$75	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$1,425**	-	-	-	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75
1.5x1000 Assets Tax	*(US$'000s)*	**$8,066**	$364	$782	$1,309	$1,248	$1,009	$803	$661	$514	$421	$338	$264	$190	$120	$43	-	-	-
Pre-Production	*(US$'000s)*	**$2,456**	$364	$782	$1,309	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$5,610**	-	-	-	$1,248	$1,009	$803	$661	$514	$421	$338	$264	$190	$120	$43	-	-	-
Contribution to the Superintendent of Companies	*(US$'000s)*	**$5,032**	$214	$490	$820	$794	$643	$513	$397	$311	$255	$206	$163	$119	$78	$30	-	-	-
Pre-Production	*(US$'000s)*	**$1,524**	$214	$490	$820	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*(US$'000s)*	**$3,508**	-	-	-	$794	$643	$513	$397	$311	$255	$206	$163	$119	$78	$30	-	-	-
Import Duties and Taxes	*(US$'000s)*	**$51,468**	$2,231	$11,224	$9,129	$2,883	$3,353	$3,228	$2,755	$2,741	$2,629	$2,588	$2,682	$2,820	$2,050	$1,330	($175)	$0	-
VAT Payable	*(US$'000s)*	**$169,211**	$4,215	$30,557	$38,789	$12,053	$11,079	$10,725	$9,539	$9,054	$8,460	$7,926	$7,902	$8,073	$6,567	$4,272	-	-	-
VAT Refund	*(US$'000s)*	**$175,441**	-	-	-	$36,799	$55,158	$21,690	$9,539	$9,054	$8,460	$7,926	$7,902	$8,073	$6,567	$4,272	-	-	-
Sovereign Adjustment	*(US$'000s)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Notes
* The Capital Expenditure numbers in this table are exclusive of applicable import duties, taxes, and VAT.

Source: DRA, 2025

22.9 Sensitivity Analysis

A sensitivity analysis was carried out, using the base case described above as a starting point, to assess the impact of changes in metal prices, total Capex, and Opex on the Project's NPV at a 5% discount rate and IRR. The impact of each variable is examined individually with an interval of ±20% and increments of 10% applied. It is to be noted that the margin of error for cost estimates at the Feasibility Study level is typically -15% +20%. However, the uncertainty in metal price forecasts remains significantly higher and is a function of price volatility.

The after-tax results of the sensitivity analyses are shown in Table 22.5, Figures 22.4 and 22.5. The NPV and IRR are most sensitive to variations in the gold price. Figure 22.4 indicates that variations in Capex and Opex have similar impacts on the after tax NPV, while Figure 22.5 shows that the after-tax IRR is more sensitive to Capex than to Opex. The economics are not particularly sensitive to either silver or copper prices as these metals make up a relatively small proportion of the revenue. The Project retains a positive NPV even at the lower end of the gold price range tested.

Table 22.5 – Sensitivity of After-Tax NPV and IRR to Metal Prices, Capex and Opex

Au Price	Units	-20%	-10%	Base	+10%	+20%
NPV @ 5.0%	US$ M	223	356	488	621	753
IRR	%	11.5	14.9	18.1	21.1	23.8
Payback	years	4.5	3.9	3.4	3.0	2.7

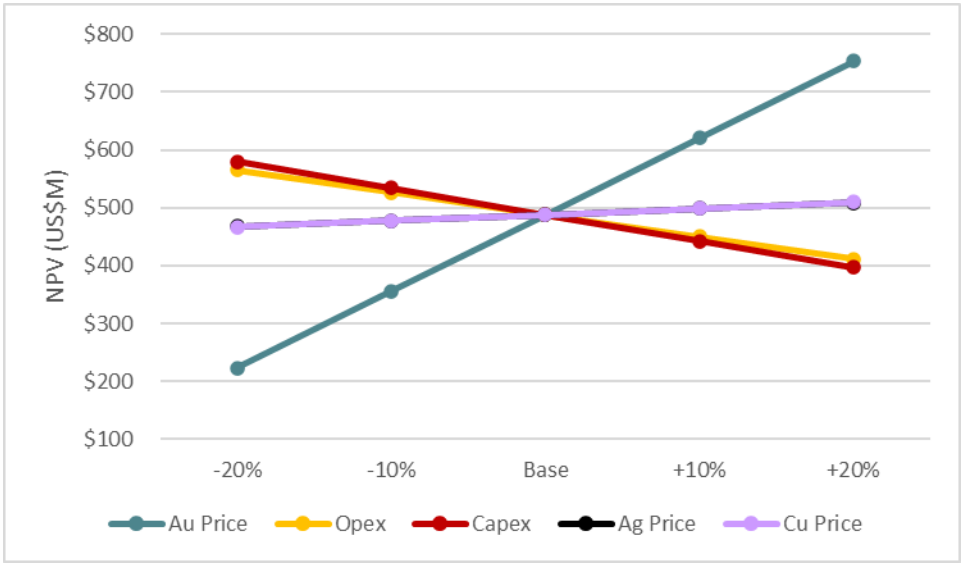
Ag Price	Units	-20%	-10%	Base	+10%	+20%
NPV @ 5.0%	US$ M	467	478	488	499	509
IRR	%	17.6	17.9	18.1	18.3	18.6
Payback	years	3.4	3.4	3.4	3.3	3.3

Cu Price	Units	-20%	-10%	Base	+10%	+20%
NPV @ 5.0%	US$ M	467	478	488	499	510
IRR	%	17.6	17.8	18.1	18.3	18.6
Payback	years	3.4	3.4	3.4	3.3	3.3

Opex	Units	-20%	-10%	Base	+10%	+20%
NPV @ 5.0%	US$ M	565	527	488	450	412
IRR	%	19.8	18.9	18.1	17.2	16.3
Payback	years	3.1	3.2	3.4	3.5	3.6

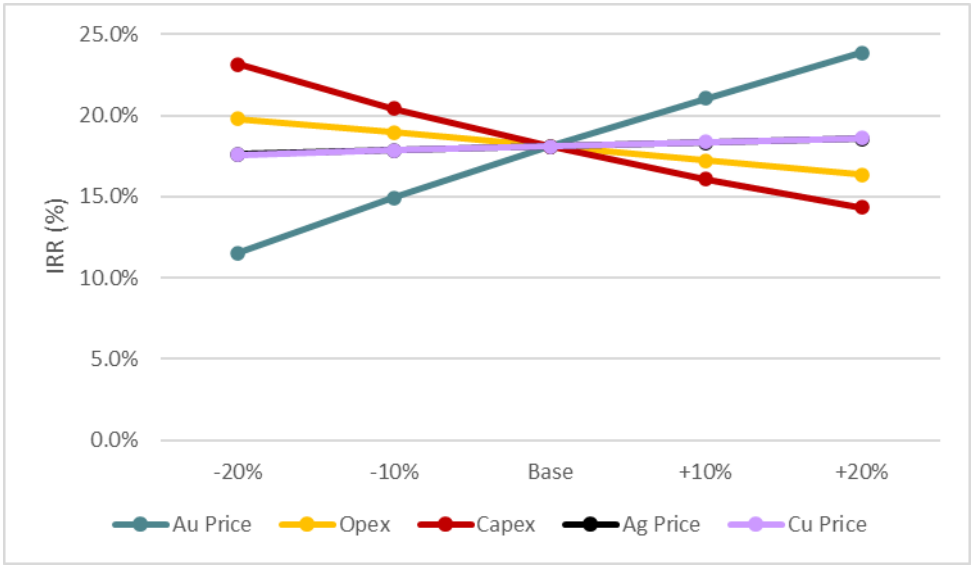
Capex	Units	-20%	-10%	Base	+10%	+20%
NPV @ 5.0%	US$ M	579	534	488	443	397
IRR	%	23.1	20.4	18.1	16.1	14.3
Payback	years	2.8	3.0	3.4	3.7	4.0

Figure 22.4 – Sensitivity of After-Tax NPV to Metal Prices, Capex and Opex



Source: DRA, 2025

Figure 22.5 – Sensitivity of After-Tax IRR to Metal Prices, Capex and Opex



Source: DRA, 2025

Table 22.6 provides the sensitivity of the project economics to a broader range of gold price levels, which encompasses the spot price as of the Effective Date of this Report. Note that silver and copper prices are maintained constant in assessing this sensitivity.

Table 22.6 – Sensitivity to a Broader Range of Gold Price (US$/oz)

Parameter	Units	US$ 1,500	US$ 1,700	US$ 1,900	US$ 2,300	US$ 2,500	US$ 3,500
NPV after-tax @ 5%	US$M	210	349	488	767	906	1,603
IRR after-tax	%	11.2	14.8	18.1	24.1	26.9	39.1
Payback Period	years	4.6	3.9	3.4	2.7	2.5	1.9

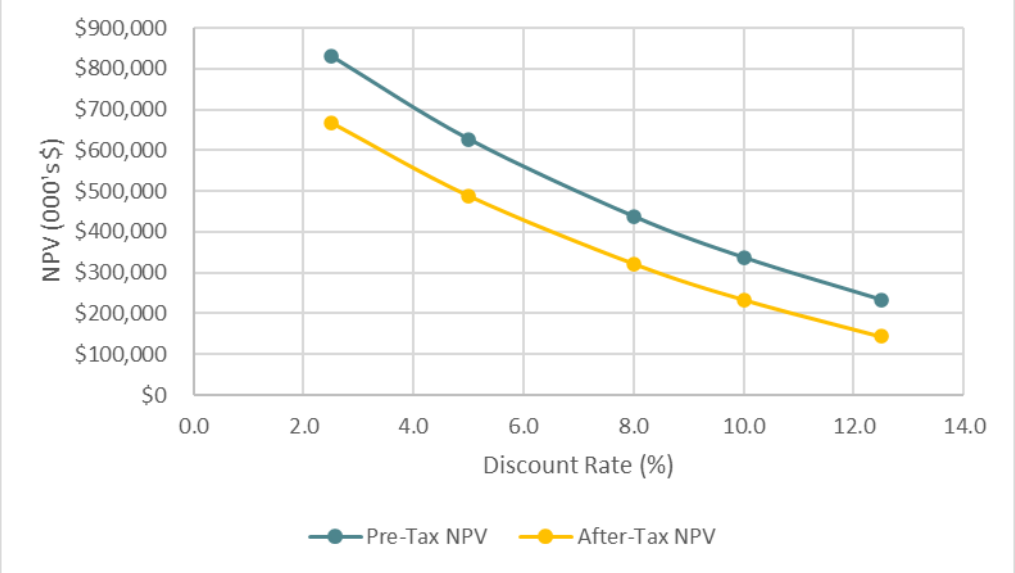
22.9.1 Discount Rate Sensitivity

The sensitivity of the pre-tax and after-tax Project NPV to various discount rates is presented in Table 22.7 and Figure 22.6.

Table 22.7 – Sensitivity of Pre-Tax and After Tax NPV to Discount Rate

Discount Rate	Pre-Tax NPV (US$ M)	After-Tax NPV (US$ M)
2.5%	831	667
5.0%	627	488
8.0%	438	322
10.0%	337	234
12.5%	233	144

Figure 22.6 – Variations in Pre-Tax and After Tax NPV with Discount Rate



Source: DRA, 2025

23 ADJACENT PROPERTIES

There are no adjacent properties as defined by NI 43-101.

24 OTHER RELEVANT DATA AND INFORMATION

This section should be read in conjunction with, and is subject to, the section entitled "Post-Effective Date Events" at the beginning of this Technical Report.

24.1 Project Execution Plan

24.1.1 INTRODUCTION

DPM's main office for the Project will be the DPM Ecuador (DPME) Offices in Quito and Cuenca. The integrated Project Management Team (which comprises DPM CM staff, shared staff from DPME, and field engineers from the engineering firm home office)) will be headed by the Project Director following an Engineering, Procurement and Construction Management (EPCM) strategy, maximising the use of Ecuadorian contractors for all field installations.

The Engineering Team will be established at the main execution centre in Canada, with support engineering in South America for detailed engineering activities and preparation of Spanish language deliverables. This Engineering and Procurement (EP) team will engage a regional South American engineer for this purpose. Specialised engineering consultants will be engaged for geotechnical engineering, water management, paste backfill of the mine, and filtered tails management.

Procurement support will be carried out from Canada for all equipment procurement during detailed engineering to support engineering data needs, and to manage vendor's conformance to Project specifications. The existing DPME contracts administration team will be expanded to coordinate the local contracts and will receive the packages contract management from the EP services provider for field administration. Procurement and transport / delivery support in Ecuador will be managed by the DPME local team.

DPME's Construction Management (CM) team will begin mobilising to support early construction activities, such as the 22 kV powerline, mass earthworks, and mine development at the mine portal area and decline. The construction of the mine portal and decline will be executed by a specialised contractor and will be transitioned to the Owner at a defined interface point before the limit of the mine resource. The DPME mining team will self-perform development of the orebody access for the pre-production phases.

Commissioning activities will be performed by an integrated team of supplier, engineering, construction and operations personnel led by DPM's commissioning manager who will ensure a smooth transfer of facilities from construction to operations. The construction team will be responsible for mechanical completion and pre-commissioning (wet commissioning) activities; operations will witness and follow the process to get familiar with the installations. With the introduction of ore and reagents (hot commissioning), the operations team will assume plant

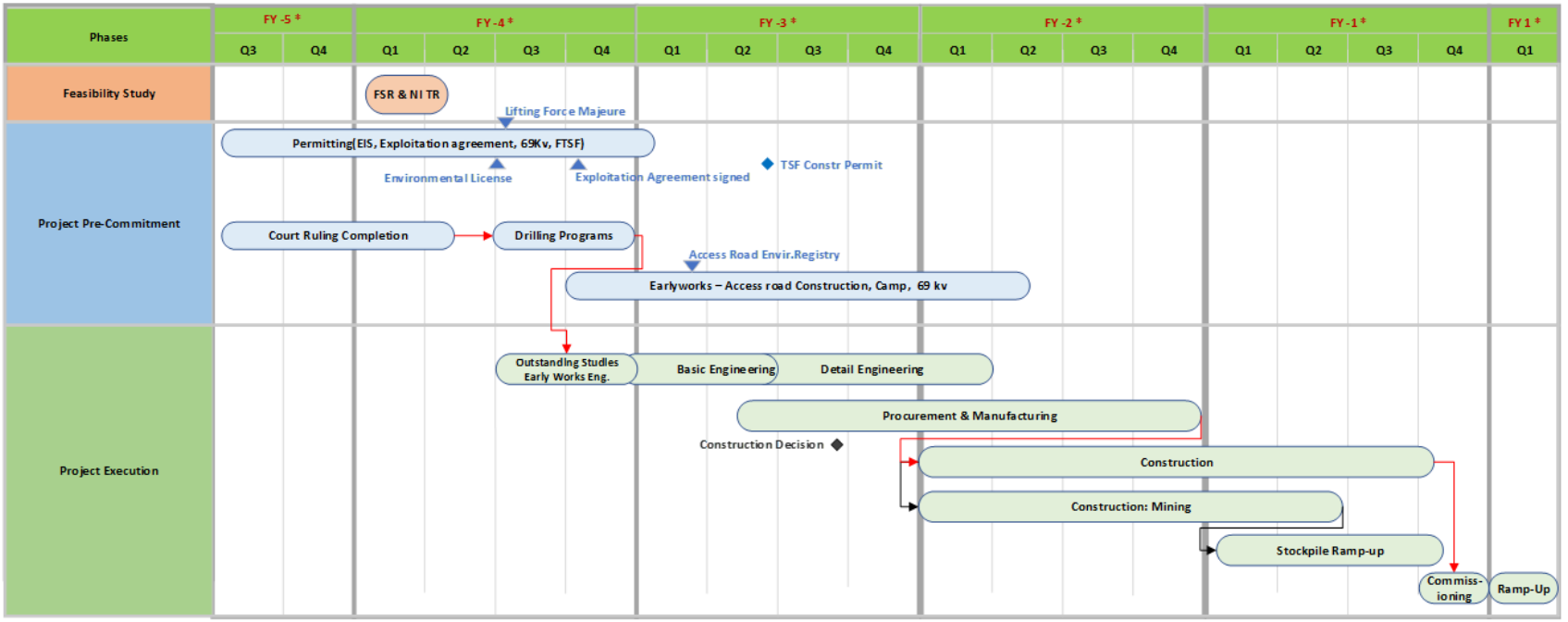
responsibility with a reduced construction team for the final punch list items and small operating improvements.

During the pre-construction phase, a 500-person camp will be built, in close proximity to the mine in order to accommodate construction personnel during the early phases of construction. The camp will later be used to host operations personnel as needed.

24.1.2 Project Schedule and Key Milestones

Project execution is divided into three (3) distinct phases: 1) Early Works, 2) Pre-Construction, and 3) Construction, each with its own set of objectives and functions, which are outlined in the following subsections. The major activities in the project are illustrated in Figure 24.1.

Figure 24.1 – Project Schedule



** Note: The schedule presented above is intended solely to illustrate the general sequence and estimated durations of key Project activities. Estimated finish date of the project and activities are dependent on when, or if, the environmental license will be re-granted. See section entitled "Post-Effective Date Events" at the beginning of this Report.*

Source: DRA, 2025

24.1.2.1 *Construction Decision*

A major milestone for the Project is the construction decision. Once the FTSF Construction Permit is granted, DPM will review and decide whether to proceed with the full construction of the Project.

To achieve the planned Project completion date, DPM will initiate selected aspects of the Project and construction, in advance of the construction decision. Once the Exploitation agreement is signed, Early Works construction and Basic Engineering will begin.

24.1.2.2 *Early Works Phase*

The Early Works construction phase will include upgrading the current exploration camp facility to a target capacity of 75 workers and supervisors. Later when permits are obtained the exploration camp will be replaced with a 500-bed permanent camp. This new facility will provide space for future facilities should mine operations growth require a future expansion.

Early Works also includes the execution of the 22 kV power transmission line, and design of 69 kV line. Reputable local consultants, engineers and contractors will be engaged to complete detailed engineering and Quality Assurance/Quality Control (QA/QC) work on critical and specialised components such as the powerline engineering and easements process.

The main access road is not included as part of the Loma Larga Project but would be upgraded in parallel with the Early Works.

24.1.2.3 *Pre-Construction Phase*

Following the Early Works phase there will be a Pre-Construction phase, which includes required water management infrastructure for site development, such as, contact water management channels and ponds, non-contact water collection and sediment control earthworks. It will also include the minimum required mining installations for mine development such as, mine dry and administration offices, fuel-bay, maintenance shelters, and temporary power distribution. Pre-Construction also includes major earthworks (mass excavation) for the mine portal area, process plant terrace, initial site roads, waste rock storage area, water treatment facilities and installation of the permanent camp of 500 beds and upgrades to the existing facilities.

This phase of construction, including priority site work, will be subject to the receipt of environmental licenses.

International third-party consultants and engineers will be engaged to complete detailed engineering and quality functions on specialised components such as tailings management facilities, paste facilities and distribution, water treatment facilities, and to support environmental permits. The Pre-Construction phase will overlap with the final weeks of the Early Works phase.

24.1.2.4 Construction Phase

The Construction phase entails installation/erection of industrial and processing facilities, underground mine development, process equipment, communications systems, ventilation and services, ore conveying, reclaim, crushing, grinding, flotation system, and other equipment. Processing facilities and supporting utilities are also included. The tails filtration and paste backfill plant facility will follow installation of the process plant. This Phase will overlap with the Pre-Construction phase, including the mechanical completion and wet commissioning.

The milestones for commencement of the critical activities are as follows:

- Detailed engineering and commitment for equipment procurement.
- Receipt of critical permits for supporting utilities and construction release.

24.1.3 PROJECT MANAGEMENT

The execution plan is based on an Engineering-Procurement services contract with the Owner´s Construction Management Team leading the management of Ecuadorian contractors in the execution and final turnover of the Project. The Integrated Project Management Team (IPMT) will minimise Project administration and management roles required to oversee and direct the engineering, procurement, and construction. It will consist of experienced managers from the Owner´s CM team, the Engineer, other consultants, and the Owner´s operations group. The organisation chart is presented in Figure 24.2.

Key Project roles include:

Project Director:

- Assumes overall accountability for preparation and execution of all general site services departments while supporting the construction, mining, and process departments throughout the Project construction and into operations.
- Has direct reports that are responsible for delivering the Project within the approved budget and schedule with the quality standards established and in a safe and socially responsible manner.

Construction Manager:

- Assumes overall responsibility for construction of the mining, processing, and all other support facilities, including development of a mechanical completion and commissioning plan, field engineering (some field engineers will come from the main engineering office but will report to the Construction Manager), and site quality control from warehousing to final installation.

Engineering Manager:

- Assumes overall responsibility for execution of all engineering deliverables, including review / approval of all designs related to infrastructure, mine, process installations and buildings. Also responsible for the work of third-party engineering consultants, engineering office – field engineering interactions and decision making, fabrication QA/QC inspectors and as-built drawings through to Project completion.

Project Control Manager:

- Assumes responsibility for overall Project planning, accounting, and controls (cost, schedule, progress, and reports).

Procurement and Contract Manager:

- Assumes overall responsibility for in-country procurement, expediting, logistics, receiving and warehousing of supplies and equipment during the Construction phase and later transitioning into operations as Supply Chain Manager. The Procurement Manager is responsible for capital procurement, expediting (documentation and fabrication), quality surveillance and logistics. The role will continue until start-up.

Operational Readiness (OR) and Commissioning Manager:

- Will join in early stages of Project, and will be responsible for the OR plan and its implementation during Project execution, and OR and commissioning team management.

Mine Manager:

- Assumes overall responsibility for preparation and execution of the pre-production scope, approval of all designs related to mine services and mine non-process buildings, and development, implementation, and execution of the mine technical services (geology and engineering), mine operations and mine maintenance.

Process Manager:

- Assumes overall responsibility for preparation and execution of OR activities related to the processing facilities and personnel management, pre-production scope and approval of all designs related to process buildings including laboratory facilities, metallurgy and execution of the process operations, process maintenance and power.

General and Administration Services

- These positions will not be filled by Loma Larga IPTM. The Finance, Human Resources (HR), and Social departments of DPME will provide supporting services as required.

Site and Camp Manager:

- Assumes overall responsibility for all camp services, transportation services, catering services for employees onsite and offsite; as well as care and maintenance of the service facilities once construction is completed.

Health and Safety (H&S) Manager:

- Assumes overall responsibility for implementation and execution of the H&S programs and execution plan during construction, commissioning, and transition to operations. The H&S Manager is also responsible for the medical clinic onsite and offsite medical services.

24.1.4 ENGINEERING

The Engineering Plan outlines the scope, organisation, and execution of the engineering activities for the Project through detailed engineering for the Project.

The assumption for this plan development is that geotechnical campaign results will not imply any significant changes to current designs. If any findings require a design review, this Project Execution Plan will be revised accordingly.

Engineering will prepare all technical documentation required for the design, procurement, construction, pre-operational verification, mechanical completion, and commissioning of the Project.

Engineering will carry out the required work for the identified phases (Basic Engineering and Detailed Engineering). In addition, they will provide certified construction drawings in accordance with industry best practices and as determined by:

- The Project master schedule.
- The construction and installation strategy.
- DPM Ecuador S.A. (Health, Safety and Security) requirements.
- Project resources.
- Contractor and vendor engineering resources and capabilities.

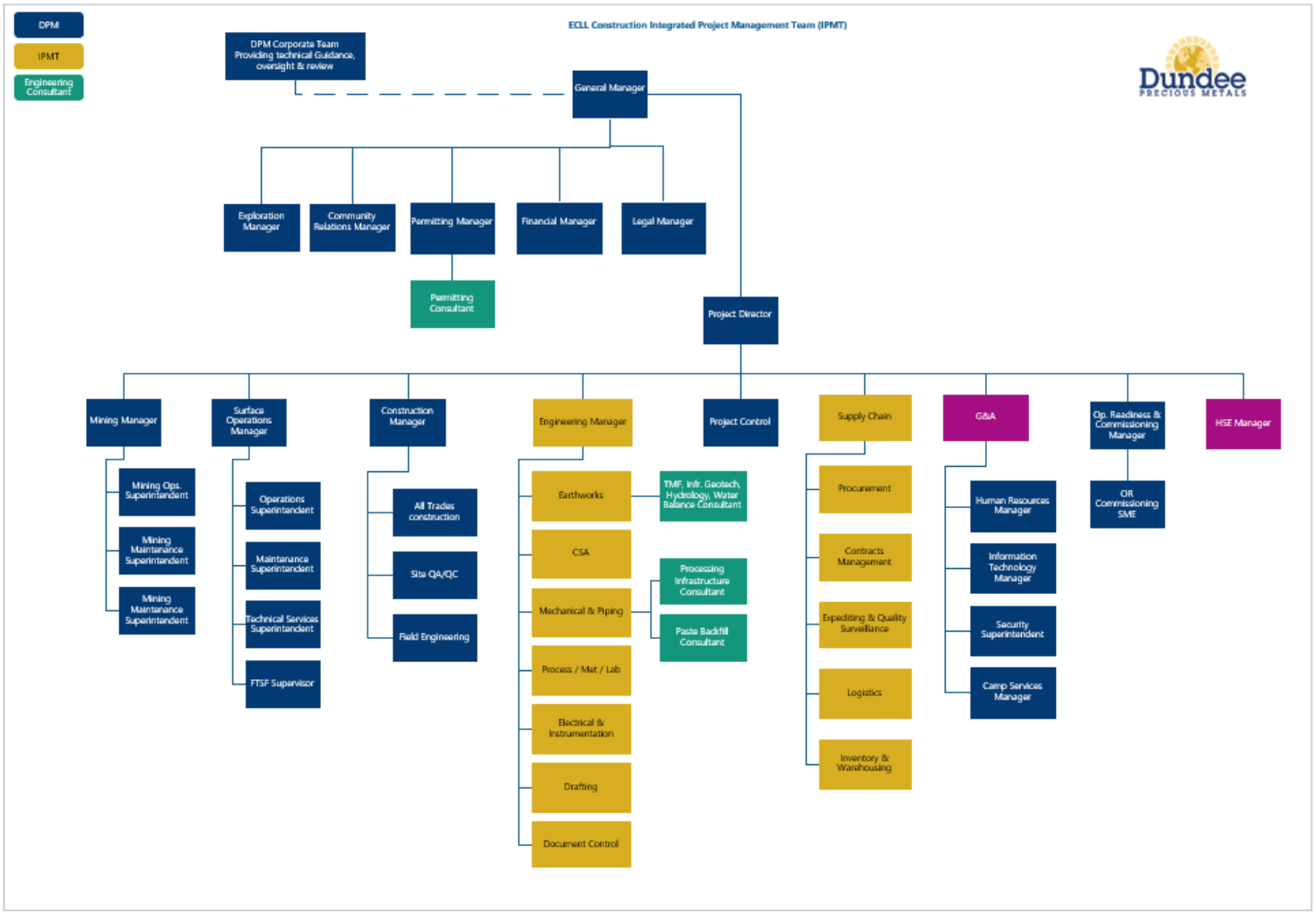
Engineering will specify and monitor the quality of engineering documents supplied by other consultants or suppliers required for plant construction and operation.

As engineering trade-offs and optimisations are completed, summary reports will be uploaded to the Document Management System to record the evolution of the engineering design.

Design criteria are based on the best practices in the mining industry, Canadian and Ecuadorian National Standards, and DPME corporate standards.

Engineering will be led by the Engineering Manager under the supervision of the Project Director.

Figure 24.2 – Project Management Organisation Chart



Source: DPM, 2025

24.1.5 PROCUREMENT AND CONTRACTS

The IPMT will manage Procurement and Contracts activities for the Project as an extension of the EP contractor's services, based in Canada and continuing by handoff to the Owner´s IPMT at the Project site.

Procurement and Contracts will be responsible for all capital procurement and construction packages. Contract Agreement will be negotiated by procurement; however, orders will be issued by DPME. Agreements may consist of Scope of Work, engineering specifications, drawings and data sheet as required by the package.

Procurement and contracts will be led by the Procurement and Contracts Manager who will report to the Project Director. The Procurement and Contracts Manager will advance the major purchase orders and contracts awarded by DPM through the EP services provider in Canada. Additional in-country procurement and contracting as required will be managed by the Owner´s IPMT.

24.1.6 CONSTRUCTION EXECUTION

During the Pre-Construction phase, DPM will implement their local communities training program. This training program targets unskilled local workers, and trains them in safety, basic craft skills, and prepares them for entry level jobs at an industrial level. Contractors will be provided with a listing of local trainees who have successfully graduated from this program and are available for hire. Mass earthworks, water management structures and the FTSF initial phase will be performed by experienced Ecuadorian contractors. The portal opening and ramp will be developed by a specialised Ecuadorian contractor. Subsequent process, building, and infrastructure work will also be performed by experienced Ecuadorian contractors where qualifications of experience, safety, financial strength, and available craft can be validated.

The execution plan is based on the Owner´s CM Team leading the management of Ecuadorian contractors in the execution and final turnover of the Project. To optimise the overall Project implementation schedule, engineering, procurement, and construction are performed and managed by the IPMT. All aspects of engineering and construction are studied to maximise efficiency with a minimised footprint, and reduced costs. This approach also covers health and engineered safety elements not only to build but also to operate and maintain. There is also a special focus on reducing re-handling of material to the minimum and reducing the use of temporary facilities by installing permanent infrastructure as early as possible, during the Pre-construction phase.

To support site development during the Construction phase, the main site access road will be upgraded, as part of a separate project. To support productivity while reducing the amount of time spent working at high altitude, modularised equipment will be procured wherever possible.

The construction team will incorporate experienced expatriate construction management and area managers, Ecuadorian supervisors, and coordinators for all trades, working with national construction contractors with skilled Ecuadorian tradesmen and labourers. Skills improvement and zero-harm safety will be emphasised throughout the construction period, and where possible, the best available candidates can apply to the operations and maintenance team at the conclusion of the construction period. This results in a large contribution of national workers to the overall construction effort, with significant skills development and opportunities for continued employment in the operating phase, or in the industrial or mining industries.

24.1.7 COMMISSIONING PLAN

The Project will be commissioned progressively by systems and subsystems using DPM commissioning procedures. A Commissioning Manager, reporting to the Project Director, will direct the day-to-day direction and activity of the commissioning and start up team.

The standard steps are typically:

- Construction activities carried out by the construction team include construction (mechanical) completion and pre-commissioning checks, proving the equipment operates correctly and is ready for wet commissioning. Systems with remaining punch list items will not be continued to wet commissioning without written acceptance by the Commissioning Manager.
- Cold/wet commissioning involves the functional testing of equipment by the integrated operations/construction commissioning team. Equipment is considered live, and all necessary controls are in place for Lockout Tagout (LOTO). Safe work practices will be evolved from construction procedures allowing for safe access and control of activities. The use of safe work permits will be strictly enforced at all times in addition to LOTO procedures. On completion of the wet commissioning, sub-area acceptance certificates will be issued resulting in the effective transfer of care custody and control to the operations team for the corresponding sub-systems.
- Hot commissioning (first feed) is carried out by the Owner's operations team that is supported by the Owner's commissioning team. The construction team and home office engineering team will also support commissioning efforts as needed to ensure a smooth start up process.
- The punch list process will be maintained throughout the commissioning cycle to make sure any defects that are discovered in hot commissioning are properly addressed by the responsible party.
- Commissioning and turnover activities present significant safety risks and all necessary LOTO procedures and acceptance certificates will be in place, properly identifying the responsible party for Transfer of Care Custody and Control (TCCC) of the equipment. This is important as there will be times when construction, commissioning and operations teams are working in common areas. Some subsystems may be in TCCC of the commissioning or

operations team while adjacent sub-systems may still be under the TCCC of the construction contractor.

- Other groups providing support during the commissioning and ramp up phases includes home office engineering and original equipment manufacturers (OEMs).

24.2 Operational Readiness

The scope of work of Operational Readiness (OR) will be to prepare people, systems, and procedures, such that all items related to process, maintenance and metallurgy are understood and fully functional in preparation for the operating phase. The success of the Project is ultimately measured by the success of the start-up and ramp up to commercial production and design rates.

The execution of the OR plan will be the responsibility of the OR Manager (in many cases it is the Process Manager) along with key process and maintenance superintendents that will be hired early in the Project. OR starts during the FS or Pre-FS phase and the Operational Readiness Plan (ORP) is finalised at the end of the FS while OR activities continue all the way through until the operation reaches sustainable production, at which point OR is completed. Some operational personnel will be hired at the onset of detailed engineering while others will follow closely behind to continue working on and developing details of the ORP.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resources

Specific conclusions for the geology and resource database related items of the Loma Larga Project are summarised below.

- Loma Larga is a high sulphidation polymetallic epithermal deposit containing significant values of gold, silver, and copper.
- The Loma Larga deposit is a stratigraphically controlled, flat lying, gently westward-dipping, north-south striking, cigar-shaped body. It also dips slightly to the north, such that the mineralised zone is closer to surface at the south end.
- The results of the Quality Control (QC) samples, together with the Quality Assurance/Quality Control (QA/QC) procedures implemented at Loma Larga, provide adequate confidence in the data collection and processing, and the assay data is suitable for Mineral Resource estimation.
- Understanding of the Project geology and mineralisation, together with the deposit type, is sufficiently well established to support Mineral Resource and Mineral Reserve estimation.

With respect to the Mineral Resources, the QP's conclusions are summarised below.

- Block grade interpolation was carried out using Ordinary Kriging (OK) for gold, silver, copper, and density. A 2.0 g/t Au wireframe model (High-Grade Zone) and a 0.8 g/t Au wireframe model (Low-Grade Zone) were used to constrain the grade and density interpolations.
- An NSR cut-off value of US$65/t is appropriate for reporting current Mineral Resources for the Project, which is based on the potential production scenario.
- Mineral Resources are estimated in four (4) zones: the High-Grade Main Zone, which is classified as Measured and Indicated Mineral Resources, the High-Grade Upper Zone, which is classified as Inferred Mineral Resources, and the Low-Grade Main and Lower Zones, which are classified as Indicated and Inferred Mineral Resources.
- Reporting within underground resource reporting shapes based on an NSR cut-off value of US$65/t, satisfying the minimum mining size and continuity criteria, Measured Mineral Resources exclusive of Mineral Reserves are estimated to be 0.3 Mt grading 4.80 g/t Au, 20.8 g/t Ag, and 0.23% Cu. Indicated Mineral Resources are estimated to be 5.7 Mt grading 2.79 g/t Au, 21.9 g/t Ag, and 0.16% Cu. Inferred Mineral Resources are estimated to be 2.2 Mt grading 2.54 g/t Au, 37.0 g/t Ag, and 0.19% Cu.
- Definitions for resource categories used in this report are consistent with those defined by CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) and incorporated by reference in NI 43 101.

- A sulphur block model was added to the Loma Larga Project to support metallurgical testwork. The local block grades are not as well supported as the payable metal block grades of the current Mineral Resource Estimate.

25.2 Mineral Processing and Metallurgical Testing

A significant amount of testwork was conducted to develop a robust and fit for purpose flowsheet for the development of the Loma Larga process plant design. The merits of sequential and bulk flotation flowsheets were examined during the program and analysed. The testwork completed supported the basis of the 2020 Feasibility Study, and 2023 FSUO.

The metallurgical programs concluded that a sequential flotation flowsheet for the recovery of separate gold bearing copper and pyrite concentrates is the preferred processing route. From the testwork, grade and recovery relationships for the copper concentrate and an understanding of the gold-pyrite concentrate recoveries was determined.

In 2022 a metallurgical program was conducted using low-grade composites from available drillhole samples completed in 2017.

The 2022 program is considered confirmatory in nature with respect to the confirmation of the flowsheet selected and assessment of process opportunities. Therefore, these tests were considered for reference purposes only and as support to the 2017-2018 program. Due to the limited quantity of the ore sample used for the 2022 metallurgical program, the 2017-2018 metallurgical program results were still considered as the basis for the 2022 Update (and thus this update also).

The objectives of the 2022 test program were to review the results from of the 2017 testwork; assess other opportunities with regards to primary grind size impact on ore flotation response, rougher concentrate regrind requirements, evaluate the number of cleaning stages required for gold-copper, and pyrite flotation circuits, assess at a high level the use of Woodgrove, Jameson, and Hydrofloat technologies, evaluate tailings settling and filtration properties, and generate a sample of tailings for Paterson & Cooke tailings testing.

The objectives of the program were met and opened several process opportunities with regards to the coarser primary grind size (125 µm P_{80}), number of the flotation cleaning stages required, and applicability of the advanced flotation technologies. Use of the Woodgrove, and Jameson flotation cell technologies were found promising.

Considering the attractive Capex for the Jameson cell, in July-August 2023 an LOM composite sample was tested at BML. The sample was compiled from the historical drillcore stored on site. The dilution test protocols were applied considering the available quantity of ore. Rougher and cleaner conditions were investigated and compared to the conventional flotation results. It was found that use of the Jameson technology benefits the rougher circuit with better recovery and allows to

produce a clean concentrate due to the froth washing. Cleaner tests suggested the same performance as it was observed for the conventional flotation. Considering the Capex advantages determined during the process trade-off study conducted in April 2023, the use of Jameson technology is recommended for both copper and pyrite flotation. Some optimisation of the rougher and cleaner conditions as well as variability tests recommended as soon as sufficient fresh sample becomes available.

25.3 Mineral Reserve Estimate

The Mineral Reserve Estimate for Loma Larga consists of selected portions of the indicated and measured resource that are above a US $75.00/tonne NSR cut-off value. This value was applied when generating the shape of stopes that will be extracted using long hole open stoping methods. The Mineral Reserves for Loma Larga are estimated at 12.59 Mt of recoverable and diluted ore grading 4.70 g/t Au, 28.56 g/t Ag, and 0.29% Cu. The Mineral Reserves are comprised of 22% in proven category (2.72 Mt grading 6.83 g/t Au, 33.23 g/t Ag and 0.43% Cu) and 78% in probable category (9.86 Mt grading 4.12 g/t Au, 27.35 g/t Ag and 0.25% Cu). Mineral Reserves are inclusive of dilution and ore loss. The latest NSR value was added to the DPM-provided Block model using Datamine software by GMD and used for further work in Deswik.

The mineral reserves are based on the mining schedule presented in Section 16. The mineral reserves are the stopes and development tonnage (including material above the cut-off and internal low-grade dilution) to which dilution and recovery factors explained below are added.

Details of the Mineral Reserves Estimate are given in Table 25.1.

Table 25.1 – Loma Larga Mineral Reserves Estimate Effective as of September 29, 2023 (GMD, 2023)

Ore Category	Tonne (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)	Cu Grade (%)	Contained Cu (Mlb)
Proven	2.72	6.83	0.60	33.23	2.91	0.43	25.58
Probable	9.86	4.12	1.31	27.35	8.67	0.25	54.39
Proven and Probable	**12.59**	**4.70**	**1.90**	**28.56**	**11.58**	**0.29**	**79.96**

1. Mineral Reserve is estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (Nov 29, 2019) and CIM Definition Standards for Mineral Resources & Mineral Reserves, May 19, 2014.
1. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the crusher.
2. Mineral Reserves account for unplanned mining dilution and ore loss that varies by orebody dimension and geotechnical properties per mining block area.
3. Mineral Reserves include long hole stopes and development in ore.
4. Mineral Reserves are reported at an NSR cut-off value of US$75/t for long hole stopes and US$55/t for development in ore.
5. Mineral Reserves are estimated using average gold price of US$1,500 per ounce, silver price of US$17.00 per ounce, and copper price of US$3.25 per pound.
6. Average bulk density is 2.71 t/m^3.
7. Numbers may not sum precisely due to rounding.
8. Metallurgical recoveries are not included in the contained values calculation.
9. The mine design and schedule for the Mineral Reserve Estimation was completed under the supervision of Dr. Lewis Meyer, CEng, QMR, FIMMM Associate Principal Engineer at Global Mine Design Ltd., who is a QP as defined under NI 43-101.
10. Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

The underground mine design and Mineral Reserve Estimate have been coordinated by GMD and prepared by the QP using industry standard/proven methods and best mining practices to a level appropriate for a Feasibility Study. The Mineral Reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources with applicable modifying factors and therefore exclude any Inferred Mineral Resources. The latter are classified as waste for reporting purposes.

All mining related costs have been estimated from first principal, considering the latest economic factors and inputs. NSR cut-off was developed by DPM and used to update the block model.

The following items are factors that may affect the Mineral Reserves:

- Geological interpretation and Mineral Resource block modelling.
- Cut-off value estimations.
- Commodity prices and market conditions assumptions.
- Operating costs, sustainable costs for development assumptions.
- Geotechnical constraints, dilution, mining recovery factors and hydrogeological assumptions.
- Metallurgical process recoveries.

- Environmental permitting.

Based on this assessment, the Mineral Reserve for the Loma Larga Project will support a 13-year mine life, including a pre-production period of two (2) years, and a production period of 11 years, at a planned production rate of 1.2 Mtpa during the steady-state production phase.

Apart from the revocation of the Environmental License subsequent to the effective date of this Report, the QP is not aware of any other permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Reserve Estimate. The QP understands that DPM is considering all its options to rectify the revocation of the environmental license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project, including the Mineral Reserve Estimate. See “Post-Effective Date Events” at the beginning of this Report.

25.4 Mining Methods

The selected long hole open stoping mining method is well suited for the orebody geometry and is known for its flexibility in stope sequencing. The current mine design maximises this flexibility by incorporating a mixture of transverse and longitudinal stope accesses in order to target high-grade ore zones as soon as possible, while maintaining the overall daily production targets. The current mine design has sought to cover as many options as possible with respect to equipment, production and waste management to ensure flexibility throughout the proposed mine life. The mine plan and expenditure schedule presented in this Report are reasonable. The mine plan has been developed based on sound engineering practices including geotechnical engineering. The mine schedule, including production ramp-up uses realistic sequencing and rates that are aligned with similar operations and mining jurisdictions.

The mine development schedule is predicated on the premise of timely award of the mining contract for decline development and the raiseboring of the first ventilation shaft, the procurement of mobile equipment by the Owner, and the recruitment of a suitable workforce.

The drill and blast designs proposed for the Project are robust and well-suited to the mine design. The stoping performance is expected to be good, with significant scope to optimise drill & blast designs throughout the mine life. These designs can be further improved by the good quality and extensive geotechnical database, as well as the provisions for geotechnical and survey monitoring equipment.

25.5 Recovery Methods

The Loma Larga process plant flowsheet and design are robust and allow for the treatment of the various ore types that will be encountered over the Project LOM. It is also considered to be conventional and fit for purpose of producing separate saleable gold-copper and pyrite concentrates.

Plant flotation circuit will use Jameson flotation cell technology which provides the benefit of reduced Capex through the small footprint and does not require flotation blowers for air supply.

The design considers one stage of copper cleaner flotation and two stages of pyrite cleaner flotation. The design of the Jameson cells selected for the duty accommodates a production increase from 3,000 to 3,400 t/d without the addition of extra cleaning stage.

25.6 Project Infrastructure

The Project infrastructure designs are developed and the infrastructure is of suitable design for the site and size of operations. Surface infrastructure such as camp, administrative buildings, and water treatment plant will be required.

Geotechnical and founding condition designs are assumed as competent, bearing conditions with no soil improvement required, as no additional geotechnical investigations have been conducted since the previous FS.

Tailings production and deposition methods were evaluated, and filtered tailings was the technology chosen. The design of the paste plant and the filter tailings storage facility was progressed and verified that adequate surface area is available for the total storage volume.

Off-site infrastructure is suitable to support the Project. Local infrastructure and authorities may require additional support.

25.7 Environmental Studies, Permitting and Social or Community Impact

- No known environmental issues were identified by the QP from the information provided by DPM for review. In the QP's opinion, there are no environmental issues that could materially impact the ability to extract the Mineral Resources and Mineral Reserves based on the review of the available documentation.
- DPM maintains an up-to-date record of the legal permits obtained to date and future permits to be obtained according to the applicable legislation and regulatory requirements. The environmental permits already approved and future environmental permits to be obtained are listed in Table 20.1.
- The Environmental License resolution for exploitation activities for the Project was issued by MAATE on June 23, 2025 and subsequently revoked by the Government of Ecuador on

October 3, 2025. The viability of the Project is dependent on the granting of the Environmental license.

- DPM developed an environmental permitting schedule listing the main permitting processes to be completed ahead of the construction stage and the anticipated timeline.
- DPM is preparing to restart negotiations with the Ecuadorian Government for the Project regarding the Mining Exploitation Agreement between DPM and the Government of Ecuador.
- Geochemical characterisation studies have been developed for the Project and a water management system protective of water quality has been engineered according to the results of the geochemical studies and predictions from water quality numerical modelling.
- The expansion of the public access road from San Gerardo requires land acquisition by DPM, which may involve the need for involuntary resettlement of some people living along the road. Although it is anticipated that potential resettlement would involve few people, it must be properly managed by DPM with close support from the social management team through a process that is transparent, equitable, and effective.
- DPM has adopted several communication mechanisms to inform the public about the Project and community development projects. DPM has implemented a consultation and grievance mechanism for the communities to provide feedback or file requests, questions, concerns, and/or complaints regarding the Project.
- DPM is working on activities to promote development of local suppliers, grow relationships with current and potential local suppliers, generate a supplier database, and promote local ventures. The Company has been implementing its social action and development plan by supporting activities associated with economic production activities, education, health, infrastructure, culture, and recreation.
- According to information provided by DPM, the Escaleras community is recognised as Indigenous by the Ecuadorian government.
- A conceptual Mine Closure Plan was developed by DPM as part of the EIA. Two (2) years prior to the planned completion of the Project (i.e. the operating phase), DPM must submit the detailed Mine Closure Plan to the environmental authority for approval.

25.8 Capital and Operating Cost Estimate

The capital and operating costs were estimated using methods described in this Report, which are typical of FS-level cost estimation exercises.

25.9 Economic Analysis

Based on the available information, the Project has an after-tax NPV of US$488 M at a discount rate of 5% with an after-tax IRR of 18.1% and Payback Period of 3.4 years from the start of operations.

The sensitivity analysis indicates that the Project economics are most sensitive to the gold price. Significant sensitivity is also observed to changes in the estimates of Capex and Opex. Even with a gold price 20% below the US$1,900/oz base case, the Project maintains a positive NPV.

25.10 Risks[1]

This section should be read in conjunction with, and is subject to, the section entitled "Post-Effective Date Events" at the beginning of this Technical Report.

During the Technical Report phase, the process engineering risk program was advanced and other engineering and operations risk reviews were conducted.

The most significant risks identified for the execution phase are captured in the risk register. For risks ranked as high or in the extreme categories, an action plan was established, and these plans will be reviewed quarterly during risk management sessions in the execution phase of the Project.

Other significant ("critical") Project risks identified are presented in Table 25.2.

Risk management activities will continue and address project, business and event risks through progressive phases of the Project (detailed engineering, construction, commissioning and operations) in accordance with Owner and EPCM contractor's procedures.

[1] **Change Post-Effective Date:**
The QP understands that DPM is considering all its options to rectify the revocation of the environmental license, including through active engagement with the government and stakeholders as well as evaluation of all legal avenues. The uncertainty with respect to obtaining the required environmental license remains a high risk to the Project. See "Post-Effective Date Events" at the beginning of this Report.

Table 25.2 – Risk Register – Critical Risks

DPM Risk #	Description	Impact	Inherent Risk Rating (unmitigated)	Mitigation/Controls:	Residual Risk Rating (after controls or mitigation)
12	Environmental regularisation process extended based on the conceptual road design.	1. Delay in the start of the Project. 2. Delay in the environmental regularisation process. 3. Delay in the start of the Cadastral Study for the acquisition of land. 4. Unplanned financial impact.	25 - Critical	1. Validation of the bidding process for a new consultant for the road design, 2. Signing of the cooperation agreement. 3. Awarding of the consultant for the road's conceptual design.	16 - High
18	Delay in obtaining environmental permits (environmental license)	1. Community disbelief in the development of Loma Larga. 2. Delegitimisation of the process. 3. Financial impact	20 - Critical	1. Strengthen relationship with Environmental authorities. 2. Restructure and consolidate the DPM team. 3. Approach with divergent groups (groups with a different position than DPM). 4. Open new scenario for development of mining activity at national level.	16 - High
49	Restriction of DPM's rights in the negotiation of the Exploitation Agreement (Taxes, Royalties, Regulations that may affect the contract)	1. Unplanned additional costs. 2. Uncertainty about the feasibility of the Project. 3. Significant delays in the subscription of the Contract. 4. Inefficiency in government management.	20 - Critical	1. Develop a political contingency plan to manage possible scenarios	16 - High
52	Self-generation of electrical energy in the Project (government requests energy self-sufficiency due to energy crisis).	1. Unplanned costs (infrastructure, operating and maintenance). 2. Operational risks. 3. Environmental Impact, additional environmental permits. 4. Security risks in facilities and labour risks. 5. Inclusion of obligations in the exploitation environmental license.	20 - Critical	1. Cost and Benefit Assessment: Conduct a detailed financial analysis to evaluate the initial, operating and maintenance costs versus the long-term benefits. 2. Search for a self-generation energy project (others) to provide energy to the project (Projects).	16 - High
86	Failure to sign the Exploitation Agreement in 2025	1. Failure to sign exploitation contract within planned timeframe. 2. Unplanned financial impact. 3. Delay in proposed objectives for the year 2025. 4. Failure to comply with investment schedule established in IPA.	20 - Critical	1. Approach with Mines authorities. 2. Generation of informal negotiation workshops. 3. Submit letter to MEM requesting the resumption of formal negotiations.	15 - High

25.10.1 Risks Related to Project Economics

The Project economic performance is highly sensitive to the gold price, as demonstrated in the sensitivity analysis. A key risk is the possibility of a significant decline in the price of gold during the life of the Project, which would negatively impact the Project economics. This risk is somewhat mitigated by the fact that the selected base case gold price of US$1,900/oz is significantly below the prevailing spot price of gold.

The current geopolitical uncertainty around trade policies has the potential to have inflationary effects on global markets. Significant cost inflation or cost overruns driven by other factors would negatively impact the Project economics given the sensitivity observed to both Development Capex and Opex.

The structure of the Production Royalty and any exemptions from import duties are subject to negotiations between DPM and the Ecuadorian government. An update of the economic analysis will be required to capture the impact of the outcome of these negotiations on the Project economics.

The precise determination of the import duties payable requires a detailed review of each Capex line item against import tax regulations. Commensurate with the level of engineering definition in the Technical Report, certain assumptions have been made to estimate the amount of import duties payable.

26 RECOMMENDATIONS

26.1 Geology and Mineral Resources

- Ensure that procedures for investigating, correcting, and documenting results of QA/QC non-compliance issues such as biases or failures are followed for future planned drillhole programs.
- Procure reference standards with grades that better reflect the range of gold grades within the Mineral Resource (i.e., 2.0 g/t Au to greater than 30 g/t Au). The QP further recommends that an analytical standard is obtained for silver and another for copper that reflect the average grades expected in the deposit, in order to quantify the accuracy of analyses.
- Complete an external check on the reference materials used on the Loma Larga Project.
- Continue the regular submission of check assays (pulp replicates) to a secondary laboratory.
- Check half core duplicate analyses using core from the existing core library, to ensure that the current practice of quarter-core analysis is accurate.
- Resurvey drillhole collars that deviated more than one metre above or below the topographic surface.
- Plan future drilling to potentially upgrade mineral resources from inferred to indicated for inclusion in the mine plan and mineral reserves.

In advancing the Project, the QP recommends the following with respect to the Loma Larga Mineral Resources:

- Develop an understanding of the work required to support upgrading areas of Indicated Mineral Resources in the High-Grade Main Zone to Measured Mineral Resources.
- Additional drilling in the High-Grade Upper Zone and Low-Grade Main and Lower Zones to upgrade the Inferred Mineral Resources to Indicated.
- Investigate the exploration potential of the following:
  - Isolated, significant, shallow high-grade gold intersections located above the High- and Low-Grade Main Zone wireframes. The geometry and orientation of this mineralisation is not well understood.
  - Isolated, significant, high-grade gold intersections located below the High- and Low-Grade Main Zone wireframes. These intersections may represent feeder zones to the high-gold grades in the High-Grade Main Zone; however, additional drilling is required to test this hypothesis.
  - A sub-horizontal planar zone of high-grade silver in the southern Low-Grade Main Zone that remains open to the south. Many of the highest silver grades are not currently incorporated in the Loma Larga mineralisation wireframes and there is potential to add a silver domain to the Loma Larga deposit and Mineral Resource estimate. The QP

recommends these be wireframed and interpolated into the block model to determine tonnage, grade, and classification category.

DPM is currently preparing to restart drilling operations once it receives regulatory permission to do so, with an initial 23 kilometres of drilling planned. Drilling will focus on testing the targets discussed above, with particular focus on infilling existing mineral resources and extending the mineralisation where the potential exists.

Additionally, as part of the project engineering requirements, geotechnical and hydrogeological drilling is planned to collect information to support the design of the underground and surface infrastructure. Furthermore, a series of dedicated metallurgical twin holes are planned to collect fresh samples for confirmatory metallurgical testwork as well as pilot scale testing and additional variability sampling. A summary table of the proposed drilling program is shown in Table 26.1.

Table 26.1 – Proposed Drilling Program

Purpose	Drillholes	Meterage
Plant Site and Ancillary Facilities		
Geotechnical	43	1,245
Hydrogeological	4	150
Filtered Tailings Storage Facility		
Geotechnical	6	95
Hydrogeological	2	120
Dual Geotech / Hydro	5	85
Condemnation	5	2,000
Loma Larga Orebody		
Metallurgical	10	2,310
Infill / Extension	68	16,600
Hydrogeological	2	400
Camp Area		
Geotechnical	5	75
Total	**150**	**23,080**

DPM has prepared a budget to support the drilling, analytical and metallurgical testwork requirements, which is due to be released once regulatory permission to conduct surface drilling activities has been granted.

26.2 Mineral Processing and Metallurgical Testing

- Representative gold-copper and pyrite concentrates, and flotation tailings samples will need to be produced in sufficient quantities for filtration testwork to be conducted by potential vendors.
- Further confirmatory testwork – comminution and flotation, through the testing of additional composite and variability samples, which could potentially improve process design conditions and produce additional design data for the single SAG mill grinding circuit considered for the concentrator design. Potential upside of additional testwork would focus on lowering capital and / or operating costs and improving recoveries.
- Continue to develop and optimise the metallurgical relationships for gold recovery in the pyrite and gold-copper concentrates.
- For the Jameson technology, some optimisation of the rougher and cleaner conditions as well as, locked cycle tests, variability tests, and pilot tests as per L150 protocol are recommended as soon as sufficient fresh sample is available.
- Ramp up to 3,400 t/d as soon as practicable (de-bottleneck process sub-systems) and validate further increases in plant throughput through de-bottlenecking.
- Evaluate processing lower grade ores to extend the LOM.

26.3 Mineral Reserve Estimate

- Following the recommendations listed for the Mineral Resources, the QP recommends a program of infill drilling be started once the orebody is reached by the ramp to better delineate the limits of the high-grade zones and refine the limits of the orebody.
- Rerun the COV estimates with the new Opex from the latest economic model (iterative process).

26.4 Mining Methods

The results of the Feasibility Study Update described in this Report have provided an update to the mining plan over the previously Feasibility Study (FS) report published in 2020. Further studies and recommendations with respect to the Loma Larga Project are discussed below. There are no known issues under consideration in the studies that would result in a material change and/or negatively affect the Project.

- Optimise haulage routes.
- Further improve initial development to maximise development advance and enable further optimisation of stope sequence.

26.5 Recovery Methods

During the next phase of the Project, it is recommended to update the process mass and water balances with the new testwork data once it becomes available. Any new information with regard to the mining plan, and any other new data should also be incorporated along with up-to-date data provided by the equipment suppliers.

Possible equipment consolidation and bundling from reputable equipment manufacturers should be considered, if economical.

26.6 Project Infrastructure

- Investigate alternative power generation technologies, including but not limited to wind and solar.
- Continue to optimise infrastructure buildings (offices, kitchens, washrooms, etc.) which were optimised for suitability as modularised prefabricated units or pre-engineered structures.
- Perform further geotechnical and survey information to characterise the existing sub superficial ground materials and bedrock properties in the areas of the FTSF, stockpiles, ponds, process plant and surface mine facilities.
- Identify specific effluent water treatment requirements based on the geochemical characterisation and the preliminary assimilative capacity study.
- Complete tailings filtration testing to verify tailings characteristics and additional geochemical, geotechnical, and hydrogeological investigation to support the FTSF design.
- Perform geotechnical investigation in the borrow areas nearby to assess quality and quantity of the available sources to produce construction materials for aggregates, underliner, overliner, riprap, filter materials, drainage gravel and other manufactured materials.
- Earthworks:
  - Utilise excavated materials for engineered fill wherever feasible. Any material that is deemed unsuitable for construction would be stockpiled on-site; which is not currently accounted for in the Feasibility Study results. Due to the limited available geotechnical information at this stage, the expected volume of these materials is difficult to estimate.
  - Once geotechnical data is available in a subsequent Project phase, evaluate stockpile requirement and find suitable location within property boundaries for these materials.
- Topsoil:

  In the next phase of the Project, confirm the amount of topsoil storage required and validate that the current stockpile location is suitable to store this material for future use in the closure phase of the Project.

- Prepare the plans and programs to support the construction, operation, maintenance and closure of the FTSF during the next phase of engineering.
- Prepare documentation and registers to assess and maintain, during the life cycle of the FTSF, compliance with GISTM and local Ecuadorian guidelines, covering governance, record keeping, knowledge base documentation management, etc.
- Validate the local availability and accessibility to rent surface operation equipment during construction and operation.
- Update water balance model to incorporate any changes in the mine plan, detailed FTSF fill plan, including updates to geochemistry of leachates based on results of ongoing kinetic testing of tailings samples and planned leachate testing on soil samples to evaluate any concerns. Soil characterisation will commence once samples are obtained from the field work planned during the drilling campaign.
- Review the possibility of renting a camp instead of constructing a camp.

26.7 Market Studies and Contracts

- DPM will engage the market and assess potential to improve commercial terms of concentrates.
- Per the current quality assumed in the model, the gold-copper concentrate will be deliverable direct to Chinese smelters without blending. Should the expected quality change or should there be changes to the Chinese import rules for gold concentrates, e.g., a decrease in the maximum arsenic content in gold concentrates, the commercial terms should be reviewed.

26.8 Environmental Studies, Permitting and Social or Community Impact

- Continue updating the record of current legal permits and future permits to be obtained. Review and update the environmental permitting schedule regularly to ensure adequate time frames for regulatory processes are captured and potential risks associated with environmental permitting approvals are flagged in a timely manner.
- Review and update regularly the community engagement plan as social risks requiring mitigations change and evolve.

26.9 Economic Analysis

The following recommendations have been identified with regards to the Project:

- Update economic analysis based on the outcome of exploitation contract negotiations between DPM and the Government of Ecuador, including items such as Production Royalty calculations and tax exemptions.

- Conduct a detailed review of the Capex and Opex estimates to develop a detailed assessment of the import duties and VAT payable for the Project, including an assessment of the schedule of required payments.
- Maintain engagement with a tax consultant to ensure that the taxation calculations applied in the analysis are aligned with the latest regulations from the Government of Ecuador.

26.10 Other Relevant Data and Information

The important milestones for commencement of the key activities are as follows:

- Detailed engineering and commitment for equipment procurement: finance availability.
- Construction activities: construction permit availability.

Critical long lead procurement equipment that drives the construction schedule should be prioritised early in Detailed Engineering, and procurement packages for this equipment should also be prioritised. It is recommended that once OEM vendors for these packages are selected, a negotiated early payment be paid upfront for expediting engineering documents and certified vendor drawings which are required for the engineering consultant to finalise IFC drawings.

27 REFERENCES

AACE International Recommended Practice No.47R-11, Rev. Aug 7, 2020 - Cost Estimate Classification System - As Applied in Engineering, Procurement, and Construction for the Mining and Mineral Processing Industries.

Agency of Mining Regulation and Control – ARCOM, 2010: Integral Mining Manual. Comprehensive Manual of Legal, Technical and Economic Procedures, 2010.

Agency of Mining Regulation and Control – ARCOM, 2014: Regulation of health and safety at work 16, 2014.

Agency of Mining Regulation and Control – ARCOM, 2015: Instructions for the Constitution of Right-of-way. Published by Resolution No. 045 DIR-ARCOM-2015 on December 29, 2015.

Agency of Mining Regulation and Control – ARCOM, 2016: Technical Guideline 006. Technical guidelines of the ARCOM. Published in the Official Gazette, supplement 789 of July 15, 2016.

Albuja, L., Almendariz, C., Montalvo, L. D., Cáceres, F., Román, C., & Luis, J., 2012: Vertebrate fauna of Ecuador (No. R591. 986 al156f e.g., 1 024969). National Polytechnic School. Quito, Ecuador.

Arribas, A. 1995: Magmas, Fluids, and Ore Deposits. Nepean, Ont.: Mineralogical Association of Canada.

Arribas, A. Jr. 1995: Characteristics of High-Sulphidation Epithermal Deposits, and Their Relation to Magmatic Fluid. In Magmas, Fluids, and Ore Deposits, Ed.: J.FM. Thompson, Mineralogical Association of Canada Short Course Vol. 23 (1995).

Astudillo, P. X., Barros, S., Siddons, D. C. and E. Zarate., 2017: Influence of Habitat Modification by Livestock on Paramo Bird Abundance in Southern Andes of Ecuador. Studies on Neotropical Fauna and Environment, 2017. https://doi.org/10.1080/01650521.2017.1382122.

Barbour, M., Gerristen, J., Snyder, B. & Stribling, J., 1999: Rapid Bioassessment Protocols for Use in Streams and Wadeable Rivers: Periphyton, Benthic Macroinvertebrates and Fish. Second Edition. EPA 841-B-99-002. Environmental Protection Agency; Office of Water; Washington, DChttps://www3.epa.gov/region1/npdes/merrimackstation/pdfs/ar/AR-1164.pdf

Bieniawski, Z.T., 1989. Engineering rock mass classifications: a complete manual for engineers and geologists in mining, civil, and petroleum engineering. John Wiley & Sons.BQE 2018a: Treatment of Mine Impacted Water at Loma Larga. Engineering Design Report, October 26, 2018.

BQE 2018: Selenium Removal from Mine Impacted Water at Loma Larga as Project Contingency, Addendum to Water Treatment Plant (WTP) Engineering Design, November 7, 2018.

Brito, J., Camacho, M. A., Romero, V., and Vallejo, A. F., 2019: Mammals of Ecuador. Version 2019.0. Museum of Zoology, Pontificia Universidad Católica del Ecuador.

Buytaert, W., Cuesta-Camacho, F. and Tobón, C., 2011: Potential impacts of climate change on the environmental services of humid tropical alpine regions. Global Ecology and Biogeography, 20(1), pp.19-33.

Camacho, M., 2014: The ecuadorian moorlands: characterization and considerations for their conservation and sustainable use. Revista Anales, 1 (372), 77-92.

Canadian Dam Association, 2007: CDA Technical Bulletins: Dam Safety.

Canadian Dam Association, 2013: Dam Safety Guidelines 2007, 2013 Edition.

Canadian Dam Association, 2014: Application of Dam Safety Guidelines to Mining Dams, Technical Bulletin.

Canadian Dam Association, 2016: CDA Technical Bulletin: Dam Safety Reviews.

Canadian Dam Association, 2014: Application of Dam Safety Guidelines to Mining Dams, Technical Bulletin.

Carrascal, D. R., del Pilar Arroyave, M., Lagoueyte, M. E. G, and P. A. Z. Jaramillo, 2011: Increased Climatic Stress on High-Andean Ecosystems in the Cordillera Central of Columbia. In book: Climate change and biodiversity in the Tropical Andes. Edition: 1. MacArthur Foundation, Inter-American Institute of Global Change Research (IAI) and Scientific Committee on Problems of the Environment (SCOPE) Editors: Herzog SK, Martínez R, Jørgensen PM, Tiessen H

Castillo, D.D, Quevedo, L. A, Gavilanes, A. V., Carrasco, J.C., Ricaurte, C. B. and S. A. Borz., 2017: Diversity, Composition, and Structure of Andean High Forest in Ecuador, South America. Bulletin of the Transilvania University of Brasov 10 (59) No. 2

Chiaradia, M., et al. 2004: Metal Sources in Mineral Deposits and Crustal Rocks of Ecuador, A Lead Isotope Synthesis. In Economic Geology, v. 99, pp 1085-1106.

Chouinard, A., Williams-Jones, A.E., Leonardson, R.W., Hodgson, C.J., Silva, P., Tellez, C., and Rojas, F., 2005: Geology and Genesis of the Multistage High-Sulfidation Epithermal Pascua Au-Ag-Cu Deposit, Chile and Argentina. Economic Geology, 100(3), 463-490.

CIM, 2014: CIM Definition Standards for Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions, adopted by CIM Council on May 10, 2014.

CIM, 2019: CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines, November 29, 2019.

Clark, L. and Pakalnis, R., 1997: April. An empirical design approach for estimating unplanned dilution from open stope hangingwalls and footwalls. In Proceedings of the 99th annual general meeting. Vancouver: Canadian Institute of Mining, Metallurgy and Petroleum.

Corbett, G. 2005: Epithermal Au-Ag Deposit Types – Implication for Exploration. In Proceedings of Proexplo Conference Peru, 2005.

Directory of the Electricity Regulation and Control Agency - ARCONEL, 2017: Regulation for the Distribution and Commercialization of Electrical Energy - Regulation No. ARCONEL 005/17 of December 7, 2017. Published on December 7, 2017.

DPM, 2021: Annual Information Form for the financial year ended December 31, 2021. Dundee Precious Metals Inc.

DPM, 2022: "*Informe Técnico para obtener Certificado de Viabilidad Técnica de la Relavera, Proyecto Minero Loma Larga*" DPMEcuador S.A., April 2022

DRA Americas Inc., 2018: Loma Larga Feasibility Study Risk Register, J01834-PM-REG-001 Rev I, Nov 21, 2018.

DRA Americas Inc., 2019: Feasibility Study on the Loma Larga Project, Azuay Province, Ecuador. Prepared on behalf of INV Metals Inc, January 2019.

DRA Americas Inc., 2020: Feasibility Study on the Loma Larga Project, Azuay Province, Ecuador. NI 43-101 Technical Report prepared by DRA for INV Metal Inc., April 2020.

DRA Americas Inc., 2021: Feasibility Study on the Loma Larga Project, Azuay Province, Ecuador. Prepared on behalf of Dundee Precious Metals Inc., November 2021.

DRA Americas Inc., 2024: Loma Larga Project – Volume I – Technical Report Feasibility Study Update Optimisation. Prepared by DRA for Dundee Precious Metals Inc., January 2024.

Eisenbeis, R., 1987: Eroding market imperfections: implications for financial intermediaries, the payments system, and regulatory reform. Proceedings, 19-62.

Encalada, A. R., 2011: Simplified protocols for the evaluation of the ecological quality of high Andean rivers (CERA-S). QUITO.

Entrix 2022: “Preliminary Environmental Design Criteria Report – Loma Larga Project”, 10485009, July 2022.

ERM, 2018: Proyecto Minero Loma Larga: Water Modelling Report. Prepared for INV Minerales Ecuador S.A. by ERM Consultants Canada Ltd.: Vancouver, British Columbia.

ERM, 2017: Review and Identification of Data Gaps of Baseline Studies for the Loma Larga Project. Prepared on behalf of INV Metals, Inc.

Ex Ministry of Non-renewable Natural Resources, 2009: General regulation of the mining law. Executive Decree 119. Published in the Official Register Supplement 67 of November 16, 2009. Last modification: January 04, 2017.

Ex Ministry of Non-renewable Natural Resources, 2009: Mining Law. Law 45 published in the Official Gazette, Supplement 517 of January 29, 2009. Last modification: April 29, 2016.

Executive function of the Presidency of the Republic, 2010: Organic Code of Territorial Organization. Law 0 published in the Official Register Supplement 303 of October 19, 2010. Last modification: January 24, 2017.

Farley, K. A. and E. F. Kelly., 2004: Effects of Afforestation of a Paramo Grassland on Soil Nutrient Status. Forest Ecology and Management 195 (3):281-290.

Frenkel, C., Guayasamín, J. M., Yánez-Muñoz, M. H., Varela-Jaramillo, A., and Ron, S. R. 2014: Pristimantis cryophilius Anfibios del Ecuador:

G Mining Services, 2023: Feasibility Study on the Loma Larga Project, Azuay Province, Ecuador. Prepared on behalf of Dundee Precious Metals Inc., April 2023

Gareca Leon, Edgar E., Fernadez, M., and Stanton S., 2010: Dendrochronological Investigation of the High Andean Tree Species Polylepis Besseri and Implications for Management and Conservation. Biodiversity and Conservation. 19. 1839-1851.

Geomingold, 2020: “Informe Final Construcción Calicatas y Muestreo Suelos”, Proyecto Loma Larga, Azuay, Ecuador. February 2020.

Geosciences North Ltd., 2018: Internal Memo: Summary of Work on INV Loma Larga – Geophysics to date. Prepared for INV Metals Limited.

GISTM, 2020: “Global Industry Standard for Tailings Management”, International Council on Mining & Metals, UN Environment Programme, Principles for Responsible Investment. August 2020.

Global Mine Design, 2023: Geotechnical evaluation of Loma Larga Feasibility Study Update Optimisation, Loma Larga Project (GMD-DPM-LLP-107-R-002), 29th September 2023

Global Mine Design, 2023: Geotechnical gap analysis report, Loma Larga Project (GMD-DPM-LLP-107-R-001), 29th September 2023

Global Mine Design, 2023: Ventilation Design Report for Loma Larga Feasibility Study Update Optimisation, Loma Larga Project (GMD-DPM-LLP-112-R-001), 29th September 2023

Golder, 2009: Assessment of Critical Hydrogeological Issues, Quimsacocha Project, Ecuador. Prepared for IAMGold Corporation, June.

Gradstein, S. R., and Leon-Yanez, S., 2017: Liverworts (Marchantiophyta) of Polylepis Pauta Forests from Ecuador with Description of Leptoscyphus Leoniae sp. nov. and Plagiochila Pautaphila sp. nov. Nova Hewigia published online July 2017. DOI: 10.1127/nova_hedwigia/2017/0432.

Granizo, T., Pacheco, C., Ribadeneira, M. B., Guerrero, M. and Suárez, L., 2002: Libro Rojo de las Aves del Ecuador. SIMBIO, Conservación Internacional, EcoCiencia, Ministerio del Ambiente, UICN. Quito, Ecuador.

Gray, M. D., 2013: QA/QC Review, Drillhole Database Inclusive of LLD-364 through 375, Loma Larga Project, Ecuador (2013). Prepared for INV Metals Limited by Resource Geosciences Incorporated.

Green, D. M., 2003: The ecology of extinction: Population fluctuation and decline in amphibians. Biological Conservation, 111, 331- 343.

Hedenquist, J.W., 2013: Update on the Loma Larga Au-Cu-Ag project, Ecuador, Report for INV, April 2013.

Hydrocarbons Regulation and Control Agency, 2014: Instruction for Purchasing Fuels in the Industrial Segment. Resolution of the Hydrocarbons Regulation and Control Agency. Official registration 387 of December 02, 2014.

IAMGOLD, 2008: Press Release: IAMGOLD releases Quimsacocha Pre-feasibility Study, Positive Results Demonstrate a Solid Project for IAMGOLD, July 29, 2008, Toronto.

IAMGOLD, 2008: Unpublished Pre-feasibility study, July 2008.

IAMGOLD, 2009: Quimsacocha Gold Project, Azuay Province, Ecuador, a technical report prepared by IAMGOLD Technical Services, February 16, 2009.

IAMGOLD, 2011: Auditoria Ambiental de Cumplimiento. Concesiones Mineras: Cerro Casco y Rio Falso. Año 2011. IAMGOLD Ecuador S.A.

IFC, 2007a: International Finance Corporation Environmental, Health, and Safety General Guidelines. World Bank Group.

IFC, 2007b: International Finance Corporation Environmental, Health, and Safety Guidelines for Mining. World Bank Group.

IFC, 2007c: International Finance Corporation Environmental, Health, and Safety Guidelines for Electric Power Transmission and Distribution. World Bank Group.

IFC, 2012a: International Finance Corporation Performance Standards on Environmental and Social Sustainability. World Bank Group.

IFC, 2012b: International Finance Corporation Performance Standard 6. Biodiversity Conservation and Sustainable Management of Living Natural Resources. World Bank Group. https://www.ifc.org/wps/wcm/connect/bff0a28049a790d6b835faa8c6a8312a/PS6_English_2012.pdf?MOD=AJPERES. Accessed June 2018.

IFC, 2012c: International Finance Corporation Guidance Note 6. Biodiversity Conservation and Sustainable Management of Living Natural Resources. World Bank Group. https://www.ifc.org/wps/wcm/connect/a359a380498007e9a1b7f3336b93d75f/Updated_GN6-2012.pdf?MOD=AJPERES. Accessed June 2018.

Iniguez, V., 2017: Multi-scale monitoring and modelling for the analysis of the hydrologic response of southern Ecuadorian Andean Páramos.

Isch, E., 2012: Climate change and management of the moorlands. CAMAREN, AVSF. Quito.

Itasca. 2017a: Loma Larga Hydrogeologic Field Activities, April 20, 2017

Itasca. 2017b: Groundwater Gradient and Water Table Contour Map for Loma Larga Project, June 20, 2017

Itasca 2017c: Conceptual Water Balance and Water Management Planning, Advanced Exploration Confirmation Program, Loma Larga, November 10, 2017

Itasca, 2018: “Simulations of Groundwater Flow Conditions for the Loma Larga Project”. Prepared for INV Metals, Inc. by Itasca Denver, Inc., April 2018.

Itasca. 2018a: Ramp Inflow Assessment for the Loma Larga Project, March 19, 2018.

Itasca. 2018b: Simulations of Groundwater Flow Conditions for the Loma Larga Project, November 2018

Itasca 2018c: Waste Rock Geochemistry Kinetic Testing Recommendations, Loma Larga Project, June 5, 2018

IUCN, 2020: IUCN Red List of Threatened Species. International Union for Conservation of Nature and Natural Resources. Website: www.iucnredlist. Org. Consultation: 2020.

León-Yánez, S., 2000: The flora of the Ecuadorian moorlands. The biodiversity of the moorlands. GTP/Abya Yala. Quito.

Loma Larga, 2016/2017: Ore Characterization for Metallurgical Testing – April 12, 2017.

Luteyn, J., 1999: Moorlands: A checklist of plant diversity, geographic distribution, and botanical literature. Memoirs of the New York Botanical Garden, 84, 1-278.

MacDonald, P.J., Bissig, T., Hart, C.J.R., Barreno, J., Viera, F., Mantilla, G., and Rogers, J., 2012: Hydrothermal evolution of the Rio Falso Fault, Quimsacocha High Sulfidation Au-Ag-Cu District, Azuay Province, Ecuador. Presented to SEG conference, Lima, Peru, September 2012.

MacDonald, P.J., Bissig, T., Hart, C.J.R., Rogers, J., Barreno, J. E., and Bell, R. C., 2013: Magmatic evolution during flattened subduction and high-sulfidation epithermal ore deposition: Quimsacocha volcanic center, Azuay Province, Ecuador. Presented to SEG conference, Whistler, B.C. September 25, 2013.

MacDonald, P.J., Hart, C., Bissig, T., 2010: Field Report on the Geology of the Quimsacocha Property, Azuay Province, Ecuador, Report for PhD at Mineral Deposit Research Unit, University of British Columbia.

Madrinan, S., Cortes, A. J., and Richardson, J.E., 2013: Paramo is the World's Fastest Evolving and Coolest Biodiversity Hotspot. Front Genet 4:192 https://www.ncbi.nlm.nih.gov/pmc/articles/PMC3793228/. Accessed June 2018.

Mena, P. & Hofstede, R., 2006: The Ecuadorian Moorlands. La Paz: Universidad Mayor de San Andrés.

Mena-Vásconez, P., Castillo, A., Flores, S., Hofstede, R., Josse, C., Lasso, S., Medina, G., Ochoa, N. and Ortiz, D., 2011: Moorlands. Landscape studied, inhabited, managed and institutionalized. EcoScience/Abya-Yala/ECOBONA. Quito.

Mena-Vásconez, P., Medina, G. and Hofstede, R. G. M., 2001: The Moorlands of Ecuador. Particularities, Problems and Perspectives. Abya Yala/Paramo Project. Quito.

MERNNR, 2020: “Instructivo para la Aprobación de Proyectos de Diseño, Construcción, Operación y Mantenimiento de Depósitos de Relaves para la Mediana y Gran Minería”, Ministerio de Energía y Recursos Naturales No Renovables, Gobierno del Ecuador. July 15, 2020.

Met Sampling Program Update – December 19, 2016 Loma Larga Met test: DDH Samples INV Metals – S Ramnath – August 19, 2013.

Minga, D., Ansaloni R., Verdugo, A. And Ulloa, C.U., 2016: Flora del Paramo del Cajas, Ecuador. Universidad del Azuay. Imprenta Don Bosco. Cuenca.

Ministry of Environment of Ecuador, 2013: System of Classification of the Ecosystems of Continental Ecuador. Quito: Undersecretariat of Natural Heritage.

Ministry of Environment, Government of Ecuador, 2004: Environmental management law. Coding 19 published in the Official Gazette Supplement 418 of September 10, 2004. Last modification: May 22, 2016.

Ministry of Environment, Government of Ecuador, 2008: Ministerial Agreement No. 26 of the issues the procedures for registering hazardous waste generators, managing hazardous waste prior to environmental licensing, and for transporting hazardous materials. Published in Official Gazette No. 334 of May 12, 2008.

Ministry of Environment, Government of Ecuador, 2014: Environmental Regulation of Mining Activities. Ministerial Agreement 37 published in the Official Gazette Supplement 213 of March 27, 2014. Last modification: July 12, 2016.

Ministry of Environment, Government of Ecuador, 2015: Ministerial Agreement of the Ministry of Environment 083b, Posted on June 8, 2015.

Ministry of Environment, Government of Ecuador, 2017: Environmental Code. Official Register: Supplement 983 on April 12, 2017.

Ministry of Mines, Government of Ecuador, 2015: Ministerial Agreement No.18 of the Ministry of Mines- Instructions for authorizations for processing plants, smelting and tailings, published in the Official Gazette 554 of July 29, 2015.

Ministry of Mines, Government of Ecuador, 2018: Strategic Mining Projects. PDAC 2018 Presentation. Accessed on June 28, 2018. https://www.mineria.gob.ec/wp-content/uploads/2018/03/Strategic-Mining-Projects-2018.pdf.

Ministry of National Defense, Government of Ecuador, 1997: Regulation of the Law on arms, ammunition, explosives and accessories. Executive Decree 169 published in the Official Gazette 32 of March 27, 1997. Last modification: June 15, 2015.

Ministry of National Defense, Government of Ecuador, 2016: Ministerial Agreement 270 of the Ministry of Defense. Posted on September 6, 2016.

Ministry of Public Health, Government of Ecuador, 2014: Ministerial Agreement No. 5212 that issues the typology to standardize health establishments by levels of care and support services of the national health system. Published by the Ministry of Public Health on December 24, 2014.

Ministry of Public Health, Government of Ecuador, 2014: Substitute regulation for granting permits of functioning to the establishments subject to sanitary control and surveillance (Ministerial Agreement No. 00004712). Published on February 11, 2014.

Ministry of Public Health, Government of Ecuador, 2016: Ministerial Agreement No. 79 Issues the sanitary regulations for the issuance of the operating permit of public and private health facilities of the national health system, home health care services, establishments providing indirect support services and prepaid health and medicine companies. Published in Official Gazette No. 834 of September 6, 2016.

Ministry of Transport and Public Works, Government of Ecuador, 2013: General specifications for construction of roads and bridges. Prepared by the Ministry of Transport and Public Works. Volume No.3 in Quito, 2013.

Ministry of Urban Development and Housing, Government of Ecuador, 2015: Ecuadorian Standard for Fire Construction- Code NEC-HS-CI. Published in October 2015.

Mosquera, G. M., Lazo, P. X, Celleri, R., Wilcox, B. P. and Crespo P., 2015: Runoff from Tropical Alpine Grasslands Increases with Areal Extent of Wetlands. Catena 125: 120-128.

National Assembly, Government of Ecuador, 2014: Comprehensive Organic Criminal Code. Published in the Official Register Supplement 180 of February 10, 2014.

National Congress Committee on legislation and codification, Government of Ecuador, 2005: Civil Code. Codification 10 published in the Official Register Supplement 46 of June 24, 2005. Last modification: May 22, 2016.

National Constitution, Government of Ecuador, 2008. Published in Legislative Decree 0 through Official Register 449 of October 20, 2008. Last modified: July 13, 2011.

National Police, Ecudaor, 2008: Organic Law of Land Transport, Traffic and Road Safety. Law published in the Official Gazette, Supplement 398 of August 07, 2008. Last modification: December 31, 2014.

NewFields 2018: Technical Memorandum: Climate Analysis Loma Larga Project Azuay Province, Ecuador. April 30, Prepared on behalf of INV Metals, Inc.

NewFields, 2018a: “Climate Analysis Technical Memorandum”, Loma Larga Project Rev 0. April 30, 2018.

NewFields, 2018b: “Additional Climate Analysis Technical Memorandum”, Loma Larga Project, Rev 0. September 27, 2018.

NewFields, 2018c: “Seismic Hazard Assessment”, Loma Larga Project, Azuay Province, Ecuador, 30 March.

NewFields, 2018d: “Rougher Tailings – Geotechnical Laboratory Testing Summary”, Loma Larga Project, Technical Memorandum, February 2, 2018.

NewFields, 2018e: Interim Results for the Geochemical Characterization of Floatation Tailings, Loma Larga Project, Azuay Province, Ecuador, Technical Memorandum, May 17.

NewFields, 2018f: Static Test Results for Prospective Quarry Sites, Loma Larga Project, Azuay Province, Ecuador, Technical Memorandum. June 15.

NewFields 2019: Geochemical Characterization of Flotation Tailings, Loma Larga Project, Azuay Province, Ecuador, February 15, 2019.

NewFields, 2022: “Climate Change Considerations for Loma Larga Project”, Loma Larga – Feasibility Update, Rev C. November 18, 2022.

Orix Presentations on sample selection for PFS testwork program: Met Sampling Program Update – December 19, 2016 Loma Larga Met test: DDH.

Samples INV Metals – S Ramnath – August 19, 2013.

Páez, N. B., and Ron, S. R., 2019: Systematics of Huicundomantis, a new subgenus of Pristimantis (Anura, Strabomantidae) with extraordinary cryptic diversity and eleven new species. ZooKeys, 868, 1.

Parisi, G., 1979: Field-theoretic approach to second-order phase transitions in two-and-three dimensional systems. Journal of Statistical Physics, 23, 49–82. https://doi.org/10.1007/BF01014429.

Paterson & Cooke, 2018: “Loma Large Paste Backfill Feasibility Study – Test Work Report”, Document No. 32-0252-00-TW-REP-0010-C, Revision C, June 6, 2018.

Paterson & Cooke Feasibility Study Report (32-0252-2210-GE-REP-0001) Rev A, 8 Sept 2018

Paterson & Cooke, 2018: Loma Larga Paste Backfill Feasibility Study – Testwork Report, Document No. 32-0252-00-TW-REP-0010-C, Revision C, June 6, 2018.

Presidency of the Republic, Ecuador, 2007: General regulations of the Cultural Heritage Law, 2007. Published in the Decree 2733 on February 27, 2007. Last modification: April 12, 2007.

Presidency of the Republic, Ecuador, 2012: Regulation on Land Transport, Traffic and Road Safety Law. Published by Executive Decree 1196 in the Official Gazette Supplement 731 of June 25, 2012.

Presidency of the Republic, Ecuador, 2015: Published by Executive Decree 650 in the Official Register Supplement 483 of April 20, 2015. Regulation to the Law of Water Resources and Use of Water.

PROMAS (Universidad de Cuenca), 2009 (March): Estudio de hidrologia - informe segundo año. Universidad de Cuenca.

PROMAS (Universidad de Cuenca), 2011a: Informe de Edafologia. Línea Base en Hidrología para los páramos de Quimsacocha y su área de Influencia. Universidad de Cuenca.

PROMAS (Universidad de Cuenca), 2011b: Linea Base En Hidrológia, Cuarto Año. Universidad de Cuenca.

Promet 101, 2018: Loma Larga - Feasibility Study Metallurgical Report Rev C1, November 29, 2018.

Propaxis, 2018: Línea Base Social Proyecto Loma Larga. Prepared on behalf of INV Metals Inc.

Propaxis, 2018b: Social Baseline Report for Loma Larga Project.

Propraxis, 2018c: Línea Base Social Proyecto Loma Larga. Prepared on behalf of INV Metals Inc.

PUCE, 2018: Libro Rojo de las Plantas Endemicas del Ecuador. https://bioweb.bio/floraweb/librorojo/home. Accessed October 2018.

Quichimbo, P. and Cisneros, P., 2013: Predicting the spatial distribution of soil depth: A case study from the South Ecuadorian Andes.

Ridgely, R. S., and Greenfield, P.J., 2001: The Birds of Ecuador. Status distribution and taxonomy. Cornell University Press.

Robert, F., et al., 2007: Models and Exploration Methods for Major Gold Deposit Types. In Proceedings of Exploration 07: Fifth Decennial International Conference on Mineral Exploration, Ed: B. Milkereit, 2007, pp 691-711.

Roscoe Postle Associates, Inc., 2016: Pre-Feasibility Study on the Loma Larga Project, Azuay Province, Ecuador, Volume 1 – Report, Prepared for INV Metals, Inc., September 30.

RPA, 2012: Technical Report on the Quimsacocha Project, Azuay Province, Ecuador. Technical report prepared by Valliant, W., Masun, K., and Postle, J. for INV Metals Inc.

RPA, 2016: Technical Report on the Loma Larga Project, Azuay Province, Ecuador. NI 43-101 Technical Report prepared by Cox J.J., Altman K.A., Masun, K., Robertson, L., and Diaz, C.A. for INV Metals Inc. August 29, 2016.

RPA, 2018: Memorandum on the Loma Larga Geochemistry Block Model prepared by K. Masun for INV Metals Limited.

RPA, 2016: Pre-feasibility Study on the Loma Larga Project, Azuay Province, Ecuador. Prepared on behalf of INV Metals, Inc.

Scott Wilson RPA, 2006: Technical Report on the Quimsacocha Project, Azuay Province, Ecuador. Technical report prepared by Valliant, W., Roscoe, W.E., and Ross, D. for IAMGOLD Corporation.

SGS Lakefield Research Limited, 2018: An Investigation into the Recovery of Gold and Copper from Samples from Loma Larga, Project 14269-002 - Final Report – October 25, 2018.

SGS Lakefield Research Limited, 2006: An Investigation into the Recovery of Gold and Copper from Quimsacocha Project Samples, Project 11035-001 – Report 1, May 10, 2006.

Sillitoe, R.H., 1999: Styles of High-Sulphidation Gold, Silver and Copper Mineralisation in Porphyry and Epithermal Environments. In Proceedings Pacrim Congress 1999, Bali Indonesia October 10-13, 1999.

SRK, 2006: Quimsacocha Gold Project, Azuay Province, Ecuador, Preliminary Assessment Report. Prepared for IAMGold Corporation, May.

Stattersfield, A., Crosby, M. J., Long, A. J. and Wege, D. C., 1998: Endemic Bird Areas of the World. Priorities for Biodiversity Conservation. Birdlife Conservation Series No. 7, Birdlife International, Cambridge.

Supreme Council of Government, Ecuador, 1980: Law on arms, ammunition, explosives and accessories Published in Supreme Decree 3757 through Official Register 311 of November 7, 1980. Last modification: March 9, 2009.

Supreme Council of Government, 1979: Regulation of Radiological Safety. Published in the Supreme Decree 3640 of the Official Registry 891 of August 8, 1979.

Terneus-Jácome, E. and Yanez, 2018: Fundamental principles around water quality, the use of bioindicators and River Ecological restoration in Ecuador. La Granja.

Tirara, D., 2017: *Guia de campo de los mamiferos del Ecuador. Ediciones Murielago Blanco. Publicacion especial sobre los mamiferos del Ecuador* 11. Quito. 600 pp.

Tirira, D., 2017: Field guide of the mammals of Ecuador. Ediciones Murciélago Blanco. Special publication on the mammals of Ecuador 11. Quito.

Tirira, D. G., 2021: Lista Roja de los mamíferos del Ecuador, en: Libro Rojo de los mamíferos del Ecuador (3a edición). Asociación Ecuatoriana de Mastozoología, Fundación Mamíferos y Conservación, Pontificia Universidad Católica del Ecuador y Ministerio del Ambiente, Agua y Transición Ecológica del Ecuador. Publicación Especial sobre los mamíferos del Ecuador 13, Quito.

Tirira, D.G., Sánchez-Sánchez, L. and Álvarez-Solas, S., 2021: An Update of the Geographic Distribution of the Red-Mantled Saddle-Back Tamarin, Leontocebus lagonotus (Callitrichidae), in Ecuador. International Journal of Primatology, 42(4), pp.600-617.Toivonen, J. M., Kessler, M., Ruokolainen, K., and D. Hertel., 2011: Accessibility Predits Structural Variation of Andean Polylepis forests. Biodiversity Conservation 20:1789-1802.

Torres-Carvajal, O., Pazmiño-Otamendi, G., and Salazar-Valenzuela, D., 2020: Reptiles of Ecuador. Version 2020. Museum of Zoology, Pontificia Universidad Católica del Ecuador.

Ulloa, C.U., Molina, S.A., Jorgensen, P.M., and Minga D., 2018: Cajas National Park Field Guide of 100 Wild Plants of the Paramo. http://www.missouribotanicalgarden.org/Portals/0/staff/PDFs/ulloa/GuiaPlantas.pdf. Accessed October 2018.

UNEP-WCMC, 2016: The State of Biodiversity in Latin America and the Caribbean: A mid-term review of progress towards the Aichi Biodiversity Targets. United Nations Environmental Programme-World Conservation Monitoring Centre, Cambridge, UK.

Universidad de Azuay, 2017: Informe Final del Monitoreo Biotico del Proyecto Minero Loma Larga. Coordinacion Dr. Pedro Astudillo Webster, Blgo. Santiogo Barros Q., Blgo. Boris Landazuri P. Escuela de Biologia, Ecologia y Gestion. Cuenca, Ecuador.

Universidad de Azuay, 2018: Informe Final del Monitoreo Biotico del Proyecto Minero Loma Larga. Coordinacion Dr. Pedro Astudillo Webster, Blgo. Santiogo Barros Q., Blgo. Boris Landazuri P. Escuela de Biologia, Ecologia y Gestion. Cuenca, Ecuador.

Western Mining Services, 2018: Loma Larga Project Assessment and Targeting Study. Confidential report prepared by Bussey, S. and Margeson, B. for INV Metals Inc.

White, N.C., and Hedenquist, J.W., 1995: Epithermal Gold Deposits: Styles, Characteristics and Exploration. SEG Newsletter, 23, 1,9-13.

White, N.C., and Hedenquist, J.W., 1990: Epithermal Environments and Styles of Mineralization: Variations and Their Causes, and Guidelines for Exploration, II. In: J.W. Hedenquist, N.C. White and G. Siddeley (Editors), Epithermal Gold Mineralization of the Circum-Pacific: Geology, Geochemistry, Origin and Exploration, J. Geochem Exploration, 36: pp 445-474.

World Bank, 2012: What a Waste, A Global Review of Solid Waste Management, Urban Development Series Knowledge Papers, No. 15. March.

28 ABBREVIATIONS

Abbreviation	Definition
$ M	Millions of Dollars
$/kWh	Dollar per Kilowatt-Hour
%	Percent
%w/w	Percent Solid per Weight
”	Inch
±	Plus-Minus
µm	Micrometre, Micron
°	Degree
°C	Degree Celsius
3D	Three Dimensional
AACE	Association for the Advancement of Cost Engineering
ABA	Acid-Base Accounting
Ag	Silver
Ag-As	Silver-Arsenic
Ag-Cu-As	Silver-Copper-Arsenic
Ai	Abrasion Index
ANFO	Ammonia Nitrate Fuel
ANRQ	*Área Nacional de Recreación Quimsacocha*
AP	Acid Potential
ARC	*Agencia de Regulación y Control*
ARCAA	*Agencia de regulación y Control del Agua* (Regulation and Control of the Environment and Water)
ARCERNNR	*Agencia de Regulación y Control de Energía y Recursos Naturales no Renovables* (Agency for Regulation and Control of Energy and Non-Renewable Natural Resources)
ARCOM	*Agencia de Regulación y Control Minero* (Mining Regulation and Control Agency)
As	*Arsenic*
ASL	Above Sea Level
Au	Gold
AUD	Australian Dollar
BAAS	Battery as a Service
BEV	Battery Electric Vehicle
BFR	Benchtop Flotation Reactor

Abbreviation	Definition
BMA	Bulk Mineral Analysis
BML	Base Metallurgical Laboratories
BMP	Best Management Practices
BSI	BSI Laboratories
BWI	Bond Ball Mill Work index
Ca	Calcium
$CaCO_3$	Calcite
CAD	Canadian Dollar
Capex	Capital Expenditure, Capital Cost Estimate
CCTV	Closed-Circuit Television
CIF	Cost, Insurance, and Freight
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
cm	Centimetre
cm/s	Centimetre per Second
CM	Construction Management
COA	Certificate of Analysis
COA	*Código orgánico del Ambiente* (Organic Code of the Environment)
COPC	Constituents of Potential Concern
COV	Coefficients of Variation
CPB	Cemented Paste Backfill
CRF	Cemented Rock Fill
CRM	Certified Reference Material
CSD	Critical Solids Density
Cu	Copper
Cu-Ag-As	Copper-Silver-Arsenic
CWi	Crushing Work Index
dB	Decibel
DCF	Discounted Cash Flow
deg	Degree
DPM	DPM Metals (formerly Dundee Precious Metals)
DPME	DPM Ecuador Holdings Inc.
DRA	DRA Americas Inc.
DSO	Deswik Stope Optimizer
DWT	Drop Weight Test

Abbreviation	Definition
E&I	Electrical and Instrumentation
EEL	Electrical Equipment List
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
ELOS	Equivalent Linear Overbreak/Slough
EMP	Environmental Management Plan
EN	Endangered
EOI	Express of Interest
EP	Engineering and Procurement
EPCM	Engineering, Procurement and Construction Management
E-Room	Electrical Room
EUR	Euro
Fe	Iron
FID	*Final Investment Decision*
FODINFA	*Fondo de Desarrollo para la Infancia*
FS	Feasibility Study
FSU	Feasibility Study Update
FSUO	Feasibility Study Update Optimisation
FTSF	Filtered Tailings Storage Facility
g	Gram
G&A	General and Administrative
g/cm^3	Gram per Cubic Centimetre
g/dmt	Gram per Dry Metric Ton
g/t	Gram per Tonne
GAAP	Generally Accepted Accounting Principles
GAD	*Gobierno Autónomo Descentralizado* (Decentralised Autonomous Governement)
GBP	British Pound
GEMS	GEOVIA GEMS 6.8.2
GISTM	Global Industry Standard on Tailings Management
GMD	Global Mine Design Ltd.
GMS	G Mining Services
GRML	General Regulations to the Mining Law

Abbreviation	Definition
GT	Geotechnical
H&S	Health and Safety
H1	First Half
H_2SO_4	Sulphuric Acid
ha	Hectare
HCl	Hydrochloric Acid
HCT	Humidity Cell Test
HDPE	High Density Polyethylene
HF	Hydrofluoric Acid
HG	Hydrogeological
hp	Horsepower
Hz	Hertz
I/O	Input/Output
ICP	Inductively Coupled Plasma
IESS	*Instituto Ecuatoriano de Seguridad Social* (Ecuadorian Social Security Institute)
IFC	International Finance Corporation
IFRS	IFRS Accounting Standards
INEC	*Instituto Nacional de Estadistica y Censos* (National Institute of Statistics and Census)
INV	INV Metals
IPA	Investment Protection Agreement
IPMT	Integrated Project Management Team
IRR	Internal Rate of Return
IUCN	International Union for Conservation of Nature
k_{80}	Particle Size 80% Fine
kg	Kilogram
$kg/m^2•h$	Kilogram per Square Metre per Hour
kg/m^3	Kilogram per Cubic Metre
km	Kilometre
km/h	Kilometre per Hour
kPa	Kilopascal
koz	Thousand Ounce

Abbreviation	Definition
kt	Kilotonne
kt/d	Kilotonne per day
kV	Kilovolt
kVA	Kilovolt-Ampere
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per tonne
kWh/y	Kilowatt-hour per year
L	Litre
L/s	Litre per Second
lb	Pound
LC	Least Concern
LCRS	Leak Collection Recovery System
LCT	Locked Cycle Test
LHD	Load Haul Dump
LMP	*Límites máximos permitidos* (Maximum Permissible Limits)
LOM	Life-of-Mine
LOTOTO	Lockout Tagout Tryout
m	Metre
m ASL	Metre above Sea Level
Mlb	Million Pound
Moz	Million Ounce
m/d	Metre per Day
m/s	Metre per Second
m^3	Cubic Metre
m^3/cell	Cubic Metre per Cell
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/s	Cubic Metre per Second
m^3/m^2/d	Cubic Metre per Square Metre per Day
MAATE	*Ministerio del Ambiente Agua y Transición Ecológica* (Ministry of Environment, Water and Ecological Transition)
MAQAN	*Muestra de agua Quinuahuaycu* (Water Sample in Quinuahuaycu)
MARC	*Muestra de agua Río Cristal (*Water Sample in Río Cristal)

Abbreviation	Definition
MARF	*Muestra de agua quebrada Rumiguaycu* (Water Sample in Rumiguaycu Creek)
MCC	Motor Control Centre
MEL	Mechanical Equipment List
MEM	*Ministerio de Energía y Minas* (Ministry of Energy and Mines)
mg	Milligram
mg/kg	Milligram per Kilogram
mg/L	Milligram per Litre
mg/m^3	Milligram per Cubic Metre
MIBC	Methyl Isobutyl Carbinol
min	Minute
MINTEL	*Ministerio de Telecomunicaciones*
mm	Millimetre
Mm^3	Million Cubic Metre
Moz	Million Ounces
MPa	Megapascal
MRE	Mineral Resource Estimate
msl	Mean Sea Level
mSv/annum	Millisievert per Annum
Mt	Megatonne
MTOP	*Ministerio de Transporte y Obras Publicas* (Ministry of Transport and Public Work)
Mtpa	Million Tonnes per Annum
MWTP	Main Water Treatment Plant
N/A	Not Applicable
Na_2SO_3	Sodium Metabisulphate
NAGpH	Net Acid Generation pH
NewFields	NewFields Mining & Technical Services LLC
NI 43-101	National instrument 43-101
NP	Neutralising Potential
NPI	Net Profit Interest
NPV	Net Present Value
NSG	Non-Sulphide Gangue
NSR	Net Smelter Return
NT	Near Threatened

Abbreviation	Definition
OBE	Operating Basis Earthquake
OEM	Original Equipment Manufacturer
OIT	Operator Interface Terminals
OK	Ordinary Kriging
Opex	Operating Expenditure, Operating Cost Estimate
OSHP	Occupational Safety and Health Plan
OR	Operational Readiness
Orix	Orix Geosciences Inc.
OSHP	Occupational Safety and Health Plan
oz/t	Ounce per Tonne
P&ID	Piping and Instrumentation Diagram
P_{80}	Passing 80%
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
Pb	Lead
PCD	Process Control Diagram
PCS	Process Control System
PDC	Process Design Criteria
PEP	Project Execution Plan
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
PGA	Peak Ground Acceleration
PIMA	Portable Infrared Mineral Analyser
PLC	Programmable Logic Controller
ppb	Parts per Billion
ppm	Parts per Million
PROMAS	*Programa de Manejo de Agua y Suelos* (Water and Soil Management Program)
PS5	Performance Standard No 5
Q1	First Quarter
Q2	Second Quarter
Q3	Third Quarter
Q4	Fourth Quarter

Abbreviation	Definition
QA/QC	Quality Assurance / Quality Control
QC	Quality Control
QP	Qualified Person
RAAM	Environmental Regulations for Mining Activities
RC	Refining Charges
RMR	Rock Mass Rating
ROM	Run of Mine
RPEEE	Reasonable Prospects of Eventual Economic Extraction
rpm	Revolution per Minute
S	Sulphur
S/E	*Subestación Eléctrica*
SAG	Semi-Autogenous Grinding
Sb	Antimony
SCADA	Supervisory Control and Data Acquisition
SENAGU	*Secretaria del Agua* (National Secretary of Water)
set/y	Set per Year
SF	Safety Factor
SFR	Staged Flotation Reactor
SG	Specific Gravity
SMC	SAG Mill Comminution
SNDGA	*Sistema Nacional Descentralizado de Gestión Ambiental* (National Decentralised Environmental Management System)
SO_2	Sulphur Dioxide
SO_4	Sulphate
SQC	Soil Quality Criteria
SUIA	*Sistema Único de Información Ambiental* (Single Environmental Information System)
t/a	Tonne per Annum
t/d	Tonne per Day
t/h	Tonne per Hour
$t/h/m^2$	Tonne per Hour per Square Metre
t/m^3	Tonne per Cubic Metre
t/y	Tonne per Year
TC	Treatment Charge

Abbreviation	Definition
TCCC	Transfer of Care Custody and Control
TDS	Total Dissolved Solids
TLV	Threshold Limit Values
TSS	Total Suspended Solids
UBR	Unified Basic Remuneration
UDS	Underground Distribution System
UG	Underground
UOP	Unit of Production
US$	US Dollars
US$/lb	US Dollars per Pound
US$/dmt	US Dollars per Dry Metric Ton
US$/oz	US Dollars per Ounce
US$/t	US Dollars per Tonne
US$ k	Thousand US Dollars
US$ M	Million US Dollars
V	Volt
VAT	Value Added Tax
VFD	Variable Frequency Drive
VSR	Van Someren Rydon
WBS	Work Breakdown Structure
wt%	Percentage by Weight
WMS	Western Mining Services
WTP	Water Treatment Plant
XRD	X-Ray Diffraction
y	Year

29 CERTIFICATES OF QUALIFIED PERSONS

20 Queen Street West / 29th floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800-8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Ryan Telford, P. Eng., PMP,* do hereby certify that:

1. I am Senior Project Manager with DRA Americas Inc. with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada M5H 3R3.
2. I am a graduate from McMaster University, Ontario, Canada with Bachelor of Engineering degree in Mechanical Engineering in 2002.
3. I am a member in good standing with Professional Engineers Ontario and registered as a Professional Engineer, license number 100079512.
4. I have worked as a Mechanical Engineer and Project Manager since my graduation. My relevant experience includes:
    - 23 years of experience in heavy industry, oil and gas, and the mining industry (15 years), including design, engineering and project management. Work experience includes the development of various feasibility studies and detailed engineering & execution projects in Canada, USA and Africa.
    - Planning, engineering and procurement experience enabling mine, process plant and infrastructure development and cost estimation.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 2 to 5, 18 (Except for 18.6 to 18.9), 21 (Except for 21.1.8 and 21.2.4), and 24. I am also responsible for the associated portions of 1 and 25 to 27 of the Technical Report.

8. I visited the property that is the subject of the Technical Report on February 25, 2025.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025, in Toronto, Ontario.

"Original Signed and Sealed on file"
Ryan Telford, P. Eng., PMP
Senior Project Manager
DRA Americas Inc.

SLR Consulting (Canada) Ltd
55 University Avenue, Suite 501, Toronto, ON M5J 2H7

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Katharine Masun, M.Sc., MSA, P.Geo.*, of Toronto, Ontario, Canada, do hereby certify:

1. I am a Principal Resource Geologist with SLR Consulting (Canada) Ltd, of Suite 501, 55 University Ave., Toronto, ON M5J 2H7.
2. I am a graduate of Lakehead University, Thunder Bay, Ontario, Canada, in 1997 with an Honours Bachelor of Science degree in Geology and in 1999 with a Master of Science degree in Geology. I am also a graduate of Toronto Metropolitan University in Toronto, Ontario, Canada, in 2010 with a Master of Spatial Analysis.
3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #1583), the Province of Newfoundland and Labrador (Reg. #08261), the Province of Saskatchewan (Reg. #55175), and Northwest Territories and Nunavut (#L5478). I have worked as a geologist for a total of 28 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a professional geologist on many mining and exploration projects around the world for due diligence and regulatory requirements,
    - Mineral Resource estimates on polymetallic carbonate replacement deposits and a variety of commodities including graphite, zinc, copper, nickel, silver, gold, rare earth elements, tin, fluorspar, diamonds, and numerous polymetallic deposits,
    - Project geologist on several field and drilling programs in North America, South America, Asia, and Australia,
    - Experienced user of several geological and resource modelling software packages including Leapfrog Geo, Micromine, and GEMS.
4. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
5. I am independent of the Corporation applying the test set out in Section 1.5 of NI 43-101.

6. I am responsible for the preparation of Sections 6 to 12, 14, and 23. I am also responsible for the associated portions of 1, 25 to 27 of the Technical Report.

7. I visited the property that is the subject of the Technical Report on February 17 to 20 2014 and February 25, 2025.

8. I have had prior involvement with the property that is the subject of the Technical Report:

   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 and issued Report Date: November 29, 2021, Document # C6090-PM-REP-001-00, prepared for Dundee Precious Metals Inc.
   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 (issued Original Report Date: April 14, 2020 and Amended Report Date: June 10, 2020), Document # J01834-PM-REP-002, Rev. 1, prepared for INV Metals Inc.
   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of January 11, 2019 and issued Report Date: January 14, 2019, Document # J01834-PM-REP-002, Rev. 0, prepared for INV Metals Inc.
   - "NI 43-101 Technical report on the Loma Larga Project, Azuay Province, Ecuador" with an effective date of August 29, 2016 and issued report date of August 29, 2016, prepared for INV Metals Inc.
   - "NI 43-101 Technical report on the Quimsacocha Project, Azuay Province, Ecuador" with an effective date July 18, 2012 and issued report date July 18, 2012, prepared for INV Metals Inc.

9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with N1 43-101.

10. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025, Toronto, Canada

"Original Signed and Sealed on file"
Katharine Masun, M.Sc. MSA, P.Geo.
Principal Resource Geologist
SLR Consulting (Canada) Ltd.

20 Queen Street West / 29th floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800-8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *David Frost, FAusIMM, B. Met. Eng.,* do hereby certify that:

1. I am Vice President – Process Engineering, with DRA Americas Inc., located at 20 Queen Street West,29th Floor, Toronto, Ontario, Canada M5H 3R3.
2. I am a graduate from the Royal Melbourne Institute of Technology (RMIT), Melbourne, Australia, with a Bachelor of Metallurgical Engineering in Metallurgy in 1993.
3. I am a registered Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM) membership #110899.
4. I have worked as a Metallurgist and Process Engineer in various capacities since my graduation from university in 1993. My relevant work experience includes:
    - 30 years of post-graduate experience of process plant operations and engineering design experience including the oversite of gold heap leach circuit processing in operations and the engineering design of several gold heap leach flowsheets.
    - Supervision and interpretation of numerous metallurgical testwork programs used for the derivation of process plant flowsheets involving gold heap leaching.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the Corporation applying all the tests in section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 13 and 17, and portions of Sections 1, 25 to 27 of the Technical Report.

8. I visited the property that is the subject of the Technical Report on June 27 to 28, 2017.

9. I have had prior involvement with the property that is the subject of the Technical Report:

   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 and issued Report Date: November 29, 2021, Document # C6090-PM-REP-001-00, prepared for Dundee Precious Metals Inc.
   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 (issued Original Report Date: April 14, 2020 and Amended Report Date: June 10, 2020), Document # J01834-PM-REP-002, Rev. 1, prepared for INV Metals Inc.
   - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of January 11, 2019 and issued Report Date: January 14, 2019, Document # J01834-PM-REP-002, Rev. 0, prepared for INV Metals Inc.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025, in Toronto, Ontario.

"Original Signed and Sealed on file"
David Frost, FAusIMM, B. Met. Eng.
Vice President – Process Engineering
DRA Americas Inc.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report - Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Dr. Lewis Meyer, ACSM, MCSM, BEng, MSc, PhD, CEng, FIMMM, QMR, MCIM,* do hereby certify:

1. I am Associate Principal Engineer with Global Mine Design Ltd., with a Corporate business address of Tremough Innovation Centre, Penryn, Cornwall, TR10 9TA, UK.
2. I am a graduate of the Camborne School of Mines with a Bachelor with Honours in Mining Engineering in 1991, a graduate of University of Newcastle Upon Tyne with a Masters in Rock Mechanics and Foundation Engineering in 1995, and a graduate of the Camborne School of Mines (University of Exeter) with a PhD in Geomechanics in 2001.
3. I am a Fellow (FIMMM) in good standing with the Institute of Materials, Minerals and Mining (IOM3) and registered as a Chartered Engineer, license number 432805. I am also a Qualified for Minerals Reporting (QMR) with IOM3.
4. I have worked as a mining engineer continuously for the past 34 years since my graduation from university. I have worked on similar projects to the Loma Larga Project in Ecuador; my experience for the purpose of the Technical Report includes:
    - Design, scheduling and reserve estimation of numerous underground mines.
    - Preparation of several scoping, prefeasibility, feasibility, and detailed design level studies for international mining projects.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying all the tests in Section 1.5 of NI 43-101.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

7. I am responsible for the preparation of Section 15 and Section 16 (with the exception of Sections 16.1, 16.2, 16.3, 16.4, 16.8, and 16.10) of NI43-101. I am also responsible for the associated portions of sections 1 and 25 to 27 of the Technical Report.
8. I personally did not visit the property that is the subject to the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th of October 2025, Penryn, Cornwall, United Kingdom.

"Original Signed and Sealed on file"
Dr. Lewis Meyer, ACSM, MCSM, BEng
MSc, PhD, CEng, FIMMM, QMR, MCIM
Associate Principal Engineer
Global Mine Design Ltd.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Phil Earl, ACSM BEng (Hons), MEng(sc), MAusIMM (CP) MIEAust,* do hereby certify:

1. I am Technical Director with Global Mine Design Ltd., with a business address of Tremough Innovation Centre, Penryn, Cornwall, TR10 9TA, UK.
2. I am a graduate of Camborne School of Mines with a Bachelor with Honours in Industrial Geology in 1999, and a graduate of Western Australia School of Mines with a Masters of Engineering Science in Mining Geomechanics in 2010.
3. I am a member in good standing with the Australasian Institute of Mining and Metallurgy and registered as a Chartered Professional Member (MAusIMM (CP)), licence number 991899. I am also a member in good standing with the Australian Institute of Engineers (#3115997).
4. I have worked either as a mining engineer or a geotechnical engineer in the civils and mining industries, continuously in my 26 years since graduation. I have worked on similar projects to the Loma Larga Project in Ecuador; my experience for the purpose of the Technical Report includes:
    - Geotechnical characterisation and design of numerous operating underground mines and mining projects
    - Mine design and scheduling of numerous mines and mining projects
    - Preparation of several scoping, prefeasibility, feasibility, and detailed design level studies for international mining projects.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying all the tests in Section 1.5 of NI 43-101.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

7. I am responsible for the preparation of Sections 16.1, 16.2, 16.3, 16.4, and 16.8. I am also responsible for the associated portions of 1 and 25 to 27 of the Technical Report.
8. I visited the property that is the subject of the Technical Report between August 21 and 24, 2023.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th of October 2025, Penryn, Cornwall, United Kingdom

"Original Signed and Sealed on file"
Phil Earl, ACSM BEng (Hons), MEng(sc),
MAusIMM (CP) MIEAust
Technical Director
Global Mine Design Ltd.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report - Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Sean McCarthy, P.Eng., PMP,* do hereby certify:

1. I am Senior Mining Advisor with Global Mine Design Ltd., with a business address of Tremough Innovation Centre, Penryn, Cornwall, TR10 9TA, UK.
2. I am a graduate of Queen's University, Kingston, Canada with a Bachelor of Applied Science, Mining Engineering in 1990.
3. I am a member in good standing with Professional Engineers Ontario and registered as a Professional Engineer, license number 100225173.
4. My relevant experience includes 30 years of experience in Mine Operations and Mine Engineering. I have worked on similar projects to the Loma Larga Project in Ecuador; my experience for the purpose of the Technical Report includes:
    - 12 Mine Engineering
    - 7 years Mine Operations
    - 5 years Capital Projects
    - 6 years Consulting
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying all the tests in Section 1.5 of NI 43-101.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

7. I am responsible for the preparation of some portions of Sections 21.1.8 and 21.2.4. I am also responsible for the associated portions of 1 and 25 to 27 of the Technical Report.
8. I visited the property that is the subject of the Technical Report on August 21 and 24, 2023.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th of October 2025, Camden East, Canada

"Original Signed and Sealed on file"

Sean McCarthy, P.Eng., PMP
Senior Mining Advisor
Global Mine Design Ltd.

Registered office:
Tremough Innovation Centre, Penryn,
Cornwall, TR10 9TA, UK.

Phone & Fax
Office (UK): +44 (0)1326 567 484

Online
info@globalminedesign.com
https://globalminedesign.com

Global Mine Design is a Limited
Company registered in England and Wales,
Company number 08665416
VAT registration number: 237 227 315

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Carl John Burkhalter, P.E.*, do hereby certify:

1. I am partner of NewFields Mining & Technical Services LLC, 9540 Maroon Circle, Suite 300, Englewood, CO 80112, USA.
2. I graduated with a Bachelor of Science in Mining Engineering in 1984 and a Master of Science in Civil and Environmental (Geotechnical) Engineering in 1987 from the University of Wisconsin- Madison.
3. I am a licensed professional engineer in good standing in Colorado in the area of Civil Engineering, P.E. (No. 29447). I am also licensed as a professional civil engineer in State of Arizona (No. 34925) and the State of South Carolina (No. 31243).
4. I have worked as civil engineer for a total of 38 years. I have worked on similar projects to the Loma Larga Project in Ecuador; my experience for the purpose of the Technical Report includes:
    - Design of over 20 Tailings Storage Facilities including performing Engineer-of-Record ("EoR") duties on 10 over my career. Currently EoR for a very similar Filtered Tailings Storage facility in Mexico.
    - Participation and author of several NI 43-101 Technical Reports related to Tailings Facilities and Water Management.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the current requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Sections 18.7, 18.8 and 18.9. I am also responsible for the associated portions of 1.7, 25.6 and 26.6 along the relevant portions of 27 of the Technical Report.

8. I visited the property that is the subject to the Technical Report April 11, 2022.

9. I have had prior involvement with the property that is the subject of the Technical Report as a tailings facility designer under the previous owner.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025.

"Original Signed and Sealed on file"
Carl John Burkhalter, P.E.
Principal Engineer, Partner
NewFields Mining & Technical Services LLC

Paterson & Cooke
KNOWLEDGE | EXPERIENCE | PASSION
Sunrise Park, Sunrise Circle, Ndabeni, 7405, Cape Town, South Africa
PO Box 845, Maitland, 7404, South Africa. **T:** +27 21 531 2955
CapeTown@PatersonCooke.com www.PatersonCooke.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") by DPM Metals Inc. (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, Leslie Correia, Pr.Eng., do hereby certify:

1. I am a Senior Engineer at Paterson & Cooke, with a Corporate business address of Sunrise Park, Sunrise Circle, Ndabeni, 7405, Cape Town, South Africa.
2. I am a graduate of the University of Stellenbosch, Stellenbosch, Western Province South Africa in 2005 with a bachelor's degree in chemical engineering.
3. I am a member in good standing of the Engineering Council of South Africa (ECSA), Registration # 20130236.
4. I have worked continuously as a Process Engineer for a total 18 years continuously since my graduation. My relevant work experience includes:
    - Designed, Implemented and Commissioned numerous paste backfill system in Africa and the Americas similar in process to the Loma Larga Project.
    - Conducted site audits and provided technical assistance for mining operations in Africa and the Americas.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of some portions of Sections 16.10 and 18.6. I am also responsible for the associated portions of 1.4.6 of the Technical Report.
8. I have not visited the property that is the subject to the Technical Report.

PATERSON & COOKE
1351-C Kelly Lake Road, Unit #2, Sudbury, Ontario P3E 5P5, Canada
Tel: +1 705 222 1720, Fax: +1 705 222 1719
Email: Sudbury@PatersonCooke.com, URL: www.PatersonCooke.com

9. I have had prior involvement with the property that is the subject of the Technical Report.
    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 and issued Report Date: November 29, 2021, Document # C6090-PM-REP-001-00, prepared for Dundee Precious Metals Inc.
    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 (issued Original Report Date: April 14, 2020 and Amended Report Date: June 10, 2020), Document # J01834-PM-REP-002, Rev. 1, prepared for INV Metals Inc.
    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of January 11, 2019 and issued Report Date: January 14, 2019, Document # J01834-PM-REP-002, Rev. 0, prepared for INV Metals Inc.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th of October 2025, Cape Town, South Africa

"Original Signed and Sealed on file"
Leslie Correia, Pr.Eng.
Senior Engineer
Paterson & Cooke

SLR Consulting (Canada) Ltd
55 University Avenue, Suite 501, Toronto, ON M5J 2H7

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Luis Vasquez, M.Sc., P. Eng.*, of Toronto, Ontario, Canada, do hereby certify:

1. I am a Principal Hydrotechnical Engineer – Water Resources Engineering with SLR Consulting (Canada) Ltd, of Suite 501, 55 University Ave., Toronto, ON M5J 2H7.
2. I am a graduate of Universidad de Los Andes, Bogotá, Colombia, in 1998 with a Bachelor of Science degree in Civil Engineering, and in 1999 with a Master of Science degree in Water Resources Engineering.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100210789).
4. I have worked as a civil engineer for a total of 26 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as an environmental consultant on numerous mining operations and projects for due diligence and regulatory requirements.
    - Preparation of environmental impact assessments for mining projects for regulatory approval in Canada and South America.
    - Preparation of mine closure plans for mining projects in Canada and Perú.
    - Preparation of several scoping, prefeasibility, feasibility, and detailed design level studies for projects in North America, South America, the Caribbean, and Asia with a focus on planning, design, and safe operation of water management systems and waste disposal facilities.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Corporation applying the test set out in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Section 20. I am also responsible for the associated portions of 1, 25, 26 and 27 of the Technical Report.
8. I did not visit the property that is the subject of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with N1 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025, Toronto, Canada

"Original Signed and Sealed on file"
Luis Vasquez, M.Sc., P. Eng.
Principal Hydrotechnical Engineer
SLR Consulting (Canada) Ltd.

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the report entitled "*Technical Report – Feasibility Study Update, Loma Larga Project, Azuay Province, Ecuador*", dated October 29, 2025 with an effective date of May 28, 2025 (the "Technical Report") prepared for DPM Metals Inc. (formerly known as Dundee Precious Metals Inc.) (the "Corporation") on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

I, *Daniel M. Gagnon, P. Eng.,* do hereby certify that:

1. I am Senior Vice President East Canada and Mining, with DRA Americas Inc., located at 555 Blvd René-Lévesque West, 6th Floor, Montreal, Quebec, Canada H2Z 1B1.
2. I am a graduate from "École Polytechnique de Montréal" with Bachelor of Mining Engineer in 1995.
3. I am a member in good standing with "*Ordre des Ingénieurs du Québec*" and registered as a Professional Engineer (# 118521).
4. I have worked in the mining industry continuously since 1995 and as a licensed mining engineer since 1999.
5. I have worked on open pit mining projects in Africa, North and South America; my experience for the purpose of the Technical Report includes:
    - Over 28 years of consulting in iron ore mines including mine planning and Mineral Reserve estimation.
    - Technical assistance in Mining Engineering and Mineral Reserves estimation for a variety of active mine operations in various countries.
    - Preparation of Capital and Operating Costs for operating mines and studies
    - Preparation of financial models for mining projects
    - Participation in the preparation of multiple NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
7. I am independent of the Corporation applying all the tests in section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 19 and 22 and portions of Sections 1, 25 to 27 of the Technical Report.

9. I visited the property that is the subject of the Technical Report on August 1 to 2, 2017.

10. I have had prior involvement with the property that is the subject of the Technical Report:

    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 and issued Report Date: November 29, 2021, Document # C6090-PM-REP-001-00, prepared for Dundee Precious Metals Inc.
    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of April 8, 2020 (issued Original Report Date: April 14, 2020 and Amended Report Date: June 10, 2020), Document # J01834-PM-REP-002, Rev. 1, prepared for INV Metals Inc.
    - "NI 43-101 Feasibility Study Technical Report on the Loma Larga Project – Azuay Province, Ecuador" with an effective date of January 11, 2019 and issued Report Date: January 14, 2019, Document # J01834-PM-REP-002, Rev. 0, prepared for INV Metals Inc.

11. I have read National Instrument 43-101 and Form 43-101F and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 29th day of October 2025, in Montreal, Quebec.

"Original Signed and Sealed on file"
Daniel M. Gagnon, P. Eng.
Senior Vice President East Canada and Mining
DRA Americas Inc.