Preliminary Economic Assessment NI 43-101 Technical Report
Aurora Cu-Mo-Ag Mine Project

Calca Province, Peru

Effective Date: April 30, 2026

Issue Date: October 15, 2026

Prepared for:

DLP Resources Inc.

Prepared by:

Qualified Persons

Terre Lane, MMSA 01407QP, SME Registered Member 4053005, GRE
Todd Harvey, PhD, SME Registered Member 4144120, GRE
Hamid Samari, PhD, QP, MMSA 01519QP, Senior Geology, GRE
Larry Breckenridge, P.E., GRE
Josh Parsons, P.Eng., SRK
Jarek Jakubec, FIMMM, SRK
Carl Kottmeier, P.Eng., MBA, SRK
Brian Prosser, PE, SRK
Scott Elfen, P.E. and P.Eng., Ausenco
Kevin Murray, P.Eng., Ausenco

Date and Signature Page

This Technical Report on the Aurora Cu-Mo-Ag Mine Project is submitted to DLP Resources Inc. and is effective April 30, 2026.

The Qualified Persons and Responsible Report Sections are:

Qualified Person	Responsible for Report Sections
Terre Lane (GRE)	Sections 1.7, 1.8, 1.9, 1.9.1, 1.11, 1.13.3, 1.13.3, 2, 3, 14.1, 15, 16.1, 19, 21.1, 21.2.1, 21.2.7, 21.2.9, 21.2.10, 21.2.11, 21.2.12, 21.2.13, 21.3.1, 21.3.6, 21.3.7, 22, 23, 24, 25.3, 25.4, 25.9, 26.3, 26.4, 26.9, and 27
Dr. Todd Harvey (GRE)	Sections 1.6, 1.13.2, 13, 17 (except for 17.3.7), 21.2.3, 21.3.3, 25.3, 26.2, and 26.6
Dr. Hamid Samari (GRE)	Sections 1.2, 1.3, 1.4, 1.5, 1.13.1, 1.14, 5, 6, 9, 10, 11, 12 (except for 12.5), 25.1, 25.2, and 26.1
Larry Breckenridge (GRE)	Sections 1.1, 1.12, 1.13.7, 1.13.8, 4, 5, 18.5, 20, 21.2.8, 21.3.5, 25.7.3, 25.8, 25.10.4, 25.10.5, 26.7.3, and 26.8
Josh Parsons (SRK)	Section 1.9.2, 1.15.5.2, 16.2 (except for 16.2.1 and 16.2.3), 25.6.2, 25.6.3, and 26.5.2
Jarek Jakubec (SRK)	Sections 1.13.5.1, 16.2.1, 25.6.1, 25.10.1, and 26.5.1
Carl Kottmeier (SRK)	Section 16.2.3 (except for 16.2.3.2)
Brian Prosser (SRK)	Section 16.2.3.2
Scott Elfen (Ausenco)	Sections 1.10, 2.3.2, 1.13.6, 12.6.4, 17.3.7, 18.1, 18.2.1, 18.3.1-18.3.4, 18.4, 21.2.4, 21.2.5, 21.3.4, 25.8.1, 25.8.3, 25.12.4, and 26.7
Kevin Murray (Ausenco)	Sections 18.2.2, 18.3.5, 21.2.6, and 25.8.2

(Signed) "Terre Lane"_____ _10/15/2026_____
Signature Terre Lane Date

(Signed) "Dr. Todd Harvey"_____ _10/15/2026_____
Signature Dr. Todd Harvey Date

(Signed) "Dr. Hamid Samari"_____ _10/15/2026_____
Signature Dr. Hamid Samari Date

(Signed) "Larry Breckenridge"_____ _10/15/2026_____
Signature Larry Breckenridge Date

(Signed) "Josh Parsons"_____ _10/15/2026_____
Signature Josh Parsons Date

(Signed) "Jarek Jakubec"_____ _10/15/2026_____
Signature Jarek Jakubec Date

(Signed) "Carl Kottmeier" 10/15/2026
Signature Carl Kottmeier Date

(Signed) "Brian Prosser" 10/15/2026
Signature Brian Prosser Date

(Signed) "Scott Elfen" 10/15/2026
Signature Scott Elfen Date

(Signed) "Kevin Murray" 10/15/2026
Signature Kevin Murray Date

DISCLAIMER

This report was prepared as a Technical Report (as such term is defined in National Instrument 43-101 - *Standards of Disclosure for Mineral Projects* ("NI 43-101")) for DLP Resources Inc (the "Company") by Global Resource Engineering (GRE), SRK Consulting (SRK), and Ausenco. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in GRE's, SRK's, and Ausenco's 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 is intended for use by the Company subject to the terms and conditions of its contracts with GRE, SRK, and Ausenco and relevant securities legislation. The contracts between the Company and GRE, SRK, and Ausenco permit the Company to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party are at that party's sole risk. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

TABLE OF CONTENTS

1.0 SUMMARY 26
1.1 Property Description and Ownership 27
1.2 History 28
1.3 Geology, Mineralization, and Deposit Type 28
1.4 Exploration, Drilling, and Sampling 30
1.4.1 Exploration 30
1.4.2 Drilling 31
1.4.3 Sample Preparation, Analyses, and Security 32
1.5 Data Verification and Quality Assurance/Quality Control 33
1.6 Mineral Processing and Metallurgical Testing 34
1.7 Previous Resource Estimates and Economic Models 35
1.8 Mineral Resource Estimate 35
1.9 Mining Methods 37
1.9.1 Open Pit 38
1.9.2 Underground 38
1.10 Infrastructure 40
1.10.1 Overview 40
1.10.2 Dry Stack Tailings Facility 41
1.10.3 Water Infrastructure and Water Availability 41
1.11 Economic Model Results 42
1.11.1 Results 42
1.11.2 Sensitivity Analyses 43
1.11.3 Conclusions of Economic Model 45
1.12 Environmental Issues, Permitting, Community Relations, and Closure 45
1.12.1 Environmental Setting 45
1.12.2 Geochemistry 46
1.12.3 Permitting 46
1.12.4 Closure 46
1.12.5 Community Relations 46
1.13 RECOMMENDATIONS 46
1.13.1 Geology 47
1.13.2 Metallurgy 48
1.13.3 Resource 48
1.13.4 Open Pit Mining 48
1.13.5 Underground Mining 49
1.13.6 Infrastructure 50
1.13.7 Water Infrastructure 51
1.13.8 Environment and Permitting 51
1.14 Total Cost for Recommended Program 53
2.0 INTRODUCTION 54

2.1 Terms of Reference 54
2.2 Purpose of Report 56
2.3 Site Visit and Details of Inspection 56
2.3.1 SRK 56
2.3.2 Ausenco 57
2.3.3 GRE 57
2.4 Sources of Information 58
2.5 Units 58
3.0 RELIANCE ON OTHER EXPERTS 59
4.0 PROPERTY LOCATION AND DESCRIPTION 60
4.1 Location 60
4.1.1 Core Tenement Ownership 61
4.1.2 Other Tenement Ownership 62
4.1.3 Manu National Park Buffer Zone 65
4.2 Peruvian Regulatory Framework 65
4.2.1 Overview 65
4.2.2 Exploration, Mining, and Surface Rights 66
4.2.3 Permitting 66
4.2.4 Royalties and Taxes 67
4.2.5 Additional Permits and Licenses 68
4.3 Environmental Permits 68
4.4 Royalties and Encumbrances 69
4.5 Environmental Liabilities 69
4.6 Risks and Additional Factors 69
4.7 Conclusions 69
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 70
5.1 Accessibility 70
5.2 Physiography 71
5.2.1 General Physiographic Regions within the Project Site 72
5.2.2 Local Ecology and Vegetation 74
5.2.3 Physiographic Impacts on Project Infrastructure and Planning 75
5.3 Surface and Mineral Rights 76
5.4 Climate 76
6.0 HISTORY 78
6.1 Introduction 78
6.2 Property Ownership 78
6.3 Overview of Exploration and Drilling 78
6.4 Surface Sampling 79
6.5 Surface Geophysics 79
6.6 Drill Testing 79
6.7 Historical Resources and Reserves Estimates 79

6.8 Production ... 79
7.0 GEOLOGIC SETTING AND MINERALIZATION ... 80
7.1 Regional Geology and Metallogeny ... 80
7.1.1 Tectonic Framework ... 80
7.1.2 Property Geology ... 81
7.2 Local Geology ... 82
7.2.1 Sedimentary Rocks (Sandia Formation) ... 82
7.2.2 Subvolcanic and Intrusive Rocks ... 83
7.2.3 Brecciation ... 86
7.2.4 Hydrothermal Alteration ... 87
7.2.5 Three-Dimensional Geological Model ... 89
7.3 Mineralization ... 92
7.3.1 Overview ... 92
7.3.2 Dimensions ... 93
7.4 QP Comments on Geology and Mineralization ... 93
8.0 DEPOSIT TYPE ... 94
8.1 QP Comments on Deposit Type ... 95
9.0 EXPLORATION ... 96
9.1 Pre-DLP Exploration ... 96
9.1.1 Surface Sampling ... 96
9.1.2 Geophysical Surveys ... 97
9.2 DLP Exploration ... 100
9.2.1 DLP Surface Sampling ... 100
9.3 QP Comments on Exploration Programs ... 103
9.3.1 Pre-DLP Exploration ... 103
9.3.2 DLP Exploration ... 103
9.3.3 Overall QP Opinion ... 103
10.0 DRILLING ... 104
10.1 Introduction ... 104
10.2 Drilling Program ... 105
10.2.1 Pre-DLP (Before 2022) ... 105
10.2.2 DLP (2022–2025) ... 107
10.3 Drilling Method (2022-2025) ... 112
10.4 Logging Procedure (2022-2025) ... 114
10.5 Core Recovery (2022-2025) ... 115
10.6 Collar Survey (2022-2025) ... 115
10.7 Down-Hole Surveys ... 116
10.8 QP Comments on Section 10 Drilling ... 117
11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY ... 118
11.1 Introduction ... 118
11.2 Sampling ... 118

11.3 Sample Preparation ... 119
11.4 Analytical Procedure ... 119
11.5 Quality Assurance and Quality Control ... 120
11.5.1 Blank Samples ... 120
11.5.2 Certified Reference Materials ... 121
11.5.3 Duplicate Samples ... 121
11.6 Database Management ... 121
11.7 Density ... 122
11.8 Sample Security ... 122
11.9 QP Comments on Section 11 Sample Preparation, Analysis, and Security ... 123
12.0 DATA VERIFICATION ... 124
12.1 Introduction ... 124
12.2 Historical Data Verification ... 124
12.3 Verification of DLP Database (2022-2025) ... 124
12.4 Verification of DLP's In-House QA/QC Program (2022–2025) ... 125
12.4.1 QA/QC Review (2022) ... 125
12.4.2 QA/QC Review (2023) ... 132
12.4.3 QA/QC Review (2024) ... 138
12.4.4 QA/QC Review (2025) ... 150
12.5 Field Visit, Geological Data Verification, and Check Assays ... 163
12.5.1 Collar Coordinate Validation ... 163
12.5.2 Geological Data Verification and Interpretation ... 167
12.5.3 Geological Logging Accuracy ... 168
12.5.4 Check Assay ... 176
12.6 Data Verification QP Opinion on Adequacy ... 181
12.6.1 Dr. Hamid Samari (GRE) ... 181
12.6.2 Dr. Todd Harvey (GRE) ... 182
12.6.3 Ms. Terre Lane (GRE) ... 182
12.6.4 Scott Elfen (Ausenco) ... 183
12.6.5 Verifications by SRK ... 183
13.0 MINERAL PROCESSING AND METALLURGICAL TESTING ... 184
13.1 Metallurgical Sampling ... 184
13.1.1 2023 Samples ... 184
13.1.2 2025 Samples ... 186
13.2 2024 Metallurgical Program ... 187
13.2.1 Mineralogy ... 187
13.2.2 Metallurgical Testing ... 193
13.2.3 Metallurgical Testing – M2 ... 197
13.2.4 Potentially Deleterious Elements in Concentrates ... 198
13.3 2025 Metallurgical Program ... 199
13.3.1 Mineralogy ... 203
13.3.2 Comminution ... 203

13.3.3 Metallurgical Testing 204
13.3.4 Potentially Deleterious Elements In Concentrates 206
13.3.5 Variability Testing 207
13.4 Recoveries 209
13.5 Recommendations 213
13.6 Sample Drill Holes 214
14.0 MINERAL RESOURCE ESTIMATE 230
14.1 Data Used 230
14.1.1 Drillhole Database 230
14.1.2 Topography 231
14.1.3 Density 232
14.2 Domain Modeling 232
14.2.1 Geological Domaining 232
14.2.2 Geological Modeling 236
14.3 Estimation Domaining 242
14.4 Statistics and Compositing 244
14.5 Evaluation of Outliers 246
14.6 Density 247
14.7 Variography 248
14.8 Block Model Parameters 250
14.9 Grade Estimation 250
14.10 Resource 252
14.10.1 Block Model Validation 252
14.11 Mineral Resource Classification 265
14.11.1 Data Completeness and Verification 265
14.11.2 Geological Confidence 265
14.11.3 Final Classification 265
14.12 Mineral Resource Statement 266
14.12.1 Open Pit 267
14.12.2 Underground 272
15.0 MINERAL RESERVE ESTIMATES 276
16.0 MINING METHODS 277
16.1 Open Pit 278
16.1.1 Site Layout and Pit Access 278
16.1.2 Whittle Pit Shell Analysis 279
16.1.3 Pit Design 280
16.1.4 Pit Resources 282
16.1.5 Mine Schedule 283
16.1.6 Open Pit High Wall Access 284
16.1.7 Waste Rock Storage Facility 285
16.1.8 Stockpile 286

16.1.9 Site Plans by Year ... 287
16.2 Underground ... 295
16.2.1 Geotechnical Basis of Design ... 295
16.2.2 Underground Mine Design ... 309
16.2.3 Underground Infrastructure Design ... 320
16.2.4 Underground Life-of-Mine Schedule ... 339
16.2.5 Underground Mobile Equipment ... 342
16.2.6 Underground Mine Personnel ... 344
16.2.7 Comments on Mining Methods (Risks and Opportunities) ... 344
16.3 Mine Dewatering ... 346
16.3.1 Groundwater Dataset ... 346
16.3.2 Conceptual Model ... 346
16.3.3 Model Methods ... 348
16.3.4 Model Results ... 348
16.3.5 Closure Phase Dewatering ... 349
17.0 RECOVERY METHODS ... 350
17.1 Introduction ... 350
17.2 Process Design Criteria ... 351
17.3 Process Description ... 357
17.3.1 Primary Crushing ... 357
17.3.2 Comminution Circuit ... 357
17.3.3 Flotation Circuit ... 358
17.3.4 Concentrate Dewatering ... 359
17.3.5 Tails Thickening ... 359
17.3.6 Process Water ... 359
17.3.7 Filter Plant ... 360
17.4 Risks ... 360
17.5 Opportunities ... 360
17.6 Recommendations ... 360
18.0 PROJECT INFRASTRUCTURE ... 361
18.1 Introduction ... 361
18.2 Off Site Infrastructure ... 363
18.2.1 Site Access ... 363
18.2.2 High Voltage Power Supply ... 363
18.3 On-Site Infrastructure ... 364
18.3.1 Site Preparation ... 364
18.3.2 On-Site Roads ... 364
18.3.3 Accommodation ... 364
18.3.4 Rope Conveyor ... 364
18.3.5 Power and Electrical ... 364
18.4 Tailings Facility ... 364
18.5 Water and Water Infrastructure ... 373

18.5.1 Methodology ... 374
18.5.2 Time Periods ... 374
18.5.3 Water Balance ... 375
18.5.4 Input Parameters and Model Methodology ... 375
18.5.5 Results ... 376
18.5.6 Makeup Water Demand ... 378
18.5.7 Water Management Conclusions ... 380
19.0 MARKET STUDIES AND CONTRACTS ... 382
20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT ... 383
20.1 Environmental Conditions, and Environmental Baseline ... 383
20.2 Environmental Studies ... 383
20.2.1 Baseline Air Quality ... 384
20.2.2 Baseline Noise ... 384
20.2.3 Baseline Soil and Soil Chemistry ... 384
20.2.4 Baseline Surface Water Flows (Quantity) ... 384
20.2.5 Surface Water Flow and Surface Water Study ... 385
20.2.6 Baseline Surface Water Quality ... 389
20.2.7 Baseline Environmental Geochemistry Studies ... 390
20.2.8 Baseline Biological Studies ... 395
20.3 Requirements for Waste and Tailings Disposal ... 395
20.3.1 Water Treatment ... 396
20.3.2 Monitoring ... 396
20.3.3 Waste Management and Closure ... 396
20.4 Project Permitting Requirements ... 396
20.4.1 Permits to Advance to Feasibility Study Stage ... 397
20.5 Social and Community Relations ... 403
20.5.1 Description of the Community ... 403
20.5.2 Community Relations ... 403
20.6 Closure ... 404
20.6.1 Regulatory Requirements ... 404
20.6.2 PEA-Level Closure Design Concepts ... 405
20.6.3 Water Management ... 407
20.6.4 Water Treatment ... 408
20.6.5 Post-Closure Monitoring ... 408
20.6.6 Unit Rate Estimates ... 408
20.6.7 Closure Cost Estimate ... 408
21.0 CAPITAL AND OPERATING COSTS ... 409
21.1 Basis of Estimate and Accuracy Classification ... 409
21.2 Capital Cost Estimate ... 409
21.2.1 Open Pit Mining ... 411
21.2.2 Underground Mining ... 412
21.2.3 Process Plant ... 414

21.2.4 Dry Stack Tailings Facility 414
21.2.5 Site Infrastructure 415
21.2.6 Power Distribution 415
21.2.7 Facility Infrastructure 415
21.2.8 Water Treatment Plant and Closure 416
21.2.9 Indirect Costs 416
21.2.10 Owner's Costs 417
21.2.11 Sustaining and Development Capital 418
21.2.12 Working Capital 418
21.2.13 Contingency 418
21.3 Operating Cost Estimate 418
21.3.1 Open Pit Mining 418
21.3.2 Underground Block Cave Mining 420
21.3.3 Concentrator Plant 421
21.3.4 Dry Stack Tailings Facility 421
21.3.5 Water Treatment 421
21.3.6 General and Administrative 424
21.3.7 Contingency 424
22.0 ECONOMIC ANALYSIS 425
22.1.1 Assumptions 425
22.1.2 Taxes 426
22.2 Results 426
22.3 Sensitivity Analyses 427
22.4 Life of Mine Cumulative Free Cash Flow 427
22.5 Conclusions of Economic Model 427
23.0 ADJACENT PROPERTIES 433
24.0 OTHER RELEVANT DATA AND INFORMATION 434
25.0 INTERPRETATIONS AND CONCLUSIONS 435
25.1 Geology and Mineralization 435
25.2 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation 435
25.3 Metallurgy 436
25.4 Resource Estimation 436
25.5 Open Pit Mining 437
25.6 Underground Mining 437
25.6.1 Underground Geotechnical 437
25.6.2 Underground Mining Plan 437
25.6.3 Underground Mining Capital and Operating Cost 438
25.7 Recovery 438
25.8 Infrastructure 439
25.8.1 Overview 439
25.8.2 Power Supply 439

25.8.3 Tailings Management ... 439
25.8.4 Water Supply and Water Management ... 439
25.9 Market Studies and Contracts ... 440
25.10 Environmental Issues ... 440
25.11 Economics ... 440
25.12 Risks and Opportunities ... 440
25.12.1 Underground Mining ... 440
25.12.2 Concentrator ... 441
25.12.3 Infrastructure ... 441
25.12.4 Tailings Management Risks ... 441
25.12.5 Water Risks ... 442
25.12.6 Environmental Risks ... 442
25.12.7 Socal Risks ... 442
25.13 Opportunities ... 442
26.0 RECOMMENDATIONS ... 443
26.1 Geology ... 443
26.2 Metallurgy ... 444
26.3 Resource ... 444
26.4 Open Pit Mining ... 444
26.5 Underground Mining ... 445
26.5.1 Geotechnical ... 445
26.5.2 Underground Mining ... 446
26.6 Concentrator ... 446
26.7 Infrastructure ... 446
26.7.1 Infrastructure Geotechnical ... 446
26.7.2 DSTF and TCSF Analyses and Studies ... 447
26.7.3 Water Infrastructure ... 448
26.8 Environment and Permitting ... 448
26.8.1 Baseline Data Collection ... 448
26.8.2 Environmental Studies ... 449
26.8.3 Permitting ... 450
26.8.4 Closure ... 450
26.9 Total Cost for Recommended Program ... 450
27.0 REFERENCES ... 451
CERTIFICATE OF QUALIFIED PERSON ... 454
CERTIFICATE OF QUALIFIED PERSON ... 456
CERTIFICATE OF QUALIFIED PERSON ... 457
CERTIFICATE OF QUALIFIED PERSON ... 458

LIST OF FIGURES

Figure 1-1: Aurora Project Underground Cave Production Schedule 39
Figure 1-2: Aurora Project Site Layout 40
Figure 1-3: Aurora Project Post-Tax NPV@8% Sensitivities 44
Figure 1-4: Aurora Project Post-Tax IRR Sensitivities 45
Figure 4-1: Aurora Property Location 60
Figure 4-2: Map of the Core "Aurora Concession" and other Concessions on the Property 61
Figure 5-1: Access Road from Cusco to the Aurora Project 70
Figure 5-2: Access Road to the Aurora Project Area 71
Figure 5-3: Site Topography 72
Figure 7-1: Topographic Map of Peru Showing the Aurora Project and Structural Domains of Peru 81
Figure 7-2: Drill-Hole Distribution, Plan View (A) and Oblique View (B) 90
Figure 7-3: Three-Dimensional Geological Model: Plan View (A), Oblique Views (B), 3D view of CPY & MPY (C), and Cross Sections: at 190000E (D), at 190200E (E), and at 190400E (F) looking East 91
Figure 8-1: Schematic Section, Porphyry Copper Deposit 94
Figure 9-1: Focus Surface Geochemistry – Copper 96
Figure 9-2: Focus Surface Geochemistry – Molybdenum 97
Figure 9-3: Focus IP Survey Conducted by Fugro 98
Figure 9-4: Line 10800 Chargeability – Long Section 99
Figure 9-5: Line 10800 Resistivity – Long Section 99
Figure 9-6: Chargeability Level Plan -200 m 100
Figure 9-7: DLP - Anomalous Copper in Rock Samples in NE Sector 101
Figure 9-8: DLP - Anomalous Molybdenum in Rock Samples in NE Sector 101
Figure 9-9: DLP - Anomalous Silver in Rock Samples in NE Sector 102
Figure 10-1: Location of Approved Drill Platforms 108
Figure 10-2: DLP Drillhole Locations by Year (2022-2025) 109
Figure 10-3: Distribution of Copper and Molybdenum Grades within the Aurora Claim Area 111
Figure 10-4: Visual Correlation between Copper Grades (solid bars) and Molybdenum Grades within the Aurora Claim Area 112
Figure 12-1: 2022 Drilling Campaign, Cu Blank Results 126
Figure 12-2: 2022 Drilling Campaign, Mo Blank Results 126
Figure 12-3: 2022 Drilling Campaign, Ag Blank Results 127
Figure 12-4: CDN-ME-1409, Cu from the 2022 Drilling Campaign 128
Figure 12-5: CDN-ME-1409, Ag from the 2022 Drilling Campaign 128
Figure 12-6: CDN-ME-1704, Cu from the 2022 Drilling Campaign 129
Figure 12-7: CDN-ME-1704, Ag from the 2022 Drilling Campaign 129
Figure 12-8: Cu Coarse Reject Duplicate Samples from the 2022 Drilling Campaign 130
Figure 12-9: Mo Coarse Reject Duplicate Samples from the 2022 Drilling Campaign 131
Figure 12-10: Ag Coarse Reject Duplicate Samples from the 2022 Drilling Campaign 131
Figure 12-11: 2023 Drilling Campaign, Cu Blank Results 132
Figure 12-12: 2023 Drilling Campaign, Mo Blank Results 133

Figure 12-13: 2023 Drilling Campaign, Ag Blank Results 133
Figure 12-14: CDN-ME-33, Cu from the 2023 Drilling Campaign 134
Figure 12-15: CDN-ME-33, Mo from the 2023 Drilling Campaign 135
Figure 12-16: CDN-ME-33, Ag from the 2023 Drilling Campaign 135
Figure 12-17: Cu Coarse Reject Duplicate Samples from the 2023 Drilling Campaign 136
Figure 12-18: Mo Coarse Reject Duplicate Samples from the 2023 Drilling Campaign 137
Figure 12-19: Ag Coarse Reject Duplicate Samples from the 2023 Drilling Campaign 137
Figure 12-20: 2024 Drilling Campaign, Cu Blank Results 138
Figure 12-21: 2024 Drilling Campaign, Mo Blank Results 139
Figure 12-22: 2024 Drilling Campaign, Ag Blank Results 139
Figure 12-23: CDN-ME-33, Cu from the 2024 Drilling Campaign 141
Figure 12-24: CDN-ME-33, Mo from the 2024 Drilling Campaign 141
Figure 12-25: CDN-ME-33, Ag from the 2024 Drilling Campaign 142
Figure 12-26: CDN-ME-1409, Cu from the 2024 Drilling Campaign 142
Figure 12-27: CDN-ME-1409, Ag from the 2024 Drilling Campaign 143
Figure 12-28: PORF-34, Cu from the 2024 Drilling Campaign 143
Figure 12-29: PORF-34, Mo from the 2024 Drilling Campaign 144
Figure 12-30: PORF-34, Ag from the 2024 Drilling Campaign 144
Figure 12-31: PORF-35, Cu from the 2024 Drilling Campaign 145
Figure 12-32: PORF-35, Mo from the 2024 Drilling Campaign 145
Figure 12-33: PORF-35, Ag from the 2024 Drilling Campaign 146
Figure 12-34: PORF-39, Cu from the 2024 Drilling Campaign 146
Figure 12-35: PORF-39, Mo from the 2024 Drilling Campaign 147
Figure 12-36: PORF-39, Ag from the 2024 Drilling Campaign 147
Figure 12-37: Cu Coarse Reject Duplicate Samples from the 2024 Drilling Campaign 148
Figure 12-38: Mo Coarse Reject Duplicate Samples from the 2024 Drilling Campaign 149
Figure 12-39: Ag Coarse Reject Duplicate Samples from the 2024 Drilling Campaign 149
Figure 12-40: 2025 Drilling Campaign, Cu Blank Results 151
Figure 12-41: 2025 Drilling Campaign, Mo Blank Results 151
Figure 12-42: 2025 Drilling Campaign, Ag Blank Results 152
Figure 12-43: DLP-01, Cu from the 2025 Drilling Campaign 153
Figure 12-44: DLP-01, Mo from the 2025 Drilling Campaign 154
Figure 12-45: DLP-01, Ag from the 2025 Drilling Campaign 154
Figure 12-46: PORF-34, Cu from the 2025 Drilling Campaign 155
Figure 12-47: PORF-34, Mo from the 2025 Drilling Campaign 155
Figure 12-48: PORF-34, Ag from the 2025 Drilling Campaign 156
Figure 12-49: PORF-35, Cu from the 2025 Drilling Campaign 156
Figure 12-50: PORF-35, Mo from the 2025 Drilling Campaign 157
Figure 12-51: PORF-35, Ag from the 2025 Drilling Campaign 157
Figure 12-52: PORF-39, Cu from the 2025 Drilling Campaign 158
Figure 12-53: PORF-39, Mo from the 2025 Drilling Campaign 158
Figure 12-54: PORF-39, Ag from the 2025 Drilling Campaign 159
Figure 12-55: PORF-43, Cu from the 2025 Drilling Campaign 159

Figure 12-56: PORF-43, Mo from the 2025 Drilling Campaign 160
Figure 12-57: Cu Coarse Duplicate Samples from the 2025 Drilling Campaign 161
Figure 12-58: Mo Coarse Duplicate Samples from the 2025 Drilling Campaign 162
Figure 12-59: Ag Coarse Duplicate Samples from the 2025 Drilling Campaign 162
Figure 12-60: Hazen Cu Results on Check Samples 179
Figure 12-61: Hazen Ag Results on Check Samples 180
Figure 12-62: Hazen Mo Results on Check Samples 181
Figure 13-1: Aurora Geological Model Showing the Locations of the Samples Used in the 2023 Metallurgical Test Program 185
Figure 13-2: Aurora Geological Model Showing the Locations of the Samples Used in the 2025 Metallurgical Test Program 186
Figure 13-3: Copper Deportment – M1 189
Figure 13-4: Copper Deportment – M2 189
Figure 13-5: Copper Deportment – M3 190
Figure 13-6: Average Chalcopyrite and Molybdenite Particle Sizes – M1 191
Figure 13-7: Average Chalcopyrite and Molybdenite Particle Sizes – M2 191
Figure 13-8: Average Chalcopyrite and Molybdenite Particle Sizes – M3 192
Figure 13-9: Chalcopyrite Liberation 192
Figure 13-10: Molybdenite Liberation 193
Figure 13-11: Bulk Cu-Mo Flotation Circuit, Followed by Cu-Mo Separation Circuit 194
Figure 13-12: Cu-Mo Separation Flowsheet 195
Figure 13-13: M2 Upgraded Flowsheet – Open Circuit Testing 198
Figure 13-14: Copper Recovery and Feed Grade of Variability and Master Composite Rougher Tests ... 208
Figure 13-15: Molybdenum Recovery and Feed Grade of Variability and Master Composite Rougher Tests 208
Figure 13-16: Variability and Locked Cycle Cleaner Test Comparison - Copper 209
Figure 13-17: Variability and Locked Cycle Cleaner Test Comparison - Molybdenum 209
Figure 13-18: Aurora Project Concentrator Flotation Recovery 212
Figure 13-19: Aurora Project Concentrator Feed Grades 213
Figure 14-1: Current and Previous Topographic Maps for the Aurora Deposit 232
Figure 14-2: DDH Assay Samples (UMAP) & Schematic Long Section Colored by Logged Geology 233
Figure 14-3: DDH Assay Samples (UMAP) & Schematic Long Section Colored by AMC Lithological Domain 234
Figure 14-4: Schematic Long Section Colored by Final Lithology Domain 235
Figure 14-5: Bivariate Plot of Ti and Nb Data for Porphyry Intrusions 236
Figure 14-6: Aurora Lithological Model Cross-Section at 8565775 Looking North 237
Figure 14-7: Aurora Lithological Model Cross-Section at 190125 Looking East 238
Figure 14-8: Drill-Hole Mineralization Domains, Plan View (A), and Oblique View (B) 240
Figure 14-9: Three-Dimensional Mineralization Model, Plan View (A), Oblique Views (B & C), and Cross Sections: at 190000 (D), at 190200 (E), and at 190400 (F) Looking East) 241
Figure 14-10: Aurora Deposit Copper Box & Whisker Plots by Estimation Domain 243
Figure 14-11: Aurora Deposit Molybdenum Box & Whisker Plots by Estimation Domain 243
Figure 14-12: Aurora Deposit Silver Box & Whisker Plots by Estimation Domain 244

Figure 14-13: Histogram of Aurora Raw Sample Lengths 245
Figure 14-14: Example of Cumulative Probability Plot (CPY Enriched Cu) 246
Figure 14-15: Aurora Deposit Cu Cross Section 253
Figure 14-16: Aurora Deposit Cu Long Section 254
Figure 14-17: Aurora Deposit Mo Cross Section 255
Figure 14-18: Aurora Deposit Mo Long Section 256
Figure 14-19: Aurora Deposit Ag Cross Section 257
Figure 14-20: Aurora Deposit Ag Long Section 258
Figure 14-21: Aurora Deposit Swath Plot X axis, CPY Primary Domain Cu 260
Figure 14-22: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Cu 261
Figure 14-23: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Cu 261
Figure 14-24: Aurora Deposit Swath Plot X axis, CPY Primary Domain Mo 262
Figure 14-25: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Mo 262
Figure 14-26: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Mo 263
Figure 14-27: Aurora Deposit Swath Plot X axis, CPY Primary Domain Ag 263
Figure 14-28: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Ag 264
Figure 14-29: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Ag 264
Figure 14-30: Aurora Deposit Resource Class Plan View 266
Figure 16-1: Aurora Project Site Features 278
Figure 16-2: Aurora Project Initial Haul Road and Equipment Pads 279
Figure 16-3: Aurora Project Open Pit Whittle Output 280
Figure 16-4: Aurora Project Designed Ultimate Pit 281
Figure 16-5: Aurora Project In-Pit Access Road 282
Figure 16-6: Aurora Project Open Pit Mine Schedule Summary 284
Figure 16-7: Aurora Project Upper Haul Road 285
Figure 16-8: Aurora Project Waste Rock Storage Facility Final Layout 286
Figure 16-9: Aurora Project Stockpile Peak Capacity Layout 287
Figure 16-10: Aurora Project Site Layout Year -3 288
Figure 16-11: Aurora Project Site Layout Year -2 289
Figure 16-12: Aurora Project Site Layout Year -1 290
Figure 16-13: Aurora Project Site Layout Year 1 291
Figure 16-14: Aurora Project Site Layout Year 2 292
Figure 16-15: Aurora Project Site Layout Year 3 293
Figure 16-16: Aurora Project Site Layout Year 4 294
Figure 16-17: Aurora Project Oblique 3D View (Looking Northeast) of the Drillholes with Geotechnical Parameters (RQD, FF/m, IRS, and UCS) 295
Figure 16-18: Aurora Project Cross-Section Showing the Main Lithologies of the 3D Model "Den_Domain Model" 297
Figure 16-19: Aurora Project Drillholes within the Area of Interest 298
Figure 16-20: Aurora Project Selected Drillholes Comparing the Intact Rock Strength Values Estimated on Site 298
Figure 16-21: Aurora Project Comparison of Logged Intact Rock Strength 299
Figure 16-22: Aurora Project Examples of Drop Test Results 299

Figure 16-23: Aurora Project Orientation of Discontinuities 300
Figure 16-24: Aurora Project Rock Mass Characterization Examples 301
Figure 16-25: Regional World Stress Map along with Aurora Property and Other Mines 302
Figure 16-26: Local World Stress Map..... 303
Figure 16-27: Aurora Project Caveability Assessement..... 304
Figure 16-28: BCF Predicted Fragmentation Curves for Different HODs..... 305
Figure 16-29: Empirical Subsidence Angle Chart 307
Figure 16-30: Aurora Project Subsidence Zoning Example..... 308
Figure 16-31: Block Cave Mining Method Example 310
Figure 16-32: Aurora Project Footprint Finder Results at 1,845 m Elevation (Plan View, 500 m Grid Spacing)..... 312
Figure 16-33: Aurora Project Underground Mine Design (Isometric View, Looking North)..... 314
Figure 16-34: Aurora Project Cave Footprint Layout (Plan View, 100-m Grid Spacing)..... 315
Figure 16-35: Aurora Project Typical Drawbell Layout (Section View, Looking North, 4-m Grid Spacing) 315
Figure 16-36: Aurora Project Typical Undercut Ring Layout (Section View, Looking North, 4-m Grid Spacing)..... 316
Figure 16-37: Aurora Project Drawbell Blasting Schedule (Plan View) 317
Figure 16-38: Aurora Project Undercut Ring Blasting Schedule (Plan View) 318
Figure 16-39: Aurora Project Mine Access Layout (Plan View, 500-m Grid Spacing)..... 319
Figure 16-40: FLSmidth TSUV (Mark V) Gyratory Crusher 321
Figure 16-41: Cross-section of Conveyor Load Showing Trough and Surcharge Angles 322
Figure 16-42: Aurora Project Conveyor Transport System (Isometric View)..... 323
Figure 16-43: Typical Transfer Arrangement 323
Figure 16-44: Aurora Project General Ventilation Circuit..... 324
Figure 16-45: Aurora Project Ramp Sump Arrangement..... 332
Figure 16-46: Aurora Project Dewatering Station Arrangement 333
Figure 16-47: Aurora Project Underground Lateral Development Schedule..... 340
Figure 16-48: Aurora Project Underground Cave Production Schedule..... 341
Figure 16-49: Aurora Project Underground Cave Production Grade Profiles..... 342
Figure 16-50: Mine Workings, Pit, and Mapacho River with 2D model Cross Section Location Map..... 347
Figure 17-1: Aurora Project Concentrator Process Flowsheet..... 351
Figure 18-1: Aurora Project Site Layout..... 362
Figure 18-2: Road Route from Cusco to the Aurora Project. 363
Figure 18-3: Aurora Project Dry Stack Tailings Facility 365
Figure 18-4: Aurora Project Dry Stack Tailings Facility 367
Figure 18-5: Aurora Project Tailings Contingency Storage Facility 370
Figure 18-6: Aurora Project Tailing Contingency Storage Facility 371
Figure 18-7: Aurora Project GoldSim Water Balance Outline..... 375
Figure 18-8: Aurora Project MIW that Needs to be Treated During the Late Development Phase..... 377
Figure 18-9: Aurora Project MIW Needing Treatment During the Block Cave 378
Figure 18-10: Aurora Project Makeup Water Needed from Sources Outside the Mestizamayo Drainage Basin 378

Figure 18-11: Aurora Project Closure Storage Surge Pond Results 380
Figure 20-1: Annual Hydrograph of the Rio Mapacho at Paracartambo (~50 Km Upstream from Aurora) 384
Figure 20-2: Aurora Site Hydrologic Basin Without Facilities 386
Figure 20-3: Aurora Project MIW by Source for Early Mine Life Stage During an Average Year 387
Figure 20-4: Aurora Project MIW by Source for Late Development Mine Stage During an Average Year 388
Figure 20-5: Aurora Project MIW by Source During the Block Cave Stage for an Average Rainfall Year . 388
Figure 20-6: Aurora Project Results of Water Quality Samples Collected by GRE 389
Figure 20-7: Aurora Project ABA Results, by Rock Type 393
Figure 20-8: Aurora Project Post Closure Block Cave Hydrologic System 395
Figure 22-1: Aurora Project Post-Tax NPV@8% Sensitivities 430
Figure 22-2: Aurora Project Post-Tax IRR Sensitivities 431
Figure 22-3: Aurora Project Cumulative Free Cash Flow 432

LIST OF PHOTOS

Photo 2-1: SRK Site Visit Photos 57
Photo 5-1: Upper Mestizamayo Drainage, with U-Shaped Valley 73
Photo 5-2: Lower Mestizamayo Drainage, with Steep V-Shaped Valley and Tributaries 73
Photo 5-3: Mapacho River at the Curve Around the Ore Deposit 74
Photo 5-4: Vegetation Patterns in the Mestizamayo Drainage 75
Photo 5-5: Mapacho River and Coropuna Hill 76
Photo 7-1: Sandia Formation 83
Photo 7-2: General View of the Early Porphyry 84
Photo 7-3: Early Porphyry, Core samples from Hole A23-005 84
Photo 7-4: Intermineral Porphyry, Core sample from Hole A24-014 85
Photo 7-5: Breccia Development in Country Rock 86
Photo 7-6: Breccia Development in Sandia Formation 87
Photo 7-7: Early Porphyry with Intermediate Argillic Alteration 88
Photo 7-8: Molybdenite & Quartz-Molybdenite + Pyrite + Chalcopyrite Veining 92
Photo 10-1: Access Roads and Drill Pads at Aurora 113
Photo 10-2: Completed Drillhole Collar 114
Photo 10-3: Collar Pick Up of Drillhole A22-003 116
Photo 12-1: Inspection of the Collar Coordinates, 2026 165
Photo 12-2: Topographic Setting of the Aurora Project and Drill Collars Selected for Field GPS Verification 166
Photo 12-3: Sandia Formation 167
Photo 12-4: Early Porphyry (MPY or DPY) 168
Photo 12-5: Core Storage in DLP's Facility in Calca, Peru 169
Photo 12-6: Visual Inspection of Selected Intervals from Hole A23-006 173
Photo 12-7: Visual Inspection of Selected Intervals from Hole A22-001 174
Photo 12-8: Visual Inspection of Selected Intervals from Hole A25-025 175

Photo 12-9: Selected Pulp Reject Samples for Check Assay 176
Photo 16-1: Outcrops of Early Porphyry (Left) and Hornfels – Sandia Formation (Right)..... 296
Photo 20-1: Aurora Project Photos of Observed Onsite Acidic Seep 391
Photo 20-2: DLP-Constructed Soccer Pitch in Parobamba, Peru 404

LIST OF TABLES

Table 1-1: Summary of Mineral Resources – AMC Technical Report 2025 35
Table 1-2: Aurora Project Pit-Constrained Statement of Mineral Resource 36
Table 1-3: Aurora Underground Mineral Resource..... 36
Table 1-4: Aurora Project Key Economic Indicators 42
Table 1-5: Estimated Costs to Complete the Aurora Project 2-Year Program Leading to PFS..... 53
Table 2-1: List of Contributing Authors 55
Table 4-1: Aurora Concession Boundaries..... 60
Table 4-2: Aurora Concession Terms..... 62
Table 4-3: DLP Concession Details..... 63
Table 5-1: Meteorological Data for the Challabamba Station 77
Table 6-1: Summary of Historical Drilling of the Aurora Concession 79
Table 9-1: Significant Results from DLP Rock Chip Sampling..... 102
Table 10-1: Summary of Drilling Completed on the Aurora Property..... 104
Table 10-2: Bear Creek Drillhole Locations..... 105
Table 10-3: Significant Drillhole Intercepts from the Bear Creek Drilling..... 105
Table 10-4: Vena Resources Drillhole Locations 106
Table 10-5: Significant Drillhole Intercepts from Vena Resources..... 106
Table 10-6: Approved Drill Platforms under DLP's Environmental Authority..... 107
Table 10-7: DLP Drillhole Specifications 109
Table 10-8: Summary of DLP Diamond Drilling by Year and Core Size (2022–2025) 113
Table 10-9: Summary of Downhole Survey Methods..... 117
Table 11-1: Summary of ALS Analytical Methods Used During the DLP Drilling Campaigns (2022–2025) 120
Table 11-2: Summary of QA/QC Samples Submitted During the DLP Drilling Campaigns (2022–2025).. 120
Table 11-3: Certified Reference Materials Used During the DLP Drilling Campaigns (2022–2025)..... 121
Table 12-1: Collar Coordinates Inspections 164
Table 12-2: Visual Inspection of Core Sample Intervals 169
Table 12-3: Check Samples Submitted to Hazen Labs..... 177
Table 12-4: Summary Table of Hazen Results with Original Assays..... 177
Table 13-1: Modal Mineralogy Results – M1 187
Table 13-2: Modal Mineralogy Results – M2 188
Table 13-3: Modal Mineralogy Results – M3 188
Table 13-4: Metallurgical Sample Assays – M1, M2, and M3 193
Table 13-5: Collector Selection Tests MC Tests 196
Table 13-6: Bulk Concentrate Flotation Results – LCT Testing..... 197
Table 13-7: Cu-Mo Separation Flotation Results – Open Circuit Testing 197

Table 13-8: Open Circuit Test Result Summary – Test No. 12 198
Table 13-9: Potentially Deleterious Elements in Concentrates 199
Table 13-10: Master Composites Head Assays 199
Table 13-11: Variability Sample Head Assays 201
Table 13-12: Grind Sizes Suggested by Mineral Liberation (AMICS) 203
Table 13-13: Ball Bond Work Index 204
Table 13-14: JK-SMC Test Results 204
Table 13-15: JK-SMC Summary 204
Table 13-16: Lock Cycle Rougher Test Summary 205
Table 13-17: Locked Cycle Cleaner Test Summary 205
Table 13-18: Open Circuit Copper-Molybdenum Separation Test Summary 206
Table 13-19: Penalty Elements 206
Table 13-20: Summary Rougher Concentrate Variability 207
Table 13-21: 2023 Sample Sources 214
Table 13-22: 2025 Sample Sources 217
Table 13-23: 2025 Variability Sample Sources 223
Table 14-1: Aurora Deposit Compositing Interval Statistics 245
Table 14-2: Aurora Deposit Estimation Clamping Summary 247
Table 14-3: Aurora Deposit Densities 247
Table 14-4: Aurora Deposit Variogram Parameters 249
Table 14-5: Block Model Parameters 250
Table 14-6: Aurora Deposit Estimation and Search Parameters 250
Table 14-7: Aurora Deposit Comparison of Composite Values to Grade Estimation Methods 259
Table 14-8: Aurora Deposit Pit Optimization Parameters 267
Table 14-9: Aurora Open Pit Mineral Resource 268
Table 14-10: Aurora Open Pit Mineral Resource Sensitivity to Cutoff Grade 269
Table 14-11: Aurora Out-of-Pit Mineral Resource 273
Table 14-12: Aurora Out-of-Pit Mineral Resource Sensitivity to Cutoff Grade 273
Table 16-1: Aurora Project Open Pit Geotechnical Design Parameters 280
Table 16-2: Aurora Project Pit Phase Resources 282
Table 16-3: Aurora Project Mine Schedule Summary 283
Table 16-4: Aurora Project Underground Geotechnical Data Availability 296
Table 16-5: Aurora Project In-situ Rock Mass Ratings 301
Table 16-6: Aurora Project Mining Rock Mass Ratings 301
Table 16-7: Aurora Project Development Profile Summary 319
Table 16-8: Aurora Project Conveyor Belt and Motor Sizing 322
Table 16-9: Aurora Project General Airflow Calculation 326
Table 16-10: Aurora Project Exhaust Fan Operating Point 328
Table 16-11: Aurora Project Ventilation Power and Operating Fan Numbers 329
Table 16-12: Aurora Project Underground Mine Mobile Equipment 343
Table 16-13: Groundwater Modeling Results 348
Table 17-1: Process Design Criteria 351
Table 18-1: DSTF Guidelines, Codes, and Standards 366

Table 18-2: Design Criteria for DSTF 366
Table 18-3: Aurora Project DSTF Stability Analysis Results 368
Table 18-4: Aurora Project Design Criteria for TCSF 370
Table 18-5: TCSF Stability Analysis Results 372
Table 18-6: Modeling Software and Their Usages in the Aurora Project 374
Table 20-1: Screening Guidelines for Acid Generation Potential Prediction 392
Table 20-2: Aurora Project Development, PFS/FS and Construction-Readiness Permits and Requirements 398
Table 20-3: Aurora Project Permits for Construction and Operation 400
Table 21-1: Aurora Project Capital Cost Summary ($millions) 410
Table 21-2: Aurora Project Open Pit Mining Capital Cost Summary ($millions) 411
Table 21-3: Aurora Project Underground Block Cave Mining Capital Cost Summary ($millions) 413
Table 21-4: Aurora Project Concentrator Capital Cost Summary ($million) 414
Table 21-5: Aurora Project Dry Stack Tailings Facility Capital Cost Summary ($millions) 414
Table 21-6: Aurora Project Site Infrastructure Capital Cost Summary ($millions) 415
Table 21-7: Aurora Project Power Supply Capital Cost Summary ($millions) 415
Table 21-8: Aurora Project Facility Infrastructure Capital Cost Summary ($millions) 416
Table 21-9: Aurora Project Water Treatment Plant and Closure Cost Summary ($millions) 416
Table 21-10: Aurora Project Indirect Costs Summary ($millions) 417
Table 21-11: Aurora Project Owner's Costs 417
Table 21-12: Aurora Project Operating Cost Summary 418
Table 21-13: Aurora Project Open Pit Mining Operating Cost Summary (million $) 419
Table 21-14: Aurora Project Underground Block Cave Mining Operating Cost Summary (million $) 422
Table 21-15: Aurora Project Concentrator Operating Cost Summary (million $) 422
Table 21-16: Aurora Project DSTF Operating Cost Summary (million $) 422
Table 21-17: Aurora Project Water Treatment Operating Cost Summary (million $) 423
Table 22-1: Aurora Project Economic Model Summary 428
Table 22-2: Aurora Project Key Economic Results 430
Table 26-1: Aurora Project Summary of Field Investigations Quantity 446
Table 26-2: Aurora Project Summary of Infrastructure Geotechnical Laboratory Tests 447
Table 26-3: Estimated Costs to Complete the Aurora Project 2-Year Program Leading to PFS 450

LIST OF ACRONYMS AND ABBREVIATIONS

$	US dollars
$/kWhr	US dollars per kilowatt-hour
µm	micron
2D	two-dimensional
3D	three-dimensional
ABA	Acid Base Accounting
ACAP	Alternative Cover Assessment Program
Ag	silver
AMICS	Automated Mineralogy System for SEM

ANA	Authoridad National de Aguas
AP	acid generating potential
ARD	Acid Rock Drainage
Ausenco	Ausenco Ltd.
Bear Creek	Bear Creek Mining Company
BWi	Bond Ball Work Index
BX	Breccia
CAMP	Center for Advanced Material Processing
CDA	Canadian Dam Association
CEMA	Conveyor Manufacturers Association
CIM	Canadian Institute of Mining, Metallurgy & Petroleum
cm	centimeter
CMC	carboxymethyl cellulose
COC	Constituent of Concern
COV	Cover
CPY	Copper Porphyry
CRM	certified reference materials
Cu	copper
DDH	diamond drillholes
DGM	Directorate General de Minas
DIA	Delcaracion de Impactos Ambientales
DIGESA	La Dirección General de Salud Ambiental
DLP	DLP Resources Inc.
dmt	dry metric tonne
DPY	Deep Porphyry
DSTF	dry stack tailings facility
EIS	Environmental Impact Statement
EN	Enriched
ET	evapotranspiration
FOCUS	Minera Focus S.A.C.
g/cm^3	grams per cubic centimeter
g/t	grams per tonne
GARD	Global Acid Rock Drainage
GNSS	Global Navigation Satellite System
GPS	global positioning system
GRE	Global Resource Engineering Ltd.
ha	hectares
HF	hornfels
HFL	Hornfels
ICP-AES	Inductively Coupled Plasma-Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma-Mass Spectrometry
ID2	inverse distance squared
IEC	International Electrotechnical Commission

INAP	Internation Network for Acid Prevention
IP	Induced Polarization
IRR	Internal Rate of Return
ISO	International Organization for Standardization
km	kilometer
L/s	liters per second
lb	pound
LCT	locked cycle test
LE	Leached
LEM	Limit Equilibrium Method
LOM	life of mine
m	meter
m^3	cubic meter
m^3/s	cubic meter per second
$m^3/s/kW$	cubic meter per second per kilowatt
masl	meters above sea level
MC	Master Composite
MINEM	Ministero de Energia y Minas
MIW	mine-impacted water
ML	metal leaching
mm	millimeter
Mo	molybdenum
MPC	Mine Power Centers
MPY	Molybdenum Porphyry
MTC	Ministerio de Telecommunicaciones
MTC	Ministero de Transporte
MVA	Mega Volt-Aperes
mV/V	millivolts per volt
NaHS	sodium hydrosulfide
NI	National Instrument
NN	nearest neighbor
NNP	Net Neutralization Potential
NP	neutralization potential
NPR	Neutralization Potential Ratio
NPV	Net Present Value
NSR	net smelter royalty
OEFA	Organismo de Evaluación y Fiscalización Ambiental or Organism for Environmental Evaluation and Control
OK	ordinary kriging
OSINERGMIN	Organismo Supervisor de la Inversion en Engergia y Minera
OSS	open-side setting
PAC	potentially acid neutralizing or potentially acid consuming
PAG	potentially acid-generating

PAX	potassium amyl xanthate
PEA	Preliminary Economic Assessment
ppm	parts per million
QA/QC	quality assurance/quality control
QP	Qualified Person
RMWi	Rod Mill Work Index
ROM	Run-of-Mine
RQD	Rock Quality Designation
SABC	semi-autogenous mill and conventional ball mill with pebble crusher
SAG	semi-autogenous grinding
SEM	scanning electron microscopy
SENACE	Servicio Nacional de Certificación Ambiental para las Inversiones Sostenibles
SERFOR	Servicio National Forestal y de Fauna Silvestre
SERNANP	Servicio Nacional de Áreas Naturales Protegidas por el Estado
SIBX	sodium isobutyl xanthate
SME	Society of Mining, Metallurgy & Exploration
SPLP	Synthetic Precipitation Leaching Procedure
SRK	SRK Consulting Ltd.
SUCAMEC	Superintendencia Nacional de Control de Servicios de Seguridad, Armas, Municiones y Explosivos de Uso Civil
TCSF	Tailings Contingency Storage Facility
TML	transportable moisture limit
tpd	tonnes per day
tph	tonnes per hour
tpy	tonnes per year
TSS	Total Suspended Solids
UMAP	Uniform Manifold Approximation and Projection
Vena	Vena Resources Inc.
VFD	Variable Frequency Drive
VOIP	Voice Over Internet Protocol
WTP	Water Treatment Plant

1.0 SUMMARY

Global Resource Engineering Ltd. (GRE), Ausenco, Ltd (Ausenco), and SRK Consulting Ltd. (SRK) were retained by DLP Resources Inc. (DLP) to complete a National Instrument 43-101 (NI 43-101) Preliminary Economic Assessment (PEA) Technical Report on the Aurora Cu-Mo-Ag Mine Project ("Aurora Project" or "the Property" or "the Project") located in Calca Province, Peru. The estimates contained in this document were prepared in accordance with the Canadian Institute of Mining, Metallurgy & Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019).

DLP Resource Inc. is a Canadian mining company focused on exploring and developing base metal mining properties in South America and Canada. Through its 99.99% owned subsidiary (Due to Peruvian legal requirements, DLP Resources Peru S.A.C. must have a minimum of two shareholders. Ian Gendall, President and CEO of DLP, holds one share in DLP Resources Peru S.A.C.) DLP Resources Peru S.A.C., the company holds exploration and mining agreements covering 12,588 hectares (ha) at the Property, located under lands owned by the Parobamba Peasant Community in the Yanatile District, Calca Province, Cusco Department, Peru. DLP is listed on the TSX Venture Exchange under the symbol DLP and on the OTCQX under the symbol DLPRF.

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The project has a large resource, and this PEA outlines the Phase I development strategy. DLP envisions several further phases following drilling and technical studies as only one quarter of the resources is used in Phase I and the deposit remains open in all directions. The total resources for the Aurora property are currently defined at 614.85 million Indicated tonnes and 1.1 billion Inferred tonnes (including both open pit and underground). Following a rigorous trade-off study of development alternatives, the Company selected a staged approach designed to unlock immediate value while positioning Aurora for a multi-decade, large-scale mining operation.

A number of mining scenarios and strategies were evaluated for this PEA, including a single large open pit recovering the majority of the resource over a 35-year life and a large, multi-panel block caving mine that would recover a smaller percentage of the resource but focus on the higher-value, higher-grade molybdenum/copper ores at depth. The selected approach combines the advantages of both the surface mine, which can be started and producing economic medium grade copper mineralized material very quickly, simultaneously with the development of a large, single-panel block caving mine in the zone of highest grade, highest value mineralized material. This approach brings forward the production of higher value material, optimizing the economic value of the project. This strategy also leaves the opportunity to significantly expand the operation and to significantly extend its economic life. Since the deposit is open in all directions horizontally and vertically, it is likely that the ultimate deposit will continue to expand with the completion of additional drilling.

Phase I represents a compelling standalone project. Over a 19-year mine life, it targets 420 million tonnes of mineralized material—approximately one-quarter of the total defined resource—delivering robust financial returns:

- Net Present Value (NPV8%): $2.7 Billion
- After Tax Internal Rate of Return (IRR): 18.5%
- Cumulative cash flow after tax of $9.05 billion
- Payback of 5.2 yrs
- Dual-Front Production Plan: The strategy maximizes early cash flow by integrating a surface open-pit mine on the upper medium-grade copper/molybdenum zones with concurrent underground block cave development. This accelerated block cave access brings the deposit's exceptionally high-grade molybdenum zone into the production schedule significantly earlier, optimizing overall NPV.
- Capital Discipline & Scalability: Staging development allows the Company to establish commercial cash flow, de-risk initial capital, and maintain complete flexibility for future surface, underground, and processing plant expansions.
- Uncapped Upside: Extensive drilling demonstrates that the deposit remains open in all four directions and at depth. As step-out and deep drilling continue, future resource expansion is expected to support a substantially larger operational footprint and an extended life-of-mine.
- By combining near-term high-margin cash generation with unconstrained expansion potential, Phase I establishes a highly bankable foundation for what promises to be a tier-one mining complex.

The Qualified Persons (QPs) responsible for the preparation of this Technical Report are:

- Terre Lane, GRE
- Dr. Todd Harvey, GRE
- Dr. Hamid Samari, GRE
- Larry Breckenridge, GRE
- Josh Parsons, SRK
- Jarek Jakubec, SRK
- Carl Kottmeier, SRK
- Scott Elfen, Ausenco
- Kevin Murray, Ausenco

1.1 Property Description and Ownership

The Property is located in the Department of Cusco, Peru, 2.5 kilometers (km) northeast of the Parobamba Community at an average altitude of 3,000 meters above sea level (masl). Politically, it lies within the Yanatile District, Calca Province, Cusco Department. The Property is situated on surface lands owned by the Parobamba Community. Exploration activities are confined to the Aurora Concession, covering 400

ha, which is owned by S.M.R.L. PAROBAMBA II and optioned to DLP Resources Perú SAC. This concession is the focus of the report, as no exploration or drilling has been conducted on DLP's surrounding tenements. DLP has a favorable social agreement with the Parobamba Community extending for six more years from July 2026. The latest agreement ended in April 2026. Beyond Aurora, DLP Resources controls an additional 12,400 ha surrounding the main claim.

DLP has secured an environmental permit for the Aurora Concession, which allows the development of 30 drilling platforms, the use of existing road access to the site, and the installation of auxiliary infrastructure, including a safety cabin and a camp covering 1,602 square meters. The environmental permit complies with Category I Mining Exploration requirements under Peruvian regulations, supported by an Environmental Impact Statement (EIS). S.M.R.L. PAROBAMBA II retains a 1.5% net smelter royalty (NSR) on the Aurora Concession, which can be bought out for US$1.5 million at any project stage. However, DLP Peru has agreed with S.M.R.L. PAROBAMBA II to renegotiate the NSR Royalty buy-back right.

The Aurora Concession is situated on the surface lands of the Parobamba Community, approximately 120 km from the city of Cusco. Access to the site is via a combination of paved and dirt roads, passing through the towns of Calca, Amparaes, and Parobamba. The journey from Lima to the project involves a 1-hour flight to Cusco, followed by a 3- to 4-hour drive to the Aurora Project camp. The paved section of Route 28B, Cusco-Calca, covers 70 km in 1.5 to 2 hours, while the gravel road from Calca through Amparaes to Parobamba spans 133 km and takes 2.5 hours. From Parobamba, the project site is accessible via a 9 km gravel road, requiring an additional 30 minutes.

The Property lies between 2,700 and 3,040 masl. The region experiences a temperate semi-dry climate with dry winters, annual rainfall between 500 and 1,000 millimeters (mm), and average temperatures ranging from 12 to 14°Celsius (C), as classified by Werren Thornthwaite's climate system.

1.2 History

Exploration and development work on the Property has been focused exclusively on the Aurora Concession, with no known historical work conducted on the other concessions. The Aurora Concession, covering 400 ha, was initially staked and recorded in 1997 and has since undergone multiple ownership transfers before being optioned by DLP Resources Perú S.A.C. in 2022. Historical exploration activities on the concession included geological mapping, surface sampling, geophysical surveys, and two drilling programs conducted by Bear Creek Mining in 2001 and Vena Resources in 2005, totaling approximately 3,616 meters. Additional work by Minera Focus S.A.C. (FOCUS) in 2012 included surface sampling, drill core logging, and geophysical surveys. Despite these efforts, no historical Mineral Resource or Mineral Reserve estimates had been reported prior to AMC's 2025 Technical Report, and no mining production has occurred. DLP Resources Perú S.A.C. is currently advancing exploration under a four-year option agreement initiated in 2022.

1.3 Geology, Mineralization, and Deposit Type

The Aurora Property is situated within a complex tectonic environment in southern Peru, at the easternmost extent of the Andean Cordillera within the Central Andean metallogenic belt. The regional geology reflects the long-lived subduction of the Nazca Plate beneath the South American Plate, resulting

in extensive magmatic activity spanning from the Paleozoic era through the Cenozoic era and the formation of numerous world-class porphyry copper deposits, including the Aurora Cu-Mo deposit. The regional stratigraphy comprises the Lower Paleozoic San José Group, the Sandia Formation, and the Ananea Formation, which have been intruded by large granitic to granodioritic plutons and locally by andesitic to dioritic subvolcanic intrusions. These rocks generally have been altered by multiple tectonic events, including the early Hercynian, the late Hercynian, and the Andean (Peruvian) deformation phases, resulting in folding, faulting, and low-grade metamorphism, the development of these deformation zones became of the key structural controls that influenced the intrusive emplacement and the hydrothermal fluid flow.

The Property geology is underlain by the Sandia Formation, consisting predominantly of black slates, phyllites, and quartz-rich sandstones with an estimated thickness of approximately 2,000 m. The Sandia Formation is intruded by andesitic-dioritic subvolcanic rocks and a multiphase granodioritic to quartz monzonitic intrusive complex collectively referred to as the Aurora Porphyry. Based on cross-cutting relationships, intrusive contacts, and mineralogical characteristics, the Aurora Porphyry subdivided into three main phases, the Early Porphyry, the Intermineral Porphyry, and the Late Intermineral Porphyry phases, representing successive intrusive events associated with the evolution of the porphyry hydrothermal system. These intrusive phases are distinguished by varying intensities of quartz veinlets and associated mineralization of chalcopyrite, molybdenite, and pyrite, and collectively exhibit progressively developed hydrothermal alteration.

The hydrothermal alteration at the Aurora Project is characteristic of calc-alkaline Cu-Mo porphyry systems and comprises a well-developed potassic, phyllic (quartz-sericite-pyrite), argillic alteration assemblages. Potassic alteration is closely associated with the higher-grade Cu-Mo mineralization and is characterized by secondary biotite, K-feldspar, magnetite, and quartz vein stockworks. Phyllic alteration is widespread throughout the deposit and commonly overprints the earlier potassic assemblages, whereas argillic alteration is developed principally along fractures and fault zones.

Mineralization at the Aurora deposit is characterized by two principal mineralization styles that developed sequentially during the evolution of the hydrothermal system. The first and earlier principal mineralization style comprises Mo-Cu porphyry mineralization hosted within both the Aurora Porphyry intrusions and the surrounding Sandia Formation. This mineralization is characterized by disseminated and stockwork molybdenite, chalcopyrite and pyrite associated with quartz veinlets and formed under relatively high-temperature hydrothermal conditions. Geochronological data carried out by ALS lab in Re-Os indicate that the Mo-Cu porphyry mineralization was emplaced at approximately 6.4 Ma, contemporaneous with the emplacement of the granodioritic to quartz monzonitic porphyry stocks.

Subsequently, the Mo-Cu porphyry mineralization is overprinted by the second principal mineralization style, the structurally controlled Cu-Ag veinlets, which have a different ore mineral assemblage consisting primarily of pyrite, chalcopyrite, and a little sphalerite. The hydrothermal processes that were responsible for the two mineralization styles occurred under different physicochemical conditions. The Mo-Cu porphyry mineralization formed earlier in a higher-temperature hydrothermal environment, whereas the Cu-Ag veinlet mineralization represented a later and lower-temperature hydrothermal phase. Surface geochemistry, particularly the distribution of molybdenum anomalies, together with the geological

mapping and the drilling results, indicates the Cu-Mo porphyry system remains open along the main northwest-southeast trend and at depth, supporting the excellent potential for additional Cu-Mo mineralization expansion.

Based on the available geological, alteration, mineralization, geochemical, and drilling information, together with observations made during the June 2, 2026 site visit, the Aurora deposit is interpreted to represent a Cu-Mo porphyry deposit. The observed multiphase intrusive complex, hydrothermal alteration assemblages, stockwork quartz-sulfide veining, brecciation, and associated Cu-Mo mineralization are consistent with the characteristics of porphyry copper-molybdenum deposits and support the interpretation that the Aurora Project hosts a Cu-Mo porphyry system.

1.4 Exploration, Drilling, and Sampling

1.4.1 Exploration

Exploration on the Aurora Property has been conducted by several operators since the early 2000s and is summarized as Pre-DLP and DLP exploration programs. Pre-DLP exploration programs established the geological potential of the Property through geological reconnaissance, surface geochemical sampling, geophysical surveys, trenching, and the initial diamond drilling programs, which confirmed the presence of a large Cu-Mo porphyry system associated with multiphase granodioritic to quartz monzonitic intrusions and widespread hydrothermal alteration.

Pre-DLP surface exploration included geological mapping, rock geochemical sampling, and airborne geophysical surveys. In 2012, FOCUS completed 120 rock-chip samples that identified significant copper and molybdenum anomalies, which were subsequently confirmed by drilling. Although the original analytical database is unavailable and was not relied upon for the Mineral Resource estimate, the program successfully identified the principal mineralized trend. During the same year, FOCUS completed approximately 43.8 line-km of airborne Induced Polarization (IP) and magnetic surveys, defining a strong chargeability and resistivity anomaly measuring approximately 2.0 km by 1.4 km that coincides with porphyry intrusions, hydrothermal alteration, and Cu-Mo mineralization. These anomalies remain open to the northwest and southeast and provided key drill targets for subsequent exploration.

Since acquiring the Property, DLP Resources Inc. has completed systematic exploration programs including detailed geological and structural mapping, surface rock geochemical sampling, petrographic and alteration studies, and extensive diamond drilling. In 2024, DLP collected 40 rock samples northeast of the existing drill area and identified a new Cu-Mo-Ag surface anomaly measuring approximately 700 m by 500 m. The highest-grade sample returned 2.25% Cu and 195 g/t Ag, with mineralization consisting of pyrite, chalcopyrite, covellite, and chalcocite, demonstrating the potential for additional mineralization beyond the currently drilled area.

The combined exploration programs have significantly improved the geological understanding of the Aurora deposit by defining the distribution of lithological units, intrusive phases, hydrothermal alteration, structural controls, and mineralization. Surface geochemical and geophysical anomalies have been validated by drilling and indicate that the Cu-Mo porphyry system remains open along its northwest-southeast trend and at depth, providing strong potential for future resource expansion.

1.4.2 Drilling

Diamond drilling on the Aurora Property has been completed by Bear Creek Mining Company (2001), Vena Resources Inc. (2005), and DLP Resources Inc. (2022–2025). Between 2001 and 2025, a total of 41 diamond drillholes totaling 26,533.18 m were completed on the Property. Historical drilling by Bear Creek and Vena Resources comprised 13 drillholes totaling 3,919.80 m, while DLP completed 28 diamond drillholes totaling 22,613.38 m between 2022 and 2025. The historical drilling completed by Bear Creek and Vena Resources confirmed the presence of significant Cu-Mo mineralization; however, these drillholes were excluded from the geological model and Mineral Resource Estimate because the original geological logging, sampling procedures, analytical data, and QA/QC information could not be independently verified by the QP. Consequently, the geological model and Mineral Resource Estimate were based exclusively on the verified DLP drilling database.

The DLP drilling programs were completed in four phases between 2022 and 2025, utilizing PQ, HQ, NQ, and limited BTW diamond core diameters. DLP personnel designed, supervised, and managed the drilling programs, while Kluane Drilling provided the contract drilling services. Drill sites were accessed by well-maintained access roads, and drillhole collars were marked with wooden stakes before drilling commenced. DLP geologists initially located drillhole collars using handheld GPS units, while Kluane Drilling positioned and aligned the drill rigs. Drilling operations were conducted continuously using two shifts per day to maximize productivity throughout each drilling campaign.

Following completion of each drillhole, steel casing was installed to secure and mark the collar, and a concrete collar monument was constructed and labeled with the drillhole identification, total depth, dip, azimuth, and completion date. Preliminary collar coordinates were recorded using handheld GPS units, and upon the completion of each drilling campaign, qualified surveyors from UGCE&C (Urano Environmental Consultants) surveyed the secured collars using a differential GPS instrument. Survey results were documented and accompanied by the appropriate calibration certificates.

Two independent downhole survey methods were employed during drilling to ensure accurate drillhole positioning. Devishot downhole surveys were completed by Kluane Peru S.A.C. at approximately 50 m intervals and formally reported to DLP. In addition, Peru Survey completed high-precision gyroscopic surveys using a GIST-M gyroscopic survey tool at approximately 10 m intervals to provide accurate north-seeking measurements and verify drillhole trajectories.

Diamond drill core was transported from the drill sites to the core logging facility where DLP geologists closely monitored drill progress and systematically processed the recovered core. Core was cleaned, reconstructed and realigned to the orientation line when present within the core boxes, then photographed, and logged using standardized geotechnical and core logging templates before the information was transferred to an electronic database. Geotechnical logging included measurements of core recovery and Rock Quality Designation (RQD), whereas core logging recorded detailed descriptions of lithology, oxidation, hydrothermal alteration, mineralization, structures, veining, and other geological characteristics. Furthermore, density measurements were collected at regular intervals, sample intervals were marked according to geological boundaries, and the core was subsequently cut using a diamond saw. One-half of the core was submitted for laboratory analysis, while the remaining half was retained in the core boxes for future reference.

Core recovery throughout the DLP drilling program (2022- 2025) was considered excellent and suitable for Mineral Resource estimation. Approximately 85% of the core intervals achieved recoveries greater than 80%, and approximately 74% achieved recoveries greater than 90%. DLP's drillhole depths ranged from 66.88 m to 1,192.15 m, with an average drillhole depth of approximately 808 m. The drilling programs were designed to provide geological information and support Mineral Resource estimation using an approximate drill spacing of 100 m × 100 m, while the 2025 drilling program further improved geological confidence and confirmed the continuity of Cu-Mo mineralization along strike and at depth.

Review of the drilling methods, surveying procedures, core handling, geological logging, sampling protocols, and recovery data indicates that the drilling database is of sufficient quality to support geological modelling and Mineral Resource estimation. The QP is not aware of any drilling, sampling, recovery, or survey factors that could materially affect the accuracy, reliability, or interpretation of the drilling results.

1.4.3 Sample Preparation, Analyses, and Security

DLP Resources Inc. developed and implemented protocols for sample preparation, analytical, QA/QC, database management, density determination, and sample security procedures for the diamond drilling programs completed between 2022 and 2025. All sampling and analytical activities were completed in accordance with DLP's established exploration procedures and protocols.

All diamond drill cores were collected using PQ, HQ, NQ, and limited BTW core diameters. The drill core was generally of good quality, providing confidence in geological logging, geotechnical measurements, and geochemical sampling. The entire length of each drillhole was sampled, with sample intervals generally established at approximately 2 m and adjusted where necessary to honor lithological, alteration, mineralization, and structural boundaries. Following geological logging and photography, the core was cut using a diamond saw, with one-half retained for record and the remaining half submitted for analysis. Between 2022 and 2025, DLP collected and submitted 11,126 diamond drill core samples for geochemical analysis.

All drill core samples were submitted to the ALS sample preparation facility in Arequipa, Peru. Sample preparation was completed in accordance with ALS preparation code PREP-31, after which prepared pulp samples were dispatched to ALS Lima for analysis. Analytical work was completed by ALS Lima, an independent ISO/IEC 17025:2017 accredited laboratory. The primary analytical method comprised ALS analytical suite ME-MS61, consisting of a 48-element package determined by four-acid digestion followed by ICP-MS analysis. Additional analytical methods, including ore-grade copper, fire assay gold, sequential copper, and ultra-trace multi-element analyses, were completed on selected samples as required.

The QA/QC program consisted of the systematic insertion of blank samples, certified reference materials (CRMs), and duplicate samples into the sample stream to monitor contamination, analytical accuracy, and analytical precision. Independent check analyses were also completed using SGS Laboratories in Lima, Peru. Review of the QA/QC data indicates that blank samples effectively monitored contamination, certified reference materials demonstrated acceptable analytical accuracy, and duplicate samples confirmed satisfactory analytical precision. No material contamination, analytical bias, or precision issues that would adversely affect the Mineral Resource Estimate were identified. Although QA/QC monitoring

was implemented throughout the drilling programs, the overall QA/QC insertion frequency for most drilling campaigns was below current industry best practice. The QP considers the QA/QC program adequate to support the Mineral Resource Estimate; however, future drilling programs should implement a minimum QA/QC insertion protocol consisting of at least one blank sample, one certified reference material (CRM), and one duplicate sample for every 20 routine samples submitted for analysis.

DLP maintained a centralized digital database containing collar, survey, geological logging, sampling, assay, density, and QA/QC information. Geological and analytical data were reviewed prior to incorporation into the project database, and QA/QC performance was routinely evaluated before assay results were accepted for geological interpretation and Mineral Resource estimation. Periodic reviews of collar coordinates, downhole surveys, geological coding, sampling intervals, assay imports, density measurements, and QA/QC records were undertaken to ensure database integrity.

Density samples were collected at nominal intervals of approximately one sample every 25 m of drill core. Individual density samples, approximately 10 cm in length, were submitted to SGS Laboratories in Arequipa, Peru, where apparent density was determined using the paraffin-coated immersion method. As of the effective date of this Technical Report, the Aurora Project density database comprises 584 apparent density measurements representing all geological domains utilized in the geological model and Mineral Resource Estimate. Measured apparent densities range from 2.29 t/m³ to 3.11 t/m³, which are consistent with the principal lithologies encountered on the Property.

DLP maintained chain-of-custody procedures throughout the sampling process. Samples were transported directly by DLP personnel from the Project to the ALS sample preparation facility in Arequipa, while the remaining drill core was transported to the secure core storage facility in Calca, Peru. The on-site logging facility and permanent core storage facility were secured, and ALS maintained its internal chain-of-custody procedures during sample preparation and shipment to the analytical laboratory in Lima.

In the opinion of the QP, the sampling methods, sample preparation procedures, analytical methods, density determination procedures, database management practices, and sample security procedures implemented by DLP during the 2022–2025 drilling campaigns are consistent with accepted industry practice. The analytical database and QA/QC results provide a reliable basis for geological interpretation and Mineral Resource estimation.

1.5 Data Verification and Quality Assurance/Quality Control

GRE's Qualified Person (QP), Dr. Hamid Samari, completed an independent data verification program to assess the accuracy, completeness, and reliability of the geological, drilling, sampling, analytical, and QA/QC data used to support this Technical Report and the Mineral Resource Estimate. The verification program focused primarily on the DLP exploration database generated between 2022 and 2025.

Limited information was available for the historical drilling programs completed by Bear Creek Mining Company (2001) and Vena Resources Inc. (2005). The available information consisted primarily of drillhole summary tables and reported drilling statistics, while the original geological logs, analytical certificates, QA/QC records, and supporting databases were not available for independent verification. Consequently, the historical drilling data were not utilized in the geological model or Mineral Resource Estimate and are presented in this Technical Report for historical and exploration context only.

DLP completed 28 diamond drillholes totaling 22,613.38 m between 2022 and 2025, comprising 11,126 analytical sample intervals. GRE's QP independently reviewed and audited the DLP databases by comparing drillhole collar coordinates, downhole survey data, geological logging, sample intervals, analytical results, and QA/QC information with the original laboratory certificates and source files. Approximately 27% of the analytical database was manually spot-checked against the original laboratory certificates and supporting records, and no transcription errors, material discrepancies, or database integrity issues were identified.

GRE's QP independently reviewed the QA/QC program implemented during the 2022 through 2025 drilling campaigns, including blank samples, certified reference materials (CRMs), duplicate samples, and independent check assays. The review confirmed that blank samples effectively monitored contamination, certified reference materials demonstrated acceptable analytical accuracy, and duplicate samples provided satisfactory analytical precision. No material contamination, analytical bias, or precision issues were identified that would adversely affect the Mineral Resource Estimate. Although QA/QC procedures were implemented throughout the drilling programs, the overall QA/QC insertion frequency for most drilling campaigns was below current industry best practice. Future drilling programs should implement a minimum QA/QC insertion protocol consisting of at least one blank sample, one certified reference material (CRM), and one duplicate sample for every 20 routine samples submitted for analysis.

During the June 2 and 3, 2026 site visit, GRE's QP independently verified the principal lithological units, hydrothermal alteration, mineralization, and structural features in the field, verified selected drillhole collar locations using handheld GPS measurements, and inspected representative drill core from the 2022 through 2025 drilling programs. The QP compared the observed geology with the geological logging and project database and independently selected 20 representative sample intervals from different areas of the deposit for independent check assay analyses. The corresponding pulp reject samples were submitted to Hazen Research, Inc. in Golden, Colorado, for independent copper, molybdenum, and silver analyses using analytical methods equivalent to those employed by ALS Laboratories. The field observations and independent check assay results confirmed the accuracy and reliability of the geological interpretations, analytical database, and QA/QC program.

In the opinion of the QP, the geological, drilling, sampling, analytical, and QA/QC data collected by DLP between 2022 and 2025 have been adequately verified and are of sufficient quality and reliability to support the geological interpretations, Mineral Resource Estimate, and conclusions presented in this Technical Report.

1.6 Mineral Processing and Metallurgical Testing

The Aurora deposit consists of several geological domains that include a main copper porphyry with minor molybdenum, and a deeper porphyry (molybdenum - dominated), as well as an enriched zone and a breccia zone. The porphyry units are described as granodiorite / quartz monzonite. A thin supergene layer is observed at the surface of the northeastern portion of the deposit, consisting of minor chalcocite. The primary copper species is chalcopyrite, while the primary molybdenum species is molybdenite there is minor enargite in the enriched zone. Small zones of breccia are observed in the contacts between the porphyry and hornfels units. Test work was conducted on all zones to examine grindability and flotation response.

The flotation flowsheet tested is typical of copper/molybdenum deposits consisting of a bulk rougher flotation of copper and molybdenum, concentrate cleaning followed by downstream molybdenum separation. In general, the flotation performance of the deposit is good, with high metal recoveries to individual copper and molybdenum concentrates. The concentrates, as developed from test work, showed no penalty elements above typical smelter thresholds. The life-of-mine (LOM) average recoveries for copper, silver, and molybdenum are 85.5%, 88.5% and 74.0%, respectively.

The samples selected were representative of the deposit in terms of spatial, domain, and grade distributions. Additional test work will be required prior to advancing the project to the next study level.

The flowsheet selected for the Aurora project was developed based on the available test work and is typical of most copper porphyry deposits containing molybdenum. The ore is subjected to primary crushing using a gyratory crusher followed by SAG and ball milling in an SABC circuit. Flotation consists of a rougher stage using large tank cells followed by concentrate cleaning in a series of Jameson or similar cells. Molybdenum separation is conducted in a second flotation circuit consisting of molybdenum rougher flotation and four stages of cleaner flotation also employing Jameson type cells. Copper and molybdenum concentrates are thickened and filtered, with the molybdenum concentrate undergoing drying in a rotary drier prior to shipment.

1.7 Previous Resource Estimates and Economic Models

The following previous mineral resource estimates were completed for the Aurora Project:

- Technical Report, NI 43-101 Technical Report on the Aurora Cu-Mo-Ag Property in the Calca Province, Peru (AMC, 2025) for DLP Resources Inc.

The reported Mineral Resource Estimate in the AMC 2025 Technical Report is shown in Table 1-1.

Table 1-1: Summary of Mineral Resources – AMC Technical Report 2025

Cut-off (NSR)	Resource Category	Tonnage (Mt)	CuEQ (%)	Cu (%)	Mo (%)	Ag (g/t)	Cu Metal (Mlb)	Mo Metal (Mlb)	Ag Metal (Moz)
$5.75/t	Inferred	1,050	0.44	0.20	0.05	2.4	4,650	1,110	80

Mt = millions of tonnes
Mlb = millions of pounds
Moz = millions of troy ounces

The AMC 2025 mineral resources are shown here for completeness. The current Mineral Resource Estimate was prepared independently and supersedes any previously prepared estimates.

1.8 Mineral Resource Estimate

GRE was contracted to complete a Mineral Resource Estimate incorporating the current drill hole database. This estimate is current as of the effective date of this report.

Table 1-2 shows the pit-constrained Mineral Resource at a NSR cutoff of $6.9.

Table 1-2: Aurora Project Pit-Constrained Statement of Mineral Resource

Class	Estimation Domain	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated	CPY Enriched	57.80	0.22	275.49	0.009	11.87	2.46	4.56	375.18
	CPY Leached	10.52	0.12	28.32	0.006	1.49	1.44	0.49	40.13
	CPY Primary	76.21	0.26	433.57	0.010	17.61	2.27	5.57	571.87
	DPY Primary	0.00				0.00	0.00	0.00	0.00
	HFL Enriched	65.00	0.33	467.83	0.018	25.12	3.23	6.75	655.61
	HFL Leached	31.98	0.15	107.06	0.023	16.48	3.05	3.13	219.94
	HFL Primary	107.52	0.21	495.03	0.021	48.92	2.33	8.07	820.33
	MPY Primary	33.56	0.24	178.02	0.059	43.92	2.64	2.85	435.15
Total Indicated		**382.58**	**0.24**	**1,985.32**	**0.020**	**165.39**	**2.55**	**31.42**	**3,118.21**
Inferred	CPY Enriched	47.53	0.22	233.68	0.010	10.46	2.09	3.19	314.90
	CPY Leached	5.22	0.12	14.06	0.008	0.86	1.31	0.22	20.42
	CPY Primary	57.79	0.25	323.23	0.015	19.69	2.21	4.11	461.08
	DPY Primary	0.00				0.00	0.00	0.00	0.00
	HFL Enriched	109.46	0.36	874.10	0.021	51.46	4.92	17.32	1,286.62
	HFL Leached	84.38	0.17	312.26	0.018	34.00	2.76	7.49	553.38
	HFL Primary	168.60	0.23	855.34	0.019	68.84	2.64	14.32	1,336.79
	MPY Primary	64.87	0.21	293.25	0.057	81.63	2.15	4.49	764.73
Total Inferred		**537.86**	**0.25**	**2,905.93**	**0.023**	**266.95**	**2.96**	**51.14**	**4,737.91**

1) The effective date of the Mineral Resources Estimate is April 30th, 2026.
2) The Qualified Persons for the estimate are Terre Lane QP-MMSA and Hamid Samari QP-MMSA of GRE.
3) Mineral resources are not mineral reserves and are not demonstrably economically recoverable.
4) Mineral resources are reported at a $6.9 NSR cutoff, calculated using a Cu price of $4.37/lb, a Mo price of $22.10/lb, a Ag price of $37/tr.oz., using variable recovery, a slope angle of 45 degrees, 0% royalty, processing cost $6.90 per tonne (includes admin).
5) Cu Equivalent (CuEq) calculated as Cu lbs + Mo lbs * (Mo price * Mo recovery * Mo payable) / (Cu price * Cu recovery * Cu payable) + Ag oz * (Ag price * Ag recovery * Ag payable) / (Cu price * Cu recovery * Cu payable)

Table 1-3 shows the out-of-pit Mineral Resource at a NSR cutoff of $25.

Table 1-3: Aurora Underground Mineral Resource

Class	Estimation Domain	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated	CPY Primary	30.24	0.25	169.24	0.05	36.53	2.44	2.37	383.07
	DPY Primary	45.02	0.05	45.38	0.13	124.28	0.34	0.50	712.72
	HFL Primary	41.21	0.24	219.94	0.04	34.09	3.18	4.21	435.42
	MPY Primary	115.79	0.08	194.03	0.19	493.14	0.74	2.74	2,848.10
Total Indicated		**232.27**	**0.12**	**628.59**	**0.134**	**688.03**	**1.32**	**9.82**	**4,379.31**
Inferred	CPY Primary	45.32	0.21	205.98	0.05	48.88	1.80	2.62	487.76
	DPY Primary	217.72	0.06	277.41	0.12	579.52	0.42	2.91	3,393.90
	HFL Primary	154.01	0.21	702.71	0.06	187.98	2.07	10.24	1,787.58
	MPY Primary	163.89	0.06	221.37	0.19	676.54	0.61	3.21	3,858.08
Total Inferred		**580.94**	**0.11**	**1,407.47**	**0.117**	**1,492.91**	**1.02**	**18.98**	**9,527.32**

1) The effective date of the Mineral Resources Estimate is April 30th, 2026.
2) The Qualified Persons for the estimate are Terre Lane QP-MMSA and Hamid Samari QP-MMSA of GRE.
3) Mineral resources are not mineral reserves and are not demonstrably economically recoverable.
4) Mineral resources are reported at a $25 NSR cutoff, calculated using a Cu price of $4.37/lb, a Mo price of $22.10/lb, a Ag price of $37/tr.oz.
5) Cu Equivalent (CuEq) calculated as Cu lbs + Mo lbs * (Mo price * Mo recovery * Mo payable) / (Cu price * Cu recovery * Cu payable) + Ag oz * (Ag price * Ag recovery * Ag payable) / (Cu price * Cu recovery * Cu payable)

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.9 Mining Methods

The Aurora Cu-Mo-Ag Mine Project will employ conventional open pit mining techniques using hydraulic shovels and rear dump rigid frame haul trucks and underground block caving.

The mine plan is designed to deliver an average of 65,000 tonnes of mineralized material to the mill per day (tpd). Mining of the open pit will commence in year -3, during construction of the mill and other mine facilities and development for the underground. Mine scheduling was designed as follows:

- Open Pit
  - In years -3, -2, and -1, mineralized material mined from the open pit would be stockpiled for processing at a later time.
  - In Years 1 through 4, following commissioning of the mill, open pit mining would be conducted at an average rate of 131.8 thousand (k) tpd (48.1 million [M] tpa), with 65 ktpd (23.7 Mtpa) to be direct to the crusher and the remaining 66.8 ktpd (25.5 Mtpa) stockpiled.
  - In Years 5 through 18, stockpiled material from the open pit would supplement the underground mined material to reach the mill feed rate of 65,000 tpd (23.7 Mtpa).
  - There is a zone of resources between the bottom of the surface mine and the top of the projected Block Caving resources which will potentially be produced through the Block Cave. For conservatism, these resources are not included as production in this study but given that the underground draw points are in hard, stable rock, it is considered likely that some portion of these resources could continue to be produced through the Block Cave since their grade exceeds the cutoff value for underground mining.
- Underground
  - Underground development of the Block Cave mine commences simultaneously with the surface mine, but due to the longer development schedule, production from the underground mine does not commence until Year 3, with the mining of 533k tonnes.
  - In Years 4 through 7, the underground mining would ramp up to full capacity, mining 4.0 M tonnes in Year 4, 7.0 M tonnes in Year 5, 7.2 M tonnes in Year 6, and 12.0 M tonnes in Year 7.
  - In Years 8 through 17, the underground mining would be conducted at a rate of 39.5 ktpd (14.4 Mtpa).

1.9.1 Open Pit

The Pit was designed to operate from commencement of operations until a date approximately in year 7 when subsidence from the Block Cave below will require ceasing surface operations. The pit was designed to provide the needed amount of mineralized material to ensure a total feed of 65,000 tpd to the mill, of which some would come from the underground mining. The total open pit mineralized tonnage would be 237.6 M tonnes, averaging 96.8 Mtpd. Mineralized material produced in excess of the 65,000 tpd capacity of the plant would be stored in a stockpile and fed to the plant to supplement the underground production in order to average 65,000 tpd. The average daily waste production rate over the life of the mine would be 98.8 ktpd. Waste rock would be placed in a waste rock storage facility (WRSF).

Open pit mining would use conventional truck and hydraulic shovel mining techniques, using 227 tonne haul trucks and 22 cubic meter (m^3) hydraulic shovels.

1.9.2 Underground

SRK has been engaged by DLP to complete the underground (UG) mine, infrastructure, and block cave design for the Aurora Project 2026 PEA.

The Aurora deposit is large and continuous, with the whole-deposit footprint screening indicating that the block model could support a viable cave footprint. In considering the Aurora deposit geometry and continuity, grade distribution, and the production rate targeted for this PEA, SRK's assessment is that block caving is the appropriate mining method for the Aurora deposit, consistent with the block cave assessment described further in Section 16. Grade distribution is zoned, with a higher-grade core that is the basis for the block cave footprint selected for this PEA.

Block caving is a large-scale, low-cost underground mining method in which a horizontal undercut is excavated beneath the mineralized zone to remove its vertical support, thereby initiating gravity-induced caving of the rock mass above. As the undercut advances, the overlying rock progressively fractures and collapses under its own weight and stress (assisted by blasting where the rock mass is not sufficiently pre-conditioned to cave on its own), and the broken mineralized material is drawn from an underlying extraction level through a grid of drawpoints connected to drawbells, which are funnel-shaped excavations that collect mineralized material from the caving column above and direct it to the drawpoint for loading. As broken mineralized material is extracted, the back of the cave progressively migrates upward through the mineralized zone, and the column above continues to draw down in a continuous, self-sustaining process. The material drawn from the drawpoints is initially mineralized material, but dilutive waste eventually enters the column from above it and/or from the sides.

The single-level block cave design for the Aurora mine was developed and scheduled using GEOVIA's PCBC™. The design comprises a single, herringbone layout, extraction level at elevation 1,845 m, and an undercut level on a single advanced-undercut horizon 18 m above the extraction level. The block cave has a 490-drawpoint / 263-drawbell footprint spanning 14 production drifts.

The underground mine access is by portal, via dual declines: one access decline and one conveyor decline. The portal collars into a crest on the mountainside approximately 1,500 m west of the cave footprint, at an elevation of 2,430 m. The dual declines have a total length of approximately 4,100 m, split into four decline sections.

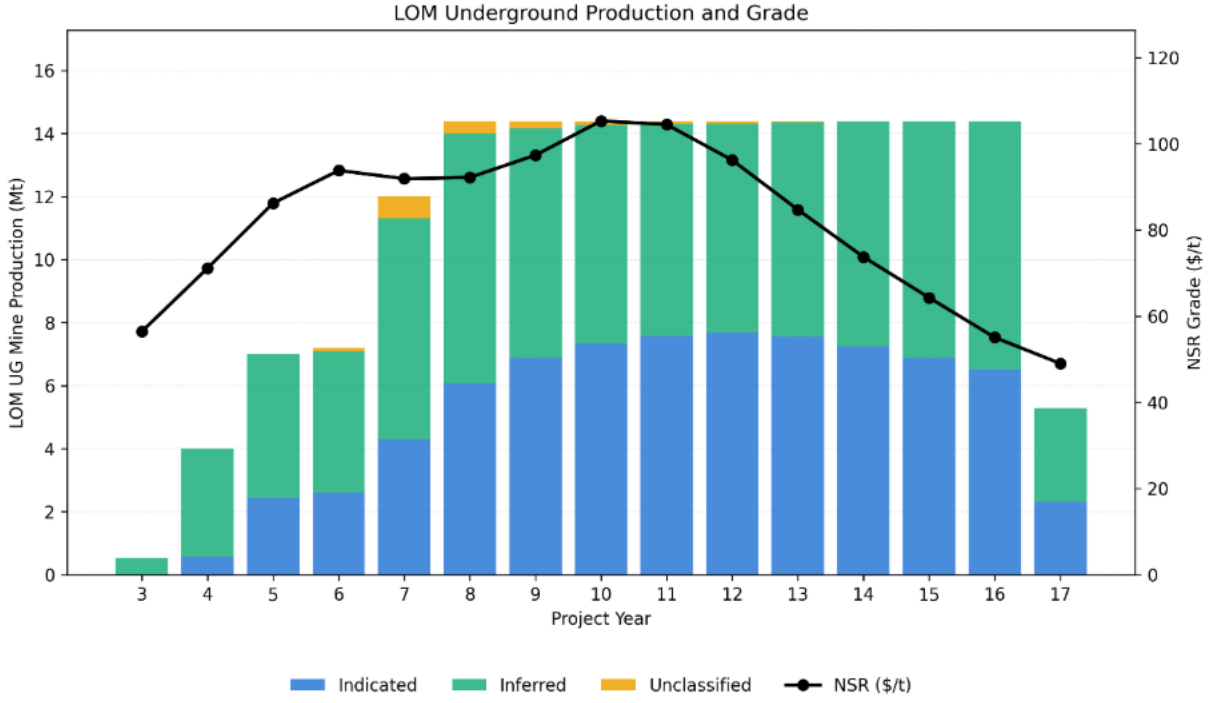
Fresh air for the underground mine is provided by the access decline, and twin fresh air raises from surface. A fresh air ventilation drive runs beneath the southern extremity of the footprint and provides fresh air to the extractions level and material handling via a series of fresh air raises. Exhaust from the working levels is collected on the footwall drives and exhausted to surface via the conveyor decline and twin exhaust raises.

The development and production sequence is designed and optimized to bring the block cave into production as quickly as possible to target the high-grade core of the deposit. Consideration was given to the overall mining rates required, the geotechnical conditions likely to be encountered, the ventilation, infrastructure, logistics, and equipment requirements. Underground lateral development is scheduled to begin at the beginning of Project Year -3 and continue through to the completion of the underground development and cave construction in Project Year 7.

Block cave establishment begins with the undercutting in Project Year 3 (Month 3), four months ahead of the first drawbell blast in Project Year 3 (Month 7). Drawbell construction rate ramp-ups to its full rate over three months. The rate of six bells per month is maintained until completion of cave construction in Project Year 7 (Month 3).

The Aurora block cave is scheduled to produce at full rate from Project Year 8 through Project Year 16 and ramping down for closure in Project Year 17. A total of 165.6 Mt of mineralized material is planned to be mined from the cave with an average NSR value of $85 per tonne. Figure 1-1 shows the LOM production, resource class distribution and NSR value.

Figure 1-1: Aurora Project Underground Cave Production Schedule



Source: SRK, 2026

1.10 Infrastructure

1.10.1 Overview

The major process facilities include the site access road, crusher facilities, process plant, filter plant, dry stack storage facility, tailings contingent facility and water management structures. Support facilities also include the administration office, laboratory and warehouse.

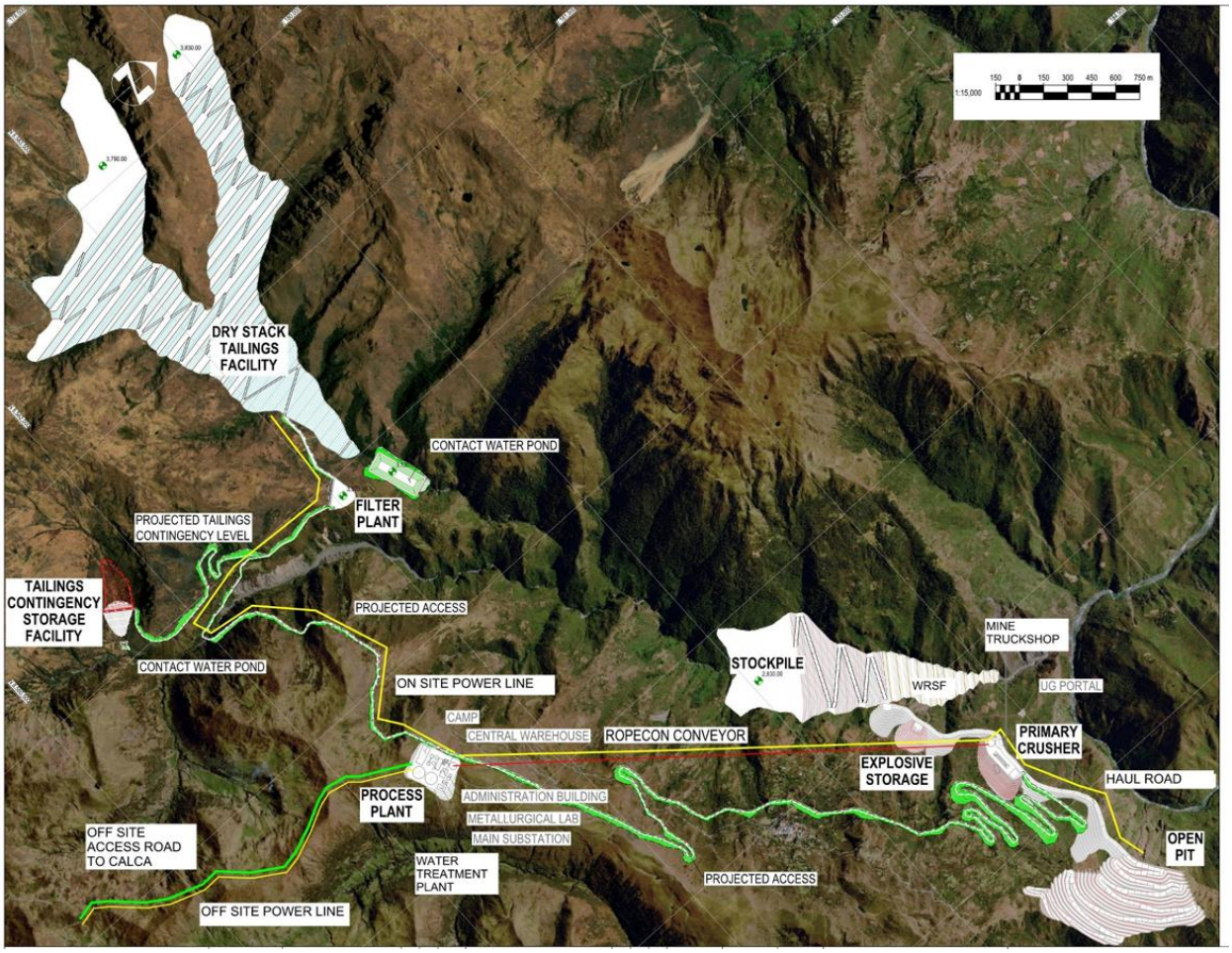
A permanent camp facility is considered on the property. The camp will be built to accommodate workers for the construction phase of the project and will be converted thereafter for use by operations personnel.

The property site can be accessed from the city of Lima to the concession area by air, from Lima to Cusco followed by land through the towns of Calca, Amparaes, and Parobamba. The total time required from Lima to Cusco is one hour by air and then an additional four and a half hours by road from Cusco to site.

Permanent electrical power is provided by a transmission line that will connect to the electrical grid in Calca. The 138-kV transmission line will connect to a substation at the site before being stepped down to 25 kV for distribution to different power requirements across the project site.

The overall site layout is shown in Figure 1-2.

Figure 1-2: Aurora Project Site Layout



Source: Ausenco, 2026

1.10.2 Dry Stack Tailings Facility

A siting and deposition trade-off study was performed as part of the PEA. Given the steep, rugged terrain, the preferred storage facility option for tailings was to place filtered tailings in a dry-stack tailings facility (DSTF). The facility is designed to provide secure, permanent storage of 393.9 million tonnes (Mt) of tailings for a 64.4 ktpd mining operation with a life of mine of 19 years. The tailings will be deposited in a steep valley, approximately 7 km southwest of the open pit and underground portal. The DSTF includes:

- Toe berm
- Non-contact water diversion channel
- Contact water diversion channels
- Underdrain to collect groundwater discharge and seepage from the filtered tailings
- Filtered tailings conveyor and stacking system
- Contact water pond
- Tailings contingency storage facility

The DSTF has been designed to contain the entire life of mine (LoM) tailings in a single facility. The facility requires a small starter berm constructed of non-acid generating (NAG) waste rock at the base of the facility. Filtered tailings will be transported from the filter plant to the edge of the DSTF by overland conveyor. Then a series of extendable grasshoppers and a radial stacking system will be used to place the filtered tailings, which will be spread and compacted by dozers and compactors in thin lifts. The facility will be constructed using a base-up stacking method.

The facility was designed with 3.H:1V local bench slopes that are 20 m high, constructed in thin lifts, and 50 m wide, benched for a global slope of 3.5H:1V. The ongoing bench raises, once completed, will be progressively closed to minimize erosion from precipitation. As the facility is raised, an underdrain will be installed to capture water before the placement of lifts. The underdrain discharges into the contact water pond at the base of the facility. In addition, contact water channels will be constructed within the DSTF and will also discharge into the contact water pond. This water will be used for mine operations, and any excess will be treated and released to the environment.

Non-contact water will be diverted around the DSTF during operations to the extent practical. Undiverted runoff will be collected in the contact water pond and utilized for mining operations.

1.10.3 Water Infrastructure and Water Availability

Based on the results of a predictive site-wide water balance, the site has abundant available water resources for mine use and mineral processing. However, abstraction from the nearby Mapacho river will be required during dry months. This large river (see Section 1.12.1) has abundant water and is “under allocated,” meaning that the river has significant available water rights.

1.11 Economic Model Results

Ms. Lane of GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Copper price of $4.90/lb, based on using a weighted average of the 3-year trailing average copper price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $4.44/lb + 40% x 1-year futures price of $6.674/lb
- Molybdenum price of $25.40/lb, based on using a weighted average of the 3-year trailing average molybdenum price and the 1-year futures price, calculated as: 80% x 3-year trailing average price of $22.08/lb + 20% x 1-year futures price of $30.50/lb
- Silver price of $45.30/oz, based on using a weighted average of the 3-year trailing average silver price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $37.03/oz + 40% x 1-year futures price of $78.406/oz
- Variable mineral recoveries calculated from the equations provided in Section 13.4 for five domains: Cu, Mo, HF, EN, and LE. Overall, mineral recoveries averaged 85.5% for copper, 88.5% for molybdenum, and 74.0% for silver.
- NSR values calculated using the following inputs:
    - Copper 96.5% payable; molybdenum 96% payable; silver 96% payable
    - $25/dmt Cu smelting
    - $0.025/dmt Cu refining
    - $169.35/dmt Cu transport
    - $0.08/dmt Mo roasting
    - $160/dmt Mo transport
    - $0.75/oz Ag refining
    - 90% Cu concentrate solids, 97% Mo concentrate solids
- All costs input to the model are in US dollars
- Sales and use taxes are not included in the model
- Taxes, depreciation, amortization, and deductions as described below.
- Project completion is in Year 18, with closure costs incurred in Years 19 through 24. Post-closure costs would be incurred in Years 20 and beyond. The model extends only to year 20.

1.11.1 Results

Table 1-4 lists the key economic results for the project.

Table 1-4: Aurora Project Key Economic Indicators

After Tax Economic Measure	Value
After-Tax NPV @8% (million $)	$2,703
IRR	18.5%
Initial Capital (million $)	$2,377
Payback Period (years)	5.2

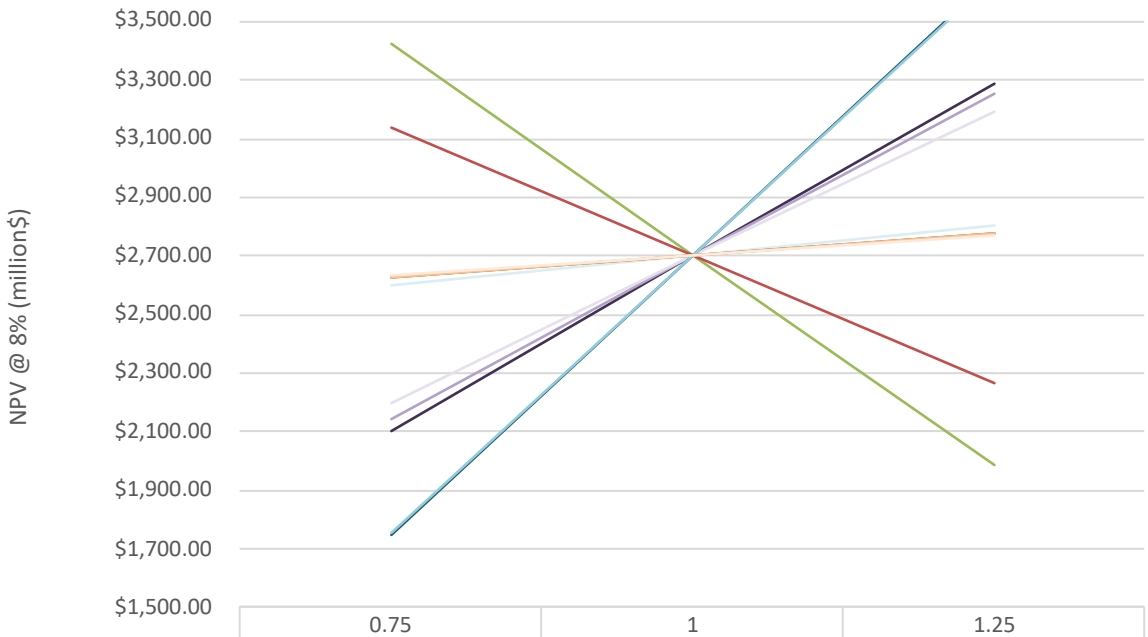
C1 Cash Cost Cu	$0.90
C1 Cash Cost Mo	$7.83

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This PEA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

1.11.2 Sensitivity Analyses

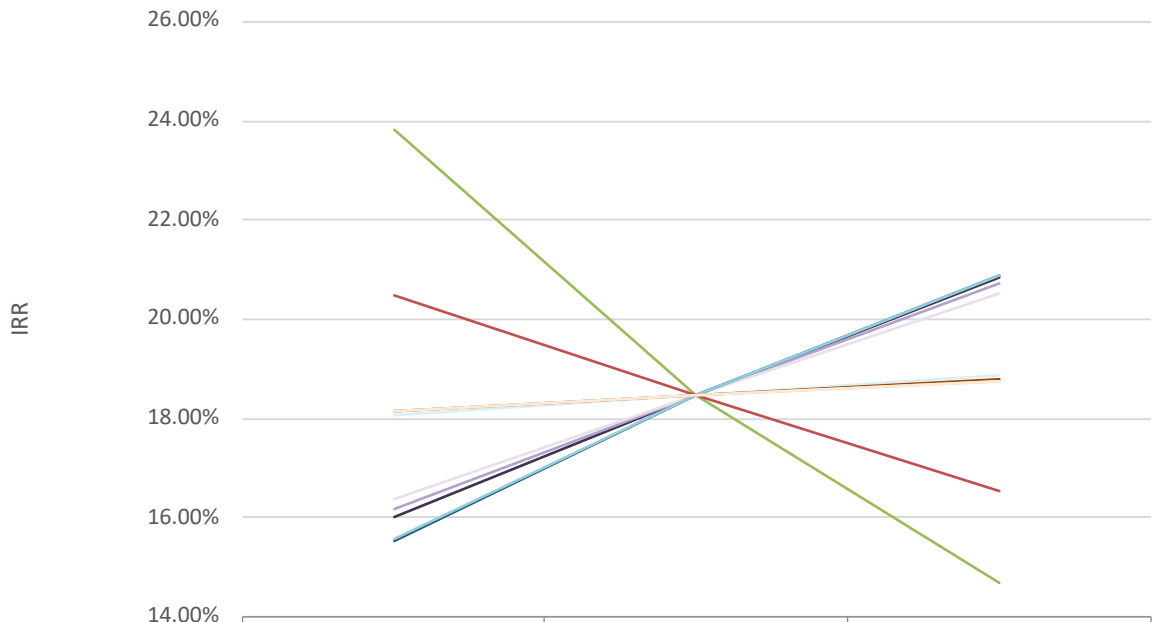
Ms. Lane of GRE evaluated the after-tax NPV@8% and IRR sensitivity to changes in capital costs, operating costs, copper price, copper grade, molybdenum price, molybdenum grade, silver price, and silver grade. The results indicate that the after-tax NPV@8% and IRR are most sensitive to molybdenum price and grade, then capital costs, then copper price and grade, with moderate sensitivity to operating cost, and minimal sensitivity to silver price and grade (NPV@8% shown in Figure 1-3; IRR shown in Figure 1-4).

Figure 1-3: Aurora Project Post-Tax NPV@8% Sensitivities



	0.75	1	1.25
Capital Costs	$3,422.38	$2,702.99	$1,988.19
Operating Costs	$3,135.67	$2,702.99	$2,263.02
Cu Price	$2,101.64	$2,702.99	$3,286.07
Cu Grade	$2,140.31	$2,702.99	$3,256.13
Cu Recovery	$2,200.44	$2,702.99	$3,191.12
Mo Price	$1,745.08	$2,702.99	$3,655.71
Mo Grade	$1,753.58	$2,702.99	$3,647.29
Mo Recovery	$2,600.99	$2,702.99	$2,804.91
Ag Price	$2,627.91	$2,702.99	$2,778.05
Ag Grade	$2,629.15	$2,702.99	$2,776.81
Ag Recovery	$2,633.18	$2,702.99	$2,772.78

Figure 1-4: Aurora Project Post-Tax IRR Sensitivities



	0.75	1	1.25
Capital Costs	23.80%	18.46%	14.69%
Operating Costs	20.49%	18.46%	16.51%
Cu Price	16.01%	18.46%	20.84%
Cu Grade	16.16%	18.46%	20.72%
Cu Recovery	16.35%	18.46%	20.52%
Mo Price	15.54%	18.46%	20.89%
Mo Grade	15.57%	18.46%	20.87%
Mo Recovery	18.05%	18.46%	18.86%
Ag Price	18.14%	18.46%	18.77%
Ag Grade	18.15%	18.46%	18.76%
Ag Recovery	18.16%	18.46%	18.75%

1.11.3 Conclusions of Economic Model

The project economics shown in the PEA are favorable, providing positive NPV values at varying prices, grades, capital costs, and operating costs.

1.12 Environmental Issues, Permitting, Community Relations, and Closure

1.12.1 Environmental Setting

The Project is located within the transition between the Amazon Basin and the Andes, and its physiography is dominated by the impacts of two major rivers: the Mapacho River and its tributary, the Mestizamayo River. The project facilities cover a range of elevations from 2,250 meters, at the confluence of the Mapacho river and the Mestizamayo river, up to 3,800 meters, at the highest level of the DSTF. The wide range of elevations means that more than one physiographic region is included in the project area:

- The “upland area” with broad U-shaped valleys which have historically been glaciated
- The“lower area” with steep-sided V-shaped valleys carved by fluvial erosion.

The Mestizamayo river runs east-west across the northern boundary of the Project and hosts most of the project infrastructure, including the DSTF, ore storage area, and the WRD. It has a total drainage basin area of 52.7 square kilometers and drains into the Mapacho River. Containing project infrastructure in one drainage basin has the following advantages:

- Lower or no disturbance within adjacent drainage basins
- Environmental impacts are concentrated within one watershed
- Easier water management and water treatment due to a single point of capture and collection for most mine impacted water.

The Mapacho River forms the northern and eastern boundary of the Project area. It is a major river with 58,000 square kilometers of drainage basin (draining an area larger than Costa Rica), Both rivers are perennial, with the Mestizamayo River showing a greater change in flow due to wet/dry seasonality than the Mapacho because of its smaller size.

Section 4.1 and Section 20.1 discuss the site physiography and environmental setting in greater detail.

1.12.2 Geochemistry

Based on screening-level geochemical sampling and analysis, it appears that the waste rock, mine tailings, underground development rock, and pit wall rock will all be Potentially-Acid Generating. As a result, the project development plan has been designed to prevent and manage water quality impacts from Acid Rock Drainage (ARD) and Metal Leaching (ML) at all stages of project development. During operations, mitigation measures include water diversion, water treatment, and the consumption of Mine Influenced Water (MIW).

1.12.3 Permitting

The site has all the necessary permits to continue exploration and development. The site also has a multi-year plan for the acquisition of the mine permits (See Section 20.4).

1.12.4 Closure

A site-wide closure plan has been designed in conjunction with the requirements of the Peruvian mine closure laws (Ministerio de Energia y Minas, 2005). During closure, mine waste facilities will be covered with an engineered evapotranspiration cover to minimize infiltration and MIW production. Long term water treatment is also required to meet surface water and groundwater discharge quality. The costs of the closure plan are included in the PEA economic model.

1.12.5 Community Relations

The site has a signed agreement with the local community for the ongoing exploration and development activities until 2032 (See Section 20.5).

1.13 RECOMMENDATIONS

The QPs have prepared recommendations within their respective areas of expertise. The following recommendations are organized by technical discipline and reflect the opinions and recommendations of

the respective QPs responsible for geology, mineral resource estimation, mining, metallurgy, and other applicable disciplines.

1.13.1 Geology

The QP recommends that DLP continue systematic exploration of the Aurora Project to improve the geological understanding of the deposit, better define the geometry, continuity, and controls of the copper-molybdenum-silver mineralization, and evaluate the lateral and vertical extensions of the known mineralized system. The recommended geological work includes the following:

- Continue diamond drilling to infill and to expand the known mineralized zones along strike and at depth, evaluate the continuity of mineralization, support future Mineral Resource classification upgrades where warranted by increased drilling density and geological confidence, and test additional exploration targets identified through geological, geochemical, and geophysical interpretation.
- Continue detailed geological mapping to improve the interpretation of lithological contacts, intrusive phases, hydrothermal alteration, and mineralization.
- Complete detailed structural mapping and structural analysis to better define the controls on intrusive emplacement and mineralization, improve geological interpretations, and optimize future drill targeting.
- Continue systematic surface rock and soil geochemical sampling in areas with limited previous coverage and around newly identified exploration targets.
- Continue refinement of the three-dimensional geological model by integrating geological, structural, alteration, geochemical, geophysical, and drilling data as additional information becomes available.
- Conduct additional petrographic and mineralogical studies to improve understanding of mineralization, alteration zoning, and the distribution of copper-, molybdenum-, and silver-bearing minerals.
- Continue collecting oriented core data, where practical, to further define structural controls on mineralization and improve future geological modeling and drill planning.
- Carry out airborne geophysical surveys (e.g., magnetics, radiometrics, magnetotellurics) to identify new exploration targets, refine structural and lithological interpretations, and support property-scale exploration.

The QP considers the Aurora Project sufficiently prospective to justify continued systematic exploration. The exploration results obtained to date support continued drilling and associated geological studies to improve the geological understanding of the deposit, better define the geometry, continuity, and controls of the porphyry copper-molybdenum-silver mineralization, and evaluate the lateral and vertical extensions of the known mineralized system.

The estimated costs of the recommended exploration work program are summarized in Table 1-5. The budget is based on DLP's planned exploration activities for 2026-2027 and includes estimated costs for

diamond drilling, geological mapping, geochemical sampling, assaying, geological modeling, metallurgical test work, environmental studies, project management, and other supporting technical activities.

1.13.2 Metallurgy

The following recommendations have been provided to aid in the advancement of the project to the next study level.

- Conduct additional comminution testing with increased sample sets and include the rod mill work index. HPGR may be applicable for this project, and the RMWi is one variable that helps identify its suitability.
- Conduct additional primary grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Conduct additional regrind grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Additional testing examining reagent suits, particularly for the molybdenum cleaning phase to depress fluorine.
- Examine the use of Jameson or Nova flotation cells to assist in fluorine rejection in the molybdenum circuit.
- Additional testing of high-grade molybdenum materials particularly from future underground areas.
- Additional variability testing, including a wide range of distributions: spatial, domains, and grades.
- Impact of material weathering of flotation performance given the potential long duration stockpile storage envisioned in the production plan.
- Conduct additional test work to define the mechanism for fluorine entrainment.
- Conduct solid/liquid separation testing to develop design criteria for thickening and filtration.
- Conduct transport moisture limit (TML) testing to define the maximum permissible moisture content of the concentrates for sea shipment.

The estimated cost of this work is $550,000, excluding drilling and sample procurement.

1.13.3 Resource

The QP makes the following recommendation: continue expansion and definition (conversion of inferred resources to M&I categories) of both the open pit and underground resources through exploration drilling.

1.13.4 Open Pit Mining

The QP makes the following recommendations:

- Conduct a geotechnical drilling program and analysis to determine appropriate pit slope parameters.
- Determine the phreatic water from drilling, perform packer tests where needed, and model potential pit inflow.
- Optimize mine design and material handling plans.

1.13.5 Underground Mining

1.13.5.1 Geotechnical

The QP makes the following recommendations relating to geotechnical risks and opportunities:

- Acquire comprehensive orebody knowledge through additional drilling and the implementation of data collection and analysis programs for rock mass, structures, and water for cave access and development.
- Undertake a dedicated drilling program targeting the proposed caving area and extending to the planned production level elevation. The program should incorporate comprehensive geotechnical investigations, including detailed geotechnical logging, oriented core, and acoustic or optical televiewer surveys to improve the understanding of the deposit's structural framework. The resulting data should be used to develop a robust three-dimensional geotechnical and structural model to support future cave design, caveability assessments, and geotechnical analyses.
- Conduct a dedicated drilling program targeting the area of interest and extending to the proposed production level elevation. These drillholes should be used to support a detailed geotechnical logging program and should include oriented core and televiewer surveys to improve the structural understanding of the deposit. This additional information should allow the development of a robust 3D geotechnical and structural model.
- Undertake more work in the next level of study to evaluate rock mass caveability and understand the steps that would have to be taken if the rock mass did not cave as expected (e.g., preconditioning via hydraulic fracturing). This work will require close collaboration between the cave mining and geotechnical consultants.
- Implement an undercutting strategy that minimizes damage to the rock mass due to induced abutment stress through advanced undercutting.
- Undertake numerical stability modelling and benchmarking.
- Develop and implement a comprehensive cave monitoring and instrumentation program prior to cave commissioning. This will enable the operator to adjust activities such as the rate and direction of undercutting and draw strategy to eliminate airblast risk and minimize potential seismic risk.
- Undertake a mudrush risk study, including an assessment of the physical properties of the caved material and the development of a surface water management strategy.
- Consider managing the subsidence zone, including water diversion, and subsidence slope management.
- Consider establishing an independent Technical Review Board to advise on compliance with industry standard block caving practices and procedures.
- Consider adding hydrogeologic characterization and monitoring to underground geotechnical drilling. This would include adding packer (lugeon) tests, oriented core, and vibrating wire piezometer installation to geotechnical boreholes.

1.13.5.2 Underground Mining

The QP makes the following recommendations relating to mining risks and opportunities:

- Undertake more work in the next level of study, as understanding of the resource is advanced, to evaluate additional block cave footprints and extend the underground mine life, and consider utilization of the infrastructure established for the initial block cave footprint.
- Further optimize mine design and development schedule to ensure main ventilation raises are established as quickly as possible to support footprint development and cave construction.
- Continue mine optimization and design work as the understanding of the resource is advanced. The footprint selection, draw strategy, and mine layout used in this study should be validated at the conclusion of each major resource model or geotechnical model update.
- Design the mine to reflect industry standards and the latest learnings from deep caves.

1.13.6 Infrastructure

1.13.6.1 Infrastructure Geotechnical

Geotechnical investigations are required within the footprint of each planned structure. These investigations should be strategically located within the footprint, applying engineering judgment and taking into account the site's topography and design characteristics.

Within the framework of the proposed geotechnical investigation program, the extracted core samples shall undergo both physical property testing and specialized tests, as determined by engineering judgment.

Samples must be transported using ASTM International (ASTM) methods to ensure proper handling and consistency. Tailings samples should be obtained from a pilot plant to reflect actual processing conditions. All samples must be representative of the deposit to provide reliable inputs for the prefeasibility-level geochemical studies.

Finally, the development of both factual and Interpretative reports based on data gathered from the geotechnical field and laboratory programs are required.

The estimated cost for this work is $1.4 million.

1.13.6.2 DSTF and TCSF Analyses and Studies

Due to the conceptual nature of this study and the limited information available at the time of writing, several assumptions have been made regarding the layout, material takeoffs, and construction of the proposed DSTF and TCSF. Geotechnical properties of construction materials will be required to support slope stability analyses and other geotechnical assessments needed to confirm that the DSTF and TCSF can be constructed as designed. Additionally, future geochemical analysis of the tailings and waste rock will be required to determine whether considerations for acid generation and metal leaching should be incorporated into the design of these facilities.

Additional studies and data collection will be required to advance project development beyond the conceptual stage. Some—though not necessarily all—of the current data gaps that should be addressed in future studies include the following:

- Perform deterministic and probabilistic local seismic hazard studies to develop design seismic parameters for infrastructure.
- Collection of hydrological information through site-specific climate and hydrological studies to support the design of ponds, diversion channels, and spillways, the development of the DSTF and TCSF water balances, and alignment with the site-wide water balance.
- Collection of hydrogeological information from desktop studies and site investigations to improve understanding of subsurface flow regimes and their implications for facility design and performance.
- Confirm geochemical characterization of tailings and waste rock from additional waste characterization studies.
- Develop seepage predictions and seepage control measures for the DSTF and TCSF based on geotechnical field and laboratory programs.
- Update the tailings deposition strategy to optimize material handling.
- The stability model should be reviewed and updated, as required, with consideration of the final deposition and stacking plan using updated data on the material properties of the tailings, construction material and the foundations for the DSTF and TCSF.
- Perform a liquefaction assessment of foundations, considering updated information on material properties for DSTF
- Perform dam breach analysis for TCSF and DSTF runout analysis.
- Solicit budgetary quotes for earthworks and geosynthetics (i.e., geomembrane, geotextile, and piping) to get more accurate pricing for the next cost estimate.
- Develop material take-offs and cost estimates (i.e., capital, sustaining capital, and operating costs) for DSTF and TCSF.

The additional information obtained and the assumptions made in this study can be verified or updated to advance the project to the next level of design. The cost of implementing the above recommendations for geotechnical engineering design is estimated at $0.6 million.

1.13.7 Water Infrastructure

Water studies must be advanced. The existing water balance requires revision after more environmental data has been collected. Section 26.8.1 discusses the necessary surface water and groundwater flow data needed to advance the surface water flow models and the site wide water balance.

1.13.8 Environment and Permitting

1.13.8.1 Baseline Data Collection

Baseline data collection for at least three years prior to mining is strongly recommended. The following environmental data must be collected:

- Surface water flow readings: As mentioned in Section 18.5, the project must take advantage of nearly all the water in the Mestizamayo drainage basin and must abstract water from the Mapacho

river. In order to acquire water rights, it is necessary to build a multi-year river flow measurement database.

- Meteorology: The current meteorological station is ~38 km away, and on-site meteorological data is required. Additionally, because the site covers two different physiographic regions (see Section 4.1.3) an upper-elevation and a lower-elevation meteorological station are recommended. The upper-elevation station should have a snow-water equivalent rain gauge.
- Environmental Geochemistry: Due to the Acid Rock Drainage (ARD) and metal leaching (ML) risk, the site should execute a wide-spread and intensive ARD and ML characterization program involving static sampling, and kinetic (long duration) geochemical testing. Testwork should cover underground development rock (from block cave access tunnels), open pit rock, pit wall rock, and mine tailings. The mine tailings geochemical test work must include an assessment of tailings variability – in particular the geochemical difference between deeper high-grade block cave tailings, and shallow lower-grade open pit tailings.
- Groundwater: No groundwater data exists. It is necessary to collect hydrogeologic data in geotechnical boreholes for he pit walls and for the underground workings. This should be done with packer tests (lugeon-style injection tests), and vibrating wire piezometer installation. Shallow groundwater wells should be installed at strategic locations around the site. These wells should be aquifer tested, and the water levels should be monitored seasonally for at least a calendar year.
- Water Quality: Few water quality samples exist. Site-wide baseline surface water quality is required from rivers, seeps, springs, and groundwater wells. Samples should be collected from seeps and springs in the Sandia formation to see of naturally-occurring ARD conditions (see Figure 20.3) are rare, or widespread. Total Suspended Solids (TSS) sampling should be performed to understand sediment transport dynamics.
- Air and Noise: Air quality samples and noise background should be collected as per the requirements of Peruvian Law.
- Social Monitoring: The site should continue a social monitoring program to better understand the local community.

1.13.8.2 Environmental Studies

For a Pre-Feasibility study, the following major studies will be required:

- Geochemical predictive models of future ARD and ML for the pit, underground workings, DSTF, and WRSF. These must be transient, site-wide, and robust enough to determine the water quality of all mine-impacted water (MIW) streams over time and space within the project.
- Predictive groundwater flow models of the dewatering vs time for the open pit, block cave development tunnels, and for the block cave.
- Revised surface water models and water balance models based on, and calibrated to, the surface water flows measured in the environmental monitoring work scope.
- Treatability test work for the MIW water treatment plant
- Air quality models for the project
- Noise models

- Sediment production and sediment management models
- Evapotranspiration cover studies to optimize the design.

1.13.8.3 Permitting

The Aurora project should continue permitting activities. At this stage, the continuation of the permitting for exploration, the baseline data collection, and the studies discussed above are the most-prudent measures to advance permitting because full-scale permit acquisition requires the data and studies above.

To the east of the Project is the Manu National Park and related buffer zone. The buffer zone is a 10 km wide band that abuts the Manu National Park and stops at the northern edge of the Mapacho River. Neither the mineral claims, nor any of the planned development for the project infringes on the buffer zone or the Manu National Park. Section 20.4 discusses additional permitting discussions which may be triggered by the buffer zone.

1.13.8.4 Closure

Studies above will improve the closure plan and the closure cost model. Once completed, the closure plan and closure cost estimate must be revised.

1.14 Total Cost for Recommended Program

The estimated costs of the recommended exploration work program are summarized in Table 1-5. The budget is based on DLP's planned exploration activities for 2027 and includes estimated costs for diamond drilling, geological mapping, geochemical sampling, assaying, geological modeling, metallurgical test work, environmental studies, project management, and other supporting technical activities.

Table 1-5: Estimated Costs to Complete the Aurora Project 2-Year Program Leading to PFS

Description	$USD
Diamond Exploration & Infill Drilling (100,000 m)	$50,000,000
Camp and Infrastructure	$5,000,000
Geological Mapping & Sampling	$1,500,000
Geophysical Survey (100 sq km)	$1,000,000
Metallurgical Test work	$3,000,000
Geotechnical Studies	$4,000,000
DSTF and TCSF Data Acquisition	$1,500,000
Environmental or Baseline Studies	$3,000,000
Permitting	$3,000,000
Community, Land & Legal Cost	$3,000,000
Pre-Feasibility Study	$5,000,000
Total	**$80,000,000**

2.0 INTRODUCTION

2.1 Terms of Reference

Global Resource Engineering Ltd. (GRE), SRK Consulting Ltd. (SRK), and Ausenco Pty Ltd. (Ausenco) were retained by DLP Resources Inc. (DLP) to complete a National Instrument 43-101 (NI 43-101) Technical Report on the Aurora Cu-Mo-Ag Project ("Aurora Project" or "the Property" or "the Project"). The Property is located in Calca Province, Peru.

The Mineral Resource Statement was prepared in accordance with the Canadian Institute of Mining, Metallurgy & Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019).

DLP Resources Inc. is a mineral exploration company operating in Southeastern British Columbia and Peru, exploring for Base Metals and Cobalt. DLP is listed on the TSX-V, trading symbol DLP and on the OTCQB, trading symbol DLPRF. Please refer to our web site www.dlpresourcesinc.com for additional information.

The Preliminary Economic Assessment (PEA) is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this PEA will be realized.

The independent Qualified Persons (QPs) as defined by NI 43-101 responsible for the preparation of this Technical Report are:

- Terre Lane, MMSA 01407QP, Society of Mining, Metallurgy & Exploration (SME) Registered Member 4053005, GRE
- Todd Harvey, PhD, SME Registered Member 4144120, GRE
- Hamid Samari, PhD, Geology QP, GRE
- Larry Breckenridge, P.E., GRE
- Josh Parsons, P.Eng., SRK
- Jarek Jakubec, Fellow of the Institute of Materials, Minerals and Mining (FIMMM), SRK
- Carl Kottmeier, P.Eng., SRK
- Brian Prosser, PE, SRK
- Scott Elfen, P.E. and P.Eng., Ausenco
- Kevin Murray, Ausenco

The QPs are collectively referred to as the "Authors" of this PEA. Several of the QPs have visited the property (see Section 2.3). In addition to their own work, the Authors have made use of information from other sources and have listed these sources in this document under "References."

Table 2-1 lists the primary "Qualified Persons" (as defined in the National Instrument 43-101) that compiled different sections of the report.

Table 2-1: List of Contributing Authors

Section	Section Name	Qualified Person
1	Summary	Terre Lane
1.1	Property Description and Ownership	Terre Lane
1.2	History	Terre Lane
1.3	Geology and Mineralization	Hamid Samari
1.4	Exploration, Drilling, and Sampling	Hamid Samari
1.5	Data Verification and Quality Assurance/Quality Control	Hamid Samari
1.6	Mineral Processing and Metallurgical Testing	Todd Harvey
1.7	Previous Resource Estimates and Economic Models	Terre Lane
1.8	Mineral Resource Estimate	Terre Lane
1.9	Mining Methods	
1.9.1	Open Pit	Terre Lane
1.9.2	Underground Block Cave	Josh Parsons
1.10	Infrastructure	Scott Elfen
1.11	Economic Model Results	Terre Lane
1.12	Environmental Issues and Permitting	Larry Breckenridge
1.13	Recommendations	All
2	Introduction	Terre Lane
3	Reliance on Other Experts	Terre Lane
4	Property Description and Location	Terre Lane
4.1	Property Location	Hamid Samari
4.2	Ownership, Mineral Rights, and Tenure	Hamid Samari
4.3	Underlying Agreements – Royalty Agreements	Hamid Samari
4.4	Environmental Liabilities	Larry Breckenridge
4.5	Permits	Larry Breckenridge
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Hamid Samari
6	History	Hamid Samari
7	Geological Setting and Mineralization	Hamid Samari
8	Deposit Types	Hamid Samari
9	Exploration	Hamid Samari
10	Drilling	Hamid Samari
11	Sample Preparation, Analyses and Security	Hamid Samari
12	Data Verification	Hamid Samari
13	Mineral Processing and Metallurgical Testing	Todd Harvey
14	Mineral Resource Estimates	Terre Lane
15	Mineral Reserve Estimates	Terre Lane
16	Mining Methods	
16.1	Open Pit	Terre Lane
16.2	Underground Block Cave	Josh Parsons, Jarek Jakubec, Carl Kottmeier, Brian Prosser
17	Recovery Methods	Todd Harvey, Scott Elfen
18	Project Infrastructure	Scott Elfen, Kevin Murray

Section	Section Name	Qualified Person
19	Market Studies and Contracts	Terre Lane
20	Environmental Studies, Permitting and Social or Community Impact	Larry Breckenridge
21	Capital and Operating Costs	All
22	Economic Analysis	Terre Lane
23	Adjacent Properties	Terre Lane
24	Other Relevant Data and Information	Terre Lane
25	Interpretation and Conclusions	All
26	Recommendations	All
27	References	Terre Lane

Note: Where multiple authors are cited, refer to author certificate for specific responsibilities.

2.2 Purpose of Report

The purpose of the Technical Report is to provide DLP and its investors with a PEA of the Project. This report contains:

- A resource estimate for the Project
- An economic evaluation for the Project costs and revenues
- An independent opinion as to the technical merits of the Project and the appropriate manner to proceed with continuing exploration and project development

It is intended that this report may be submitted to those Canadian stock exchanges and regulatory agencies that may require it. It is further intended that DLP may use the report for any lawful purpose to which it is suited.

2.3 Site Visit and Details of Inspection

2.3.1 SRK

The site visit team consisted of William Espinoza (DLP), Scott Elfen (Ausenco), Hamid Samari (GRE), and Humberto Pichuante (SRK). The team visited the Aurora property from 2 to 3 June 2026.

The purpose of this visit for the SRK professional was to:

- Develop an understanding of the project's geological framework, deposit setting, and key geological controls relevant to rock mass characterization and caveability assessments.
- Review selected drillholes withing the proposed caving area to support rock mass characterization

The first day of the site visit included a tour of the project area, selected drilling platforms, and one of the proposed portal locations. The second day was spent at the core logging facility in Calca (Photo 2-1), where selected drillholes were reviewed and partially logged to support the geotechnical assessment of the proposed caving area.

SRK was given full access to relevant data and conducted interviews with DLP personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyse historical and current geotechnical data.

Photo 2-1: SRK Site Visit Photos

Left: project area (looking northeast)
Middle and right: core facilities and some of the core boxes reviewed during the site visit
Source: SRK, 2026

2.3.2 Ausenco

Scott Elfen visited the site to review surface geotechnical features to verify proposed infrastructure siting. These include:

- June 2, 2026: The QP visited the site to inspect the site for potential infrastructure and dry stack tailings facility (DSTF).
- June 3, 2026: The QP visited DLP Resource's core to review the open pit core and look at general site geology of the overburden and rock for design of infrastructure and DSTF.

2.3.3 GRE

Dr. Hamid Samari, QP, conducted a personal inspection of the Aurora Project and associated core storage facilities on June 2 and 3, 2026. The purpose of the site inspection was to review the exploration activities completed by DLP Resources Inc., observe the geological setting and mineralization of the Aurora deposit, and independently verify selected exploration information used in this Technical Report. The site inspection included the following principal activities:

- Visited the Aurora Project on June 2, 2026, accompanied by DLP geologist William Espinoza.
- Reviewed the project geology and observed representative lithologies, alteration, mineralization, and structural features.
- Inspected and independently verified the locations of 15 selected diamond drillhole collars using a handheld global positioning system (GPS).
- Visited DLP's core storage facility in Calca, Peru, on June 3, 2026, and examined representative diamond drill core from the Aurora drilling programs.
- Reviewed core logging, handling, sampling, and storage procedures and selected pulp samples for independent check-assay verification.
- Compared field observations with the geological information and exploration data provided for the Aurora Project.

Larry Breckenridge visited the site on August 25, 2025, with the following objectives:

- to observe the site physiography and mine infrastructure
- to preliminarily identify potential plant and crusher sites, waste storage sites, and tailings storage sites
- to assess the water conditions and to take environmental geochemistry samples to characterize the potential for Acid Rock Drainage and metal leaching impacts to water quality resulting form the development of the mine
- to observe the hydrology of the project area

2.4 Sources of Information

Drill hole and assay data for the property were generated by various owners and operators and provided to Chris Harman and GRE by DLP. Geologic maps, reports, results from geochemical sampling, geophysical work, and metallurgical testwork were supplied by DLP, including available work by previous owners and operators.

2.5 Units

All currency amounts and commodity prices are stated in US dollars unless otherwise stated. Quantities are stated in metric units. Commodity weights of measure are in grams (g), troy ounces (tr. oz.), or pounds (lbs) unless otherwise stated.

3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Report:

- The following disclosure is made in respect of this Expert: Mr. Oscar Benavides and Tomas Denegri Vargas Rodrigo, Elias & Medrano Lawyers, Lima, Peru, as advised in a letter to DLP with an effective date of 19 August 2026.
- Report, opinion, or statement relied upon information on mineral tenure and status, title issues, royalties and mining concessions.
- Extent of reliance: full reliance following a review by the QPs.
- Portion of Report to which disclaimer applies: Section 4.1.1 and 4.1.2.

4.0 PROPERTY LOCATION AND DESCRIPTION

4.1 Location

The Aurora Property is located in the Department of Cusco, Peru (Figure 4-1). It is located 2.5 kilometers (km) northeast of the Parobamba Community, at an average altitude of 3,000 meters (m) above sea level (masl). Politically, the project belongs to the Yanatile District, Calca Province, Cusco Department. The project is situated on surface lands belonging to the Parobamba Community.

Figure 4-1: Aurora Property Location

Source: DLP, 2025

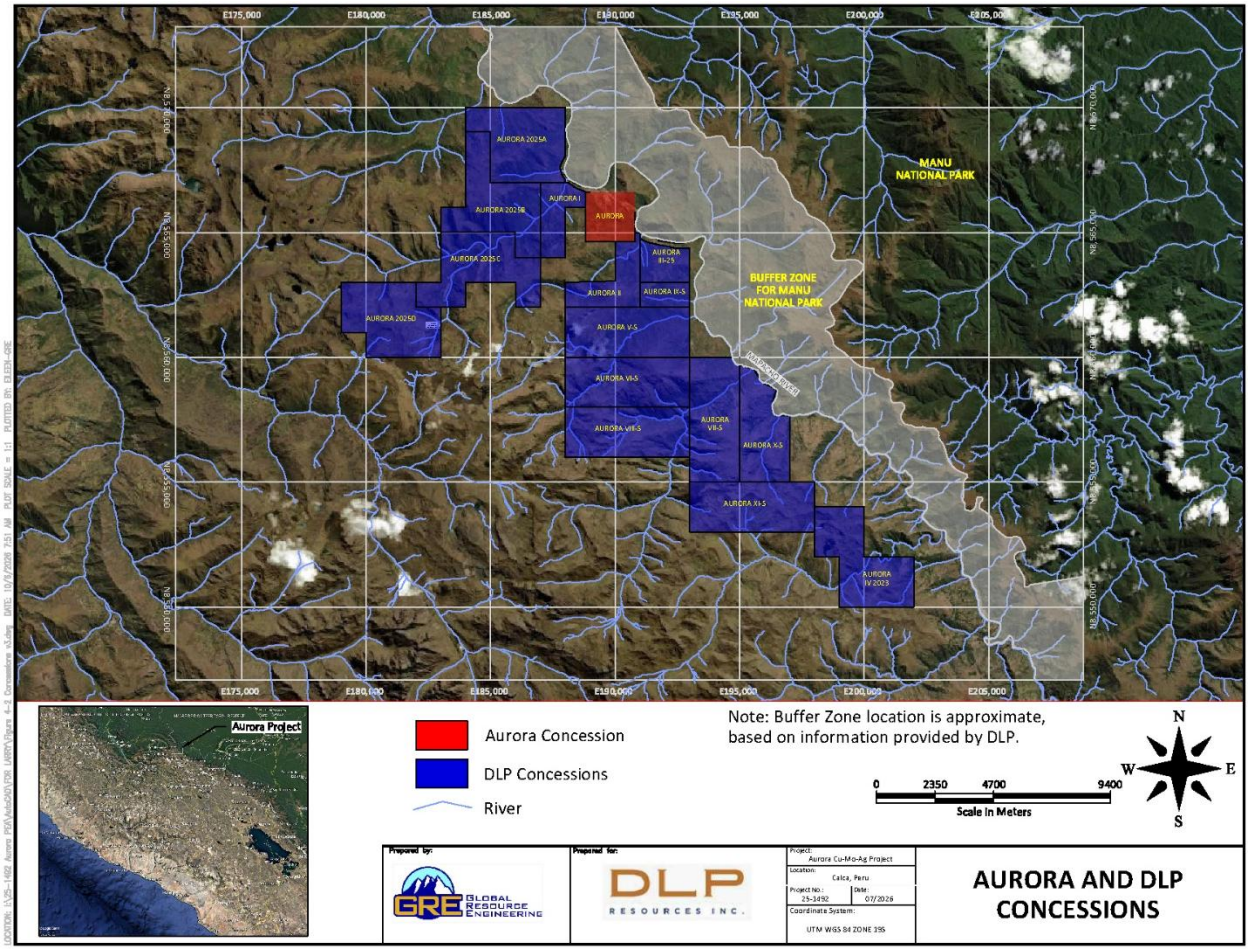
The Aurora Property, which is the subject of this report, is a large land package of 12,588 ha. It is comprised of a core "Aurora" concession and surrounded by additional concessions. To the knowledge of the QP, no exploration activities have been undertaken on the DLP concessions external to, but adjoining, the core Aurora concession. For this reason, with the exception of this section, the discussion in the rest of the report primarily relates to the core Aurora Concession outlined in Table 4-1 and presented in red in Figure 4-2.

Table 4-1: Aurora Concession Boundaries

Vertex	East	North
1	190808.51	8566628.18
2	190808.51	8564628.14

Vertex	East	North
3	188808.49	8564628.14
4	188808.49	8566628.19

Figure 4-2: Map of the Core "Aurora Concession" and other Concessions on the Property



Source: DLP, 2026.

Figure 4-2 shows the concession boundaries and the site physiography.

4.1.1 Core Tenement Ownership

The Aurora Concession, on which exploration drilling has been conducted, comprises a single concession bearing the same name, "Aurora." It is owned by DLP Resources Perú SAC, as it acquired the mining concession on June 5, 2026. Ownership of the Aurora concession is in the process of being recorded with the Public Registry (application No. 2026-02037982). Once registration is finalized, ownership of Aurora concession will be enforceable before the State and third parties. Only the Aurora Concession contains the Mineral Resource Estimate, but the other concessions are held by DLP for project infrastructure or regional exploration.

The Aurora Concession's approximate center is located at UTM coordinates 189850E and 8565700N. It occupies an area of 400 ha and is defined by the vertices shown in Table 4-1, with coordinates in UTM WGS84, Zone 19S.

In addition to the optioned Aurora claim, DLP Resources Peru SAC holds 100% of an additional 11,958 Ha in and around the main Aurora claim. These claims are shown in Figure 4-2.

4.1.2 Other Tenement Ownership

The Aurora Concession is immediately adjoined to the northwest and southeast by exploration concessions held by DLP. The concessions are held for periods of 30 years, as outlined in Table 4-2 with the details of the size and jurisdiction outlined in Table 4-3. According to Rodrigo, Elias & Medrano Lawyers, 2026, the tenements are in "Good Standing".

Table 4-2: Aurora Concession Terms

Concession		Code	10-year Expiration Date	30-year Expiration Date
1	AURORA	40005597	31-Dec-18	31-Dec-38
2	AURORA I	10149022	31-Dec-33	31-Dec-53
3	AURORA II	10149122	31-Dec-34	31-Dec-54
4	AURORA V-S	10139723	31-Dec-34	31-Dec-54
5	AURORA VI-S	10139623	31-Dec-34	31-Dec-54
6	AURORA VII-S	10139823	31-Dec-34	31-Dec-54
7	AURORA VIII-S	10139523	31-Dec-34	31-Dec-54
8	AURORA XI-S	10146023	31-Dec-34	31-Dec-54
9	AURORA IV 2023	10140223	31-Dec-34	31-Dec-54
10	AURORA IX-S	10142423	31-Dec-34	31-Dec-54
11	AURORA X-S	10146123	31-Dec-35	31-Dec-55
12	AURORA 2025A	10063825	31-Dec-35	31-Dec-55
13	AURORA 2025B	10063925	31-Dec-35	31-Dec-55
14	AURORA 2025C	10064025	31-Dec-35	31-Dec-55
15	AURORA 2025D	10064425	31-Dec-35	31-Dec-55

Source: Rodrigo, Elias & Medrano Lawyers, 2026

Table 4-3: DLP Concession Details

Concession	Code	Effective Area (Ha.)	Location District / Province / Department	Registered Titleholder	Registry File (Cusco Office)	Outstanding Validity Fees (US$)		Outstanding Production Penalties (PEN)	
						2025	2026	2025	2026
AURORA	40005597	400	Yanatile / Calca Cusco	S.M.R.L. Parobamba II	20003146	---	---	103,000	107,000
AURORA I	10149022	387.955	Yanatile / Calca / Cusco	DLP Peru	11318023	---	---	Not applicable	Not applicable
AURORA II	10149122	469.918	Yanatile, Challabamba / Calca, Paucartambo / Cusco	DLP Peru	11349705	---	---	Not applicable	Not applicable
AURORA V-S	10139723	1,000	Yanatile, Challabamba / Calca, Paucartambo / Cusco	DLP Peru	11321616	---	---	Not applicable	Not applicable
AURORA VI-S	10139623	1,000	Challabamba / Paucartambo / Cusco	DLP Peru	11321618	---	---	Not applicable	Not applicable
AURORA VII-S	10139823	1,000	Challabamba / Paucartambo / Cusco	DLP Peru	11321619	---	---	Not applicable	Not applicable
AURORA VIII-S	10139523	1,000	Challabamba / Paucartambo / Cusco	DLP Peru	11321620	---	---	Not applicable	Not applicable
AURORA XI-S	10146023	1,000	Challabamba / Paucartambo / Cusco	DLP Peru	11321622	---	---	Not applicable	Not applicable
AURORA IV 2023	10140223	1,000	Challabamba / Paucartambo / Cusco	DLP Peru	11326372	---	---	Not applicable	Not applicable
AURORA IX-S	10142423	200	Challabamba / Paucartambo / Cusco	DLP Peru	11343846	---	---	Not applicable	Not applicable
AURORA 2025B	10063925	1,000	Yanatile / Calca / Cusco	DLP Peru	11363478	---	---	Not applicable	Not applicable
AURORA 2025C	10064025	1,000	Yanatile / Calca / Cusco	DLP Peru	11357395	---	---	Not applicable	Not applicable
AURORA 2025D	10064425	1,000	Lares, Yanatile / Calca / Cusco	DLP Peru	11356110	---	---	Not applicable	Not applicable

Concession	Code	Effective Area (Ha.)	Location District / Province / Department	Registered Titleholder	Registry File (Cusco Office)	Outstanding Validity Fees (US$)		Outstanding Production Penalties (PEN)	
						2025	2026	2025	2026
AURORA X-S	10146123	900	Challabamba / Paucartambo / Cusco	DLP Peru	11363477	---	---	Not applicable	Not applicable
AURORA 2025A	10063825	1,000	Yanatile / Calca / Cusco	DLP Peru	11363479	---	---	Not applicable	Not applicable
Pediment									
Mining Pediment	**Code**	**Requested Area (Ha.)**	**Location District / Province / Department**	**Titleholder**	**Registry file**	**Validity fees (US$)**			
						2025	**2026**		
AURORA III-25	10378425	400	Yanatile, Challabamba / Calca, Paucartambo / Cusco	Bertha Isabel Paredes Solano	Not registered	---	---		

Source: Rodrigo, Elias & Medrano Lawyers, 2026

4.1.3 Manu National Park Buffer Zone

As seen in Figure 4-2, the buffer zone for the Manu National Park is located on the north side of the Mapacho River. No concessions, resources, nor elements of the site infrastructure or footprint extend north of the Mapacho River into the buffer zone. Section 20.0 discusses special permitting considerations associated with the site's proximity to the buffer zone.

4.2 Peruvian Regulatory Framework

This subsection presents the Peruvaian Regulatory framework in general. Section 4.3 discusses the current permits for the project, and Section 20.2 discusses the future permits required to execute the LOM mine plan.

4.2.1 Overview

Mining concessions in Peru grant, subject to obtaining complementary permits, their holder the right to explore and exploit mineral resources within a designated area. The mining concession, however, is distinct and independent from the surface land under which it is located. Concession holders are required to pay an annual validity fee to maintain their claims, and the concession title can be registered with the Mining Registry of the Peruvian Public Registry. Titleholders must comply with production obligations, which include meeting specific output levels within a 30-years period as from the year following the issuance of the mining concession title. Failure to meet these obligations within the first 10 years results in penalties, and concessions can expire if fees (validity fees and/or production penalties) are unpaid for two consecutive years or production requirements are unmet within the aforementioned 30-year deadline.

Before initiating mining activities, concession holders must obtain several permits, including environmental management instruments and approval for surface land use. Mining concessions do not confer ownership over nor the right to use the surface land under which they are located, and holders must negotiate access with the landowner, whether it's a private entity, peasant community, or the State. In addition to these, the Ministry of Energy and Mines requires prior consultation with indigenous communities (when applicable) and citizenship participation mechanisms must be conducted throughout the preparation and evaluation of environmental studies. Compliance of environmental regulations and obligations is monitored by the Organism for Environmental Evaluation and Control (OEFA), while safety and labor obligations are monitored by the Surveillance Organism for the Investment in Mining and Energy (OSINERGMIN) and the National Superintendency of Labor Fiscalization (SUNAFIL). Water rights are separate from the mining concession and must be secured through the National Water Authority.

Mining companies in Peru are subject to several taxes, including the mining royalty, which is calculated as a percentage of operational profits and distributed among local, provincial, and regional governments as well as national universities. The Special Mining Tax applies to companies without tax stability agreements, while the Special Mining Contribution is levied on companies who had such agreements in place when the Special Mining Tax was created. These taxes vary based on the operational margin and are calculated quarterly. The royalty system ensures that mining operations contribute to both local development and government funding.

4.2.2 Exploration, Mining, and Surface Rights

The mining concession grants the holder the right to explore and exploit the mineral resources located within a determined area. It is a different and separate property from the real estate under which it is located, and the concession only grants its holder a property right over the resources extracted from the subsoil. Mining concessions do not confer property or use rights over the surface land, which must be acquired separately through purchase, land use or easement agreements.

The current validity fee for mining concessions / titles is $3 per hectare (ha) per year, which is payable on a yearly basis (June 30) to maintain the claims in good standing.

Titleholders of mining concessions must comply with two primary obligations: the annual payment of the good standing validity fee and adherence to minimum annual production. The validity fee, due by 30 June each year, varies by classification: $3/ha for general regime titleholders, $1/ha for small producers, and $0.50/ha for artisanal producers.

The minimum production obligation mandates specific output levels based on the type of mining activity. Metallic production thresholds range from one Tax Unit (currently PEN 5,500) per hectare per year for general regime holders to 5% of UIT per hectare per year for artisanal producers. Non-metallic thresholds are comparatively lower. Titleholders must achieve these production levels within 10 years from the concession grant date. Failure to meet the minimum production requirement results in penalty fees, unless the titleholder has invested at least 10 times the penalty amount per hectare annually.

Mining concessions expire if the validity fee or production penalties are unpaid for two consecutive years or if minimum production is not met within 30 years of the grant. Expired concessions cannot be reclaimed by the former titleholder or their immediate relatives for two years after being declared available.

4.2.2.1 Aurora Concession Mining and Surface Rights

The Aurora concession status is detailed in Table 4-2 and Table 4-3. There are no outstanding validity fees for the Project. The project is situated on surface lands belonging to the Parobamba Community. Social relations with the Parobamba Community are very favorable; DLP Resources Perú SAC has signed a consecutive six-year agreement with the community (June 2026 to June 2032).

4.2.3 Permitting

Before starting mining operations, concession holders must obtain authorization from the Ministry of Energy and Mines (Minem). This process, when applicable, requires prior consultation with indigenous communities, facilitated by the Ministry of Culture (Mincul). While the Indigenous community's opinion is sought, it is not mandatory for authorization, as Minem has discretionary power in the matter. Additionally, mining concessions do not confer property or use rights over the surface land, which must be acquired separately through purchase, land use or easement agreements.

Environmental regulations set by Minem and the Ministry of the Environment mandate the submission and approval of environmental studies, such as an Environmental Impact Declaration (DIA) or a Semi-detailed Environmental Impact Assessment (EIA-sd), depending on the scale of exploration activities. These studies must address technical, social, and environmental impacts and outline contingency measures through an Environmental Management Plan. Oversight is conducted by OEFA, which monitors

compliance, conducts audits, and enforces fines. Mining companies are also required to submit and implement a Closure Plan backed by environmental guarantees to ensure post-mining rehabilitation.

While the Peruvian government retains ownership of the land and mineral resources, concession holders own the extracted minerals. Water rights, managed by the National Water Authority (ANA), are independent of mining concessions. Companies must secure separate authorizations for water use and adhere to regulations regarding water consumption, yearly payments to ANA, and standards for mineral residue deposits outlined in the Law of Water Resources.

Water usage rights have been secured in accordance with Directorial Resolution No. 0872-2024-ANA-AAA-UV. The extension of the water permit will be applied for once the extension to the current DIA is approved which is in processing and anticipated to be granted in October 2026.

Note: Water permits are independent of mining rights. They are granted once an approved management instrument (Extension of DIA) is in place, and fees are paid annually based on the authorized consumption volume. In our case, we hold a water use authorization (Directorial Resolution No. 0872-2024-ANA-AAA-UV) and intend to renew it once we obtain the extension for the DIA (Declaración de Impacto Ambiental).

4.2.3.1 Aurora Concession Environmental Permits

DLP have an environmental permit to develop 30 platforms on the Aurora Concession (See Section 10 - Table 10.2), and the use of the existing road that connects the community of Parobamba with the mining exploration area. The existing road has a platform width of 4 to 5 meters and requires cleaning of ditches in the last section in an approximate area of 2 km. The environmental permit also enables the installation of auxiliary components such as a safety cabin and a camp of 1602 m^2, and an esplanade area located at the detour of the existing road about 2 km from the platforms projected for drilling.

DLP has an Environmental Impact statement (EIS). The EIS has been developed and framed within the requirements for Category I Mining Exploration projects in accordance with the indications of the Environmental Regulations for Mining Exploration Activities, approved by supreme decree No. 042-2017-EM and its amendment D.S. No. 019-2020-EM.

4.2.3.2 Other Concessions Environmental Permits

As no active exploration is currently being conducted outside the Aurora Concession, no environmental permits have been obtained for the other concessions.

4.2.4 Royalties and Taxes

In addition to corporate income tax, mining companies in Peru are required to pay mining taxes quarterly based on their operational profits. These taxes include:

- Royalty: Established under the 2004 Mining Royalty Law (revised in 2011), this tax applies an effective rate of 1% to 12% on quarterly operational profit, depending on the operational margin across 16 brackets. A minimum royalty payment of 1% of sales is mandatory. Royalties are distributed among local, provincial, and regional governments, as well as national universities.

- Special Mining Tax: This tax applies to mining companies without tax stability agreements. The rate ranges from 2% to 8.4% of quarterly operational profit, depending on the operational margin across 17 brackets.
- Special Mining Contribution: This levy applies to companies with pre-existing tax stability agreements. The rate ranges from 4% to 13.12% of quarterly operational profit, based on the operational margin across 17 brackets.

4.2.4.1 Aurora Concession Royalties

S.M.R.L. PAROBAMBA II has been granted a 1.5% net smelter royalty. This net smelter royalty can be bought out for US$1.5 million at any stage of the project's life.

4.2.5 Additional Permits and Licenses

The title of a mining concession by itself does not authorize exploration or exploitation activities. Concession holders must obtain additional permits and administrative approvals, which include:

- Approval of an environmental management instrument.
- A certificate confirming the non-existence of archaeological remains.
- Authorization to use the surface land from its owner.
- Other permits and licenses as required by applicable laws, depending on the nature and location of the activities.

Once these prerequisites are fulfilled, the concession holder can request operational authorizations from the General Mining Bureau of the Ministry of Energy and Mines. These authorizations, which validate legal mining activities, are issued for specific purposes:

- Exploration: Authorization for exploration mining activities.
- Exploitation: Authorization for exploitation mining activities.
- Beneficiation: Title and operational authorization for mining beneficiation.
- Transportation: Title and operational authorization for mining transportation.
- General Work: Title for general mining-related work.

4.3 Environmental Permits

DLP have an environmental permit to develop 30 platforms on the Aurora Concession (See Section 10 - Table 10.2), the use of the existing road access that connects the community of Parobamba with the mining exploration area. The existing road has a platform width of 4 to 5 meters and requires cleaning of ditches in the last section in an approximate area of 2 km. The environmental permit also enables the installation of auxiliary components such as a safety cabin and a camp of 1602 m_2, and an esplanade area located at the detour of the existing road about 2 km from the platforms projected for drilling. The following approvals were obtained for exploration:

- The EIA for exploration was approved by "Resolución Gerencial N.° 76-2022-GORE CUSCO-GREMH/G", dated May 12th, 2022, and by No. 042-2017-EM and its amendment D.S. No. 019-2020-EM.
- DLP received approval of the Informe Tecnico Sustentatorio (ITS). This approved a schedule update for the workplan, and was approved by "Resolución Gerencial Regional N.° 00326-2024-GR CUSCO/GREMH", dated 4 December, 2024.
- The authorization to start exploration on 11 additional platforms was covered by "Resolución Gerencial Regional N.° 00166-2025-GR CUSCO/GREMH", dated July 11, 2025.
- DLP applied for another ITS which includes a schedule update and additional platforms. This permit received a favorable technical assessment, and approval is pending (procedure started on 8 June, 2026, File No 2913).
- The Authroidad National de Agua (ANA) authorized permission to use water in "La resolución Directoral N° 0872-2024.ANA-AAA-UV" in 2024.

DLP has surface rights from the community, and a community agreement (See Section 20.5).

4.4 Royalties and Encumbrances

S.M.R.L. PAROBAMBA II has been granted a 1.5% net smelter royalty. This net smelter royalty can be bought out for a minimum amount of US$1.5 million (m) at any stage of the project's life.

4.5 Environmental Liabilities

No environmental liabilities exist on site. Some naturally occurring acid-rock drainage impacted seeps exist in the general project area (see Section 20.2) but these are not related to the primary concession, nor are they from past mining activities.

4.6 Risks and Additional Factors

To the extent known by the QP, there are no other significant factors or risks that may affect access, title, or the right or ability to perform work on the Property.

4.7 Conclusions

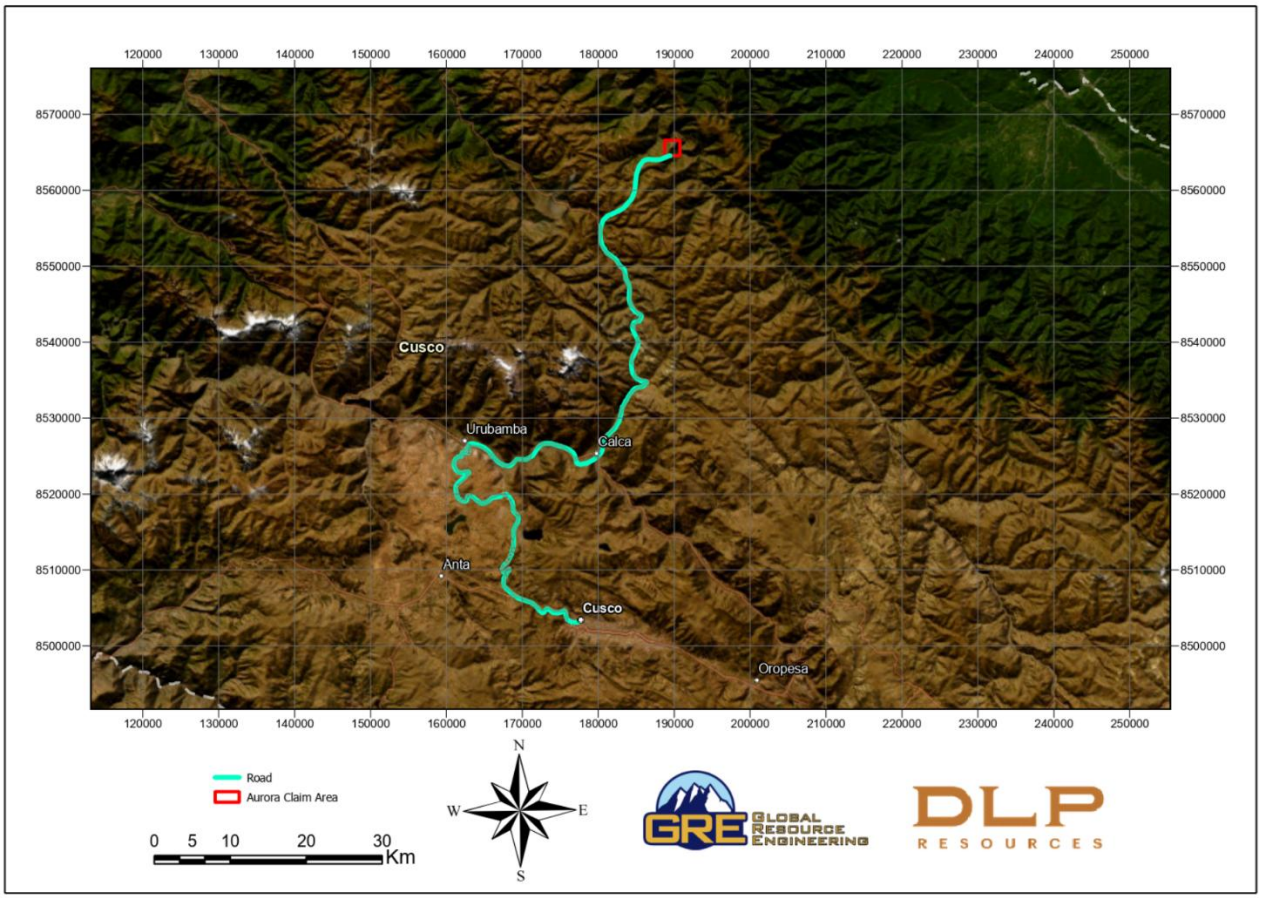
For the core Aurora Concession, all relevant permits have been obtained to conduct the required work on the Property. To the extent known by the QP, there are no other significant factors or risks that may affect access, title, or the right or ability to perform work on the Property. There are also no known environmental liabilities to which the Property is subject.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is located on the surface terrain of the small community of Parobamba, approximately 120 km from the city of Cusco. It is accessed through paved and dirt roads, passing through the towns of Calca, Amparaes, and the village of Parobamba (Figure 5-1).

Figure 5-1: Access Road from Cusco to the Aurora Project

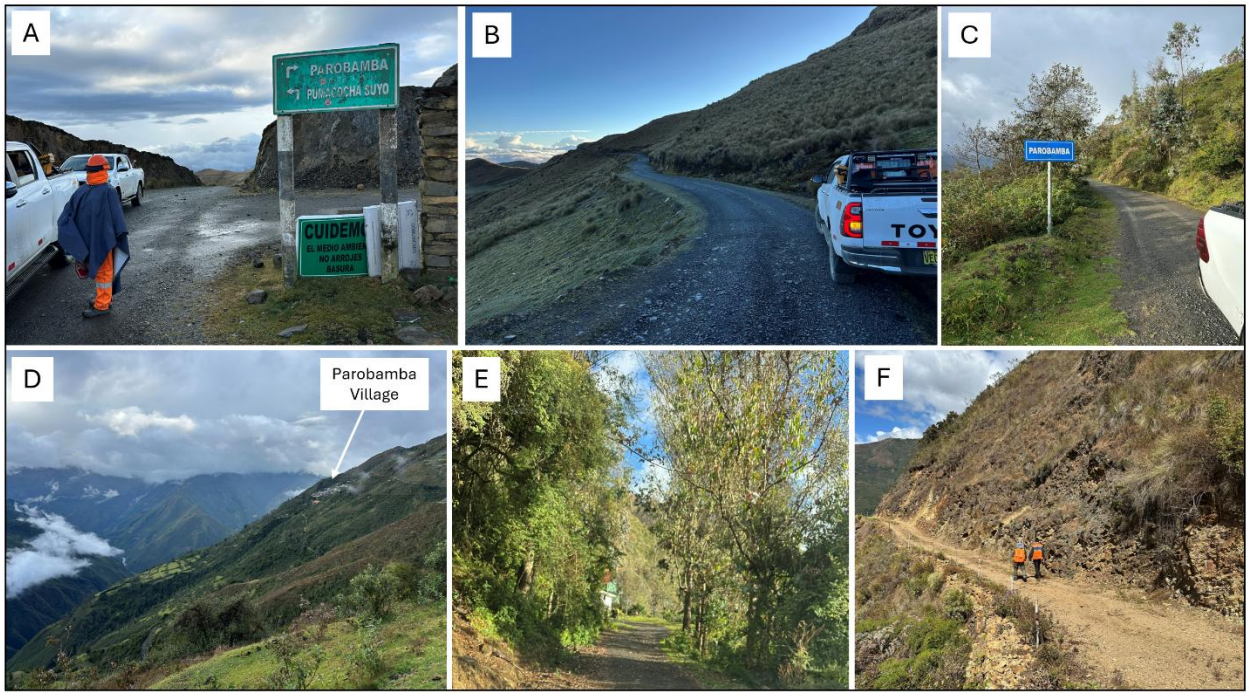


Access from the city of Lima to the concession area begins with a one-hour flight from Lima to Cusco, followed by approximately 4.5 hours of ground travel. From Cusco, the route follows Route 28B (Cusco to Calca) for approximately 70 km (about 1.5 to 2 hours) along a paved road. Beyond Calca, the travel continues along a gravel road through Amparaes and Parobamba, covering approximately 133 km, for about 2.5 hours. From Parobamba, the Project area is accessed via an additional 9 km of gravel road, which requires about 30 minutes of travel to arrive at the Aurora camp.

The drive from Cusco is through the mountainous terrain of the Peruvian Andes. Departing from Cusco, the route climbs in elevation before descending into a valley that hosts the townships of Pisac, Qoya, and Calca. From Calca, the journey rises in elevation through a mountainous pass, reaching a maximum

elevation of 4,586 meters above sea level (masl) at the Abra Amparaes pass. The road then meanders back into a valley containing the township of Amparaes. The Property is located in the community of Parobamba, a subsistence farming community with approximately 165 dwellings (Figure 5-2). The Property has access to electricity and running water.

Figure 5-2: Access Road to the Aurora Project Area

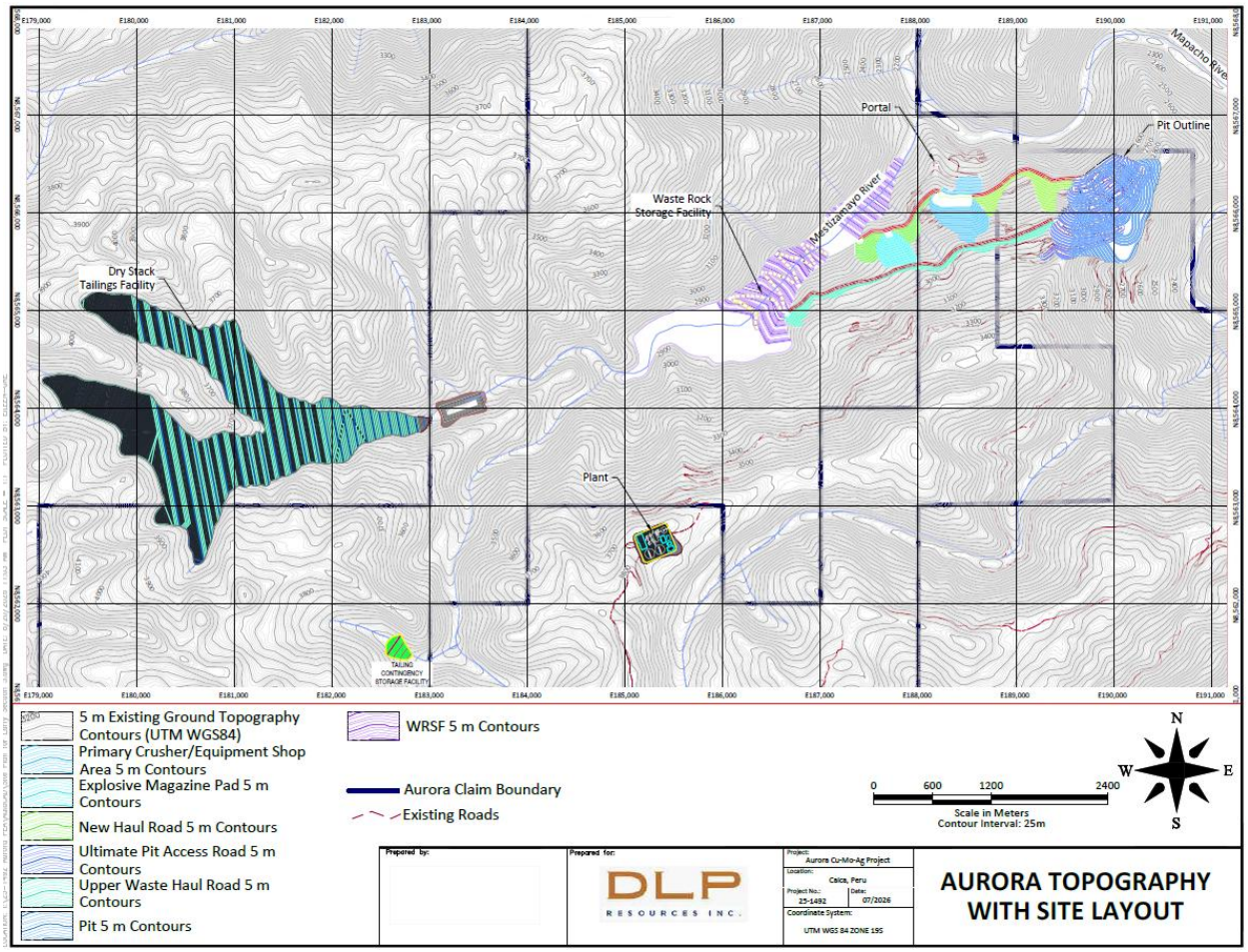


Source: GRE, 2026
A-C): Access Road to Parobamba Village, D): Location of Parobamba Village, E-F): Typical Access Road within the Project Area

5.2 Physiography

Figure 5-3 shows a topographic map of the project area with the major rivers in the project area: The Meztizimayo and the Mapacho Rivers.

Figure 5-3: Site Topography



Source: GRE, 2026

5.2.1 General Physiographic Regions within the Project Site

The area where the future project is located has an elevation of between 2250 meters (at the confluence of the Mapacho river and the Mestizamayo River up to 3800 meters, at the highest level of the DSTF. As a result, the wide range of elevations means that more than one physiographic region is included in the project area.

- The “upland area” with broad U-shaped valleys which have historically been glaciated, and
- The“lower area” with steep-sided V-shaped valleys carved by fluvial erosion.

These regions are driven by different geomorphology along the length of the Mestizamayo River and the Mapacho River. The Mestizamayo river runs east-west across the northern boundary of the site and hosts the DSTF and the WRD. It has a total drainage basin area of 52.7 square kilometers and drains into the Mapacho River. The Mapacho River is a major river with 58,000 square kilometers of drainage basin (draining an area larger than Costa Rica) which runs roughly north-south along the eastern border of the project. Both rivers are perennial, with the Mestizamayo river showing a greater change in flow due to wet/dry seasonality than the Mapacho because of its smaller size.

Photo 5-1 through Photo 5-3 show photographs of the river basins and the physiographic regions described above.

Photo 5-1: Upper Mestizamayo Drainage, with U-Shaped Valley

Source: GRE, 2026

Photo 5-2: Lower Mestizamayo Drainage, with Steep V-Shaped Valley and Tributaries

Source: GRE, 2026

Photo 5-3: Mapacho River at the Curve Around the Ore Deposit

Source: GRE, 2026

5.2.2 Local Ecology and Vegetation

According to the Holdridge system and the Ecological Map of Peru, the life zones within the Project's area of influence are: humid moorland and thorny steppe.

5.2.2.1 Humid Moorland-Subalpine Tropical

Geographically, this area is circumscribed to the high Andean region and along the Western Cordillera of the Andes. It is characterized by the presence of steep slopes, as well as hilly areas. It is the ecology type for the Upland areas of the project. The vegetation in this area is characterized by grasslands. Local farmers raise potatoes on small plots in this area, and graze llamas and alpacas.

5.2.2.2 Thorny Steppe-Low Tropical Montane

This life zone has an altitude between 2,000 and 3,100 masl. The maximum average annual biotemperature is 18.2°C and the minimum annual average is 12.1°C. This is the ecology zone for the lowland area of the project, down to the Mapacho River. According to the Holdridge diagram, this life zone has an average potential evapotranspiration variable between two and four times the value of precipitation. This area has a mixture of vegetation. There are some trees, shrubs, grass, and farming fields. Slope aspect (north facing or south facing) tends to define the vegetation, as does the steepness of the slope, with trees on the steepest slopes and cultivated fields on the flat slopes. Photo 5-4**Error! Reference source not found.** shows a photograph of the south side of the Mestizamayo drainage, showing clear signs of slope-aspect controlled vegetation.

Photo 5-4: Vegetation Patterns in the Mestizamayo Drainage

Source: GRE, 2026

5.2.3 Physiographic Impacts on Project Infrastructure and Planning

The lower reaches of the site have steep topography, which necessitated the following design decisions:

- Filtered tailings must be used because neither the upland nor the lowland valleys have sufficient storage capacity to hold the life of mine (LOM) of conventional tailings (which have low density), but the U-shaped upper valleys (see Photo 5-1) have sufficient storage for the dry-stack tailings.
- The waste rock dump must fill the valley of the Mestizamayo drainage basin because no other viable alternatives exist.
- Mine Infrastructure, such as the processing plant, must be located on the ridgetops because the side-slopes of the valleys are too steep.
- Haul roads require considerable “cut” earthworks. It is unlikely fill-based roadbeds will be stable on the side slope of a 1:1 H:V hillside.
- Steep access ramps must be made, especially near the ore deposited on Coropuna Hill (3682 m in elevation), which is bordered to the north, northwest, and northeast by the steep canyon of the Mapacho River (at an elevation of ~ 2225 masl). (see Photo 5-5, showing the oxidation of the ore deposit and the Mapacho river, looking from the north to the south).

Photo 5-5: Mapacho River and Coropuna Hill

Source: GRE, 2026

5.3 Surface and Mineral Rights

At present, the site has sufficient surface rights to continue development of the project (see Section 4.0). Power availability is sufficient to continue exploration, but not sufficient for mineral processing. Water is abundant (see Section 18.0 and 20.0) due to the presence of both the Mestizamayo and Mapacho Rivers. The site is remote and does not have a readily-available population of workers experienced in mining. Section 20.0 discusses the community in greater detail.

5.4 Climate

According to Warren Thornthwaite's climate classification method, the Project's area of influence has a temperate semi-dry climate with a dry winter, with total annual rainfall of 500 to 1,000 millimeters (mm) and average annual temperatures of 12 to 14 °Celsius (C).

The nearest meteorological station is the Challabamba meteorological station, located in the district of Challabamba, province of Paucartambo, department of Cusco, Latitude: 13° 6' 30.77'', Longitude: 71° 43' 12.01'', Altitude 2,699 masl. It is approximately 38 kilometers from the site and at a similar elevation. At this station, average temperature varies from 11.78°C to 15.34°C, corresponding to the months of July and December, respectively. Both Urano (Urano, 2022) and GRE have analyzed the climate data at the Challabamba monitoring station. Urano averaged the last five years, and GRE used the driest, average, and wettest year in the 7-year record available (Table 5-1).

Table 5-1: Meteorological Data for the Challabamba Station

Month	GRE Rainfall Analysis			Urano Rainfall (Urano, 2022)
	Dry Year Rain (mm/month)	Average Year Rain (mm/month)	Wet Year Rain (mm/month)	Average of Last Five Years (mm/month)
January	201.9	142.3	253.4	196.12
February	181.4	267.6	205	251.6
March	110.9	117.6	150.7	132.34
April	15.5	46.2	42.9	59.3
May	14.8	16.1	49.6	23.22
June	0.4	15.8	37.1	27.56
July	17.1	7.6	31	25.18
August	27.1	2.9	7.3	25.32
September	26.7	46.2	23.3	26.98
October	9.7	20.1	134	77.8
November	21.1	167.7	130.4	126.08
December	92.4	62.7	94.2	145.65
Total	**717 mm/yr**	**912.8 mm/yr**	**1158.9 mm/yr**	**1117.5 mm/year**

As seen above, the values used by GRE in the water analysis (see Section 18.0) and the values used in the Declaracion de Impactos Ambientales are similar.

The weather on the Property does not affect the length of the exploration season, nor is it likely to have a significant impact on the construction or operation of the project.

6.0 HISTORY

6.1 Introduction

As discussed in Section 4.0, the work on the Property has been wholly focused on the Aurora Concession. To the QP's knowledge, there has been no historical work done on the other concessions. The following sections pertain only to the Aurora Concession.

6.2 Property Ownership

The Aurora 400 ha concession was first staked and officially recorded in Ingemmet (Peru's tenure management system) on 3 July 1997 by Luis Alberto Elorrieta Valencia. Subsequent to registering the claim by Mr Luis Valencia, there have been transfers of the claim to various entities both private and public from 1997 to when DLP Resources optioned the claim from S.M.R.l Parobamba II in 2022.

History of the Aurora claim owners from 1997 to 2024 is:

- 03/07/1997: Luis Alberto Elorrieta Valencia
- 10/01/2001: Cabiese Crovetto Guillermo Fernando
- 10/01/2001: Bear Creek Mining Company Sucursal Del Peru (Bear Creek)
- 02/03/2006: COMPAÑIA LAS DUNAS DE ORO S.A.C.
- 05/04/2006: Luis Alberto Elorrieta Valencia
- 21/05/2013: Minera Focus S.A.C. (Focus)
- 16/04/2019: S.M.R.L Parobamba II
- 16/04/2022: DLP Resources Peru S.A.C.

DLP Resources Perú SAC acquired the mining concession via a four-year option agreement with S.M.R.L. PAROBAMBA II signed on 21 November 2021. The effective date of the Option Agreement was 22 May 2022, when all permits for drilling were obtained.

6.3 Overview of Exploration and Drilling

Exploration, including drilling, has been undertaken in four distinct phases at the Aurora Concession:

- Bear Creek first conducted exploration on the Aurora Concession in 2001, drilling six drillholes for a total of 1,560 meters.
- Vena Resources followed this program in 2005 and drilled an additional seven drillholes totaling 2,359 meters.
- Minera Los Quechua optioned the project briefly in 2008. However, no information is available on the company's work.
- In 2012, Minera FOCUS SAC (FOCUS) acquired the Aurora Concession, conducting surface mapping and sampling (120 samples), logging of remaining diamond drill cores (539 meters), and geophysics (IP & MAG), covering 43.8 km with FUGRO.

6.4 Surface Sampling

FOCUS collected 120 surface geochemical samples during 2012. The results of this work are discussed in Section 9.1.1.

6.5 Surface Geophysics

In 2012, FOCUS conducted a ground Induced Polarization (IP) and magnetics survey. The surveys were conducted by Fugro and totaled 43.8-line km. No further geophysics have been conducted on the Aurora Project since this time. The results of this work are discussed in Section 9.2.

6.6 Drill Testing

Two historical drill programs have been conducted on the Aurora Concession since 2001. The drill programs are outlined in Table 6-1.

Table 6-1: Summary of Historical Drilling of the Aurora Concession

Year	Company	Number of Drillholes	Total Depth (m)	Min Depth (m)	Avg Depth (m)	Max Depth (m)
2001	Bear Creek	5	1256.1	84.0	251.2	407.0
2005	Vena Resources	7	2359.7	160.0	337.1	604.0

Due to the inability to verify the 2001 and 2005 drill programs, these drillholes have been excluded from generating the geological model and Mineral Resource estimate. The collar locations and significant intercepts are outlined in Section 10.0.

6.7 Historical Resources and Reserves Estimates

There are no known historical estimates of Mineral Resources or Mineral Reserves at the Property.

6.8 Production

No historical mining has been undertaken.

7.0 GEOLOGIC SETTING AND MINERALIZATION

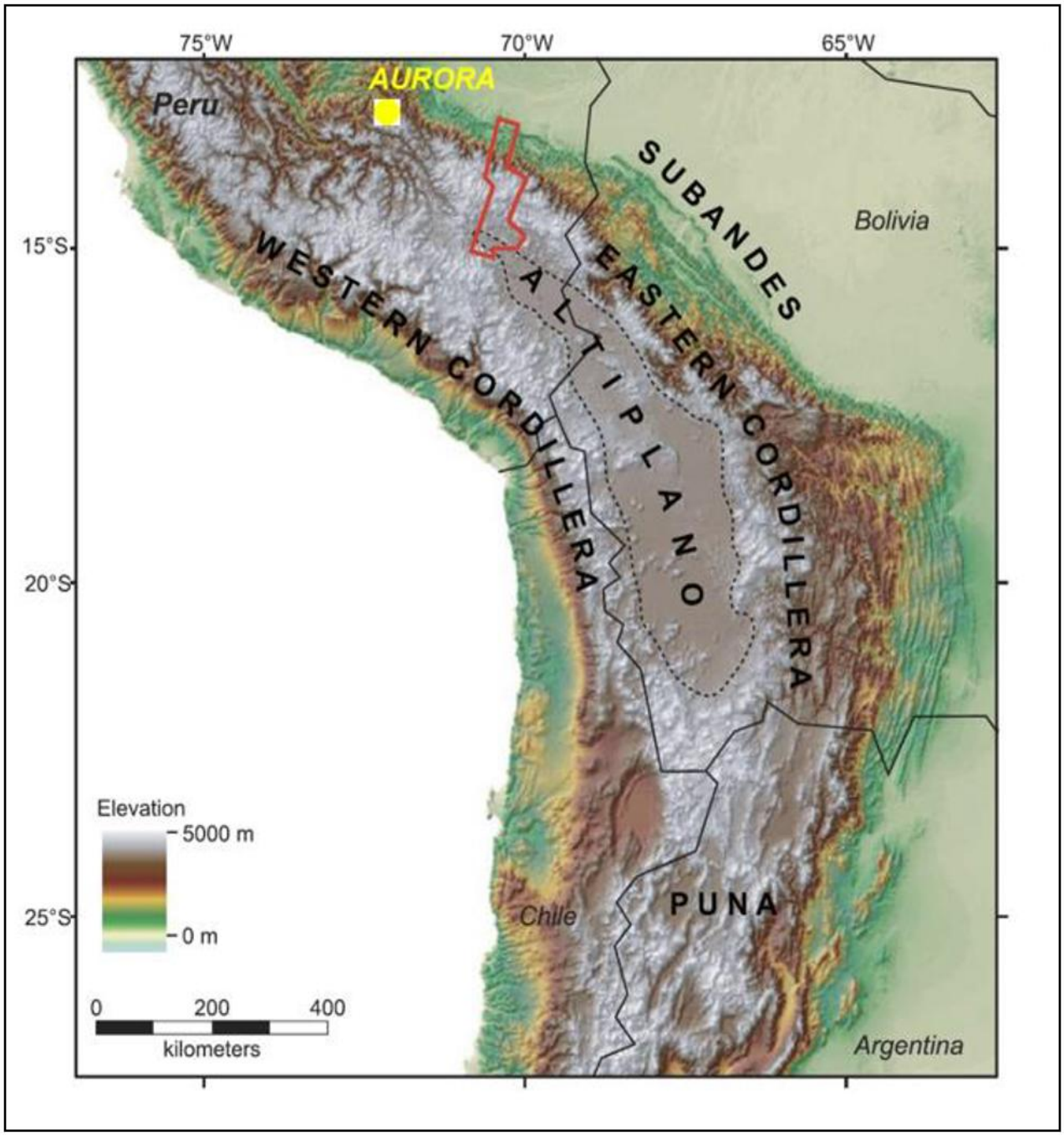
The following description of the regional and local geology, lithology, structure, mineralization, and alteration at the Aurora Project is based primarily on the 2025 NI 43-101 Technical Report prepared by AMC for DLP Resources Inc. (AMC, 2025). GRE's QP, Dr. Hamid Samari, reviewed the 2025 report and the available supporting geological information in detail, including observations made during a site visit to the Aurora Project on June 2, 2026. Based on this review, the geological descriptions and interpretations were found to be reasonable and appropriate for use in this Technical Report. Where warranted, the text has been updated and expanded to incorporate additional geological observations and provide further clarification.

7.1 Regional Geology and Metallogeny

7.1.1 Tectonic Framework

The Aurora Project is located within the Eastern Cordillera of southern Peru, a complex tectonic belt at the eastern margin of the Andean orogen (Figure 7-1). The regional tectonic framework reflects a prolonged history of convergence between the Nazca Plate and the South American Plate, resulting in multiple episodes of subduction, magmatism, deformation, and crustal uplift from the Paleozoic through the Cenozoic. These tectono-magmatic processes have played a fundamental role in the development of numerous mineral deposits throughout the region, including copper (Cu)-molybdenum (Mo)-silver (Ag) porphyry systems such as the Aurora Project.

Figure 7-1: Topographic Map of Peru Showing the Aurora Project and Structural Domains of Peru



Note: The red outline relates to geology, further discussed in the original paper.
Source: Perez, Horton, McQuarries, Stubner, and Ehlers, 2016

7.1.2 Property Geology

The following description of the property geology is based primarily on geological observations and investigations conducted within the Aurora Concession, where exploration activities have been concentrated. The geological framework is interpreted to be generally representative of the broader DLP concession package; however, this interpretation has not been verified across the remaining concessions because geological mapping, sampling, and drilling outside the Aurora Concession have been limited.

7.1.2.1 Main Lithological Units

The stratigraphic units present range from the Lower Paleozoic to the recent Quaternary. The Lower Paleozoic consists sequentially of the San José Group, Sandia Formation, and Ananea Formation, forming a thick sequence of 6,000 meters. The San José Group (2,000 meters) comprises black slates and fine sandstones. The Sandia Formation (2,000 meters), of Late Ordovician age, consists primarily of fine quartz sandstones with interbedded slaty shales in smaller proportions. The Ananea Formation (2,000 meters) comprises black slates with interbedded quartz sandstones and gneisses (Salas, et al., 1999).

In the study area, there are large intrusive bodies ranging in composition from granites to alkaline granites, coarse-grained, holocrystalline, and leucocratic. A second group is represented by subvolcanic magmatism of basic composition and fine-grained, emplaced as sills and dikes within the Lower Paleozoic rocks (Salas, et al., 1999).

7.1.2.2 Main Structures

Structural characteristics indicate the effects of three tectonic phases: the Eo-Hercynian phase, responsible for the folding and metamorphism of Lower Paleozoic rocks; the late Hercynian phase, responsible for the folding and faulting of Upper Paleozoic rocks; and the Peruvian phase, responsible for the structuring of the Palotoa syncline, which hosts the Aurora Deposit, and accentuation of folding and faulting of pre-Cretaceous terrains (Salas, et al., 1999).

7.2 Local Geology

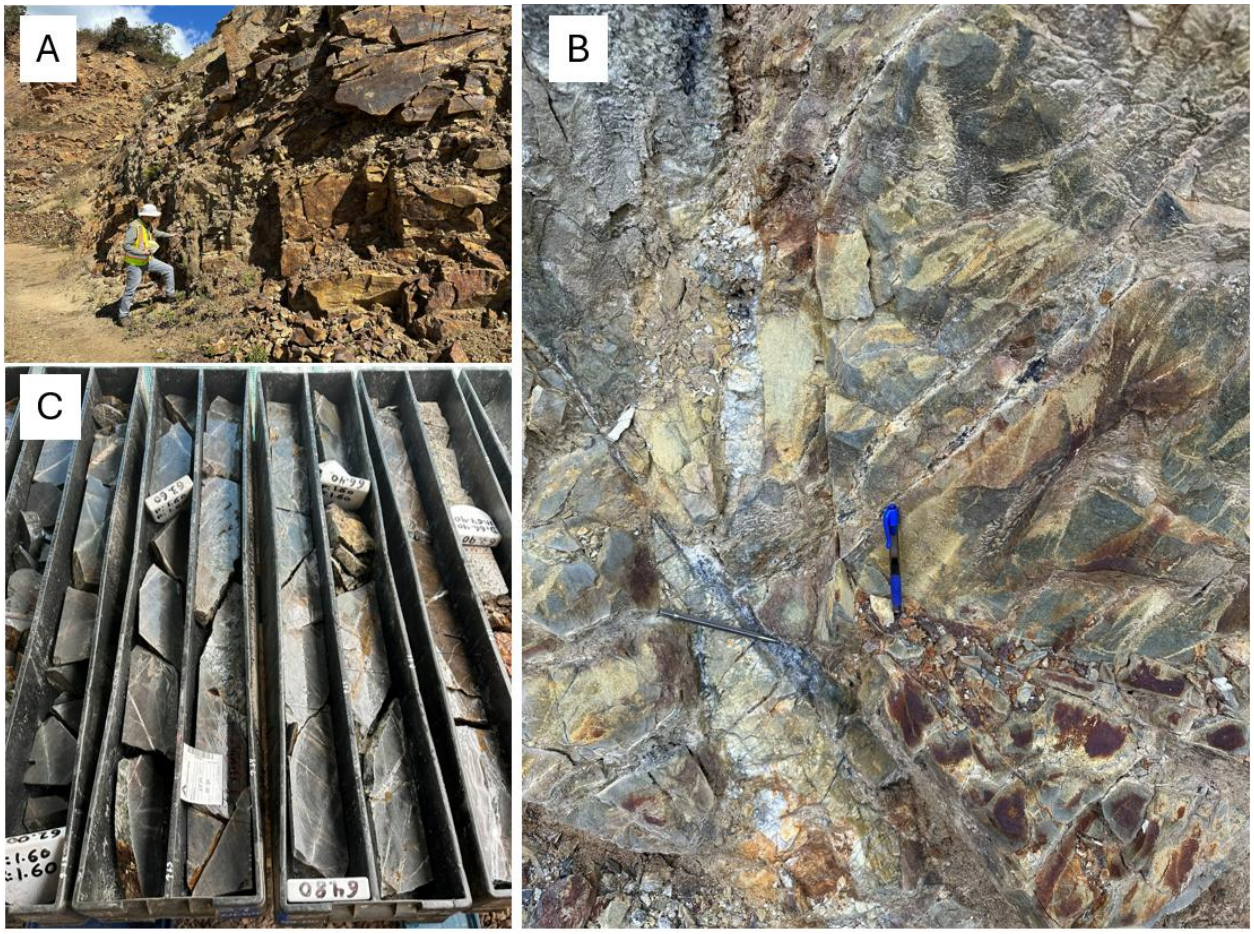
The Aurora concession area shows the following geological characteristics.

There are three main rock types: the Sandia Formation (sedimentary rocks), sub-volcanic rocks, and intrusive rocks that intrude the Sandia Formation.

7.2.1 Sedimentary Rocks (Sandia Formation)

The Sandia Formation was named by Laubacher (1978) for a Lower Paleozoic stratigraphic unit composed of quartzites and slates. Regional reconnaissance has identified the sedimentary characteristics of this formation, which may reach a thickness of 2,000 meters. The lower part of the Sandia Formation comprises black slates interbedded with minor proportions of gray sandstones in 3 to 5 centimeter (cm) thick layers. Lenticular bedding is also observed within the slate packages. The slate packages generally exhibit millimeter-scale lamination, with an approximate thickness of 500 meters. Higher up, alternations of sandstones and slates are present. The sandstones are quartzitic and occur in 5- to 10-cm thick layers, forming metric packages with occasional limonite laminations. This section has an approximate thickness of 1,200 meters (Photo 7-1). The upper part consists of black slates in metric packages with interbedded fine-grained sandstones (Salas, et al., 1999).

Photo 7-1: Sandia Formation



Source: GRE, 2026

A) Outcrop view, B) Quartz veins cutting the Formation, and C) Drill core from A22-001

7.2.2 Subvolcanic and Intrusive Rocks

The subvolcanic rocks are andesitic to dioritic in composition. They appear as sills and dikes intruding into only the Sandia Formation.

The intrusive rocks are Miocene granodioritic to quartz-monzonitic intrusions collectively referred to as the "Aurora Porphyry" in the area of interest. These rocks cut through the Sandia Formation and have been emplaced along the Andean trend (northwest-southeast). These rocks are discussed in more detail below.

7.2.2.1 Early Porphyry

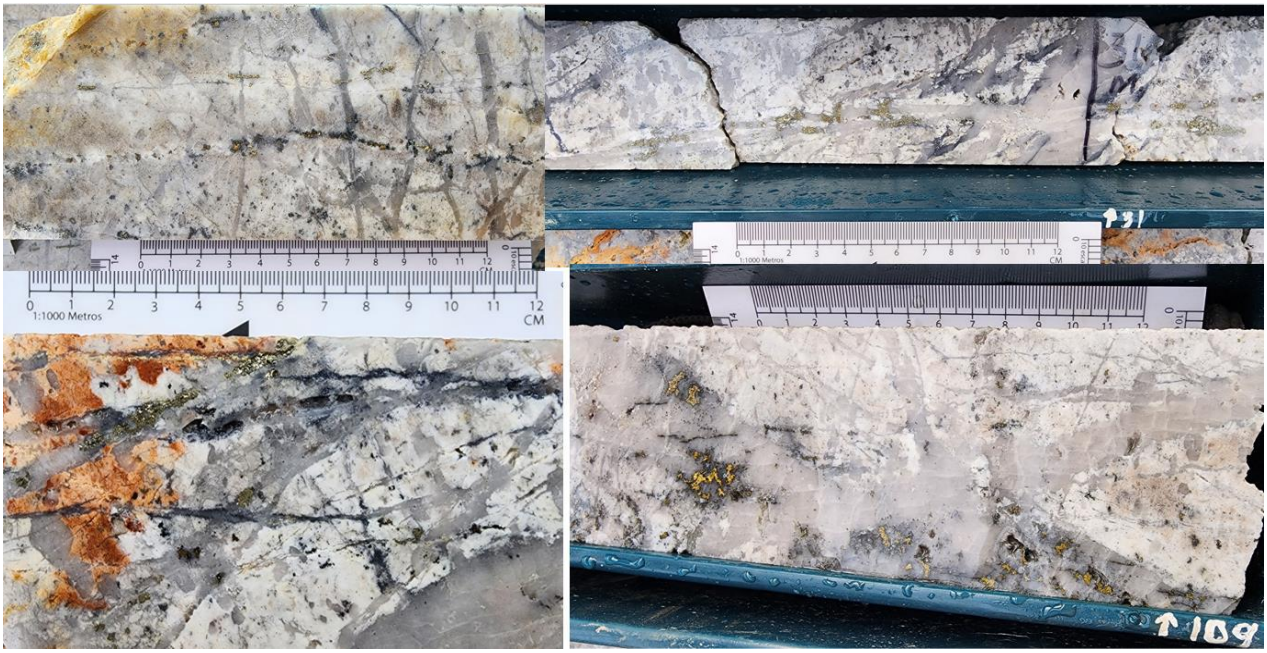
Early Porphyry (QEFP1: Quartz Eyes Feldspar Porphyry) is granodiorite to quartz monzonite in composition (Photo 7-2). It is common for quartz veinlets ranging from millimeters to < 5 cm in width to cut this unit. The quartz in the veinlets is subhedral, sometimes with granular or saccharoidal textures. The veinlets are usually straight and lack alteration halos. The veinlets can reach up to 20% abundance by volume (Photo 7-3). The characteristic mineralization is molybdenite, found on the edges or disseminated at the center of the veinlets, sometimes associated with pyrite. Weak chalcopyrite is disseminated within the veinlets.

Photo 7-2: General View of the Early Porphyry

Source: GRE, 2026

Alteration is potassic, primarily with secondary biotite (shreddy biotite) and, to a lesser extent, pinkish potassic feldspar. Potassic alteration sometimes coexists with quartz-sericite alteration, although the latter tends to dominate. Sericite exhibits euhedral crystals and varies in color from whitish greyish to greenish grey, depending on the intensity of intermediate argillic overprinting. The Early Porphyry outcrops are at the NW part of the concession and had been intersected in drilling.

Photo 7-3: Early Porphyry, Core samples from Hole A23-005



Source: DLP, 2024

7.2.2.2 Intermineral Porphyry

Intermineral Porphyry (QEFP2: Quartz Eyes Feldspar Porphyry) is granodiorite to quartz monzonite in composition. It is common for quartz veinlets ranging from millimeters to < 0.5 cm in width to cut this unit. The quartz is subhedral, sometimes with granular or saccharoidal textures. The veinlets are usually straight, lack alteration halos, and can reach up to 2% abundance by volume (Photo 7-4). The characteristic mineralization is molybdenite, found on the edges or center of the veinlets, sometimes associated with pyrite. Weak chalcopyrite is disseminated within the veinlets. Alteration is quartz-sericite. Sericite presents euhedral crystals and is mainly whitish. Intermediate argillic overprinting is observed.

Photo 7-4: Intermineral Porphyry, Core sample from Hole A24-014

Source: GRE, 2026

There is also an Intermineral Porphyry with alteration and mineralization mainly characterized by macroscopic chalcopyrite disseminated and associated with quartz-sericite alteration. Sericite exhibits euhedral crystals selectively or pervasively replacing feldspars and plagioclase; it varies in color from whitish greyish, to greenish grey, depending on the intensity of intermediate argillic overprinting. Remnants of potassic alteration, mainly with secondary biotite (shreddy biotite), are also observed. Additionally, millimeter-scale quartz veinlets are present, reaching up to 2% abundance.

The Intermineral Porphyry is widespread in the concession, mainly it's in the upper part of the system, cutting or in contact with the country rocks of Sandia Formation.

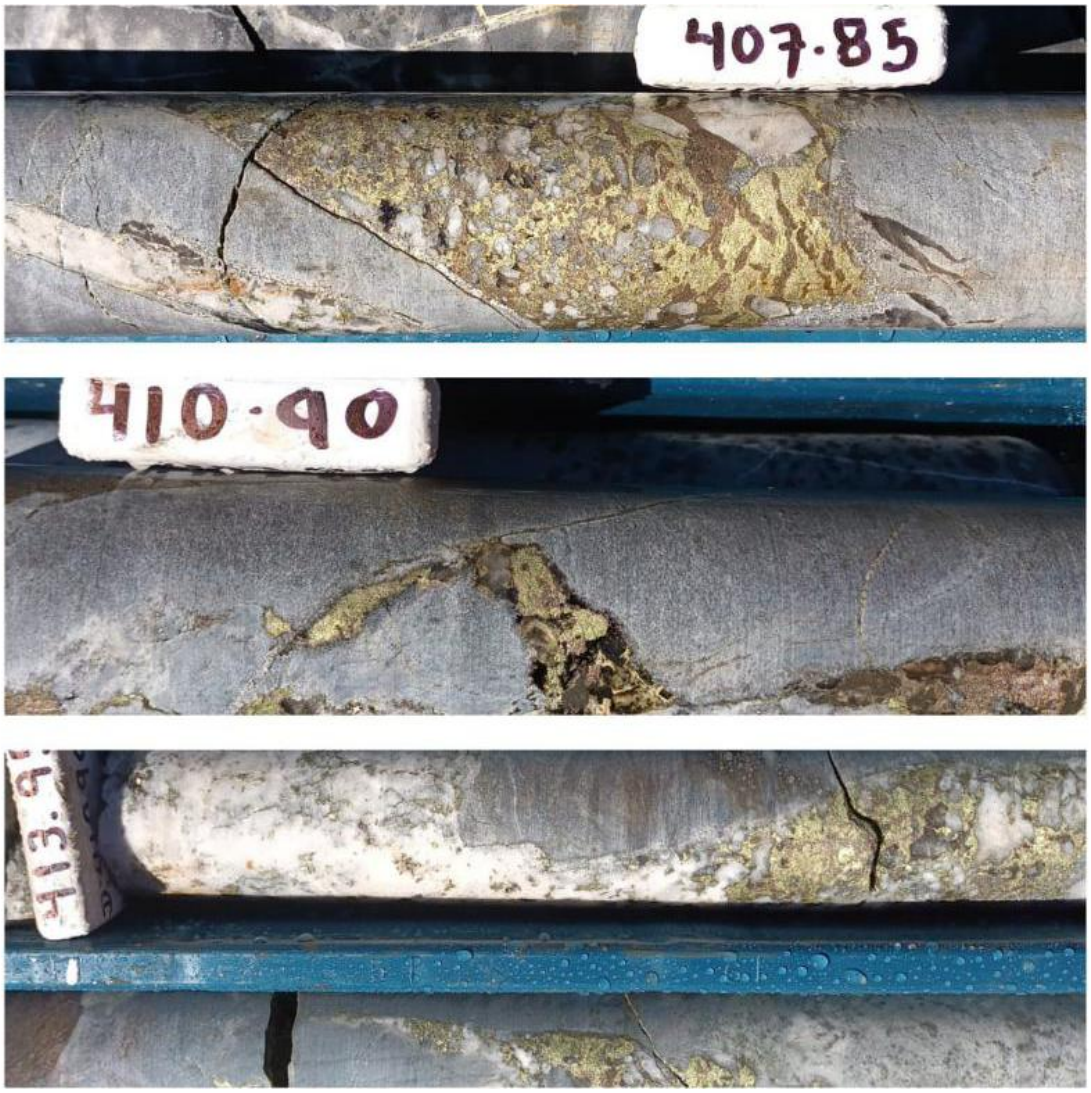
7.2.2.3 Late Intermineral Porphyry

Late Intermineral Porphyry (QEFP3: Quartz Eyes Feldspar Porphyry) is granodiorite to quartz monzonite in composition. It is commonly cut by pyrite veinlets of up to < 0.3 cm in width. These veinlets are straight and lack alteration halos. The veinlets can reach up to 1% abundance by volume. The characteristic mineralization is pyrite and traces of disseminated chalcopyrite. Alteration is weak chloritization associated with igneous biotite. Weak intermediate argillic overprinting is present. The late Intermineral Porphyry is limited to a few holes and occurs as dikes.

7.2.3 Brecciation

Zones of brecciation are observed on the contacts of the porphyry units and the surrounding Sandia Formation (Photo 7-5 and Photo 7-6). The breccias start as intrusive-dominated, grading through to hornfels-dominated breccias.

Photo 7-5: Breccia Development in Country Rock



Source: DLP 2025

Photo 7-6: Breccia Development in Sandia Formation

Source: GRE, 2026

7.2.4 Hydrothermal Alteration

The Aurora Cu-Mo-Ag Porphyry is primarily characterized by potassic, intermediate argillic, and quartz-sericite-pyrite alterations. Below is a description of each zone and its implications.

7.2.4.1 Potassic Alteration

Potassic alteration is characterized by the presence of secondary biotite and potassium feldspar; the replacement in the original rock has been selective to pervasive. Disseminated magnetite is associated with this alteration. The potassic alteration has been observed in the deeper part of the system that overprints the intrusive rocks.

7.2.4.2 Intermediate Argillic Alteration

This alteration is superimposed on the potassic alteration and exhibits montmorillonite, illite, chlorite, with possible traces of residual feldspars. The replacement of the original minerals has been moderate (Photo 7-7). Pyrite is associated with this alteration.

Photo 7-7: Early Porphyry with Intermediate Argillic Alteration

Source: DLP, 2024

7.2.4.3 Quartz-Sericite-Pyrite Alteration

The quartz-sericite-pyrite alteration is characterized by the presence of sericite, both white and greenish grey. It mainly appears as halos around some veinlets. Pyrite is associated with this alteration. Additionally, mica (muscovite) is frequently observed in association with some veinlets, along with quartz and fluorite, which can be considered a type of “greisen” alteration. In general, two distinct sericite alteration styles are recognized in the system. The first is associated with higher temperature and is characterized by a greenish to greyish color that commonly hosts chalcopyrite mineralization. The second is associated with lower temperature and is characterized by white color overprinting and replacing the earlier mineralization with pyrite mineralization.

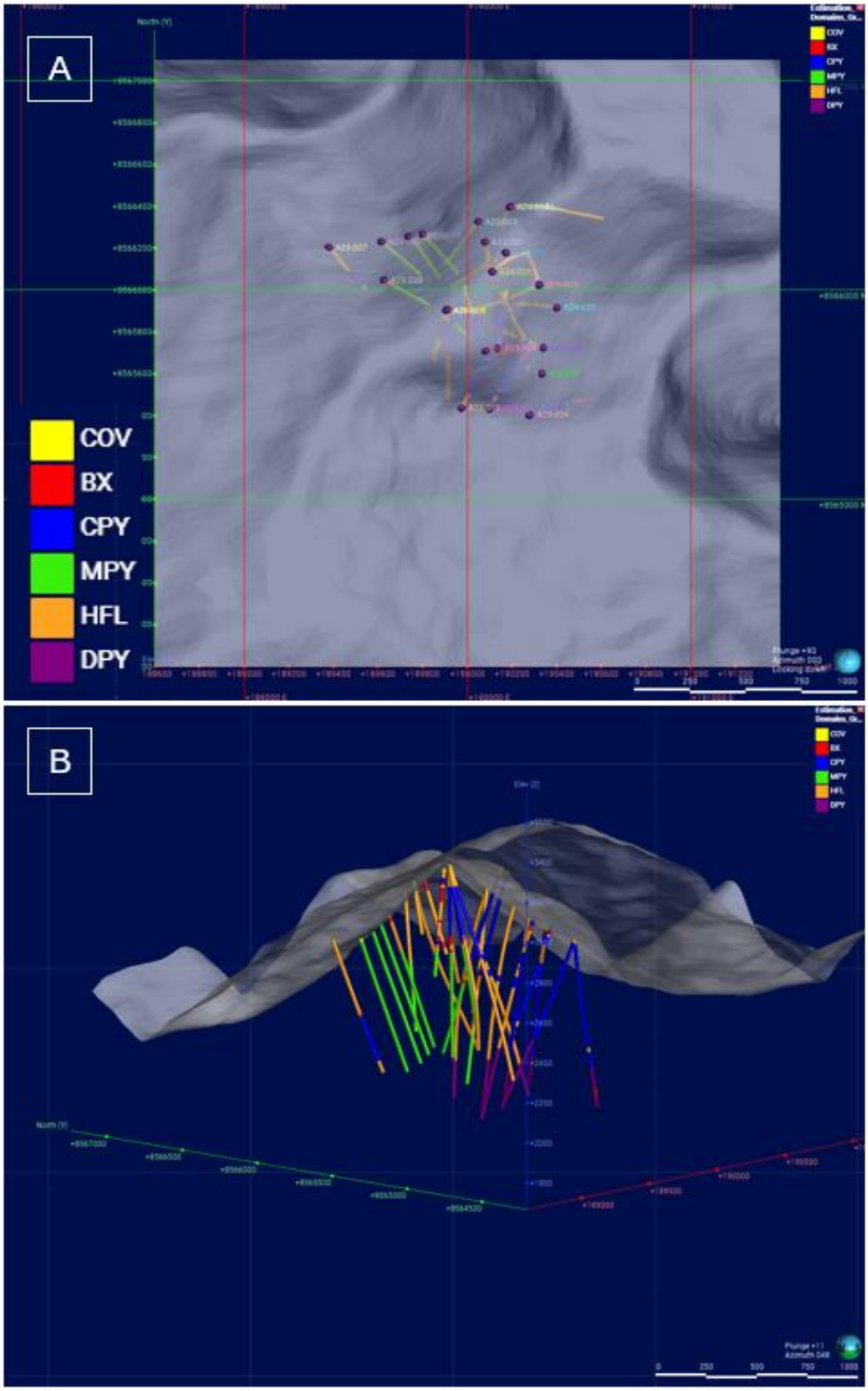
7.2.5 Three-Dimensional Geological Model

GRE's QP updated and interpreted the three-dimensional (3D) geological model using the revised DLP drill-hole database, incorporating all drilling completed between 2022 and 2025, including 28 diamond drill holes totaling 22,613.38 m. The updated model integrates these data with historical drilling completed by previous operators to refine the geometry and spatial distribution of the principal lithological units across the Aurora deposit. The distribution of the historical and recent drill holes used to constrain the geological interpretation is shown in Figure 7-2, which illustrates both the plan and oblique views of the drill-hole database.

The geological model was developed in Leapfrog Geo using implicit modeling techniques based on detailed lithological logging from all available drill holes. Geological contacts were constrained by drill-hole observations and refined through iterative review of plan views, cross sections, and three-dimensional visualizations to ensure consistency with the available geological information. The model was prepared by the GRE Qualified Person and subsequently used as the geological framework for the 2026 Mineral Resource Estimate.

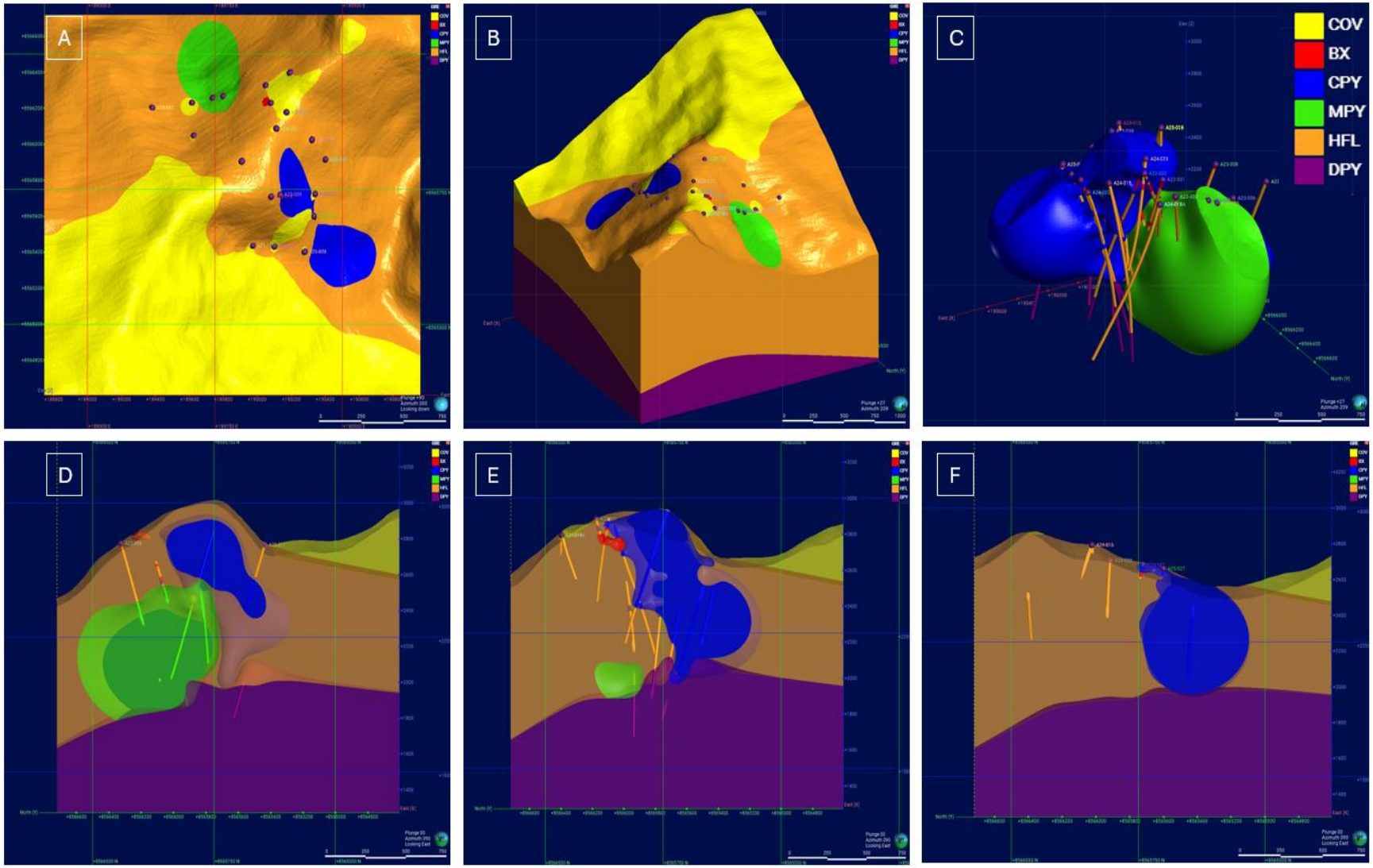
The geological model includes the principal lithological domains recognized during drilling, consisting of Cover (COV), Breccia (BX), Copper Porphyry (CPY), Molybdenum Porphyry (MPY), Hornfels (HFL), and Deep Porphyry (DPY). The CPY and MPY represent the principal intrusive phases associated with the porphyry system, whereas the DPY represents a deeper intrusive phase recognized in the recent drilling. The HFL domain comprises thermally metamorphosed rocks of the Sandia Formation that form the principal country rock surrounding the intrusive complex. BX represents localized intrusive and hydrothermal breccia bodies, while COV consists of post-mineral unconsolidated surficial deposits. The completed three-dimensional geological model, including plan views, oblique views, and representative cross sections, is presented in Figure 7-3, illustrating the geometry, continuity, and spatial relationships of the principal lithological units across the Aurora deposit.

Figure 7-2: Drill-Hole Distribution, Plan View (A) and Oblique View (B)



Source: GRE, 2026

Figure 7-3: Three-Dimensional Geological Model: Plan View (A), Oblique Views (B), 3D view of CPY & MPY (C), and Cross Sections: at 190000E (D), at 190200E (E), and at 190400E (F) looking East



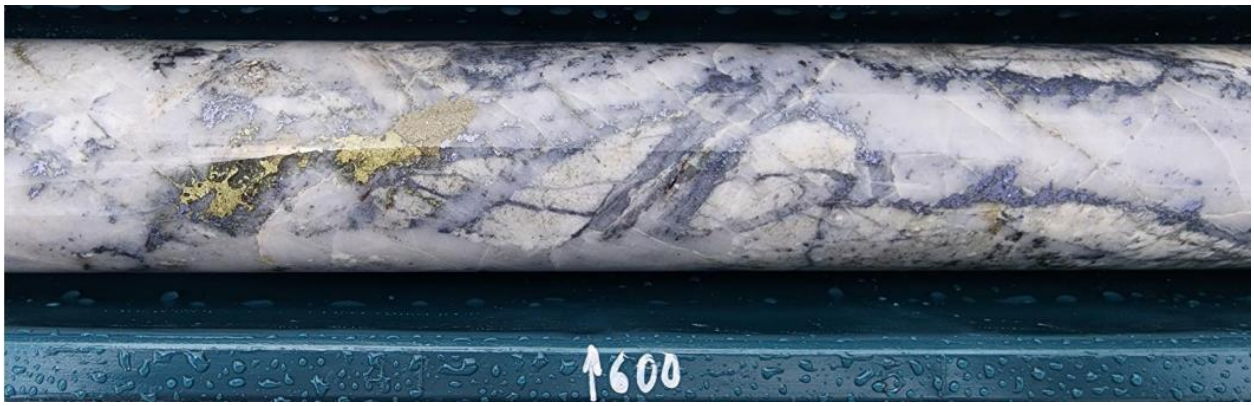
Source: GRE, 2026

7.3 Mineralization

7.3.1 Overview

On the Aurora Concession, two main types of mineralization have been identified; 1) Mo-Cu Porphyry developed in the intrusive and also in the Sandia Formation (Photo 7-8), and 2) Cu-Ag associated with veinlets that cut through the porphyry and are primarily developed in the Sandia Formation. These types of mineralization are spatially and temporally related to Miocene intrusive rocks. A geochronological dating (ALS Minerals, 2013) performed on molybdenite extracted from the Mo-Cu Porphyry veinlets at Aurora recorded an age of 6.4 Ma via the Re / Os method. This age of hydrothermal activity of the Mo-Cu Porphyry seems to correlate with the emplacement of granodioritic-quartz monzonitic stocks. In contrast, the absolute ages of the Cu-Ag mineralization stages are unknown.

Photo 7-8: Molybdenite & Quartz-Molybdenite + Pyrite + Chalcopyrite Veining



Note: Intermediate argillic alteration in core sample.
Source: DLP, 2024

Detailed logging of alteration and mineralization at the Aurora Project, focused on cross-cutting relationships, reveals that the Mo-Cu Porphyry deposit developed first and is overprinted by Cu-Ag veinlets, which extend to depths exceeding 1 km, where they still cut through the Mo-Cu Porphyry.

The alteration and mineralization characteristics of the Aurora Mo-Cu Porphyry allow for its development to be characterized in two stages. The first stage corresponds to a set of early porphyries, with straight quartz veinlets and Mo-Cu mineralization. The veinlets account for up to 20% volume abundance and mainly appear as stockwork. Mo is observed either along the edges, in the center of the veinlets, or disseminated within them. Cu is disseminated in the veinlets and possibly remobilized. The veinlets lack alteration halos. The second stage is characterized by a set of inter-mineral porphyries, with fewer straight veinlets, amounting to no more than 3% abundance. These porphyries cut through the earlier ones.

Cu-Ag veinlets cut through all veinlets and alteration assemblages associated with the porphyry. They exhibit pyrite-pyrrhotite-chalcopyrite-sphalerite mineralization, and their ore mineralogy is more complex than that of the porphyry.

The evolution of hydrothermal processes for both types of mineralization is clearly distinguishable. The Mo-Cu Porphyry developed at high temperatures, when the intrusive exhibited ductile-brittle characteristics and the hydrothermal solutions were in equilibrium with the igneous rock.

It is likely that a considerable portion of the Cu-Ag mineralization in the system has been eroded, as it is observed in smaller proportions in the northwestern part of the Aurora Concession. The significant development of these veinlets in the host rock, such as the Sandia Formation, suggests an important source of hydrothermal fluids, implying that the Mo-Cu Porphyry likely extends further along the NW-SE trend and at depth.

Surface evidence corroborates the dimensions of the porphyry-type system, as geochemical rock samples indicate molybdenum anomalies (> 100 parts per million [ppm]) in both the intrusive rocks and the Sandia Formation.

7.3.2 Dimensions

Current geological investigations conducted on the Aurora Concession, through drilling and surface sampling, confirmed the presence of both types of mineralization over a northing length of 1.5 km and an easting length of 1.5 km and from surface to a depth of 1.1 km. It is noted that mineralization is open to the north, southeast, and west, as well as below the maximum depth of drilling.

7.4 QP Comments on Geology and Mineralization

GRE's QP, Dr. Hamid Samari, reviewed the available geological information, including historical reports, geological logging records, drill core, analytical results, and observations made during the site visit conducted on June 2, 2026. Using the updated DLP geological database, the QP also updated the three-dimensional geological model to incorporate all drilling completed between 2022 and 2025. Based on this review and interpretation, the QP considers the geological model and geological interpretation presented in this report to be reasonable and appropriate for the current stage of exploration. The available drilling has adequately defined the principal lithological units, intrusive phases, hydrothermal alteration assemblages, structural features, and mineralization styles within the Aurora Concession and provides a sound geological framework for the 2026 Mineral Resource Estimate.

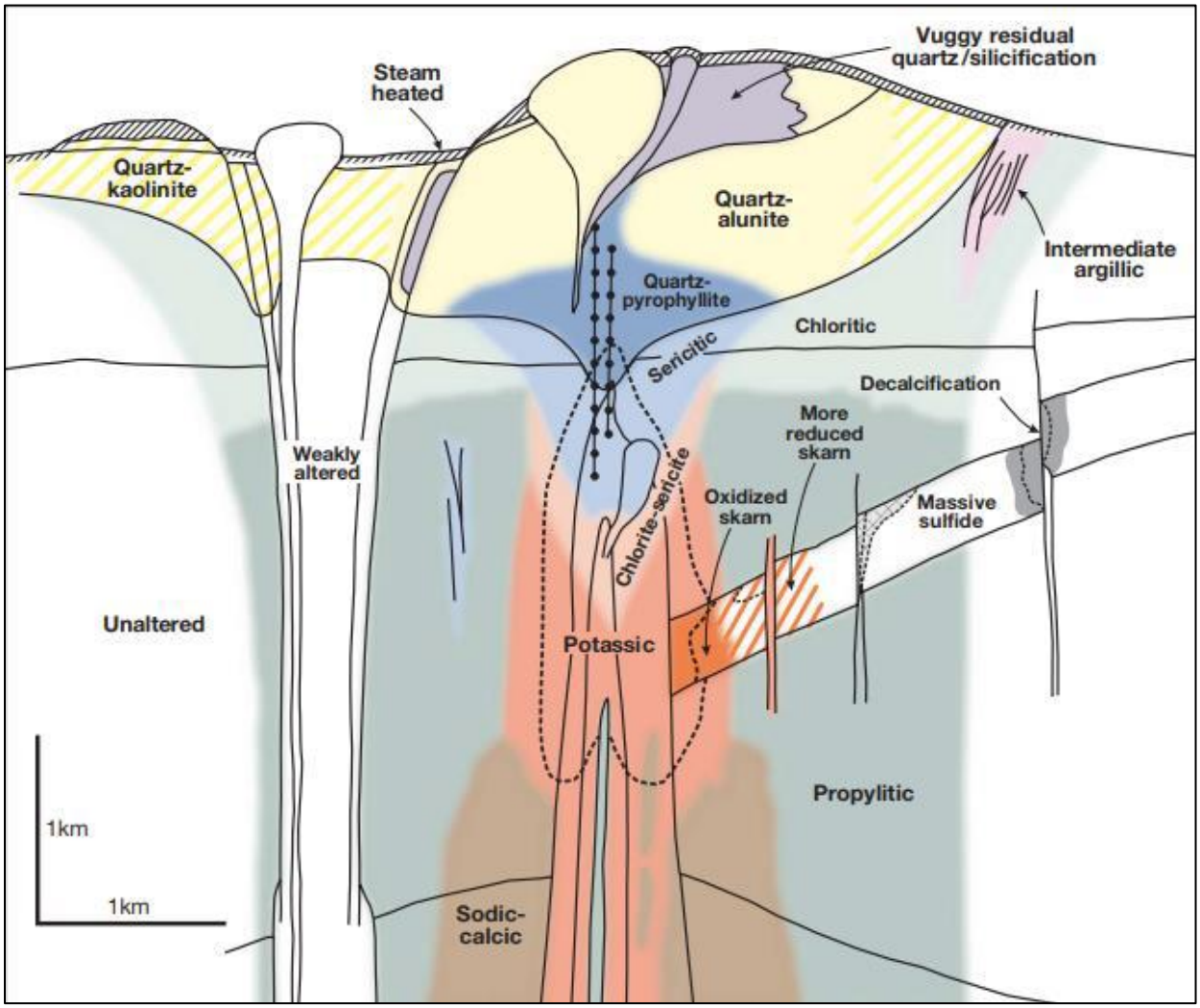
The QP notes that a detailed property-scale geological map integrating the distribution of lithological units, lithological contacts, hydrothermal alteration zones, major structural features, drill-hole collar locations, and the updated three-dimensional geological interpretation is currently unavailable. Preparation of such an integrated geological map is recommended to further improve the geological understanding of the deposit, support future exploration targeting and drill planning, and provide a stronger framework for future geological interpretation and future Mineral Resource updates.

8.0 DEPOSIT TYPE

The following description of the deposit type is based primarily on the 2025 NI 43-101 Technical Report prepared by AMC for DLP Resources Inc. (AMC, 2025). GRE's QP, Dr. Hamid Samari, reviewed the 2025 report together with the available geological information and observations made during the June 2, 2026, site visit, and found the deposit model to be compatible with the geology of the Aurora Project.

The Aurora deposit is a Cu-Mo Porphyry deposit. Porphyry Cu-Mo deposits are characterized by their association with multiple intrusions and dikes, typically composed of diorite to quartz monzonite with porphyritic textures. Breccia zones, containing angular or locally rounded fragments, are often present and commonly linked to the intrusion. These zones are usually hydrothermal in origin and may manifest as pebble dikes. Sulfide mineralization typically occurs between or within the breccia fragments (Figure 8-1).

Figure 8-1: Schematic Section, Porphyry Copper Deposit



Source: Sillitoe, 2010

Alteration zones are a key feature of Porphyry copper deposits. An outer zone of epidote–chlorite mineral alteration is typically present, while a quartz–sericite alteration zone occurs closer to the center and may

overprint earlier zones. At the core of the deposit, a potassic alteration zone characterized by secondary biotite and orthoclase is commonly associated with the highest concentrations of mineralization.

Fractures play an important role in mineralization, often filled or coated with sulfides or quartz veins containing sulfides. Closely spaced fractures with multiple orientations are typically linked to the highest-grade mineralization. In the upper portions of these deposits, supergene enrichment processes may occur. This involves the dissolution of metals near the surface, followed by their transport and precipitation below the water table.

8.1 QP Comments on Deposit Type

GRE's QP, Dr. Hamid Samari, reviewed the available geological, alteration, mineralization, geochemical, and drilling information, together with observations made during the site visit conducted on June 2, 2026. Based on this review, the QP considers the Aurora Project to be consistent with the characteristics of a Cu-Mo porphyry deposit. The observed intrusive phases, hydrothermal alteration assemblages, stockwork quartz-sulfide veining, brecciation, and associated Cu-Mo mineralization support the interpretation that the Aurora Project hosts a Cu-Mo porphyry system.

9.0 EXPLORATION

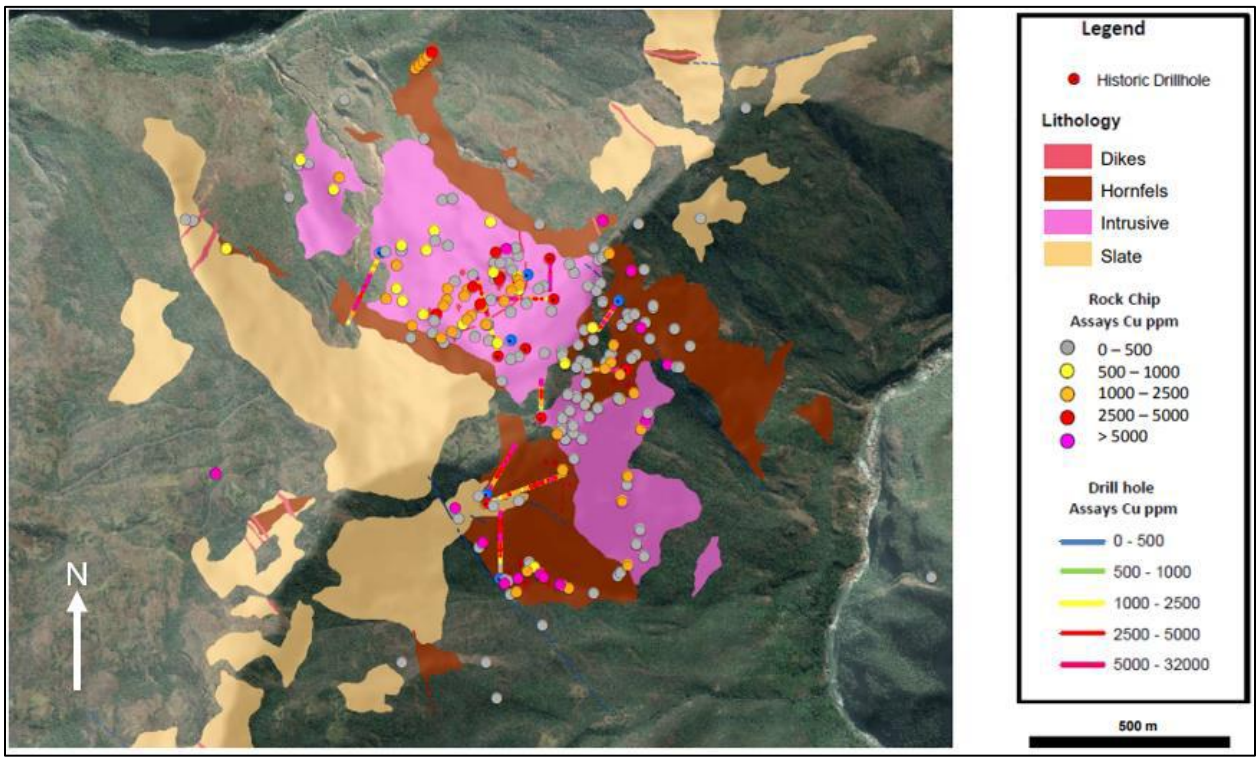
The description of exploration activities presented in this chapter is based primarily on the 2025 NI 43-101 Technical Report prepared by AMC for DLP Resources Inc. (AMC, 2025). The GRE QP, Dr. Hamid Samari, reviewed the 2025 Technical Report together with the available exploration data, including geological mapping, geochemical, geophysical, and drilling information. Based on this review, the exploration interpretations and results presented herein were found to be reasonable and appropriate for use in this Technical Report.

9.1 Pre-DLP Exploration

9.1.1 Surface Sampling

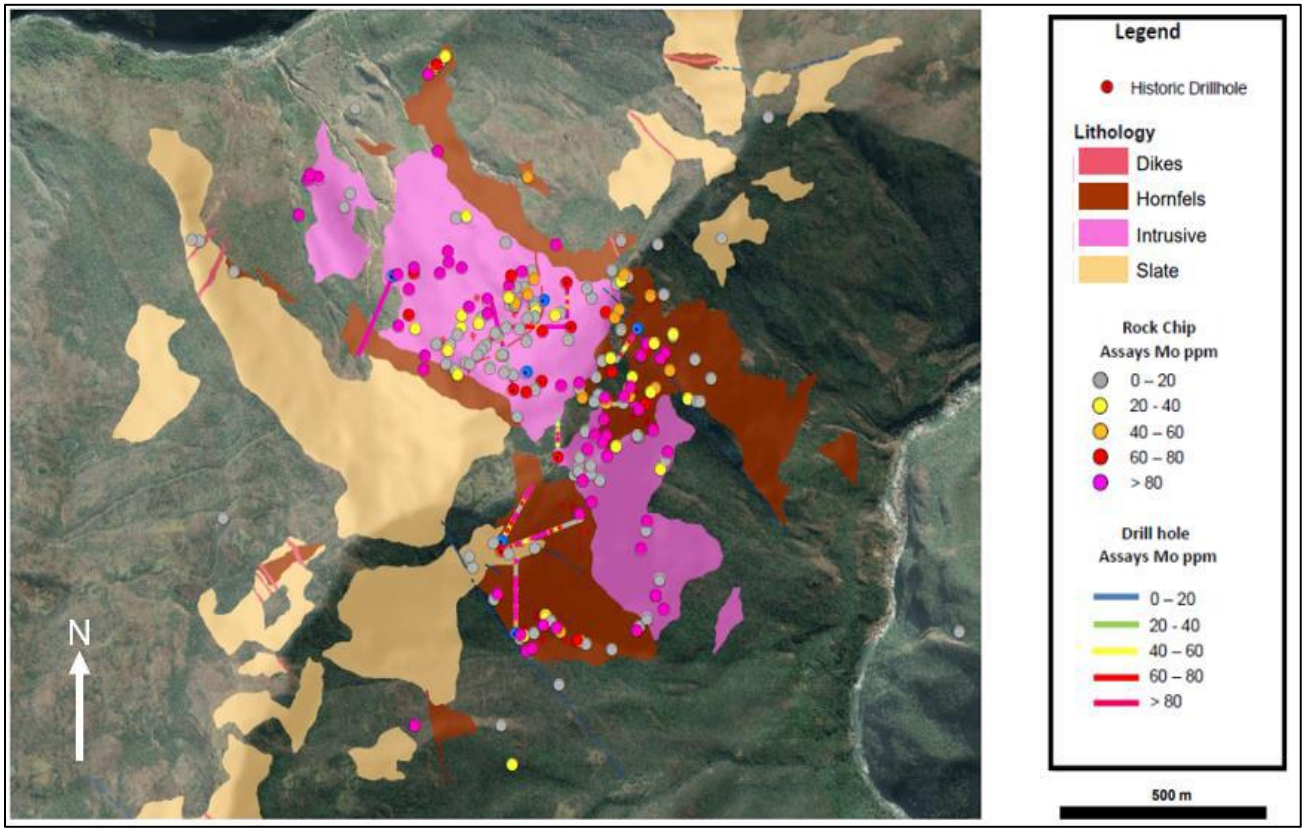
According to the 2013 FOCUS investor presentation, FOCUS completed a rock chip sampling program in 2012 comprising 120 rock chip samples. The surface sampling identified a copper (Figure 9-1) and molybdenum (Figure 9-2) anomaly. This anomaly was later confirmed by the drilling activities undertaken by DLP in subsequent years.

Figure 9-1: Focus Surface Geochemistry – Copper



Note: The field of view is consistent with the Aurora Concession boundary.
Source: AMC, 2025 (modified from FOCUS investor presentation, 2013)

Figure 9-2: Focus Surface Geochemistry – Molybdenum



Note: The field of view is consistent with the Aurora Concession boundary.
Source: AMC, 2025 (modified from FOCUS investor presentation, 2013)

The original analytical data supporting the historical FOCUS surface geochemical results presented in Figure 9-1 and Figure 9-2 were not available for independent verification by the QP at the time of reporting. Consequently, these historical geochemical results are presented for background information only and were not used in the geological interpretation or Mineral Resource Estimate.

9.1.2 Geophysical Surveys

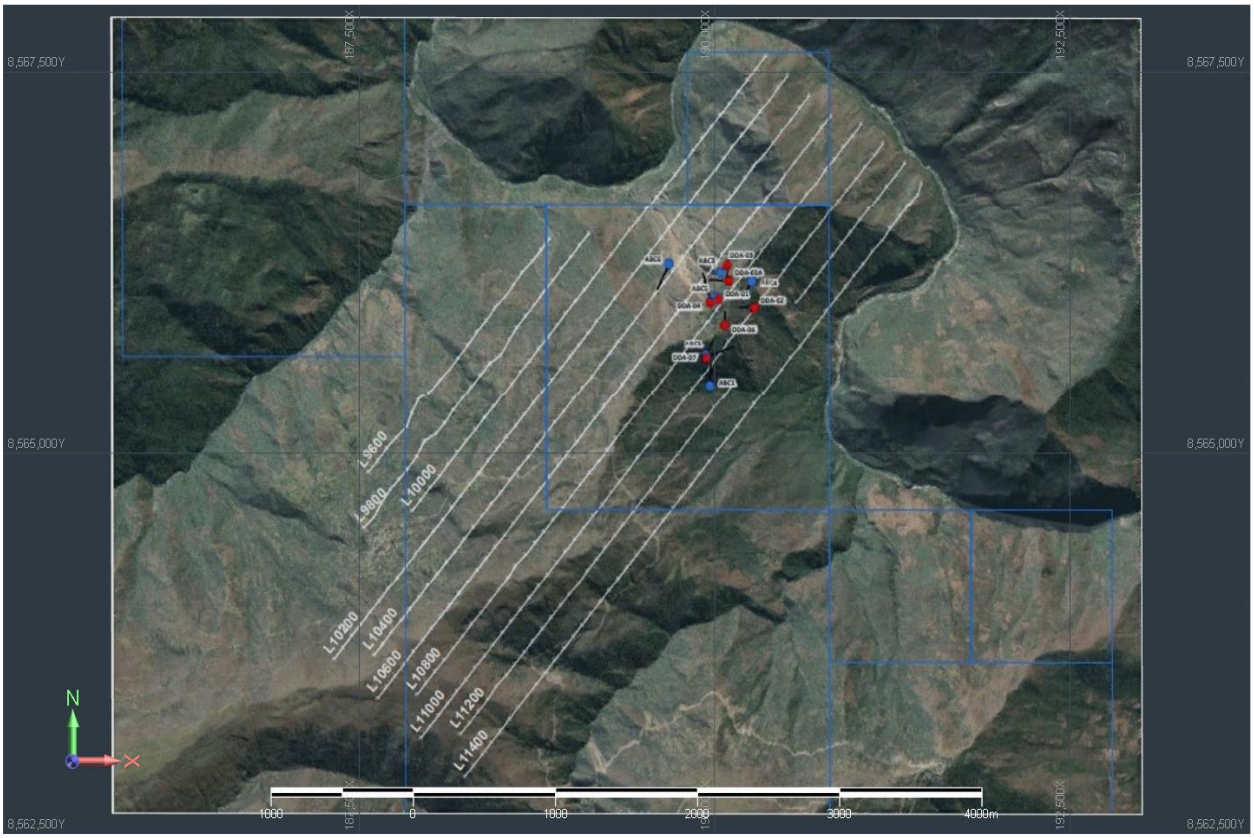
In 2012, FOCUS conducted a ground- IP survey. The survey was conducted by Fugro and totaled 43.8 line-km (The geophysical survey identified strong chargeability and resistivity anomalies up to 2 km x 1.4 km in area that remain open to the northwest and southeast, coincident with outcropping coarse-grained porphyritic intrusive rocks and overlying zones of mineralized hornfels. The anomaly is defined by strong chargeability readings from 30 millivolts per volt (mV/V) to over 70 mV/V and resistivity lows from 0 - 300 ohms per meter commencing between 100 m – 150 m below the surface and coincident with the start of the disseminated copper and molybdenum sulfide mineralization seen in the historical drillholes, for example, drillhole DDA-1 (Figure 9-4 to Figure 9-6). The stronger molybdenum values seen at deeper elevations in several historical diamond drillholes could indicate a deep molybdenum-rich core below the copper zone.

Figure 9-3). Two-thirds of the survey was completed using the 3D IP method with offset pole-dipole at 100-m spacings, and a more traditional two-dimensional (2D) IP method at smaller 50-m dipole spacings was undertaken in the northeast part of the grid over the main area of drilling due to difficulties with

signal penetration with the 3D method. As a result, depth penetration of more than 250 meters could not be achieved in the area of the main porphyry mineralization.

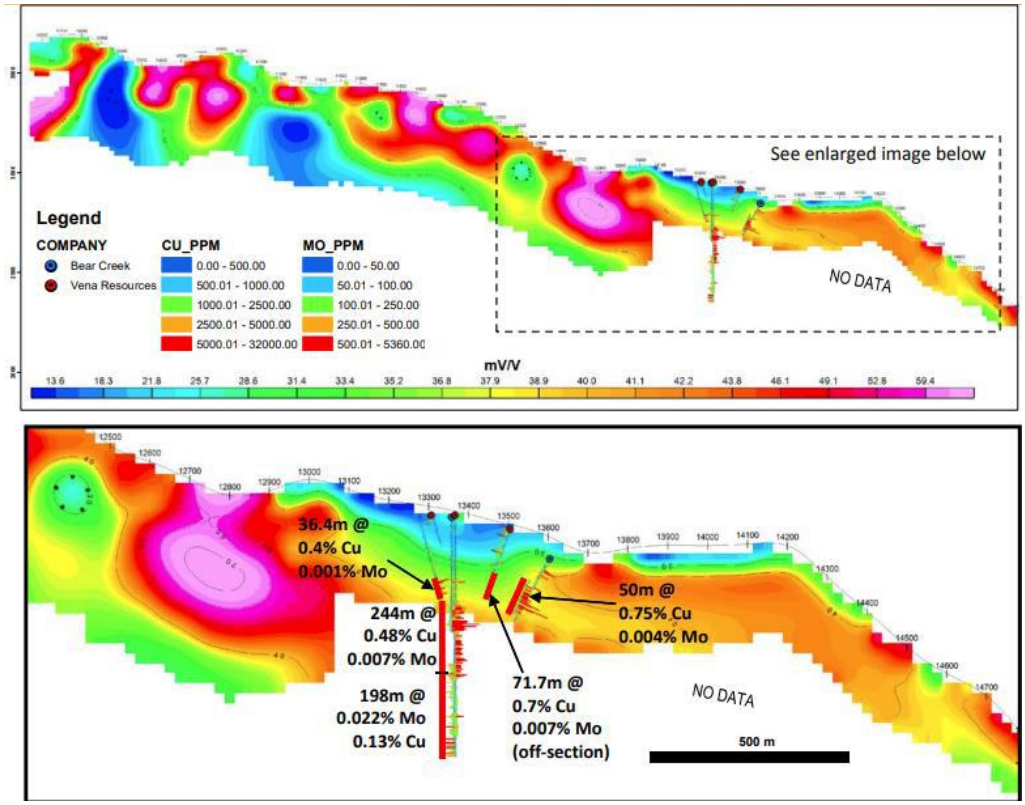
The geophysical survey identified strong chargeability and resistivity anomalies up to 2 km x 1.4 km in area that remain open to the northwest and southeast, coincident with outcropping coarse-grained porphyritic intrusive rocks and overlying zones of mineralized hornfels. The anomaly is defined by strong chargeability readings from 30 millivolts per volt (mV/V) to over 70 mV/V and resistivity lows from 0 - 300 ohms per meter commencing between 100 m – 150 m below the surface and coincident with the start of the disseminated copper and molybdenum sulfide mineralization seen in the historical drillholes, for example, drillhole DDA-1 (Figure 9-4 to Figure 9-6). The stronger molybdenum values seen at deeper elevations in several historical diamond drillholes could indicate a deep molybdenum-rich core below the copper zone.

Figure 9-3: Focus IP Survey Conducted by Fugro



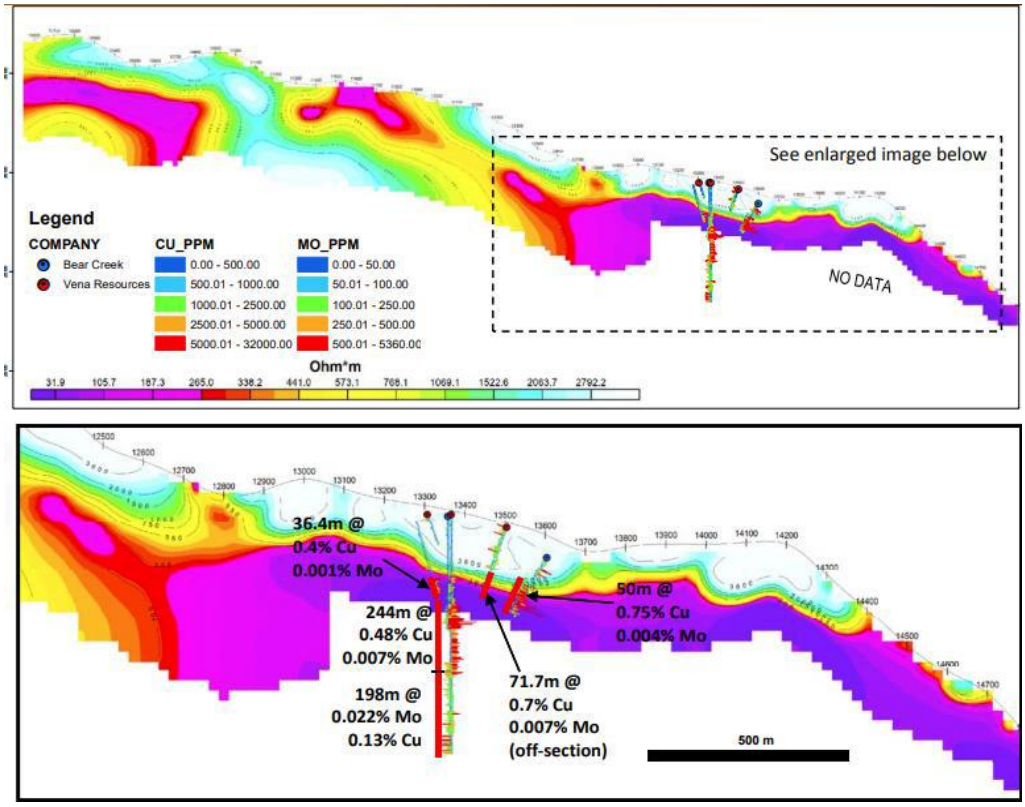
Notes: Blue lines represent concessions and blue drillhole collars = Bear Creek's drilling, red collars = Vena Resources drilling.
Source: FOCUS, 2012

Figure 9-4: Line 10800 Chargeability – Long Section



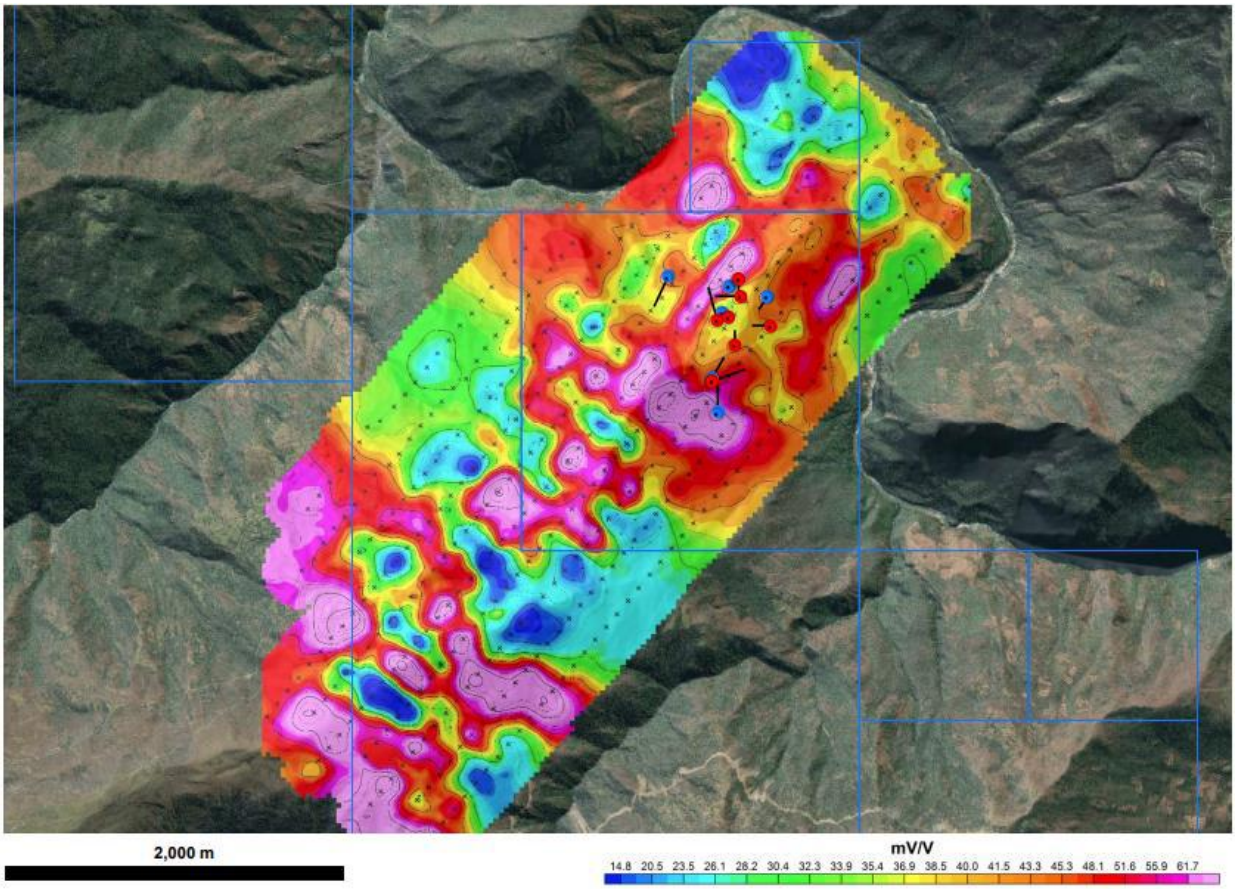
Source: Focus investor presentation, 2013

Figure 9-5: Line 10800 Resistivity – Long Section



Source: FOCUS investor presentation, 2013

Figure 9-6: Chargeability Level Plan -200 m



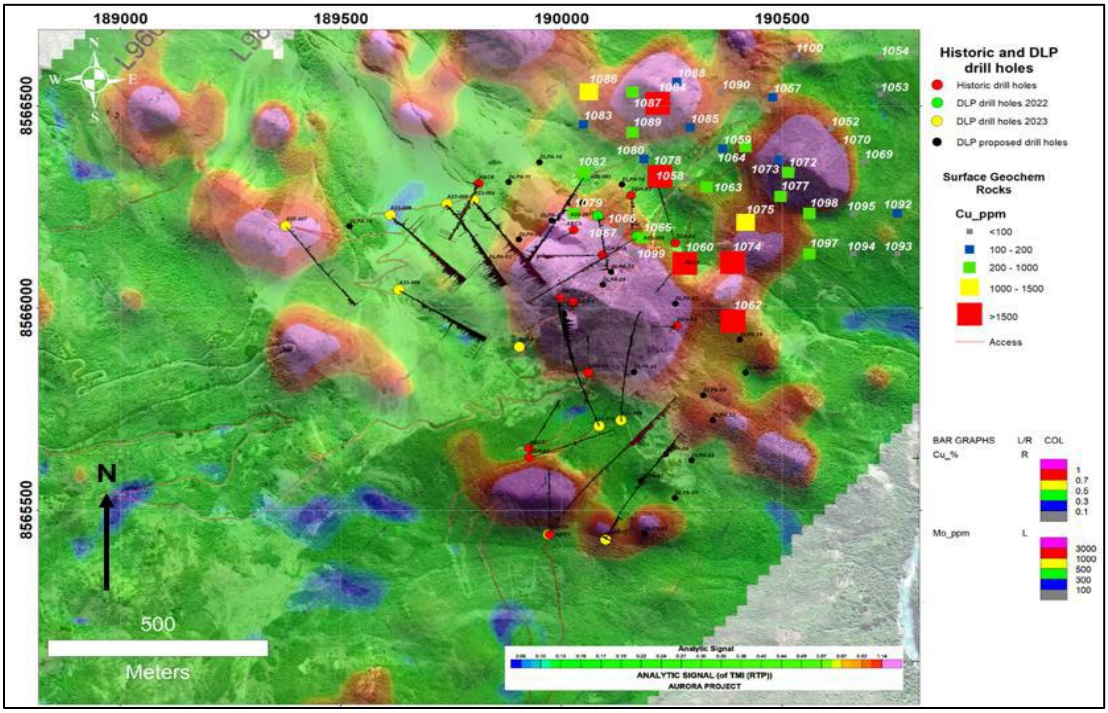
Source: Focus investor presentation, 2013

9.2 DLP Exploration

9.2.1 DLP Surface Sampling

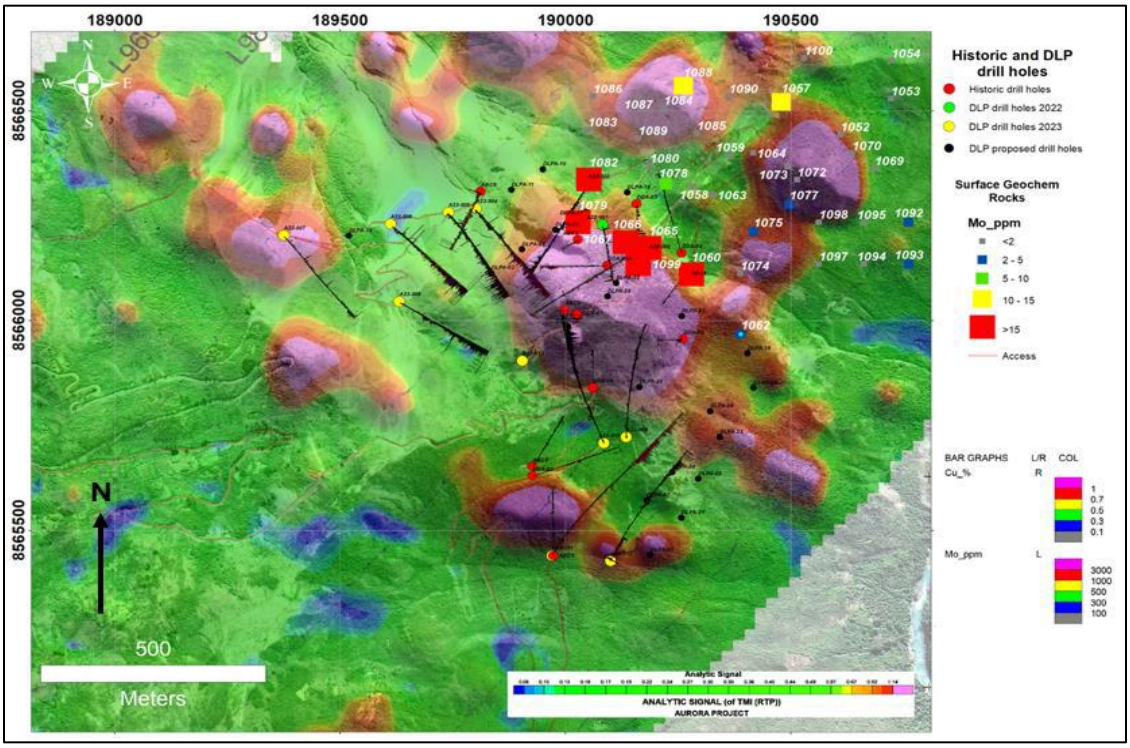
More recently, DLP Resources has collected 40 rock samples taken to the northeast of the DLP drilling. This sampling identified an area of anomalous Cu, Mo, and Ag extending over approximately 700 m x 500 m (Figure 9-7 to Figure 9-9). This area also overlies a magnetic analytic signal anomaly similar to that associated with the main Aurora porphyry system, further supporting the potential for an additional mineralized intrusive center in the northeastern part of the property.

Figure 9-7: DLP - Anomalous Copper in Rock Samples in NE Sector



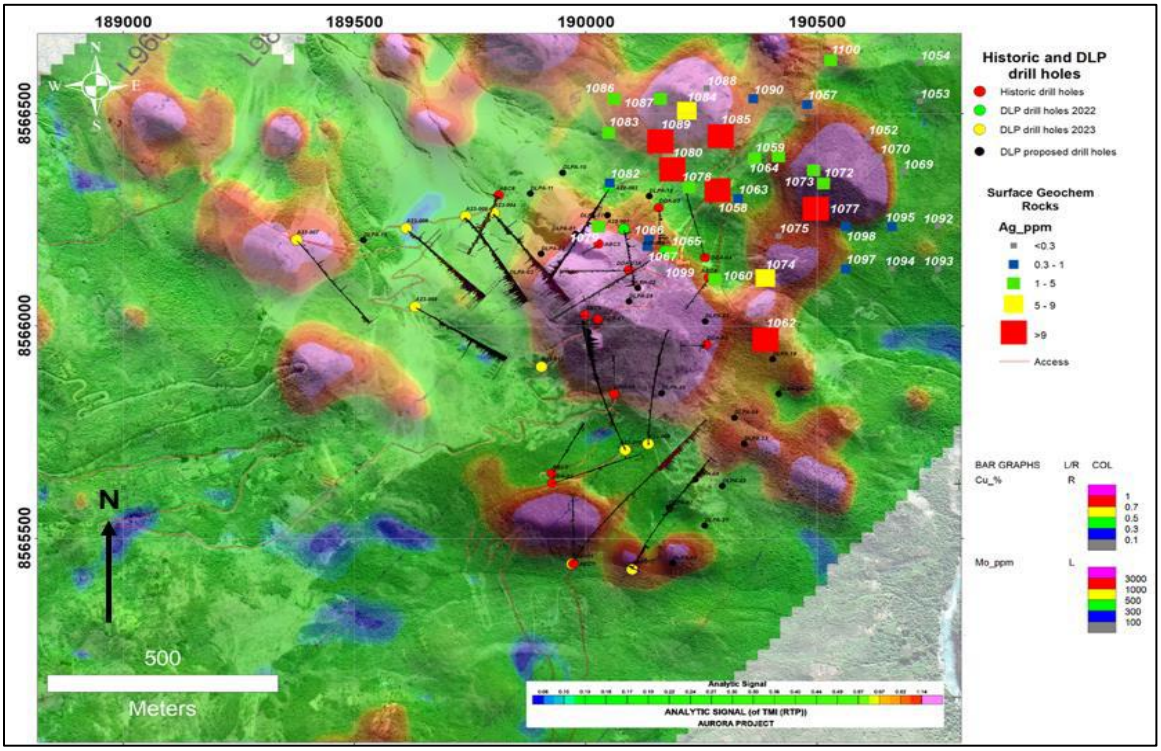
Notes: Magnetic analytic signal data with drillholes shown & anomalous copper in rock samples in NE sector.
Source: DLP, 2024

Figure 9-8: DLP - Anomalous Molybdenum in Rock Samples in NE Sector



Notes: Magnetic analytic signal data with drillholes shown & anomalous molybdenum in rock samples in NE sector.
Source: DLP, 2024

Figure 9-9: DLP - Anomalous Silver in Rock Samples in NE Sector



Notes: Magnetic analytic signal data with drillholes shown and anomalous silver in rock samples in NE sector.
Source: DLP, 2024

Sampling was conducted over a 10-m x 10-m surface area.

Rock chip sample 1062 returned 2.25% Cu and 195 grams per tonne (g/t) Ag. This sample was taken from a moderately silicified and brecciated hornfels with mineralization consisting of pyrite-chalcopyrite-covellite-chalcocite and copper oxides. Further rock chip samples with highly anomalous Cu, Ag, and / or Mo results are shown in Table 9-1.

Table 9-1: Significant Results from DLP Rock Chip Sampling

Sample Number	Cu (%)	Ag (g/t)	Mo (ppm)
1084	1.52	5.53	1.01
1060	0.27	2.09	74.3
1065	0.05	2.28	137
1067	0.01	0.79	53.1
1074	0.34	5.56	1.78
1075	0.13	0.13	4.86
1078	0.19	4.77	5.59
1086	0.12	4.61	1.23
1099	0.01	0.29	1.82

9.3 QP Comments on Exploration Programs

9.3.1 Pre-DLP Exploration

The historical exploration programs completed prior to DLP Resources successfully identified the principal geological and geochemical characteristics of the Aurora Cu-Mo porphyry system. Integrated surface rock geochemical sampling and IP surveys defined coherent copper and molybdenum geochemical anomalies together with significant chargeability and resistivity anomalies that spatially coincide with the porphyry intrusive complex and associated hornfels alteration. These exploration targets were subsequently confirmed by DLP Resources through diamond drilling, demonstrating that the historical exploration programs effectively identified the principal mineralized system within the Aurora Project and provided an excellent foundation for subsequent exploration.

9.3.2 DLP Exploration

The exploration programs completed by DLP Resources have significantly advanced the geological understanding of the Aurora Project through systematic geological investigations, surface geochemical sampling, and extensive diamond drilling. These programs confirmed the continuity of copper and molybdenum mineralization within the Aurora porphyry system and identified additional copper, molybdenum, and silver surface geochemical anomalies northeast of the currently drilled area. These results indicate that the mineralized footprint extends beyond the limits of the currently delineated deposit and that additional mineralized intrusive centers may exist elsewhere within the property. Collectively, the exploration programs completed by DLP Resources demonstrate that the Aurora Project remains highly prospective for the discovery of additional porphyry-style copper-molybdenum mineralization and the further expansion of the currently defined mineralized system.

9.3.3 Overall QP Opinion

The historical and DLP exploration programs have significantly improved the geological understanding of the Aurora Cu-Mo porphyry system and successfully identified and partially delineated the principal mineralized porphyry intrusive center. The integrated geological, geochemical, geophysical, and drilling datasets demonstrate that the exploration methods employed have been appropriate and effective for defining the known mineralized system and identifying additional areas with exploration potential.

Although the principal porphyry system has been successfully identified and partially delineated, significant portions of the property remain underexplored owing primarily to steep topography, dense vegetation, and limited surface access. Consequently, several areas have received only limited geological mapping, structural interpretation, hydrothermal alteration mapping, and surface geochemical coverage. Additional geological mapping, structural interpretation, hydrothermal alteration mapping, and systematic surface rock geochemical sampling in these areas would further improve the geological understanding of the Aurora Project. Integration of these datasets with the existing geophysical and drilling data would refine the geological model, improve the understanding of the geological controls on mineralization, support the identification of additional exploration targets, and assist in evaluating the lateral and vertical extent and continuity of the mineralized system.

10.0 DRILLING

10.1 Introduction

Drilling on the Aurora Property has been carried out by three operators over approximately 25 years. Initial exploration drilling was conducted by Bear Creek Mining Company (Bear Creek) in 2001 and subsequently by Vena Resources Inc. (Vena) in 2005. More recently, DLP has undertaken systematic diamond drilling programs between 2022 and 2025 aimed at evaluating and delineating the Cu-Mo porphyry mineralization at Aurora.

A total of 41 diamond drillholes totaling 26,533.18 m have been completed on the Aurora Property between 2001 and 2025 (Table 10-1). Of this total, DLP completed 28 diamond drillholes totaling 22,613.38 m between 2022 and 2025, representing approximately 85% of the total drilling completed on the Property and forming the basis of the current geological interpretation and Mineral Resource Estimate. During these drilling programs, DLP collected and submitted a total of 11,126 core samples for geochemical analysis, comprising 807 in 2022, 4,073 in 2023, 4,114 in 2024, and 2,132 in 2025.

Table 10-1: Summary of Drilling Completed on the Aurora Property

Operator	Year	Number of Drillholes	Total Drilled Length (m)
Bear Creek Mining Company	2001	6	1,560.10
Vena Resources Inc.	2005	7	2,359.70
DLP Resources Inc.	2022	3	1,651.90
DLP Resources Inc.	2023	10	8,258.35
DLP Resources Inc.	2024	10	8,394.53
DLP Resources Inc.	2025	5	4,308.60
Total	**2001–2025**	**41**	**26,533.18**

This section provides a summary of the historical drilling programs completed by Bear Creek and Vena and a detailed description of the drilling programs completed by DLP. To the knowledge of the QP, no drilling has been undertaken elsewhere on the Property outside the Aurora Concession.

The following discussion is restricted to the Aurora Concession, not the surrounding tenements. To the knowledge of the QP, no drilling has been undertaken on the rest of the Property.

This section describes diamond drill programs completed by DLP at the Aurora Concession. It is noted that, due to the inability to verify the geological logging and sample analysis, the drilling completed by Bear Creek and Vena Resources has been excluded for use in the generation of the geological model or Mineral Resource Estimate. The description of these two programs is brief, as the discussion is immaterial to the Mineral Resource.

10.2 Drilling Program

10.2.1 Pre-DLP (Before 2022)

10.2.1.1 Bear Creek Mining Company (2001)

In 2001, Bear Creek completed six diamond drillholes (ABC-1 to ABC-6) totaling 1,560.10 m on the Aurora Property. The drilling was predominantly shallow and represented the first known drill testing of the Aurora porphyry system. The reported drill results indicated the presence of copper and molybdenum mineralization associated with porphyry-style alteration and provided the initial evidence for the potential lateral and vertical extent of the mineralized system.

The drillhole collar locations are presented in Table 10-2, and reported significant drill intercepts are summarized in Table 10-3. Reported intercepts include 181.0 m grading 0.30% Cu and 0.01% Mo in drillhole ABC-1, 224.4 m grading 0.27% Cu and 0.007% Mo in drillhole ABC-2, and 302.4 m grading 0.23% Cu and 0.082% Mo in drillhole ABC-6. Drillholes ABC-4 and ABC-5 also reported locally higher copper grades over shorter intervals, including 56.1 m grading 0.72% Cu and 0.008% Mo in drillhole ABC-5.

The Bear Creek drilling program represents the earliest known drilling completed on the Aurora Property and established the initial exploration framework for subsequent drilling programs. However, the QP has not independently verified the historical geological logging, sampling procedures, analytical results, quality assurance/quality control (QA/QC) procedures, or original drill core from the Bear Creek program. Accordingly, the reported historical drill results are presented solely for historical context and to document the exploration history of the Property and have not been used in the geological interpretation, geological modeling, or Mineral Resource Estimate presented in this Technical Report.

Table 10-2: Bear Creek Drillhole Locations

Hole ID	Easting	Northing	RL	Total depth (m)	Platform
ABC1	189973	8565440	2803.5	290	ABC-1
ABC2	189927	8565652	3022.5	407	ABC-2
ABC3	190028	8566193	2896.3	84	ABC-3
ABC4	190266	8566114	2881.7	186	ABC-4
ABC5	189999	8566026	2995.2	289.1	ABC-5
ABC6	189812	8566308	2620.5	304	ABC-6

Table 10-3: Significant Drillhole Intercepts from the Bear Creek Drilling

Hole	From (m)	To (m)	Interval (m)	Cu (%)	Mo (%)
ABC 1	109	290	181	0.3	0.01
ABC 2	181.6	406	224.4	0.27	0.007
ABC 3	0	83.5	83.5	0.12	0.006
ABC 4	30	54	24	0.45	0.01
ABC 5	173	289.1	116.1	0.45	0.005
including	233	289.1	56.1	0.72	0.008
ABC 6	1.5	303.9	302.4	0.23	0.082
including	90	174	84	0.42	0.106

10.2.1.2 Vena Resources Inc. (2005)

In 2005, Vena Resources Inc. ("Vena") completed seven diamond drillholes (DDA-01 to DDA-07) totaling 2,359.70 m on the Aurora Property. The drillholes ranged in depth from approximately 160 m to 604 m, with an average drillhole depth of approximately 337 m. The reported drilling represented a significant expansion of the historical drilling program and further indicated the presence of copper and molybdenum mineralization associated with the Aurora porphyry system. The drilling also suggested the potential for mineralization to extend both laterally and at depth.

The drillhole collar locations are presented in Table 10-4, and reported significant drill intercepts are summarized in Table 10-5. The reported drill results include several broad intervals of copper and molybdenum mineralization over substantial drill lengths and contributed to the understanding of the geometry and potential scale of the Aurora porphyry system.

Table 10-4: Vena Resources Drillhole Locations

Hole ID	Easting	Northing	Elevation (m)	Total depth (m)	Platform
DDA-01	190026	8566015	2997.8	604	DDA-01
DDA-02	190262	8565956	2844.7	219	DDA-02
DDA-03	190158	8566278	2883.8	160	DDA-03
DDA-03A	190093	8566132	2961	298	DDA-03A
DDA-04	190258	8566161	2887.8	401	DDA-04
DDA-06	190061	8565840	3047.4	268.4	DDA-06
DDA-07	189927	8565630	3004.8	409.3	DDA-07

Table 10-5: Significant Drillhole Intercepts from Vena Resources

Hole	From (m)	To (m)	Interval (m)	Cu (%)	Mo (%)
DDA-01	406	244	0.48	0.007	0.52
including	*266*	*20*	*1.01*	*0.006*	*1.04*
	406	*604*	*0.13*	*0.022*	*0.25*
DDA-02	62	42	0.5	0.028	0.64
DDA-03	159.8	141.8	0.5	0.004	0.52
including	*88*	*50*	*0.75*	*0.004*	*0.77*
DDA-3A	297.7	215.7	0.36	0.005	0.3
including	*226*	*297.7*	*0.7*	*0.007*	*0.73*
DDA-04	396	192	0.19	0.012	0.25
DDA-06	268.4	36.4	0.4	0.001	0.41
DDA-07	402	192	0.21	0.006	0.23

The QP has not independently verified the historical geological logging, sampling procedures, analytical results, QA/QC procedures, or original drill core from the Vena drilling program. Accordingly, the reported historical drill results are presented solely for historical context and to document the exploration history of the Property and have not been used in the geological interpretation, geological modeling, or Mineral Resource Estimate presented in this Technical Report.

10.2.2 DLP (2022–2025)

Since acquiring the Aurora Property, DLP has completed systematic diamond-drilling programs in 2022, 2023, 2024, and 2025 to evaluate and delineate the copper-molybdenum porphyry mineralization at Aurora. DLP has completed 28 diamond drillholes for a total drilled length of 22,613.38 m.

The drilling programs were designed as early-stage resource delineation drilling aimed at evaluating the geometry, continuity, and extent of the Aurora porphyry system. Drilling has generally been completed on an approximate nominal drillhole spacing of 100 m × 100 m and has utilized PQ, HQ, NQ, and limited BTW core sizes depending on drilling depth and ground conditions.

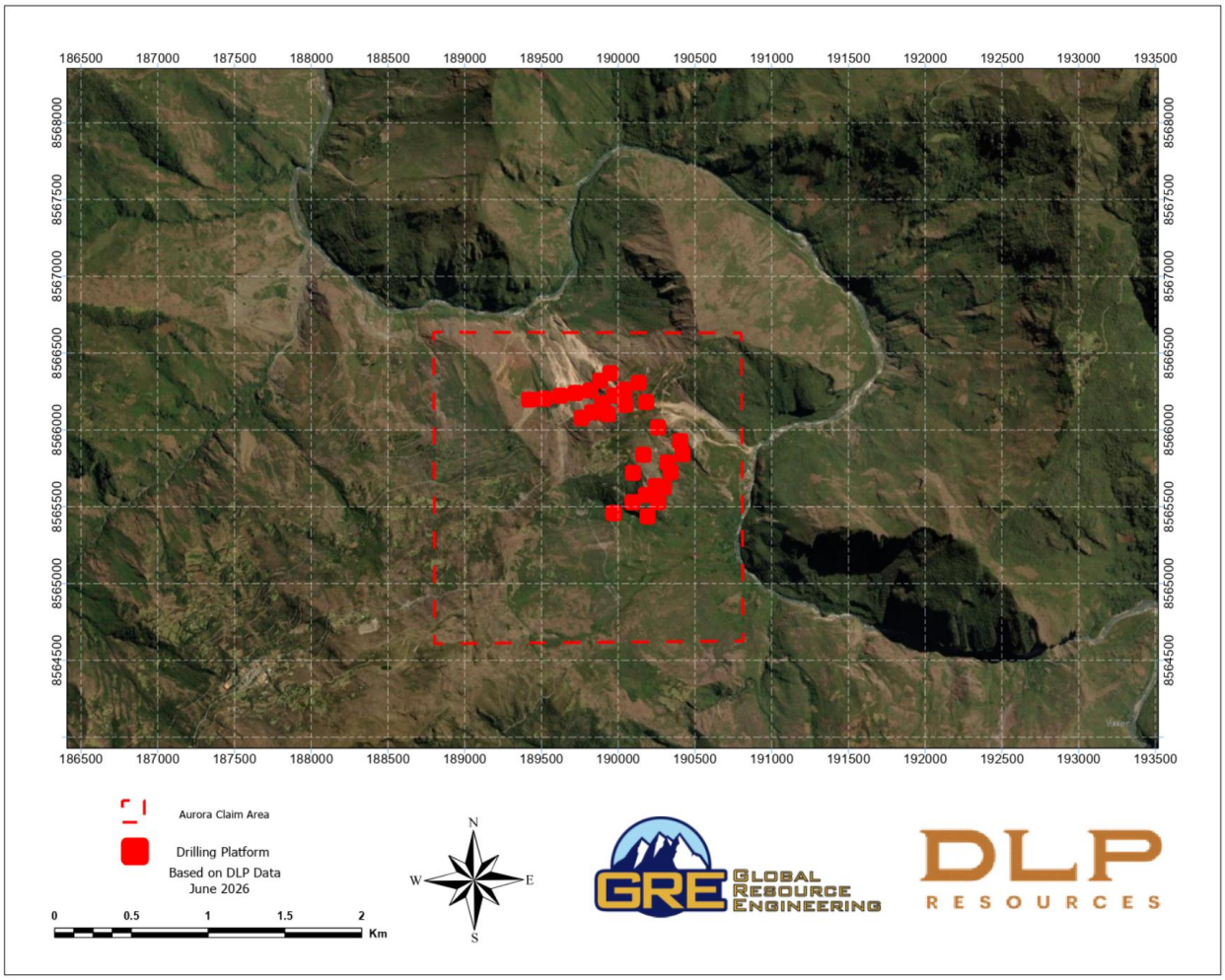
Drillhole depths range from 66.88 m to 1,192.15 m and average approximately 807.62 m. Due to the steeply dipping nature of the intrusive units and the steep topography across the Property, several drillholes were completed parallel to sub-parallel to the interpreted orientation of the intrusive bodies and mineralized domains. Consequently, the true thickness of the mineralized intervals and geological units remains incompletely constrained in certain areas of the deposit.

The locations of DLP's approved drill platforms are presented in Table 10-6 and illustrated in Figure 10-1. Thirty drill platforms have been approved under DLP's environmental authority and were established to provide access to the interpreted mineralized zones while complying with the requirements of the Peruvian environmental permitting process. Due to the steep and mountainous topography of the Aurora Property, drill platforms and access roads are generally limited in size and were constructed to provide safe and practical access for drilling operations.

Table 10-6: Approved Drill Platforms under DLP's Environmental Authority

Platform	East	North	Platform	East	North
DLPA-01	189978	8566217	DLPA-16	189423	8566193
DLPA-02	189903	8566170	DLPA-17	190047	8566261
DLPA-03	189835	8566114	DLPA-18	190137	8566306
DLPA-04	189772	8566074	DLPA-19	190404	8565922
DLPA-05	189968	8565459	DLPA-20	190188	8565442
DLPA-06	190093	8565528	DLPA-21	190257	8565530
DLPA-07	190180	8565572	DLPA-22	190295	8565623
DLPA-08	190237	8565639	DLPA-23	190342	8565723
DLPA-09	190321	8565784	DLPA-24	190417	8565840
DLPA-10	189950	8566361	DLPA-25	190164	8565842
DLPA-11	189880	8566312	DLPA-26	190097	8565720
DLPA-12	189819	8566257	DLPA-27	190258	8566011
DLPA-13	189725	8566238	DLPA-28	190185	8566178
DLPA-14	189625	8566222	DLPA-29	190049	8566159
DLPA-15	189519	8566202	DLPA-30	189931	8566098

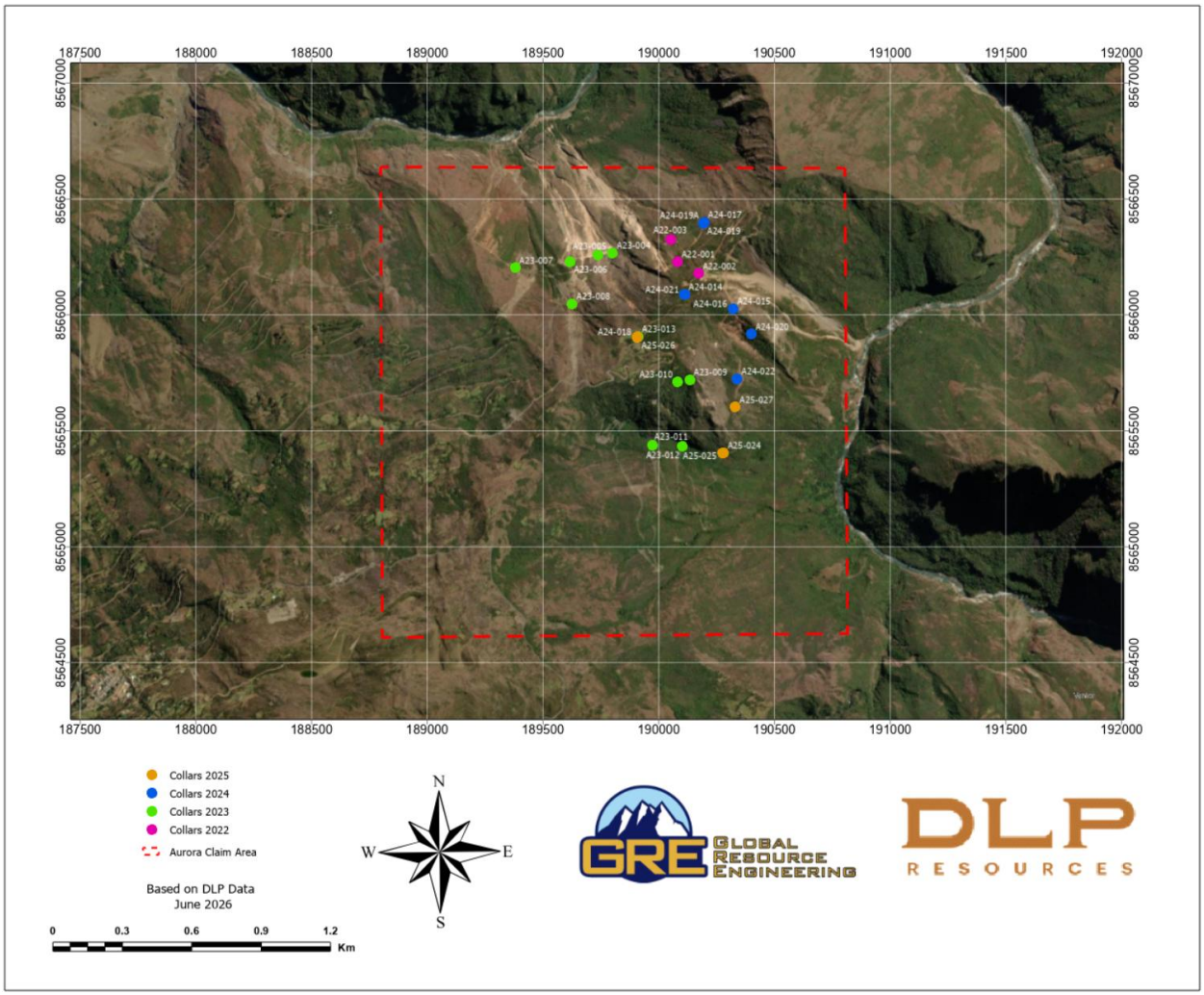
Figure 10-1: Location of Approved Drill Platforms



Source: GRE, 2026

The locations of DLP drillholes completed between 2022 and 2025 are presented in Figure 10-2, and the drillhole specifications are summarized in Table 10-7. The drillholes are concentrated within the central portion of the Aurora Concession and were designed to systematically evaluate the geometry, continuity, and extent of the copper-molybdenum porphyry mineralization.

Figure 10-2: DLP Drillhole Locations by Year (2022-2025)



Source: GRE, 2026

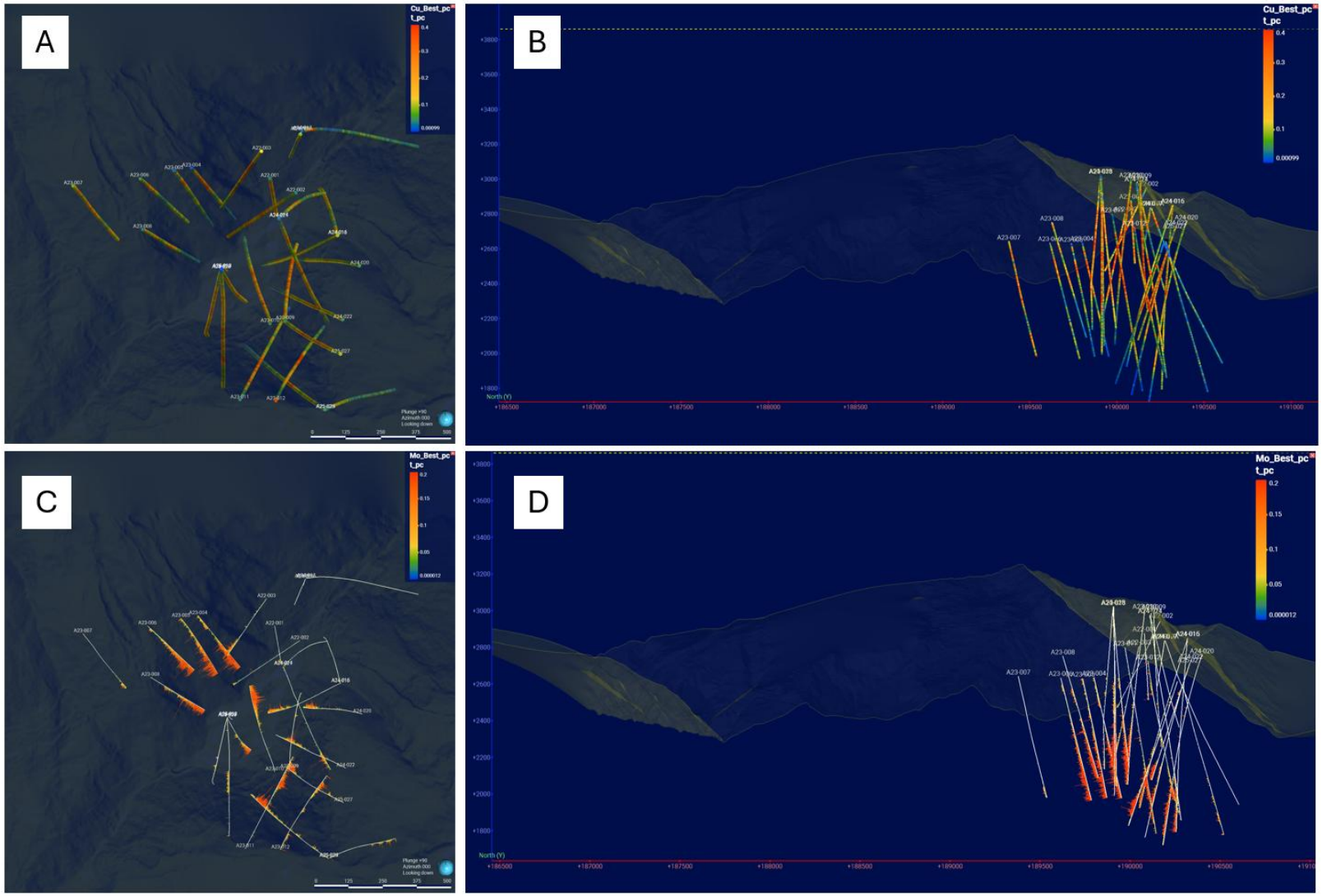
Table 10-7: DLP Drillhole Specifications

Hole ID	Easting	Northing	Elevation (m)	Total depth (m)	Platform	Az	Dip
A22-001	190080	8566229	2878.176	388	DLPA-33	177	-70.0
A22-002	190171.5	8566177	2952.572	561.6	DLPA-28	235	-60.0
A22-003	190050	8566326	2806.849	702.3	DLPA-17	210	-70.0
A23-004	189800	8566266	2636.461	700.9	DLPA-12	140	-70.0
A23-005	189737.4	8566256	2630.432	693.4	DLPA-13	140	-70.0
A23-006	189615.1	8566230	2635.281	708.55	DLPA-14	140	-70.0
A23-007	189379.4	8566202	2643.344	708.25	DLPA-16	140	-70.0
A23-008	189626.5	8566047	2753.411	703.65	DLPA-21	120	-70.0
A23-009	190133.2	8565720	2999.542	790.55	DLPA-26	0	-70.0
A23-010	190079.7	8565710	3002.284	1002.55	DLPA-26A	340	-70.0
A23-011	189972.3	8565437	2800.245	1081.7	DLPA-05	28	-70.0
A23-012	190100.2	8565433	2725.371	887.6	DLPA-06	20	-70.0
A23-013	189905.9	8565902	3022.283	981.2	DLPA-31	0	-90.0
A24-014	190111.8	8566088	2976.684	641.3	DLPA-32	50	-70.0

Hole ID	Easting	Northing	Elevation (m)	Total depth (m)	Platform	Az	Dip
A24-015	190320.8	8566026	2851.556	972.5	DLPA-27	245	-70.0
A24-016	190319.6	8566024	2851.668	1080.15	DLPA-27	345	-70.0
A24-017	190195.5	8566399	2837.207	987.25	DLPA-18	85	-70.0
A24-018	189906.2	8565902	3022.348	1118	DLPA-31	185	-70.0
A24-019	190191.2	8566395	2837.283	66.88	DLPA-18	200	-70.0
A24-019A	190191.8	8566396	2837.287	267.45	DLPA-18	200	-70.0
A24-020	190399.2	8565916	2759.798	1061.35	DLPA-19	285	-70.0
A24-021	190111	8566086	2976.542	1192.15	DLPA-32	170	-70.0
A24-022	190338.5	8565724	2726.512	1007.5	DLPA-23	300	-70.0
A25-023	189907.6	8565904	3022.35	928.65	DLPA-31	178	-74.0
A25-024	190278.8	8565404	2652.184	909.65	DLPA-20	74	-74.0
A25-025	190275.7	8565402	2652.269	901.6	DLPA-20	308	-66.0
A25-026	189906.8	8565902	3022.36	889.5	DLPA-31	142	-80.5
A25-027	190330.9	8565601	2709.612	679.2	DLPA-22	300	-69.0

Plan, oblique, and sectional views showing the distribution of copper and molybdenum grades from the DLP drilling programs are presented in Figure 10-3 and Figure 10-4, respectively. The figures illustrate the spatial distribution of copper and molybdenum assay grades within the Aurora Claim Area. Copper grades occur over a broad vertical interval extending from near surface to depth, whereas higher molybdenum grades are generally observed in the deeper portions of the drilled volume.

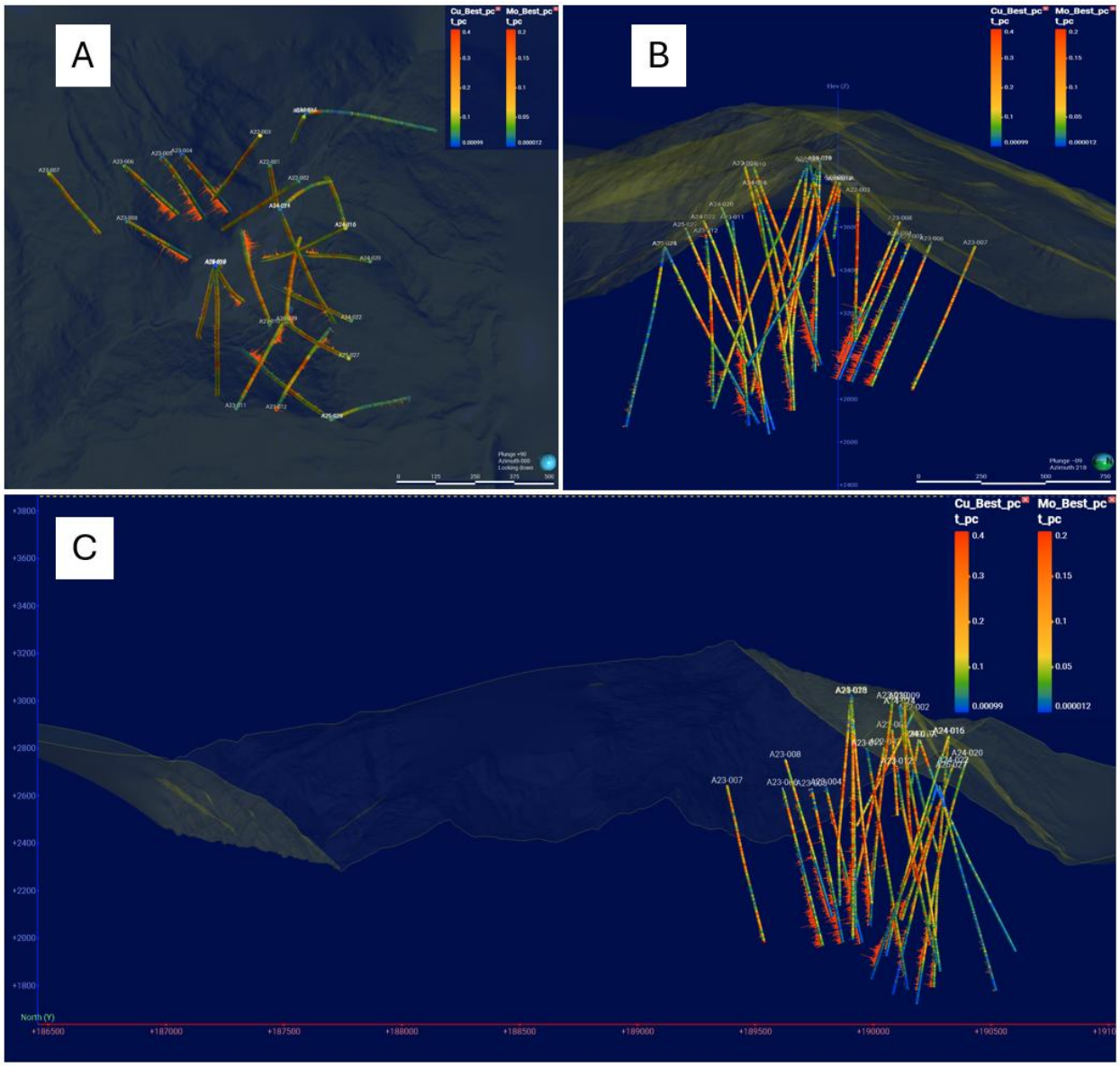
Figure 10-3: Distribution of Copper and Molybdenum Grades within the Aurora Claim Area



Source: GRE, 2026

A) Plan View for Cu Grades, B) North-Looking Section View of Cu, C) Plan View of Mo Grades, and D) North-Looking Section View of Mo Grades.

Figure 10-4: Visual Correlation between Copper Grades (solid bars) and Molybdenum Grades within the Aurora Claim Area



Source: GRE, 2026

A) Plan View, B) Oblique View, and C) North-Looking Section View

10.3 Drilling Method (2022-2025)

Diamond drill programs completed at the Aurora Property were designed and managed by DLP personnel. Drilling was conducted by the contract drilling company, Kluane Drilling. Drillholes were planned from approved drill platforms established under DLP's environmental authority, with access roads developed as required to provide safe access for personnel and equipment. Due to the steep and mountainous terrain, drill pads and access roads are generally limited in size and were constructed only to accommodate the drill rig, support equipment, and safe personnel access (Photo 10-1). The drill platforms are commonly developed on narrow ridge crests and steep hillsides and are connected by single-lane access roads that generally follow the natural topography. Consequently, the footprint of individual drill

pads is relatively small, minimizing ground disturbance and reducing the need for extensive earthworks. Once access is achieved, the drillhole collar is marked with a wooden stake.

Photo 10-1: Access Roads and Drill Pads at Aurora

Source: GRE, 2026

Between 2022 and 2025, DLP completed 28 diamond drillholes totaling 22,613.38 m. Drilling was conducted from elevations ranging from approximately 2,630 m to 3,022 m above sea level. Drillhole depths range from 66.88 m to 1,192.15 m and average approximately 807.62 m. Drillholes were designed to evaluate the geometry, continuity, and extent of the Aurora porphyry system from multiple orientations. Drillhole azimuths range from 0° to 345°, although most holes were drilled toward the southeast and southwest quadrants. Drillhole inclinations generally range from -60° to -80.5°, with most holes drilled at approximately -70° and one hole completed vertically (-90°). Drilling utilized PQ, HQ, NQ, and limited BTW core sizes, depending on drill depth and ground conditions (Table 10-8).

Table 10-8: Summary of DLP Diamond Drilling by Year and Core Size (2022–2025)

Year	Number of Drillholes	Meters drilled	PQ	HQ	NQ	BTW
2022	3	1,651.90	805.75	846.15	N/A	N/A
2023	10	8,258.35	3,297.45	4,853.00	107.9	N/A
2024	10	8,394.53	137.68	3,377.35	4,823.65	55.85
2025	5	4,308.60	N/A	2,120.90	2,187.70	N/A
Total	28	22,613.38	4,240.88	11,197.4	7,119.25	55.85

DLP geologists use handheld GPS units to locate drillhole collars. The drill contractor subsequently positions the drill rig, and DLP geologists confirm the drill alignment, including inclination and azimuth, prior to drilling. Drilling operations are conducted continuously in two separate shifts, operating 24 hours per day and seven days per week.

Upon completion of each drillhole, steel casing is installed by the drilling contractor to preserve the collar location and maintain drillhole integrity. DLP personnel subsequently construct a concrete monument and mark the collar with the drillhole identification number, depth, dip, azimuth, and completion date, as shown in Photo 10-2.

Photo 10-2: Completed Drillhole Collar

Source: GRE, 2026

10.4 Logging Procedure (2022-2025)

DLP personnel visit active drill sites at least once daily to monitor drilling progress and collect completed core boxes. Diamond drill core is transported by the drilling contractor from the drill sites to DLP's core logging facility located in the camp, near Parobamba Village.

Upon arrival at the core facility, the core boxes are checked, recorded on a core handover form, and signed by the receiving DLP personnel. The core boxes are then transferred to the logging area where detailed geological logging, geotechnical measurements, photography, processing, and sampling are completed. Logging data are initially recorded manually on standardized paper templates and subsequently entered into an electronic database maintained by DLP.

The DLP core logging and processing procedure comprises the following steps:

- The core is cleaned and broken core pieces are reassembled to their original orientation where possible.
- The recovered core length for each drill run is measured and core recovery is calculated.
- Depth blocks are checked and drillhole depths are marked on the core.
- Rock Quality Designation (RQD) measurements are completed and recorded.

- A detailed geological log is completed using standardized DLP logging templates. The geological log includes a graphic log and records lithology, oxidation, alteration, mineralization, structures, veins, and other relevant geological observations.
- The geologist defines sample intervals and marks sample boundaries and cutting lines based on lithological contacts, mineralization, alteration, structures, and other geological characteristics.
- Density samples are collected at nominal intervals of approximately one sample every 25 m and submitted to SGS for analysis (see Section 11.0).
- Prior to cutting, wet photographs of the entire drill core are collected using a high-resolution digital camera.
- The core is cut longitudinally using a diamond saw and sampled in accordance with DLP's sampling procedures (see Section 11.0).
- The remaining half core and associated records are transferred to DLP's secure core storage facility in Calca (see Section 11.0).
- Sample shipments are prepared and dispatched to the analytical laboratory on a regular basis. Sampling procedures, shipment protocols, chain of custody, and security procedures are described in Section 11.0.

10.5 Core Recovery (2022-2025)

Core recovery for the DLP drilling programs is considered acceptable for the purposes of geological interpretation and Mineral Resource estimation. Approximately 90% of the core recovery measurements exceeded 80% recovery, and approximately 80% of the measurements exceeded 90% recovery. Core recovery was generally high throughout the drilling programs and is not considered to have introduced a material bias into the geological interpretation or analytical results.

10.6 Collar Survey (2022-2025)

Following completion of drilling activities, DLP geologists record preliminary drillhole collar coordinates using handheld GPS units (Photo 10-3). These preliminary coordinates are used for drillhole management and planning purposes during active drilling.

Upon completion of a drill program or a series of drillholes, final collar surveys are completed by Urano Environmental Consultants, an independent surveying company with appropriately qualified personnel and experience in topographic and mining surveys. Final collar locations are surveyed using differential GPS (DGPS) methods, and the results are documented in formal survey reports accompanied by the applicable calibration and quality-control documentation.

Photo 10-3: Collar Pick Up of Drillhole A22-003

Source: DLP, 2023

10.7 Down-Hole Surveys

All downhole surveys completed at the Aurora Project were undertaken by contractors independent of DLP. Two downhole survey methods have been employed during the DLP drilling programs: Devishot surveys and gyroscopic surveys (see Table 10-9).

Devishot surveys were completed by the drilling contractor, Kluane Peru S.A.C. Survey measurements were generally collected at approximately 50-m intervals downhole as part of the drilling process. Survey results were provided to DLP in formal survey reports as part of the drilling services.

Gyroscopic surveys were completed by Peru Survey, an independent surveying contractor, using a GIST-M gyroscopic survey tool. The GIST-M is a high-precision north-seeking gyroscopic instrument capable of acquiring multi-shot and continuous survey measurements in real time. Gyroscopic surveys were generally collected at approximately 10-m intervals downhole.

The downhole survey data were used to define the three-dimensional geometry of the drillholes and are considered by the QP to be appropriate and adequate for geological interpretation and Mineral Resource Estimation.

Table 10-9: Summary of Downhole Survey Methods

Survey Method	Contractor	Tool	Typical Survey Interval
Devishot	Kluane Peru S.A.C.	Devishot	50 m
Gyroscopic	Peru Survey	GIST-M Gyroscope	10 m

10.8 QP Comments on Section 10 Drilling

In the QP's opinion, drilling was conducted in accordance with industry-standard exploration practices and provides adequate coverage of the Aurora copper-molybdenum porphyry system. The drilling programs completed by DLP between 2022 and 2025 were designed to evaluate the geometry, continuity, and extent of the mineralized system and have successfully demonstrated the presence of laterally and vertically extensive porphyry-style copper and molybdenum mineralization.

Collar and downhole survey methods generally provided reliable drillhole locations, and the drilling methods resulted in acceptable core recovery and sample quality. Logging procedures were systematic and provided consistent geological, alteration, structural, and mineralization descriptions suitable for geological interpretation and Mineral Resource estimation.

The collected drilling data adequately reflect the dimensions, continuity, and style of the Aurora porphyry system. DLP drillholes were completed from multiple orientations, including northwest- and southeast-directed holes, to evaluate the geometry and continuity of the mineralized system. Although the steep mountainous topography of the Property imposes practical constraints on drill platform locations and achievable drill orientations, the current drilling provides sufficient information to support the geological interpretation and Mineral Resource Estimate presented in this Technical Report.

The Aurora porphyry system remains open both laterally and at depth. Additional exploration drilling from new platforms and alternative drill orientations may further refine the understanding of the geometry, continuity, and true thickness of the geological and mineralized domains and assist in evaluating the full extent of the mineralized system.

No material factors were identified in the drilling, sampling, recovery, or data collection procedures that could significantly affect the geological interpretation or Mineral Resource Estimation presented in this Technical Report.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Introduction

The following discussion is restricted to the Aurora Concession and pertains exclusively to the diamond drilling programs completed by DLP Resources Inc. between 2022 and 2025. To the knowledge of the QP, no sampling activities have been undertaken elsewhere on the Property outside the Aurora Concession.

This section describes the sampling methods, sample preparation procedures, analytical methods, quality assurance and quality control procedures, database management practices, density determination methods, and sample security protocols employed by DLP during the 2022 to 2025 drilling campaigns. All sampling and analytical activities were completed in accordance with DLP's established exploration procedures and protocols.

11.2 Sampling

All drilling completed by DLP at the Aurora Project was undertaken by the contract drilling company, Kluane Peru S.A.C., using diamond core drilling methods. Drilling was completed using a combination of PQ (85 mm), HQ (63.5 mm), NQ (47.6 mm), and BTW (42.13 mm) core sizes, selected according to drilling depth and ground conditions. HQ core was the predominant core size and accounted the 49.53% of the drilled intervals, with NQ, PQ, and BTW accounting for approximately 31.48%, 18.75% and 0.24%, respectively. The drilling programs produced generally good-quality diamond drill core suitable for geological logging, geotechnical measurements, and geochemical sampling.

Diamond core recovery was measured and recorded during the core logging process. Core recoveries were considered satisfactory, with more than 75% of the measurements exceeding 90% recovery and more than 85% exceeding 80% recovery by length. In the opinion of the QP, the core recovery achieved during the DLP drilling programs is adequate and is considered representative of the mineralized intervals intersected.

Diamond drill core sampling was completed at DLP’s core logging facility located at the Parobamba field camp as part of the standard core processing workflow. Following geological logging, geologists identified sample intervals based on lithological contacts, alteration, mineralization, structures, and other geological characteristics. Sample intervals were generally established at a nominal length of approximately 2 m, although intervals varied locally to honor geological boundaries. The entire length of each drillhole was sampled and submitted for geochemical analysis.

Sample information, including drillhole identification and sample intervals, was recorded in pre-numbered sample books containing unique sample identification numbers and tear-off tags. Following geological logging and photography, the core was cut longitudinally using a diamond core saw. One-half of the core was consistently retained for reference, and the remaining half was collected and placed into pre-labeled sample bags bearing unique sample identification numbers. Sample intervals, sample numbers, and sample bags were cross-checked by DLP personnel before sealing.

The outer tear-off sample tag was retained in the core box at the beginning of the sample interval, and the inner tag was placed within the sample bag. Individual sample bags were sealed using staples and

tape, after which QA/QC samples were inserted into the sample stream. Sample bags were subsequently placed into large poly-weave bags for transport to the analytical laboratory. Individual sample batches generally comprised between 80 and 130 samples.

Between 2022 and 2025, DLP collected and submitted a total of 11,126 diamond drill core samples for geochemical analysis. Sample lengths ranged from approximately 0.85 m to 10 m, with an average sample length of approximately 2.02 m. More than 95% of the samples were approximately 2 m in length.

11.3 Sample Preparation

All diamond drill core samples collected by DLP during the 2022 to 2025 drilling campaigns were submitted to the ALS sample preparation facility in Arequipa, Peru. ALS Arequipa, Peru, is part of ALS Global, an independent commercial analytical laboratory specializing in analytical geochemistry services and is independent of DLP.

Sample preparation was completed in accordance with ALS preparation code PREP-31. Under this procedure, samples are crushed to better than 70% passing 2 mm, riffle split to obtain an approximately 250-g sub-sample, and subsequently pulverized to better than 85% passing 75 microns (µm). The prepared pulp samples were then dispatched to ALS Lima, Peru for geochemical analysis.

11.4 Analytical Procedure

Analytical work was completed by ALS Lima, Peru, an independent commercial analytical laboratory and part of ALS Global. ALS Lima is accredited in accordance with the International Organization for Standardization and International Electrotechnical Commission standard ISO/IEC 17025:2017, General Requirements for the Competence of Testing and Calibration Laboratories.

The analytical program used by DLP consisted primarily of ALS analytical suite ME-MS61, comprising a 48-element package determined by four-acid digestion followed by Inductively Coupled Plasma-Mass Spectrometry (ICP-MS) analysis. This analytical suite was applied to all routine drill core samples.

Additional analytical methods were employed on selected samples depending on grade and project requirements. Ore-grade copper analyses were completed using ALS method Cu-OG62 on samples exceeding the upper detection limit of the routine analytical suite. Gold analyses were completed using fire assay with either atomic absorption finish (Au-AA24) or Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES) finish (Au-ICP21) on selected samples. Sequential copper analyses (Cu-PKG06LI) were undertaken during the early stages of exploration to evaluate copper speciation and distribution within the deposit; however, this work was subsequently discontinued after investigations indicated limited oxide copper mineralization within the system. Ultra-trace multi-element analyses were also completed on selected samples using ALS analytical method ME-MS41.

A summary of the analytical methods employed by DLP is presented in Table 11-1.

Table 11-1: Summary of ALS Analytical Methods Used During the DLP Drilling Campaigns (2022–2025)

Drill Campaign	ALS Analysis Code	Elements	Description	Protocol Notes	Number of Samples
2022-2025	ME-MS61	Cu, Mo, Ag + 45 others	48-element four-acid ICP-MS	All Samples	10,401
	CU-PKG06LI	Cu	Cu Sequential Analysis	2,133	CU-PKG06LI
	Au-AA24	Au	Au 50 g FA AA finish	625	Au-AA24
	Cu-OG62	Cu	Ore Grade Cu - Four Acid	88	Cu-OG62
	Au-ICP21	Au	Au 30 g FA ICP-AES Finish	85	Au-ICP21
	ME-MS41	Cu, Mo, Ag + 45 others	Ultra Trace Aqua Regia ICP-MS	51	ME-MS41

11.5 Quality Assurance and Quality Control

The QA/QC program was designed to monitor analytical accuracy, precision, and potential contamination throughout the sampling, sample preparation, and analytical process. The QA/QC program implemented by DLP during the 2022 to 2025 drilling campaigns consisted of the systematic insertion of blank samples, certified reference materials (CRMs), and duplicate samples into the sample stream. A summary of the QA/QC samples inserted during the DLP drilling campaigns is presented in Table 11-2.

Table 11-2: Summary of QA/QC Samples Submitted During the DLP Drilling Campaigns (2022–2025)

Year	Core Samples	Blank	CRMs	Coarse Reject Duplicate	Pulp Reject Duplicate	% Of Total Assays
2022	807	15	14	14	N/A	5.33
2023	4073	79	69	74	N/A	5.45
2024	4114	349	337	346	N/A	20.08
2025	2132	64	119	60	N/A	11.40

11.5.1 Blank Samples

Blank samples were inserted into the sample stream by DLP personnel to monitor potential contamination during sample preparation and analytical procedures. A total of 507 blank samples were submitted during the 2022 to 2025 drilling campaigns. Blank sample insertion rates ranged from approximately 0.37 to 1.70 blanks per 20 routine samples. The 2024 drilling program exceeded the commonly adopted industry guideline of approximately one blank sample per 20 routine samples, whereas the blank insertion frequencies during the 2022, 2023, and 2025 drilling programs were below this threshold.

DLP utilized a commercially purchased crushed quartz aggregate as the blank reference material for the QA/QC program. The blank material was independently certified by Intertek Australia and has certified values of 7 ppm Cu and 1 ppm Mo. The blank material was inserted into the sample stream to monitor potential contamination during sample preparation and analytical procedures. The certification analyses of 20 blank samples demonstrated very low background concentrations of copper and molybdenum and confirmed that the material is suitable for use as a blank reference material in the Aurora QA/QC program.

11.5.2 Certified Reference Materials

CRMs obtained from independent commercial suppliers were inserted into the sample stream by DLP personnel to monitor analytical accuracy and analytical bias. A total of 539 CRM samples were submitted during the 2022 to 2025 drilling campaigns. The QA/QC program used seven commercially prepared CRMs and one project-specific reference material that collectively covered a range of copper, molybdenum, and silver grades representative of the mineralization encountered at the Aurora Project.

The CRMs used during the DLP drilling campaigns included CDN-CM-33, CDN-ME-1409, CDN-ME-1704, PORF-34, PORF-35, PORF-39, PORF-43, and DLP-01. Certified copper values ranged from 0.092% to 0.692%, certified molybdenum values ranged from 0.0042% to 0.138%, and certified silver values ranged from 0.915 g/t to 11.6 g/t. A summary of the certified reference materials employed during the DLP drilling campaigns is presented in Table 11-3.

Table 11-3: Certified Reference Materials Used During the DLP Drilling Campaigns (2022–2025)

	Cu (%)		Mo (%)		Ag (g/t)	
CRM	Expected value	SD	Expected value	SD	Expected value	SD
CDN-CM-33	0.346	0.014	0.025	0.002	2.3	0.3
CDN-ME-1409	0.242	0.01	-	-	11.6	1.6
CDN-ME-1704	0.692	0.028	-	-	11.6	1.3
PORF-34	0.414	0.0127	0.014	0.00057	3.87	0.144
PORF-35	0.616	0.016	0.013	0.0005	7.26	0.23
PORF-39	0.231	0.005	0.017	0.0009	1.218	0.1
PORF-43	0.226	0.006	0.0042	0.0002	-	-
DLP-01	0.092	0.002	0.138	0.0032	0.915	0.132

11.5.3 Duplicate Samples

Duplicate coarse reject samples were collected to evaluate the precision and repeatability of the sampling, sample preparation, and analytical procedures. The duplicate program comprised coarse reject duplicate samples during the 2022 to 2025 drilling campaign. A total of 494 coarse reject duplicate samples were submitted during the DLP drilling programs.

Coarse-reject duplicates were generated from crushed reject material and used to assess the repeatability of sample preparation and analytical procedures. The duplicate samples were inserted in accordance with DLP's QA/QC procedures and formed an integral component of the Aurora Project QA/QC program.

11.6 Database Management

DLP maintains a centralized digital database for the Aurora Project containing collar, downhole survey, geological logging, sampling, assay, density, and QA/QC information generated during the 2022 to 2025 drilling campaigns. Geological logging and sample information are recorded using standardized logging and sampling procedures to promote consistency in data capture and minimize transcription errors.

Geological, geotechnical, and sampling information are reviewed by project geologists prior to incorporation into the project database. Drillhole collar locations, downhole surveys, and geological intervals are periodically reviewed for completeness and consistency.

Analytical data are received directly from ALS laboratories in digital format and incorporated into the project database while retaining the original laboratory identifiers and assay records. Assay results are reviewed in conjunction with the QA/QC program, including the performance of blanks, CRMs, and duplicate samples, prior to their use in geological interpretation and Mineral Resource Estimation. Where QA/QC results indicate potential analytical issues or where blank or CRM results fall outside established acceptance criteria, the affected analytical batches are reviewed and appropriate follow-up actions, including review of the affected analytical batches and, where considered necessary, re-assay of selected samples or analytical batches, are undertaken.

QA/QC information is continuously reviewed as new analytical data are received. Periodic reviews of the database, including verification of collar coordinates, downhole surveys, geological coding, sampling intervals, assay imports, density measurements, and QA/QC records, are undertaken to identify and correct any inconsistencies or data try errors.

11.7 Density

DLP collected density samples as part of the diamond drill core logging and sampling procedures. Density samples were generally collected at nominal intervals of approximately one sample every 25 m of diamond drill core. Individual density samples, approximately 10 cm in length, were labeled, wrapped in plastic film and fiber tape, and placed in core boxes for shipment to the laboratory.

Density samples were submitted to the SGS laboratory in Arequipa, Peru, for apparent density determination. Apparent density measurements were determined using the paraffin-coated immersion method (laboratory code GQ_GEP). The density samples were transported and delivered to the laboratory by DLP personnel. SGS is an independent commercial laboratory and is not affiliated with DLP.

As of the effective date of this Technical Report, the Aurora Project density database comprises 729 apparent density measurements representing all geological domains utilized in the geological model and Mineral Resource Estimate. The measured apparent densities range from 2.29 grams per cubic centimeter (g/cm^3) to 3.11 g/cm^3, which fall within the expected range for the principal lithologies encountered at the Aurora Project.

11.8 Sample Security

DLP managed all aspects of sample collection and transportation from the Aurora Project to the ALS sample preparation facility in Arequipa, Peru. All samples for analysis were collected at the Aurora Project and delivered directly to the ALS facility by DLP personnel. Sample shipments were transported by site geologists via road transport to the ALS sample preparation facility in Arequipa, located approximately 6.5 hours from the Project.

Following sampling, the remaining diamond drill core was transported from the Aurora Project to the permanent core storage facility in Calca, Peru, located approximately 1.5 hours from the Project site. During active drilling operations, core from up to two active drillholes could be temporarily stored at DLP's

field facility at the Parobamba camp for geological logging and sampling. The Calca core storage facility consists of a secure, fenced, and locked building and is equipped to accommodate additional geological logging and core examination activities if required. The on-site logging facility at DLP's Parobamba field camp is similarly secured and remains locked when personnel are not present.

Samples were dispatched to ALS several times per week during active drilling campaigns and were delivered in person to the ALS sample preparation facility in Arequipa. Following receipt at ALS, the samples were prepared for analysis, sealed in containers with tamper-evident labels, and transported to the ALS analytical laboratory in Lima, Peru, in accordance with ALS's internal chain-of-custody and sample handling procedures.

11.9 QP Comments on Section 11 Sample Preparation, Analysis, and Security

In the opinion of the GRE's QP, Dr. Hamid Samari, the sampling methods, sample preparation procedures, analytical methods, density determination procedures, database management practices, and sample security procedures implemented by DLP during the 2022 to 2025 drilling campaigns were generally conducted in accordance with accepted industry practices for an exploration-stage copper-molybdenum-silver porphyry project. The analytical database and QA/QC results provide a reasonable basis for geological interpretation and Mineral Resource Estimation. Nevertheless, the QP has identified several opportunities to further strengthen the QA/QC program and analytical dataset for future drilling campaigns, including the following:

- Implement a minimum QA/QC insertion protocol of at least one blank sample, one CRM, and one duplicate sample for every 20 routine samples submitted for analysis.
- Continue using the DLP-prepared blank reference material for routine QA/QC monitoring. To provide an additional independent control on potential contamination during sample preparation and analytical procedures, the QP recommends the periodic insertion of a commercially prepared and independently certified blank reference material in future drilling campaigns.
- Continue using CRMs that cover the range of copper, molybdenum, and silver grades encountered at the Aurora Project and periodically review CRM performance to ensure that the selected reference materials remain appropriate for the mineralization styles and grade ranges being evaluated.
- Review the performance of CRM PORF-34 for silver analyses, as the analytical results exhibit greater variability and poorer agreement with the certified silver value than observed for the other analytical parameters. DLP should consider re-evaluating the suitability of PORF-34 as a silver reference material, including obtaining additional certification data, undertaking independent check analyses, or replacing the CRM with an alternative reference material that provides more robust control of higher-grade silver analyses.
- Continue periodic reviews of the QA/QC program and database management procedures to ensure that analytical data remain suitable for geological interpretation and future Mineral Resource Estimation work.

12.0 DATA VERIFICATION

12.1 Introduction

This section summarizes the data verification procedures completed by GRE's QP, Dr. Hamid Samari, to assess the accuracy, completeness, and reliability of the exploration data utilized in this Technical Report. The review included historical exploration information and DLP exploration data comprising drillhole collar locations, downhole survey information, geological logging records, analytical data, QA/QC information, and supporting technical documentation.

The objective of the verification program was to identify any material errors, inconsistencies, or deficiencies that could adversely affect the geological interpretations, conclusions, and recommendations presented in this Technical Report.

12.2 Historical Data Verification

Historical exploration work on the Aurora Property included six diamond drillholes completed by Bear Creek in 2001 and seven diamond drillholes completed by Vena in 2005. Information available to GRE's QP included historical drillhole summary tables, selected analytical results, and descriptions presented in the 2024 Technical Report. However, original drillhole databases, detailed geological logs, analytical certificates, and complete supporting documentation were not available for review.

Consequently, GRE's QP was unable to independently verify the historical drilling and analytical data to the same level as the more recent DLP exploration data. The 2024 Technical Report also concluded that the historical drilling data were not used in the Mineral Resource Estimate because of the limited availability of supporting data and the inability to adequately verify the historical information. Based on the information available for review, GRE's QP is also of the opinion that the historical drilling data are insufficiently documented and cannot be adequately verified; therefore, the historical drillholes completed by Bear Creek and Vena are not suitable for use in the current Mineral Resource Estimate and are presented in this Technical Report for historical and exploration context only.

12.3 Verification of DLP Database (2022-2025)

From 2022 through 2025, DLP completed 28 diamond drillholes totaling 22,613.38 m on the Aurora Property and comprising 11,126 core sample intervals. The drilling included three drillholes totaling 1,651.90 m and comprising 807 core sample intervals in 2022, ten drillholes totaling 8,258.35 m and comprising 4,073 core sample intervals in 2023, ten drillholes totaling 8,394.53 m and comprising 4,114 core sample intervals in 2024, and five drillholes totaling 4,308.60 m and comprising 2,132 core sample intervals in 2025. The data were provided to GRE in digital format and included drillhole collar coordinates, downhole survey data, geological logging, analytical results, and QA/QC information for the 2022 to 2025 drilling programs.

GRE's QP independently reviewed the DLP databases by comparing the digital databases with the original analytical certificates and source files. The review included verification of collar coordinates, downhole survey data, geological logging, sample intervals, assay data, and QA/QC information. Approximately 27% of the analytical database was manually spot-checked against the original laboratory certificates and source records for accuracy, and no transcription errors or material discrepancies were identified.

12.4 Verification of DLP's In-House QA/QC Program (2022–2025)

DLP implemented an in-house QA/QC program for the 2022 through 2025 drilling campaigns on the Aurora Property. The QA/QC program consisted of the systematic insertion of blank samples, CRMs, and duplicate samples into the sample stream to monitor potential contamination and assess the analytical accuracy and precision of the assay data. All core samples, including QA/QC samples, were submitted to the ALS sample preparation facility in Arequipa, Peru, and the prepared pulp samples were subsequently dispatched to ALS Lima, Peru, for geochemical analysis. The corresponding analytical certificates and QA/QC datasets were available for independent review.

GRE's QP independently reviewed the complete QA/QC datasets for the 2022 to 2025 drilling campaigns and prepared and evaluated QA/QC control charts for blank samples, CRMs, and duplicate samples for copper, molybdenum, and silver analyses. The results of the QA/QC review for each drilling campaign are presented in the following sections.

12.4.1 QA/QC Review (2022)

12.4.1.1 Blank Samples (2022)

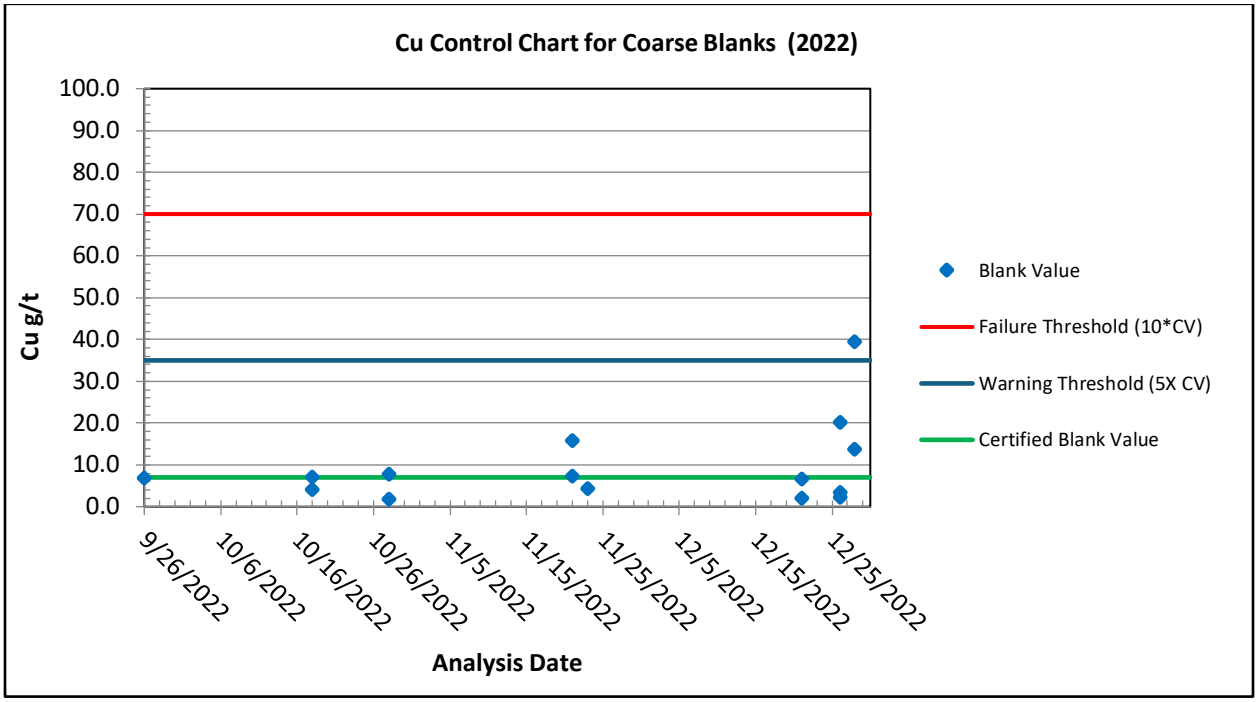
A total of 15 blank samples were inserted during the 2022 drilling campaign. The blank material consisted of crushed quartz prepared by DLP and independently certified by Intertek Genalysis Perth, Western Australia. The certified blank values are 7 ppm Cu and 1 ppm Mo, and the blank samples were inserted at a rate of approximately one blank sample per 54 routine samples from the 2022 drilling program.

For the evaluation of blank sample performance, GRE's QP adopted warning and failure thresholds based on multiples of the certified blank values for copper and molybdenum. Warning thresholds were established at five times the certified value (5×CV), corresponding to 35 ppm Cu and 5 ppm Mo, whereas failure thresholds were established at ten times the certified value (10×CV), corresponding to 70 ppm Cu and 10 ppm Mo. Silver was not certified in the blank reference material; therefore, GRE's QP adopted a threshold based of five times the analytical detection limit (5×DL). The analytical detection limit for Ag was 0.05 ppm, resulting in an Ag threshold of 0.25 ppm.

Figure 12-1 to Figure 12-3 present the copper, molybdenum, and silver blank sample results for the 2022 drilling campaign. One copper blank sample exceeded the warning threshold of 35 ppm Cu; however, no blank samples exceeded the failure threshold of 70 ppm Cu. No molybdenum or silver blank samples exceeded their respective thresholds. Overall, the blank sample results indicate that contamination during sample preparation and analytical procedures was not a material issue during the 2022 drilling campaign.

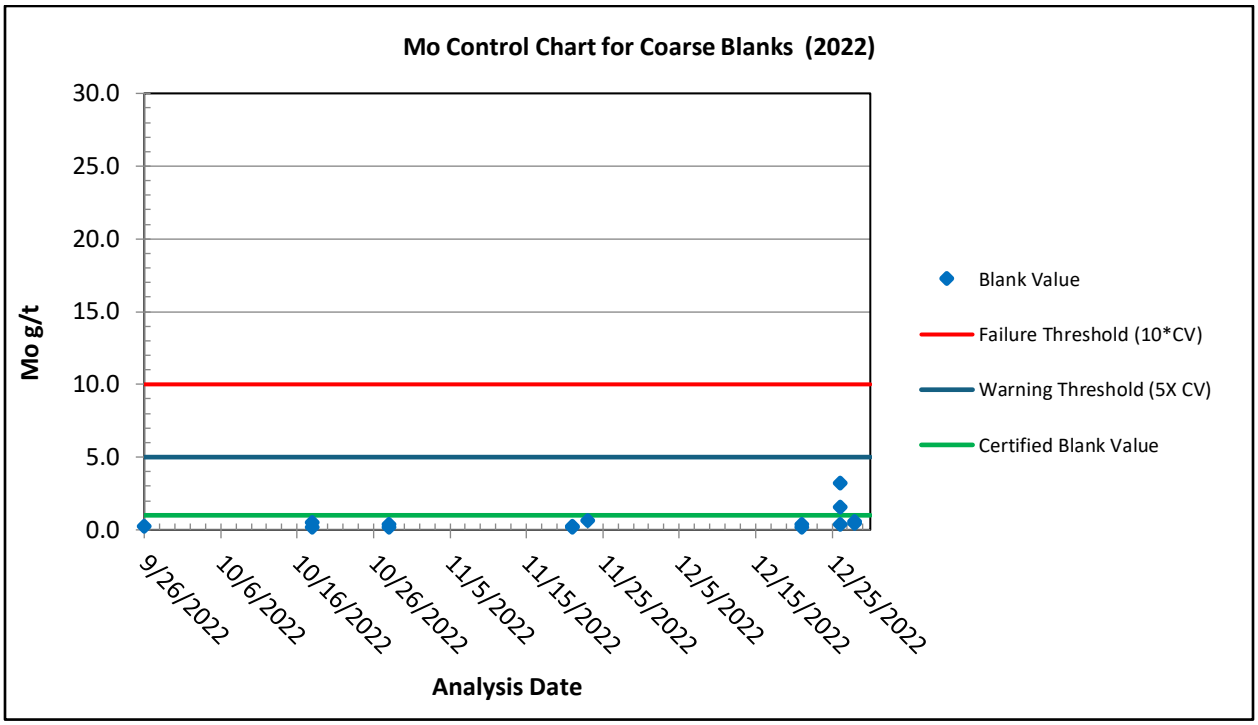
The 2022 drilling campaign's blank sample insertion rate falls below the generally accepted industry guideline of approximately one blank sample per 20 routine samples.

Figure 12-1: 2022 Drilling Campaign, Cu Blank Results



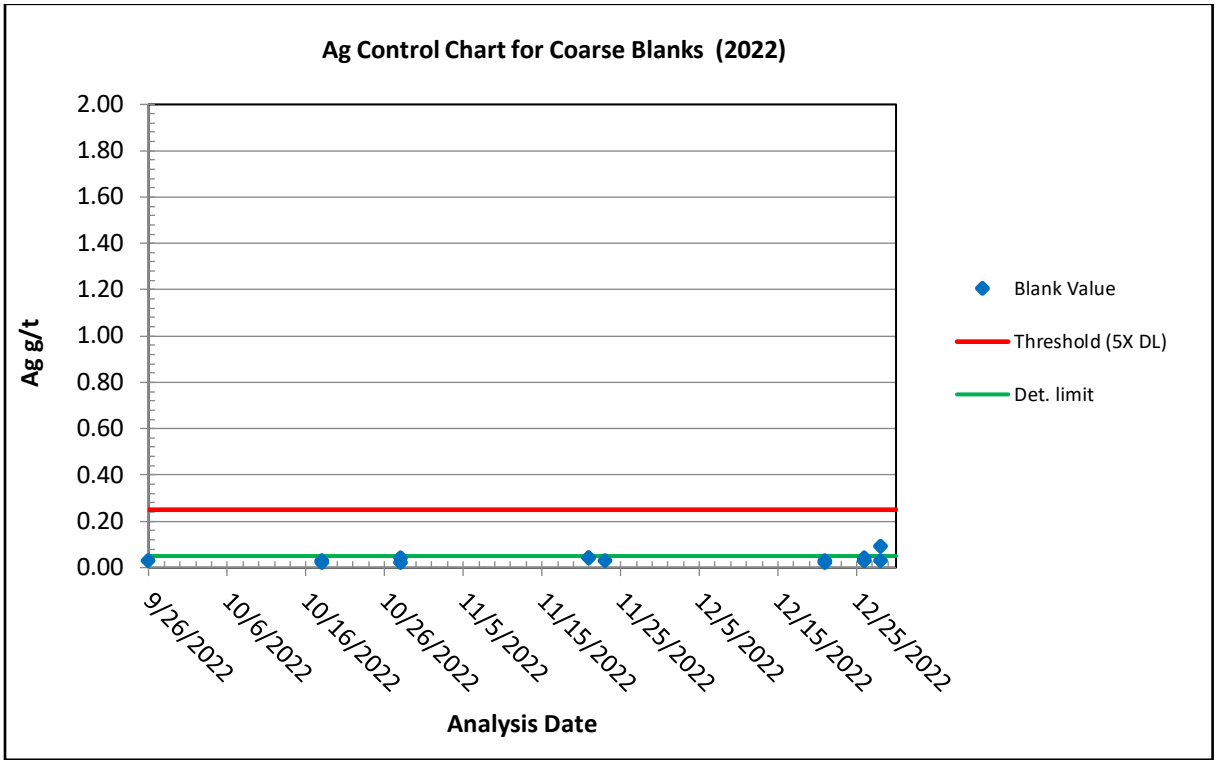
Source: GRE, 2026

Figure 12-2: 2022 Drilling Campaign, Mo Blank Results



Source: GRE, 2026

Figure 12-3: 2022 Drilling Campaign, Ag Blank Results



Source: GRE, 2026

12.4.1.2 Certified Reference Materials (2022)

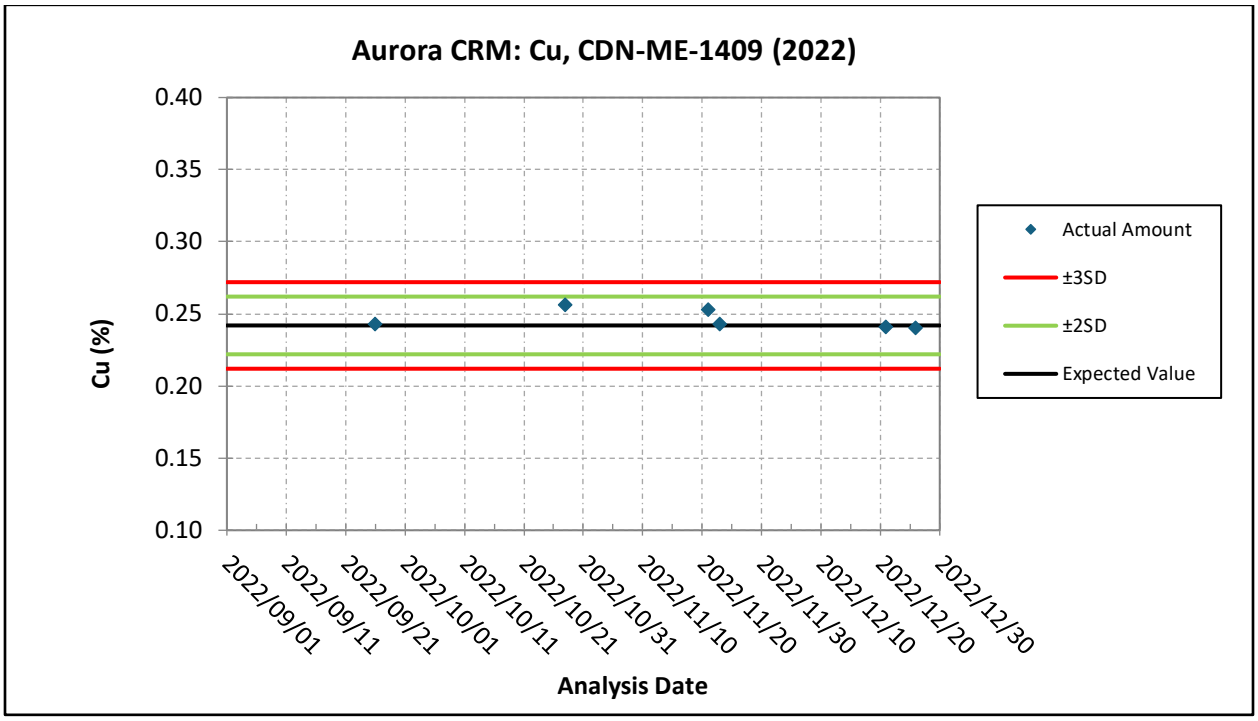
A total of 14 CRM samples were inserted during the 2022 drilling campaign, at a rate of approximately one CRM sample per 58 routine samples. The CRMs used during the 2022 drilling campaign were CDN-ME-1409 and CDN-ME-1704, which contain certified concentrations of copper and silver that are representative of the mineralization encountered on the Aurora Property.

GRE's QP independently reviewed the CRM analytical results and prepared control charts to evaluate the analytical accuracy of the assay data. The CRM results were assessed by comparing the reported analytical values with the certified values and their corresponding ±2 standard deviation (±2SD) and ±3 standard deviation (±3SD) control limits. Figure 12-4 to Figure 12-7 present the control charts for copper and silver analyses of the CDN-ME-1409 and CDN-ME-1704 certified reference materials.

Review of the CRM control charts indicates that all CRMs analyses were within the established ±3SD control limits, and the majority of the analytical results were within the ±2SD control limits. No systematic analytical bias or material analytical issues were identified. Overall, the CRM results demonstrate acceptable analytical accuracy and indicate that the assay data generated during the 2022 drilling campaign are reliable for geological interpretation and subsequent resource estimation.

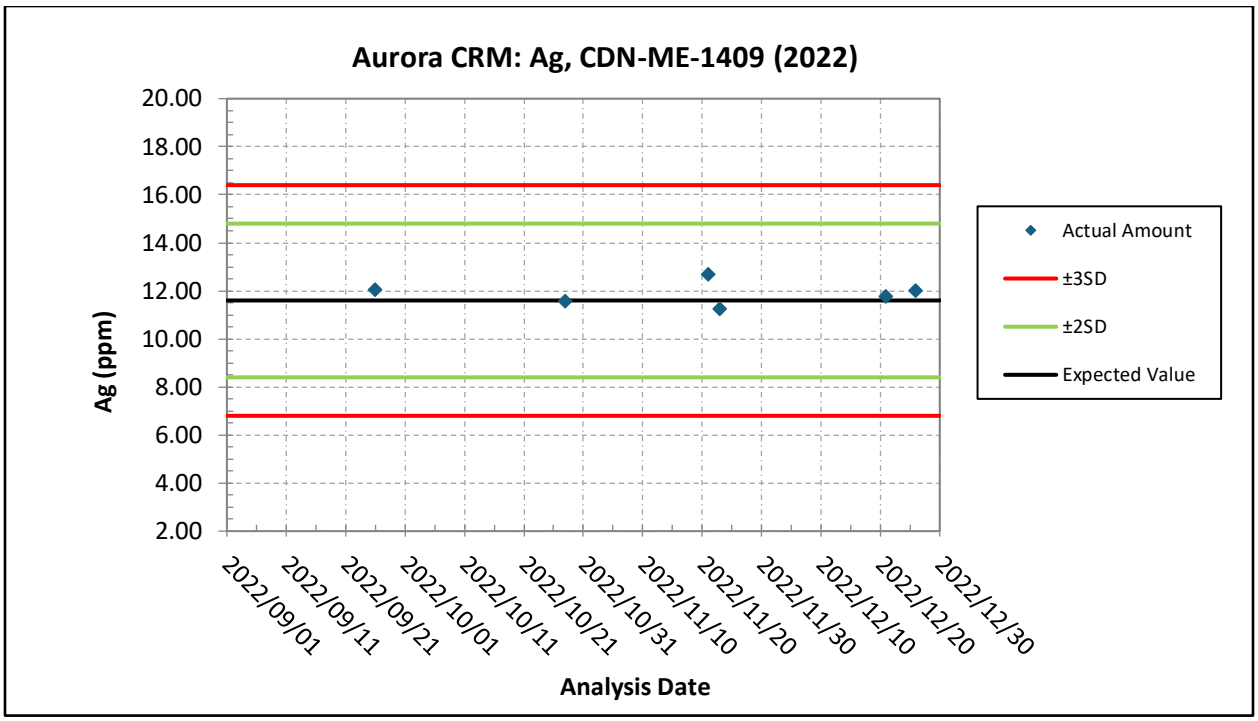
The 2022 drilling campaign's CRM insertion rate falls below the generally accepted industry guideline of approximately one CRM sample per 20 routine samples.

Figure 12-4: CDN-ME-1409, Cu from the 2022 Drilling Campaign



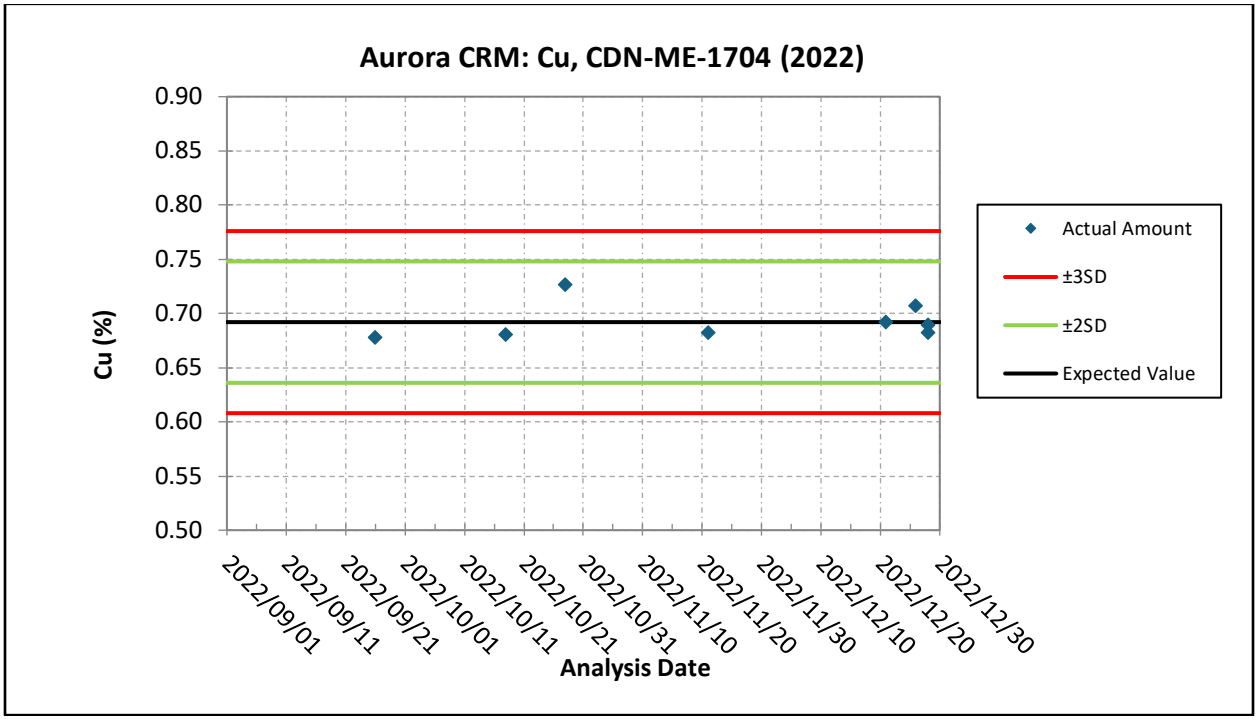
Source: GRE, 2026

Figure 12-5: CDN-ME-1409, Ag from the 2022 Drilling Campaign



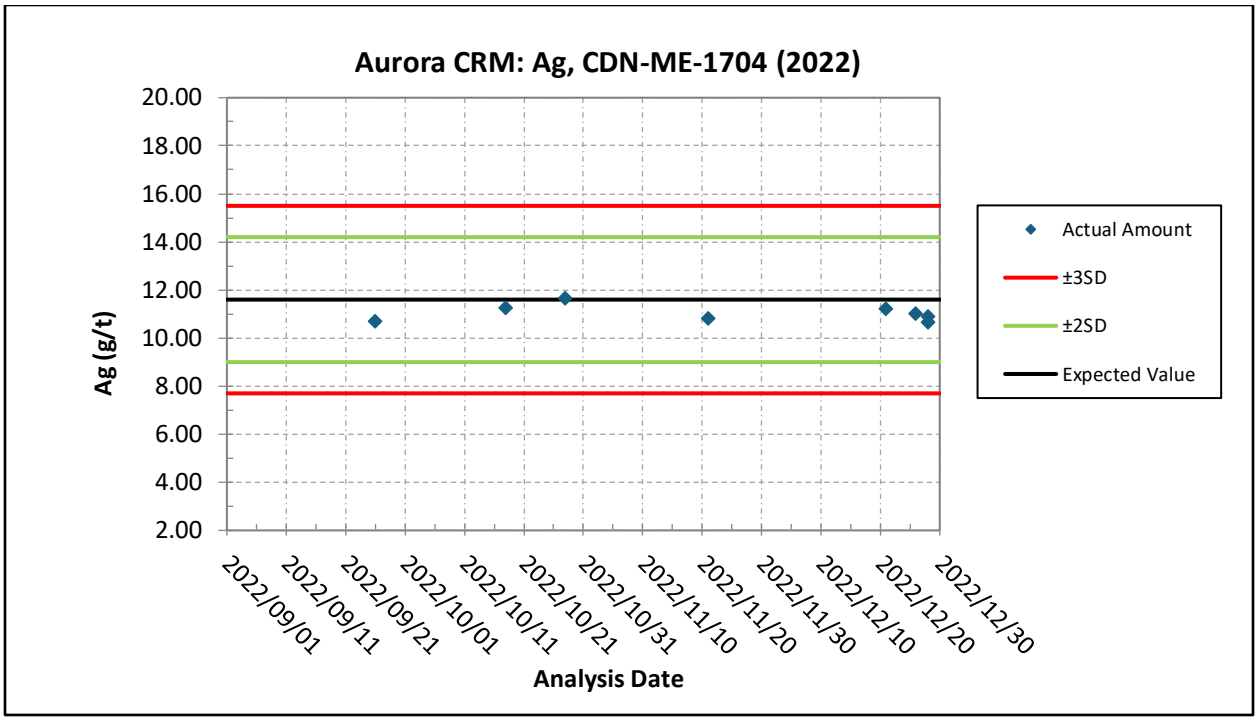
Source: GRE, 2026

Figure 12-6: CDN-ME-1704, Cu from the 2022 Drilling Campaign



Source: GRE, 2026

Figure 12-7: CDN-ME-1704, Ag from the 2022 Drilling Campaign



Source: GRE, 2026

12.4.1.3 Coarse Reject Duplicate Samples (2022)

A total of 14 coarse reject duplicate samples were submitted during the 2022 drilling campaign, representing an insertion rate of approximately 1.7 duplicate samples per 100 routine samples

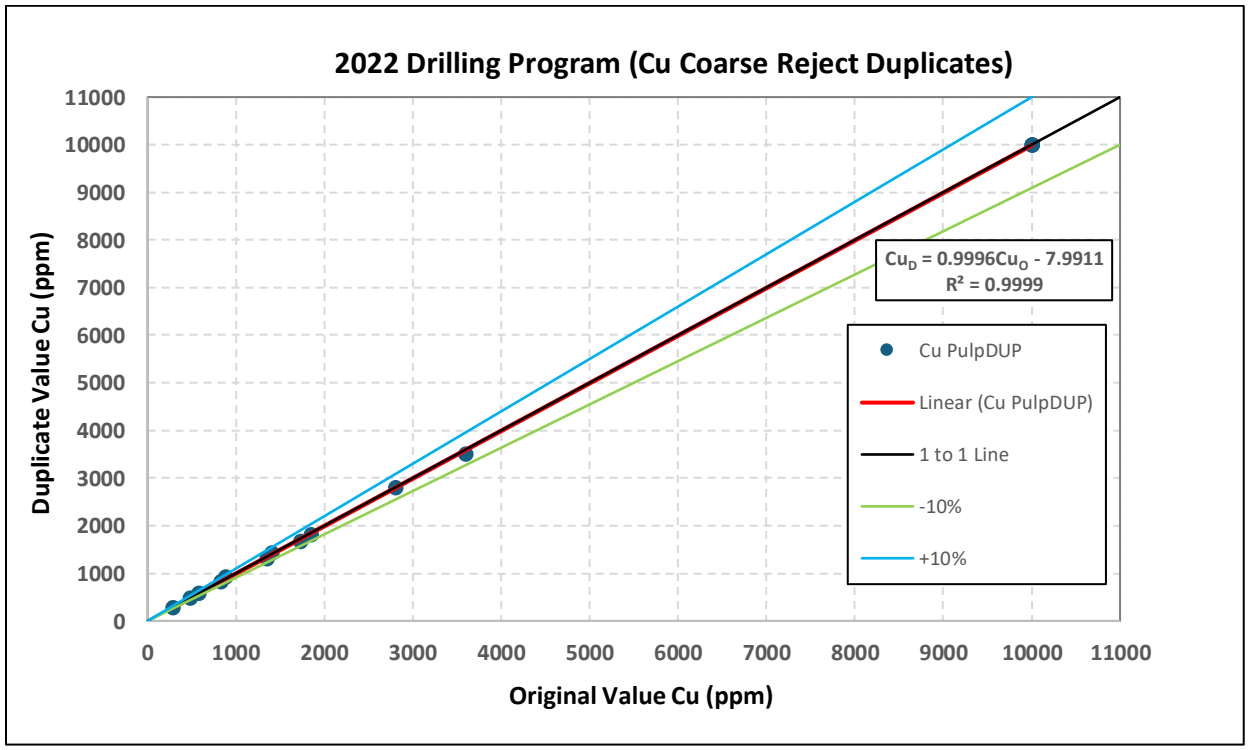
(approximately one duplicate sample per 58 routine samples). Coarse reject duplicate samples were prepared by re-splitting the crushed reject material from the original samples and were submitted to ALS for analysis.

Figure 12-8 to Figure 12-10 present Q-Q plots comparing the original and duplicate analyses for copper, molybdenum, and silver. The duplicate datasets exhibit excellent correlations, with coefficient of determination (R^2) values of 0.9999 for copper, 0.9997 for molybdenum, and 0.9901 for silver. Nearly all duplicate pairs plot within the ±10% control limits and closely follow the 1:1 line.

Overall, the coarse reject duplicate results demonstrate excellent analytical precision and reproducibility across the range of analyzed concentrations. No material analytical precision issues or systematic biases were identified, and the duplicate results indicate that the assay data generated during the 2022 drilling campaign are suitable for geological interpretation and subsequent resource estimation.

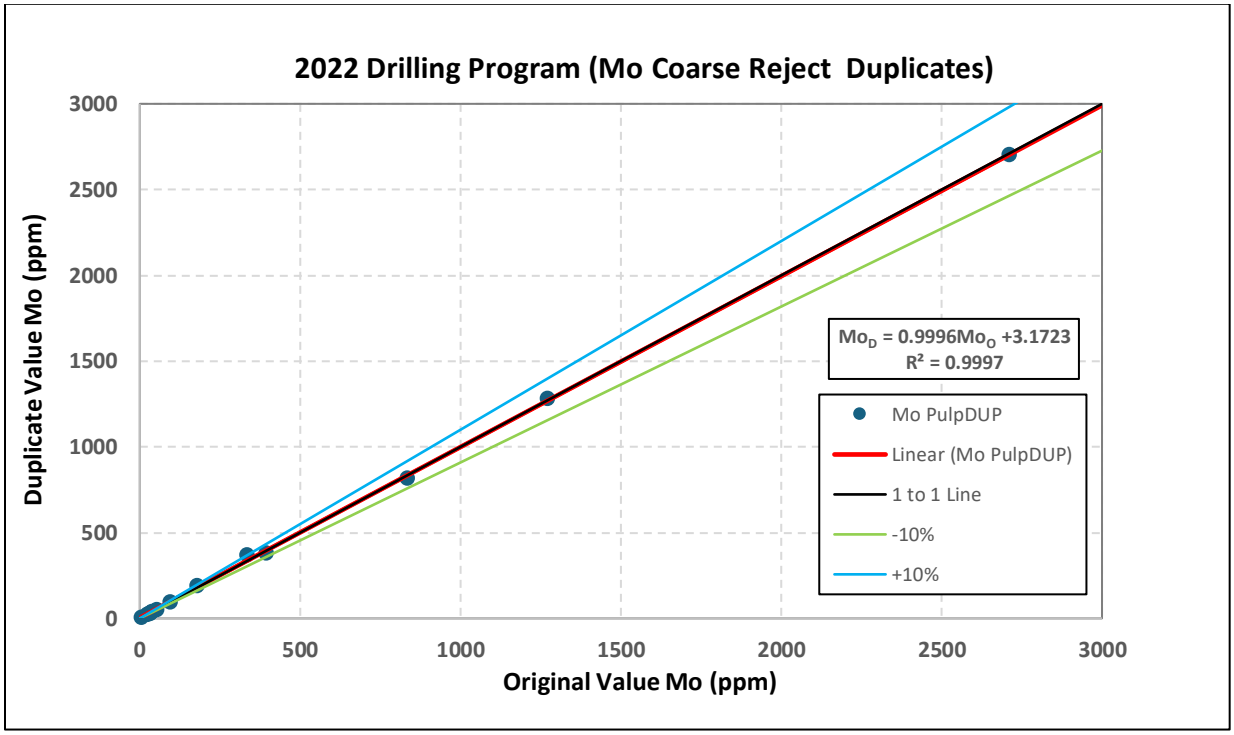
The 2022 drilling campaign's coarse reject duplicate insertion rate falls below the generally accepted industry guideline of approximately one duplicate sample per 20 routine samples.

Figure 12-8: Cu Coarse Reject Duplicate Samples from the 2022 Drilling Campaign



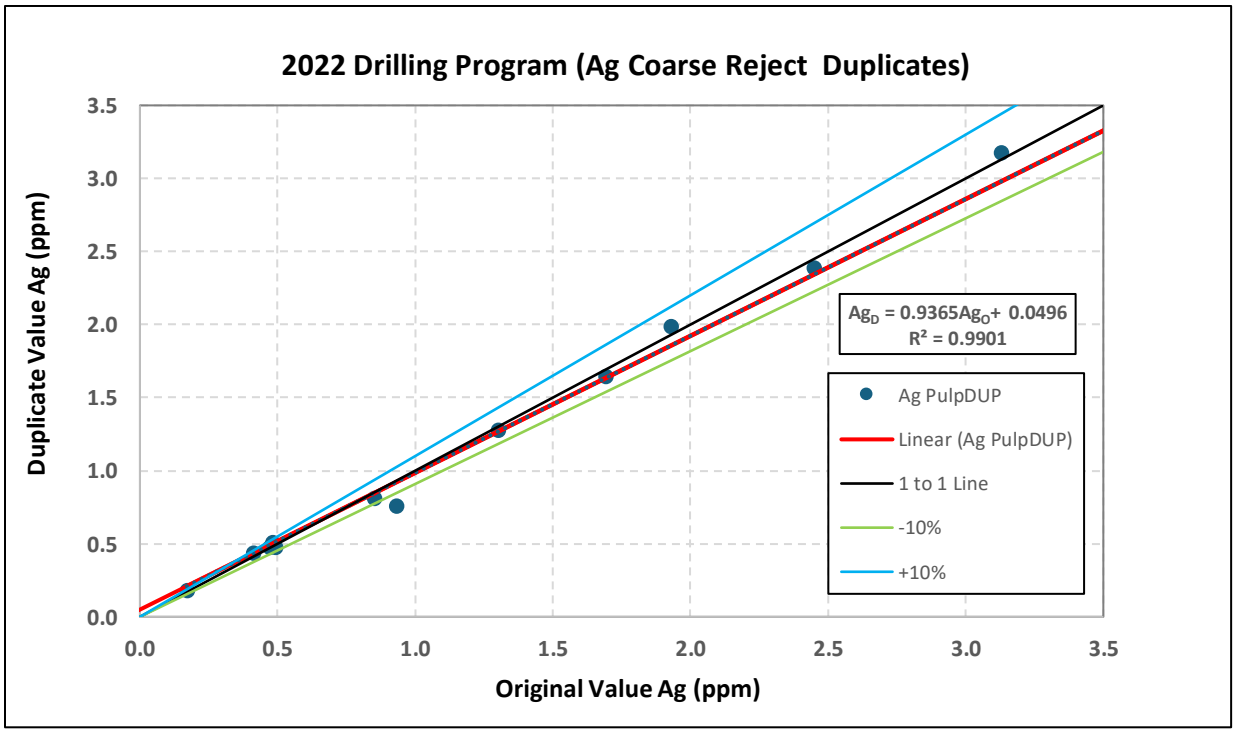
Source: GRE, 2026

Figure 12-9: Mo Coarse Reject Duplicate Samples from the 2022 Drilling Campaign



Source: GRE, 2026

Figure 12-10: Ag Coarse Reject Duplicate Samples from the 2022 Drilling Campaign



Source: GRE, 2026

12.4.2 QA/QC Review (2023)

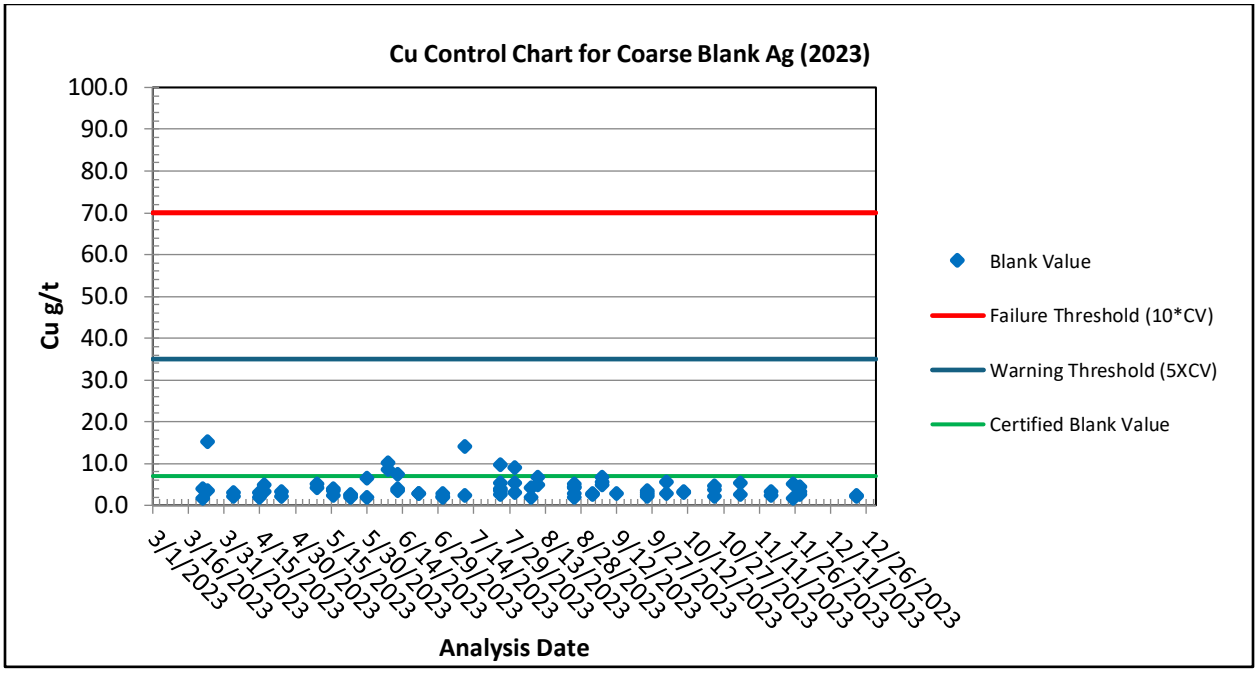
12.4.2.1 Blank Samples (2023)

A total of 79 blank samples were inserted during the 2023 drilling campaign at a rate of approximately one blank sample per 52 routine samples. The same blank material and evaluation criteria described in Section 12.4.1.1 were utilized for the 2023 drilling campaign.

Figure 12-11 to Figure 12-13 present the copper, molybdenum, and silver blank sample results for the 2023 drilling campaign. No copper blank samples exceeded the warning or failure thresholds. Two molybdenum blank samples exceeded the warning threshold of 5 ppm Mo; however, no samples exceeded the failure threshold of 10 ppm Mo. No silver blank samples exceeded the 0.25 ppm Ag threshold. Overall, the blank sample results indicate that contamination during sample preparation and analytical procedures was not a material issue during the 2023 drilling campaign.

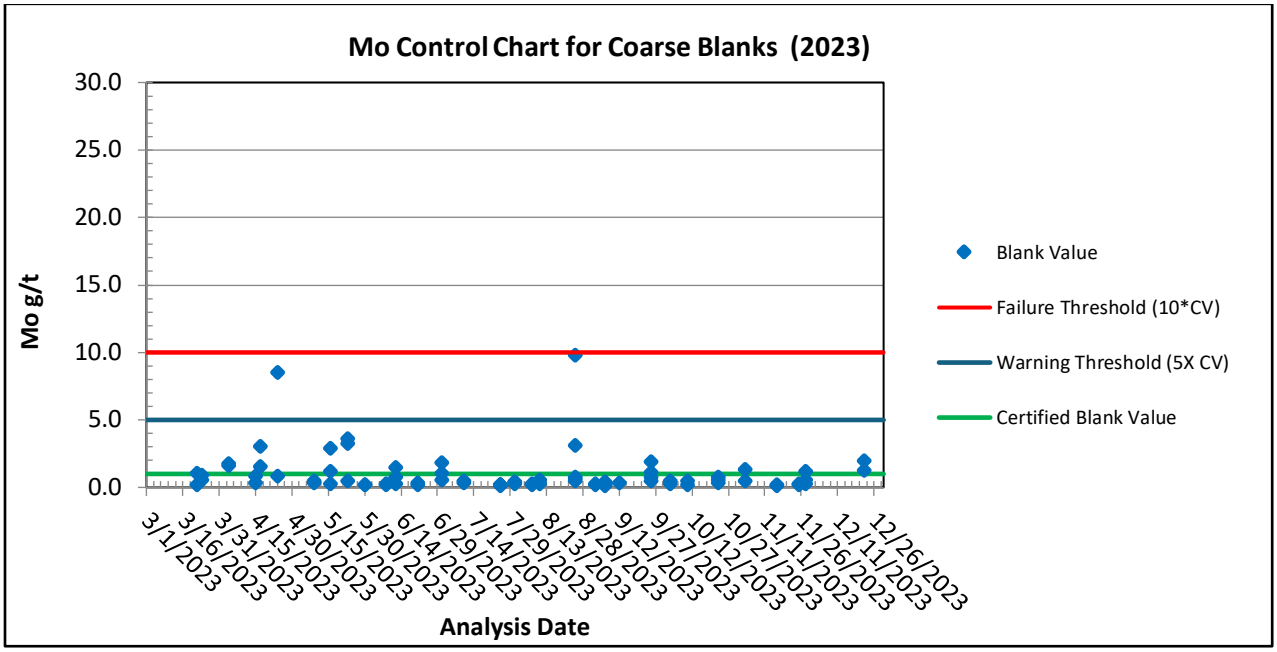
The 2023 drilling campaign's blank sample insertion rate falls below the generally accepted industry guideline of approximately one blank sample per 20 routine samples.

Figure 12-11: 2023 Drilling Campaign, Cu Blank Results



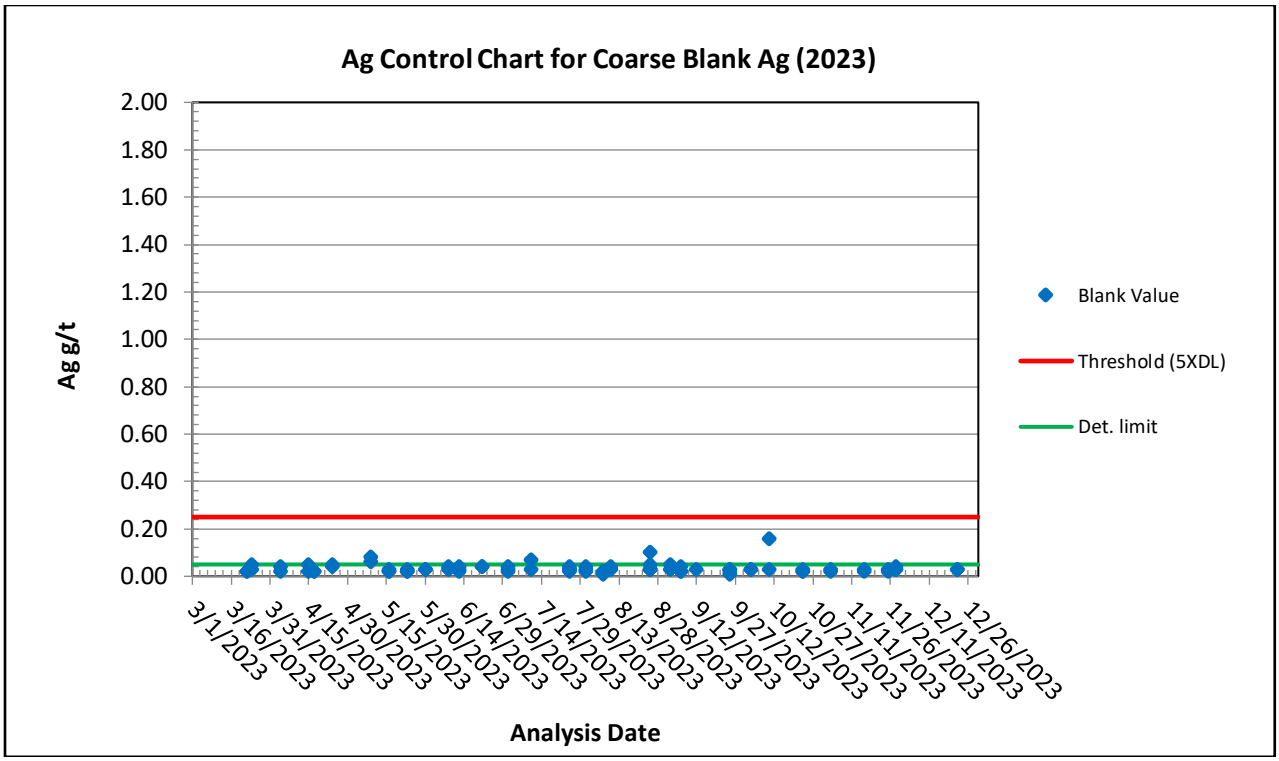
Source: GRE, 2026

Figure 12-12: 2023 Drilling Campaign, Mo Blank Results



Source: GRE, 2026

Figure 12-13: 2023 Drilling Campaign, Ag Blank Results



Source: GRE, 2026

12.4.2.2 Certified Reference Materials (2023)

A total of 69 CRM samples were inserted during the 2023 drilling campaign at a rate of approximately one CRM sample per 59 routine samples. The CRM program consisted solely of CDN-CM-33. The CRM analyses

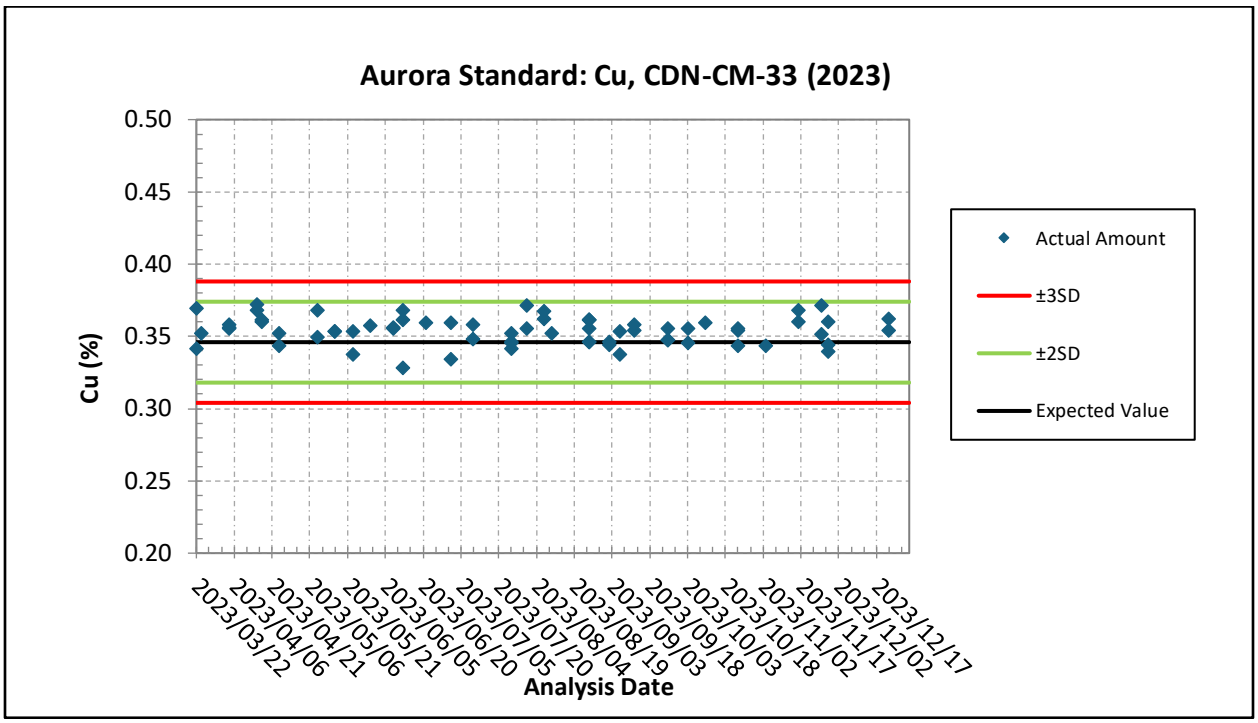
were assessed by comparing the reported analytical values with the certified values and their corresponding ±2SD and ±3SD control limits.

Figure 12-14 to Figure 12-16 present the copper, molybdenum, and silver analytical results for the CDN-CM-33 CRM. All CRM analyses were within the established ±3SD control limits and the majority of results were within the ±2SD control limits, indicating acceptable analytical accuracy and no evidence of systematic analytical bias during the 2023 drilling campaign.

Overall, the 2023 CRM results demonstrate acceptable analytical accuracy and indicate that the assay data generated during the 2023 drilling campaign are reliable for geological interpretation and subsequent resource estimation.

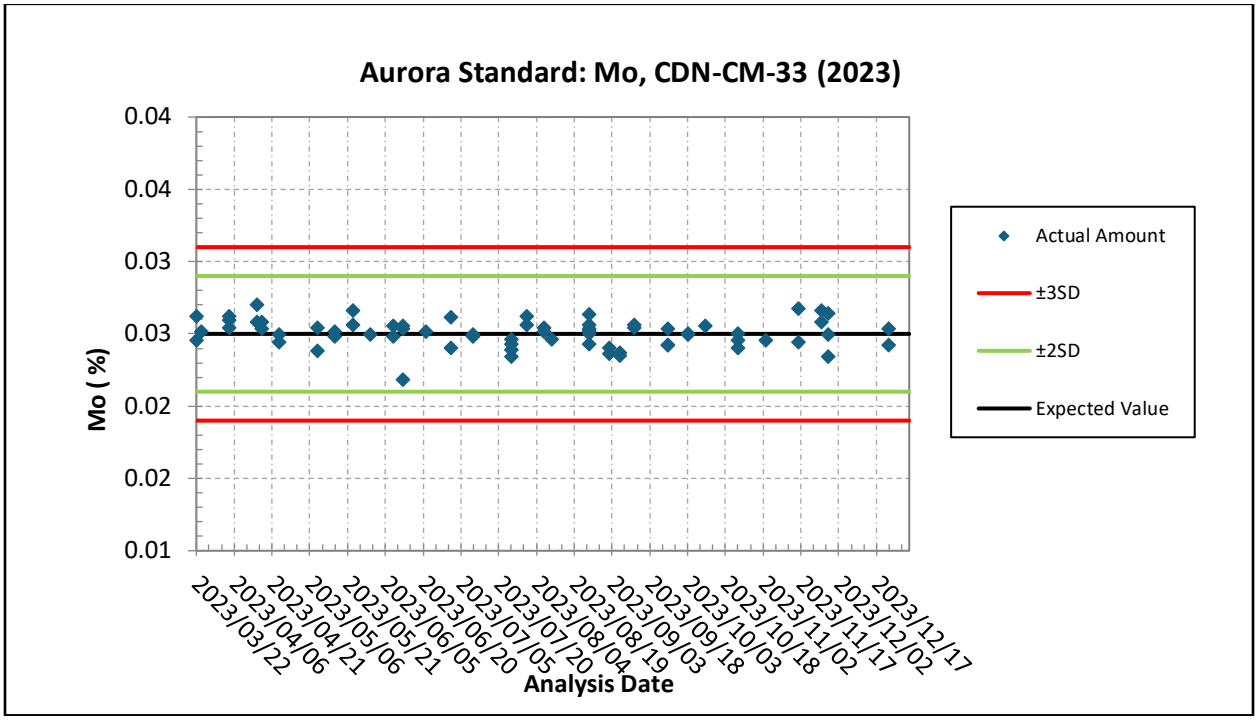
The 2023 drilling campaign's CRM insertion rate falls below the generally accepted industry guideline of approximately one CRM sample per 20 routine samples.

Figure 12-14: CDN-ME-33, Cu from the 2023 Drilling Campaign



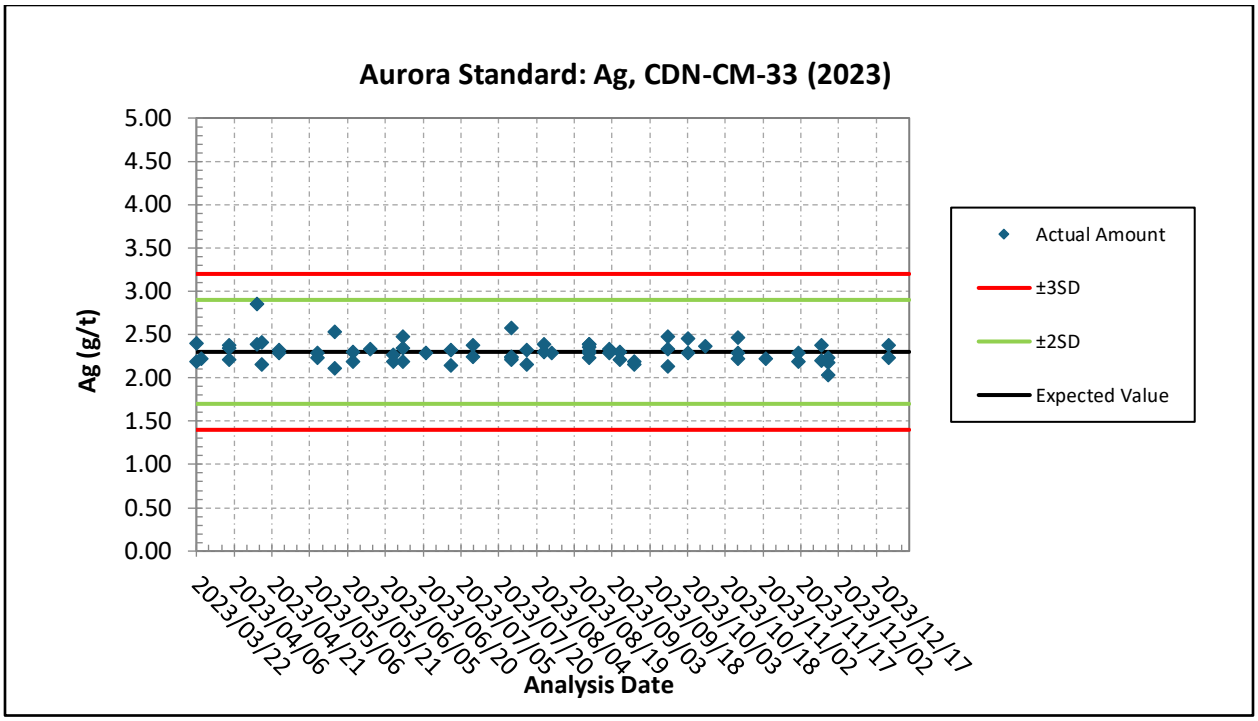
Source: GRE, 2026

Figure 12-15: CDN-ME-33, Mo from the 2023 Drilling Campaign



Source: GRE, 2026

Figure 12-16: CDN-ME-33, Ag from the 2023 Drilling Campaign



Source: GRE, 2026

12.4.2.3 Coarse Reject Duplicate Samples (2023)

A total of 74 coarse reject duplicate samples were submitted during the 2023 drilling campaign, representing an insertion rate of approximately 1.8 duplicate samples per 100 routine samples

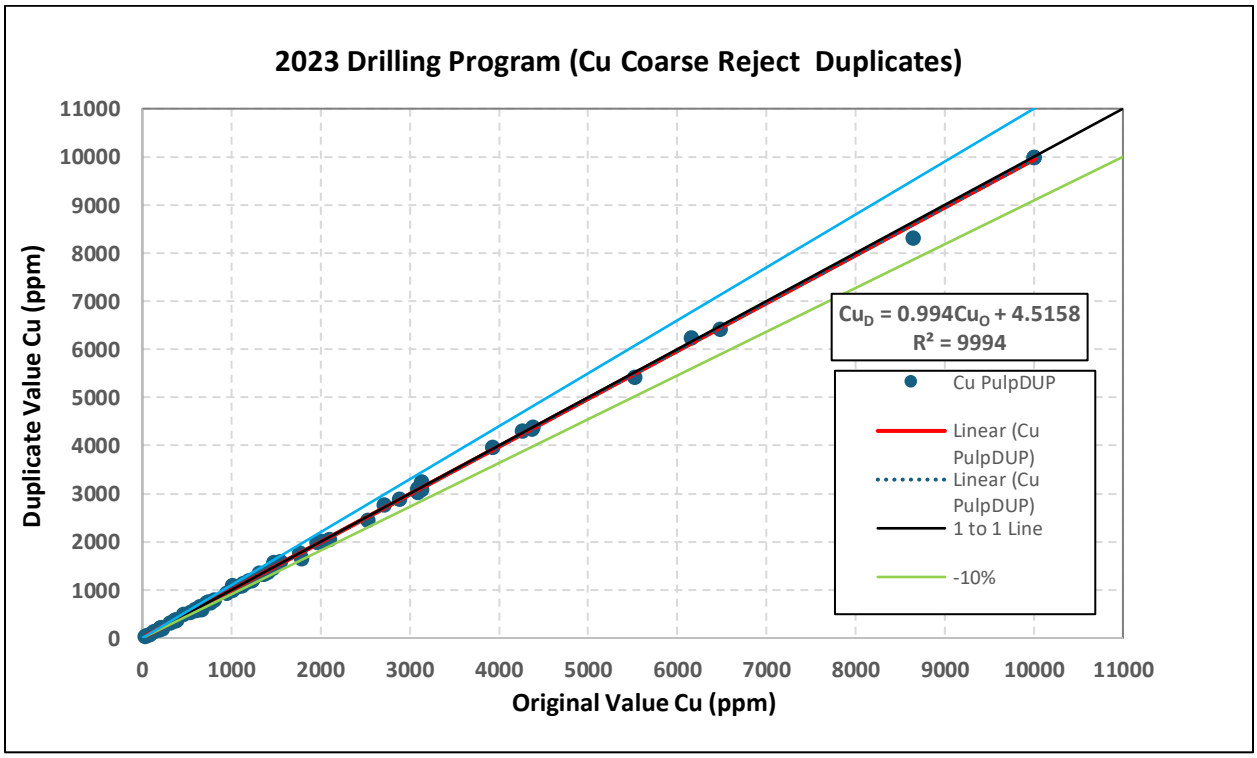
(approximately one duplicate sample per 55 routine samples). Coarse reject duplicate samples were prepared by re-splitting the crushed reject material from the original samples and were submitted to ALS for analysis.

Figure 12-17 to Figure 12-19 present Q-Q plots comparing the original and duplicate analyses for copper, molybdenum, and silver. The duplicate datasets exhibit excellent correlations, with coefficient of determination (R^2) values of 0.9994 for copper, 1.0000 for molybdenum, and 0.9995 for silver. Nearly all duplicate pairs plot within the ±10% control limits and closely follow the 1:1 line.

Overall, the coarse reject duplicate results demonstrate excellent analytical precision and reproducibility across the range of analyzed concentrations. No material analytical precision issues or systematic biases were identified, and the duplicate results indicate that the assay data generated during the 2023 drilling campaign are suitable for geological interpretation and subsequent resource estimation.

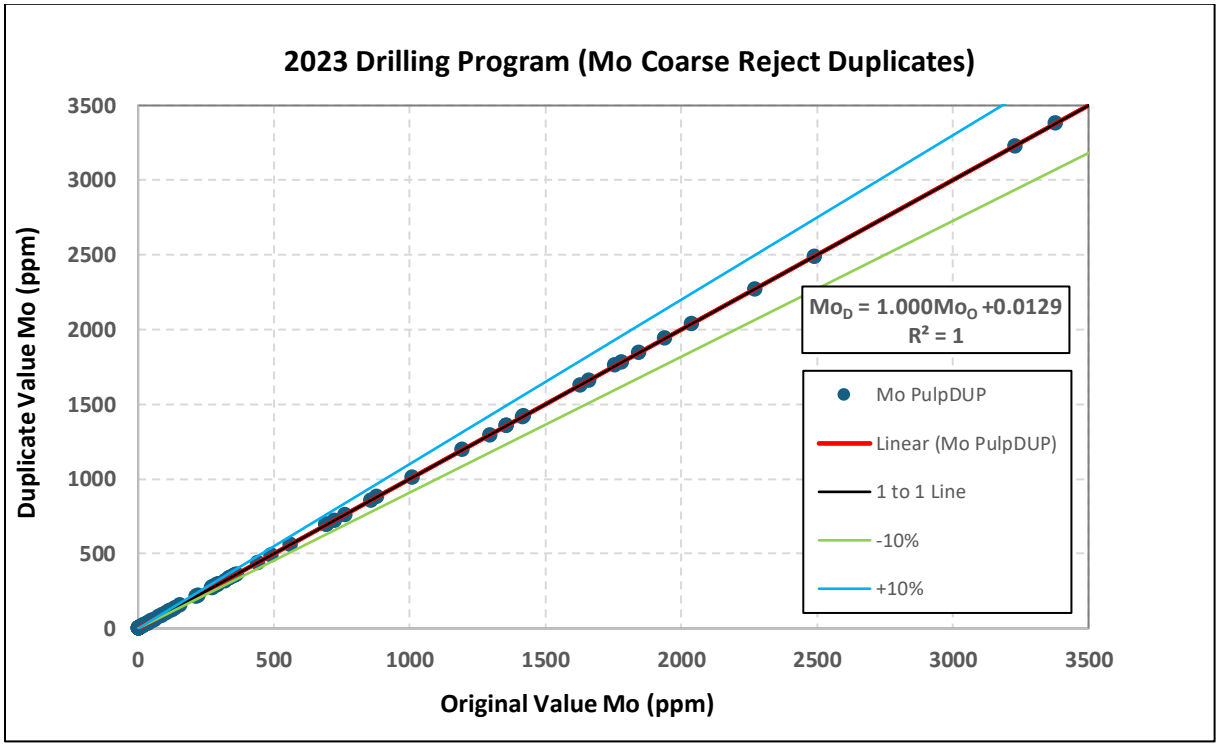
The 2023 drilling campaign's coarse reject duplicate insertion rate falls below the generally accepted industry guideline of approximately one duplicate sample per 20 routine samples.

Figure 12-17: Cu Coarse Reject Duplicate Samples from the 2023 Drilling Campaign



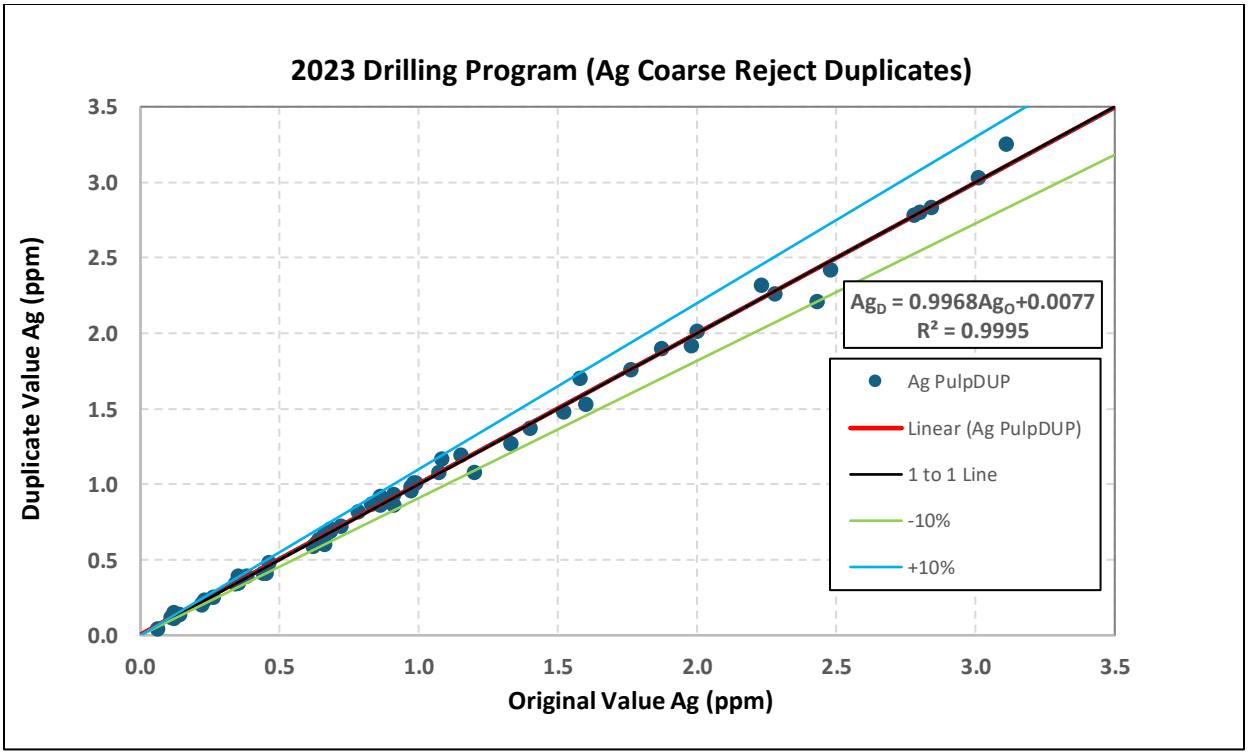
Source: GRE, 2026

Figure 12-18: Mo Coarse Reject Duplicate Samples from the 2023 Drilling Campaign



Source: GRE, 2026

Figure 12-19: Ag Coarse Reject Duplicate Samples from the 2023 Drilling Campaign



Source: GRE, 2026

12.4.3 QA/QC Review (2024)

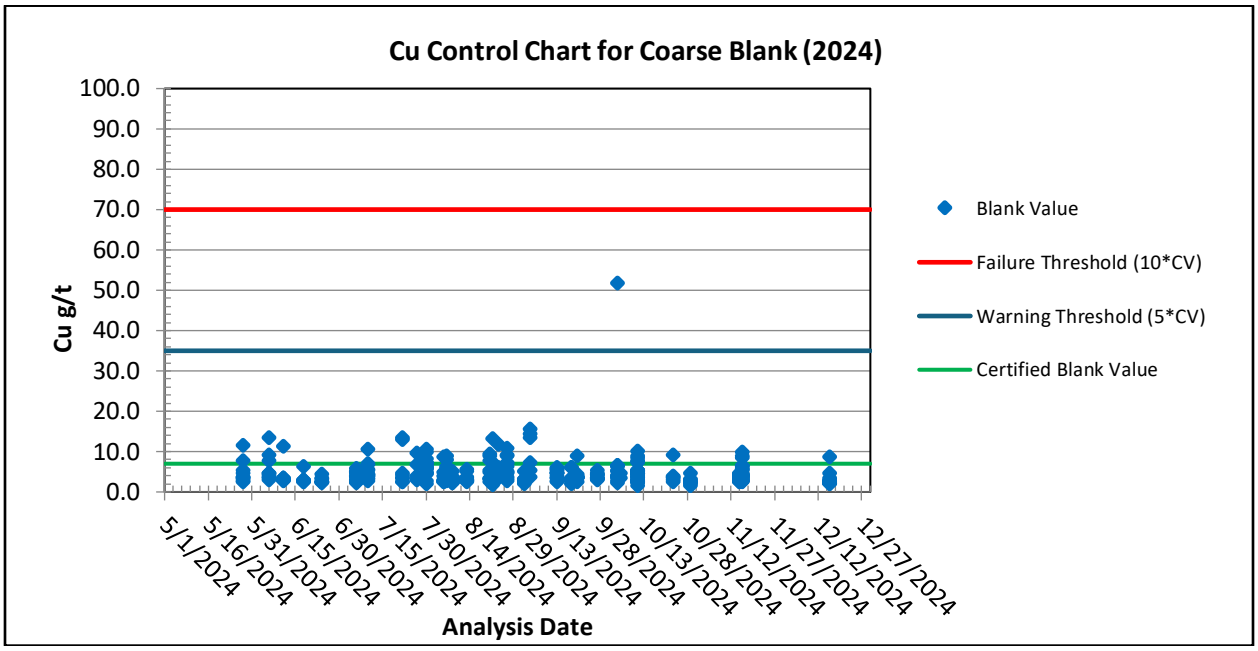
12.4.3.1 Blank Samples (2024)

A total of 349 blank samples were inserted during the 2024 drilling campaign at a rate of approximately one blank sample per 12 routine samples. The blank material consisted of crushed quartz prepared by DLP and independently certified by Intertek Genalysis Perth, Western Australia. The certified blank values are 7 ppm Cu and 1 ppm Mo. For the evaluation of blank sample performance, warning and failure thresholds of 5×CV and 10×CV were adopted for copper and molybdenum, respectively, whereas a threshold of 0.25 ppm Ag (5×DL) was adopted for silver.

Figure 12-20 to Figure 12-22 present the copper, molybdenum, and silver blank sample results for the 2024 drilling campaign. One copper blank sample exceeded the warning threshold of 35 ppm Cu; however, no samples exceeded the failure threshold of 70 ppm Cu. Several molybdenum blank samples exceeded the warning threshold of 5 ppm Mo, and one blank sample exceeded the failure threshold of 10 ppm Mo. One silver blank sample exceeded the threshold of 0.25 ppm Ag.

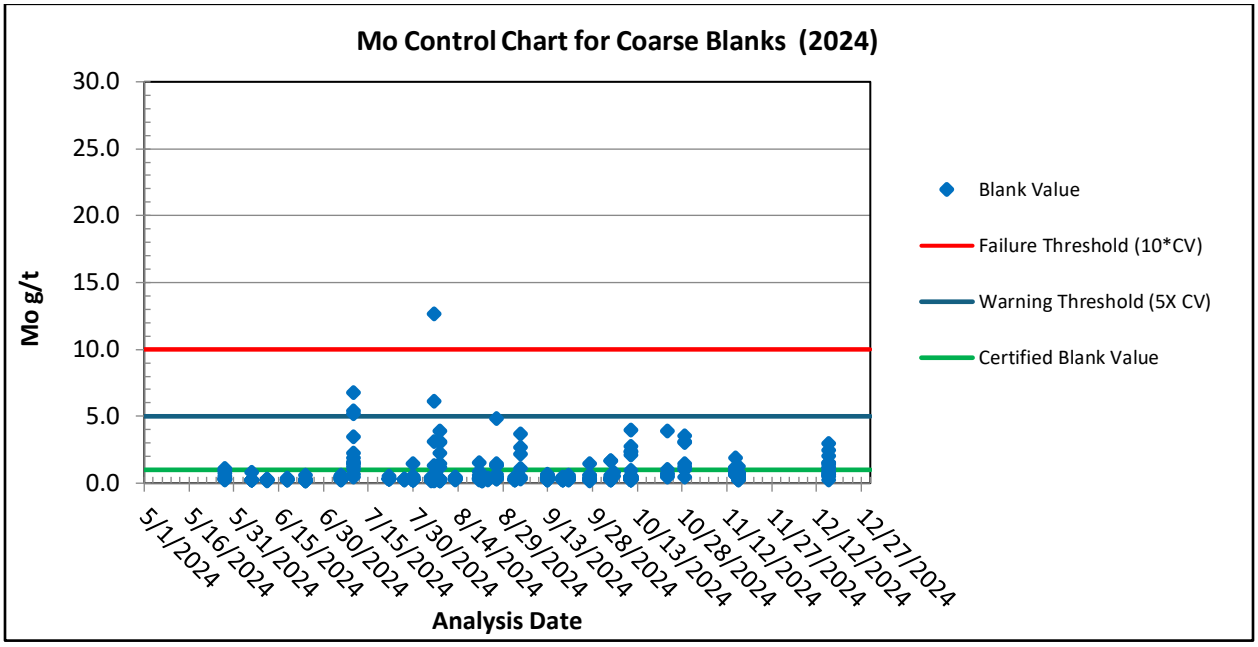
Although several isolated warning and failure events were identified, the vast majority of blank sample results remained below the established thresholds and do not indicate systematic contamination during sample preparation or analytical procedures. Overall, the blank sample results indicate that contamination was not a material issue during the 2024 drilling campaign.

Figure 12-20: 2024 Drilling Campaign, Cu Blank Results



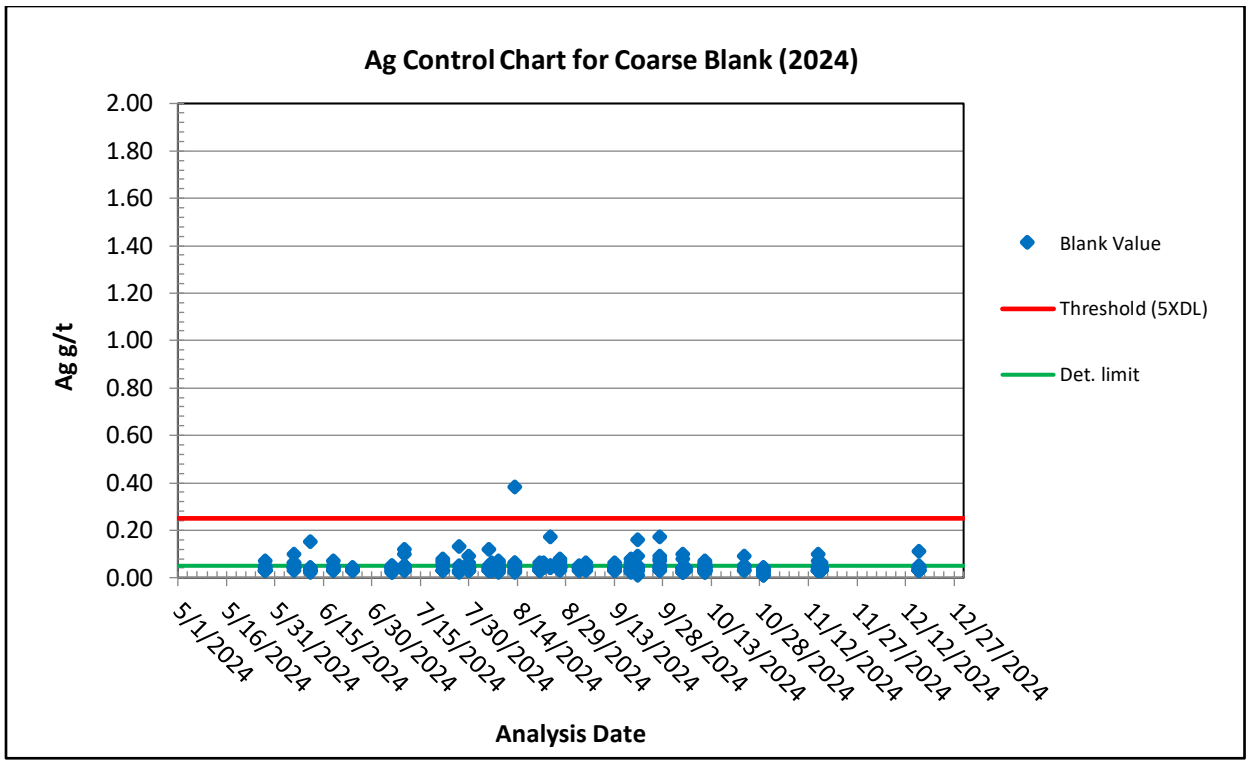
Source: GRE, 2026

Figure 12-21: 2024 Drilling Campaign, Mo Blank Results



Source: GRE, 2026

Figure 12-22: 2024 Drilling Campaign, Ag Blank Results



Source: GRE, 2026

12.4.3.2 Certified Reference Materials (2024)

A total of 337 CRM samples were inserted during the 2024 drilling campaign at a rate of approximately one CRM sample per 12 routine samples. The CRM program consisted of CDN-CM-33, CDN-ME-1704,

PORF-34, PORF-35, and PORF-39. The CRM analyses were assessed by comparing the reported analytical values with the certified values and their corresponding ±2SD and ±3SD control limits.

GRE's QP independently reviewed the complete CRM dataset for the 2024 drilling campaign and prepared and evaluated control charts for copper, molybdenum, and silver analyses. The control charts are presented in Figure 12-23 to Figure 12-36.

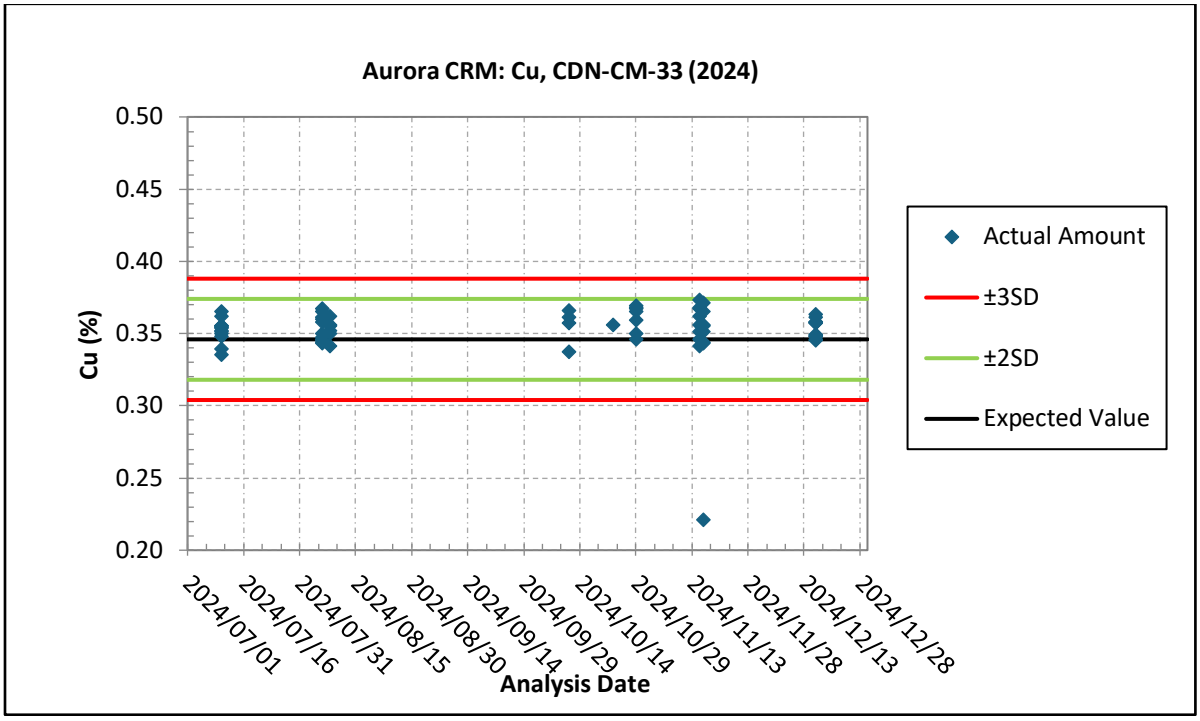
The majority of CRM analyses clustered closely around their respective certified values and fell within the established control limits, demonstrating good analytical accuracy and precision. A single CDN-CM-33 sample submitted on November 16, 2024, returned Cu, Mo, and Ag values that all fell outside the ±3SD control limits. Because all three analytes in the same CRM sample exhibited the same anomalous behavior, GRE's QP considers this result likely attributable to sample misidentification or labeling error rather than analytical bias or laboratory performance.

In addition, one CRM analysis for each of Cu, Mo, and Ag associated with PORF-35 exceeded the ±3SD control limits. These occurrences were isolated and do not indicate persistent analytical bias or systematic analytical errors.

The PORF-34 silver CRM exhibited greater variability than the other CRMs, with several analyses exceeding the +3SD control limit. Although these elevated Ag results were not associated with any evident temporal trend or laboratory bias and are not considered material to the overall reliability of the assay database, the observed dispersion warranted additional review. Accordingly, re-assaying the affected analytical batches may be considered where possible, and the potential use of an alternative high-grade silver CRM in future drilling programs should be evaluated to provide additional verification of analytical accuracy and better represent the upper range of silver grades encountered at the Aurora Project.

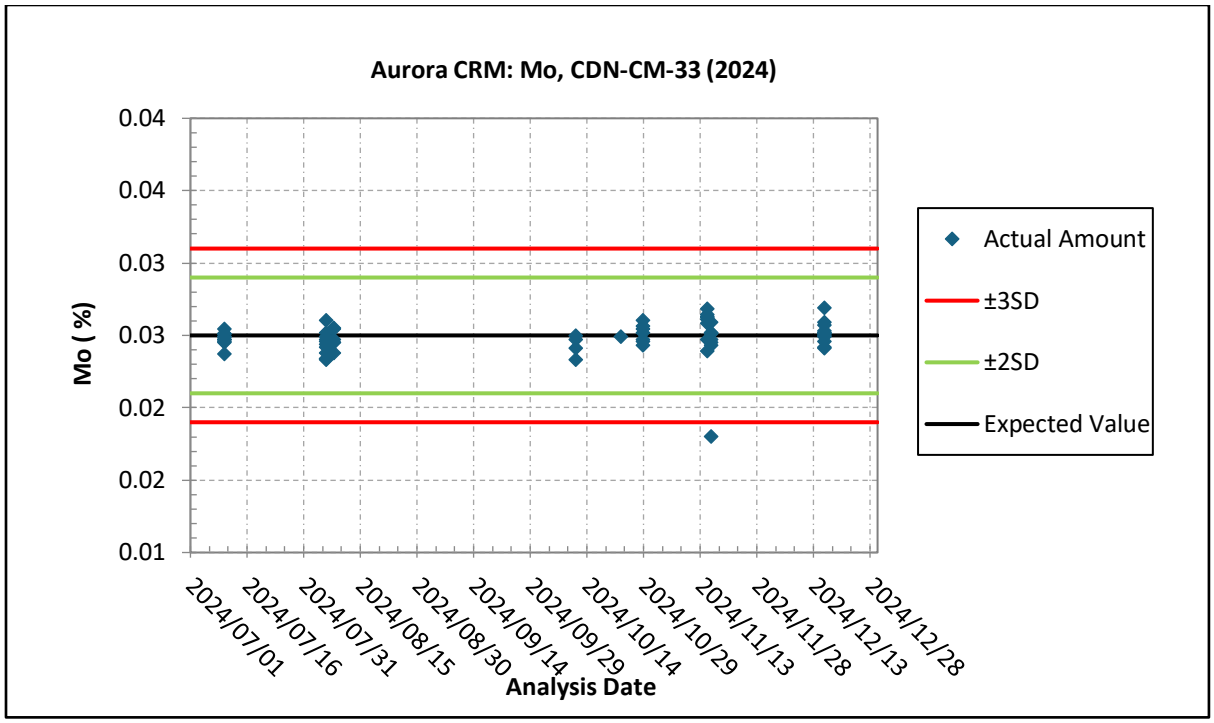
Overall, the 2024 CRM results indicate acceptable analytical performance and demonstrate that the analytical data are accurate and reliable. No material issues were identified that would adversely affect the suitability of the assay database for geological interpretation and Mineral Resource Estimation.

Figure 12-23: CDN-ME-33, Cu from the 2024 Drilling Campaign



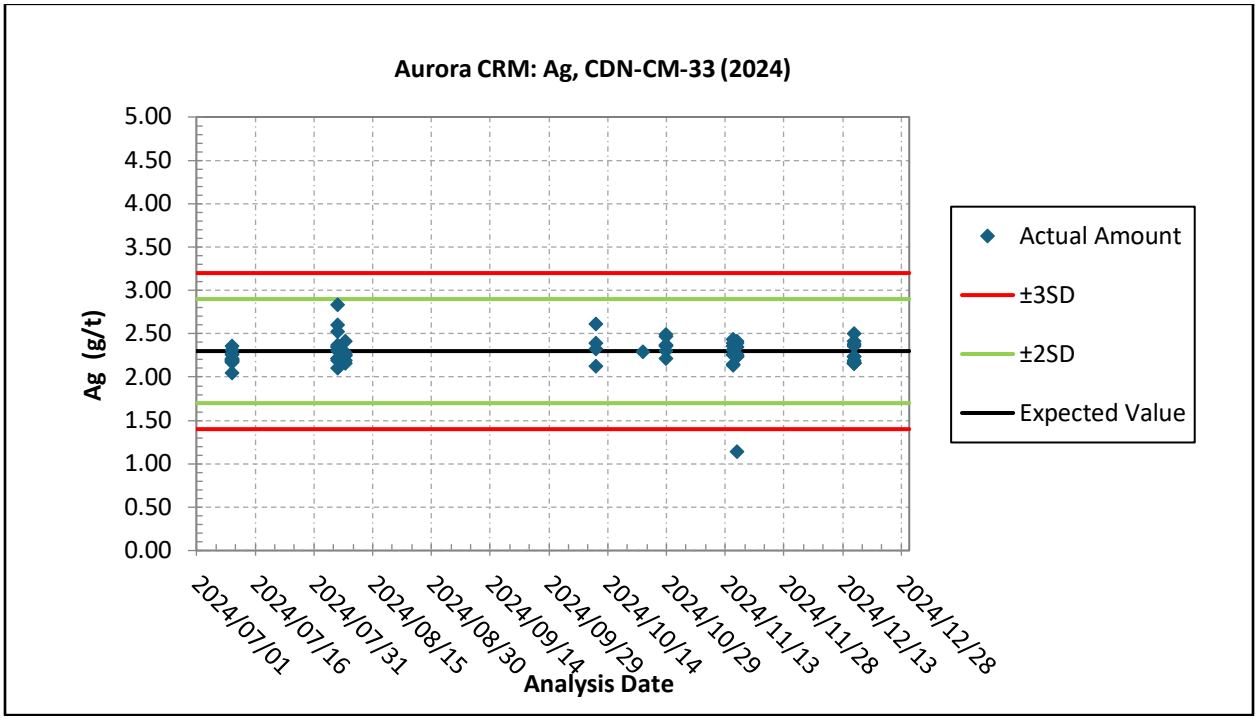
Source: GRE, 2026

Figure 12-24: CDN-ME-33, Mo from the 2024 Drilling Campaign



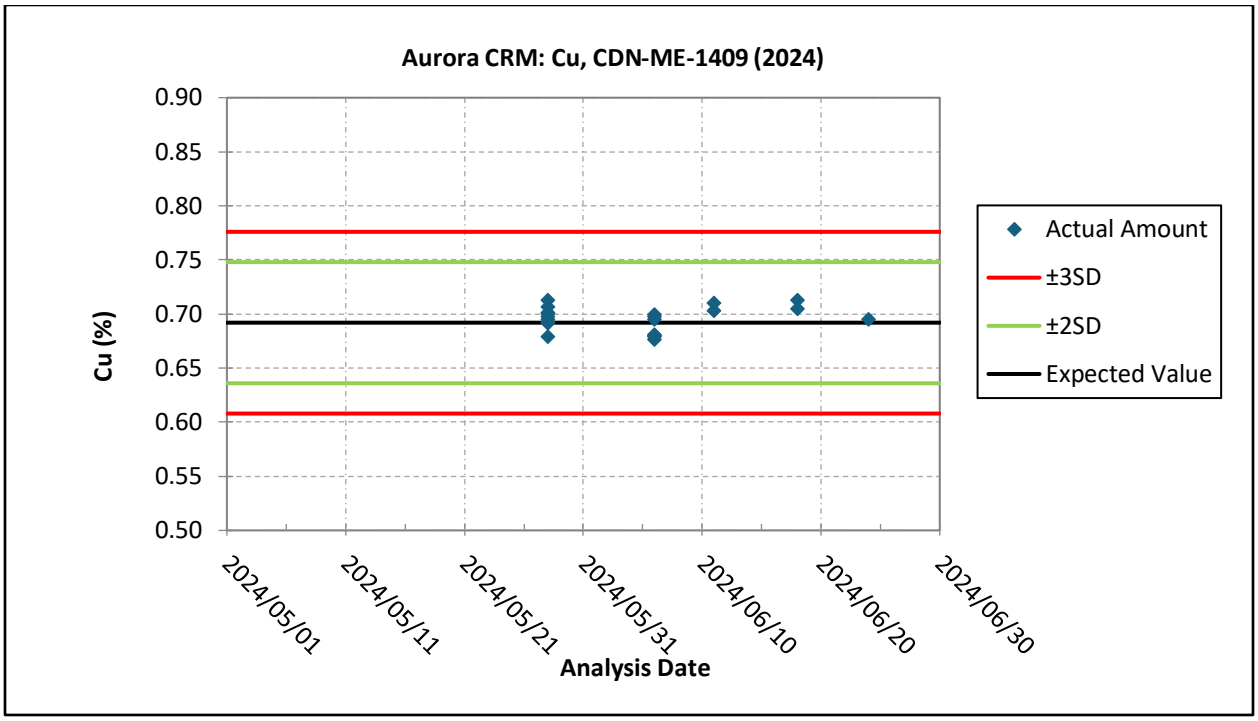
Source: GRE, 2026

Figure 12-25: CDN-ME-33, Ag from the 2024 Drilling Campaign



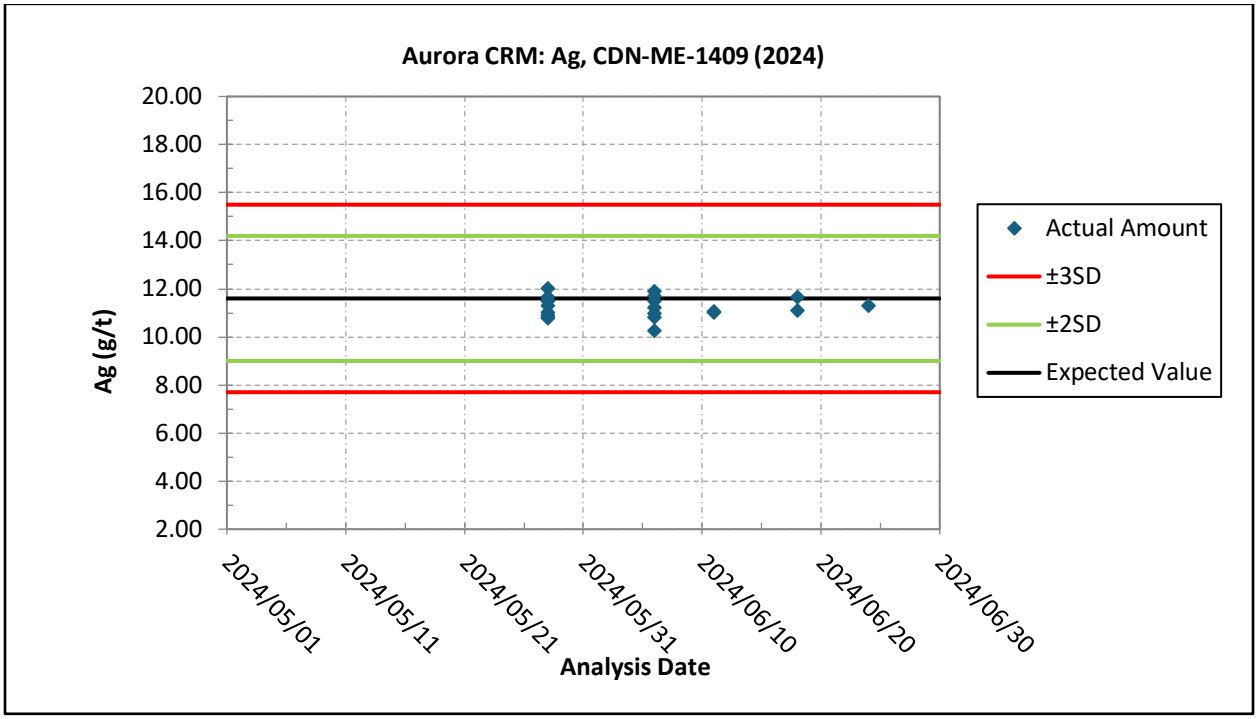
Source: GRE, 2026

Figure 12-26: CDN-ME-1409, Cu from the 2024 Drilling Campaign



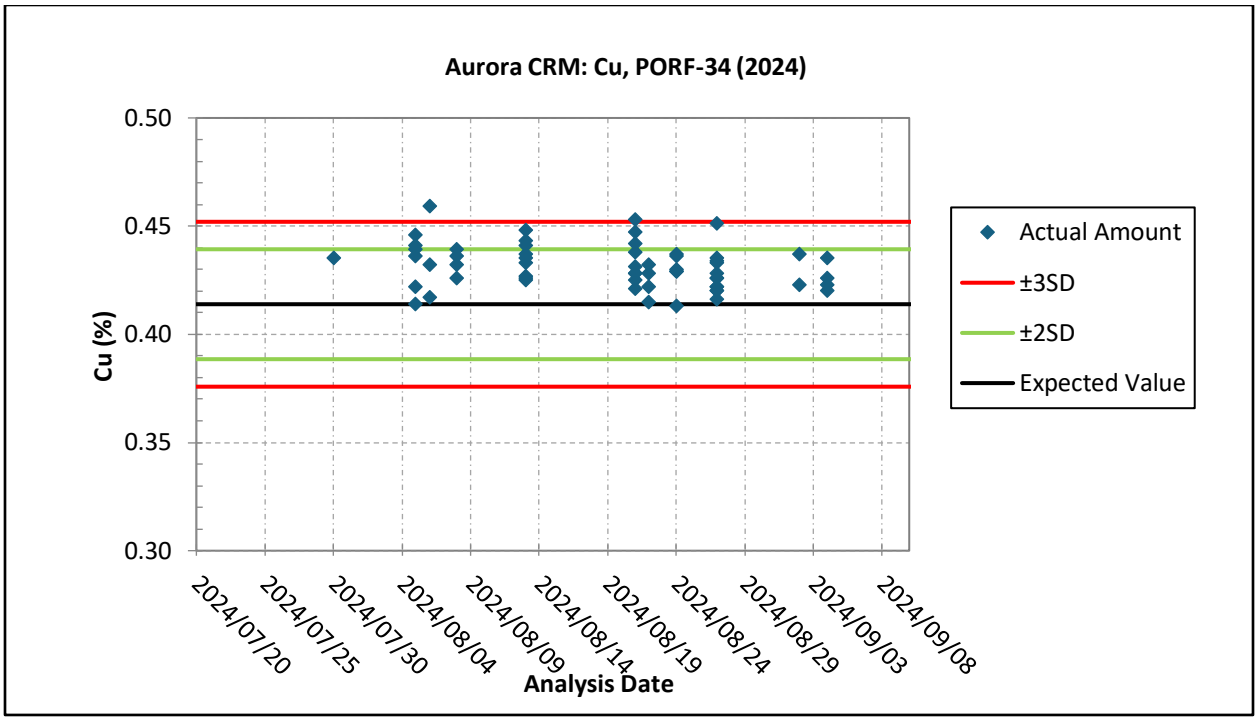
Source: GRE, 2026

Figure 12-27: CDN-ME-1409, Ag from the 2024 Drilling Campaign



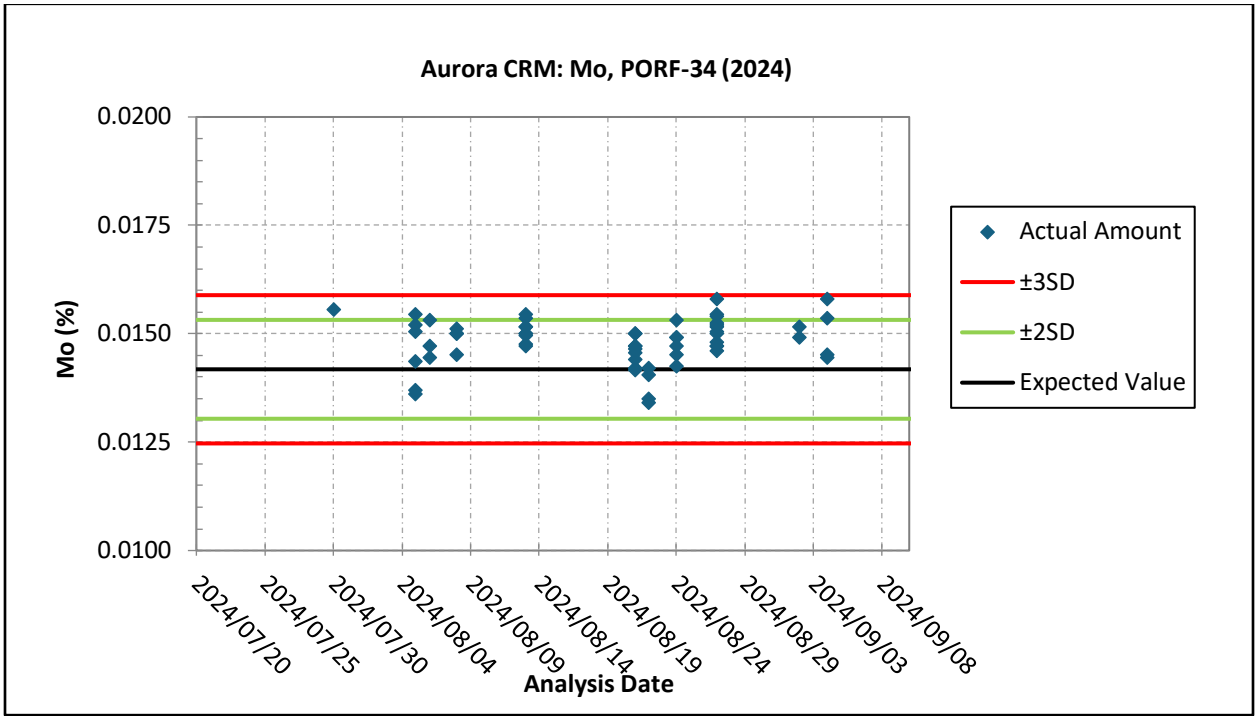
Source: GRE, 2026

Figure 12-28: PORF-34, Cu from the 2024 Drilling Campaign



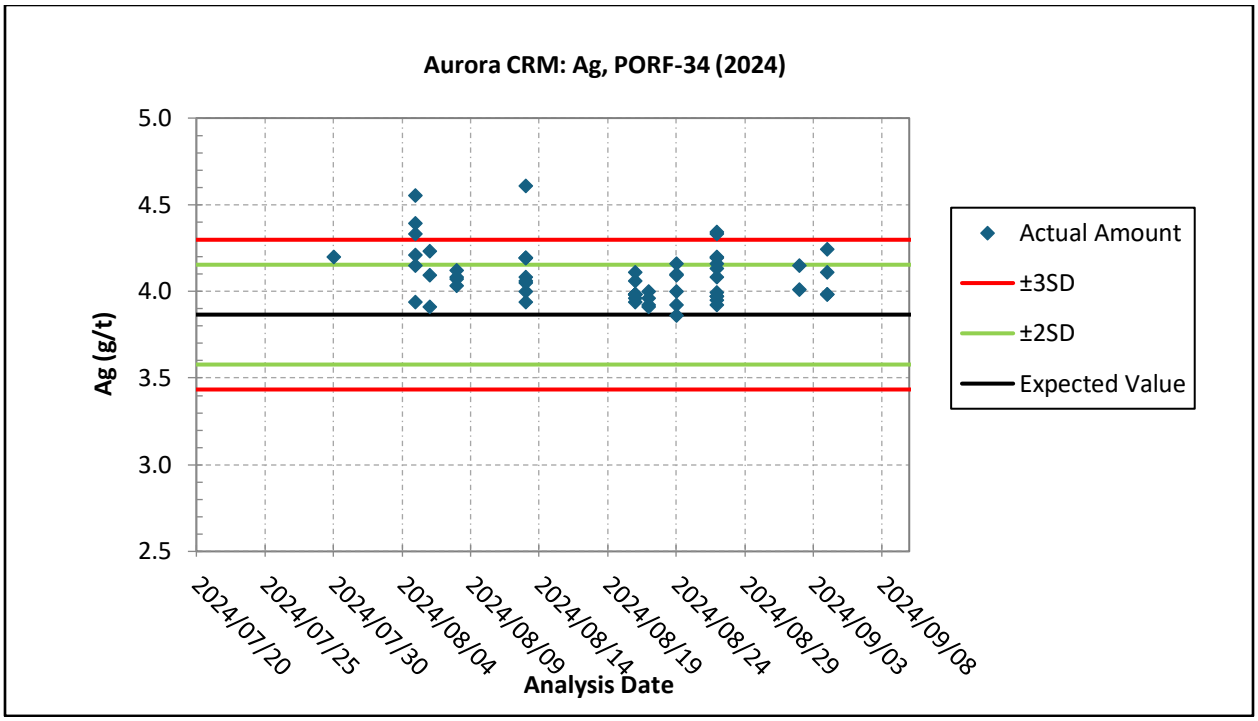
Source: GRE, 2026

Figure 12-29: PORF-34, Mo from the 2024 Drilling Campaign



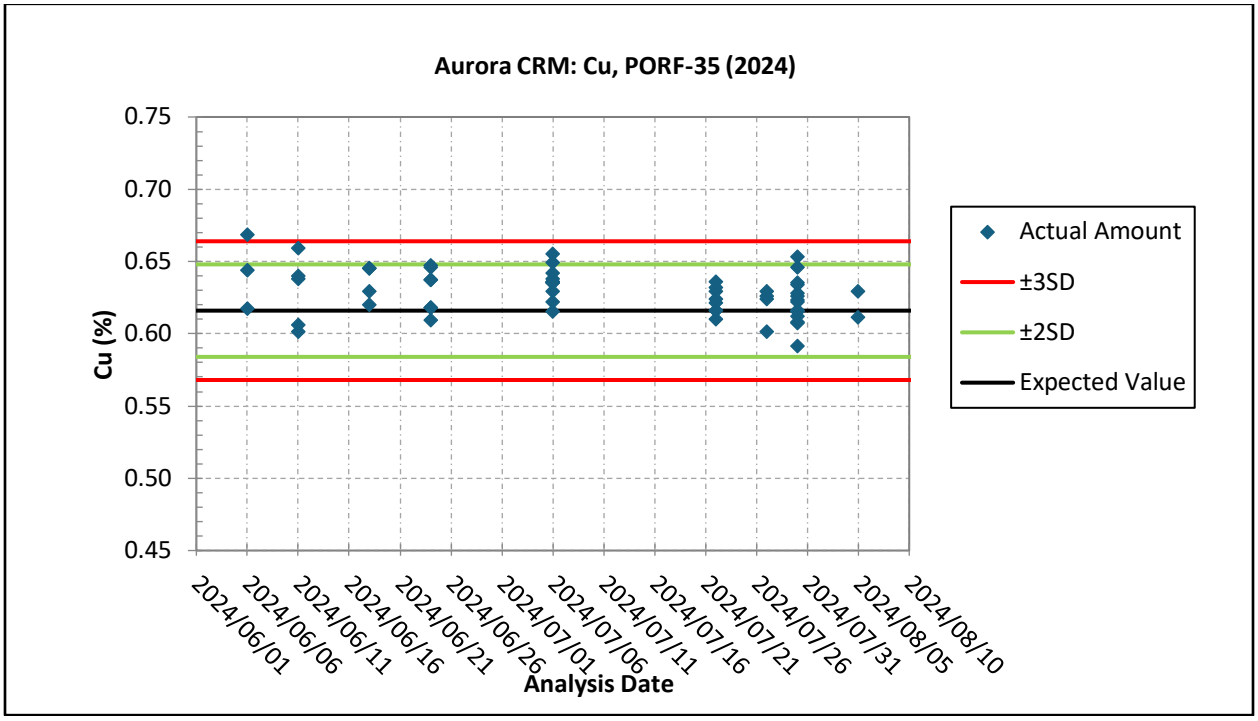
Source: GRE, 2026

Figure 12-30: PORF-34, Ag from the 2024 Drilling Campaign



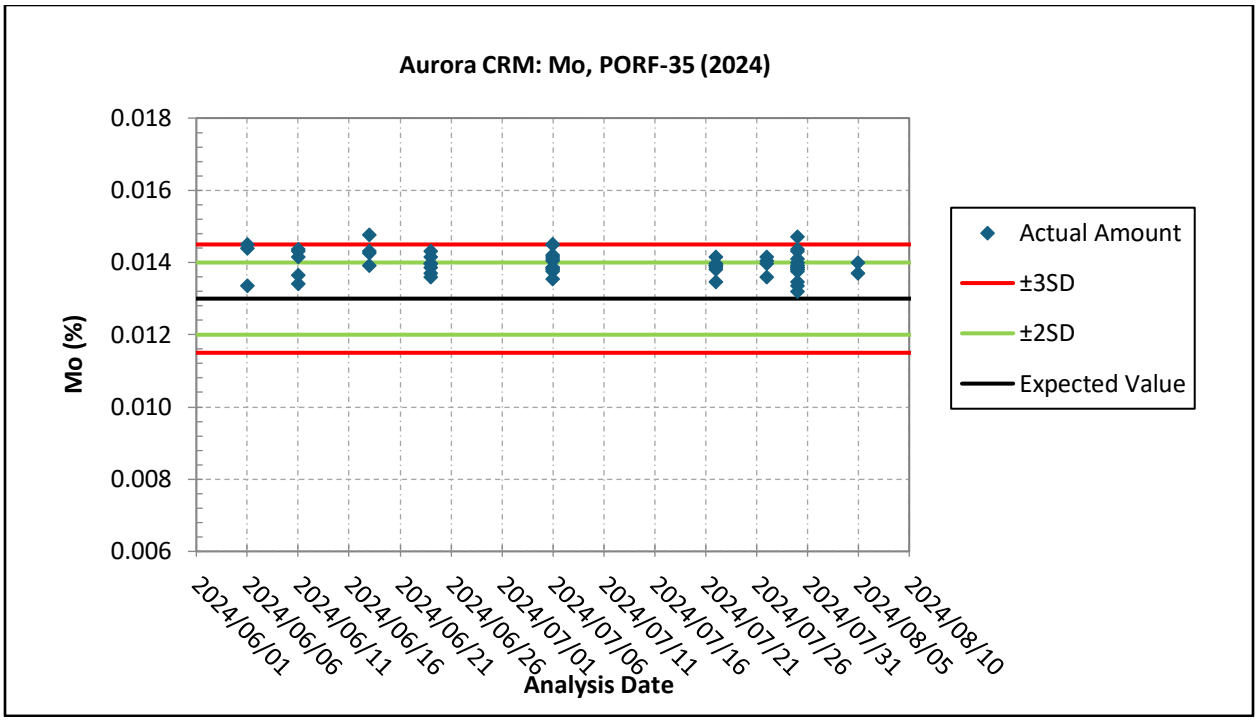
Source: GRE, 2026

Figure 12-31: PORF-35, Cu from the 2024 Drilling Campaign



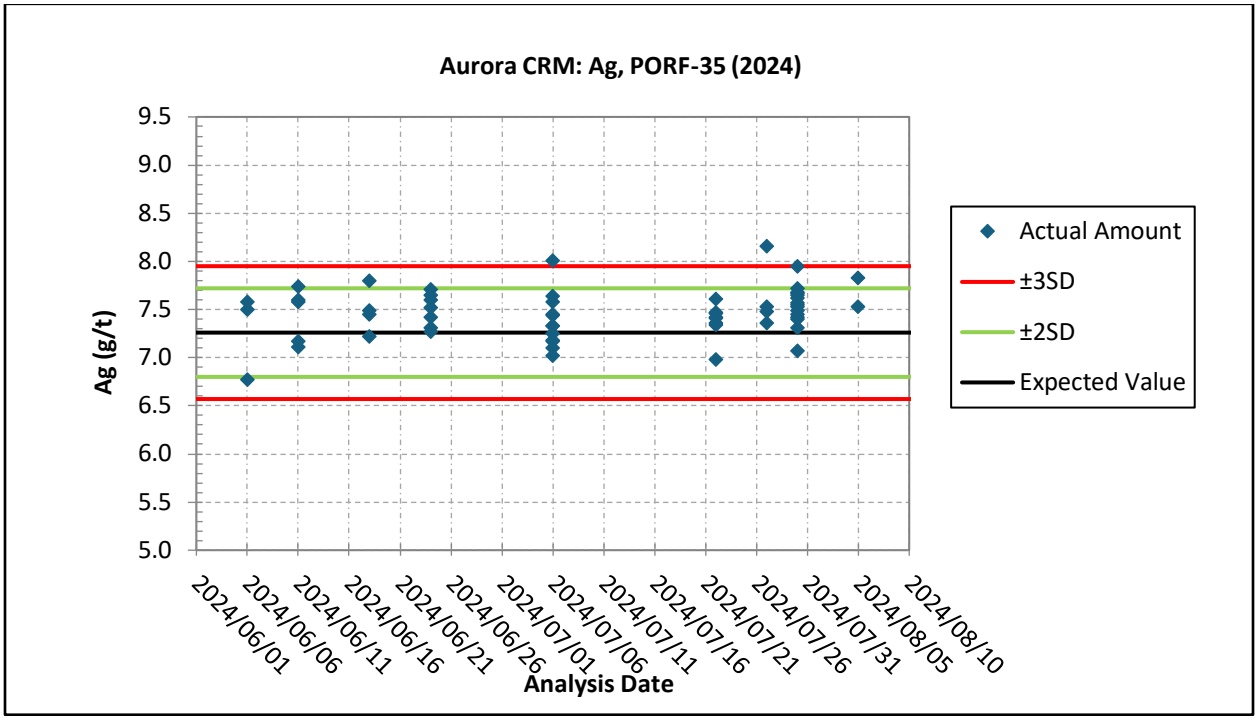
Source: GRE, 2026

Figure 12-32: PORF-35, Mo from the 2024 Drilling Campaign



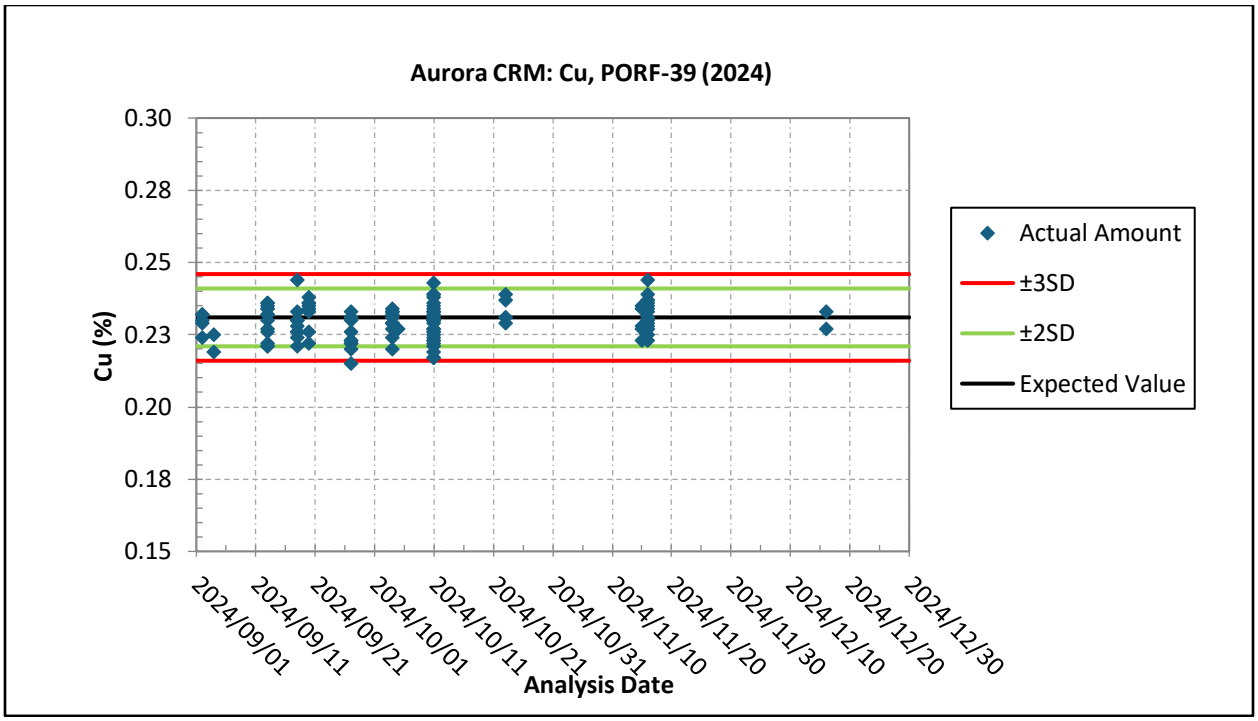
Source: GRE, 2026

Figure 12-33: PORF-35, Ag from the 2024 Drilling Campaign



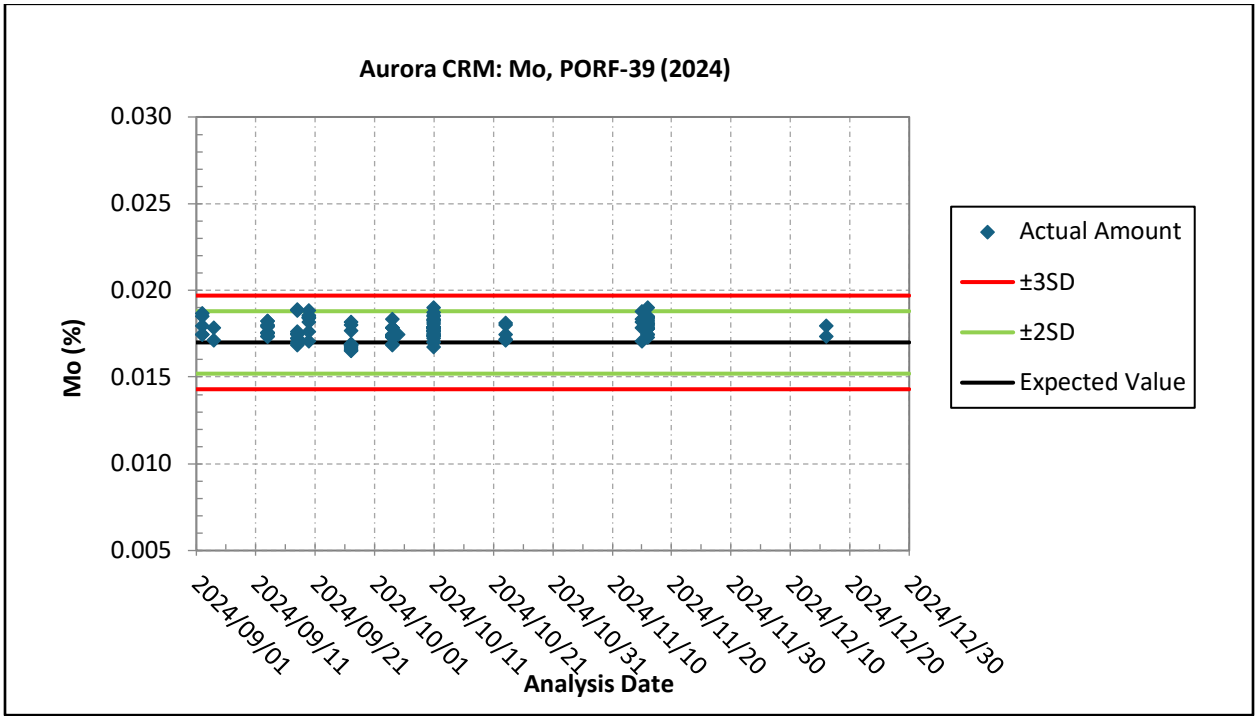
Source: GRE, 2026

Figure 12-34: PORF-39, Cu from the 2024 Drilling Campaign



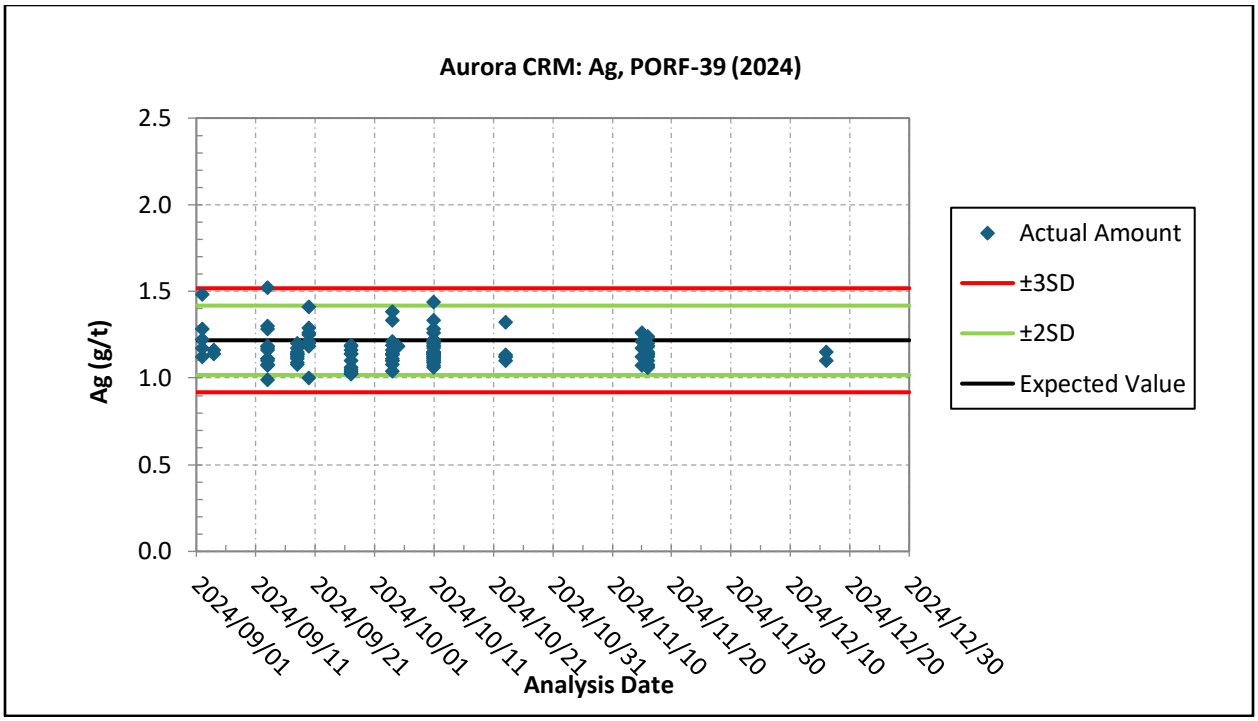
Source: GRE, 2026

Figure 12-35: PORF-39, Mo from the 2024 Drilling Campaign



Source: GRE, 2026

Figure 12-36: PORF-39, Ag from the 2024 Drilling Campaign



Source: GRE, 2026

12.4.3.3 Coarse Reject Duplicate Samples (2024)

A total of 346 coarse reject duplicate samples were submitted during the 2024 drilling campaign, representing an insertion rate of approximately 8.4 duplicate samples per 100 routine samples

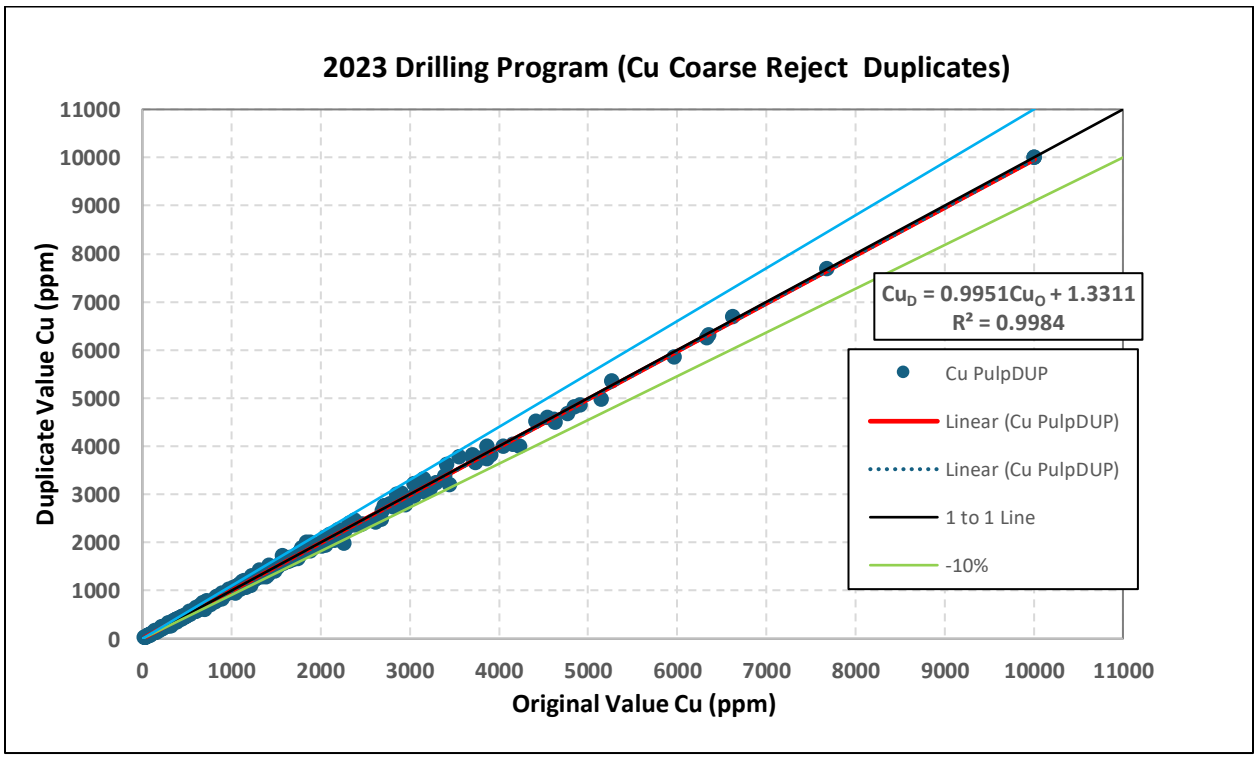
(approximately one duplicate sample per 12 routine samples). Coarse reject duplicate samples were prepared by re-splitting the crushed reject material from the original samples and were submitted to ALS for analysis.

Figure 12-37 to Figure 12-39 present Q-Q plots comparing the original and duplicate analyses for copper, molybdenum, and silver. The duplicate datasets exhibit excellent correlations, with coefficients of determination (R^2) of 0.9964 for silver, 0.9984 for copper, and 0.9966 for molybdenum. Nearly all duplicate pairs plot within the ±10% control limits and closely follow the 1:1 line. A limited number of minor deviations from the ±10% envelopes are present in the silver dataset, particularly within the intermediate-grade range, and are considered to reflect normal analytical variability rather than systematic analytical bias.

Overall, the coarse duplicate results demonstrate excellent analytical precision and reproducibility across the range of analyzed concentrations. No material analytical precision issues or systematic biases were identified, and the duplicate results indicate that the assay data generated during the 2024 drilling campaign are suitable for geological interpretation and subsequent Mineral Resource Estimation.

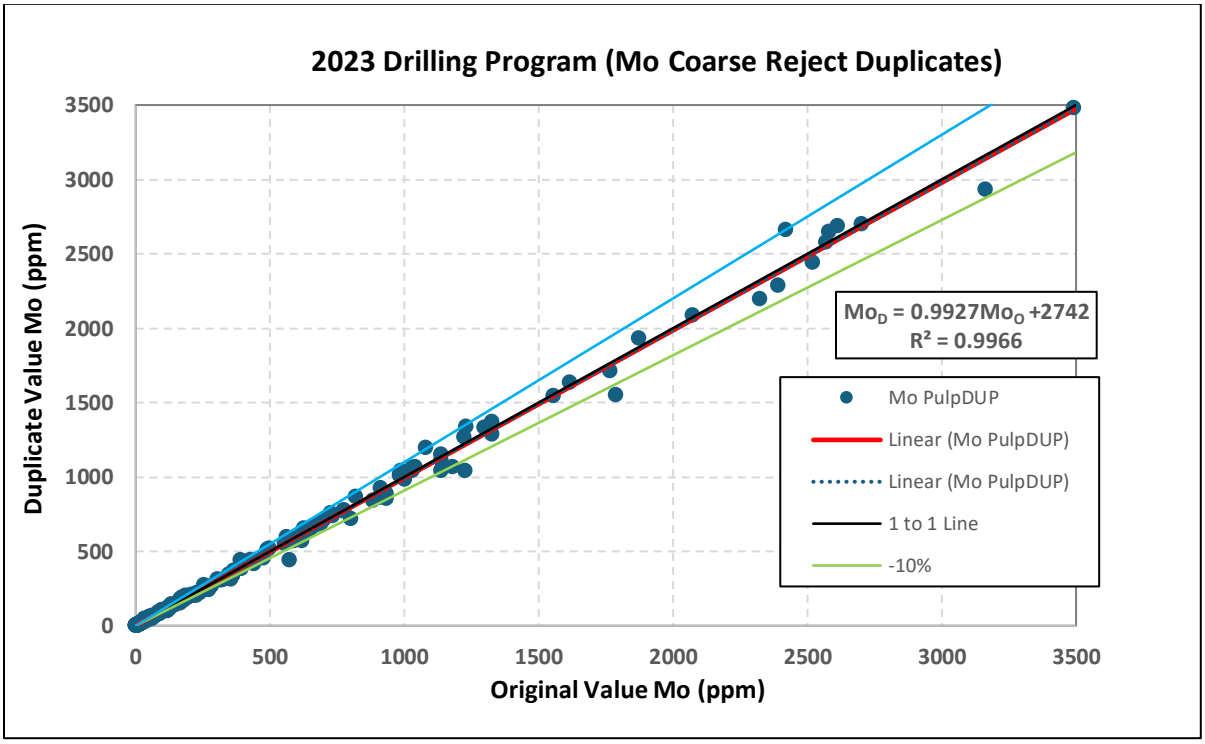
The 2024 coarse reject duplicate insertion rate exceeds the generally accepted industry guideline of approximately one duplicate sample per 20 routine samples and provides a robust assessment of laboratory analytical repeatability and precision.

Figure 12-37: Cu Coarse Reject Duplicate Samples from the 2024 Drilling Campaign



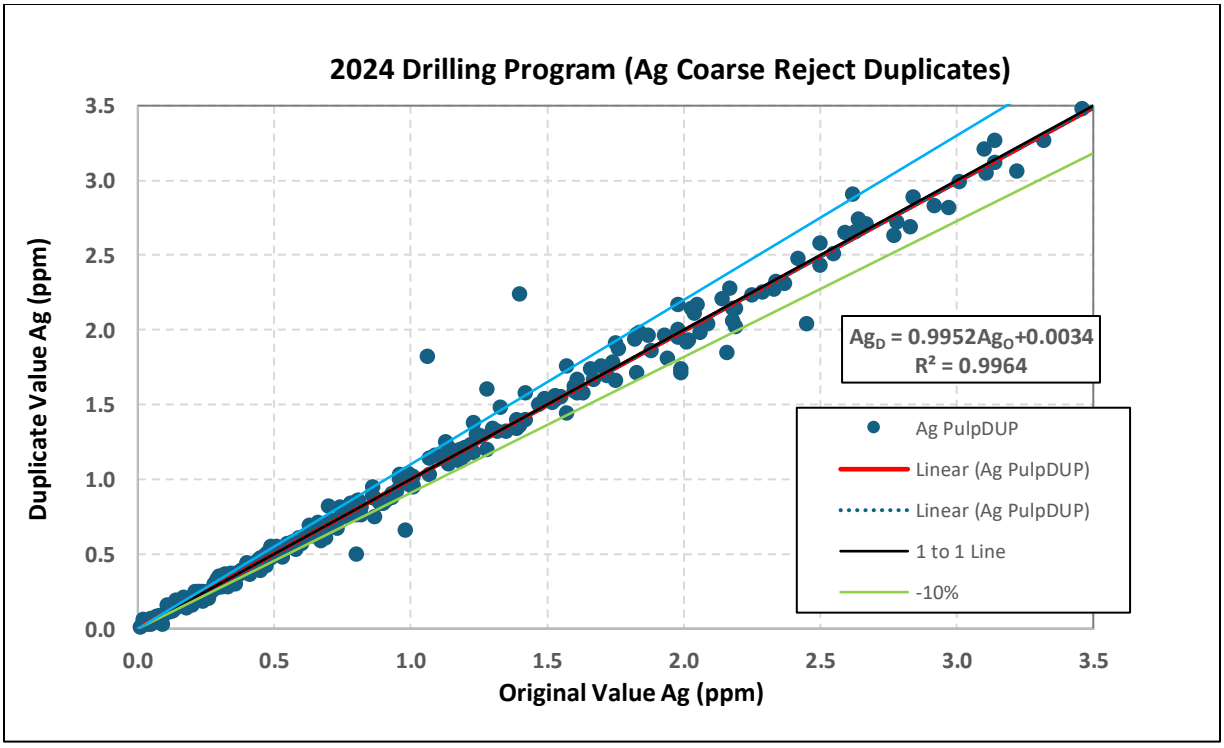
Source: GRE, 2026

Figure 12-38: Mo Coarse Reject Duplicate Samples from the 2024 Drilling Campaign



Source: GRE, 2026

Figure 12-39: Ag Coarse Reject Duplicate Samples from the 2024 Drilling Campaign



Source: GRE, 2026

12.4.4 QA/QC Review (2025)

12.4.4.1 Blank Samples (2025)

A total of 64 blank samples were inserted during the 2025 drilling campaign, representing an insertion rate of approximately 3.0 blank samples per 100 routine samples (approximately one blank sample per 33 routine samples). Coarse blank samples consisting of certified barren material were inserted into the sample stream to monitor potential contamination during sample preparation and analytical procedures. Blank sample performance was evaluated using certified values and predefined warning and failure thresholds of 5× and 10× the certified values for Cu and Mo, respectively, and 5× the analytical detection limit for Ag.

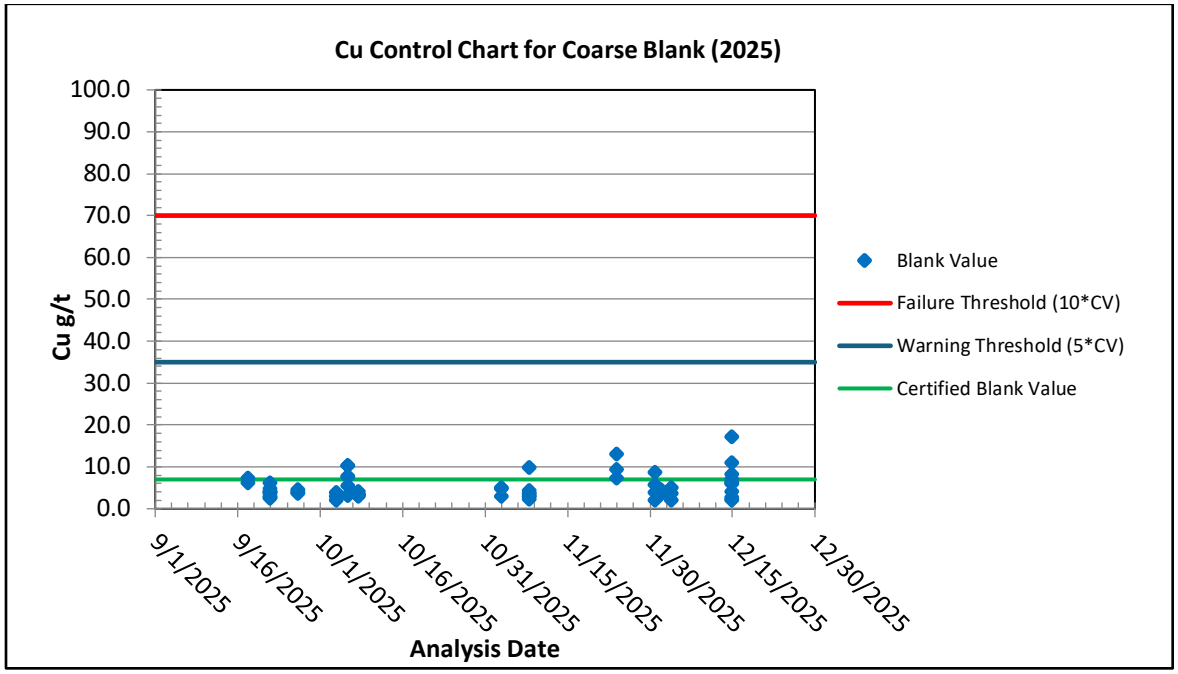
Figure 12-40 to Figure 12-42 present the control charts for Cu, Mo, and Ag blank samples. The majority of blank analyses returned values close to the certified blank values and remained well below the established warning and failure thresholds. No blank samples exceeded the failure thresholds for Cu, Mo, or Ag.

Several molybdenum blank samples approached or slightly exceeded the warning threshold of 5 ppm Mo, and a limited number of copper blank samples returned moderately elevated values relative to the certified blank value. However, these occurrences were isolated, did not exceed the failure thresholds, and do not indicate persistent contamination or systematic carryover during sample preparation or analytical procedures. Silver blank analyses remained consistently low and well below the established threshold throughout the drilling campaign.

Overall, the 2025 blank sample results indicate that sample preparation and analytical procedures were adequately controlled and that no material contamination issues were identified. The blank sample results demonstrate acceptable analytical performance and indicate that the assay data generated during the 2025 drilling campaign are suitable for geological interpretation and subsequent Mineral Resource Estimation.

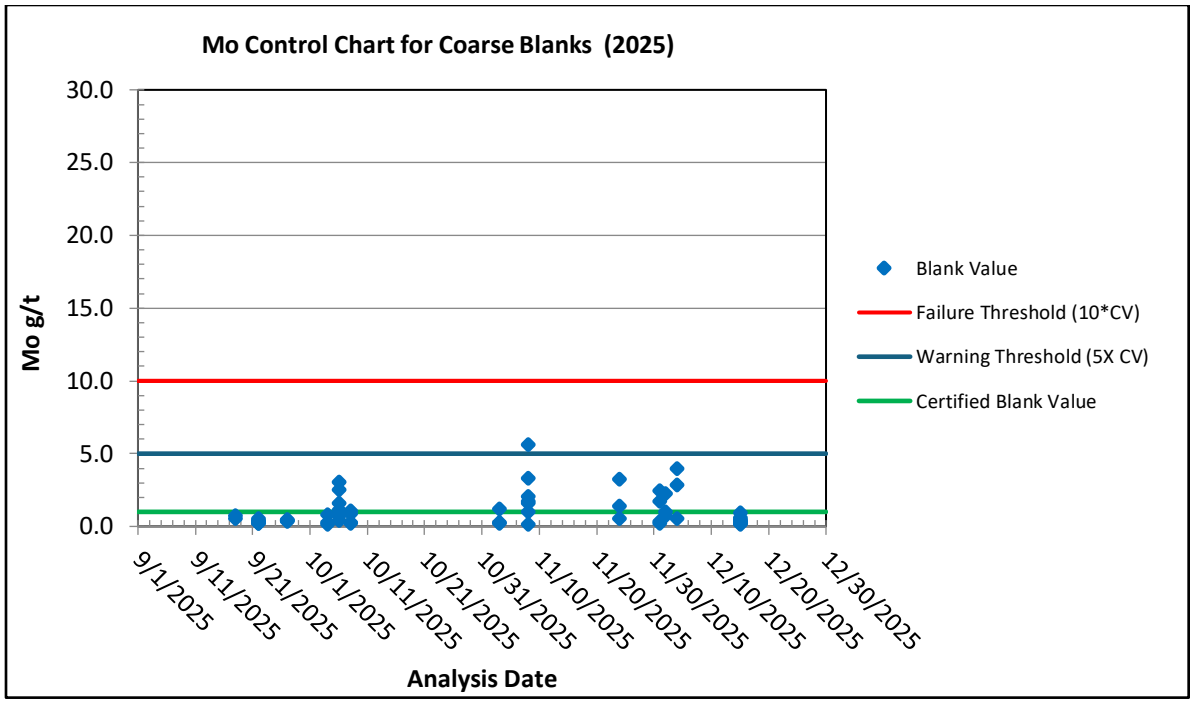
Although the 2025 blank sample insertion rate was lower than the generally accepted industry practice of approximately one blank sample per 20 routine samples, the absence of blank failures indicates that contamination was adequately monitored during the drilling campaign. Increasing the blank insertion rate in future drilling programs would provide a more robust assessment of potential contamination and analytical carryover.

Figure 12-40: 2025 Drilling Campaign, Cu Blank Results



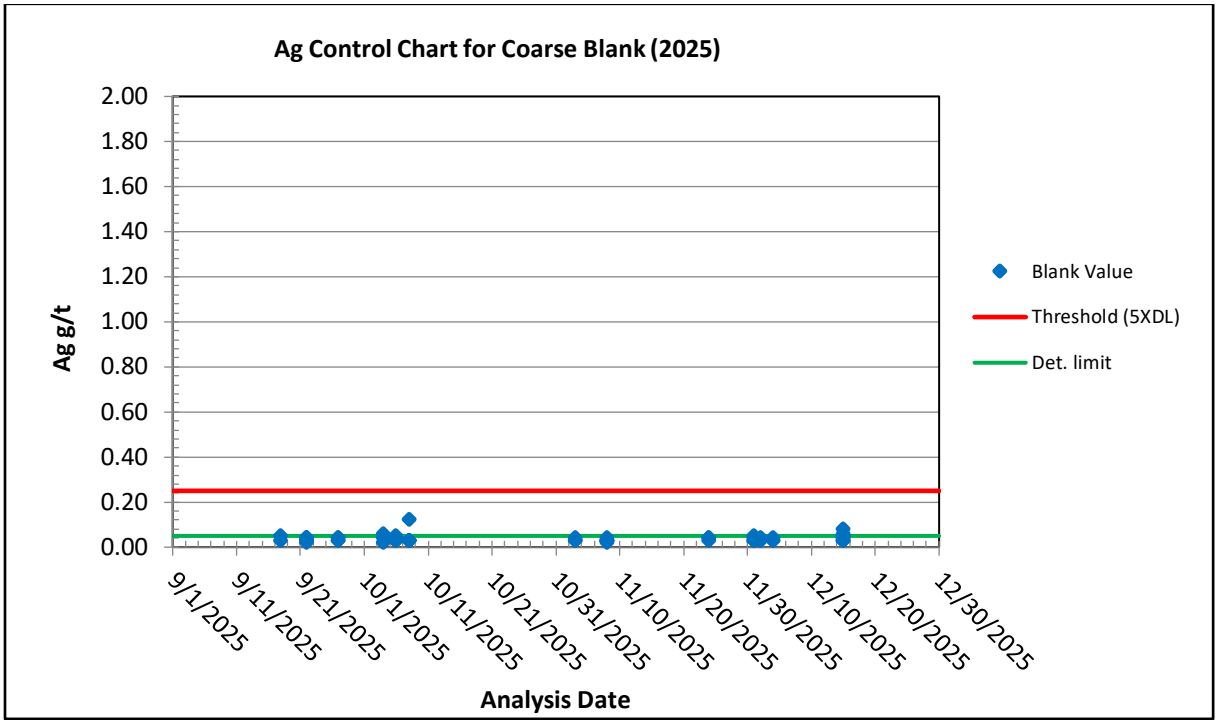
Source: GRE, 2026

Figure 12-41: 2025 Drilling Campaign, Mo Blank Results



Source: GRE, 2026

Figure 12-42: 2025 Drilling Campaign, Ag Blank Results



Source: GRE, 2026

12.4.4.2 Certified Reference Materials (2025)

A total of 119 CRM samples were inserted during the 2025 drilling campaign, representing an insertion rate of approximately 5.6 CRM samples per 100 routine samples (approximately one CRM sample per 18 routine samples). The CRM program consisted of DLP-01, PORF-34, PORF-35, PORF-39, and PORF-43. The CRM analyses were assessed by comparing the reported analytical values with the certified values and their corresponding ±2SD and ±3SD control limits.

GRE's QP independently reviewed the complete CRM dataset for the 2025 drilling campaign and prepared and evaluated control charts for copper, molybdenum, and silver analyses. The control charts are presented in Figure 12-43 to Figure 12-56.

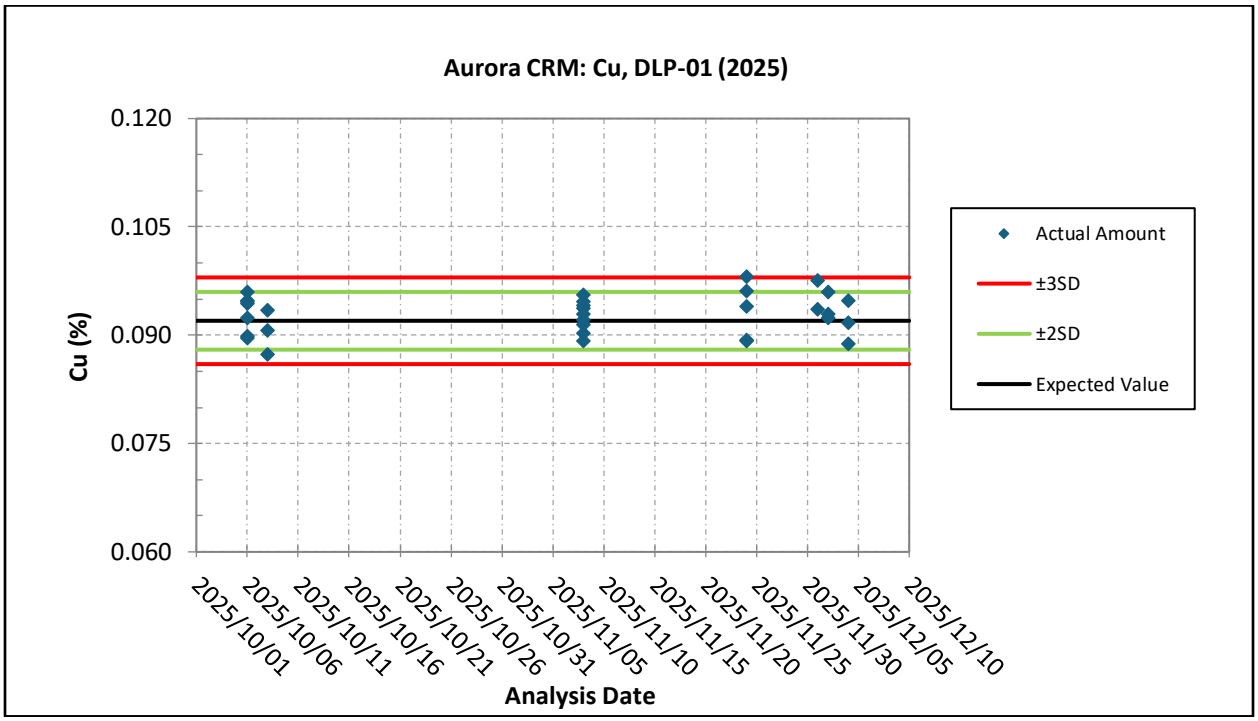
The majority of CRM analyses clustered closely around their respective certified values and fell within the established control limits, demonstrating good analytical accuracy and precision. The DLP-01, PORF-35, PORF-39, and PORF-43 CRMs exhibited acceptable to excellent analytical performance, with most analyses plotting within the ±2SD control limits and all analyses remaining within the ±3SD control limits. No evidence of persistent analytical bias, analytical drift, or systematic laboratory errors was identified.

The PORF-34 CRM also demonstrated acceptable analytical performance. Copper and molybdenum analyses were tightly clustered around their certified values and remained within the control limits. Silver analyses exhibited slightly greater dispersion, with a small number of isolated analyses approaching or marginally exceeding the +3SD control limit. However, these occurrences were sporadic, showed no temporal clustering, and do not indicate a persistent analytical issue or systematic laboratory bias.

Compared to the 2024 drilling campaign, the performance of the PORF-34 silver CRM showed a marked improvement during the 2025 drilling campaign. The analytical results were generally tightly clustered around the certified value and exhibited substantially less dispersion than observed in 2024. Accordingly, the PORF-34 CRM is considered suitable for continued use as a silver reference material in future drilling programs. Nevertheless, continued monitoring of its performance is recommended, and the inclusion of an additional high-grade silver CRM could provide further verification of analytical accuracy across the upper range of silver mineralization encountered at the Aurora Project.

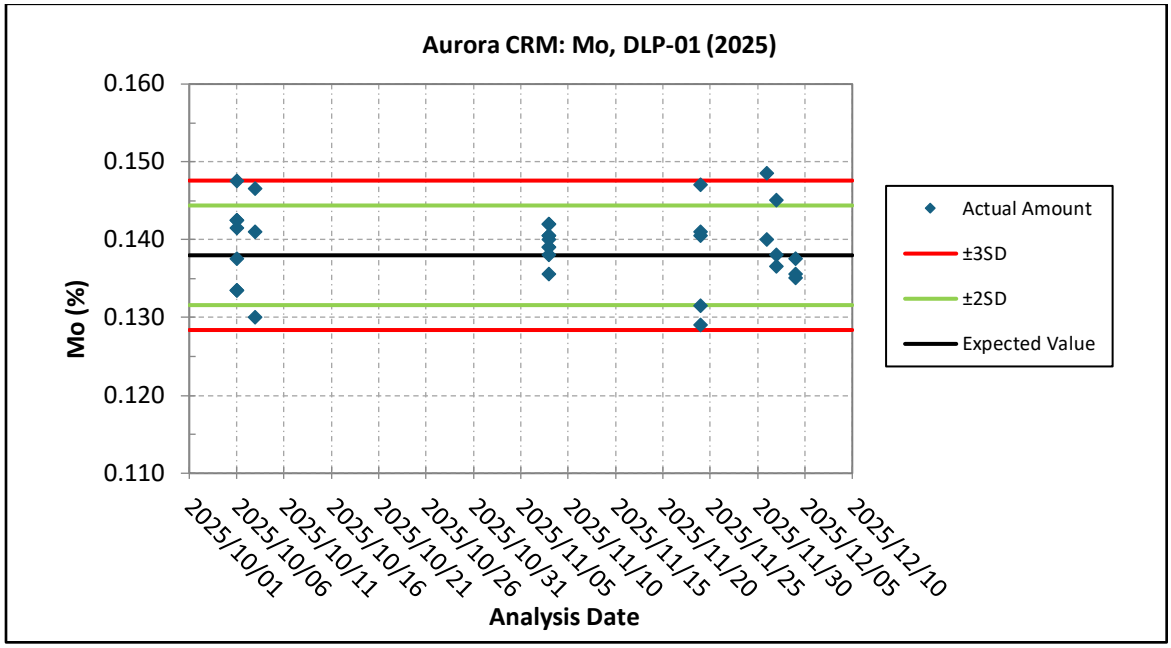
Overall, the 2025 CRM results indicate acceptable to excellent analytical performance and demonstrate that the analytical data are accurate and reliable. No material issues were identified that would adversely affect the suitability of the assay database for geological interpretation and Mineral Resource Estimation. The 2025 CRM insertion rate exceeded the generally accepted industry guideline of approximately one CRM sample per 20 routine samples and provided a robust assessment of analytical accuracy and laboratory performance.

Figure 12-43: DLP-01, Cu from the 2025 Drilling Campaign



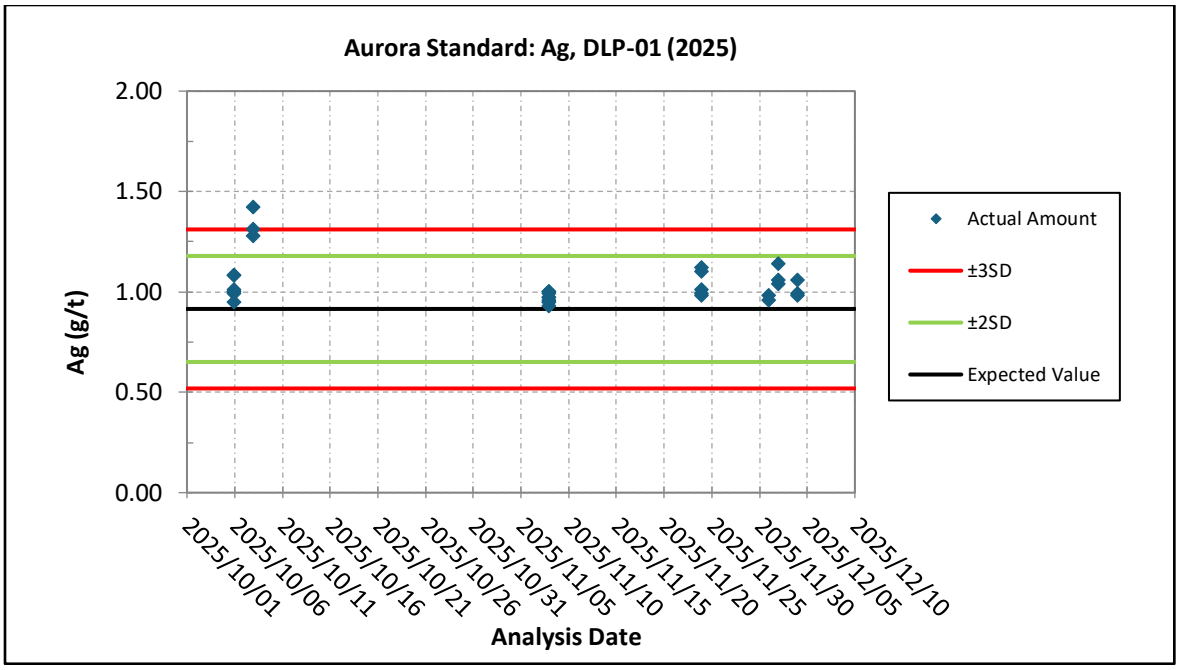
Source: GRE, 2026

Figure 12-44: DLP-01, Mo from the 2025 Drilling Campaign



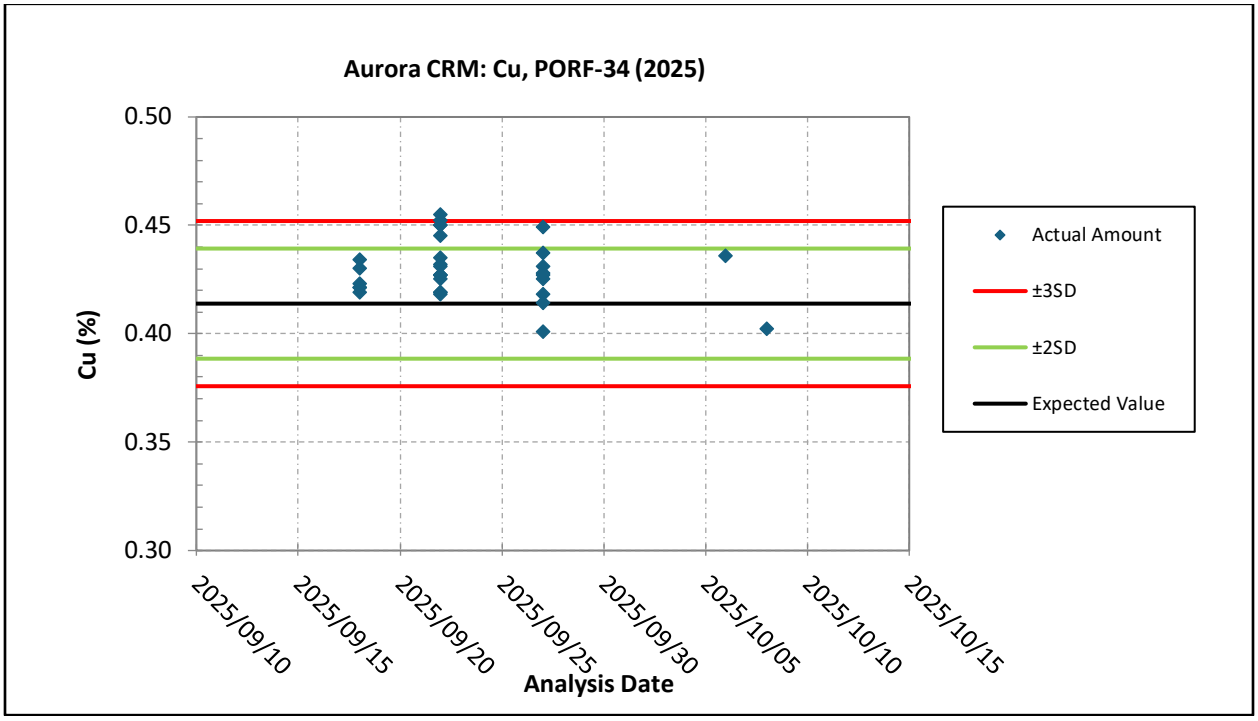
Source: GRE, 2026

Figure 12-45: DLP-01, Ag from the 2025 Drilling Campaign



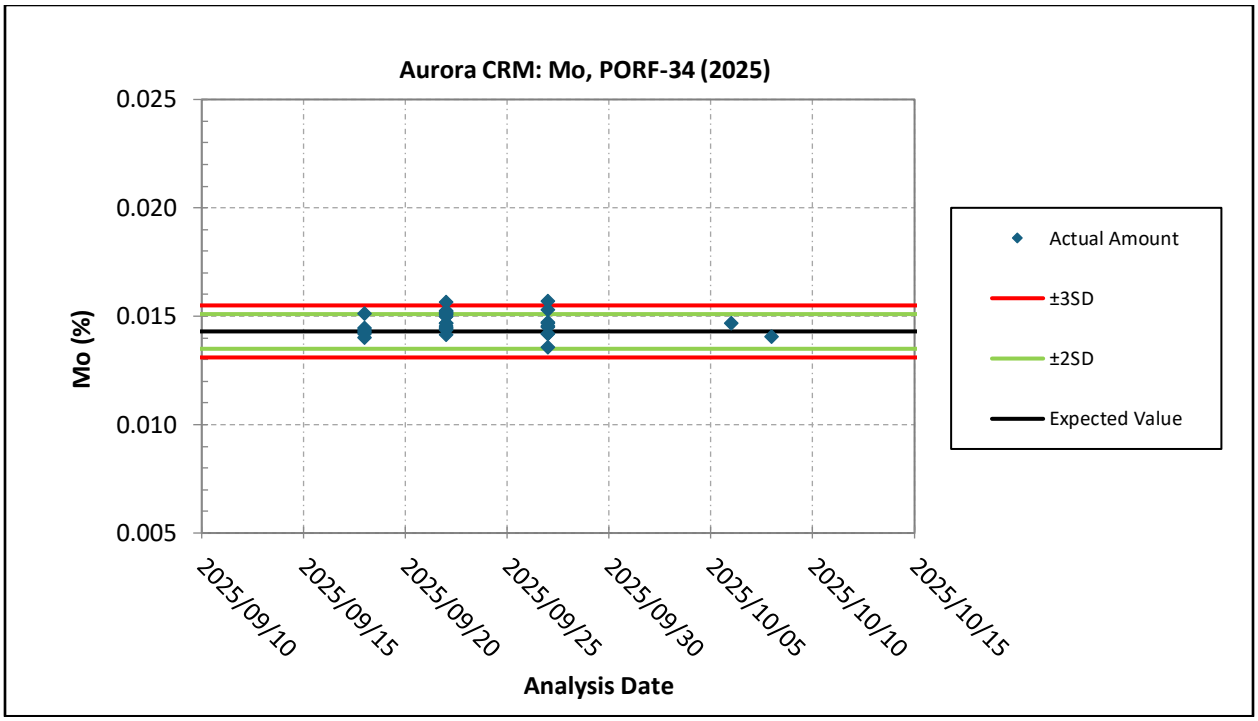
Source: GRE, 2026

Figure 12-46: PORF-34, Cu from the 2025 Drilling Campaign



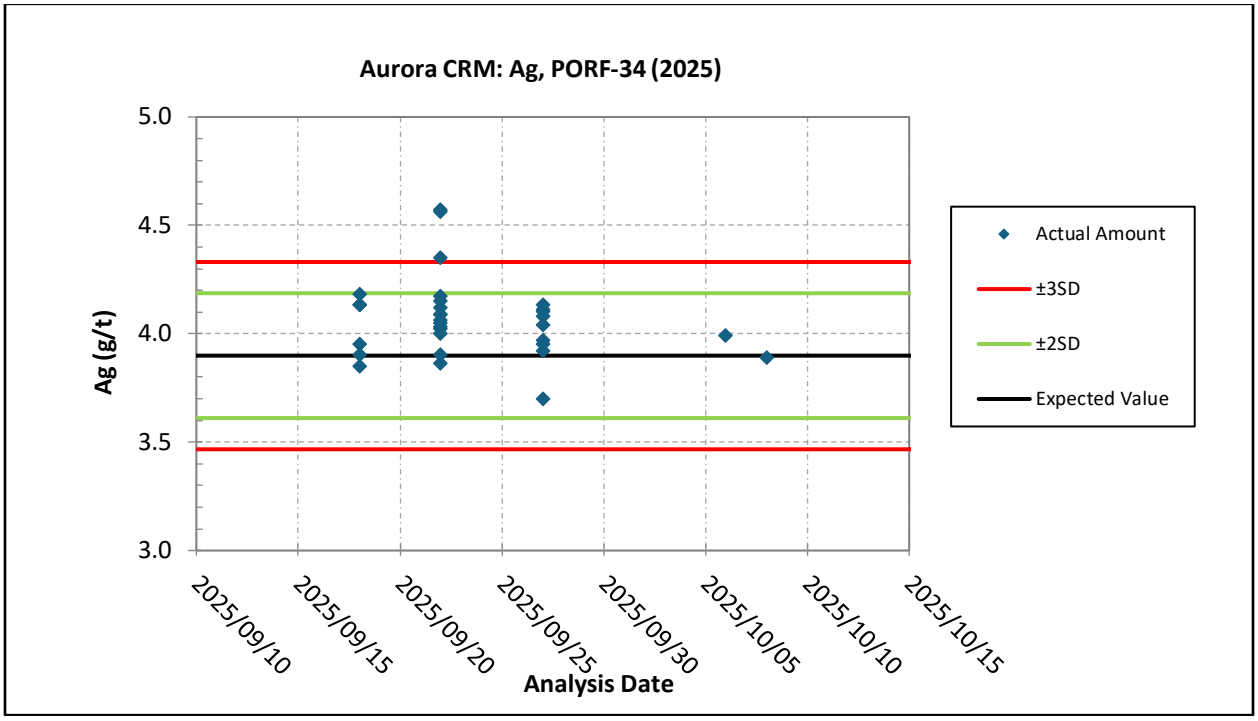
Source: GRE, 2026

Figure 12-47: PORF-34, Mo from the 2025 Drilling Campaign



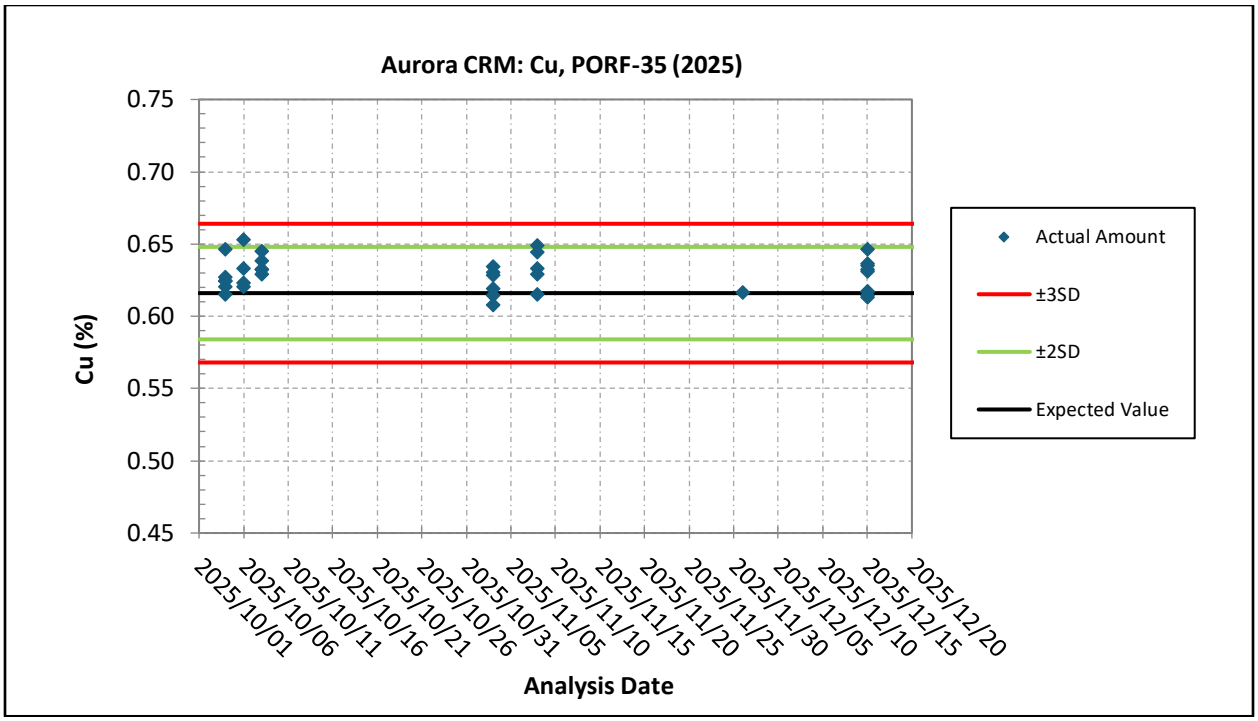
Source: GRE, 2026

Figure 12-48: PORF-34, Ag from the 2025 Drilling Campaign



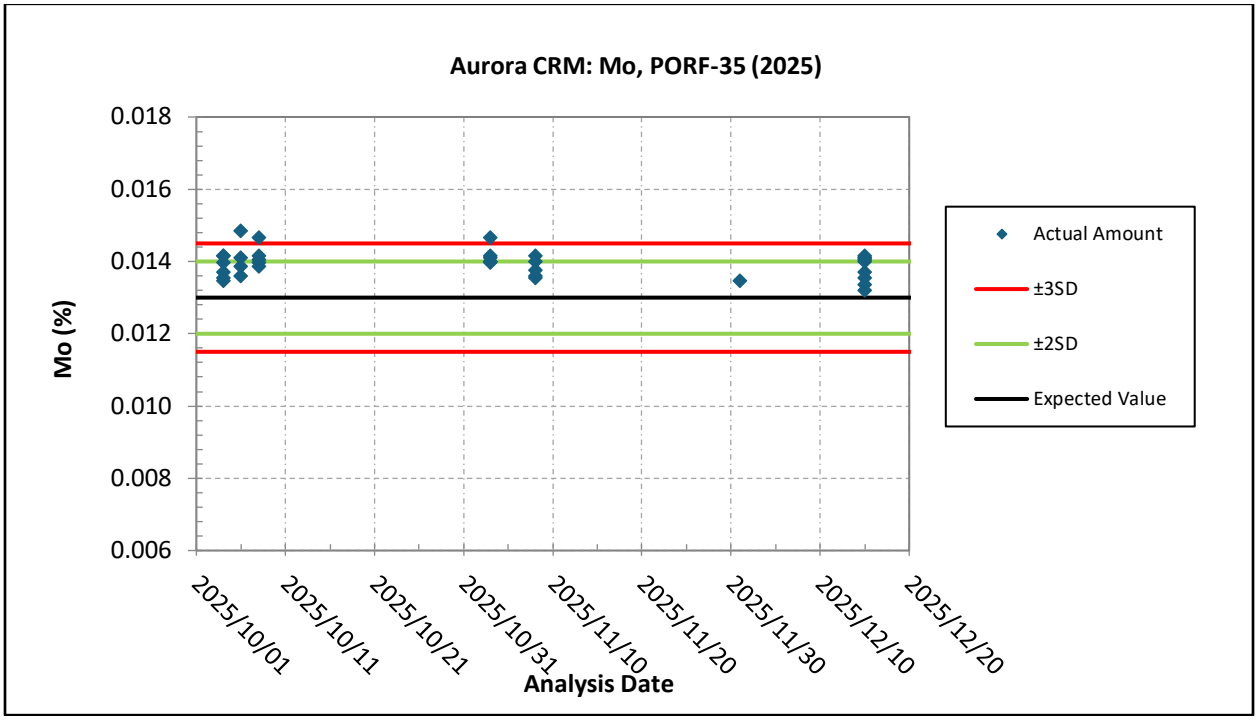
Source: GRE, 2026

Figure 12-49: PORF-35, Cu from the 2025 Drilling Campaign



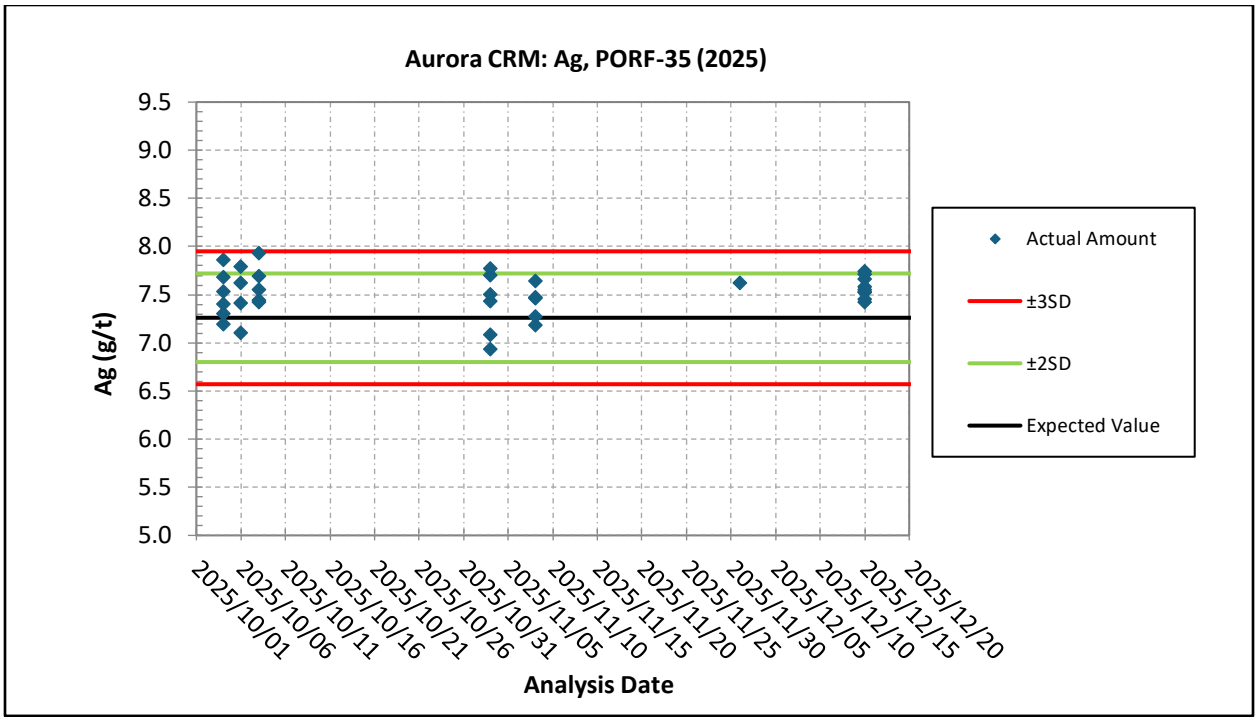
Source: GRE, 2026

Figure 12-50: PORF-35, Mo from the 2025 Drilling Campaign



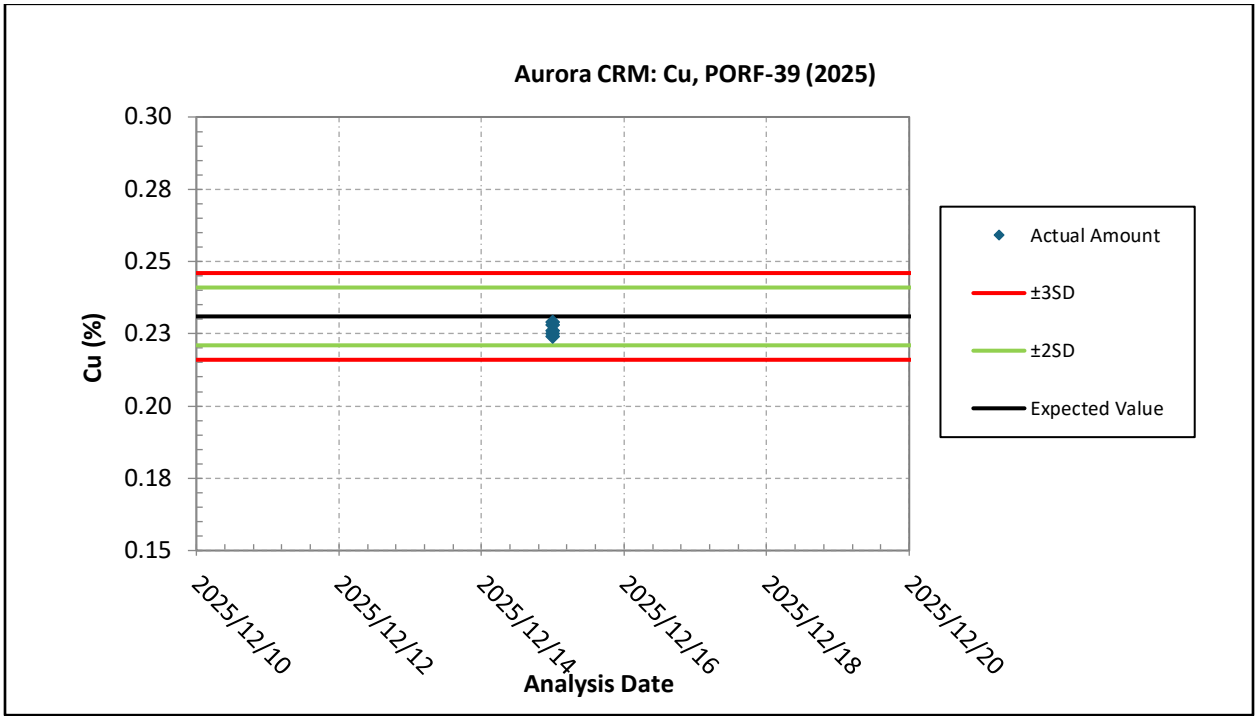
Source: GRE, 2026

Figure 12-51: PORF-35, Ag from the 2025 Drilling Campaign



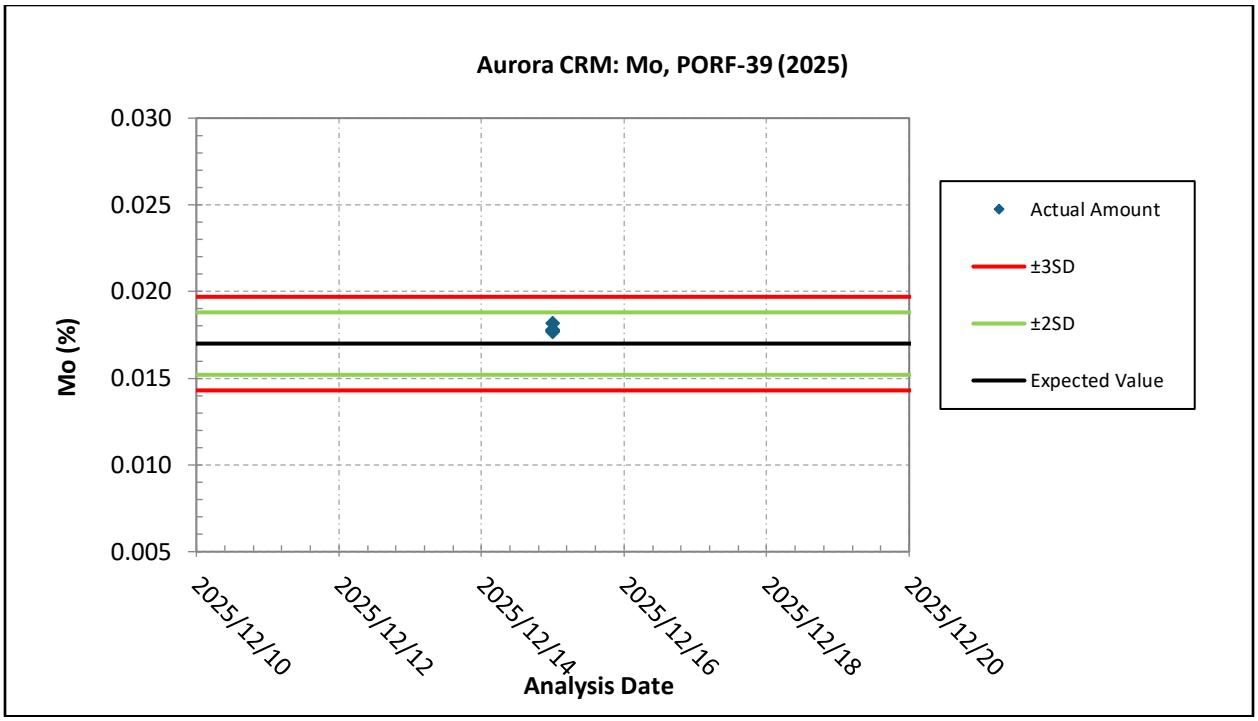
Source: GRE, 2026

Figure 12-52: PORF-39, Cu from the 2025 Drilling Campaign



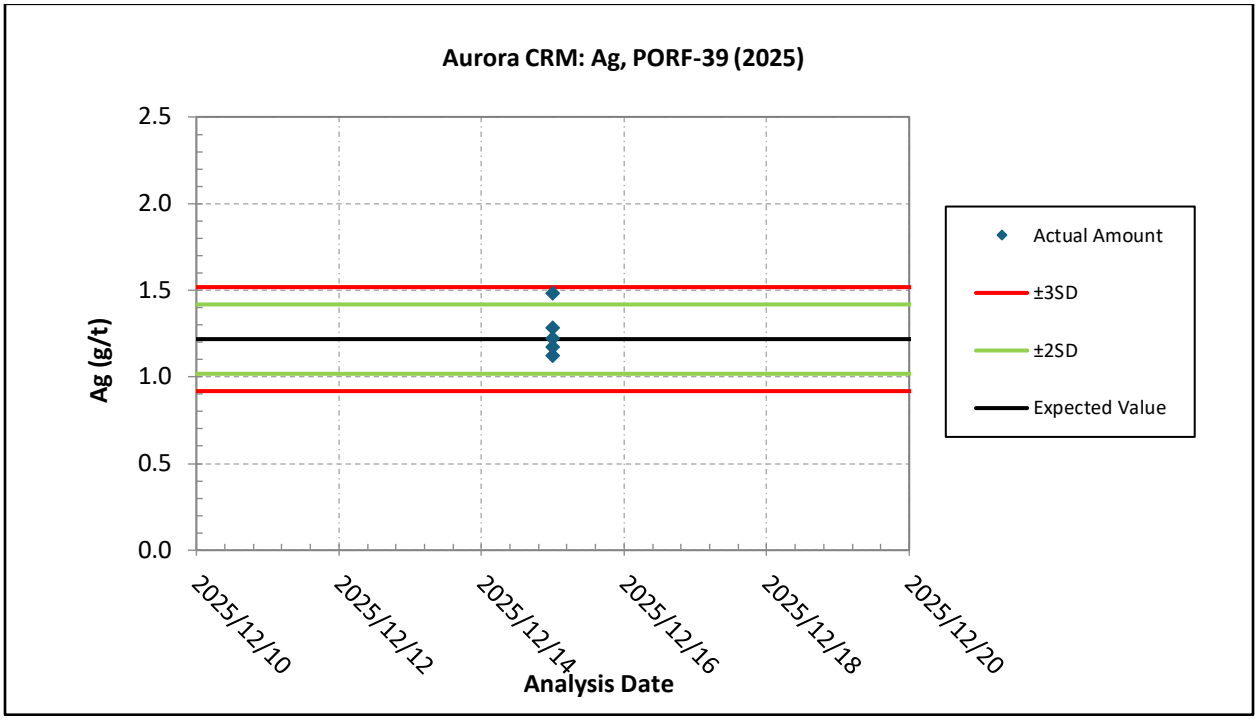
Source: GRE, 2026

Figure 12-53: PORF-39, Mo from the 2025 Drilling Campaign



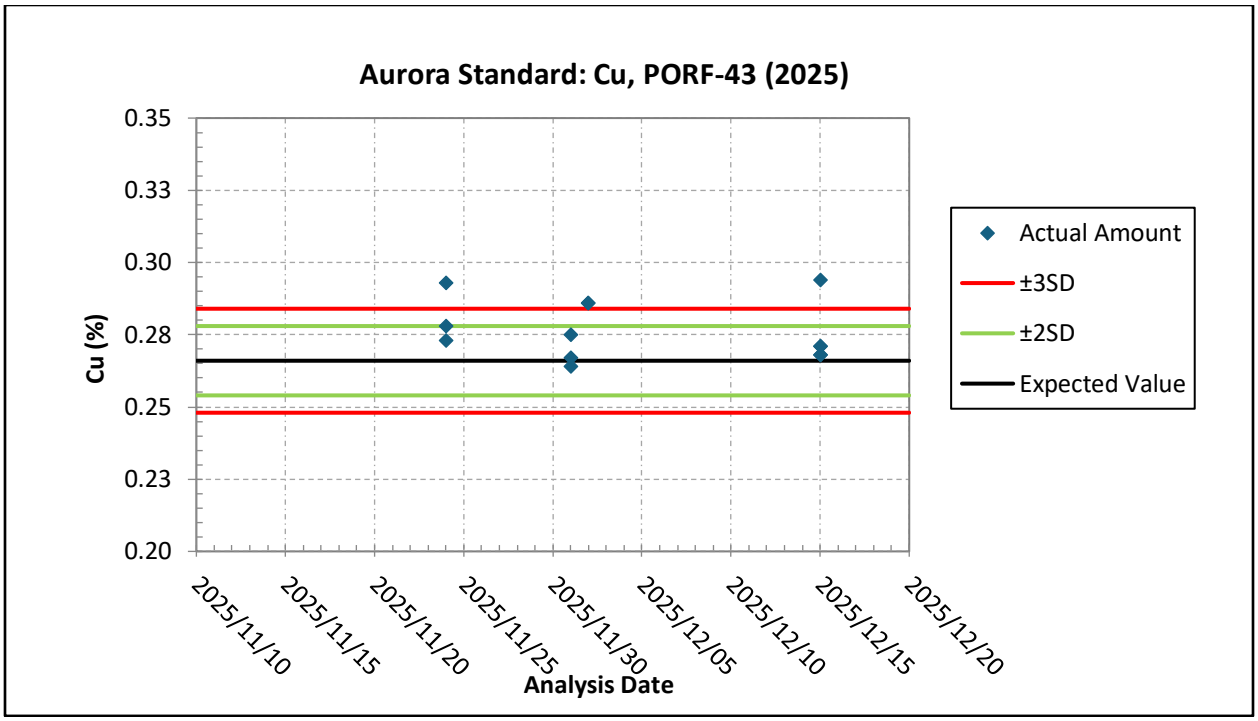
Source: GRE, 2026

Figure 12-54: PORF-39, Ag from the 2025 Drilling Campaign



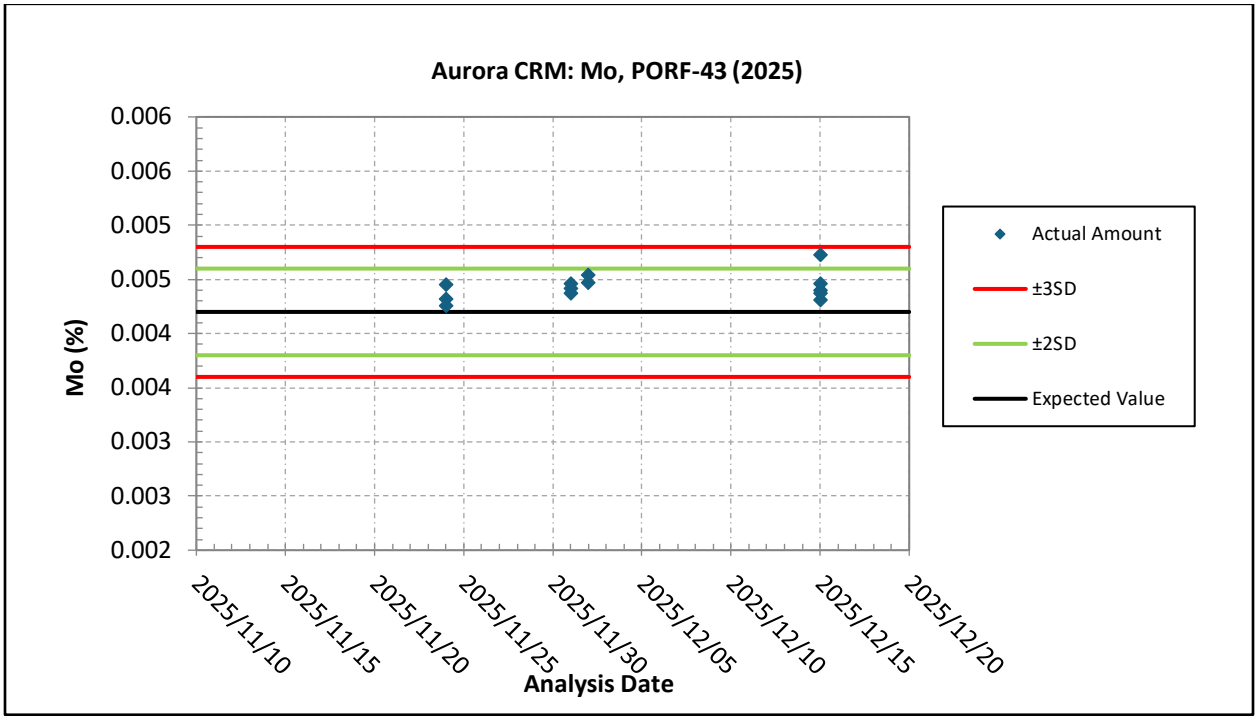
Source: GRE, 2026

Figure 12-55: PORF-43, Cu from the 2025 Drilling Campaign



Source: GRE, 2026

Figure 12-56: PORF-43, Mo from the 2025 Drilling Campaign



Source: GRE, 2026

12.4.4.3 Coarse Reject Duplicate Samples (2025)

A total of 60 coarse reject duplicate samples were submitted during the 2025 drilling campaign, representing an insertion rate of approximately 2.8 duplicate samples per 100 routine samples (approximately one duplicate sample per 36 routine samples). Coarse reject duplicate samples were prepared by re-splitting the crushed reject material from the original samples and were submitted to ALS for analysis.

Figure 12-57 to Figure 12-59 present Q-Q plots comparing the original and coarse duplicate analyses for copper, molybdenum, and silver. The duplicate datasets exhibit strong to excellent correlations, with coefficient of determination (R^2) values of 0.9957 for copper, 0.7239 for molybdenum, and 0.9861 for silver.

The copper and silver coarse reject duplicate datasets display excellent agreement between original and duplicate analyses. Nearly all duplicate pairs plot close to the 1:1 line and fall within the ±10% control limits, indicating excellent precision and reproducibility across the analyzed grade ranges. A small number of duplicate pairs exhibit slightly greater dispersion at moderate to high grades; however, these occurrences are isolated and do not indicate systematic analytical bias or material precision issues.

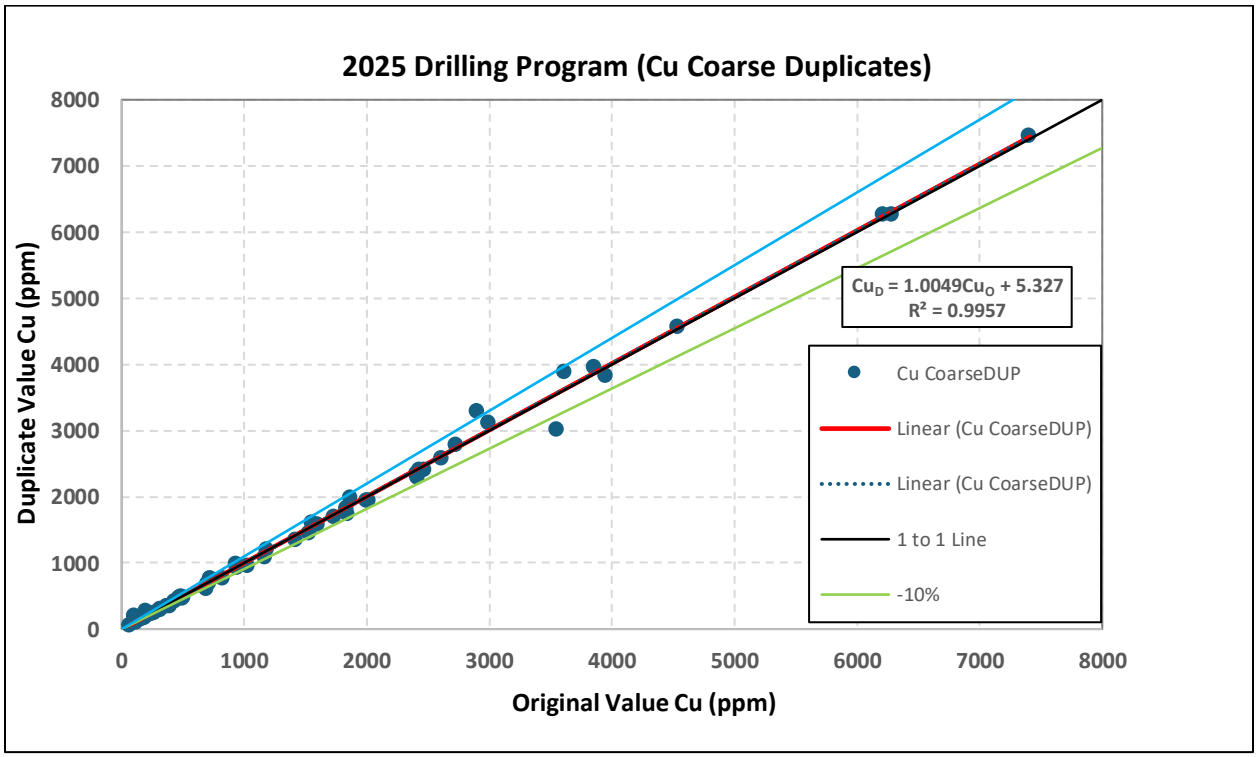
The molybdenum coarse duplicate data exhibit greater scatter than the copper and silver datasets, including several duplicate pairs that plot outside the ±10% control limits and one notable outlier at moderate molybdenum grades. The increased dispersion is interpreted to reflect the heterogeneous distribution of molybdenite within quartz veins and veinlets and the inherent sampling variability associated with coarse reject duplicates rather than analytical issues. Molybdenite commonly occurs as

localized concentrations and discontinuous accumulations along quartz veins and fractures, which can result in increased variability between duplicate aliquots prepared from the crushed reject material.

Overall, the coarse duplicate results demonstrate acceptable to excellent precision and reproducibility for the 2025 drilling campaign. No evidence of systematic analytical bias or material analytical precision issues was identified, and the duplicate results indicate that the assay data generated during the 2025 drilling campaign are suitable for geological interpretation and subsequent Mineral Resource Estimation.

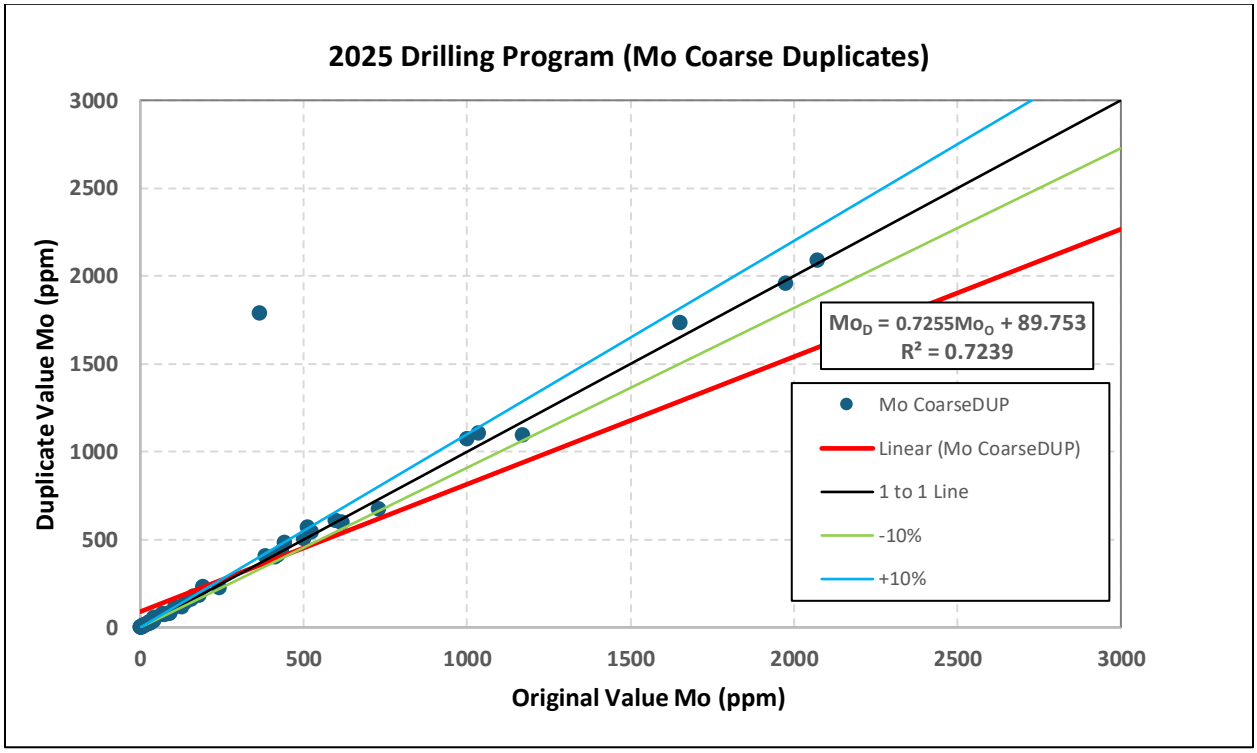
The 2025 drilling campaign's coarse reject duplicate insertion rate falls below the generally accepted industry guideline of approximately one duplicate sample per 20 routine samples. Increasing the insertion frequency of coarse reject duplicates in future drilling programs is recommended to provide a more robust assessment of sampling precision and mineralization heterogeneity.

Figure 12-57: Cu Coarse Duplicate Samples from the 2025 Drilling Campaign



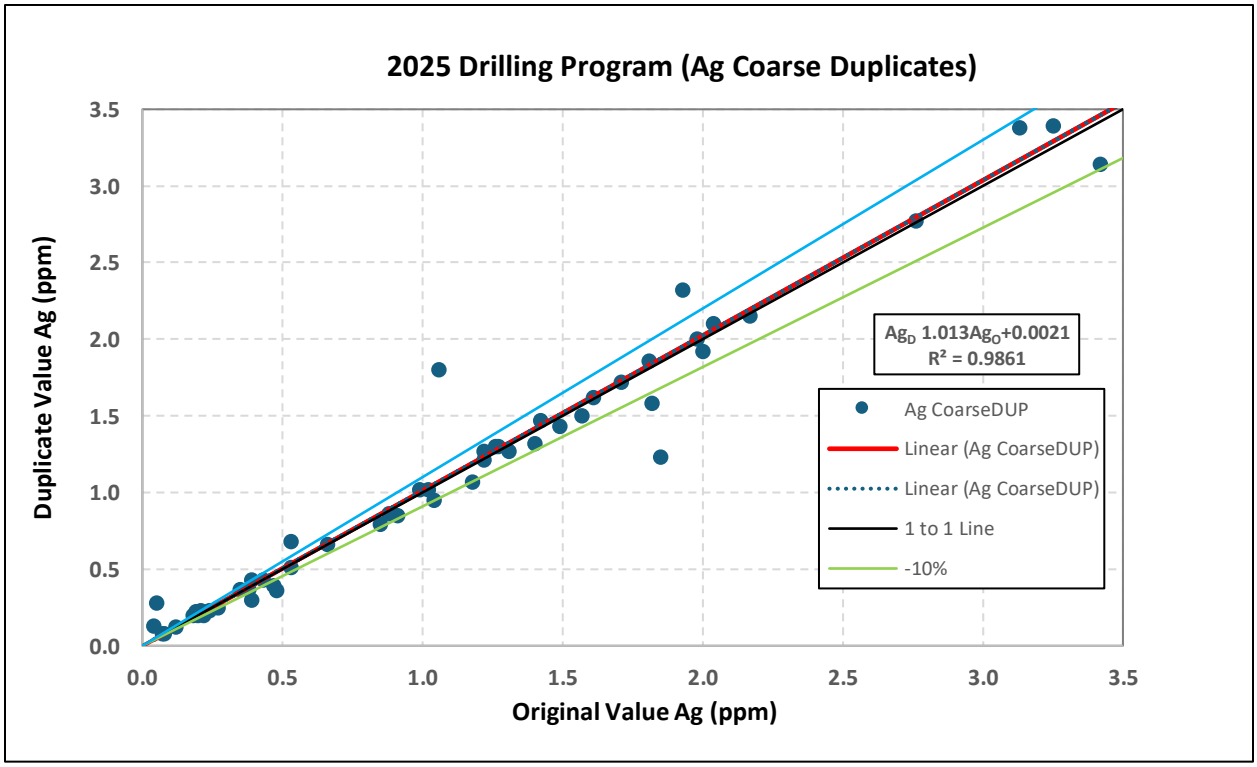
Source: GRE, 2026

Figure 12-58: Mo Coarse Duplicate Samples from the 2025 Drilling Campaign



Source: GRE, 2026

Figure 12-59: Ag Coarse Duplicate Samples from the 2025 Drilling Campaign



Source: GRE, 2026

12.5 Field Visit, Geological Data Verification, and Check Assays

GRE's QP, Dr. Hamid Samari, conducted site inspections of the Aurora Project on June 2, 2026, at the project site and on June 3, 2026, at the core storage facility in Calca, Peru, accompanied by DLP's geologist, Mr. William Espinoza. The purpose of the site visits was to review the exploration programs completed by DLP from 2022 through 2025 and to independently verify selected exploration data. During the site visits, Dr. Samari inspected the geology and mineralization exposed at the project site, reviewed geological maps and drill hole collar locations, verified selected collar coordinates and geological logging records, examined representative drill core, and collected selected pulp samples for independent check assay analyses.

12.5.1 Collar Coordinate Validation

GRE's QP used a handheld GPS (Garmin GPSMAP 64st; WGS 84, Zone 19 South) to verify the locations of selected drill hole collars. Geographic coordinates for the collar locations of 15 selected diamond drill holes completed during the 2022 through 2025 drilling programs were recorded in the field and compared with the collar coordinates contained in the project database (Table 12-1 and Photo 12-1).

Horizontal coordinate differences ranged from approximately 1.0 m to 10.3 m, with an average difference of approximately 4.0 m, while elevation differences ranged from approximately 8.6 m to 27.4 m, with an average difference of approximately 21.0 m.

The Aurora Project is situated in rugged mountainous terrain characterized by steep slopes, narrow ridges, and significant local relief (Photo 12-2). Access to many drill collar locations is limited, and several collar sites could only be approached from nearby access roads constructed along steep hillsides rather than directly at the collar locations. Consequently, the handheld GPS measurements were, in some cases, collected a short distance from the surveyed collar positions. On the steep terrain present at the Aurora Project, even small horizontal offsets can produce relatively large apparent elevation differences. In addition, handheld GPS units generally provide lower vertical accuracy than horizontal accuracy. Therefore, the combination of steep topographic gradients, differences between the GPS measurement locations and the surveyed collar positions, and the inherent limitations of handheld GPS measurements can reasonably account for the observed elevation differences. Based on the GRE QP's field observations, these differences are not considered indicative of any material errors in the collar survey database.

Furthermore, final drill hole collar locations contained in the project database were surveyed by UGCE&C, an independent surveying company, using differential GPS (DGPS) methods and documented in formal survey reports with associated calibration and quality-control documentation. In the opinion of the QP, the collar coordinates contained in the project database are acceptable and sufficiently accurate for the purposes of this Technical Report, and the observed differences do not materially affect the validity or reliability of the drill hole database.

Table 12-1: Collar Coordinates Inspections

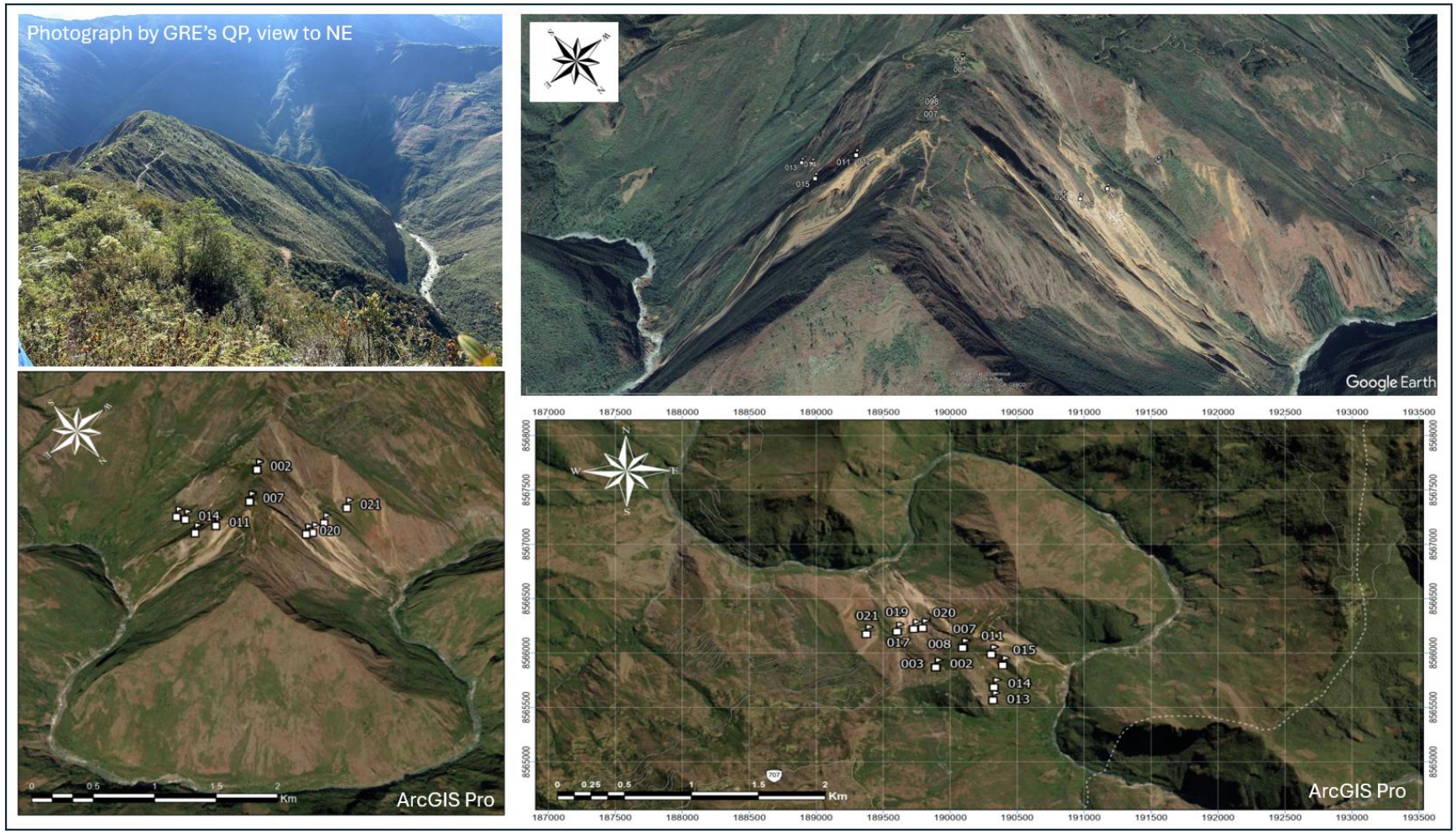
General Holes Information						Taken from Collar Excel Spreadsheet, DLP Database, WGS 84, Zone 19 South			Taken from the hand-held GPS, WGS 84, Zone 19 South by GRE's QP			Difference Distance (m)	Difference Elevation (m)
No.	Hole ID	Drill Year	Actual TD (m)	Company	GRE GPS Station #	Easting	Northing	Elevation (m)	Easting	Northing	Elevation (m)		
1	A23-004	2023	700.9	DLP	GPS20	189800.0	8566266.1	2636.5	189806.0	8566266.0	2615.0	6.03	21.46
2	A23-005	2023	693.4	DLP	GPS19	189737.4	8566256.5	2630.4	189739.0	8566258.0	2613.0	2.19	17.43
3	A23-006	2023	708.6	DLP	GPS17	189615.2	8566229.7	2635.3	189616.0	8566233.0	2617.0	3.37	18.28
4	A23-007	2023	708.3	DLP	GPS21	189379.4	8566202.1	2643.3	189386.0	8566210.0	2623.0	10.32	20.34
5	A23-013	2023	981.2	DLP	GPS03	189909.7	8565906.3	3022.2	189904.0	8565902.0	2995.0	7.14	27.17
6	A24-014	2024	641.3	DLP	GPS07	190111.8	8566087.5	2976.7	190111.0	8566088.0	2953.0	0.96	23.68
7	A24-015	2024	972.5	DLP	GPS11	190320.8	8566026.0	2851.6	190321.0	8566027.0	2836.0	1.02	15.56
8	A24-016	2024	1080.2	DLP	GPS12	190319.6	8566023.9	2851.7	190318.0	8566024.0	2833.0	1.58	18.67
9	A24-018	2024	1118.0	DLP	GPS01	189906.2	8565902.0	3022.3	189905.0	8565905.0	2995.0	3.20	27.35
10	A24-020	2024	1061.4	DLP	GPS15	190399.3	8565916.3	2757.9	190404.0	8565923.0	2738.0	8.17	19.88
11	A24-021	2024	1192.2	DLP	GPS08	190111.1	8566085.8	2976.7	190106.0	8566083.0	2952.0	5.78	24.65
12	A24-022	2024	1007.5	DLP	GPS14	190338.6	8565724.1	2729.2	190339.0	8565722.0	2712.0	2.17	17.24
13	A25-023	2025	928.7	DLP	GPS04	189907.6	8565903.9	3022.4	189904.0	8565905.0	2995.0	3.75	27.35
14	A25-026	2025	889.5	DLP	GPS02	189906.8	8565902.4	3022.4	189907.0	8565905.0	2995.0	2.63	27.36
15	A25-027	2025	679.2	DLP	GPS13	190330.9	8565601.2	2709.6	190331.0	8565603.0	2701.0	1.83	8.61

	Distance	Elevation
Maximum Difference	10.32	27.36
Minimum Difference	0.96	8.61
Average Difference	4.01	21.00

Photo 12-1: Inspection of the Collar Coordinates, 2026

Source: GRE, 2026

Photo 12-2: Topographic Setting of the Aurora Project and Drill Collars Selected for Field GPS Verification



Source: GRE, 2026

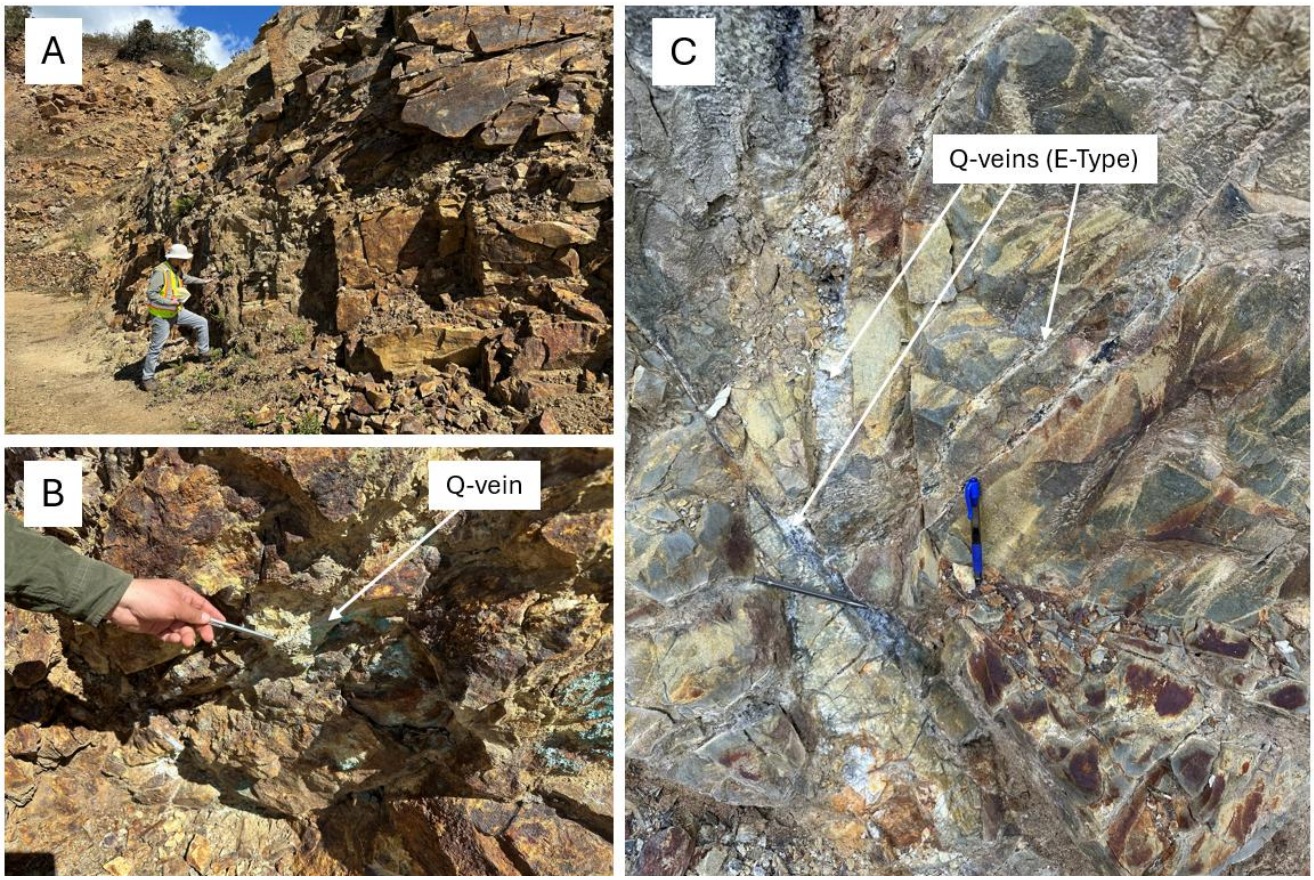
12.5.2 Geological Data Verification and Interpretation

From June 2 to 3, 2026, GRE's QP, Dr. Hamid Samari, spent one day at the project site to examine selected drill collar locations, geological formations, mineralization, and structural features exposed within the project area.

During the site visit, field observations generally confirmed the geological units and interpretations presented in the Leapfrog geological model. The lithology of exposed bedrock, alteration types, and observed structural features were generally consistent with descriptions provided in previous project reports (DLP, 2024). Dr. Samari did not observe any field evidence that would materially alter or refute the current interpretation of the local geological setting (Photo 12-3 and Photo 12-4).

Although the current geological interpretation is considered reasonable for the purposes of this Technical Report, the QP is of the opinion that the preparation of detailed, large-scale geological and structural-geology maps would further improve understanding of the lithological and structural controls on mineralization and provide a valuable framework for future drill planning and exploration activities.

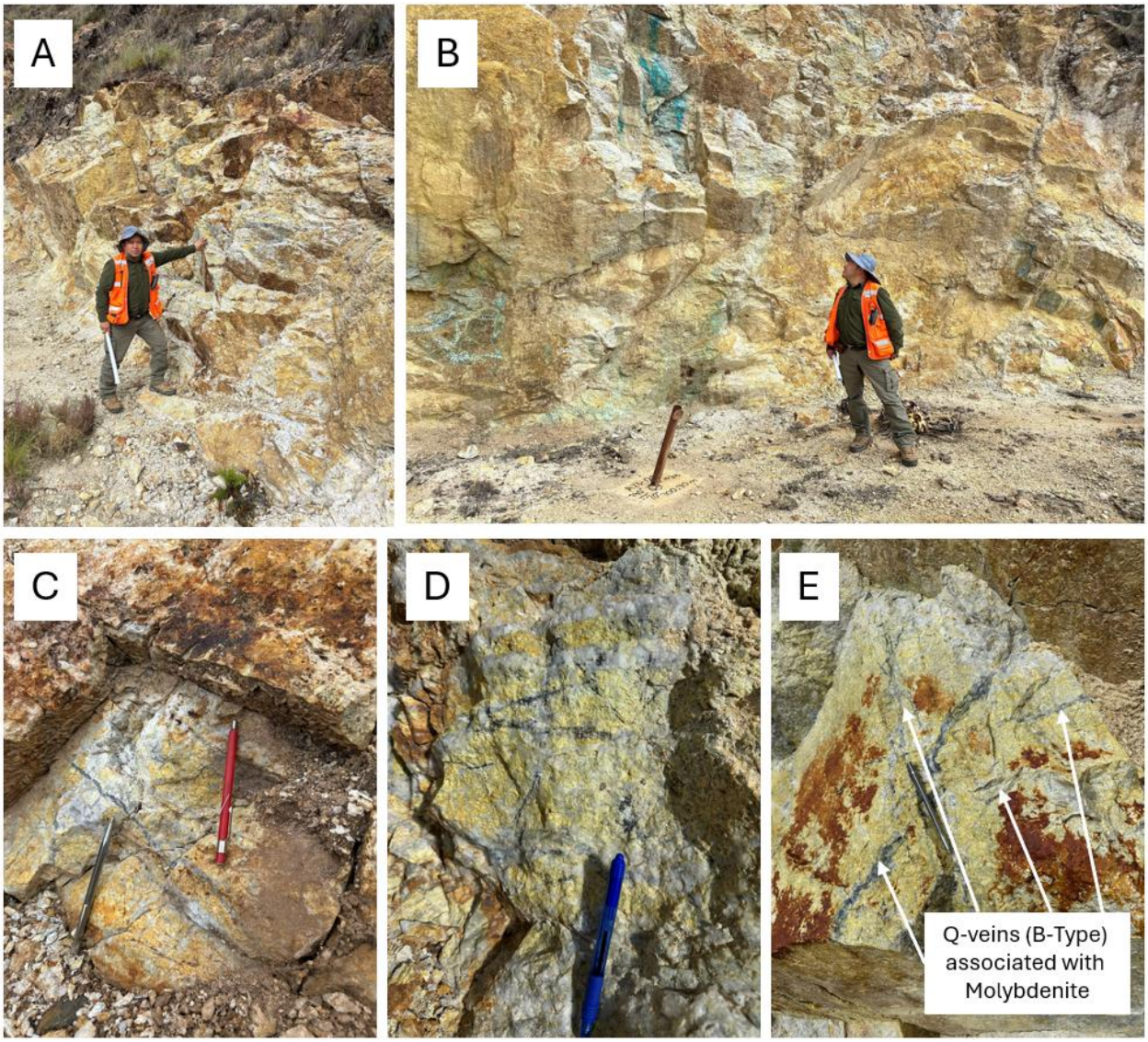
Photo 12-3: Sandia Formation



A) General view of the outcrop B) Mineralization within the Sandia Formation showing copper carbonate-hydroxide minerals and narrow quartz veins associated with copper-silver mineralization C) E-type quartz veins associated with chalcocite and pyrite mineralization

Source: GRE, 2026

Photo 12-4: Early Porphyry (MPY or DPY)



A, B) General views of the outcrop. C, D, E) Mineralization within the MPY shows molybdenite associated with B-type quartz veins. Panels D and E show iron oxide staining (limonite and hematite) with localized yellow coatings interpreted in the field as ferrimolybdite.

Source: GRE, 2026

12.5.3 Geological Logging Accuracy

During the site visit, GRE's QP, Dr. Hamid Samari, spent one full day at DLP's core storage facility in Calca, Peru, inspecting representative drill core from the 2022 through 2025 drilling programs (Photo 12-5).

Dr. Samari compared selected core intervals stored in core boxes at the Calca facility with the corresponding geological logs contained in DLP's database. A total of approximately 333 m of drill core from 18 diamond drill holes completed during the 2022 through 2025 drilling campaigns was visually inspected. The review included verification of lithology, alteration, mineralization, and significant structural features recorded in the geological logs. No material discrepancies were identified, and the

observed core characteristics were generally consistent with the geological information recorded in the DLP database (see Table 12-2 and Photo 12-6 through Photo 12-8).

Photo 12-5: Core Storage in DLP's Facility in Calca, Peru

Source: GRE, 2026

Table 12-2: Visual Inspection of Core Sample Intervals

Drilling Date	Hole ID	From (m)	To (m)	Sample Length (m)	Lithology	Mineralization
2022	A22-001	22.5	24.5	2.1	BX	Enriched
		24.5	26.3	1.8	BX	Enriched
		26.3	27.5	1.2	BX	Enriched
		27.5	29.2	1.7	BX	Enriched
		29.2	31.2	2.1	BX	Enriched
		61.3	64.0	2.7	HFL	Enriched
		64.0	66.4	2.4	HFL	Enriched
		66.4	68.5	2.1	HFL	Enriched
		108.7	110.7	2.0	HFL	Enriched
		110.7	112.7	2.0	HFL	Enriched
		112.7	114.7	2.0	HFL	Enriched
		114.7	116.7	2.0	HFL	Enriched
		116.7	118.7	2.0	HFL	Enriched
		118.7	120.7	2.0	HFL	Enriched
		120.7	122.6	2.0	HFL	Enriched
		122.6	124.3	1.7	HFL	Enriched
	A22-002	338.0	340.0	2.0	HFL	Primary
		340.0	342.0	2.0	HFL	Primary
		342.0	344.0	2.0	HFL	Primary
		344.0	346.0	2.0	HFL	Primary
		346.0	348.0	2.0	HFL	Primary
	A22-003	256.0	258.0	2.0	HFL	Enriched

Drilling Date	Hole ID	From (m)	To (m)	Sample Length (m)	Lithology	Mineralization
		258.0	260.0	2.0	HFL	Enriched
		260.0	262.0	2.0	HFL	Enriched
		262.0	264.5	2.5	HFL	Enriched
		264.5	267.0	2.5	HFL	Enriched
		267.0	269.5	2.5	HFL	Enriched
		269.5	271.6	2.1	BX	Enriched
		271.6	274.0	2.4	HFL	Enriched
		274.0	276.0	2.0	HFL	Enriched
		276.0	278.0	2.0	HFL	Enriched
		278.0	280.0	2.0	HFL	Enriched
		280.0	282.0	2.0	HFL	Enriched
		282.0	284.0	2.0	HFL	Enriched
		284.0	286.0	2.0	HFL	Enriched
		286.0	288.0	2.0	HFL	Enriched
		288.0	290.0	2.0	HFL	Enriched
2023	A23-006	99.0	101.0	2.0	MPY	Enriched
		101.0	103.0	2.0	MPY	Enriched
		103.0	105.0	2.0	MPY	Enriched
		105.0	107.0	2.0	MPY	Enriched
		107.0	109.0	2.0	MPY	Enriched
		109.0	111.0	2.0	MPY	Enriched
		111.0	113.0	2.0	MPY	Enriched
		113.0	115.0	2.0	MPY	Enriched
		115.0	117.0	2.0	MPY	Enriched
		117.0	119.0	2.0	MPY	Enriched
	A23-007	274.0	276.0	2.0	HFL	Primary
		276.0	278.0	2.0	HFL	Primary
		278.0	280.0	2.0	HFL	Primary
		280.0	282.0	2.0	HFL	Primary
		282.0	284.0	2.0	HFL	Primary
		284.0	286.0	2.0	HFL	Primary
		286.0	288.0	2.0	HFL	Primary
		288.0	290.0	2.0	HFL	Primary
		290.0	292.0	2.0	HFL	Primary
		292.0	294.0	2.0	HFL	Primary
	A23-008	177.0	179.0	2.0	HFL	Enriched
		179.0	181.0	2.0	HFL	Enriched
		181.0	183.5	2.5	HFL	Enriched
		183.5	186.0	2.5	HFL	Primary
		186.0	188.6	2.6	HFL	Primary
		188.6	191.0	2.5	MPY	Primary
		191.0	193.0	2.0	MPY	Primary
	A23-009	377.0	379.0	2.0	CPY	Primary
		379.0	381.0	2.0	CPY	Primary
		381.0	383.0	2.0	CPY	Primary
		383.0	385.0	2.0	CPY	Primary

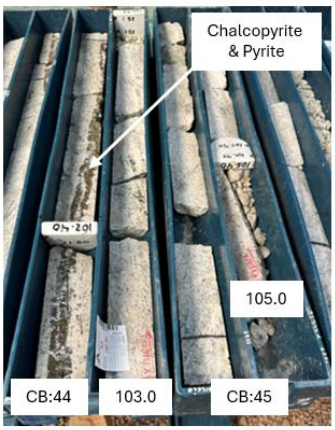
Drilling Date	Hole ID	From (m)	To (m)	Sample Length (m)	Lithology	Mineralization
		385.0	387.0	2.0	CPY	Primary
		387.0	389.0	2.0	CPY	Primary
	A23-011	778.0	780.0	2.0	CPY	Primary
		780.0	782.0	2.0	CPY	Primary
		782.0	784.0	2.0	CPY	Primary
		784.0	786.0	2.0	CPY	Primary
		786.0	788.0	2.0	CPY	Primary
		788.0	790.0	2.0	CPY	Primary
		790.0	792.0	2.0	CPY	Primary
		792.0	794.0	2.0	CPY	Primary
		794.0	796.0	2.0	CPY	Primary
		796.0	798.0	2.0	CPY	Primary
		798.0	800.0	2.0	CPY	Primary
		800.0	802.0	2.0	CPY	Primary
		802.0	804.0	2.0	CPY	Primary
		804.0	806.0	2.0	CPY	Primary
		806.0	808.0	2.0	CPY	Primary
2024	A24-014	87.0	89.0	2.0	BX	Leached
		89.0	91.0	2.0	BX	Leached
		91.0	93.0	2.0	BX	Leached
		93.0	95.0	2.0	BX	Leached
		95.0	97.5	2.5	BX	Leached
		97.5	100.0	2.5	BX	Leached
		100.0	102.5	2.5	BX	Leached
	A24-016	898.0	900.0	2.0	DPY	Primary
		900.0	902.0	2.0	DPY	Primary
		902.0	904.0	2.0	DPY	Primary
		904.0	906.0	2.0	DPY	Primary
		906.0	908.0	2.0	DPY	Primary
		908.0	910.0	2.0	DPY	Primary
		910.0	912.0	2.0	DPY	Primary
		912.0	914.0	2.0	DPY	Primary
	A24-017	73.0	75.0	2.0	HFL	Enriched
		75.0	77.0	2.0	HFL	Enriched
		77.0	79.0	2.0	HFL	Enriched
		79.0	81.0	2.0	HFL	Enriched
		81.0	83.0	2.0	HFL	Enriched
		83.0	85.0	2.0	HFL	Enriched
	A24-019A	86.0	88.0	2.0	HFL	Enriched
		88.0	90.0	2.0	HFL	Enriched
		90.0	92.0	2.0	HFL	Enriched
		92.0	94.0	2.0	HFL	Primary
		94.0	96.0	2.0	HFL	Primary
	A24-020	443.0	445.0	2.0	HFL	Primary
		445.0	447.0	2.0	HFL	Primary
		447.0	449.0	2.0	HFL	Primary

Drilling Date	Hole ID	From (m)	To (m)	Sample Length (m)	Lithology	Mineralization
		449.0	451.0	2.0	HFL	Primary
		451.0	453.0	2.0	HFL	Primary
		453.0	455.0	2.0	HFL	Primary
2025	A25-023	364.0	366.0	2.0	HFL	Primary
		366.0	368.0	2.0	BX	Primary
		368.0	370.0	2.0	BX	Primary
		370.0	372.0	2.0	BX	Primary
		372.0	374.0	2.0	BX	Primary
		374.0	376.0	2.0	BX	Primary
		376.0	378.0	2.0	BX	Primary
		378.0	380.0	2.0	HFL	Primary
	A25-024	212.0	214.0	2.0	CPY	Enriched
		214.0	216.0	2.0	CPY	Enriched
		216.0	218.0	2.0	CPY	Enriched
		218.0	220.0	2.0	CPY	Enriched
	A25-025	378.0	380.0	2.0	CPY	Enriched
		380.0	382.0	2.0	CPY	Enriched
		382.0	384.0	2.0	CPY	Enriched
		384.0	386.0	2.0	CPY	Enriched
		386.0	388.0	2.0	CPY	Enriched
		772.0	774.0	2.0	DPY	Primary
		774.0	776.0	2.0	DPY	Primary
		776.0	778.0	2.0	DPY	Primary
		778.0	780.0	2.0	DPY	Primary
		780.0	782.0	2.0	DPY	Primary
		782.0	784.0	2.0	DPY	Primary
		784.0	786.0	2.0	DPY	Primary
		786.0	788.0	2.0	DPY	Primary
		788.0	790.0	2.0	DPY	Primary
	A25-026	777.0	779.0	2.0	MPY	Primary
		779.0	781.0	2.0	MPY	Primary
		781.0	783.0	2.0	MPY	Primary
		783.0	785.0	2.0	MPY	Primary
		785.0	787.0	2.0	MPY	Primary
		787.0	789.0	2.0	MPY	Primary
		789.0	791.0	2.0	MPY	Primary
	A25-027	606.0	608.0	2.0	MPY	Primary
		608.0	610.0	2.0	MPY	Primary
		610.0	612.0	2.0	MPY	Primary
		612.0	614.0	2.0	MPY	Primary
		614.0	616.0	2.0	MPY	Primary
		616.0	618.0	2.0	MPY	Primary
		618.0	620.0	2.0	DPY	Primary
		620.0	622.0	2.0	DPY	Primary
		622.0	624.0	2.0	DPY	Primary
		624.0	626.0	2.0	DPY	Primary

Drilling Date	Hole ID	From (m)	To (m)	Sample Length (m)	Lithology	Mineralization
		626.0	628.0	2.0	DPY	Primary
		628.0	630.0	2.0	DPY	Primary
		630.0	632.0	2.0	DPY	Primary
		632.0	634.0	2.0	DPY	Primary

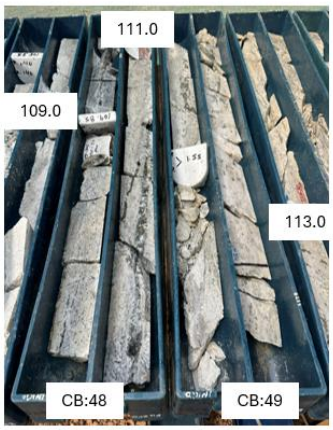
Photo 12-6: Visual Inspection of Selected Intervals from Hole A23-006

98.65-100.45 & 110.45-102.20m
Cu@0.44% & Mo@0.021%
from 99.0-101.0m



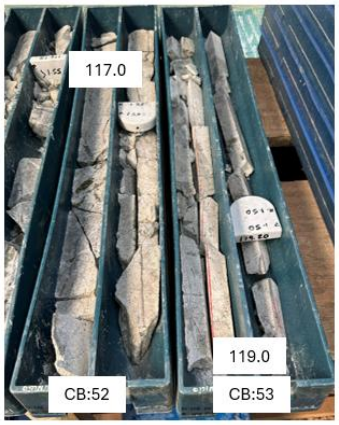
102.20-103.95 & 103.95-105.90m
Cu@1% & Mo@0.02 from 101.0-103.0
Cu@0.41% & Mo@0.017%
from 103.0-105.0m

105.90-107.60 & 107.60-109.30m
Cu@0.32% & Mo@0.015%
from 107.0-109.0m



109.30-111.05 & 111.05-112.75m
Cu@1% & Mo@0.02 from 109.1-111.0m
Cu@0.18% from 111.0-113.0m

112.75-114.50 & 114.50-116.35
Cu @ 0.67% & Mo @ 0.4%
from 113.0-115.0m



116.35-118.10 & 118.10-119.85m
Cu@1% & Mo @ 0.0 4from 115.0-117.0m
Cu@0.14 & Mo@0.02 from 117.0-119.0m

Source: GRE, 2026

Photo 12-7: Visual Inspection of Selected Intervals from Hole A22-001

Source: GRE, 2026

Photo 12-8: Visual Inspection of Selected Intervals from Hole A25-025

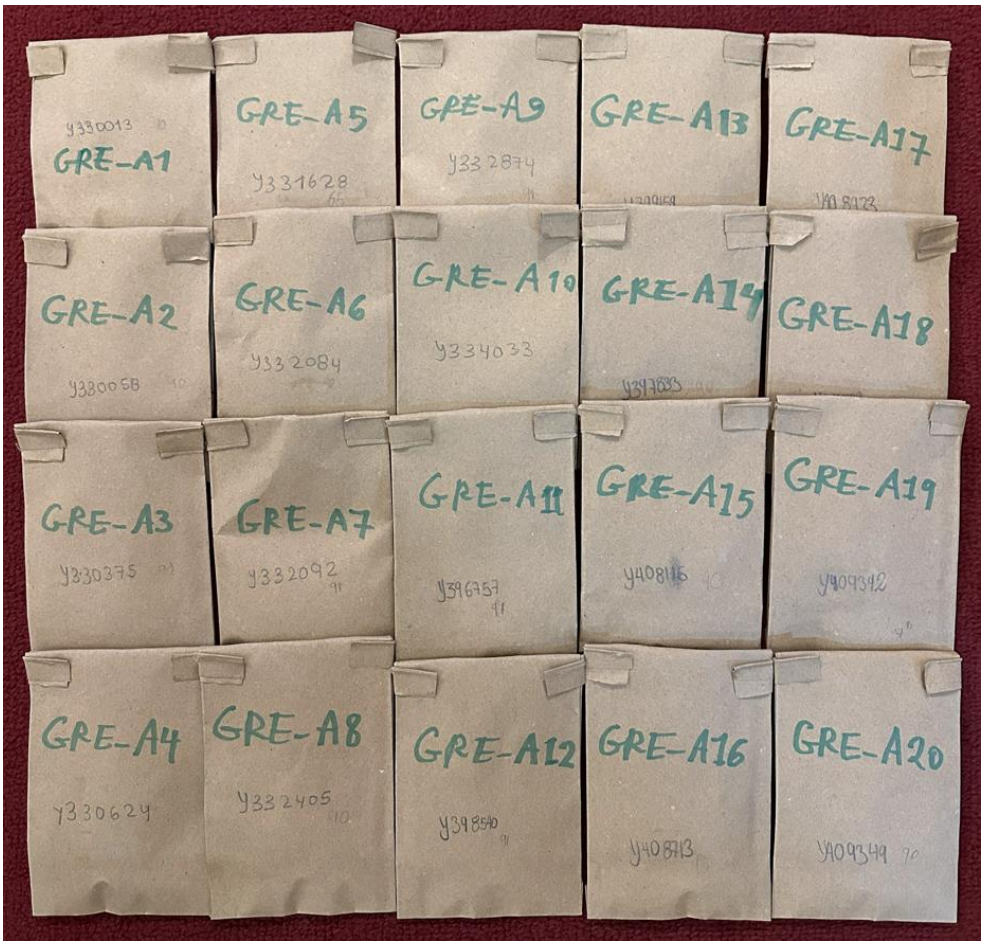
Source: GRE, 2026

12.5.4 Check Assay

GRE's QP, Dr. Hamid Samari, selected 20 sample intervals for independent check assay analyses. The selected intervals were obtained from diamond drill holes completed during the 2022 through 2025 drilling programs and were chosen from different areas of the deposit to provide representative spatial coverage of the mineralized system (Photo 12-9).

The corresponding pulp rejects were retrieved from DLP's sample archive and relabeled by Dr. Samari to maintain sample anonymity. A total of 20 check samples were subsequently shipped by Dr. Samari to Hazen Research, Inc. (Hazen) in Golden, Colorado, USA, for independent copper, molybdenum, and silver analyses using the ME-MS61 analytical method (ICP-MS following four-acid digestion), which is equivalent to the analytical method employed by ALS in Peru (Table 12-3).

Photo 12-9: Selected Pulp Reject Samples for Check Assay



Source: GRE, 2026

Hazen received all check samples on June 8, 2026.

On July 2, 2026, GRE's QP, Hamid Samari, received Hazen's analytical report on the 20 selected samples by ME-MS 61 analysis (Table 12-4).

Table 12-3: Check Samples Submitted to Hazen Labs

No.	Hole ID	DLP Sample #	From (m)	To (m)	Sample Length (m)	Rock Type	Mineralization	GRE Sample ID	Type of Sample	Request Analysis ME-MS 61 (ICP-MS) with 4-Acid Digestion		
										Ag	Cu	Mo
1	A22-001	Y330013	26.3	27.5	1.2	BX	Enriched	GRE-A1	Pulp	✓	✓	✓
2	A22-001	Y330058	118.7	120.7	2.0	HFL	Enriched	GRE-A2	Pulp	✓	✓	✓
3	A22-002	Y330375	340.0	342.0	2.0	HFL	Primary	GRE-A3	Pulp	✓	✓	✓
4	A22-003	Y330624	262.0	264.5	2.5	HFL	Enriched	GRE-A4	Pulp	✓	✓	✓
5	A23-006	Y331628	109.0	111.0	2.0	MPY	Enriched	GRE-A5	Pulp	✓	✓	✓
6	A23-007	Y332084	274.0	276.0	2.0	HFL	Primary	GRE-A6	Pulp	✓	✓	✓
7	A23-007	Y332092	290.0	292.0	2.0	HFL	Primary	GRE-A7	Pulp	✓	✓	✓
8	A23-008	Y332405	183.5	186.0	2.5	HFL	Primary	GRE-A8	Pulp	✓	✓	✓
9	A23-009	Y332874	379.0	381.0	2.0	CPY	Primary	GRE-A9	Pulp	✓	✓	✓
10	A23-011	Y334033	800.0	802.0	2.0	CPY	Primary	GRE-A10	Pulp	✓	✓	✓
11	A24-016	Y396757	906.0	908.0	2.0	DPY	Primary	GRE-A11	Pulp	✓	✓	✓
12	A24-017	Y398540	79.0	81.0	2.0	HFL	Enriched	GRE-A12	Pulp	✓	✓	✓
13	A24-019A	Y399159	88.0	90.0	2.0	HFL	Enriched	GRE-A13	Pulp	✓	✓	✓
14	A24-020	Y397833	443.0	445.0	2.0	HFL	Primary	GRE-A14	Pulp	✓	✓	✓
15	A25-024	Y408116	216.0	218.0	2.0	CPY	Enriched	GRE-A15	Pulp	✓	✓	✓
16	A25-025	Y408713	382.0	384.0	2.0	CPY	Enriched	GRE-A16	Pulp	✓	✓	✓
17	A25-025	Y408933	776.0	778.0	2.0	DPY	Primary	GRE-A17	Pulp	✓	✓	✓
18	A25-026	Y411922	781.0	783.0	2.0	MPY	Primary	GRE-A18	Pulp	✓	✓	✓
19	A25-027	Y409342	608.0	610.0	2.0	MPY	Primary	GRE-A19	Pulp	✓	✓	✓
20	A25-027	Y409349	620.0	622.0	2.0	DPY	Primary	GRE-A20	Pulp	✓	✓	✓

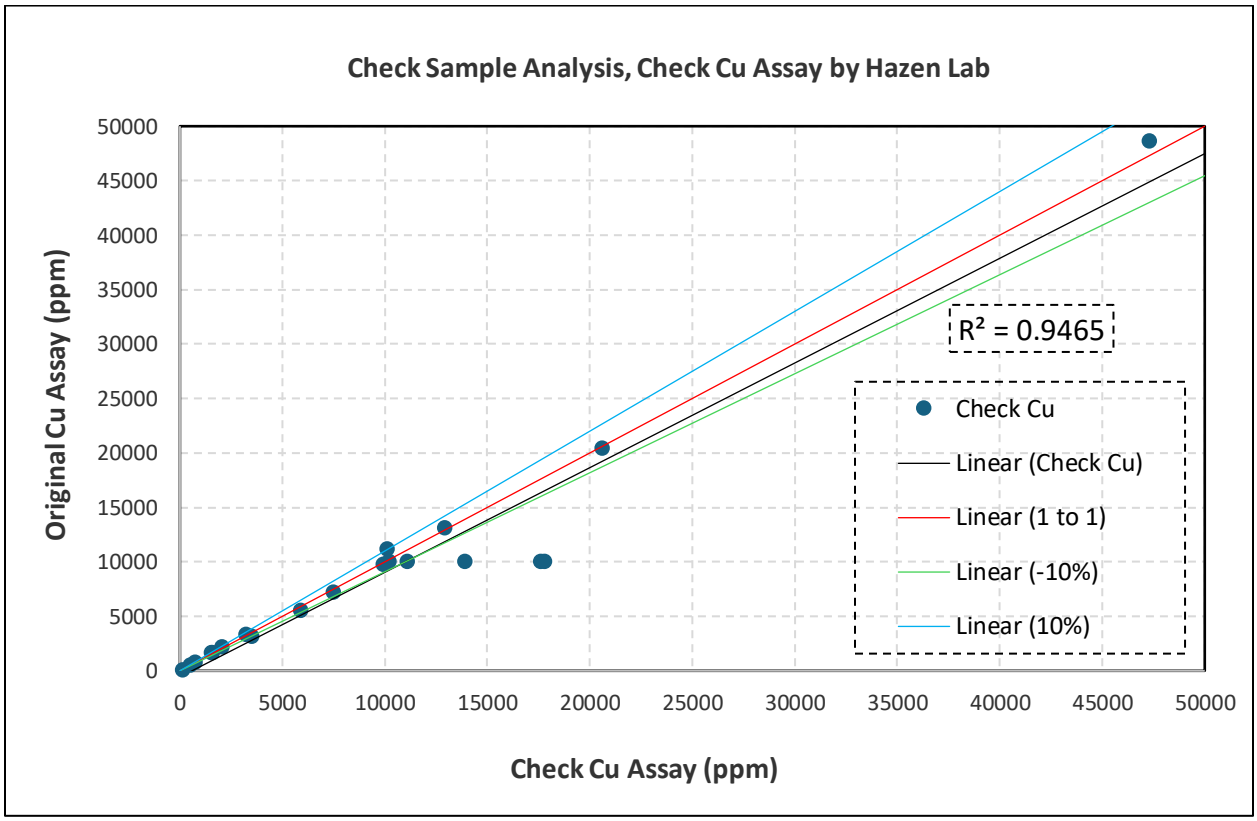
Table 12-4: Summary Table of Hazen Results with Original Assays

No.	Hole ID	DLP Sample #	From (m)	To (m)	GRE Sample ID	Original Assays from Database (ppm) ME-MS 61 (ICP-MS)			Hazen Results (ppm) ME-MS 61 (ICP-MS)		
						Ag (ppm)	Cu (ppm)	Mo (ppm)	Ag (ppm)	Cu (ppm)	Mo (ppm)
1	A22-001	Y330013	26.25	27.48	GRE-A1	1.12	10000.00	70.60	<4.00	11100.00	81.00
2	A22-001	Y330058	118.65	120.65	GRE-A2	4.97	9740.00	52.90	5.00	9900.00	51.00
3	A22-002	Y330375	340.00	342.00	GRE-A3	11.90	7210.00	33.50	14.00	7500.00	33.00

No.	Hole ID	DLP Sample #	From (m)	To (m)	GRE Sample ID	Original Assays from Database (ppm) ME-MS 61 (ICP-MS)			Hazen Results (ppm) ME-MS 61 (ICP-MS)		
						Ag (ppm)	Cu (ppm)	Mo (ppm)	Ag (ppm)	Cu (ppm)	Mo (ppm)
4	A22-003	Y330624	262.00	264.50	GRE-A4	6.34	10000.00	17.55	6.00	13900.00	19.00
5	A23-006	Y331628	109.00	111.00	GRE-A5	21.90	10000.00	265.00	20.00	17600.00	240.00
6	A23-007	Y332084	274.00	276.00	GRE-A6	4.96	5510.00	25.50	5.00	5900.00	28.00
7	A23-007	Y332092	290.00	292.00	GRE-A7	14.15	10000.00	51.50	13.00	10200.00	56.00
8	A23-008	Y332405	183.50	186.00	GRE-A8	38.50	13100.00	409.00	38.00	12900.00	410.00
9	A23-009	Y332874	379.00	381.00	GRE-A9	2.56	3160.00	48.90	<4.00	3490.00	51.00
10	A23-011	Y334033	800.00	802.00	GRE-A10	0.55	754.00	3560.00	<4.00	750.00	3700.00
11	A24-016	Y396757	906.00	908.00	GRE-A11	0.93	1660.00	1325.00	<4.00	1580.00	1500.00
12	A24-017	Y398540	79.00	81.00	GRE-A12	1.58	48600.00	1.88	<4.00	47300.00	4.00
13	A24-019A	Y399159	88.00	90.00	GRE-A13	0.45	20400.00	1.58	<4.00	20600.00	4.00
14	A24-020	Y397833	443.00	445.00	GRE-A14	6.33	3290.00	5.13	6.00	3240.00	6.00
15	A25-024	Y408116	216.00	218.00	GRE-A15	18.45	10000.00	159.00	18.00	17800.00	150.00
16	A25-025	Y408713	382.00	384.00	GRE-A16	8.47	11200.00	337.00	8.00	10100.00	300.00
17	A25-025	Y408933	776.00	778.00	GRE-A17	0.05	69.00	5720.00	<4.00	130.00	5600.00
18	A25-026	Y411922	781.00	783.00	GRE-A18	1.10	1615.00	2270.00	<4.00	1530.00	2400.00
19	A25-027	Y409342	608.00	610.00	GRE-A19	1.64	2140.00	3890.00	<4.00	2040.00	4100.00
20	A25-027	Y409349	620.00	622.00	GRE-A20	0.30	493.00	3050.00	<4.00	510.00	3200.00

The check assay comparison between the original copper assays, analyzed by DLP using the ME-ICP61 ICP-AES method, and the independent Hazen Laboratory results, analyzed using the same analytical method, demonstrates excellent analytical agreement and confirms the overall reliability of the copper assay database. Most samples plot very close to the 1:1 reference line and fall within the ±10% error envelope, indicating excellent reproducibility between the original and check assay results. The regression coefficient (R^2 = 0.9465) indicates a very strong correlation between the two datasets (Figure 12-60).

Figure 12-60: Hazen Cu Results on Check Samples



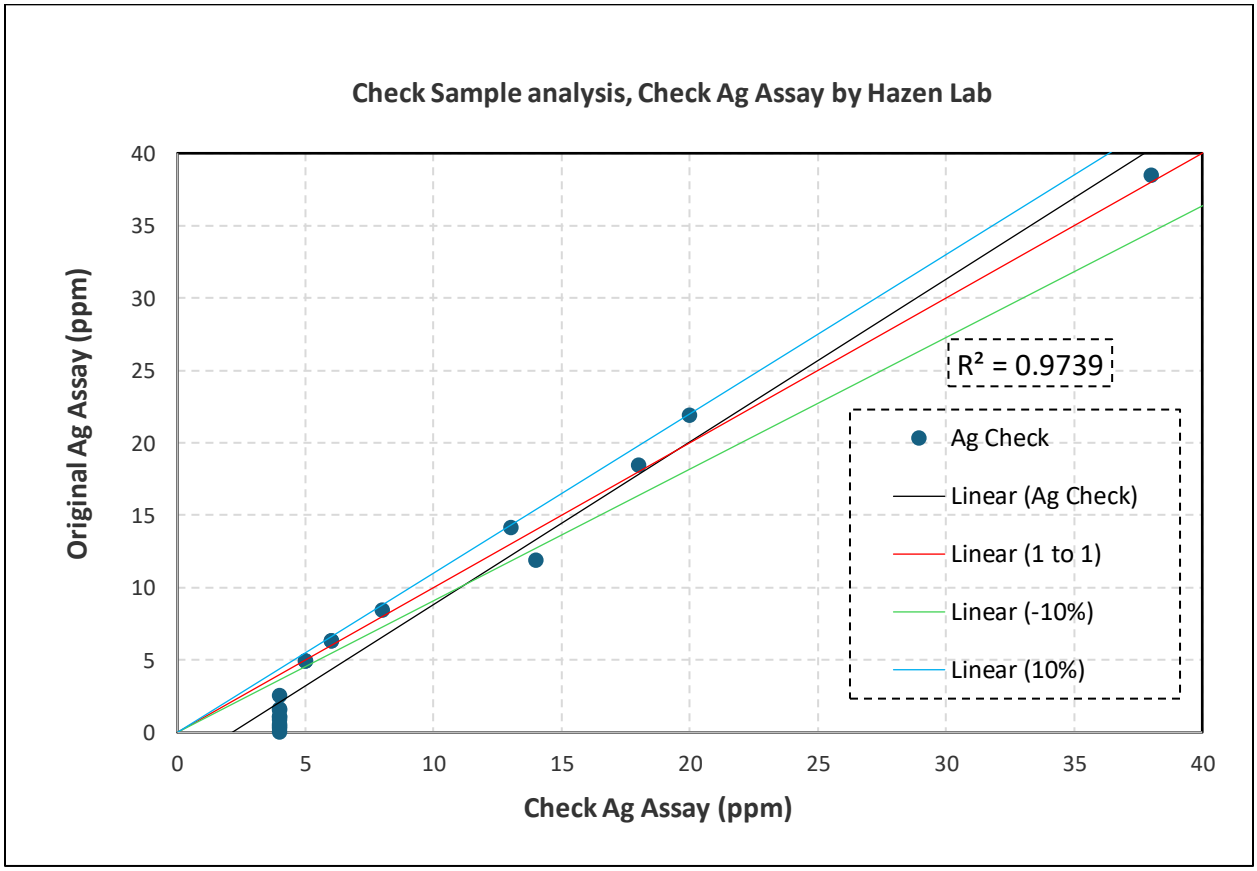
Source: GRE, 2026

An additional observation from the check assay program relates to the reporting of high-grade copper values. Five check assay samples recorded as 10,000 ppm Cu in the DLP database were reanalyzed by Hazen Laboratory and returned quantitative copper values of 11,100, 13,900, 17,600, 10,200, and 17,800 ppm, confirming that the original copper concentrations exceeded the 10,000-ppm reporting limit recorded in the database. These results indicate that the 10,000 ppm values in the DLP database represent a reporting cap rather than the actual copper concentrations for these intervals. Review of the DLP assay database identified 48 assay intervals recorded as 10,000 ppm Cu, including 32 intervals from the 2022 drilling program, 15 intervals from the 2023 drilling program, and one interval from the 2025 drilling program. In contrast, the database also contains 67 assay intervals from the 2023, 2024, and 2025 drilling programs with reported copper values greater than 10,000 ppm, reaching a maximum of 61,500 ppm. The GRE's QP recommends that, for future drilling programs, DLP request that ALS report actual quantitative copper values for assays exceeding 10,000 ppm, where available, rather than truncating the results at the reporting limit, thereby preserving the complete analytical dataset.

Aside from the reporting limitation discussed above, only minor deviations from the 1:1 reference line are observed and are considered consistent with normal analytical and subsampling variability between laboratories. No significant systematic analytical bias is evident. Overall, the check assay program provides a high level of confidence in the accuracy and reliability of the copper assay database and supports its use for geological interpretation and Mineral Resource Estimation.

The check assay comparison between the original silver assays, analyzed by DLP using the ME-MS61 (ICP-MS) method, and the independent Hazen Laboratory results, analyzed using the same analytical method, demonstrates excellent analytical agreement and confirms the reliability of the silver assay database. Most samples plot very close to the 1:1 reference line and fall within the ±10% error envelope, indicating excellent reproducibility between the original and check assay results. The regression coefficient (R^2 = 0.9739) indicates a very strong correlation between the two datasets (Figure 12-61). Minor deviations from the 1:1 reference line are observed for a limited number of low-grade samples because Hazen Laboratory reports silver concentrations below its analytical reporting limit as <4 ppm. For graphical comparison, these values were plotted at the reporting limit (4 ppm), resulting in apparent deviations at low silver grades. These differences reflect the analytical reporting limit rather than systematic analytical bias. Overall, the check assay program demonstrates excellent agreement between the original and independent assays and confirms that the silver assay database is suitable for geological interpretation and Mineral Resource Estimation.

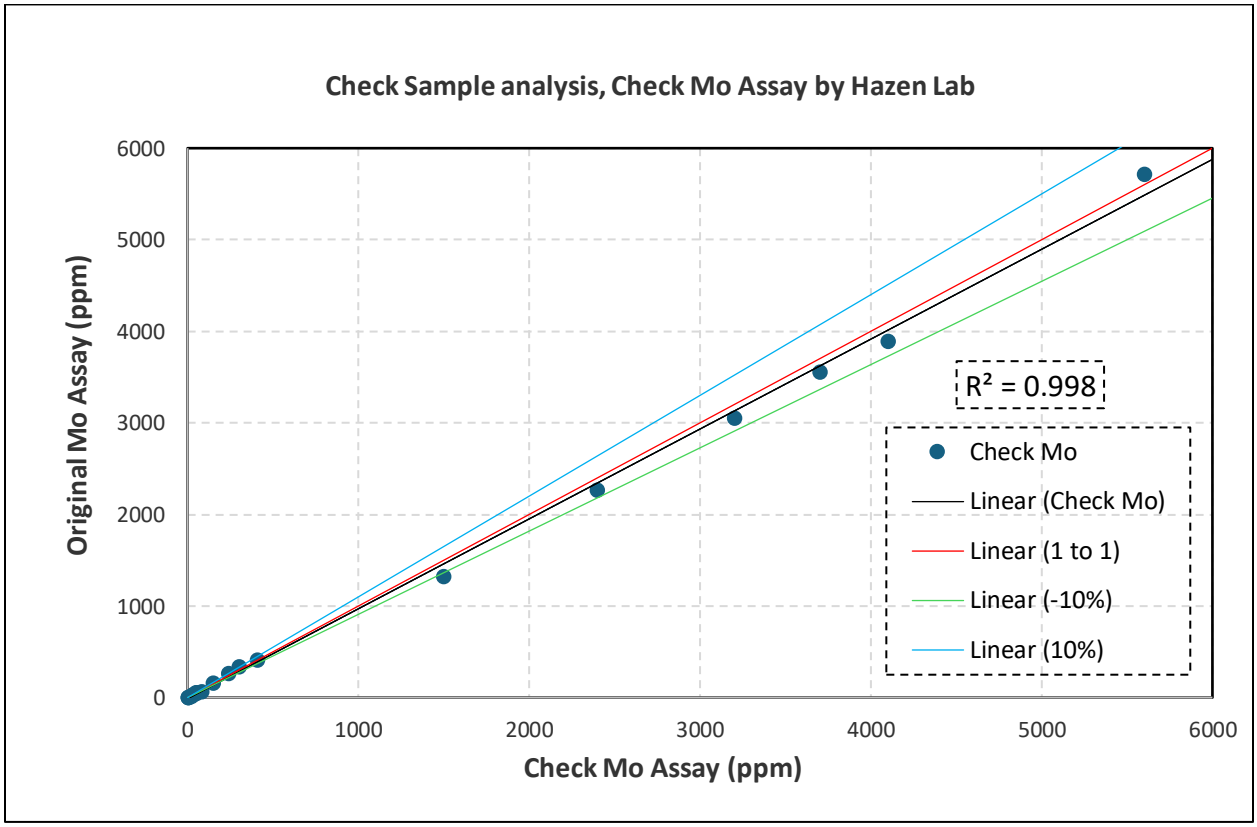
Figure 12-61: Hazen Ag Results on Check Samples



Source: GRE, 2026

The check assay comparison between the original molybdenum assays, analyzed by DLP using the ME-MS61 (ICP-MS) method, and the independent Hazen Laboratory results, analyzed using the same analytical method, demonstrates excellent analytical agreement and confirms the reliability of the molybdenum assay database. Most samples plot very close to the 1:1 reference line and fall within the ±10% error envelope, indicating excellent reproducibility between the original and check assay results. The regression coefficient (R^2 = 0.998) indicates an exceptionally strong correlation between the two datasets (Figure 12-62). Only very minor deviations from the 1:1 reference line are observed and are considered consistent with normal analytical and subsampling variability between laboratories. No significant systematic analytical bias is evident. Overall, the check assay program provides a very high level of confidence in the accuracy and reliability of the molybdenum assay database and supports its use for geological interpretation and Mineral Resource Estimation.

Figure 12-62: Hazen Mo Results on Check Samples



Source: GRE, 2026

12.6 Data Verification QP Opinion on Adequacy

12.6.1 Dr. Hamid Samari (GRE)

GRE's QP, Dr. Hamid Samari, independently reviewed and evaluated the available drilling, sampling, geological, analytical, and QA/QC information associated with the Aurora Project databases, including data from the diamond drilling programs completed by DLP from 2022 through 2025. The review did not identify any material issues that would adversely affect the integrity of the Aurora Project databases.

The QP independently reviewed and audited the Aurora Project databases by comparing the digital databases with original analytical certificates, collar survey data, geological logs, and other source files. The review included verification of collar coordinates, downhole surveys, geological logging, sample intervals, assay databases, and QA/QC information. Approximately 27% of the analytical database was manually spot-checked against original laboratory certificates and source records for accuracy, and no transcription errors, material discrepancies, or database integrity issues were identified.

The QP also independently reviewed and evaluated the complete QA/QC datasets for the 2022 through 2025 drilling programs, including blank samples, certified reference materials (CRMs), and duplicate samples. QA/QC control charts were prepared and evaluated for all QA/QC sample types, and the results demonstrated satisfactory analytical accuracy and precision, with no evidence of material contamination, analytical bias, or other material issues that would adversely affect the integrity of the databases.

Furthermore, the QP conducted independent site inspections of the Aurora Project and DLP's core storage facility in Calca, Peru. These site visits included verification of selected drill hole collar locations, review of geological interpretations, inspection of representative diamond drill core, verification of geological logging records, and review of sample handling and storage procedures. Selected pulp samples were submitted for independent check assays, and the results demonstrated acceptable agreement with the original analytical results and did not indicate any systematic analytical bias.

Based on the independent database audits, QA/QC reviews, site inspections, geological verification activities, and independent check assay results, the GRE QP is of the opinion that the Aurora Project databases are sufficiently accurate, reliable, and suitable for use in Mineral Resource Estimation.

12.6.2 Dr. Todd Harvey (GRE)

Dr. Todd Harvey, the Metallurgy QP, believes that the metallurgical testing was completed for the Aurora project by a number of well-known commercial metallurgical laboratories. Dr. Harvey reviewed the sample selection and compositing used in the metallurgical test work and found that the selection of samples was representative for this type of deposit and geology. Dr. Harvey performed several mathematical tests to validate the metallurgical balances presented in the test work and found the data presented in the metallurgical reports to be consistent with practices performed by reputable independent test laboratories. A complete discussion of the test work is provided in Section13.0. The work appears to be professionally completed and is well documented, is supported by similar operational data, and is suitable for estimation of flotation copper and molybdenum recovery calculations in this PEA.

12.6.3 Ms. Terre Lane (GRE)

Open pit mining methods, costs, and infrastructure needs were verified by comparison to other similar sized open pit copper/molybdenum mines operating in Peru and experience of the QP, Ms. Lane. Cost data used in the report was sourced from the most recent Infomine cost data report. All costs used in the analysis were verified and reviewed by Ms. Lane and were assessed to be current and appropriate for use. Finally, after the economic study was performed, the overall operating costs for different aspects of the operation (mining and general & admin) were benchmarked against similar sized mines and recent technical reports to determine if they were similar; the results did benchmark well to other operations and economic studies.

The taxation rates used and applied were values available from Peruvian government sources at the time of the economic analysis.

12.6.4 Scott Elfen (Ausenco)

Mr Scott Elfen, P.E. and P.Eng., SME and Global Technical leader at Ausenco Canada ULC., visited the site as described in Section 2.3. Geotechnical information for the infrastructure was based on the site visit, site topography, and mine production, and was used in the design of the infrastructure. In the opinion of the QP responsible for the geotechnical information, the data is sufficient for the design of the infrastructure contained in this report.

12.6.5 Verifications by SRK

12.6.5.1 Site Visit

The site visit was completed by Humberto Pichuante (SRK). The team visited the Aurora property from 2-3 June 2026. The purpose of this visit for SRK professional was to:

- Develop an understanding of the project's geological framework, deposit setting, and key geological controls relevant to rock mass characterization and caveability assessments.
- Review selected drillholes withing the proposed caving area to support rock mass characterization

The first day of the site visit included a tour of the project area, selected drilling platforms, and one of the proposed portal locations. The second day was spent at the core logging facility in Calca where selected drillholes were reviewed and partially logged to support the geotechnical assessment of the proposed caving area.

SRK was given full access to relevant data and conducted interviews with DLP personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyse historical and current geotechnical data.

SRK was given full access to relevant data and conducted interviews with DLP personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyse historical and current geotechnical data.

12.6.5.2 SRK Comments

The site visit provided SRK with an opportunity to review the principal lithological units and alterations exposed in both outcrop and drillcore. Although the geotechnical data is very limited at this stage of the project, this review facilitated a high-level assessment of the available geological and geotechnical data, as well as the rock mass conditions within the proposed caving area, for the purposes of caveability evaluation.

Based on the information reviewed, SRK is of the opinion that the data pertaining to the mine design is sufficient to support this PEA.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The original Aurora deposit domain classifications consisted of three porphyry domains: Deep Porphyry (molybdenum - dominated) (DPY), Copper Porphyry (copper-dominated) (CPY), and Molybdenum Porphyry (molybdenum-dominated with minor copper) (MPY) (AMC, 2025). The domains were revised in 2026 to include Leached (LE), Enriched (EN), Breccia (BX), Copper Porphyry (copper-dominated) (CPY), Moly Porphyry (molybdenum-dominated with minor copper) (MPY), and Deep Porphyry (molybdenum-dominated) (DPY). Details on these domains is provided in Section 14.0.

The main porphyry intrusions are hosted in a hornfels country rock, with mineralization bleeding into the country rock. The porphyry units are described as granodiorite / quartz monzonite. A thin supergene layer is observed at the surface of the northeastern portion of the deposit, consisting of chalcocite. The primary copper species is chalcopyrite, while the primary molybdenum species is molybdenite. Small zones of breccia are observed in the contacts between the porphyry and hornfels units. Alteration is predominantly potassic, with minimal advanced argillic alteration observed.

Laboratorio Plenge, Miraflores Lima Peru (Plenge), an established mineral analysis and metallurgical research laboratory in Peru, has carried out early-stage metallurgical evaluation of the Aurora deposit. Two test programs have been conducted by Plenge, one started in late 2023 (C. H. Plenge & CIA. S.A., 2024), an additional phase of testing in late 2024 (C. H. Plenge & CIA. S.A., 2024) and the most recent in 2025 (C. H. Plenge & CIA. S.A., 2026). Eagle Engineering conducted a mineralogy study in 2023 (Eagle Engineering, 2023) and the Center for Advanced Materials Processing (CAMP) conducted additional mineralogy in 2024 (CAMP, 2024). Additional studies included grinding test work (SMC) conducted by the University of Queensland (The University of Queensland, 2025) and comminution sizing conducted by Alex G Doll Consulting Limited (Alex G Doll Consulting Limited, 2026).

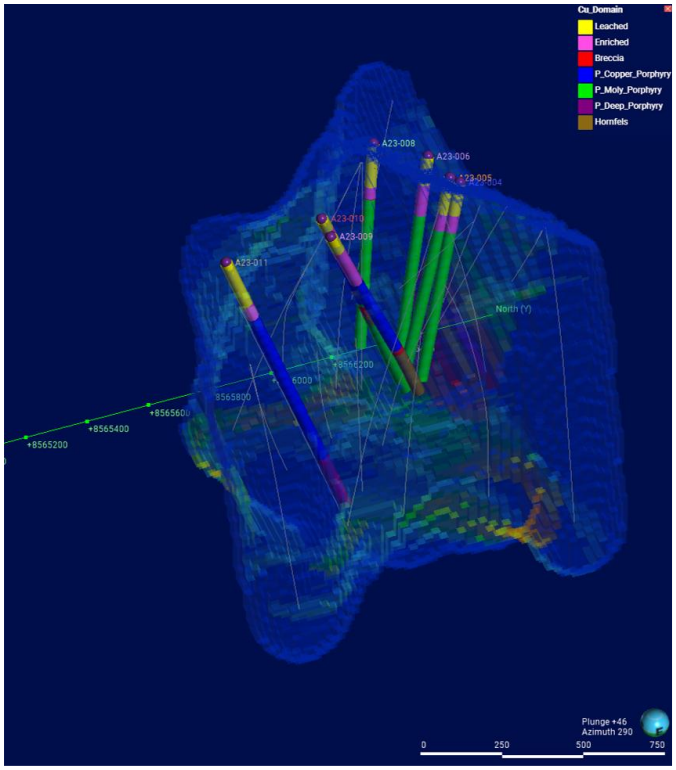
13.1 Metallurgical Sampling

13.1.1 2023 Samples

Three metallurgical composite samples were assembled in 2023 for use in initial, exploratory metallurgical testing of the Aurora deposit. The samples were diamond drill core composites assembled from drillholes shown in Figure 13.1. The samples, labelled M1, M2, and M3, were characterized by DLP and Plenge as follows:

- M1 (Muestra #1) – Partial Leached, from the top of the Molybdenum Porphyry (Mo Porphyry domain), yellow in Figure 13-1.
- M2 (Muestra #2)– Mixed Enriched, from the upper portion of the Mo Porphyry, blue in Figure 13-1.
- M3 (Muestra #3)– Moly Rich, from the lower portions of the Mo Porphyry and the Copper Porphyry (Cu Porphyry), green in Figure 13-1.

Figure 13-1: Aurora Geological Model Showing the Locations of the Samples Used in the 2023 Metallurgical Test Program



Note: Partial Leached =M1, Mixed enriched = M2, Moly Rich = M3.

Source: GRE, 2026.

These early samples were not fully representative of the mineralized domains of the Aurora deposit as a whole. The following general observations can be made (AMC, 2025):

- The samples are drawn from within the Mineral Resource area.
- Diamond drill core making up the samples has predominantly been drawn from the Mo Porphyry. M3 is partially comprised of diamond drill core from the Cu Porphyry.
- Mineralization in the samples is likely to represent mineral species that will be processed. Assay grades and relative proportions of mineral species in the domains cannot be reliably determined from analysis of M1, M2, and M3 which will provide only indicative data at this stage of the project.
- Cu Porphyry material has not been exclusively sampled and tested. Some material was tested as part of a blend in M3.
- The enriched zone at the top of the Mo Porphyry was not exclusively sampled in this program and tested. Some material was tested as part of a blend in M2.
- Sample M1 consisted mainly of the leached zone.

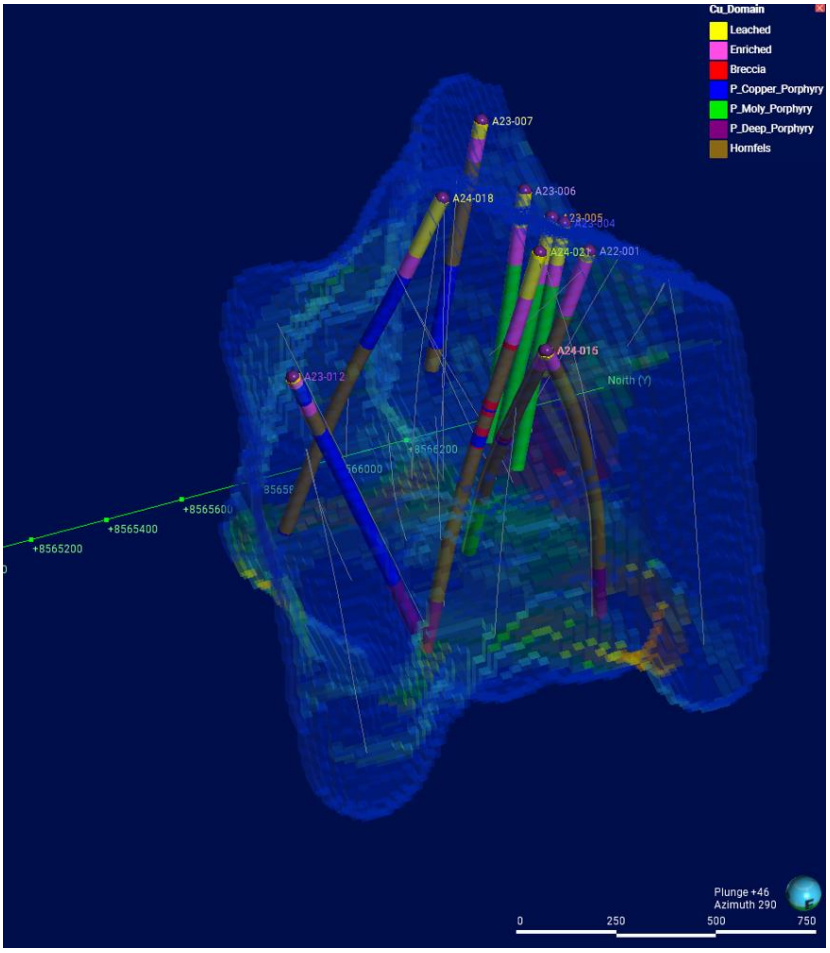
13.1.2 2025 Samples

Four metallurgical composite samples were assembled in 2025 for use in follow-up metallurgical testing of the Aurora deposit. The samples were diamond drill core composites assembled from drillholes shown

in**Error! Reference source not found.** . The samples, labeled Cu, Mo, HF, and EN, were characterized by DLP and Plenge as follows:

- Cu – Copper-dominated with minor molybdenum, from the upper portion of the Copper Porphyry, blue.
- Mo – Molybdenum-dominated with minor copper, from the lower portion of the Molybdenum Porphyry, green and burgundy.
- HF – Hornfels, Copper-dominated with minor molybdenum, from the Hornfels domain, brown.
- EN – Enriched, Copper-dominated with minor molybdenum, high grade copper, from the Enriched domain, pink.

Figure 13-2: Aurora Geological Model Showing the Locations of the Samples Used in the 2025 Metallurgical Test Program



Source: GRE, 2026

The 2025 samples were sourced from the diamond drillholes (DDH) shown in Table 13-22 at the end of the section. Additionally, several variability samples were selected for additional testing as shown in Table 13-23.

13.2 2024 Metallurgical Program

13.2.1 Mineralogy

In 2024, Plenge had modal mineralogical analysis (mineral distribution, grain size, mineral associations) conducted on M1, M2, and M3 using AMICS (Automated Mineralogy System for SEM [scanning electron microscopy]). No additional mineralogical investigations were conducted. The samples were examined in four size fractions as follows:

- +60 mesh (+250 microns [µm])
- -60 mesh/+100 mesh (150 µm)
- -100 mesh/+200 mesh (75 µm)

-200 mesh**Error! Reference source not found.**, Table 13-2, and Table 13-3 show the results of modal analysis. The following general metallurgical observations can be made:

- Samples are predominantly quartz and other silicates such as orthoclase and muscovite.
- Pyrite is the predominant sulfide mineral.
- Copper minerals include bornite, chalcocite, chalcopyrite, chrysocolla, and enargite.
- Molybdenum as molybdenite was observed in all samples.

Table 13-1: Modal Mineralogy Results – M1

Mineral	Chemistry	+60 mesh	-60/+100 mesh	-100/+200 mesh	-200 mesh
Andalusite	Al2SiO5	0.40	0.29	0.63	0.69
Arsenopyrite	FeAsS	0.00	0.00	0.04	0.00
Biotite	K(Mg,Fe)3Si3O10(OH)2	0.01	0.00	0.00	0.00
Bornite	Cu5FeS4	0.04	0.01	0.02	0.10
Calcite	CaCO3	0.38	0.01	0.23	0.23
Chalcocite	Cu2S	0.02	0.02	0.08	0.03
Chalcopyrite	CuFeS2	0.41	0.65	0.32	0.26
Chrysocolla	CuSiO3*2H2O	0.00	0.01	0.04	0.04
Clinochlore	(Mg,Fe)5A1Si3O10(OH)2	0.00	0.00	0.01	0.01
Enargite	Cu3AsS4	0.02	0.01	0.06	0.00
Fayalite	Fe2SiO5	0.00	0.03	0.05	0.04
Hematite	Fe2O3	0.00	0.04	0.10	0.00
Molybdenite	MoS2	0.01	0.21	0.04	0.07
Muscovite	KA12Si3O10(OH)2	4.38	7.59	15.96	13.49
Orthoclase	KA1Si3O8	25.70	29.92	26.78	26.13
Plagioclase	(Na,Ca)A1Si3O8	0.60	0.19	0.20	0.21
Pyrite	FeS2	1.80	2.01	1.08	0.72
Quartz	SiO2	66.00	58.80	54.22	57.83
Rustile	TiO2	0.21	0.20	0.12	0.05
Shhalerite	ZnS	0.00	0.00	0.01	0.09
Zircon	ZrSiO4	0.02	0.01	0.01	0.01

Source: Plenge, 2024

Table 13-2: Modal Mineralogy Results – M2

Mineral	Chemistry	+60 mesh	-60/+100 mesh	-100/+200 mesh	-200 mesh
Andalusite	A12SiO5	0.03	0.10	0.39	0.35
Apatite	Ca5(PO4)3OH	0.01	0.00	0.09	0.01
Arsenopyrite	FeAsS	0.00	0.00	0.00	0.09
Biotite	K(Mg,Fe)3Si3O10(OH)2	0.24	0.00	0.07	0.04
Bornite	Cu5FeS4	0.10	0.22	0.02	0.12
Calcite	CaCO3	0.22	0.37	0.53	0.71
Chalcocite	Cu2S	0.13	0.14	0.44	0.28
Chalcopyrite	CuFeS2	1.14	0.48	0.81	1.45
Chrysocolla	CuSiO3*2H2O	0.00	0.00	0.01	0.17
Clinochlore	(Mg,Fe)5A1Si3O10(OH)2	0.03	0.14	0.04	0.30
Enargite	Cu3AsS4	0.02	0.01	0.00	0.02
Epidote	Ca2A12FeSi3O10(OH)	0.00	0.00	0.00	0.01
Fayalite	Fe2SiO5	0.00	0.00	0.02	0.04
Molybdenite	MoS2	0.07	0.02	0.18	0.07
Muscovite	KA12Si3O10(OH)2	14.97	8.49	12.88	18.43
Orthoclase	KA12Si3O8	9.98	14.10	16.88	16.48
Plagioclase	(Na,Ca)A1Si3O8	0.38	0.12	0.27	0.50
Pyrite	FeS2	6.27	4.77	3.83	2.06
Quartz	SiO2	66.32	70.91	63.39	58.76
Rhodochrosite	MnCO3	0.00	0.00	0.00	0.00
Rustile	TiO2	0.06	0.12	0.05	0.07
Shhalerite	ZnS	0.02	0.00	0.10	0.00
Zircon	ZrSiO4	0.01	0.02	0.00	0.03

Source: Plenge, 2024

Table 13-3: Modal Mineralogy Results – M3

Mineral	Chemistry	+60 mesh	-60/+100 mesh	-100/+200 mesh	-200 mesh
Andalusite	A12SiO5	0.00	0.00	0.07	0.03
Apatite	Ca5(PO4)3OH	0.05	0.01	0.00	0.05
Biotite	K(Mg,Fe)3Si3O10(OH)2	0.21	0.08	0.46	0.38
Calcite	CaCO3	0.88	1.00	1.23	0.95
Chalcopyrite	CuFeS2	0.02	0.16	0.15	0.41
Chrysocolla	CuSiO3*2H2O	0.04	0.01	0.08	0.04
Clinochlore	(Mg,Fe)5A1Si3O10(OH)2	0.07	0.03	0.03	0.32
Epidote	Ca2A12FeSi3O10(OH)	0.00	0.00	0.00	0.02
Fayalite	Fe2SiO5	0.02	0.01	0.00	0.02
Hematite	Fe2O3	0.08	0.00	0.15	0.10
Molybdenite	MoS2	0.07	0.01	0.27	0.75
Muscovite	KA12Si3O10(OH)2	3.72	4.15	5.98	8.00
Orthoclase	KA12Si3O8	38.45	31.05	34.83	37.12
Plagioclase	(Na,Ca)A1Si3O8	2.51	1.91	2.20	2.50
Pyrite	FeS2	0.96	2.81	1.92	1.46
Quartz	SiO2	52.85	58.72	52.61	47.65
Rustile	TiO2	0.07	0.05	0.02	0.19

Mineral	Chemistry	+60 mesh	-60/+100 mesh	-100/+200 mesh	-200 mesh
Shhalerite	ZnS	0.00	0.00	0.00	0.01

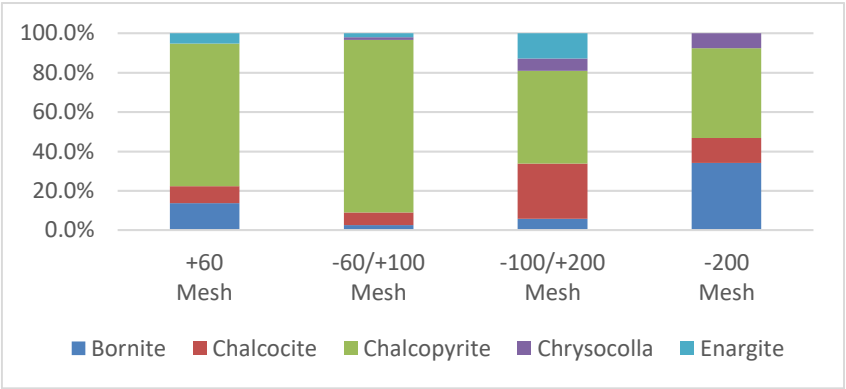
Source: Plenge, 2024

13.2.1.1 Copper Deportment

Figure 13-3, Figure 13-4, and Figure 13-5 show the deportment of Cu by size range in each sample. The following general observations can be made:

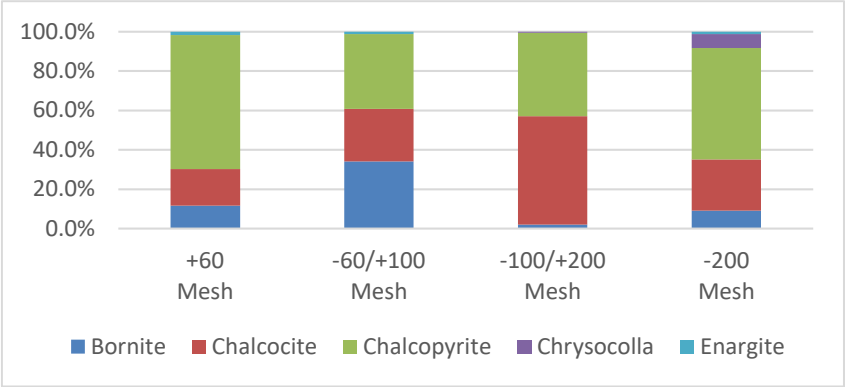
- M1 (characterized as the enriched chalcocite domain) is predominantly chalcopyrite with chalcocite and bornite. Bornite, chalcocite and chrysocolla appear preferentially in the finer fractions.
- In M2, the Cu is distributed more evenly, and minor enargite was observed.
- In M3, Chalcopyrite is the predominant Cu species (high-Mo domain).

Figure 13-3: Copper Deportment – M1



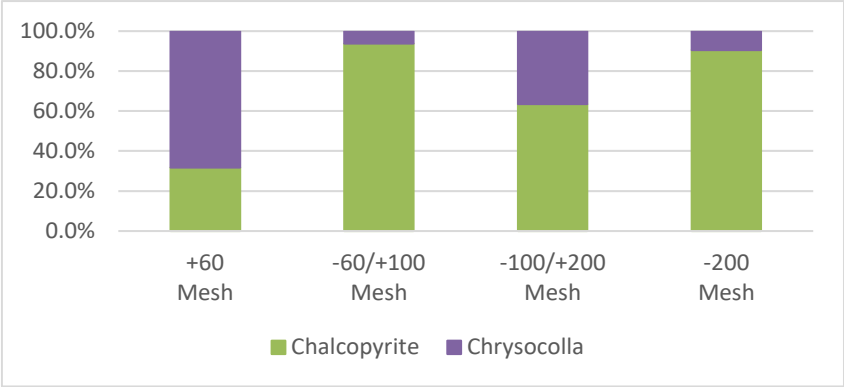
Source: Plenge, 2024.

Figure 13-4: Copper Deportment – M2



Source: Plenge, 2024

Figure 13-5: Copper Deportment – M3



Source: Plenge, 2024.

13.2.1.2 Molybdenum Association

Molybdenite mineral association was determined for M1, M2, and M3. Molybdenite particles were found to mostly be associated with orthoclase and quartz particles for all samples.

13.2.1.3 Mineral Liberation Size

Average particle sizes for the likely most abundant copper and molybdenum minerals, and likely most economically valuable minerals – chalcopyrite and molybdenite were determined for the four selected size ranges. Results are displayed in Figure 13-6, Figure 13-7, and Figure 13-8. Pledge noted the following:

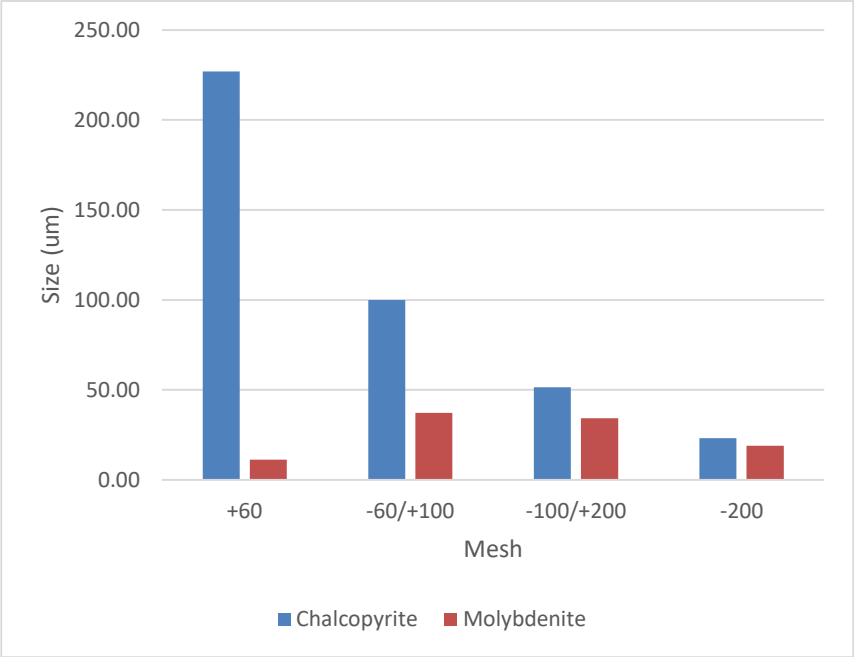
- In general, chalcopyrite particles are coarser than molybdenite particles in each range examined.
- For M1, chalcopyrite particles ranged from ~227 µm to ~23 µm. Molybdenite particles ranged from ~37 µm to ~19 µm.
- For M2, chalcopyrite particles ranged from ~128 µm to ~36 µm. Molybdenite particles ranged from ~40 µm to ~25 µm.
- For M3, chalcopyrite particles ranged from ~86 µm to ~64 µm. Molybdenite particles ranged from ~53 µm to ~27 µm.

The chalcopyrite particle sizes differ between ore domains, and, as a result, the primary grind will need to consider the minimum liberation necessary for effective flotation recovery. Fine grinding, especially regrinding prior to molybdenum flotation, will be required for successful liberation and flotation recovery of chalcopyrite and molybdenite particles.

Target mineral associations (with each other and with the major constituents (pyrite, quartz, orthoclase, muscovite, calcite)) were investigated with the general conclusions as follows:

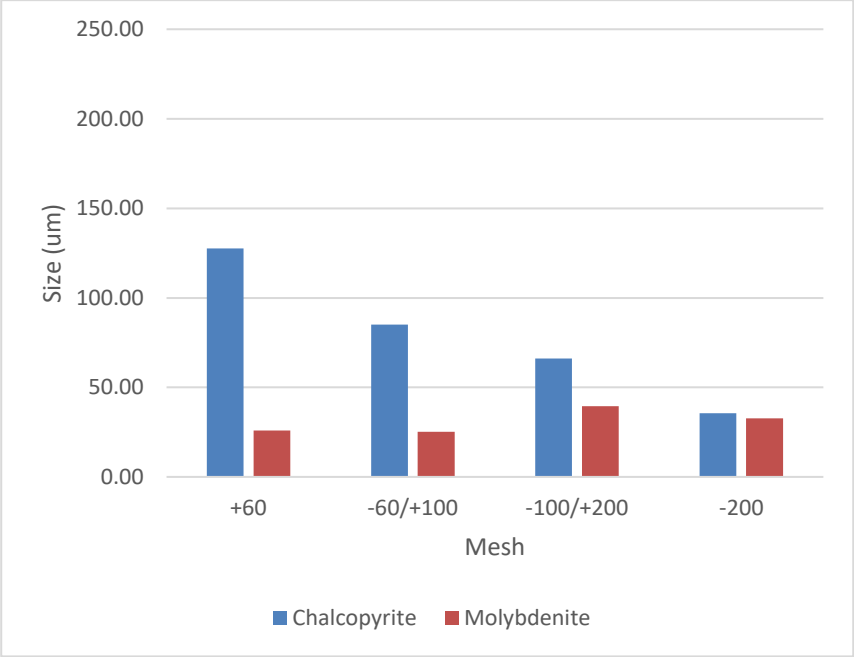
- Chalcopyrite in M1 is associated with Cu minerals, orthoclase, and quartz.
- Chalcopyrite in M2 is associated with Cu minerals and quartz.
- Chalcopyrite in M3 is associated with Cu minerals, pyrite, orthoclase, muscovite, and quartz.
- Molybdenite in all samples is predominantly associated with orthoclase and quartz.

Figure 13-6: Average Chalcopyrite and Molybdenite Particle Sizes – M1



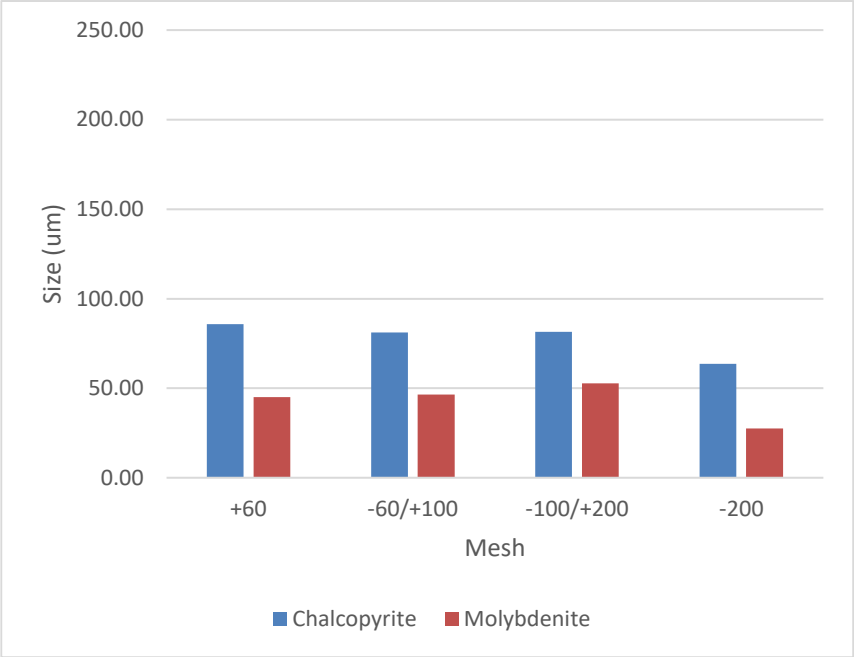
Source: Plenge, 2024

Figure 13-7: Average Chalcopyrite and Molybdenite Particle Sizes – M2



Source: Plenge, 2024

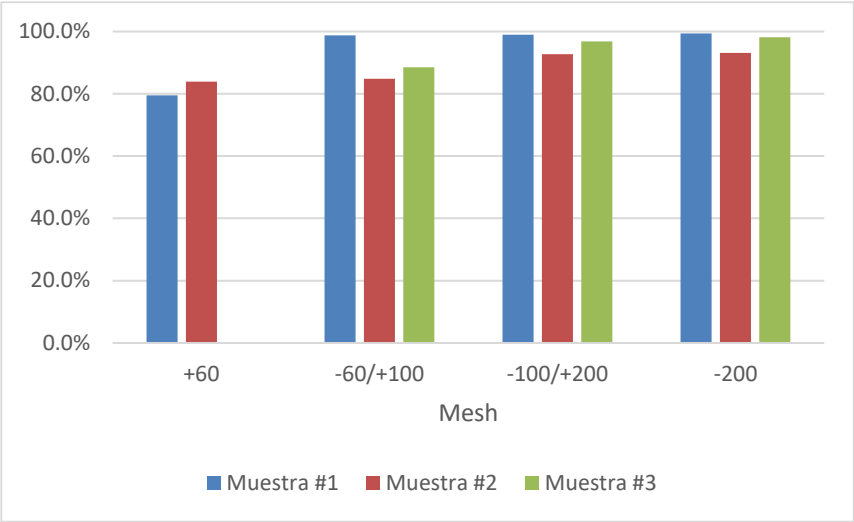
Figure 13-8: Average Chalcopyrite and Molybdenite Particle Sizes – M3



Source: Plenge, 2024

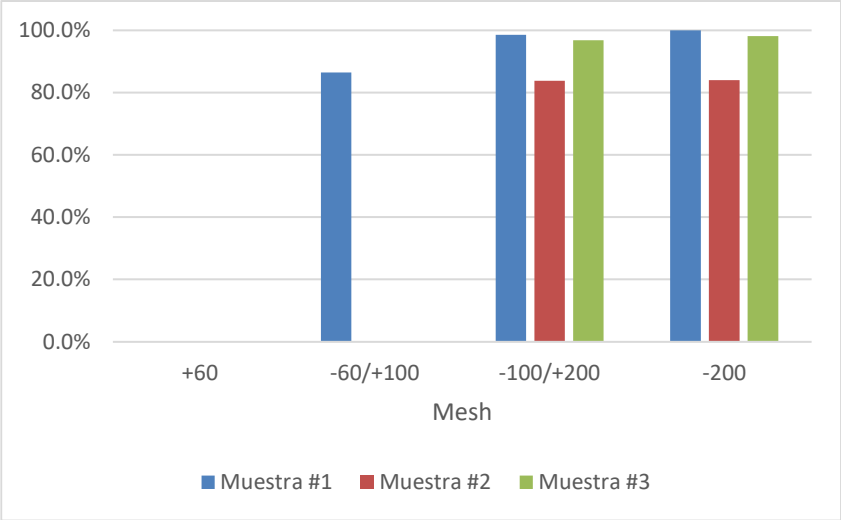
Liberation extents for all three samples (M1, M2, and M3) were developed for each of the particle size ranges and are presented in Figure 13-9 and Figure 13-10.

Figure 13-9: Chalcopyrite Liberation



Source: Plenge, 2024

Figure 13-10: Molybdenite Liberation



Source: Plenge, 2024

According to the data, M1 has excellent liberation for chalcopyrite at the -60/+100 mesh while molybdenite was liberated at the -100/+200 mesh size. For M2 and M3, liberation of chalcopyrite and molybdenite occurs at the -100/+200 mesh size.

13.2.2 Metallurgical Testing

Plenge conducted metallurgical testing on the M1, M2, and M3 samples in 2023 and 2024 and reported findings on 27 August 2024. The testing established sample head assays and examined mineralogy in addition to focusing on development of a first-pass flotation flowsheet to produce copper and molybdenum concentrates of saleable quality. Table 13-4 shows assays for the composite metallurgical samples.

Table 13-4: Metallurgical Sample Assays – M1, M2, and M3

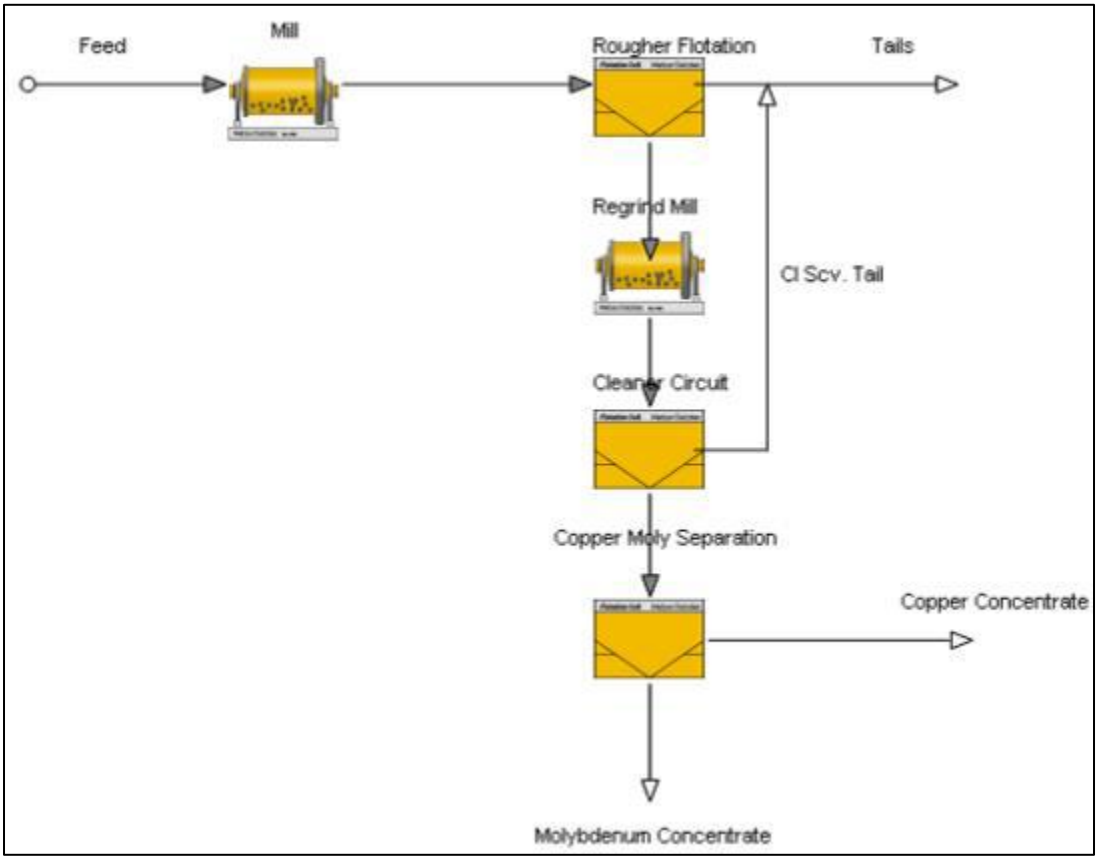
Element	M-1	M-2	M-3
Ag g/t	3.00	6.90	0.50
Au g/t	0.01	0.01	0.01
Cu %	0.21	0.62	0.07
Fe %	1.41	2.59	1.15
Mo %	0.0566	0.0517	0.2015
S total %	1.17	2.33	1.21

Source: Plenge, 2024

Testing established a flotation circuit arrangement to produce a bulk Cu-Mo concentrate and then to separate as a Mo concentrate and a Cu concentrate (see Figure 13-11 and Figure 13-12). The test also targeted to optimize operating parameters, including flotation reagent types and dosages.

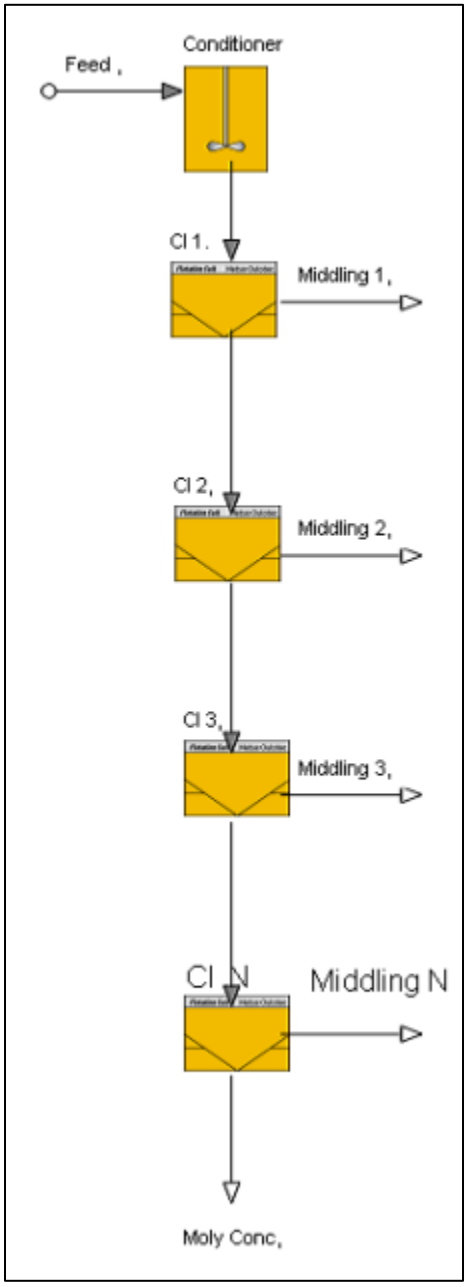
The circuit arrangement is typical for production of a bulk sulfide concentrate, with primary grinding to a P_{80} size of 150 µm, rougher flotation using PAX (potassium amyl xanthate) and SIBX (sodium isobutyl xanthate) collectors and Aerofloat®208 (a dithiophosphate flotation promoter), regrinding of rougher concentrate to P_{80} size of 38 µm and Cu-Mo separation flotation using four-stage cleaner flotation and NaHS (sodium hydrosulfide) as sulfide depressant.

Figure 13-11: Bulk Cu-Mo Flotation Circuit, Followed by Cu-Mo Separation Circuit



Source: Plenge, 2024

Figure 13-12: Cu-Mo Separation Flowsheet



Source: Plenge, 2024

A series of reagent selection tests were conducted using a Master Composite (MC). The Master Composite was prepared by blending equal parts of each of the three composites; M-1, M-2, and M-3. Rougher reagent collection and dosage was explored by seven flotation kinetic tests. The tests were run at grind of P_{80} size of 106 µm to reduce mineral liberation effects. All tests were run at pH=9.5 using 10 g/t fuel oil as molybdenum collector. The results of the tests are shown in Table 13-5. It should be noted that no grind vs recovery analysis was conducted in the 2024 or 2025 flotation test work. The primary grind selection was established based on the mineral liberation study.

Table 13-5: Collector Selection Tests MC Tests

Collector	Distribution (%)											
	T= 6 min				T=9 min				T= 12 min			
	Wt (%)	Ag	Cu	Mo	Wt (%)	Ag	Cu	Mo	Wt (%)	Ag	Cu	Mo
A-3894	8.4	83.9	95.3	92.5	11.1	85.3	95.9	93.3	13.8	86.1	96.4	93.9
Z-6	10.8	86.5	96.1	93.0	12.6	87.8	96.7	93.6	14.8	88.3	97.1	94.2
A-404	7.6	81.6	94.9	92.1	10.7	83.8	95.8	93.1	12.2	85.7	96.5	93.7
A-208	8.5	80.2	95.3	92.2	10.3	81.8	96.1	92.9	12.4	82.6	96.7	93.4
Z-3	7.1	80.4	96.2	91.2	8.5	81.8	96.7	91.9	9.3	83.1	97.1	92.2
Z-11	6.9	81.1	95.9	91.8	8.2	82.5	96.6	92.5	9.6	83.1	97.1	93.1
Z-14	6.7	81.0	96.3	91.9	7.9	81.9	96.8	92.5	9.9	82.5	97.3	93.3

Source: Plenge, 2024

The tests suggest that A-208, fuel oil and mix of PAX and SIBX are the best collectors for copper, molybdenum and silver in the MC. The dosages selected can be found in the Plenge report (C. H. Plenge & CIA. S.A., 2024).

The optimum conditions for rougher flotation for the MC are listed below (Test 32C):

- Primary Grind P80= 133 µm
- pH=9.5
- Collectors
    - Xanthate SIBX (Z-14) 27 g/ t
    - A-208 11 g/ t
    - F. Oi l 10g/ t
- Frother MIBC 17 g/ t
- Flotation Time 12 min

A series of batch cleaning tests were conducted examining various regrind times. A regrind particle size of 38 µm was selected as the optimum.

A six-cycle locked cycle test (LCT) program using the flowsheet shown in Figure 13-11 was conducted. Summarized results of the testing are shown in Table 13-7.

- M1 material produced a concentrate grading 17.7% Cu, 3.8% Mo, and 153 g/t Ag and achieved 81.8% Cu recovery, 62.4% Mo recovery, and 53.3% Ag recovery.
- M2 material produced a concentrate grading 29.5% Cu, 1.52% Mo, and 234 g/t Ag and achieved 82.1% Cu recovery, 50.5% Mo recovery, and 68.0% Ag recovery. The low Mo recovery was noted by Plenge who recommended further testing using this sample.
- M3 material produced a concentrate grading 7.5% Cu, 28.0% Mo, and 57.7 g/t Ag and achieved 85.6% Cu recovery, 89.8% Mo recovery, and 61.0% Ag recovery.

Cu and Mo recoveries achieved were reasonable for indicative-level flotation testing and would be expected to improve with further testing and development of the circuit and parameters.

Table 13-6: Bulk Concentrate Flotation Results – LCT Testing

Composite	Wt %	RC	Ag %	Cu %	Fe %	Mo %	S Total %	% Distribution		
								Ag	Cu	Mo
M-1	0.95	104.77	153	17.7	31.3	3.8	42.2	53.3	81.8	62.4
M-2	1.74		234	29.5	28.5	1.52	36.1	68	82.1	50.5
M-3	0.65	152.83	57.7	7.5	19.3	28	41.3	61	85.6	89.8
Average	1.12		148	18.2	26.4	11.1	39.9	60.8	83.2	67.6

Note: RC = Repeatability Cycle
Source: Plenge, 2024

Separation testing on Cu-Mo bulk concentrate from M1 and M3 was successfully completed using the four-stage flowsheet shown in Figure 13-13. Summarized results are shown in Table 13-7. Mo concentrate grades of 52.6% and 49.7% were achieved, which are close to commercial grades and can be expected to improve with further development testing. Mo overall recoveries were 61.2% for M1 and 84.0% for M3. A stable LCT trial using M2 could not be conducted, presumably due to the slow-floating nature of the Mo species present.

Table 13-7: Cu-Mo Separation Flotation Results – Open Circuit Testing

Sample	Product	Wt %	Ag %	Cu %	Fe %	Mo %	S %	% Distribution		
								Ag	Cu	Mo
M1	Conc Mo	0.1	0.5	0.8	1.6	52.6	38.2	0.0	0.2	61.2
	Conc Cu-Mo	1.4	113.9	13.0	36.3	2.8	44.9	62.5	87.7	69.6
M3	Conc Mo	0.4	0.5	0.5	2.6	49.7	39.1	0.3	2.6	84.0
	Conc Cu-Mo	1.0	32.0	6.9	23.9	21.9	42.5	62.2	92.6	96.1

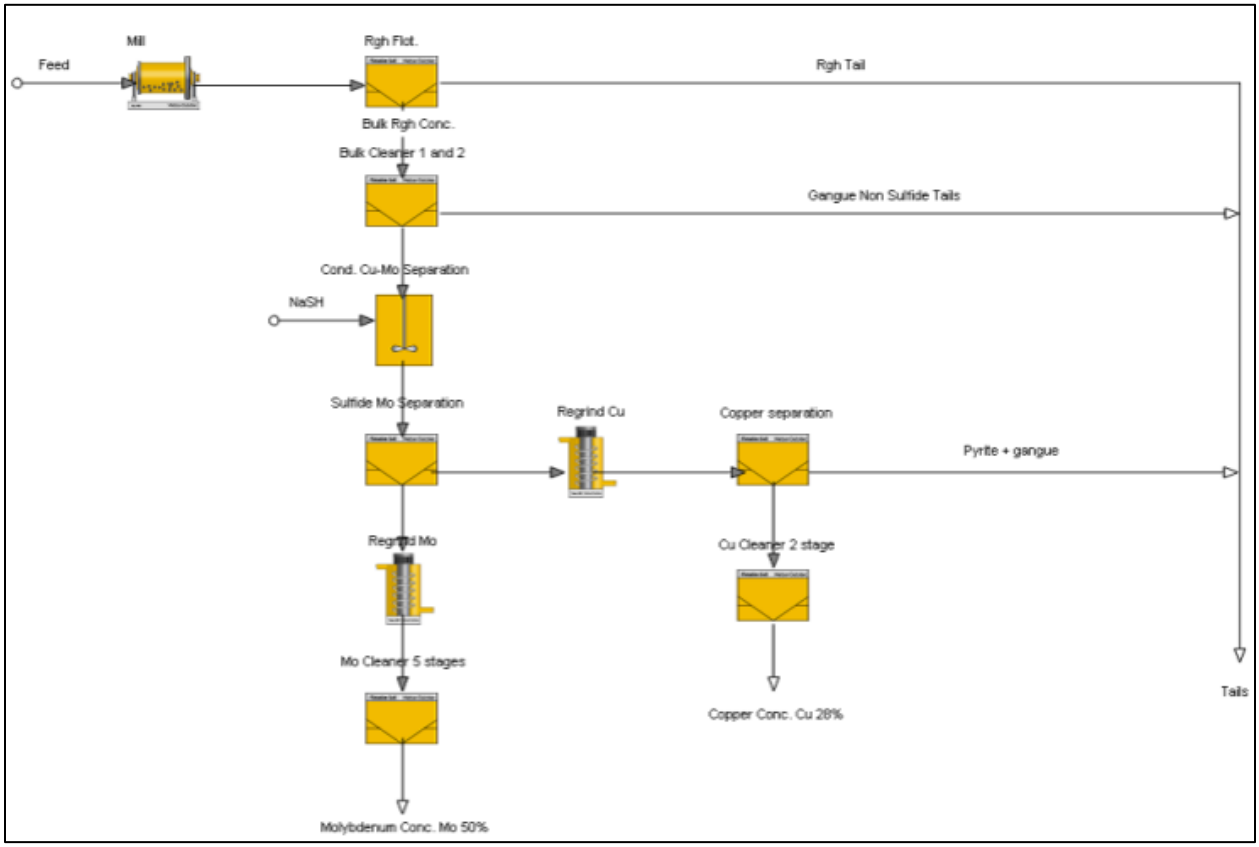
Source: Plenge, 2024

13.2.3 Metallurgical Testing – M2

Plenge conducted a further series of flotation tests using the M2 composite sample to further optimize the production of saleable grade Mo concentrate. A more complex, upgraded flowsheet was employed as shown in Figure 13-13. The arrangement consists of the following basic circuits:

- Ball mill grinding to P_{80} size of 150 µm.
- Bulk rougher flotation to produce Cu-Mo concentrate.
- Two-stage cleaner flotation with tailings discard.
- Cu-Mo separation by flotation with NaHS.
- Mo concentrate regrind to P_{80} size of 43 µm.
- Five-stage Mo concentrate cleaning.
- Cu concentrate regrind to P_{80} size of 25 µm.
- Cu concentrate re-float with pyrite and silicate gangue depression.

Figure 13-13: M2 Upgraded Flowsheet – Open Circuit Testing



Source: Plenge, 2024

Table 13-8 shows the results of the most successful open-circuit test (No.12). Mo concentrate reached a grade of 49.7%, which is comparable to the M1 and M3 results and could be expected to improve further in an LCT trial. Mo recovery of 78.0% was achieved, which is also comparable with M1 and M3. Cu concentrate with a grade of 28.7% was achieved at an overall recovery of 74.4%.

Table 13-8: Open Circuit Test Result Summary – Test No. 12

Product	Wt %	Assays					Distribution %				
		Ag %	Cu %	Fe %	Mo %	S %	Ag	Cu	Fe	Mo	S
Conc. Mo	0.09	12.5	0.24	2.42	49.7	35.2	0.18	0.03	0.08	78	1.2
Cl. Middling Mo	0.37	109	5.6	17.9	1.31	19.16	6.6	3.3	2.6	8.7	2.9
Con. Cu	1.6	229	28.7	30.4	0.07	38.7	60.3	74.4	19.4	2	25.3
C Middling Cu	3.6	37	3	40.4	0.05	44.9	21.9	16.2	58.1	3	66
Cl. Middling Bulk	3	7.1	0.31	1.84	0.03	0.58	3.5	1.5	2.2	1.7	0.7
Tail	91.4	0.5	0.03	0.48	0.004	0.11	7.5	4.6	17.6	6.4	3.9
Head Calc.	100	6.1	0.62	2.5	0.055	2.45	100	100	100	100	100
Head Assay		6.9	0.62	2.6	0.057	2.33					

Source: Plenge, 2024

13.2.4 Potentially Deleterious Elements in Concentrates

Table 13-9 shows assays for potentially deleterious elements in the concentrates produced during the testing program. All the concentrations of these elements are below typical penalty levels.

Table 13-9: Potentially Deleterious Elements in Concentrates

Element	Unit	M1		M2		M3	
		Mo con	Cu con	Mo con	Cu con	Mo con	Cu con
Cl	%	<0.01	<0.01	<0.01	0.21	0.05	0.04
F	%	0.02	<0.01	-	-	0.50	<0.01
As	%	0.014	0.19	0.016	0.089	0.01	0.09
Mg	%	0.03	0.02	-	-	0.03	0.02
Zn	%	0.02	0.34			0.02	0.31
Pb	%	-	-	0.02	0.05	-	-
Cu	%	-	-	0.20	-	-	-
Si	%	-	-	4.10	0.20	-	-
Hg	ppm	0.19	0.15	0.10	0.10	0.15	0.66

Source: Plenge, 2024

13.3 2025 Metallurgical Program

A metallurgical investigation was conducted on twelve samples representing four distinct geometallurgical units (High Cu (Cu), High Mo (Mo), Hornfels (HF), Enriched (EN)) from the Aurora Porphyry Copper-Molybdenum Project to assess their comminution, flotation, and separation characteristics. The samples exhibited variable grades, with copper content ranging from 0.06% to 0.55%, molybdenum from 172 ppm to 1,751 ppm, and silver from 0.7 g/t to 3.8 g/t. Notably, the MC-EN sample contained arsenic, and MC-HF was rich in fluorine. Table 13-10 shows the master composite head assays.

Table 13-10: Master Composites Head Assays

Element	Units	Composite Cu	Composite Mo	Composite HF	Composite EN
Ag	g/t	2.6	0.7	3.8	3.7
Au	g/t	0.009	0.007	0.012	0.016
Cu	%	0.34	0.06	0.41	0.55
Fe	%	1.84	1.23	5.26	1.9
Mo	ppm	267	1751	274	172
S total	%	1.21	0.67	2.05	2.19
C total	%	0.11	0.06	0.21	0.02
F	%	0.61	0.8	1.22	0.84
Hg	ppm	0.04	0.02	0.02	0.03
Cl	ppm	<20	<20	<20	<20
As	ppm	22	<5	17	83
Al	%	5.53	5.3	7.75	5.42
Ca	%	0.52	0.74	1.28	0.28
K	%	5.5	5.4	4.6	4.35
Mg	%	0.11	0.1	1.78	0.29
Mn	%	0.01	0.01	0.03	0.01
Na	%	0.63	0.74	0.61	0.27
Pb	%	0	0	0	0.01
Ti	%	0.02	0.03	0.31	0.06
Zn	%	0.02	0.01	0.02	0.01
As	ppm	<5	<5	<5	<5
Ba	ppm	1292	892	1420	609

Element	Units	Composite Cu	Composite Mo	Composite HF	Composite EN
Be	ppm	2	2	5	4
Bi	ppm	<5	<5	<5	<5
Cd	ppm	<5	<5	<5	<5
Co	ppm	<5	<5	<5	<5
Cr	ppm	<2	<2	<2	<2
Ni	ppm	10	7	94	15
P	ppm	223	204	919	339
Sb	ppm	<5	<5	<5	<5
Sc	ppm	1	1	13	3
Sn	ppm	8	<5	<5	<5
Sr	ppm	<5	<5	<5	<5
V	ppm	10	11	172	32
W	ppm	12	11	5	43
Y	ppm	<1	<1	<1	<1
Zr	ppm	2	2	3	2

Source: Plenge, 2024

An additional set of samples was also selected for variability testing (

Table 13-11).

Table 13-11: Variability Sample Head Assays

Element	Units	Cu-M-Y400989	Cu-H-Y400990	Cu-L-Y400991	Mo-M-Y400992	Mo-L-Y400993	Mo-H-Y400994	HF-M-Y400995	HF-L-Y400996	HF-H-Y400997	EN-L-Y400998	EN-M-Y400999	EN-H-Y401000
Ag	g/t	2	3.7	2	0.5	0.5	1	3	1.5	7	5.2	3	3
Au	g/t	0.007	0.011	0.009	0.006	0.007	0.007	0.012	0.012	0.012	0.016	0.016	0.017
Cu	%	0.22	0.57	0.22	0.08	0.02	0.09	0.35	0.22	0.67	0.58	0.45	0.61
Fe	%	1.99	1.66	1.86	1.41	1.13	1.14	4.59	5.21	5.99	1.9	2.26	2.59
Mo	ppm	191	596	14	1659	1337	2258	654	149	20	828	491	23
S total	%	1.41	0.83	1.39	0.55	0.62	0.85	1.11	1.92	3.1	2.59	2.33	1.65
C total	%	0.12	0.14	0.07	0.04	0.08	0.05	0.08	0.1	0.44	0.02	0.01	0.03
F	%	1.1	0.2	0.52	0.3	1.1	0.99	1.34	1.34	0.99	1.1	0.56	0.86
Hg	ppm	0.07	0.04	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.04	0.02	0.02
Cl	ppm	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20
G.E.	g/cc	2.65	2.65	2.66	2.57	2.59	2.66	2.78	2.84	2.89	2.67	2.65	2.77
Al	%	5.76	5.37	5.45	5.66	5.4	4.84	7.02	7.85	8.37	4.48	4.44	7.33
Ca	%	1.08	0.21	0.27	0.41	1.03	0.77	1.19	1.71	0.93	0.43	0.16	0.26
K	%	4.91	5.57	6.01	5.24	5.61	5.36	5.32	4.25	4.22	4.56	3.99	4.49
Mg	%	0.13	0.12	0.09	0.11	0.09	0.1	1.96	1.99	1.4	0.1	0.13	0.64
Mn	%	0.01	0.01	<0.01	0.01	0.01	0.01	0.03	0.03	0.02	0.01	0.01	0.01
Na	%	0.76	0.7	0.43	1.18	0.55	0.49	0.84	0.82	0.16	0.14	0.11	0.57
Pb	%	0.01	0	0	0	0.01	0	0	0	0	0	0.01	0.01
Ti	%	0.03	0.02	0.02	0.04	0.02	0.02	0.28	0.39	0.25	0.02	0.02	0.13
Zn	%	0.03	0.02	0.02	0.01	0.01	<0.01	0.02	0.01	0.02	0.02	0.01	0.01
As	ppm	34	11	22	<5	<5	<5	<5	30	22	36	108	104
Ba	ppm	1244	1003	1629	907	889	879	2014	666	1579	546	461	820
Be	ppm	3	1	2	2	2	2	4	6	6	3	3	5
Bi	ppm	<5	<5	<5	<5	<5	<5	<5	<5	<5	5	<5	10
Cd	ppm	6	6	5	5	5	5	6	5	6	6	6	6
Co	ppm	17	9	11	8	4	9	23	30	29	19	27	16
Cr	ppm	34	30	26	32	44	32	113	118	102	47	55	67
Ni	ppm	5	7	17	6	8	7	122	87	72	6	8	30
P	ppm	227	220	223	200	198	213	929	1016	811	106	297	615
Sb	ppm	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5
Sc	ppm	1	1	1	1	1	1	12	13	15	1	1	8
Sn	ppm	13	5	6	<5	<5	<5	6	7	9	15	12	10
Sr	ppm	282	204	152	248	225	169	170	186	74	92	137	129

Element	Units	Cu-M-Y400989	Cu-H-Y400990	Cu-L-Y400991	Mo-M-Y400992	Mo-L-Y400993	Mo-H-Y400994	HF-M-Y400995	HF-L-Y400996	HF-H-Y400997	EN-L-Y400998	EN-M-Y400999	EN-H-Y401000
V	ppm	12	12	7	12	11	9	211	134	170	17	18	60
W	ppm	13	11	13	9	9	14	7	<1	3	9	14	105
Y	ppm	<1	<1	<1	<1	4	1	4	5	<1	<1	<1	<1
Zr	ppm	2	2	2	2	3	2	2	3	3	2	2	3

Source: Plenge, 2024

13.3.1 Mineralogy

Modal Mineralogy using AMICS was conducted on four size fractions of each composite to provide quantitative mineral identification, and association, particle sizes as well as liberation information:

Key findings:

- Data Correlation: Excellent agreement between AMICS-calculated copper assays and provided wet chemistry data across all samples.
- Modal Mineralogy: Dominant minerals vary by group. High Cu and High Mo samples are primarily quartz and orthoclase, with minor muscovite and albite. High Fluorine samples feature quartz, biotite, muscovite, and orthoclase, with minor albite and almandine (not explicitly listed but inferred). High Enargite samples are mainly quartz and orthoclase, with minor muscovite, pyrite, and pyrrhotite. Copper-bearing minerals identified are chalcopyrite, bornite, and chalcocite.
- Copper Deportment: In high copper, high molybdenum, and high fluorine groups, copper is predominantly present as chalcopyrite (>90-100%), with minor bornite and chalcocite in some fractions. In High Enargite, copper is more evenly distributed among chalcopyrite (19-74%), bornite (29-71%), and chalcocite (5-49%).
- Liberation: Chalcopyrite liberation is generally high (>90%) in finer fractions across most groups but varies. High Copper shows high liberation in -100/+200 mesh except for low-copper subsamples. High Moly is well-liberated overall. High Fluorine has high liberation only in high-copper subsamples, with lower in mid/low. High enargite copper sulfides liberate at ~85%+ in -200/+400 mesh, with variability by mineral (chalcopyrite often highest).
- Particle Size: Average sizes for copper minerals decreased with finer fractions (e.g., chalcopyrite from ~100-130 µm in +100 mesh to ~11-20 µm in -400 mesh).
- Mineral Associations: Chalcopyrite in non-enargite groups associates mainly with quartz, orthoclase, and muscovite. In enargite, chalcopyrite associates with bornite and chalcocite. Bornite and chalcocite primarily associate with each other and chalcopyrite.
- These results are similar to those obtained in the 2024 analysis.

Based on these results the required grind sizes were developed as shown in Table 13-12.

Table 13-12: Grind Sizes Suggested by Mineral Liberation (AMICS)

Zone	Grind P_{80} µm	Rougher Flotation	Regrind P_{80} µm	Cleaner Flotation
Cu	140-160	Good Cu liberation early; Mo follows.	30-35	Consistent >95% in finest fractions.
Mo	160-180	Mo liberates coarser Cu secondary.	28-32	Fine for high-grade Mo separation.
HF	130-150	Variable Cu; moderate grind balances.	30-35	Achieves 95% sulfide mass purity.
EN	100-120	Intergrowths demand finer primary.	25-30	Finest to break locks for cleaners.

Source: Plenge, 2024

13.3.2 Comminution

Comminution tests indicated that the ore is generally soft to medium soft, with an average JK-SMC A x b= 38.6, Bond Ball Work Index (BWi) of 11.66 kWh/t and Abrasion Index 0.260g.

Table 13-13: Ball Bond Work Index

Sample	Ai		BWi*
	P_{80} µm	grams	kWh/t
Composite Cu	13359	0.3187	11.71
Composite Mo	13310	0.3704	10.53
Composite HF	12952	0.1058	12.31
Composite EN	11859	0.2426	12.08
Average	12870	0.2594	11.66
Standard Deviation	699.0	0.1151	0.79

*BWI, Work index in kilowatts-hour per metric ton, Mesh Size=150 microns

Source: Plenge, 2024

The average BWi ranged from 10.53 to 12.31 kWh/t metric which denotes an ore that can be characterized as medium soft with no significant variations between the four composites. The Ai abrasion index ranged from 0.1058 g to 0.3704 g. The results of the comminution testing are found in Table 13-14 and Table 13-15.

Table 13-14: JK-SMC Test Results

Sample Designation	*DWi* (kWh/m^3)	*DWi* (%)	*Mi* Parameters (kWh/t)				Estimated
			Mia	Mih	Mic	SG	Mib
Composite Cu	6.75	51.0	19.8	14.7	7.6	2.66	15.0
Composite EN	5.63	36.0	16.9	12.1	6.2	2.70	14.0
Composite HF	10.13	88.0	25.4	20.4	10.6	2.88	18.0
Composite Mo	6.68	50.0	19.7	14.6	7.5	2.66	15.0

Source: Plenge, 2024

Table 13-15: JK-SMC Summary

Sample	A	b	A*b	Ta	SCSE kWh/t
Composite Cu	85.5	0.46	39.3	0.38	9.88
Composite Mo	91.6	0.43	39.4	0.46	9.87
Composite HF	70.4	0.40	28.2	0.25	12.27
Composite EN	74.6	0.64	47.7	0.38	9.12
Average	80.5	0.48	38.7	0.37	10.29
Standard Deviation	9.7	0.11	8.0	0.09	1.37

Source: Plenge, 2024

Alex Doll Consulting utilized these results to size the comminution circuit (see Section 17.0). The copper and molybdenum domains are the main material sources and they have similar hardness. The design of the grinding circuit needs to consider the impact of ore blending on the available grinding energy.

13.3.3 Metallurgical Testing

A series of flotation tests were conducted on the original four composites as well as the additional variability samples. The flotation parameters mimicked those employed in the 2024 test program. A series of open circuit and locked-cycle rougher flotation tests were conducted. The results of the locked cycle tests are shown in Table 13-16.

Table 13-16: Lock Cycle Rougher Test Summary

Sample	Wt. %	Feed Assays					Concentrate Assays					% Distribution		
		Ag g/t	Cu %	Fe %	Mo %	S total %	Ag g/t	Cu %	Fe %	Mo %	S total %	Ag	Cu	Mo
Cu	6.54	2.6	0.34	1.84	0.0267	1.21	30.5	5.05	17.06	0.38	16.43	91.4	97.8	93.1
Mo	4.79	0.7	0.06	1.23	0.1751	0.67	8.8	1.03	12.73	3.92	14.89	69.5	94.2	97.1
HF	11.12	3.8	0.41	5.26	0.0274	2.05	28.6	3.66	19.25	0.24	15.54	90.0	98.3	88.5
EN	8.99	3.7	0.55	1.9	0.0172	2.19	33.6	5.54	16.6	0.51	19.06	90.4	90.4	95.4
Average	7.86	2.7	0.34	2.6	0.0616	1.53	25.4	3.82	16.41	1.26	16.48	85.3	95.2	93.5

Source: Plenge, 2024

The various composites performed well for both copper and molybdenum recovery. Silver recovery was excellent for all of the composites except for the Mo. The typical copper-molybdenum flowsheet relies on the bulk flotation of both metals and a high rougher recovery is critical to the overall performance of the circuit.

The results from the locked cycle tests were employed to test batch cleaner flotation to upgrade the copper and molybdenum concentrations in the cleaner concentrate. Ultimately, the cleaner concentrate is subjected to copper and molybdenum separation. The locked cycle cleaner tests are shown in Table 13-17.

Table 13-17: Locked Cycle Cleaner Test Summary

Composite	Wt. %	Feed Assays							Concentrate Assays							% Distribution		
		Ag g/t	Cu %	Fe %	Mo %	S %	F %	Hg ppm	Ag g/t	Cu %	Fe %	Mo %	S %	F %	Hg ppm	Ag	Cu	Mo
Cu	1.05	2.6	0.34	1.84	0.0267	1.21	0.61	0.04	158.1	30.09	29.1	1.74	34.43	0.08	0.12	77.0	93.4	69.0
Mo	0.54	0.7	0.06	1.23	0.1751	0.67	0.8	0.02	56	8.63	13.4	34.25	38.49	0.1	0.08	50.3	88.2	90.7
HF	1.76	3.8	0.41	5.26	0.0274	2.05	1.22	0.02	170.3	22.9	33.7	1.47	35.89	0.14	0.07	86.2	97.0	85.3
EN	1.73	3.7	0.55	1.9	0.0172	2.19	0.84	0.03	155	27.05	28	2.56	38.31	0.07	0.09	83.0	85.0	92.6
Avg	1.27	2.7	0.34	2.6	0.0616	1.53	0.87	0.03	134.8	22.17	26.03	10.01	36.78	0.1	0.09	74.1	90.9	84.4

Source: Plenge, 2024

Standard locked cycle flotation achieved satisfactory overall average recoveries of 90.9% for copper, 84.4% for molybdenum, and 74.1% for silver into a copper-molybdenum concentrate with R.C 78.7:1. The primary grind was P_{80} of 150 µm, regrind P_{80} of 30 µm in a typical rougher flotation followed by regrind of rougher concentrate and cleaners.

The low grade of the copper in the Mo composite resulted in a lower overall copper recovery and similarly in the EN composite the high grade copper resulted in slightly lower overall copper recovery, likely due to a slower floating mineral such as chalcocite or enargite. The same is true for the molybdenum, lower grade Mo in the Cu composite resulted in lower Mo recovery to the concentrate.

Copper-molybdenum separation was successfully demonstrated using NaHS at pH 9 with multiple cleaning stages coupled with magnetic separation. Commercial copper concentrates were obtained assaying from 20.5-36.8% copper, 153-231g/t silver while molybdenum concentrate grades spanned from

48.4-56.6% molybdenum. The feed for the separation test was obtained from a large test. An open circuit separation test was conducted as shown in Table 13-18.

Table 13-18: Open Circuit Copper-Molybdenum Separation Test Summary

Composite	Concentrate	Wt. %	Assay Ag g/t	Assay Cu %	Assay Mo %	Assay S %	Assay F %	% Distribution Ag	% Distribution Cu	% Distribution Mo
Cu	Conc Mo	0.03	27.0	2.98	48.92	36.59	0.11	0.4	0.3	53.0
	Conc Cu	0.91	153.4	30.36	0.37	36.66	0.04	67.4	86.9	11.9
Mo	Conc Mo	0.27	8.2	0.76	56.09	39.64	0.08	4.0	4.6	82.0
	Conc Cu	0.16	130.6	20.47	7.61	36.81	0.13	37.3	74.0	6.6
HF	Conc Mo	0.02	28.2	2.81	48.4	36.34	0.13	0.2	0.2	44.2
	Conc Cu	1.22	231.4	28.29	0.2	35.85	0.09	77.4	83.6	8.9
EN	Conc Mo	0.05	6.0	0.35	56.58	40.13	0.08	0.1	0.0	56.8
	Conc Cu	1.06	200.3	36.79	0.55	35.24	0.07	62.6	69.5	12.1

Source: Plenge, 2024

The recovery of molybdenum to the molybdenum concentrate for the copper composite was low at 53%. A similar result was exhibited for the Hornfels composite and the Enriched composite. Higher recoveries of Mo are possible despite the lower grades, but it may require a finer regrind to obtain liberation or a longer flotation retention time due to the slow floating nature of molybdenite. There was no indication provided as to the effectiveness or necessity of the magnetic separation stage.

13.3.4 Potentially Deleterious Elements In Concentrates

The main potential penalty elements are fluorine for all concentrates except for the Cu copper concentrate. Arsenic is notable only for EN copper concentrate but within commercial limits.

Table 13-19: Penalty Elements

Element	Units	Composite Cu Conc Cu	Composite Cu Conc Mo	Composite Mo Conc Cu	Composite Mo Conc Mo	Composite HF Conc Cu	Composite HF Conc Mo	Composite EN Conc Cu	Composite EN Conc Mo
Cl	ppm	<20	<20	<20	<20	<20	<20	<20	<20
F	%	0.04	0.11	0.13	0.08	0.09	0.13	0.07	0.08
Hg	ppm	0.08	0.07	0.21	0.08	0.07	0.04	0.09	0.08
As	%	0.033	0.008	0.028	0.004	0.004	0.013	0.114	0.006
Mg	%	0.02	0.11	0.06	0.03	0.05	0.19	0.04	0.03
Zn	%	0.73	0.34	0.96	0.04	0.55	0.15	1.05	0.02

Source: Plenge, 2024

Sodium silicate, sodium hexametaphosphate and quebracho were used for fluorine depression in the copper/molybdenum cleaning stages in LCT-2 for composites MC-Cu and MC-Mo with mixed success as fluorine assays were not low enough to avoid penalties but the molybdenum recovery was negatively impacted. The fluorine content in the copper concentrate can be reduced to acceptable levels by depressing fluorite using depressants such as CMC (carboxymethyl cellulose), Quebracho, and sodium silicate.

In many instances excess fluorine is the result of mechanical entrainment rather than mineralogical entrainment. In most cases, a more dilute flotation with a wash system will eliminate this issue (Jameson

cell). In extreme cases, an acid leach can be utilized to remove the fluorine entrainment. Fluorine in the molybdenum concentrate was effectively reduced from 0.1% to 0.04% by sulfuric acid at 60°C. The leach also increased the molybdenum concentrate grade from 56% to 58% because of the removal of carbonates and alkaline salts with negligible molybdenum losses.

Further investigation is required to determine the fluorine deportment.

13.3.5 Variability Testing

An additional three samples for each domain were selected for confirmation testing. The rougher flotation recoveries of variability samples agreed closely to that obtained for the master composites. The overall average recoveries for silver, copper and molybdenum were 80.0%, 94.1% and 94.3%, respectively (Table 13-20).

Table 13-20: Summary Rougher Concentrate Variability

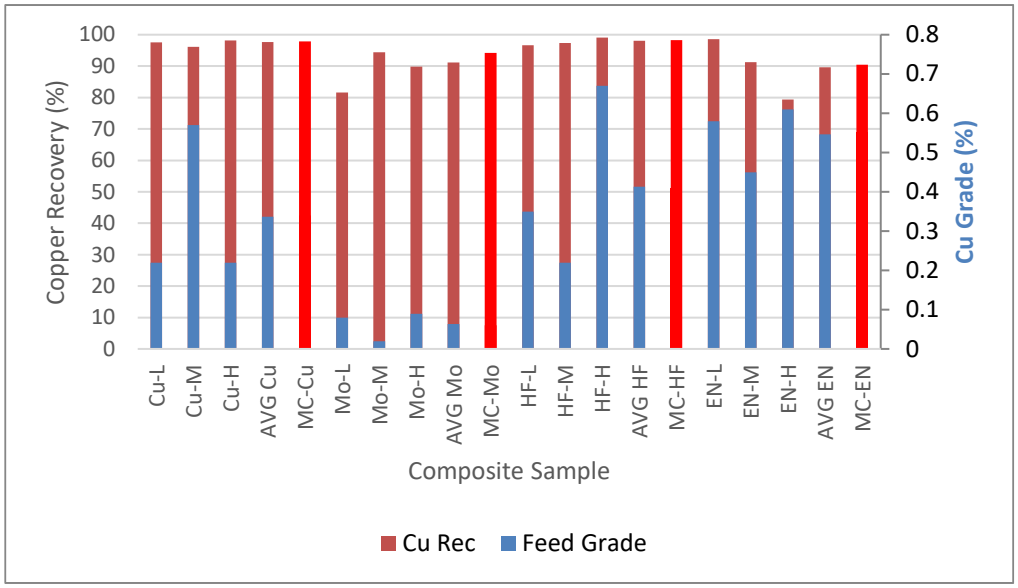
		Feed Assays					Concentrate Assays					Distribution		
Sample	Wt. %	Ag g/t	Cu %	Fe %	Mo %	S total %	Ag g/t	Cu %	Fe %	Mo %	S total %	Ag	Cu	Mo
Cu-L	7.85	2.00	0.22	1.99	0.0191	1.41	20.5	2.77	18.9	0.03	16	77.6	97.5	69.7
Cu-M	6.9	3.70	0.57	1.66	0.0596	0.83	21.9	3.03	17.1	0.27	17.7	88.9	96.1	89.8
Cu-H	7.3	2.00	0.22	1.86	0.0014	1.39	40.7	7.68	13.5	0.73	13.6	90.2	98.2	93
AVG	7.35	2.57	0.34	1.84	0.0267	1.21	27.7	4.49	16.5	0.34	15.8	86.2	97.6	91.3
MC-Cu	**6.54**	**2.6**	**0.34**	**1.84**	**0.0267**	**1.21**	**30.5**	**5.05**	**17.06**	**0.38**	**16.43**	**91.4**	**97.8**	**93.1**
Mo-L	5.2	0.50	0.08	1.41	0.1659	0.55	3.45	0.25	7.5	2.55	9.3	47.8	81.6	97.3
Mo-M	4.8	0.50	0.02	1.13	0.1337	0.62	10.6	1.5	11.9	3.78	14.5	57.9	94.4	97.9
Mo-H	5.1	1.00	0.09	1.14	0.2258	0.85	13.6	1.47	8.9	4.76	12.3	74.9	89.8	97.8
AVG	5.03	0.67	0.06	1.23	0.17513	0.67	9.18	1.07	9.4	3.69	12	63.5	91.1	97.7
MC-Mo	**4.79**	**0.7**	**0.06**	**1.23**	**0.1751**	**0.67**	**8.8**	**1.03**	**12.73**	**3.92**	**14.89**	**69.5**	**94.2**	**97.1**
HF-L	8.97	3.00	0.35	4.59	0.0654	1.11	15	2.36	19.9	0.16	17.4	79.4	96.6	89.7
HF-M	8.7	1.50	0.22	5.21	0.0149	1.92	32.2	4.18	16.7	0.79	13.7	89.5	97.3	95.9
HF-H	13.87	7.00	0.67	5.99	0.002	3.1	46.7	4.88	23.9	0.01	18.6	96.2	99.1	67
AVG	10.51	3.83	0.41	5.26	0.027	2.03	31.3	3.81	20.2	0.32	16.5	91.9	98.1	94
MC-HF	**11.12**	**3.8**	**0.41**	**5.26**	**0.0274**	**2.05**	**28.6**	**3.66**	**19.25**	**0.24**	**15.54**	**90**	**98.3**	**88.5**
EN-L	10.2	5.20	0.58	1.9	0.0828	2.59	49	5.91	19.8	0.89	23.7	94.1	98.6	97.8
EN-M	7.8	3.00	0.45	2.26	0.0491	2.33	32.3	5.09	23.3	0.62	14.4	59.9	91.2	90.2
EN-H	8.6	3.00	0.61	2.59	0.0023	1.65	25.8	5.59	12.4	0.02	13.5	68.6	79.4	56.2
AVG	8.87	3.73	0.55	2.25	0.044	2.19	35.7	5.53	18.5	0.51	17.2	78.3	89.6	94.3
MC-EN	**8.99**	**3.7**	**0.55**	**1.9**	**0.0172**	**2.19**	**33.6**	**5.54**	**16.6**	**0.51**	**19.06**	**90.4**	**90.4**	**95.4**
AVG Overall	**7.94**	**2.70**	**0.34**	**2.64**	**0.07**	**1.53**	**26**	**3.72**	**16.1**	**1.2**	**15.4**	**80**	**94.1**	**94.3**

Source: Plenge, 2024

The results are shown graphically in Figure 13-14 and Source: GRE, 2026

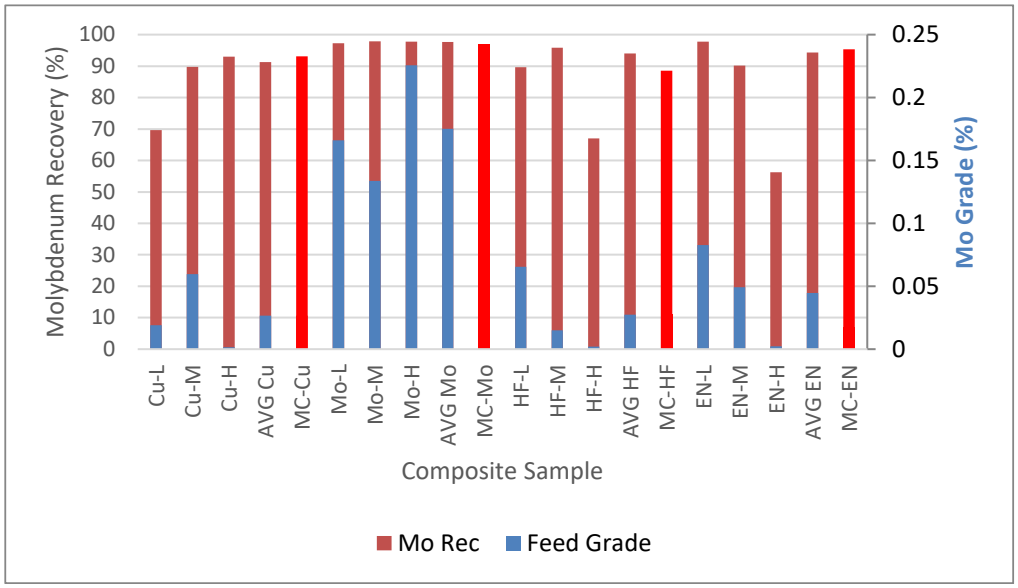
Figure 13-15.

Figure 13-14: Copper Recovery and Feed Grade of Variability and Master Composite Rougher Tests



Source: GRE, 2026

Figure 13-15: Molybdenum Recovery and Feed Grade of Variability and Master Composite Rougher Tests

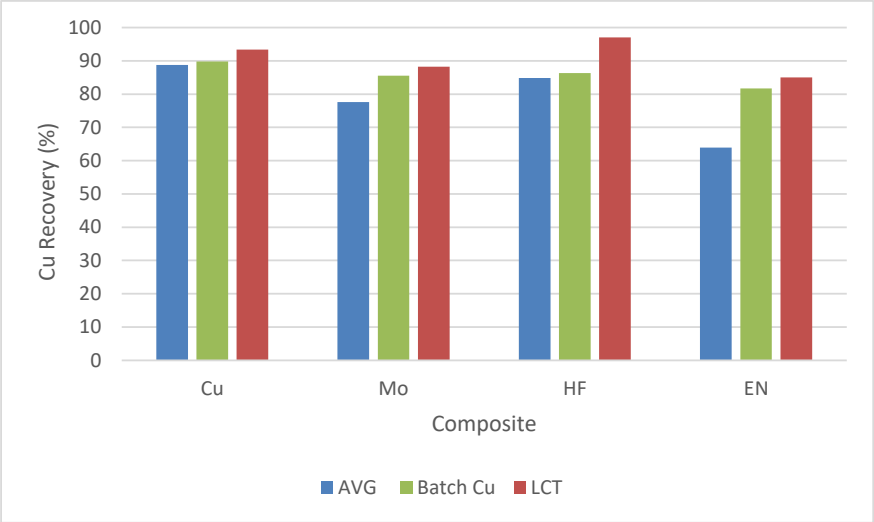


Source: GRE, 2026

In general, the variability samples responded as predicted. They again highlight the impact of copper and molybdenum feed grades on rougher recovery. Lower grade materials tend to have a lower overall rougher recovery. This is most likely related to liberation as with a lower grade, a greater proportion of metal is in the unliberated fraction.

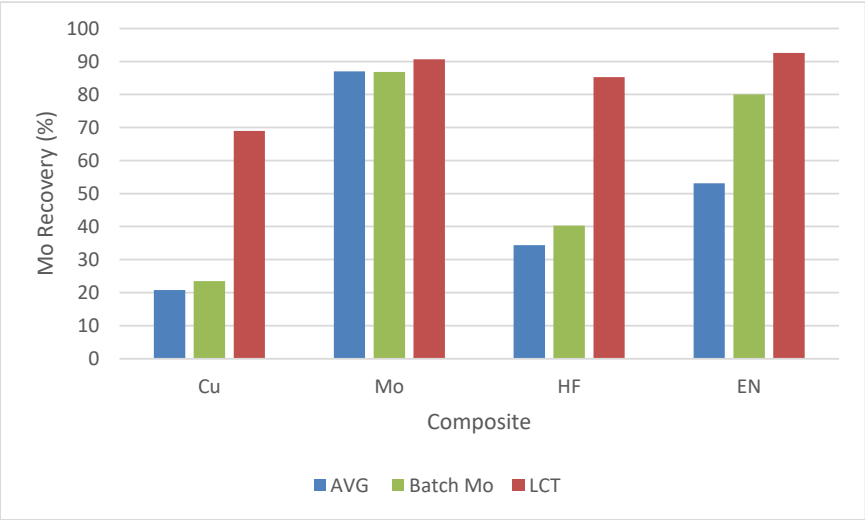
In cleaner testing, the variability tests performed very similar to the locked cycle tests for copper but not for molybdenum. The reason for this is because the molybdenum circuit tends to have a large circulating load of molybdenum and in open circuit tests this is not recovered. These results are compared to LCT tests graphically in Figure 13-16 and Figure 13-17.

Figure 13-16: Variability and Locked Cycle Cleaner Test Comparison - Copper



Source: GRE, 2026

Figure 13-17: Variability and Locked Cycle Cleaner Test Comparison - Molybdenum



Source: GRE, 2026

13.4 Recoveries

The previous technical report (AMC, 2025) employed an averaged recovery for each metal and did not include any ore type dependency. This was based on the work conducted on the M1, M2, and M3 composites, which were not ore type specific but rather blends. Further, no variability testing was conducted so it was impossible to develop a relationship to feed grades. Unfortunately, this resulted in an overstatement of the overall recovery and in particular silver. Silver reports to both the copper and molybdenum concentrates but it is only payable in the copper concentrate and thus, the overall payable recovery is reduced.

The previous recoveries employed are as follows:

- Copper – 85.6%
- Molybdenum – 84.0%

- Silver – 60.8%

With additional testing, domain specific composites and variability samples, a better representation of the metal recoveries is possible. Further, to develop a process flow diagram, more variables need to be defined including concentrate mass pull, concentrate grades, and impurity levels. Although there was a significant amount of test work conducted, there were still not enough tests to develop a comprehensive geometallurgical analysis. However, statistical analysis (regression analysis) was employed to develop the best relationships possible. The typical comparisons were employed including recovery versus mass pull, recovery versus grade, grade versus mass pull for each domain.

The result was a series of equations that were employed to develop a net smelter return within the block model to allow for the development of the mine plan.

Rougher Mass Pull (x = Cu Grade)

Domain	Mass Pull
Cu	y = 1.4294x + 6.6926
Mo	y = 5.313x + 4.6404
HF	y = 12.196x + 5.6341
EN	y = 19.585x - 0.9731
LE	y = 1.4294x + 6.6926

Rougher Cu Recovery

Domain	Rougher Cu Rec	Intercept	MP Factor	Cu Grade Factor
Cu	Rec = int + MP * factor + Cu Grade * factor	-29.75	19.25	-32.5
Mo	Rec = int + MP * factor + Cu Grade * factor	246.5721005	-32.1011741	48.8158
HF	Rec = int + MP * factor + Cu Grade * factor	90.25863147	0.99847667	-7.47245
EN	Rec = int + MP * factor + Cu Grade * factor	91.0	1.0	-11.0
LE	Rec = int + MP * factor + Cu Grade * factor	71.87	4.6158795	0.0

Rougher Mo Recovery (x = copper recovery)

Domain	Rougher Moly Rec
Cu	y = 1.6475x - 68.44
Mo	y = 0.0485x + 93.367
HF	y = -9.0331x + 968.94
EN	y = 2.2659x - 118.81
LE	y = 1.6475x - 68.44

Rougher Ag Recovery (x = copper recovery)

Domain	Rougher Ag Rec

Cu	y = 0.8713x + 5.3298
Mo	y = 1.2097x - 46.345
HF	y = 5.8312x - 481.66
EN	y = 1.3643x - 37.691
LE	y = 0.8713x + 5.3298

Domain	Cu Cleaner Mass Pull	Cln/Rgh Cu
Cu	MP Cu Cln = Rougher MP * Cln/Rgh Cu	0.145259939
Mo	MP Cu Cln = Rougher MP * Cln/Rgh Cu	0.104384134
HF	MP Cu Cln = Rougher MP * Cln/Rgh Cu	0.10971223
EN	MP Cu Cln = Rougher MP * Cln/Rgh Cu	0.117908788
LE	MP Cu Cln = Rougher MP * Cln/Rgh Cu	0.145259939

Cleaner Bulk Concentrate Cu Recovery

Domain	Cu Cleaner Recovery	Cu factor
Cu	Cu Rougher Recovery * factor	0.96
Mo	Cu Rougher Recovery * factor	0.92
HF	Cu Rougher Recovery * factor	0.96
EN	Cu Rougher Recovery * factor	0.90
LE	Cu Rougher Recovery * factor	0.94

Cleaner Bulk Concentrate Mo Recovery

Domain	Mo Cleaner Recovery	Mo factor
Cu	Mo Rougher Recovery * factor	0.85
Mo	Mo Rougher Recovery * factor	0.95
HF	Mo Rougher Recovery * factor	0.85
EN	Mo Rougher Recovery * factor	0.90
LE	Mo Rougher Recovery * factor	0.85

Cleaner Bulk Concentrate Ag Recovery

Domain	Ag Cleaner Recovery to Cu Conc	Ag Factor
Cu	Ag Rougher Recovery * factor	0.91
Mo	Ag Rougher Recovery * factor	0.72
HF	Ag Rougher Recovery * factor	0.90
EN	Ag Rougher Recovery * factor	0.90
LE	Ag Rougher Recovery * factor	0.91

Final Concentrate Mo Recovery

Domain	Mo Cleaner Recovery	Final Conc Factor

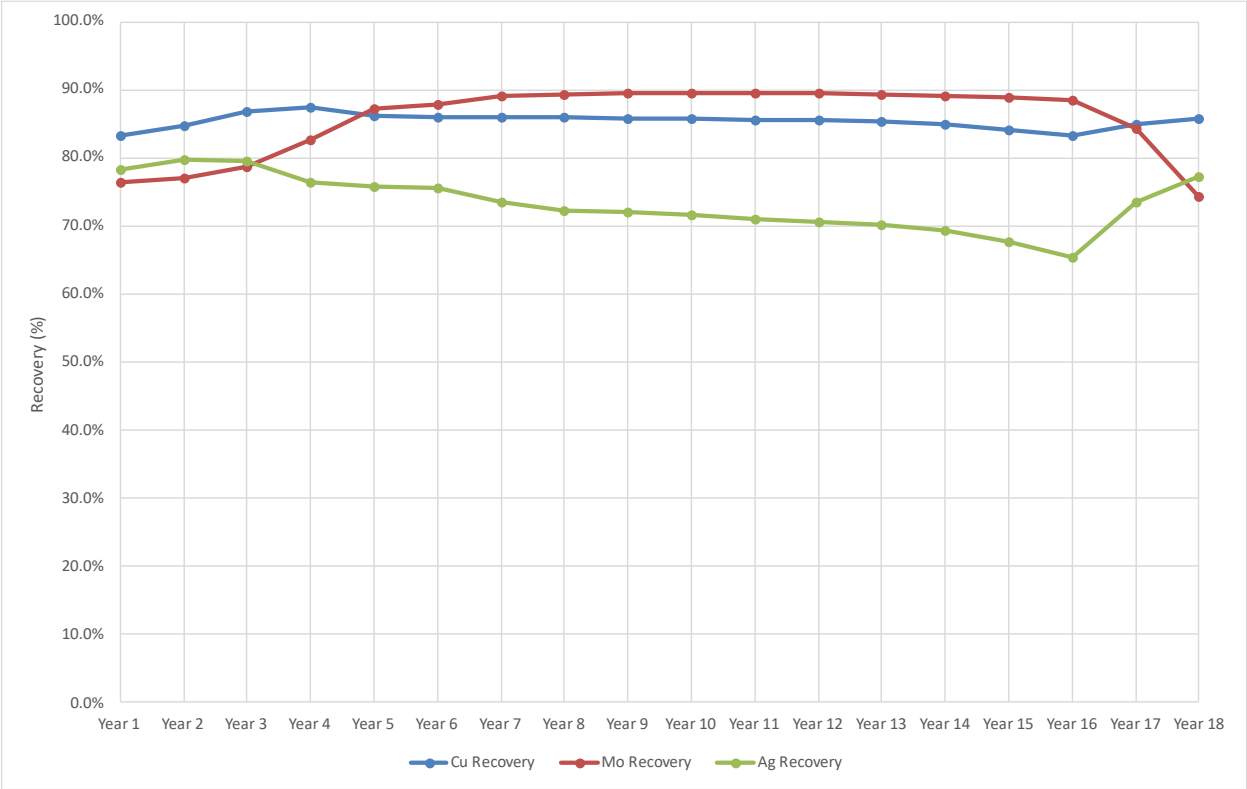
Cu	Mo Bulk Recovery * factor	0.93
Mo	Mo Bulk Recovery * factor	0.97
HF	Mo Bulk Recovery * factor	0.93
EN	Mo Bulk Recovery * factor	0.95
LE	Mo Bulk Recovery * factor	0.95

Final Concentrate Cu/Ag Rejection

Domain	Cu/Ag Rejection in Mo Circuit
Cu	0.99
Mo	0.98
HF	0.98
EN	0.98
LE	0.99

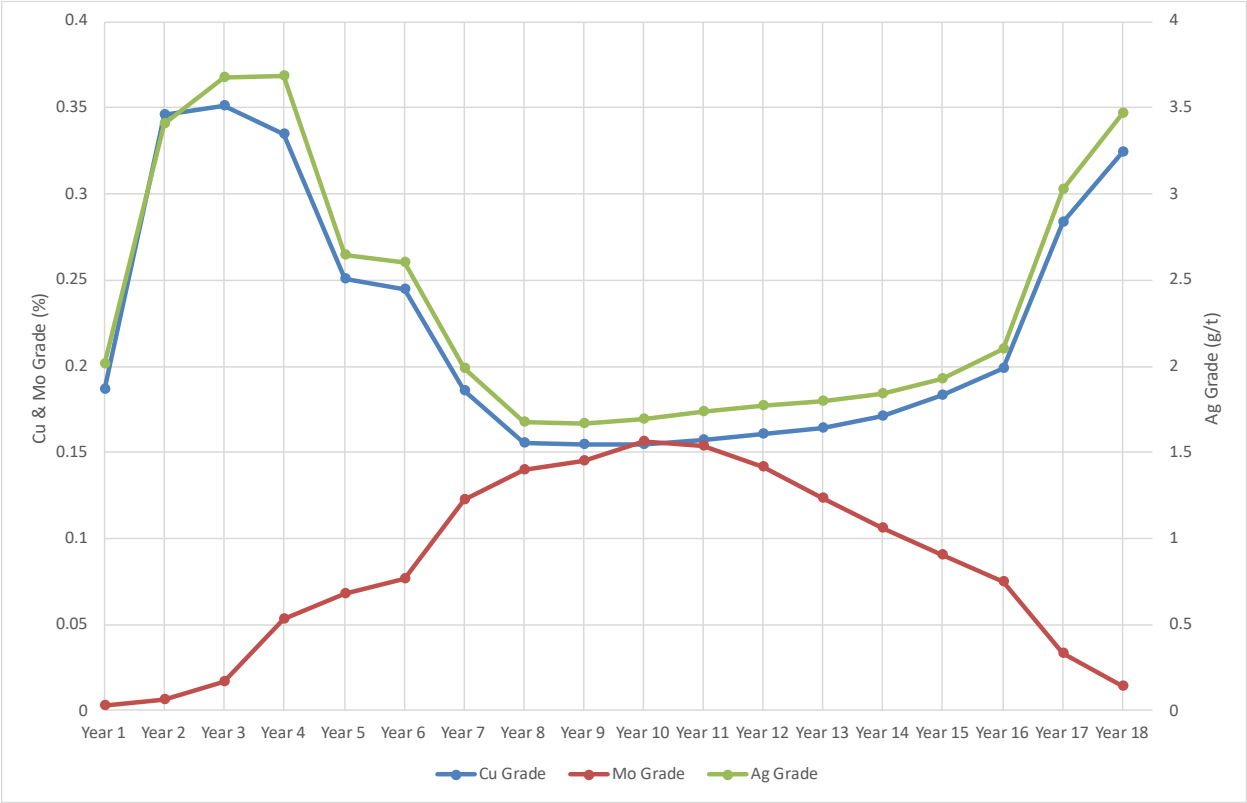
Based on the mine plan developed the annual recoveries are shown in Figure 13-18 and the feed grades are show in Figure 13-19. The LOM average recoveries for copper, silver, and molybdenum are 85.5%, 88.5% and 74.0%, respectively.

Figure 13-18: Aurora Project Concentrator Flotation Recovery



Source: GRE, 2026

Figure 13-19: Aurora Project Concentrator Feed Grades



Source: GRE, 2026

13.5 Recommendations

The following recommendations have been provided to aid in the advancement of the project to the next study level.

- Conduct additional comminution testing with increased sample sets and include the rod mill work index. HPGR may be applicable for this project and the rod mill work index (RMWi) is one variable that helps identify its suitability.
- Conduct additional primary grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Conduct additional regrind grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Additional testing examining reagent suits, particularly for the molybdenum cleaning phase to depress fluorine.
- Examine the use of Jameson or Nova flotation cells to assist in fluorine rejection in the molybdenum circuit.
- Additional testing of high-grade molybdenum materials particularly from future underground areas.
- Additional variability testing, including a wide range of distributions: spatial, domains, and grades.

- Impact of material weathering of flotation performance given the potential long duration stockpile storage envisioned in the production plan.
- Conduct additional test work to define the mechanism for fluorine entrainment.
- Conduct solid/liquid separation testing to develop design criteria for thickening and filtration.
- Conduct transportable moisture limit (TML) testing to define the maximum permissible moisture content of the concentrates for sea shipment.

13.6 Sample Drill Holes

Table 13-21: 2023 Sample Sources

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
Partial Leached_M1					
Y330866	A23-004	28.00	30.00	2.00	Leached
Y330869	A23-004	34.00	36.00	2.00	Leached
Y330870	A23-004	36.00	38.00	2.00	Leached
Y330873	A23-004	40.00	42.00	2.00	Leached
Y330875	A23-004	44.00	46.00	2.00	Leached
Y330877	A23-004	48.00	50.00	2.00	Leached
Y330878	A23-004	50.00	52.00	2.00	Leached
Y330879	A23-004	52.00	54.00	2.00	Leached
Y330880	A23-004	54.00	56.00	2.00	Leached
Y330884	A23-004	62.00	64.00	2.00	Leached
Y330887	A23-004	68.00	70.00	2.00	Leached
Y330889	A23-004	72.00	74.00	2.00	Leached
Y330892	A23-004	76.00	78.00	2.00	Leached
Y330896	A23-004	84.00	86.00	2.00	Leached
Y330900	A23-004	92.00	94.00	2.00	Leached
Y330907	A23-004	106.00	108.50	2.50	Leached
Y331230	A23-005	24.00	26.00	2.00	Leached
Y331232	A23-005	28.00	30.00	2.00	Leached
Y331234	A23-005	32.00	34.00	2.00	Leached
Y331235	A23-005	34.00	36.00	2.00	Leached
Y331238	A23-005	38.00	40.00	2.00	Leached
Y331239	A23-005	40.00	42.00	2.00	Leached
Y331243	A23-005	48.00	50.00	2.00	Leached
Y331250	A23-005	62.00	64.00	2.00	Leached
Y331251	A23-005	64.00	66.00	2.00	Leached
Y331252	A23-005	66.00	68.00	2.00	Leached
Y331269	A23-005	98.00	100.00	2.00	Leached
Y331270	A23-005	100.00	102.00	2.00	Leached
Y331274	A23-005	108.00	110.00	2.00	Leached
Y331593	A23-006	38.00	40.00	2.00	Leached
Y331595	A23-006	42.00	45.00	3.00	Leached
Y331596	A23-006	45.00	47.00	2.00	Leached
Y331597	A23-006	47.00	49.00	2.00	Leached
Y331598	A23-006	49.00	51.00	2.00	Leached
Y331603	A23-006	58.00	60.00	2.00	Leached

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
Y331607	A23-006	69.00	71.00	2.00	Leached
Y331610	A23-006	75.00	77.00	2.00	Leached
Y331611	A23-006	77.00	79.00	2.00	Leached
Y331621	A23-006	95.00	97.00	2.00	Leached
Y331633	A23-006	119.00	121.00	2.00	Enriched
Mixed/Enriched_M2					
Y330909	A23-004	110.90	113.00	2.10	Enriched
Y330912	A23-004	115.00	117.50	2.50	Enriched
Y330924	A23-004	140.00	142.00	2.00	Enriched
Y330927	A23-004	146.00	148.00	2.00	Enriched
Y330933	A23-004	156.00	158.00	2.00	Enriched
Y330934	A23-004	158.00	160.00	2.00	Enriched
Y330936	A23-004	162.00	164.00	2.00	Enriched
Y330937	A23-004	164.00	166.00	2.00	Enriched
Y330961	A23-004	210.00	212.00	2.00	P_Moly_Porphyry
Y330967	A23-004	222.00	224.00	2.00	P_Moly_Porphyry
Y330969	A23-004	226.00	228.00	2.00	P_Moly_Porphyry
Y330973	A23-004	232.00	234.00	2.00	P_Moly_Porphyry
Y330974	A23-004	234.00	236.00	2.00	P_Moly_Porphyry
Y330978	A23-004	242.00	244.00	2.00	P_Moly_Porphyry
Y330979	A23-004	244.00	246.00	2.00	P_Moly_Porphyry
Y330980	A23-004	246.00	248.00	2.00	P_Moly_Porphyry
Y330981	A23-004	248.00	250.00	2.00	P_Moly_Porphyry
Y330985	A23-004	256.00	258.00	2.00	P_Moly_Porphyry
Y330986	A23-004	258.00	260.00	2.00	P_Moly_Porphyry
Y330987	A23-004	260.00	262.00	2.00	P_Moly_Porphyry
Y331294	A23-005	146.00	148.00	2.00	Enriched
Y331295	A23-005	148.00	150.00	2.00	Enriched
Y331296	A23-005	150.00	152.00	2.00	Enriched
Y331301	A23-005	158.00	160.00	2.00	Enriched
Y331302	A23-005	160.00	162.00	2.00	Enriched
Y331303	A23-005	162.00	164.00	2.00	Enriched
Y331304	A23-005	164.00	166.00	2.00	Enriched
Y331313	A23-005	182.00	184.00	2.00	Enriched
Y331618	A23-006	91.00	93.00	2.00	Enriched
Y331623	A23-006	99.00	101.00	2.00	Enriched
Y331636	A23-006	125.00	127.00	2.00	Enriched
Y331639	A23-006	131.00	133.00	2.00	Enriched
Y331671	A23-006	191.00	193.00	2.00	Enriched
Y331675	A23-006	199.00	201.00	2.00	Enriched
Y331693	A23-006	231.00	233.00	2.00	P_Moly_Porphyry
Y332488	A23-008	343.00	345.00	2.00	P_Moly_Porphyry
Y332492	A23-008	351.00	353.00	2.00	P_Moly_Porphyry
Y332846	A23-009	325.00	327.00	2.00	P_Copper_Porphyry
Primary Moly Rich_M3					
Y331111	A23-004	497.00	499.00	2.00	P_Moly_Porphyry
Y331112	A23-004	499.00	502.00	3.00	P_Moly_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
Y331117	A23-004	510.00	512.00	2.00	P_Moly_Porphyry
Y331118	A23-004	512.00	514.00	2.00	P_Moly_Porphyry
Y331123	A23-004	520.00	522.00	2.00	P_Moly_Porphyry
Y331186	A23-004	640.00	642.00	2.00	P_Moly_Porphyry
Y331187	A23-004	642.00	644.00	2.00	P_Moly_Porphyry
Y331188	A23-004	644.00	646.00	2.00	P_Moly_Porphyry
Y331468	A23-005	478.00	480.00	2.00	P_Moly_Porphyry
Y331469	A23-005	480.00	482.00	2.00	P_Moly_Porphyry
Y331470	A23-005	482.00	484.00	2.00	P_Moly_Porphyry
Y331477	A23-005	496.00	498.00	2.00	P_Moly_Porphyry
Y331478	A23-005	498.00	500.00	2.00	P_Moly_Porphyry
Y331550	A23-005	635.00	637.00	2.00	P_Moly_Porphyry
Y331551	A23-005	637.00	639.00	2.00	P_Moly_Porphyry
Y331563	A23-005	661.00	663.00	2.00	P_Moly_Porphyry
Y331565	A23-005	665.00	667.00	2.00	P_Moly_Porphyry
Y331830	A23-006	500.00	502.00	2.00	P_Moly_Porphyry
Y331833	A23-006	506.00	508.00	2.00	P_Moly_Porphyry
Y331836	A23-006	512.00	514.00	2.00	P_Moly_Porphyry
Y331848	A23-006	534.00	536.00	2.00	P_Moly_Porphyry
Y331849	A23-006	536.00	538.00	2.00	P_Moly_Porphyry
Y332559	A23-008	480.00	482.00	2.00	P_Moly_Porphyry
Y332561	A23-008	482.00	484.00	2.00	P_Moly_Porphyry
Y332571	A23-008	502.00	504.00	2.00	P_Moly_Porphyry
Y332581	A23-008	520.00	522.00	2.00	P_Moly_Porphyry
Y332588	A23-008	534.00	536.00	2.00	P_Moly_Porphyry
Y332609	A23-008	574.00	576.00	2.00	P_Moly_Porphyry
Y332612	A23-008	580.00	582.00	2.00	P_Moly_Porphyry
Y332636	A23-008	626.00	628.00	2.00	P_Moly_Porphyry
Y332649	A23-008	650.00	652.00	2.00	P_Moly_Porphyry
Y332672	A23-008	694.00	696.00	2.00	P_Moly_Porphyry
Y333505	A23-010	789.00	791.00	2.00	P_Moly_Porphyry
Y333506	A23-010	791.00	793.00	2.00	P_Moly_Porphyry
Y333508	A23-010	795.00	797.00	2.00	P_Moly_Porphyry
Y333509	A23-010	797.00	799.00	2.00	P_Moly_Porphyry
Y333513	A23-010	803.00	805.00	2.00	P_Moly_Porphyry
Y333514	A23-010	805.00	807.00	2.00	P_Moly_Porphyry
Y333515	A23-010	807.00	809.00	2.00	P_Moly_Porphyry
Y333517	A23-010	811.00	813.00	2.00	P_Moly_Porphyry
Y333520	A23-010	817.00	819.00	2.00	P_Moly_Porphyry
Y333524	A23-010	825.00	827.50	2.50	P_Moly_Porphyry
Y333528	A23-010	835.00	837.50	2.50	P_Moly_Porphyry
Y333596	A23-010	964.00	966.00	2.00	P_Moly_Porphyry
Y333597	A23-010	966.00	968.00	2.00	P_Moly_Porphyry
Y333602	A23-010	976.00	978.00	2.00	P_Moly_Porphyry
Y333605	A23-010	982.00	984.00	2.00	P_Moly_Porphyry
Y333607	A23-010	986.00	988.00	2.00	P_Moly_Porphyry
Y333994	A23-011	726.00	728.00	2.00	P_Copper_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
Y334008	A23-011	752.00	754.00	2.00	P_Copper_Porphyry
Y334017	A23-011	768.00	770.00	2.00	P_Copper_Porphyry
Y334025	A23-011	784.00	786.00	2.00	P_Copper_Porphyry
Y334035	A23-011	804.00	806.00	2.00	P_Copper_Porphyry
Y334048	A23-011	828.00	830.00	2.00	P_Copper_Porphyry
Y334072	A23-011	874.00	876.00	2.00	P_Deep_Porphyry
Y334102	A23-011	930.00	932.00	2.00	P_Deep_Porphyry
Y334114	A23-011	954.00	956.00	2.00	P_Deep_Porphyry

Table 13-22: 2025 Sample Sources

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
Y401000	A22-001	89.1	91.3	2.2	Enriched
	A22-001	91.3	94.05	2.75	Enriched
	A22-001	94.05	96.05	2	Enriched
	A22-001	96.05	98.7	2.65	Enriched
	A22-001	98.7	100.35	1.65	Enriched
	A22-001	100.35	102.35	2	Enriched
	A22-001	102.35	104.65	2.3	Enriched
	A22-001	104.65	106.65	2	Enriched
	A22-001	106.65	108.65	2	Enriched
	A22-001	108.65	110.65	2	Enriched
	A22-001	110.65	112.65	2	Enriched
	A22-001	112.65	114.65	2	Enriched
	A22-001	114.65	116.65	2	Enriched
	A22-001	116.65	118.65	2	Enriched
	A22-001	118.65	120.65	2	Enriched
	A22-001	120.65	122.6	1.95	Enriched
	A22-001	122.6	124.3	1.7	Enriched
	A22-001	124.3	126.3	2	Enriched
	A22-001	126.3	128.3	2	Enriched
	A22-001	128.3	130.3	2	Enriched
	A22-001	130.3	132.3	2	Enriched
	A22-001	132.3	134.3	2	Enriched
	A22-001	134.3	135.35	1.05	Enriched
	A22-001	135.35	137.35	2	Enriched
	A22-001	137.35	139.35	2	Enriched
	A22-001	139.35	141.8	2.45	Enriched
	A22-001	141.8	143.8	2	Enriched
	A22-001	143.8	145.8	2	Enriched
	A22-001	145.8	147.8	2	Enriched
	A22-001	147.8	149.8	2	Enriched
	A22-001	149.8	151.8	2	Enriched
Y400998	A23-004	110.9	113	2.1	Enriched
	A23-004	113	115	2	Enriched
	A23-004	115	117.5	2.5	Enriched
	A23-004	117.5	120	2.5	Enriched

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A23-004	120	122	2	Enriched
	A23-004	122	124	2	Enriched
	A23-004	124	126	2	Enriched
	A23-004	126	128	2	Enriched
	A23-004	128	130	2	Enriched
	A23-004	130	132	2	Enriched
	A23-004	132	134	2	Enriched
	A23-004	134	136	2	Enriched
	A23-004	136	138	2	Enriched
	A23-004	138	140	2	Enriched
	A23-004	140	142	2	Enriched
	A23-004	142	144	2	Enriched
	A23-004	144	146	2	Enriched
	A23-004	146	148	2	Enriched
	A23-004	148	150	2	Enriched
	A23-004	150	152	2	Enriched
	A23-004	152	154	2	Enriched
	A23-004	154	156	2	Enriched
	A23-004	156	158	2	Enriched
	A23-004	158	160	2	Enriched
Y400999	A23-005	136	138	2	Enriched
	A23-005	138	140	2	Enriched
	A23-005	140	142	2	Enriched
	A23-005	142	144	2	Enriched
	A23-005	144	146	2	Enriched
Y400994	A23-006	647	649	0.11	P_Moly_Porphyry
	A23-006	649	651	0.67	P_Moly_Porphyry
	A23-006	651	653	0.47	P_Moly_Porphyry
	A23-006	653	655	0.65	P_Moly_Porphyry
	A23-006	655	657	0.98	P_Moly_Porphyry
	A23-006	657	659	1.04	P_Moly_Porphyry
	A23-006	659	661	1.3	P_Moly_Porphyry
	A23-006	661	663	1.39	P_Moly_Porphyry
	A23-006	663	665	1.73	P_Moly_Porphyry
	A23-006	665	667	0.36	P_Moly_Porphyry
	A23-006	667	669	1.24	P_Moly_Porphyry
	A23-006	669	671	0.88	P_Moly_Porphyry
	A23-006	671	673	0.73	P_Moly_Porphyry
	A23-006	673	675	2.34	P_Moly_Porphyry
	A23-006	675	677	0.99	P_Moly_Porphyry
	A23-006	677	679	0.52	P_Moly_Porphyry
	A23-006	679	681	0.82	P_Moly_Porphyry
	A23-006	681	683	0.64	P_Moly_Porphyry
	A23-006	683	685	0.47	P_Moly_Porphyry
	A23-006	685	687	0.53	P_Moly_Porphyry
	A23-006	687	689	0.68	P_Moly_Porphyry
	A23-006	689	691	0.62	P_Moly_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A23-006	691	693	0.65	P_Moly_Porphyry
	A23-006	693	695	0.84	P_Moly_Porphyry
	A23-006	695	697	1.4	P_Moly_Porphyry
	A23-006	697	699	0.63	P_Moly_Porphyry
	A23-006	699	701	0.76	P_Moly_Porphyry
	A23-006	701	703	0.45	P_Moly_Porphyry
	A23-006	703	705	0.54	P_Moly_Porphyry
	A23-006	705	707	0.61	P_Moly_Porphyry
	A23-006	707	708.5	0.85	P_Moly_Porphyry
Y400997	A23-007	206	208	5.59	Hornfels
	A23-007	208	210	23.6	Hornfels
	A23-007	210	212	32.9	Hornfels
	A23-007	212	214	4.3	Hornfels
	A23-007	214	216	1.84	Hornfels
	A23-007	216	218	3.38	Hornfels
	A23-007	218	220	1.7	Hornfels
	A23-007	220	222	3.58	Hornfels
	A23-007	222	224	2.14	Hornfels
	A23-007	224	226	2.28	Hornfels
	A23-007	226	228	1.12	Hornfels
	A23-007	228	230	11.3	Hornfels
	A23-007	230	232	6.44	Hornfels
	A23-007	232	234	1.96	Hornfels
	A23-007	234	236	1.52	Hornfels
	A23-007	236	238	3.42	Hornfels
	A23-007	238	240	4.5	Hornfels
	A23-007	240	242	2.92	Hornfels
	A23-007	242	244	1.6	Hornfels
	A23-007	244	246	0.88	Hornfels
	A23-007	246	248	1.13	Hornfels
	A23-007	248	250	2.08	Hornfels
	A23-007	250	252	2.9	Hornfels
	A23-007	252	254	4.88	Hornfels
	A23-007	254	256	5.39	Hornfels
Y400989	A23-007	566	568	2.15	P_Copper_Porphyry
	A23-007	568	570	1.38	P_Copper_Porphyry
	A23-007	570	572	1.72	P_Copper_Porphyry
	A23-007	572	574	2.08	P_Copper_Porphyry
	A23-007	574	576	4.18	P_Copper_Porphyry
	A23-007	576	578	6.07	P_Copper_Porphyry
	A23-007	578	580	1.62	P_Copper_Porphyry
	A23-007	580	582	4.04	P_Copper_Porphyry
	A23-007	582	584	3.16	P_Copper_Porphyry
	A23-007	584	586	2.24	P_Copper_Porphyry
	A23-007	586	588	1.01	P_Copper_Porphyry
	A23-007	588	590	1.35	P_Copper_Porphyry
	A23-007	590	593	0.85	P_Copper_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A23-007	593	594.9	4.04	P_Copper_Porphyry
	A23-007	594.9	596.9	0.75	P_Copper_Porphyry
	A23-007	596.9	599	1.88	P_Copper_Porphyry
	A23-007	599	601	2.7	P_Copper_Porphyry
	A23-007	601	603	2.67	P_Copper_Porphyry
	A23-007	603	605	0.85	P_Copper_Porphyry
	A23-007	605	607	1.65	P_Copper_Porphyry
	A23-007	607	609	1.13	P_Copper_Porphyry
	A23-007	609	611	0.66	P_Copper_Porphyry
	A23-007	611	613	0.94	P_Copper_Porphyry
	A23-007	613	615	3.04	P_Copper_Porphyry
	A23-007	615	617	1.37	P_Copper_Porphyry
	A23-007	617	619	0.83	P_Copper_Porphyry
	A23-007	619	621	1.32	P_Copper_Porphyry
	A23-007	621	623	1.53	P_Copper_Porphyry
	A23-007	623	625	1.23	P_Copper_Porphyry
Y400990	A23-012	456	458	1.73	P_Copper_Porphyry
	A23-012	458	460	2.27	P_Copper_Porphyry
	A23-012	460	462	2.3	P_Copper_Porphyry
	A23-012	462	464	2.55	P_Copper_Porphyry
	A23-012	464	466	2.36	P_Copper_Porphyry
	A23-012	466	468	3.93	P_Copper_Porphyry
	A23-012	468	470	4.35	P_Copper_Porphyry
	A23-012	470	472	3.43	P_Copper_Porphyry
	A23-012	472	474	3.15	P_Copper_Porphyry
	A23-012	474	476	2.85	P_Copper_Porphyry
	A23-012	476	478	4.75	P_Copper_Porphyry
	A23-012	478	480	8.8	P_Copper_Porphyry
	A23-012	480	482	3.14	P_Copper_Porphyry
	A23-012	482	484	3.05	P_Copper_Porphyry
	A23-012	484	486	4.27	P_Copper_Porphyry
	A23-012	486	488	4.94	P_Copper_Porphyry
	A23-012	488	490	4.96	P_Copper_Porphyry
	A23-012	490	492	3.86	P_Copper_Porphyry
	A23-012	492	494	2.4	P_Copper_Porphyry
	A23-012	494	496	3.85	P_Copper_Porphyry
	A23-012	496	498	3.42	P_Copper_Porphyry
	A23-012	498	500	3.65	P_Copper_Porphyry
	A23-012	500	502	3.53	P_Copper_Porphyry
	A23-012	502	504	3.7	P_Copper_Porphyry
	A23-012	504	506	3.28	P_Copper_Porphyry
	A23-012	506	508	2.22	P_Copper_Porphyry
Y400993	A24-015	919	921	0.19	P_Moly_Porphyry
	A24-015	921	923	0.19	P_Moly_Porphyry
	A24-015	923	925	0.12	P_Moly_Porphyry
	A24-015	925	927	0.4	P_Moly_Porphyry
	A24-015	927	929	0.24	P_Moly_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A24-015	929	931	0.2	P_Moly_Porphyry
	A24-015	931	933	0.28	P_Moly_Porphyry
	A24-015	933	935	0.46	P_Moly_Porphyry
	A24-015	935	937	0.73	P_Moly_Porphyry
	A24-015	937	939	0.1	P_Moly_Porphyry
	A24-015	939	941	0.07	P_Moly_Porphyry
	A24-015	941	943	0.13	P_Moly_Porphyry
	A24-015	943	945	0.18	P_Moly_Porphyry
	A24-015	945	947	0.19	P_Moly_Porphyry
	A24-015	947	949	0.18	P_Moly_Porphyry
	A24-015	949	951	0.14	P_Moly_Porphyry
	A24-015	951	953	0.1	P_Moly_Porphyry
	A24-015	953	955	0.15	P_Moly_Porphyry
	A24-015	955	957	0.56	P_Moly_Porphyry
	A24-015	957	959	2.07	P_Moly_Porphyry
	A24-015	959	961	0.07	P_Moly_Porphyry
	A24-015	961	963	0.07	P_Moly_Porphyry
	A24-015	963	965	1.46	P_Moly_Porphyry
	A24-015	965	967	0.07	P_Moly_Porphyry
	A24-015	967	969	0.2	P_Moly_Porphyry
	A24-015	969	971	0.07	P_Moly_Porphyry
Y400992	A24-016	1028	1030	1.74	P_Deep_Porphyry
	A24-016	1030	1032	1.04	P_Deep_Porphyry
	A24-016	1032	1034	0.44	P_Deep_Porphyry
	A24-016	1034	1036	0.34	P_Deep_Porphyry
	A24-016	1036	1038	0.35	P_Deep_Porphyry
	A24-016	1038	1040	0.37	P_Deep_Porphyry
	A24-016	1040	1042	0.5	P_Deep_Porphyry
	A24-016	1042	1044	0.46	P_Deep_Porphyry
	A24-016	1044	1046	0.44	P_Deep_Porphyry
	A24-016	1046	1048	0.26	P_Deep_Porphyry
	A24-016	1048	1050	0.81	P_Deep_Porphyry
	A24-016	1050	1052	0.21	P_Deep_Porphyry
	A24-016	1052	1054	0.18	P_Deep_Porphyry
	A24-016	1054	1056	0.24	P_Deep_Porphyry
	A24-016	1056	1058	0.28	P_Deep_Porphyry
	A24-016	1058	1060	0.66	P_Deep_Porphyry
	A24-016	1060	1062	0.4	P_Deep_Porphyry
	A24-016	1062	1064	0.57	P_Deep_Porphyry
	A24-016	1064	1066	0.99	P_Deep_Porphyry
	A24-016	1066	1068	0.5	P_Deep_Porphyry
	A24-016	1068	1070	0.37	P_Deep_Porphyry
	A24-016	1070	1072	0.52	P_Deep_Porphyry
	A24-016	1072	1074	0.36	P_Deep_Porphyry
	A24-016	1074	1076	0.65	P_Deep_Porphyry
Y400991	A24-018	364	366	1.14	P_Copper_Porphyry
	A24-018	366	368	0.86	P_Copper_Porphyry

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A24-018	368	370	1.35	P_Copper_Porphyry
	A24-018	370	372	1.56	P_Copper_Porphyry
	A24-018	372	374	1.49	P_Copper_Porphyry
	A24-018	374	376	1.47	P_Copper_Porphyry
	A24-018	376	378	2.47	P_Copper_Porphyry
	A24-018	378	380	1.53	P_Copper_Porphyry
	A24-018	380	382	1.25	P_Copper_Porphyry
	A24-018	382	384	2.16	P_Copper_Porphyry
	A24-018	384	386	1.04	P_Copper_Porphyry
	A24-018	386	388	2.19	P_Copper_Porphyry
	A24-018	388	390	1.87	P_Copper_Porphyry
	A24-018	390	392	1.94	P_Copper_Porphyry
	A24-018	392	394	2.51	P_Copper_Porphyry
	A24-018	394	396	3.42	P_Copper_Porphyry
	A24-018	396	398	2.67	P_Copper_Porphyry
Y400995	A24-018	496	498	2.61	Hornfels
	A24-018	498	500	2.5	Hornfels
	A24-018	500	502	2.71	Hornfels
	A24-018	502	504	6.42	Hornfels
	A24-018	504	506	2.26	Hornfels
	A24-018	506	508	5.92	Hornfels
	A24-018	508	510	3.61	Hornfels
	A24-018	510	512	2.51	Hornfels
	A24-018	512	514	2.98	Hornfels
	A24-018	514	516	1.29	Hornfels
	A24-018	516	518	1.97	Hornfels
	A24-018	518	520	2.99	Hornfels
	A24-018	520	522	1.5	Hornfels
	A24-018	522	524	1.6	Hornfels
	A24-018	524	526	2.63	Hornfels
	A24-018	526	528	3.97	Hornfels
	A24-018	528	530	4.6	Hornfels
	A24-018	530	532	4.58	Hornfels
	A24-018	532	534	2.1	Hornfels
	A24-018	534	536	2.99	Hornfels
	A24-018	536	538	3.65	Hornfels
	A24-018	538	540	4.73	Hornfels
	A24-018	540	542	2.97	Hornfels
Y400996	A24-021	737	739	1.39	Hornfels
	A24-021	739	741	1.55	Hornfels
	A24-021	741	743	2.88	Hornfels
	A24-021	743	745	2.19	Hornfels
	A24-021	745	747	1.74	Hornfels
	A24-021	747	749	2.67	Hornfels
	A24-021	749	751	3.2	Hornfels
	A24-021	751	753	2.47	Hornfels
	A24-021	753	755	1	Hornfels

Sample	DDH	From (m)	To (m)	Interval (m)	Current Domin
	A24-021	755	757	1.56	Hornfels
	A24-021	757	759	0.67	Hornfels
	A24-021	759	761	0.83	Hornfels
	A24-021	761	763	0.97	Hornfels
	A24-021	763	765	1.7	Hornfels
	A24-021	765	767	0.63	Hornfels
	A24-021	767	769	0.98	Hornfels
	A24-021	769	771	0.81	Hornfels
	A24-021	771	773	0.86	Hornfels
	A24-021	773	775	1.4	Hornfels
	A24-021	775	777	2.13	Hornfels
	A24-021	777	779	2.04	Hornfels
	A24-021	779	779.7	0.7	Hornfels

Table 13-23: 2025 Variability Sample Sources

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A23-007	566	568	2.15	P_Copper_Porphyry
	A23-007	568	570	1.38	P_Copper_Porphyry
	A23-007	570	572	1.72	P_Copper_Porphyry
	A23-007	572	574	2.08	P_Copper_Porphyry
	A23-007	574	576	4.18	P_Copper_Porphyry
	A23-007	576	578	6.07	P_Copper_Porphyry
	A23-007	578	580	1.62	P_Copper_Porphyry
	A23-007	580	582	4.04	P_Copper_Porphyry
	A23-007	582	584	3.16	P_Copper_Porphyry
	A23-007	584	586	2.24	P_Copper_Porphyry
	A23-007	586	588	1.01	P_Copper_Porphyry
	A23-007	588	590	1.35	P_Copper_Porphyry
	A23-007	590	593	0.85	P_Copper_Porphyry
	A23-007	593	594.9	4.04	P_Copper_Porphyry
A23-007	A23-007	594.9	596.9	0.75	P_Copper_Porphyry
	A23-007	596.9	599	1.88	P_Copper_Porphyry
	A23-007	599	601	2.7	P_Copper_Porphyry
	A23-007	601	603	2.67	P_Copper_Porphyry
	A23-007	603	605	0.85	P_Copper_Porphyry
	A23-007	605	607	1.65	P_Copper_Porphyry
	A23-007	607	609	1.13	P_Copper_Porphyry
	A23-007	609	611	0.66	P_Copper_Porphyry
	A23-007	611	613	0.94	P_Copper_Porphyry
	A23-007	613	615	3.04	P_Copper_Porphyry
	A23-007	615	617	1.37	P_Copper_Porphyry
	A23-007	617	619	0.83	P_Copper_Porphyry
	A23-007	619	621	1.32	P_Copper_Porphyry
	A23-007	621	623	1.53	P_Copper_Porphyry
	A23-007	623	625	1.23	P_Copper_Porphyry
A23-012	A23-012	456	458	1.73	P_Copper_Porphyry

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A23-012	458	460	2.27	P_Copper_Porphyry
	A23-012	460	462	2.3	P_Copper_Porphyry
	A23-012	462	464	2.55	P_Copper_Porphyry
	A23-012	464	466	2.36	P_Copper_Porphyry
	A23-012	466	468	3.93	P_Copper_Porphyry
	A23-012	468	470	4.35	P_Copper_Porphyry
	A23-012	470	472	3.43	P_Copper_Porphyry
	A23-012	472	474	3.15	P_Copper_Porphyry
	A23-012	474	476	2.85	P_Copper_Porphyry
	A23-012	476	478	4.75	P_Copper_Porphyry
	A23-012	478	480	8.8	P_Copper_Porphyry
	A23-012	480	482	3.14	P_Copper_Porphyry
	A23-012	482	484	3.05	P_Copper_Porphyry
	A23-012	484	486	4.27	P_Copper_Porphyry
	A23-012	486	488	4.94	P_Copper_Porphyry
	A23-012	488	490	4.96	P_Copper_Porphyry
	A23-012	490	492	3.86	P_Copper_Porphyry
	A23-012	492	494	2.4	P_Copper_Porphyry
	A23-012	494	496	3.85	P_Copper_Porphyry
	A23-012	496	498	3.42	P_Copper_Porphyry
	A23-012	498	500	3.65	P_Copper_Porphyry
	A23-012	500	502	3.53	P_Copper_Porphyry
	A23-012	502	504	3.7	P_Copper_Porphyry
	A23-012	504	506	3.28	P_Copper_Porphyry
	A23-012	506	508	2.22	P_Copper_Porphyry
A24-018	A24-018	364	366	1.14	P_Copper_Porphyry
	A24-018	366	368	0.86	P_Copper_Porphyry
	A24-018	368	370	1.35	P_Copper_Porphyry
	A24-018	370	372	1.56	P_Copper_Porphyry
	A24-018	372	374	1.49	P_Copper_Porphyry
	A24-018	374	376	1.47	P_Copper_Porphyry
	A24-018	376	378	2.47	P_Copper_Porphyry
	A24-018	378	380	1.53	P_Copper_Porphyry
	A24-018	380	382	1.25	P_Copper_Porphyry
	A24-018	382	384	2.16	P_Copper_Porphyry
	A24-018	384	386	1.04	P_Copper_Porphyry
	A24-018	386	388	2.19	P_Copper_Porphyry
	A24-018	388	390	1.87	P_Copper_Porphyry
	A24-018	390	392	1.94	P_Copper_Porphyry
	A24-018	392	394	2.51	P_Copper_Porphyry
	A24-018	394	396	3.42	P_Copper_Porphyry
A24-016	A24-016	1028	1030	1.74	P_Deep_Porphyry
	A24-016	1030	1032	1.04	P_Deep_Porphyry
	A24-016	1032	1034	0.44	P_Deep_Porphyry
	A24-016	1034	1036	0.34	P_Deep_Porphyry
	A24-016	1036	1038	0.35	P_Deep_Porphyry
	A24-016	1038	1040	0.37	P_Deep_Porphyry

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A24-016	1040	1042	0.5	P_Deep_Porphyry
	A24-016	1042	1044	0.46	P_Deep_Porphyry
	A24-016	1044	1046	0.44	P_Deep_Porphyry
	A24-016	1046	1048	0.26	P_Deep_Porphyry
	A24-016	1048	1050	0.81	P_Deep_Porphyry
	A24-016	1050	1052	0.21	P_Deep_Porphyry
	A24-016	1052	1054	0.18	P_Deep_Porphyry
	A24-016	1054	1056	0.24	P_Deep_Porphyry
	A24-016	1056	1058	0.28	P_Deep_Porphyry
	A24-016	1058	1060	0.66	P_Deep_Porphyry
	A24-016	1060	1062	0.4	P_Deep_Porphyry
	A24-016	1062	1064	0.57	P_Deep_Porphyry
	A24-016	1064	1066	0.99	P_Deep_Porphyry
	A24-016	1066	1068	0.5	P_Deep_Porphyry
	A24-016	1068	1070	0.37	P_Deep_Porphyry
	A24-016	1070	1072	0.52	P_Deep_Porphyry
	A24-016	1072	1074	0.36	P_Deep_Porphyry
	A24-016	1074	1076	0.65	P_Deep_Porphyry
A24-015	A24-015	919	921	0.19	P_Moly_Porphyry
	A24-015	921	923	0.19	P_Moly_Porphyry
	A24-015	923	925	0.12	P_Moly_Porphyry
	A24-015	925	927	0.4	P_Moly_Porphyry
	A24-015	927	929	0.24	P_Moly_Porphyry
	A24-015	929	931	0.2	P_Moly_Porphyry
	A24-015	931	933	0.28	P_Moly_Porphyry
	A24-015	933	935	0.46	P_Moly_Porphyry
	A24-015	935	937	0.73	P_Moly_Porphyry
	A24-015	937	939	0.1	P_Moly_Porphyry
	A24-015	939	941	0.07	P_Moly_Porphyry
	A24-015	941	943	0.13	P_Moly_Porphyry
	A24-015	943	945	0.18	P_Moly_Porphyry
	A24-015	945	947	0.19	P_Moly_Porphyry
	A24-015	947	949	0.18	P_Moly_Porphyry
	A24-015	949	951	0.14	P_Moly_Porphyry
	A24-015	951	953	0.1	P_Moly_Porphyry
	A24-015	953	955	0.15	P_Moly_Porphyry
	A24-015	955	957	0.56	P_Moly_Porphyry
	A24-015	957	959	2.07	P_Moly_Porphyry
	A24-015	959	961	0.07	P_Moly_Porphyry
	A24-015	961	963	0.07	P_Moly_Porphyry
	A24-015	963	965	1.46	P_Moly_Porphyry
	A24-015	965	967	0.07	P_Moly_Porphyry
	A24-015	967	969	0.2	P_Moly_Porphyry
	A24-015	969	971	0.07	P_Moly_Porphyry
A23-006	A23-006	667	669	1.24	P_Moly_Porphyry
	A23-006	669	671	0.88	P_Moly_Porphyry
	A23-006	671	673	0.73	P_Moly_Porphyry

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A23-006	673	675	2.34	P_Moly_Porphyry
	A23-006	675	677	0.99	P_Moly_Porphyry
	A23-006	677	679	0.52	P_Moly_Porphyry
	A23-006	679	681	0.82	P_Moly_Porphyry
	A23-006	681	683	0.64	P_Moly_Porphyry
	A23-006	683	685	0.47	P_Moly_Porphyry
	A23-006	685	687	0.53	P_Moly_Porphyry
	A23-006	687	689	0.68	P_Moly_Porphyry
	A23-006	689	691	0.62	P_Moly_Porphyry
	A23-006	691	693	0.65	P_Moly_Porphyry
	A23-006	693	695	0.84	P_Moly_Porphyry
	A23-006	695	697	1.4	P_Moly_Porphyry
	A23-006	697	699	0.63	P_Moly_Porphyry
	A23-006	699	701	0.76	P_Moly_Porphyry
	A23-006	701	703	0.45	P_Moly_Porphyry
	A23-006	703	705	0.54	P_Moly_Porphyry
	A23-006	705	707	0.61	P_Moly_Porphyry
	A23-006	707	708.5	0.85	P_Moly_Porphyry
	A23-006	647	649	0.11	P_Moly_Porphyry
	A23-006	649	651	0.67	P_Moly_Porphyry
	A23-006	651	653	0.47	P_Moly_Porphyry
	A23-006	653	655	0.65	P_Moly_Porphyry
	A23-006	655	657	0.98	P_Moly_Porphyry
	A23-006	657	659	1.04	P_Moly_Porphyry
	A23-006	659	661	1.3	P_Moly_Porphyry
	A23-006	661	663	1.39	P_Moly_Porphyry
	A23-006	663	665	1.73	P_Moly_Porphyry
	A23-006	665	667	0.36	P_Moly_Porphyry
A24-018	A24-018	496	498	2.61	Hornfels
	A24-018	498	500	2.5	Hornfels
	A24-018	500	502	2.71	Hornfels
	A24-018	502	504	6.42	Hornfels
	A24-018	504	506	2.26	Hornfels
	A24-018	506	508	5.92	Hornfels
	A24-018	508	510	3.61	Hornfels
	A24-018	510	512	2.51	Hornfels
	A24-018	512	514	2.98	Hornfels
	A24-018	514	516	1.29	Hornfels
	A24-018	516	518	1.97	Hornfels
	A24-018	518	520	2.99	Hornfels
	A24-018	520	522	1.5	Hornfels
	A24-018	522	524	1.6	Hornfels
	A24-018	524	526	2.63	Hornfels
	A24-018	526	528	3.97	Hornfels
	A24-018	528	530	4.6	Hornfels
	A24-018	530	532	4.58	Hornfels
	A24-018	532	534	2.1	Hornfels

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A24-018	534	536	2.99	Hornfels
	A24-018	536	538	3.65	Hornfels
	A24-018	538	540	4.73	Hornfels
	A24-018	540	542	2.97	Hornfels
A24-021	A24-021	737	739	1.39	Hornfels
	A24-021	739	741	1.55	Hornfels
	A24-021	741	743	2.88	Hornfels
	A24-021	743	745	2.19	Hornfels
	A24-021	745	747	1.74	Hornfels
	A24-021	747	749	2.67	Hornfels
	A24-021	749	751	3.2	Hornfels
	A24-021	751	753	2.47	Hornfels
	A24-021	753	755	1	Hornfels
	A24-021	755	757	1.56	Hornfels
	A24-021	757	759	0.67	Hornfels
	A24-021	759	761	0.83	Hornfels
	A24-021	761	763	0.97	Hornfels
	A24-021	763	765	1.7	Hornfels
	A24-021	765	767	0.63	Hornfels
	A24-021	767	769	0.98	Hornfels
	A24-021	769	771	0.81	Hornfels
	A24-021	771	773	0.86	Hornfels
	A24-021	773	775	1.4	Hornfels
	A24-021	775	777	2.13	Hornfels
	A24-021	777	779	2.04	Hornfels
A23-007	A23-007	206	208	5.59	Hornfels
	A23-007	208	210	23.6	Hornfels
	A23-007	210	212	32.9	Hornfels
	A23-007	212	214	4.3	Hornfels
	A23-007	214	216	1.84	Hornfels
	A23-007	216	218	3.38	Hornfels
	A23-007	218	220	1.7	Hornfels
	A23-007	220	222	3.58	Hornfels
	A23-007	222	224	2.14	Hornfels
	A23-007	224	226	2.28	Hornfels
	A23-007	226	228	1.12	Hornfels
	A23-007	228	230	11.3	Hornfels
	A23-007	230	232	6.44	Hornfels
	A23-007	232	234	1.96	Hornfels
	A23-007	234	236	1.52	Hornfels
	A23-007	236	238	3.42	Hornfels
	A23-007	238	240	4.5	Hornfels
	A23-007	240	242	2.92	Hornfels
	A23-007	242	244	1.6	Hornfels
	A23-007	244	246	0.88	Hornfels
	A23-007	246	248	1.13	Hornfels
	A23-007	248	250	2.08	Hornfels

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
	A23-007	250	252	2.9	Hornfels
	A23-007	252	254	4.88	Hornfels
A23-004	A23-004	110.9	113	3.36	Enriched
	A23-004	113	115	2.55	Enriched
	A23-004	115	117.5	2.67	Enriched
	A23-004	117.5	120	1.02	Enriched
	A23-004	120	122	2.5	Enriched
	A23-004	122	124	2.57	Enriched
	A23-004	124	126	73.7	Enriched
	A23-004	126	128	5.67	Enriched
	A23-004	128	130	1.03	Enriched
	A23-004	130	132	1.82	Enriched
	A23-004	132	134	1.58	Enriched
	A23-004	134	136	2.31	Enriched
	A23-004	136	138	1.82	Enriched
	A23-004	138	140	1.72	Enriched
	A23-004	140	142	2.34	Enriched
	A23-004	142	144	2.89	Enriched
	A23-004	144	146	5.59	Enriched
	A23-004	146	148	3.15	Enriched
	A23-004	148	150	1.85	Enriched
	A23-004	150	152	1.21	Enriched
	A23-004	152	154	0.91	Enriched
	A23-004	154	156	0.75	Enriched
	A23-004	156	158	1.89	Enriched
A23-005	A23-005	146	148	9.33	Enriched
	A23-005	148	150	4.44	Enriched
	A23-005	150	152	7.36	Enriched
	A23-005	152	154	1.18	Enriched
	A23-005	154	156	2.42	Enriched
	A23-005	156	158	2.81	Enriched
	A23-005	158	160	8.36	Enriched
	A23-005	160	162	3.28	Enriched
	A23-005	162	164	15.25	Enriched
	A23-005	164	166	5.89	Enriched
	A23-005	166	168	2.22	Enriched
	A23-005	168	170	1.62	Enriched
	A23-005	170	172	2.32	Enriched
	A23-005	172	174	2.58	Enriched
	A23-005	174	176	1.46	Enriched
	A23-005	176	178	0.99	Enriched
	A23-005	178	180	1.17	Enriched
	A23-005	136	138	2.97	Enriched
	A23-005	138	140	0.72	Enriched
	A23-005	140	142	2.5	Enriched
	A23-005	142	144	3.02	Enriched
	A23-005	144	146	3.11	Enriched

Smple	DDH	From (m)	To (m)	Interval (m)	Domin
A22-001	A22-001	91.3	94.05	3.14	Enriched
	A22-001	94.05	96.05	4.53	Enriched
	A22-001	96.05	98.7	3.23	Enriched
	A22-001	98.7	100.35	2.73	Enriched
	A22-001	100.35	102.35	3.87	Enriched
	A22-001	102.35	104.65	3.98	Enriched
	A22-001	104.65	106.65	5.99	Enriched
	A22-001	106.65	108.65	3.11	Enriched
	A22-001	108.65	110.65	1.72	Enriched
	A22-001	110.65	112.65	6.67	Enriched
	A22-001	112.65	114.65	1.5	Enriched
	A22-001	114.65	116.65	1.42	Enriched
	A22-001	116.65	118.65	2.61	Enriched
	A22-001	118.65	120.65	4.97	Enriched
	A22-001	120.65	122.6	1.69	Enriched
	A22-001	122.6	124.3	3.47	Enriched
	A22-001	124.3	126.3	2.02	Enriched
	A22-001	126.3	128.3	3.84	Enriched
	A22-001	128.3	130.3	3.36	Enriched
	A22-001	130.3	132.3	1.89	Enriched
	A22-001	132.3	134.3	3.34	Enriched
	A22-001	134.3	135.35	2.49	Enriched
	A22-001	135.35	137.35	1.31	Enriched
	A22-001	137.35	139.35	2.2	Enriched
	A22-001	139.35	141.8	9.28	Enriched
	A22-001	141.8	143.8	2.29	Enriched
	A22-001	143.8	145.8	3.87	Enriched
	A22-001	145.8	147.8	1.92	Enriched
	A22-001	147.8	149.8	1.07	Enriched
	A22-001	149.8	151.8	0.96	Enriched

14.0 MINERAL RESOURCE ESTIMATE

The Mineral Resource statement presented herein represents the updated mineral resource estimate for the Aurora deposit located in Calca Province, Peru. The Aurora deposit Mineral Resource estimate was prepared by GRE under the supervision of GRE's Principal Geologist, Hamid Samari, and Principal QP, Terre Lane, who takes responsibility for the estimate. The most recent previous Mineral Resource estimate was contained in the report titled "NI 43-101 Technical Report on the Aurora Cu-Mo-Ag Property in the Calca Province, Peru" (AMC, 2025).

The mineral resources were estimated in conformity with the CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian National Instrument 43-101. This mineral resource estimate includes inferred mineral resources. Inferred resources are defined as resources that are "…reasonably expected that the majority of the inferred mineral resources could be upgraded to indicated." (CIM, 2014) by additional drilling. There is also no certainty that the inferred mineral resources will be converted to the measured or indicated categories through further drilling or into mineral reserves, once economic considerations are applied. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The project presently has no mineral reserves. Whittle Pit optimization was applied to the open pit mineral resource estimate to assess the reasonable prospects for economic extraction for the resource.

The effective date of the Mineral Resource Statement is April 30, 2026. In the opinion of the qualified persons Terre Lane and Hamid Samari, the mineral resource estimate reported here is a reasonable representation of the global mineral resources found in the Aurora deposit at the current level of sampling.

The estimate was completed in Leapfrog Geo version: 2025.3 software using the Leapfrog Edge extension (Leapfrog).

The QP for the Mineral Resource estimate is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially impact the estimate. However, potential factors that may influence the estimate include metal price assumptions, changes in the interpretation of mineralization geometry and continuity, variations in metallurgical recovery and operating cost assumptions, confidence in modifying factors (such as securing surface rights for mining infrastructure), delays or challenges in reaching agreements with local or regulatory authorities and stakeholders, and changes to land tenure or permitting requirements.

14.1 Data Used

14.1.1 Drillhole Database

GRE performed a data validation of the drill hole database prepared by DLP for the Aurora deposit and determined it to be of suitable accuracy to perform a mineral resource estimate for the property. More detail regarding the validation of the drill hole database can be found in Section 12.0. The sample data used for the estimation consisted of surface diamond drillholes (DDH) drilled by DLP. DLP maintains the drilling database in Microsoft Excel sheets and provided the data to GRE as Excel files. Twenty-eight

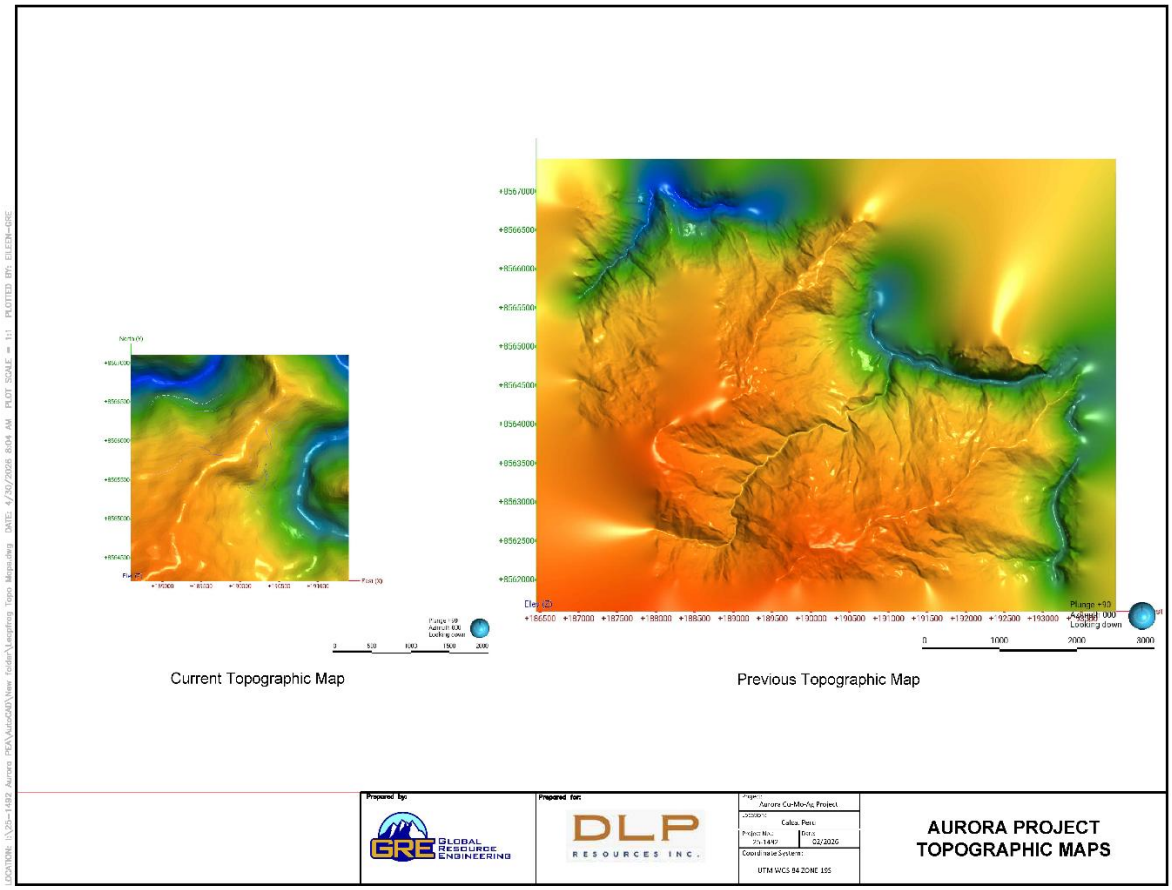
drillholes, drilled between 2022 and 2025, for a total of 22,613.38 m, were used to inform the estimate (see Section 10.0, Table 10-8).

14.1.2 Topography

DLP provided GRE with several topographic maps with varying levels of areal coverage and resolution. The previous 2025 Mineral Resource estimate used the topographic map with the least areal coverage but highest resolution, referred to as "aurora_wo_4594_5m_contours_2023jun26." GRE compared that topographic map with the other maps provided with greater areal coverage and determined that the previous topographic map did not align well vertically with the other maps, with vertical displacement on the order of 50 to 80 meters. Because GRE would be using the topographic map for layout of mine, mill, and administrative facilities as well as for pit design, GRE determined that use of the "aurora_wo_4594_5m_contours_2023jun26" topography would result in large discrepancies in elevations where that topographic map abutted other topographic maps; therefore, GRE used a topographic map with greater areal coverage but lower resolution, referred to as "existing ground topo." Because this topographic map is, as stated above, on the order of 50 to 80 meters lower in elevation than the previously used topographic map, all drill holes in the database were adjusted within Leapfrog to match the topographic elevation of the new topographic map.

Figure 14-1 show the previously used topographic map along with a cropped portion of the new topographic map.

Figure 14-1: Current and Previous Topographic Maps for the Aurora Deposit



14.1.3 Density

DLP has collected 584 density measurements from the modeled geological domains. The collection of density measurements is described in Section 11.0. Estimation of in situ dry bulk density in the resource block model is discussed in Section 14.6.

14.2 Domain Modeling

14.2.1 Geological Domaining

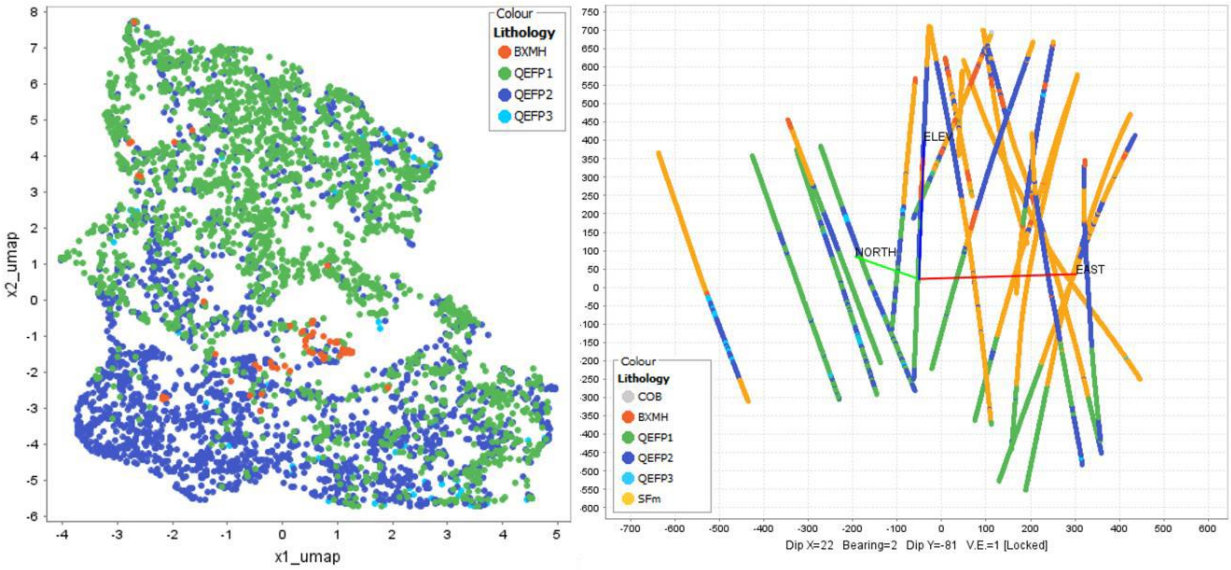
Mineralization at the Aurora deposit is hosted in a series of porphyry intrusions, breccias, and the surrounding hornfels country rock. Geological logging identified three phases of porphyry intrusions. For the previous technical report (AMC, 2025), the AMC QP conducted multivariate analysis using the geochemical data to help refine the limits of the different porphyry phases. The current Mineral Resource estimate uses the results of that analysis, which is described below exactly as described in the 2025 Technical Report. Note that references to the QP in this quoted section refer to the QP of the 2025 Technical Report Mineral Resource Estimate.

The Uniform Manifold Approximation and Projection (UMAP) dimension-reduction algorithm was used to visualize chemical data for samples logged as porphyry intrusion. Five elements were chosen – Ti, Nb, Sc, Al and Th. These elements are useful for identifying magmatic fractionation trends in mineralized and

hydrothermally altered porphyry systems. The algorithm transforms the five chemical variables into two dimensions, x1_umap and x2_umap.

The geological logging codes are projected onto the points on the UMAP in Figure 14-2. There is a clear clustering of the different logged porphyry phases in UMAP space with minor overlap. QEFP1 samples occur in two different clusters, suggesting this logging code includes two chemically distinct phases.

Figure 14-2: DDH Assay Samples (UMAP) & Schematic Long Section Colored by Logged Geology

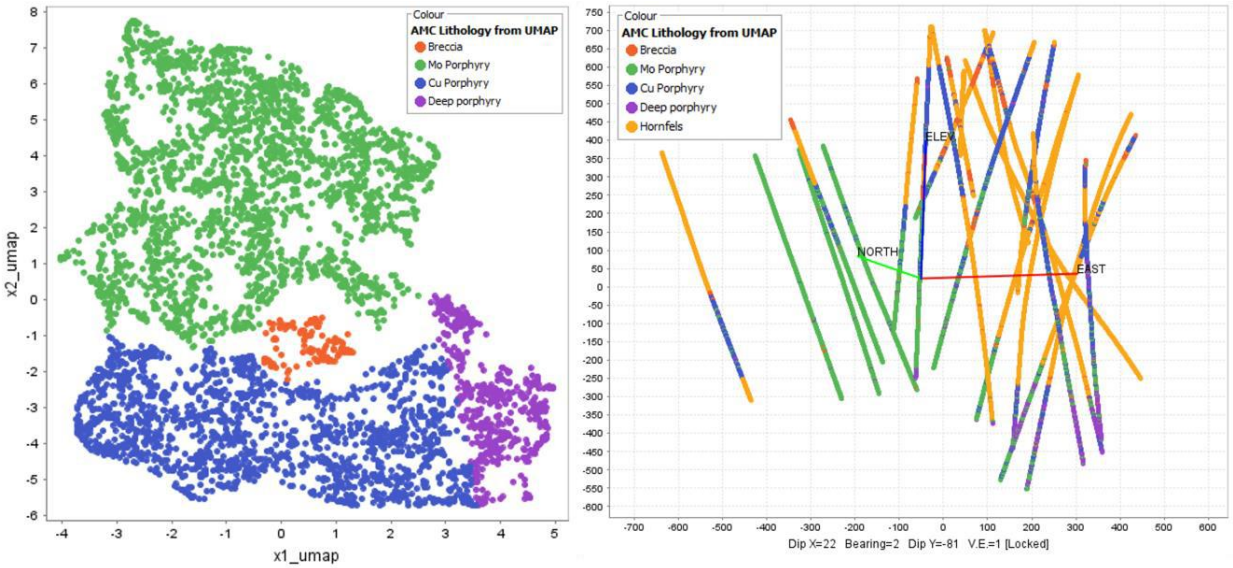


Source: AMC, 2025

The QP refined the lithology logging by defining groups of samples in UMAP space that have similar chemical and geological characteristics (Figure 14.3). To avoid confusion, these new lithology domains were named as the Cu Porphyry, Mo Porphyry, and Deep Porphyry (Figure 14-3).

The Cu Porphyry includes the majority of samples logged as QEFP2 and is associated with Cu-rich mineralization. The Mo Porphyry consists of the samples logged as QEFP1, excluding samples from deeper deposit areas, and is associated with Mo-rich mineralization. The Deep Porphyry consists of the samples logged as QEFP1 from the deeper areas of the deposit. These samples form a separate cluster of samples in UMAP space, which are more chemically similar to the Cu Porphyry, but the Deep Porphyry is associated with Mo-rich mineralization similar to the Mo Porphyry.

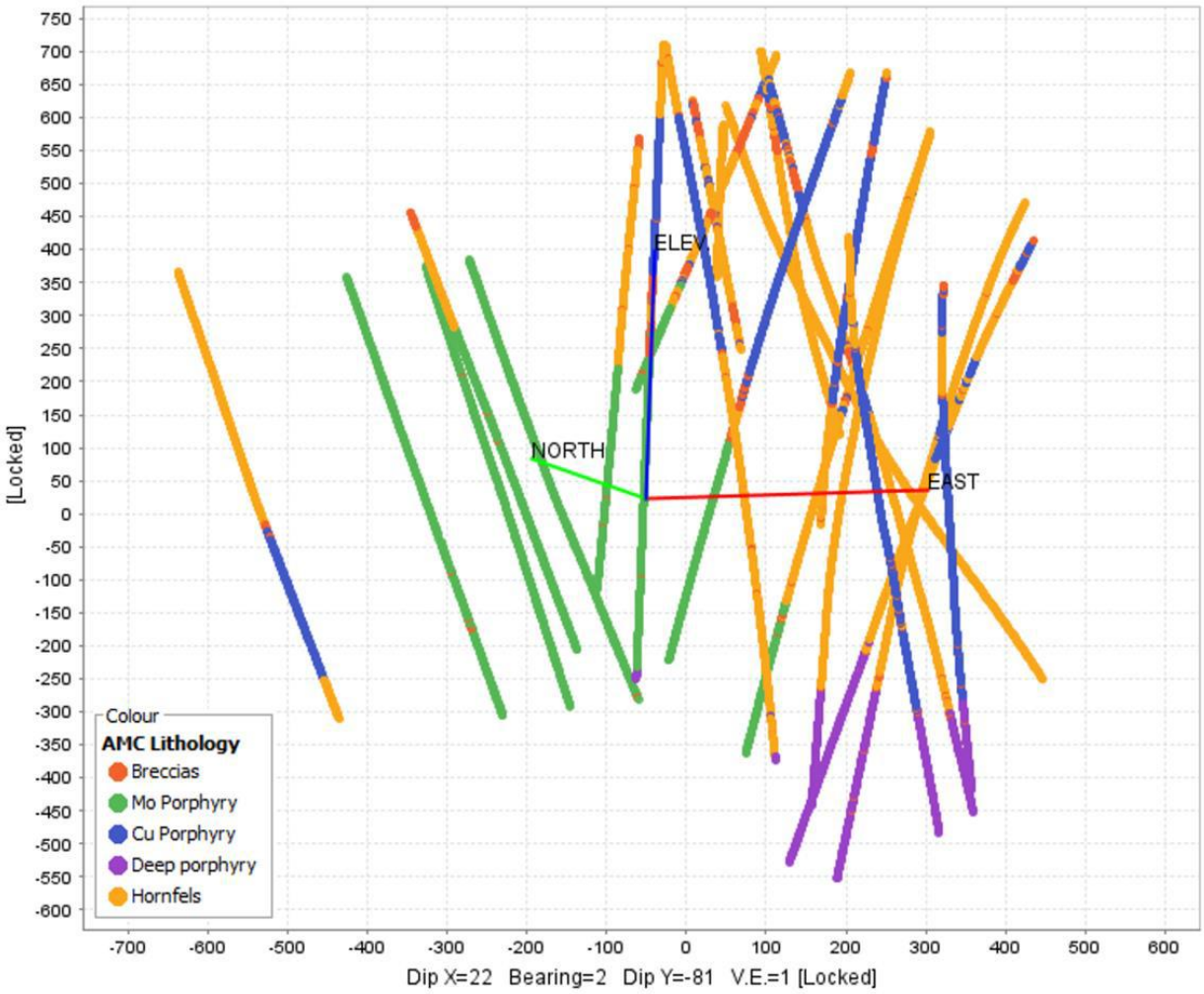
Figure 14-3: DDH Assay Samples (UMAP) & Schematic Long Section Colored by AMC Lithological Domain



Source: AMC, 2025.

The new lithological domains were further refined and simplified after viewing the groups projected onto the drillholes. The final interpretation that was used as the basis of the 3D geological modelling is shown in Figure 14-4.

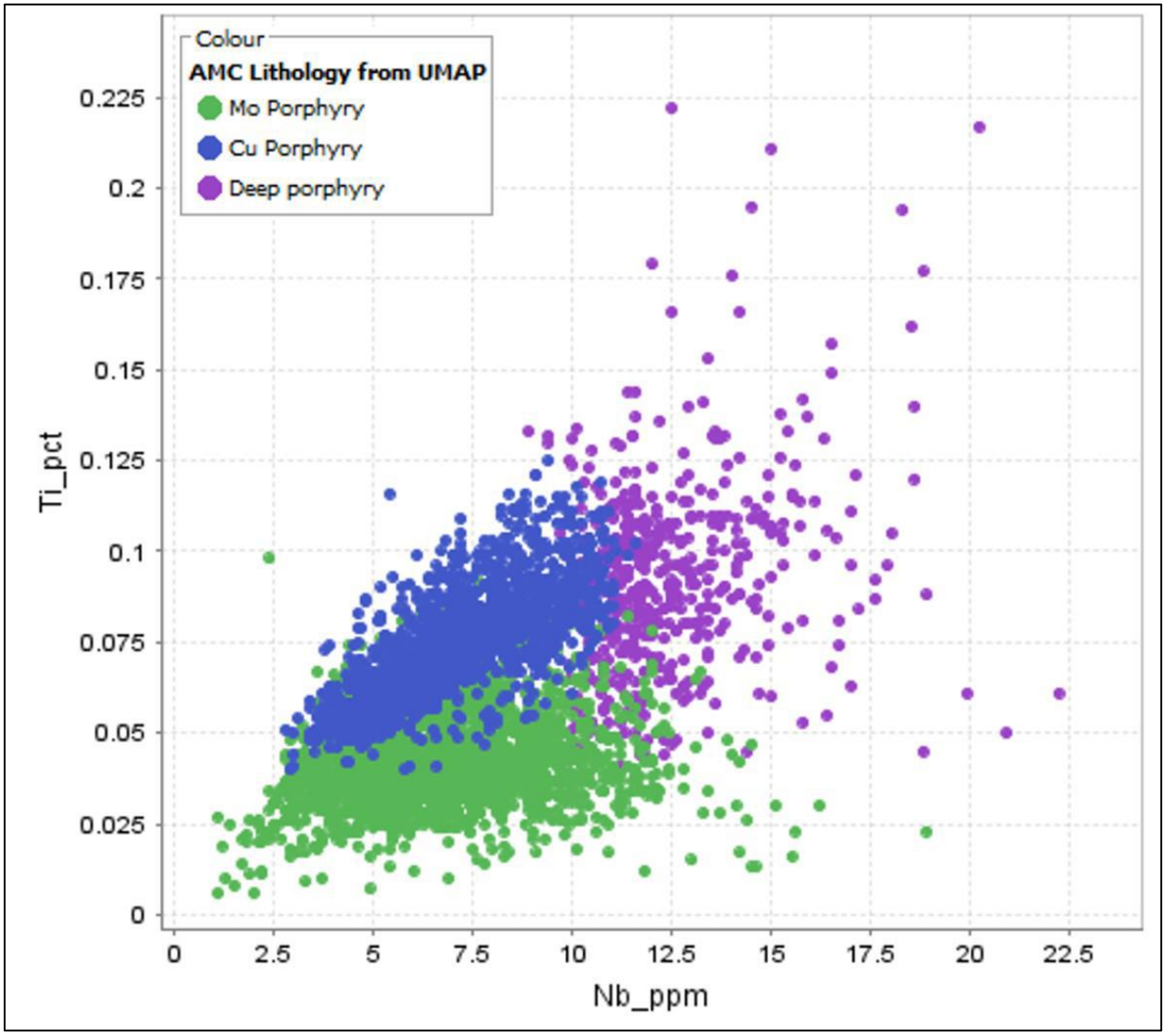
Figure 14-4: Schematic Long Section Colored by Final Lithology Domain



Source: AMC, 2025

Overall, the chemistry of the different porphyry phases is only subtly different. A review of bivariate relationships such as Figure 14-5 suggests that the Mo Porphyry is the most fractionated phase, the Cu Porphyry is relatively less fractionation, and the Deep Porphyry has seen the lowest degree of magmatic fractionation of the three phases. The QP recommends U-Pb zircon dating of the three porphyry phases to better understand their relative timing relationships.

Figure 14-5: Bivariate Plot of Ti and Nb Data for Porphyry Intrusions



Source: AMC, 2025

14.2.2 Geological Modeling

14.2.2.1 Lithology

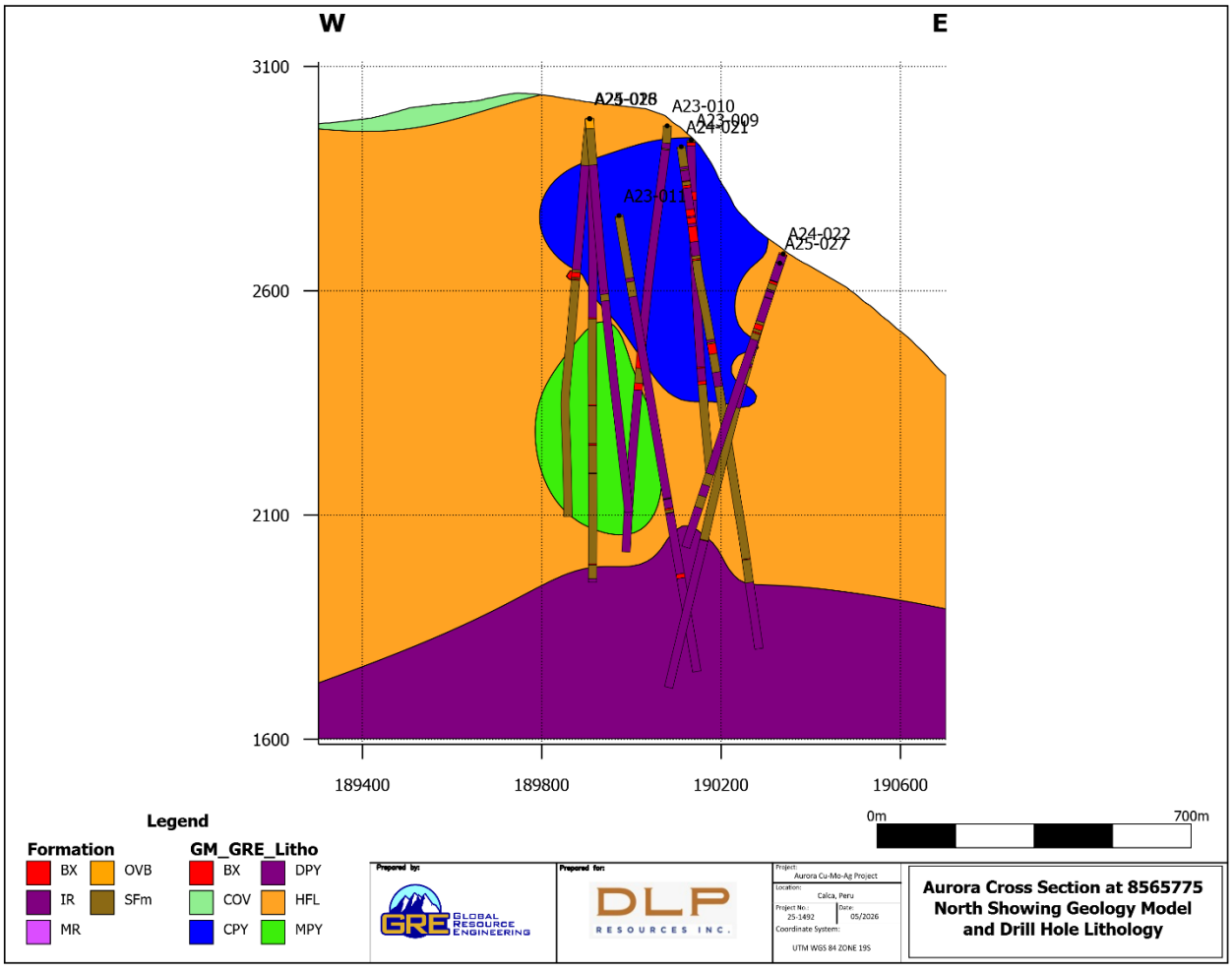
Prior to geological modeling, the refined lithological logging groups described in Section 14.2.1 were reviewed, assigned to the Aurora drill-hole database, and imported into Leapfrog Geo. GRE's Qualified Person (QP) updated the three-dimensional (3D) geological model and geological interpretation using the revised DLP drill-hole database, incorporating all drilling completed between 2022 and 2025, including 28 diamond drill holes totaling 22,613.38 m. The updated geological model integrates these data with historical drilling completed by previous operators to refine the geometry and spatial distribution of the principal lithological units across the Aurora deposit.

The Cover (COV) and Hornfels (HFL) domains were modeled as stratigraphic surfaces, whereas the Breccia (BX), Copper Porphyry (CPY), Molybdenum Porphyry (MPY), and Deep Porphyry (DPY) domains were modeled as intrusive bodies using Leapfrog Geo's implicit modeling algorithms. The lithological domains

were constrained by detailed drill-hole logging and refined through iterative review of plan views, cross sections, and three-dimensional visualizations to ensure consistency with the available geological information. Where necessary, the lithological coding of a limited number of drill-hole intervals was manually adjusted to ensure that the geological domains honored the logged lithology (Figure 14-6 and Figure 14-7).

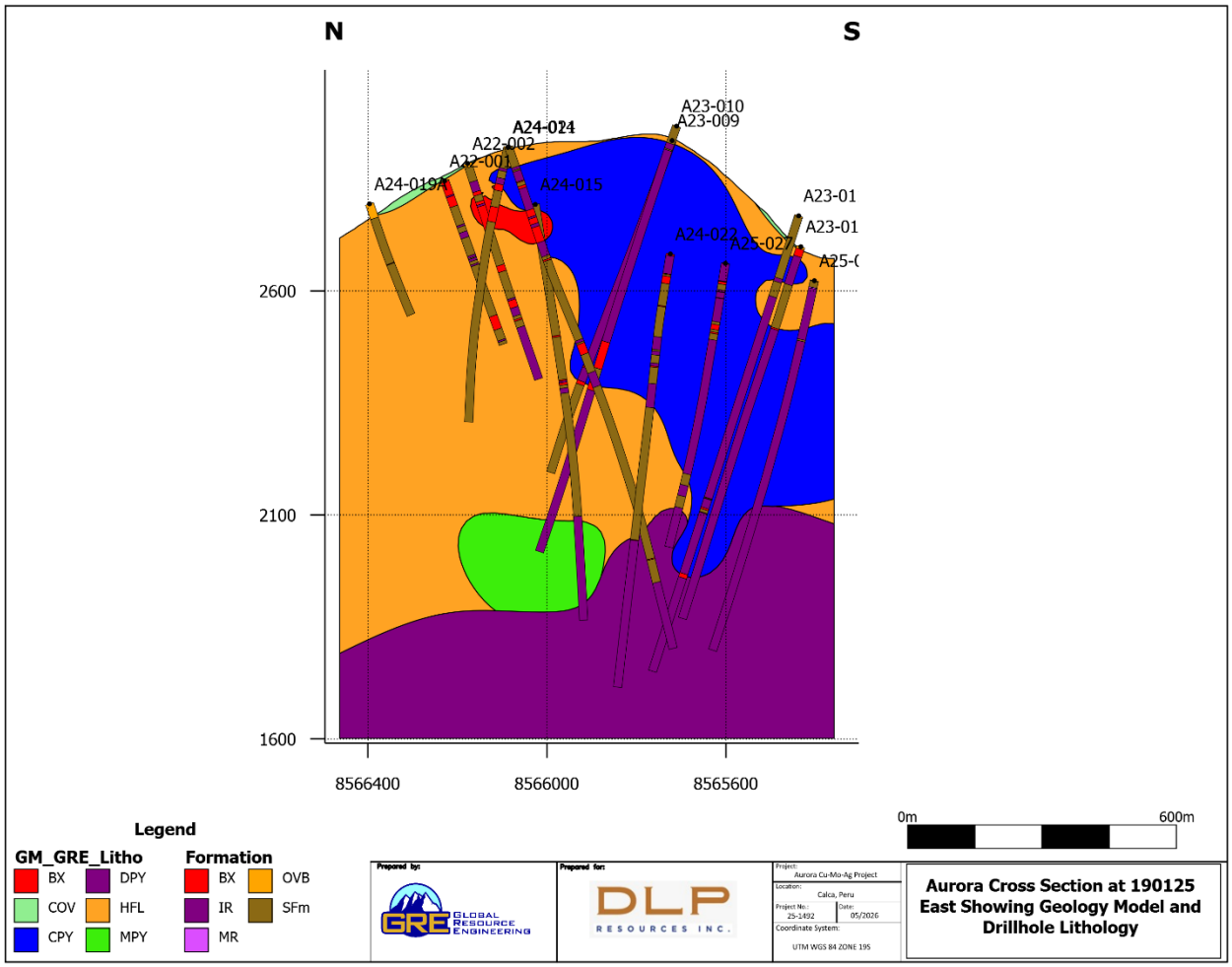
The completed geological model comprises Cover (COV), Breccia (BX), Copper Porphyry (CPY), Molybdenum Porphyry (MPY), Hornfels (HFL), and Deep Porphyry (DPY). The model provided the geological framework for the estimation domains and the 2026 Mineral Resource Estimate.

Figure 14-6: Aurora Lithological Model Cross-Section at 8565775 Looking North



Source: GRE, 2026

Figure 14-7: Aurora Lithological Model Cross-Section at 190125 Looking East



Source: GRE, 2026

14.2.2.2 Mineralization

Mineralization domains were modeled in Leapfrog Geo using the updated geological framework described in Section 14.2.2.1. GRE's QP updated the three-dimensional mineralization model using the revised DLP drill-hole database, incorporating all drilling completed between 2022 and 2025. The mineralization model defines three estimation domains based on oxidation characteristics and mineralogical interpretation: Primary, Enriched, and Leached.

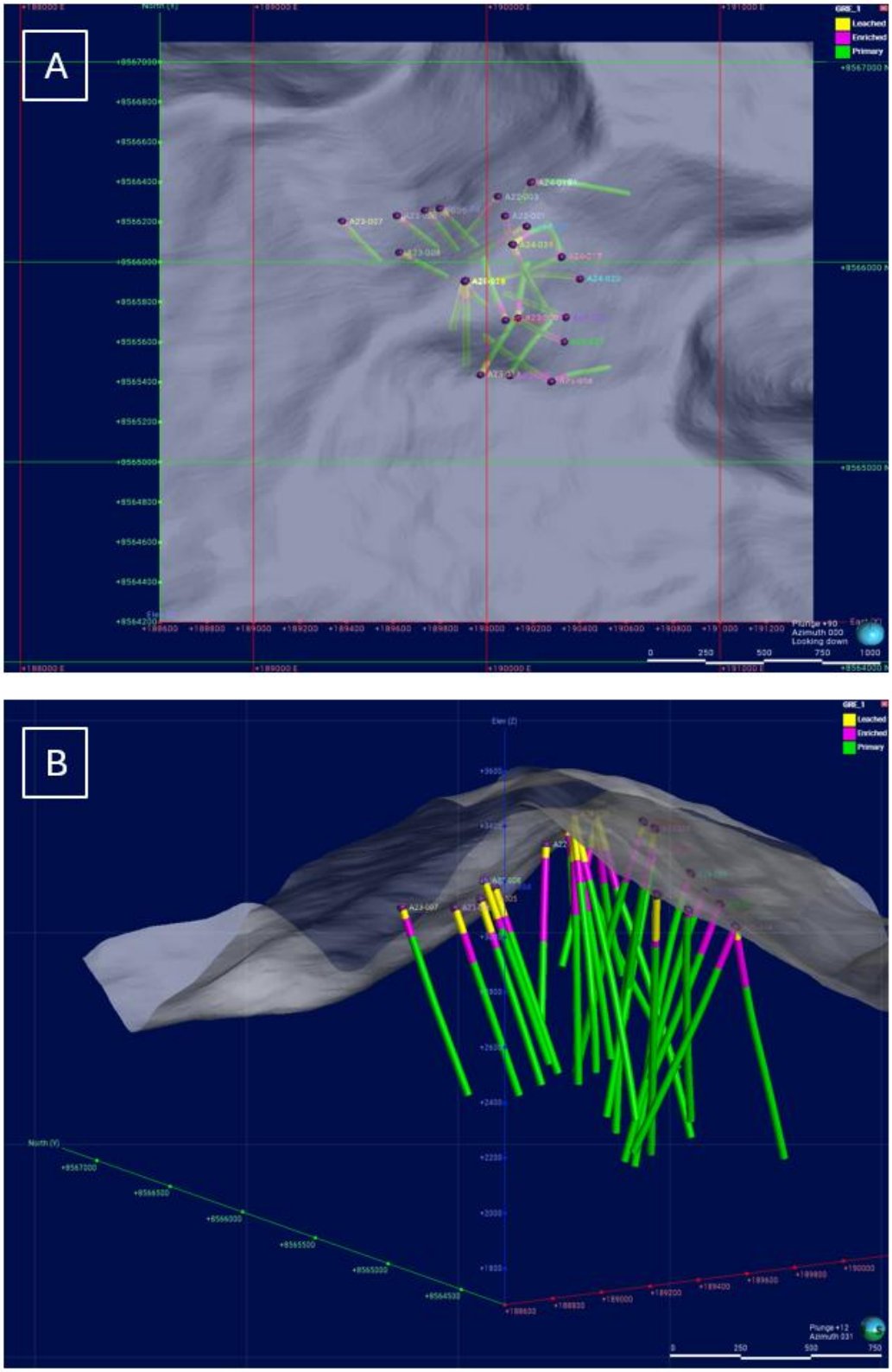
The mineralization domains were interpreted from detailed drill-hole geological logging, oxidation logging, sulfide mineralogy, and supporting assay information. Domain boundaries were constrained by drill-hole observations and refined through iterative review of plan views, cross sections, and three-dimensional visualizations to ensure consistency with the geological interpretation. Where necessary, the domain coding of a limited number of drill-hole intervals was manually adjusted to ensure that the mineralization domains honored the logged oxidation and mineralization characteristics.

The Primary domain represents the unoxidized sulfide portion of the porphyry system and constitutes the largest mineralization domain at the Aurora deposit. The Enriched domain represents the supergene

enrichment zone developed immediately above the primary sulfide mineralization, whereas the Leached domain defines the near-surface oxidized zone resulting from weathering and leaching processes. The spatial distribution of these domains reflects the vertical weathering profile developed across the deposit and provides the basis for estimation domain coding.

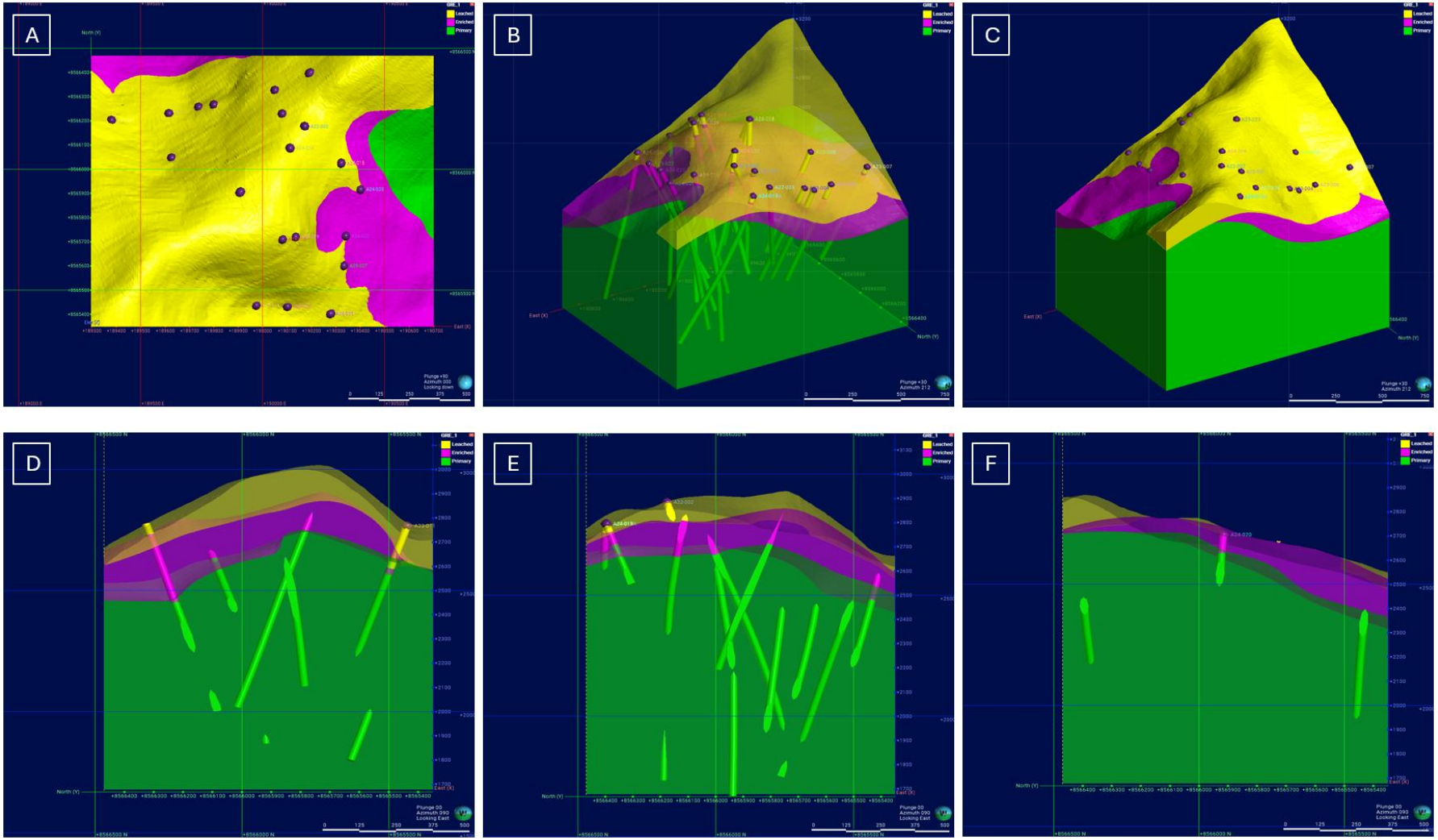
The distribution of the mineralization domains within the drill-hole database is illustrated in Figure 14-8, which presents plan and oblique views of the interpreted oxidation domains. The completed three-dimensional mineralization model, including plan views, oblique views, and representative cross sections, is shown in Figure 14-9, illustrating the geometry, continuity, and spatial relationships of the Primary, Enriched, and Leached domains used to constrain grade estimation for the 2026 Mineral Resource Estimate.

Figure 14-8: Drill-Hole Mineralization Domains, Plan View (A), and Oblique View (B)



Source: GRE, 2026

Figure 14-9: Three-Dimensional Mineralization Model, Plan View (A), Oblique Views (B & C), and Cross Sections: at 190000 (D), at 190200 (E), and at 190400 (F) Looking East)



Source: GRE, 2026

14.3 Estimation Domaining

The five mineralized lithologies (breccia, copper porphyry [CPY], moly porphyry [MPY], deep porphyry [DPY], and hornfels [HFL]) were combined with the three mineralogies (leached, enriched, and primary) to form 13 preliminary domains:

- Breccia Enriched
- Breccia Leached
- Breccia Primary
- CPY Enriched
- CPY Leached
- CPY Primary
- DPY Primary
- HFL Enriched
- HFL Leached
- HFL Primary
- MPY Enriched
- MPY Leached
- MPY Primary

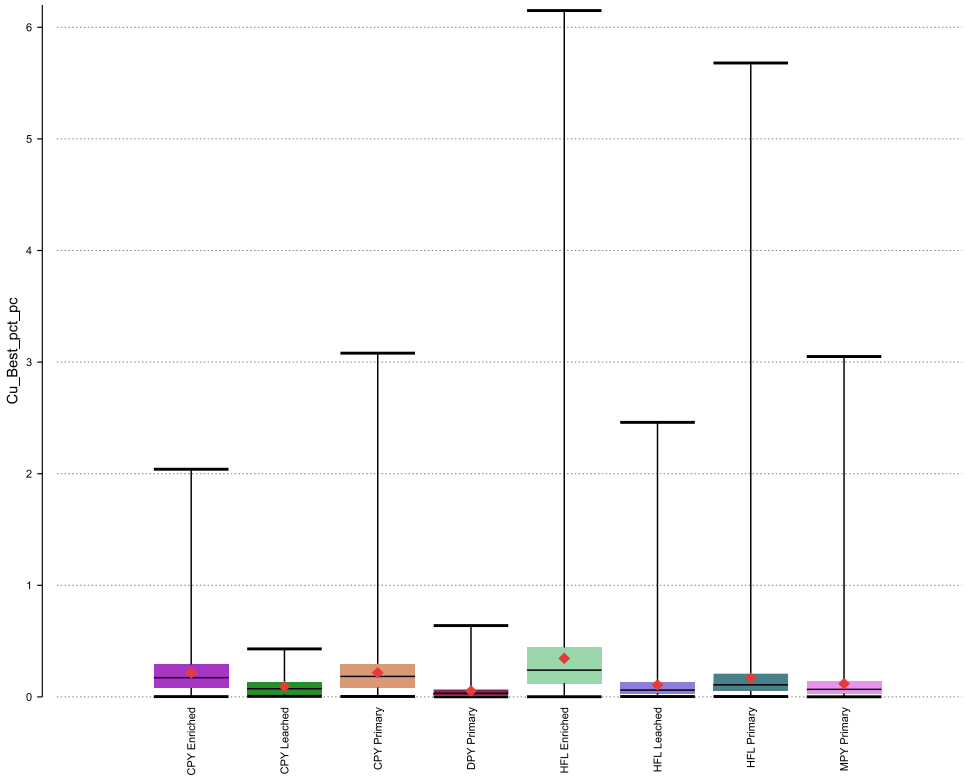
The DPY lithology did not intersect the Enriched or Leached mineralogies.

The three breccia domains and the MPY enriched and leached domains were very small and contained insufficient data to use as separate estimation domains. As a result, the GRE QP combined those preliminary domains with the HFL domains to form eight final estimation domains:

- CPY Enriched
- CPY Leached
- CPY Primary
- DPY Primary
- HFL Enriched – combined breccia enriched, HFL enriched, and MPY enriched
- HFL Leached – combined breccia leached, HFL leached, and MPY leached
- HFL Primary – combined breccia primary and HFL primary
- MPY Primary

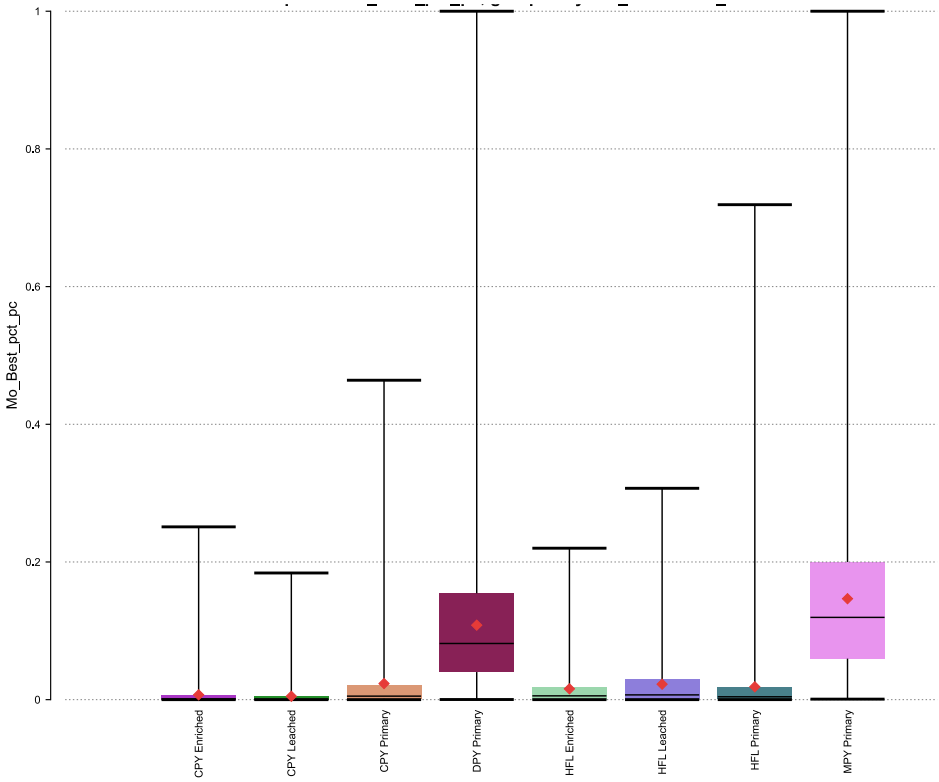
Box and whisker plots for the eight estimation domains are shown in Figure 14-10 through Figure 14-12.

Figure 14-10: Aurora Deposit Copper Box & Whisker Plots by Estimation Domain



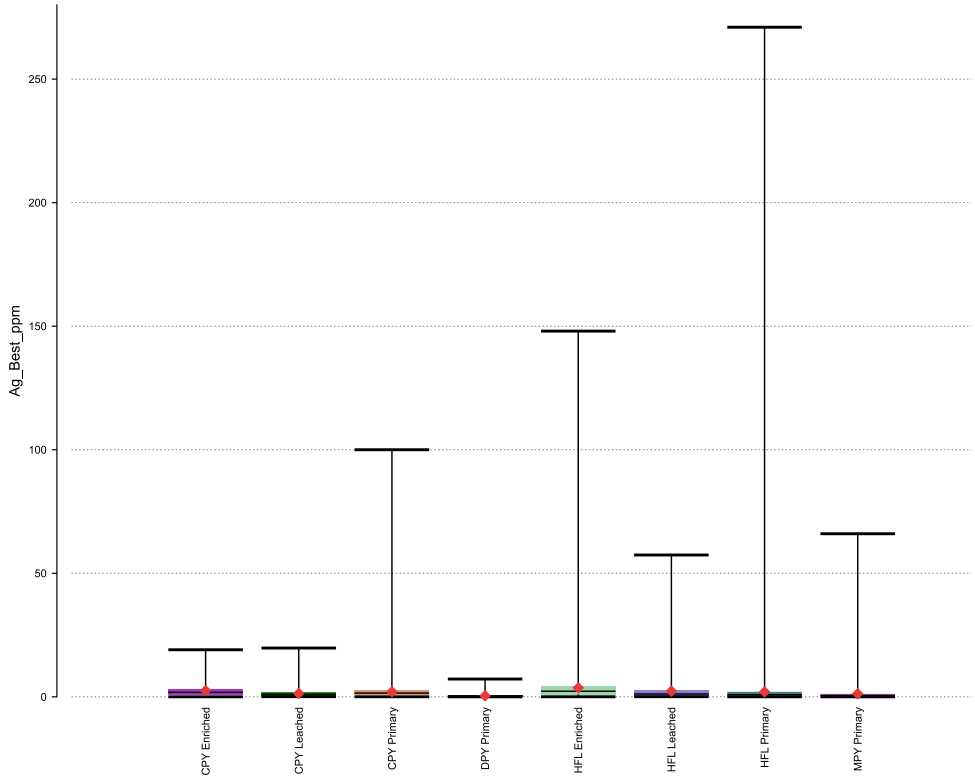
Source: GRE, 2026

Figure 14-11: Aurora Deposit Molybdenum Box & Whisker Plots by Estimation Domain



Source: GRE, 2026

Figure 14-12: Aurora Deposit Silver Box & Whisker Plots by Estimation Domain

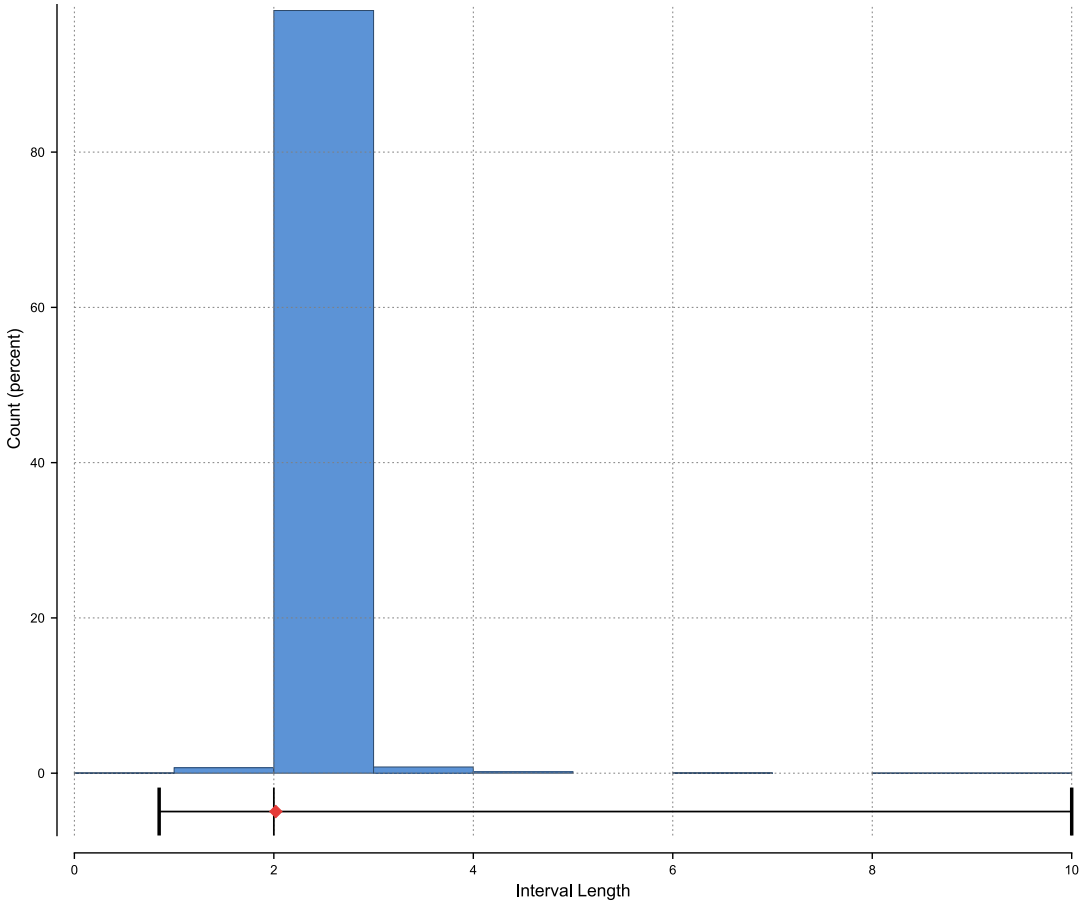


Source: GRE, 2006

14.4 Statistics and Compositing

Sample data was composited to intervals of equal length to ensure that the samples used in statistical analysis and estimations were equally weighted. To accomplish compositing, Ms. Lane of GRE first examined the interval histogram to determine the most common assay length, which was 2 meters (Figure 14-13). Ms. Lane of GRE evaluated various compositing lengths to avoid splitting assays. It was decided that compositing on a 10-meter interval represented a significant decrease in the variance of the data while not adversely decreasing the mean of the data set. Therefore, Ms. Lane of GRE selected a 10-meter composite interval.

Figure 14-13: Histogram of Aurora Raw Sample Lengths



Source: GRE, 2026

Table 14-1: Aurora Deposit Compositing Interval Statistics

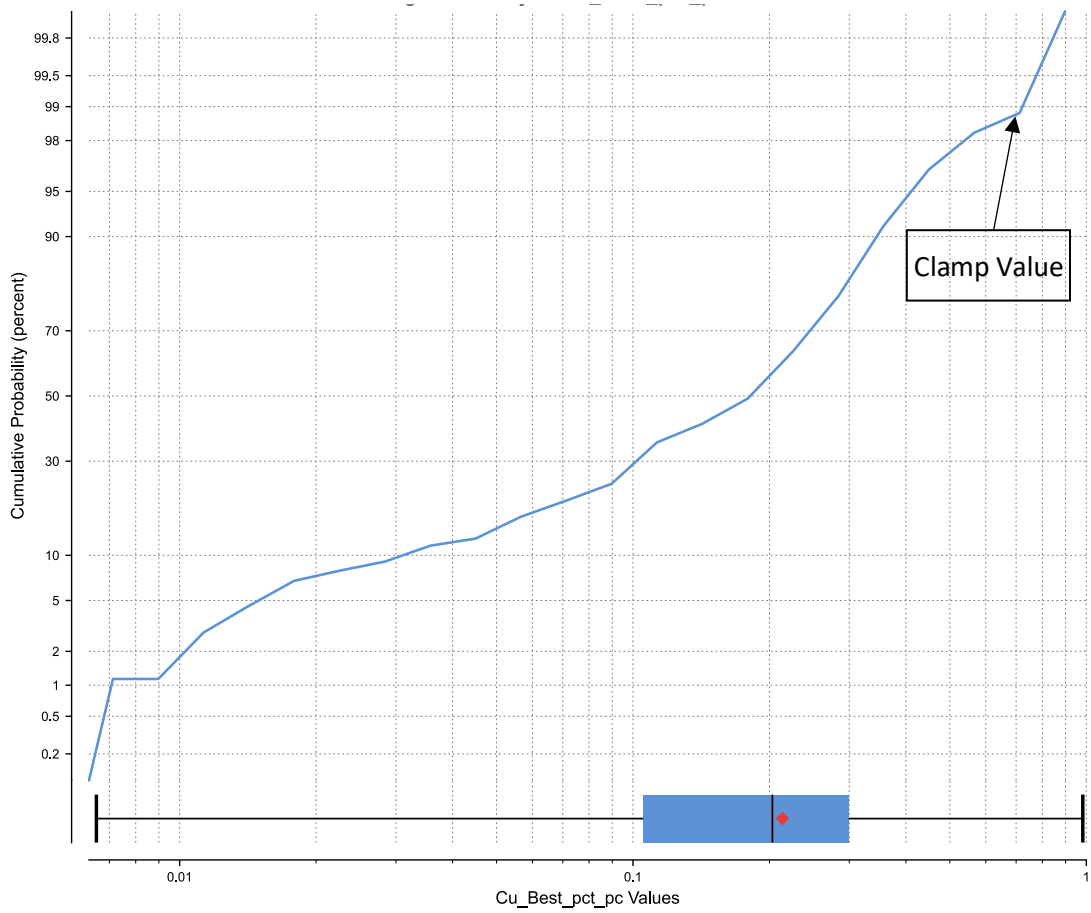
	Original Assays			4-m Composites			10-m Composites		
Name	**Ag_ppm**	**Cu_pct**	**Mo_pct**	**Ag_ppm**	**Cu_pct**	**Mo_pct**	**Ag_ppm**	**Cu_pct**	**Mo_pct**
Count	11,126	11,126	11,126	5,617	5,617	5,617	2,250	2,250	2,250
Length	22,476.03	22,476.03	22,476.03	22,473.38	22,473.38	22,473.38	22,503.38	22,503.38	22,503.38
Mean	1.891	0.167	0.050	1.892	0.167	0.050	1.890	0.167	0.050
Standard deviation	5.358	0.219	0.084	3.884	0.190	0.079	3.007	0.163	0.076
Coefficient of variation	2.833	1.312	1.674	2.053	1.135	1.573	1.592	0.976	1.512
Variance	28.708	0.048	0.007	15.084	0.036	0.006	9.045	0.027	0.006
Minimum	0.005	0.00099	0.000012	0.01	0.001145	0.0000145	0.021	0.002382	0.0000332
Lower quartile	0.38	0.0431	0.00159	0.445	0.0488	0.001885	0.519	0.05412	0.0021905
Median	0.96	0.1015	0.01115	1.0775	0.11075	0.01231	1.206	0.11656	0.0134
Upper quartile	2.09	0.219	0.0635	2.2	0.2275	0.0674	2.325	0.23899	0.0689
Maximum	271	6.15	1	147.57875	3.1875	0.6485	68.878	1.6653	0.47

This comparison shows that compositing, while not significantly changing the mean or quartiles, does drastically reduce the maximum value of grades.

14.5 Evaluation of Outliers

Grade clipping was investigated for copper, molybdenum, and silver after compositing was completed to reduce the effect of any high-grade outliers still present within the populations. Cumulative probability plots were generated for Cu, Mo, and Ag in each estimation domain (see Figure 14-14 for an example). A break in the population was identified as indicating the presence of high-grade outliers. GRE used the grade clamping feature in Leapfrog to reduce the effect of these outliers on the results.

Figure 14-14: Example of Cumulative Probability Plot (CPY Enriched Cu)



Leapfrog's grade clamping feature reduces bias by constraining the effect of high values at a distance. It reduces any high values to the value threshold. It limits the samples that will be considered to those within a specified distance percentage of the search ellipsoid size, and only those outside that distance if they are within the value threshold. If a sample point is beyond the distance threshold and the point's value exceeds the value threshold, it will be clamped to the value threshold.

Table 14-2 shows the clamping values for each metal in each domain.

Table 14-2: Aurora Deposit Estimation Clamping Summary

Domain	Clamping Value	Number Clamped
Cu		
CPY Enriched	0.9	2
CPY Leached	0.16	11
CPY Primary	0.7	2
DPY Primary	0.225	4
HFL Enriched	1.0	4
HFL Leached	0.7	1
HFL Primary	0.7	3
MPY Primary	0.7	4
Mo		
CPY Enriched	0.055	6
CPY Leached	0.045	2
CPY Primary	0.25	2
DPY Primary	0.29	2
HFL Enriched	0.09	7
HFL Leached	0.08	9
HFL Primary	0.18	2
MPY Primary	-	
Ag		
CPY Enriched	7.2	4
CPY Leached	4.5	1
CPY Primary	10.8	4
DPY Primary	1.8	4
HFL Enriched	10.5	7
HFL Leached	10.8	5
HFL Primary	10.6	10
MPY Primary	9.0	5

14.6 Density

Average in situ dry bulk densities were calculated for each domain, as shown in Table 14-3. The results for each domain fit well with GRE's experience with similar rock types.

Table 14-3: Aurora Deposit Densities

Domain	Min Density	Max Density	Avg Density	Number of Samples
HFL Enriched	2.51	3.08	2.80	34
HFL Leached	2.2	2.99	2.74	36
HFL Primary	2.58	3.09	2.81	181
COV Leached	2.19	2.21	2.20	13
CPY Enriched	2.2	3.01	2.62	51
CPY Leached	2.48	2.68	2.58	14
CPY Primary	2.52	2.95	2.64	138
DPY Primary	2.29	2.91	2.61	68

Domain	Min Density	Max Density	Avg Density	Number of Samples
MPY Primary	2.45	2.85	2.63	142

14.7 Variography

The majority of domains were estimated with standard variograms where enough samples were present to inform the variograms. For domains with relatively few samples, a good fit for the data was found using pairwise relative variograms. The pairwise relative variogram helps to smooth the variogram by scaling γ(h) using the square of the mean of each sample pair of the data from calculating γ(h). This makes the interpretation of the variogram model easier, and all variances calculated this way are relative to the mean of the sample pairs within the distribution.

Variogram analysis was completed on all of the estimation domains. The nugget for each variogram was determined by examining the downhole variograms and determining where the short-range trend crossed the y-axis. The sill for each variogram was found using a global variogram. To the extent possible, variograms were oriented along the strike and dip of the visually observed grade trend of the composites, with the major axis oriented along the direction of maximum continuity.

Table 14-4 summarizes the variogram parameters for each estimation domain and metal.

Table 14-4: Aurora Deposit Variogram Parameters

Estimation Domain	Nugget	Sill		Dip	Dip Azimuth	Pitch	Range Structure 1			Range Structure 2		
		Structure 1	Structure 2				Major Axis	Semi-Major Axis	Minor Axis	Major Axis	Semi-Major Axis	Minor Axis
Cu												
CPY Enriched	0.135	0.0001	0.716	0	0	90	350	340	34	450	410	230
CPY Leached	0.117	0.241	0.261	0	0	67	355	510	25	640	615	80
CPY Primary	0.244	0.223	0.469	6	0	130	323	155	30	600	475	200
DPY Primary	0.154	0.016	0.865	4	0	160	238	200	46	725	400	133
HFL Enriched	0.136	0.473	NA	0	0	70	700	660	270	NA	NA	NA
HFL Leached	0.124	0.011	0.436	6	0	130	170	500	20	715	675	90
HFL Primary	0.225	0.231	0.719	6	0	130	600	600	37	720	720	270
MPY Primary	0.093	0.458	0.383	25	5	114	346	300	56	550	360	180
Mo												
CPY Enriched	0.227	0.247	0.584	3	185	67	280	85	26	480	320	120
CPY Leached	0.234	0.470	0.611	3	185	67	87	65	23	500	490	50
CPY Primary	0.049	0.066	0.610	3	5	67	235	250	40	780	490	145
DPY Primary	0.115	0.149	0.774	12	300	110	304	200	75	500	480	200
HFL Enriched	0.163	0.001	1.04	0	0	70	267	175	120	700	550	300
HFL Leached	0.150	0.144	0.705	7	30	67	77	314	33	485	475	65
HFL Primary	0.059	0.979	NA	20	30	67	640	560	180	NA	NA	NA
MPY Primary	0.110	0.077	0.816	12	0	154	500	300	10	600	480	190
Ag												
CPY Enriched	0.124	0.521	NA	5	0	90	380	230	110	NA	NA	NA
CPY Leached	0.126	0.002	0.109	5	0	90	136	140	30	440	280	55
CPY Primary	0.104	0.014	0.343	70	180	115	250	220	143	410	410	240
DPY Primary	0.225	0.797	NA	17	28	70	490	400	200	NA	NA	NA
HFL Enriched	0.152	0.740	NA	0	0	70	830	590	200	NA	NA	NA
HFL Leached	0.182	0.169	0.274	0	0	150	590	370	24	708	444	100
HFL Primary	0.195	0.026	0.491	20	180	135	770	625	60	880	630	180
MPY Primary	0.163	0.837	NA	11	90	90	500	440	200	NA	NA	NA

14.8 Block Model Parameters

A 3D block model was developed to represent the deposit using a block size of 25 m x 25 m x 15 m. The block model is un-rotated and contains no sub-blocking. The block model parameters are shown in Table 14-5.

Table 14-5: Block Model Parameters

Parameter	X (Easting)	Y (Northing)	Z (Elevation / RL)
Origin	188850	8564600	3520
Number of blocks	74	80	168

14.9 Grade Estimation

Estimation within mineralized domain boundaries was performed using an inverse distance squared (ID2) method. Two estimation passes were performed:

- For the first pass:
  - For Cu, the search distances were set to a minimum of 200 meters or the variogram range in each direction. Search parameters were set to a minimum of 4 samples, a maximum of 12 samples, and a limit of 3 samples per drillhole. Variable anisotropy was applied for each of the leached and enriched domains.
  - For Mo and Ag, the search distances were set to the variogram range in each direction. Search parameters were set to a minimum of 4 samples, a maximum of 12 samples, and a limit of 3 samples per drillhole. Variable anisotropy was applied for each of the leached and enriched domains.
- For the second pass, the search distances were set to the variogram range in each direction. Search parameters were set to a minimum of 2 samples, a maximum of 6 samples, and either a limit of 2 samples per drillhole or no limit on drillholes. Variable anisotropy was applied for each of the leached and enriched domains.

The ID2 estimation parameters for each estimation domain are given in Table 14-10.

Table 14-6: Aurora Deposit Estimation and Search Parameters

Estimation Domain	Pass	Dynamic Anisotropy	Major	Semi-Major	Minor	Min Samples	Max Samples	Max Samples per Drillhole
Cu								
CPY Enriched	Pass 1	Yes	200	200	200	4	12	3
	Pass 2	Yes	450	410	230	2	6	2
CPY Leached	Pass 1	Yes	200	200	80	4	12	3
	Pass 2	Yes	640	615	80	2	6	2
CPY Primary	Pass 1	No	200	200	200	4	12	3
	Pass 2	No	600	475	200	2	6	-
DPY Primary	Pass 1	No	200	200	133	4	12	3
	Pass 2	No	725	400	133	2	6	2
HFL Enriched	Pass 1	Yes	200	200	200	4	12	3

Estimation Domain	Pass	Dynamic Anisotropy	Major	Semi-Major	Minor	Min Samples	Max Samples	Max Samples per Drillhole
	Pass 2	Yes	700	660	270	2	6	2
HFL Leached	Pass 1	Yes	200	200	90	4	12	3
	Pass 2	Yes	715	675	90	2	6	-
HFL Primary	Pass 1	No	200	200	200	4	12	3
	Pass 2	No	720	720	270	2	6	2
MPY Primary	Pass 1	No	200	200	180	4	12	3
	Pass 2	No	550	360	180	2	6	2
Mo								
CPY Enriched	Pass 1	Yes	480	320	120	4	12	3
	Pass 2	Yes	480	320	120	2	6	-
CPY Leached	Pass 1	Yes	500	490	50	4	12	3
	Pass 2	Yes	500	490	50	2	6	-
CPY Primary	Pass 1	No	780	490	145	4	12	3
	Pass 2	No	780	490	145	2	6	-
DPY Primary	Pass 1	No	500	480	200	4	12	3
	Pass 2	No	500	480	200	2	6	-
HFL Enriched	Pass 1	Yes	700	550	300	4	12	3
	Pass 2	Yes	700	550	300	2	6	-
HFL Leached	Pass 1	Yes	485	475	65	4	12	3
	Pass 2	Yes	485	475	65	2	6	-
HFL Primary	Pass 1	No	640	560	180	4	12	3
	Pass 2	No	640	560	180	2	6	-
MPY Primary	Pass 1	No	600	480	190	4	12	3
	Pass 2	No	600	480	190	2	6	-
Ag								
CPY Enriched	Pass 1	Yes	380	230	110	4	12	3
	Pass 2	Yes	380	230	110	2	6	-
CPY Leached	Pass 1	Yes	440	280	55	4	12	3
	Pass 2	Yes	440	280	55	2	6	-
CPY Primary	Pass 1	No	410	410	240	4	12	3
	Pass 2	No	410	410	240	2	6	-
DPY Primary	Pass 1	No	490	400	200	4	12	3
	Pass 2	No	490	400	200	2	6	-
HFL Enriched	Pass 1	Yes	830	590	200	4	12	3
	Pass 2	Yes	830	590	200	2	6	-
HFL Leached	Pass 1	Yes	708	444	100	4	12	3
	Pass 2	Yes	708	444	100	2	6	-
HFL Primary	Pass 1	No	880	630	180	4	12	3
	Pass 2	No	880	630	180	2	6	-
MPY Primary	Pass 1	No	500	440	200	4	12	3
	Pass 2	No	500	440	200	2	6	-

After each domained estimator was constructed, a combined estimator was used to assign a hierarchical value to each domained estimation to produce a single metal value.

14.10 Resource

14.10.1 Block Model Validation

Validation of the estimated block grades for the Aurora deposit was completed for each of the estimation domains. The resource block model estimate was validated by:

- Completing a series of visual inspections by comparisons of Cu, Mo, and Ag assay and composite grades to estimated block values across the deposit in vertical sections.
- Statistical comparison of parameters such as means, quantiles, and variance between 10-m composites, Nearest Neighbor (NN), Inverse Distance squared (ID2), and Ordinary Kriged (OK) estimators to ensure that the grade estimations are representative of the composites they are based on.
- Comparing average composite sample values with average estimated block grades along east, north, and elevation orientations using swath grade trend plots.

14.10.1.1 Visual Inspection

The model was examined in section views to compare to drill hole locations and grades. Section views for each of the estimation domains are shown in Figure 14-15 though Figure 14-20. Comparison of the model grade from the assays did not reveal any major discrepancies.

Figure 14-15: Aurora Deposit Cu Cross Section

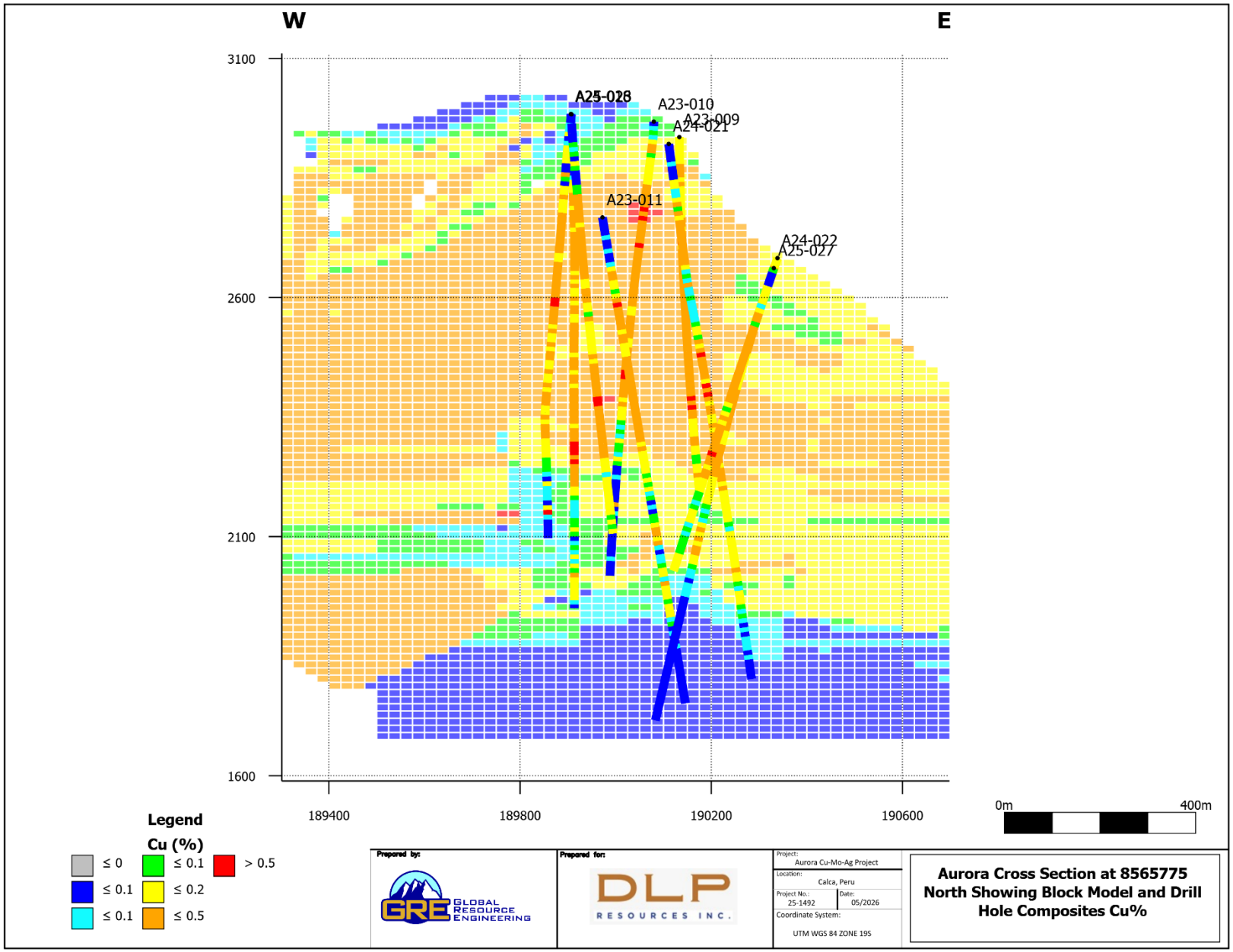


Figure 14-16: Aurora Deposit Cu Long Section

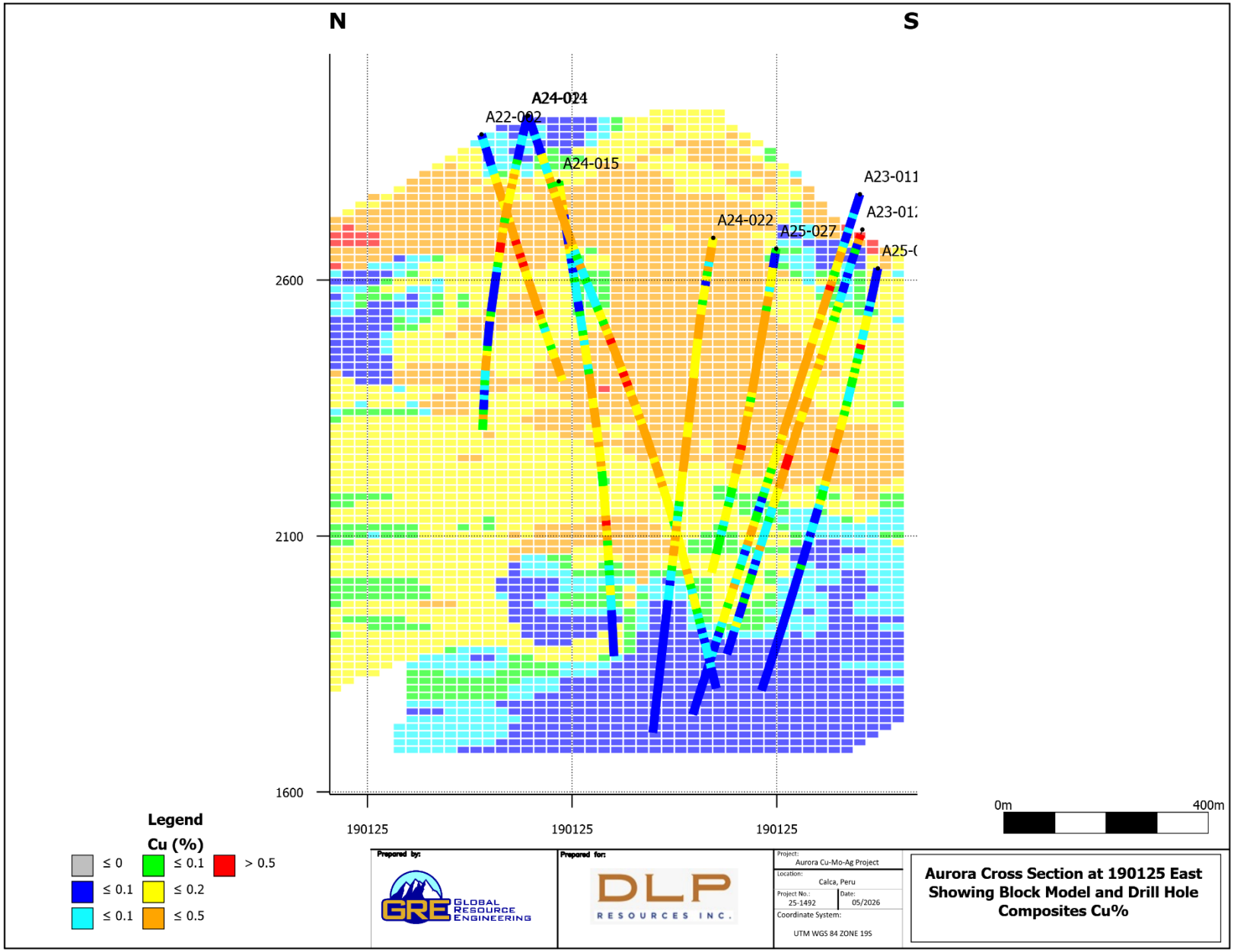


Figure 14-17: Aurora Deposit Mo Cross Section

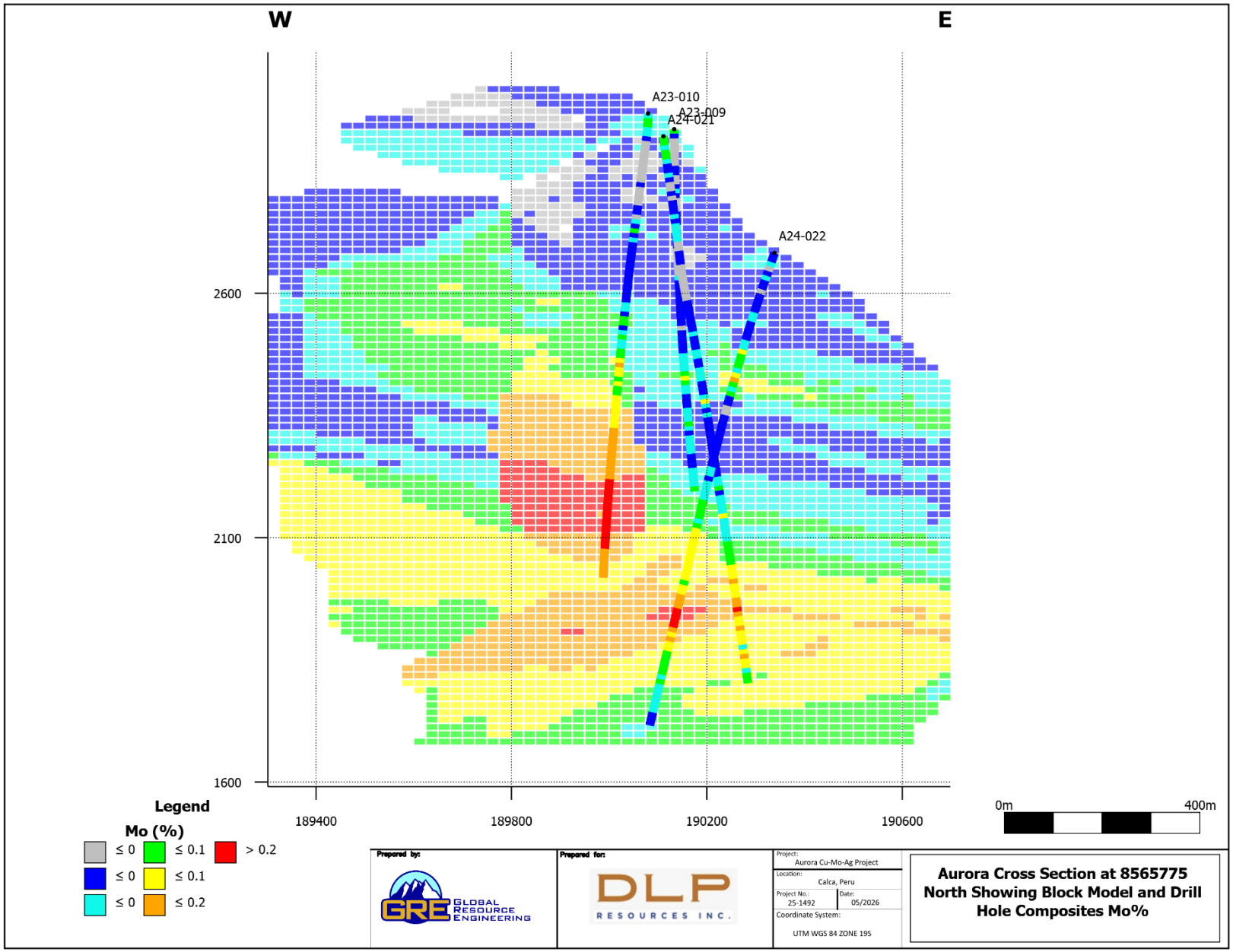


Figure 14-18: Aurora Deposit Mo Long Section

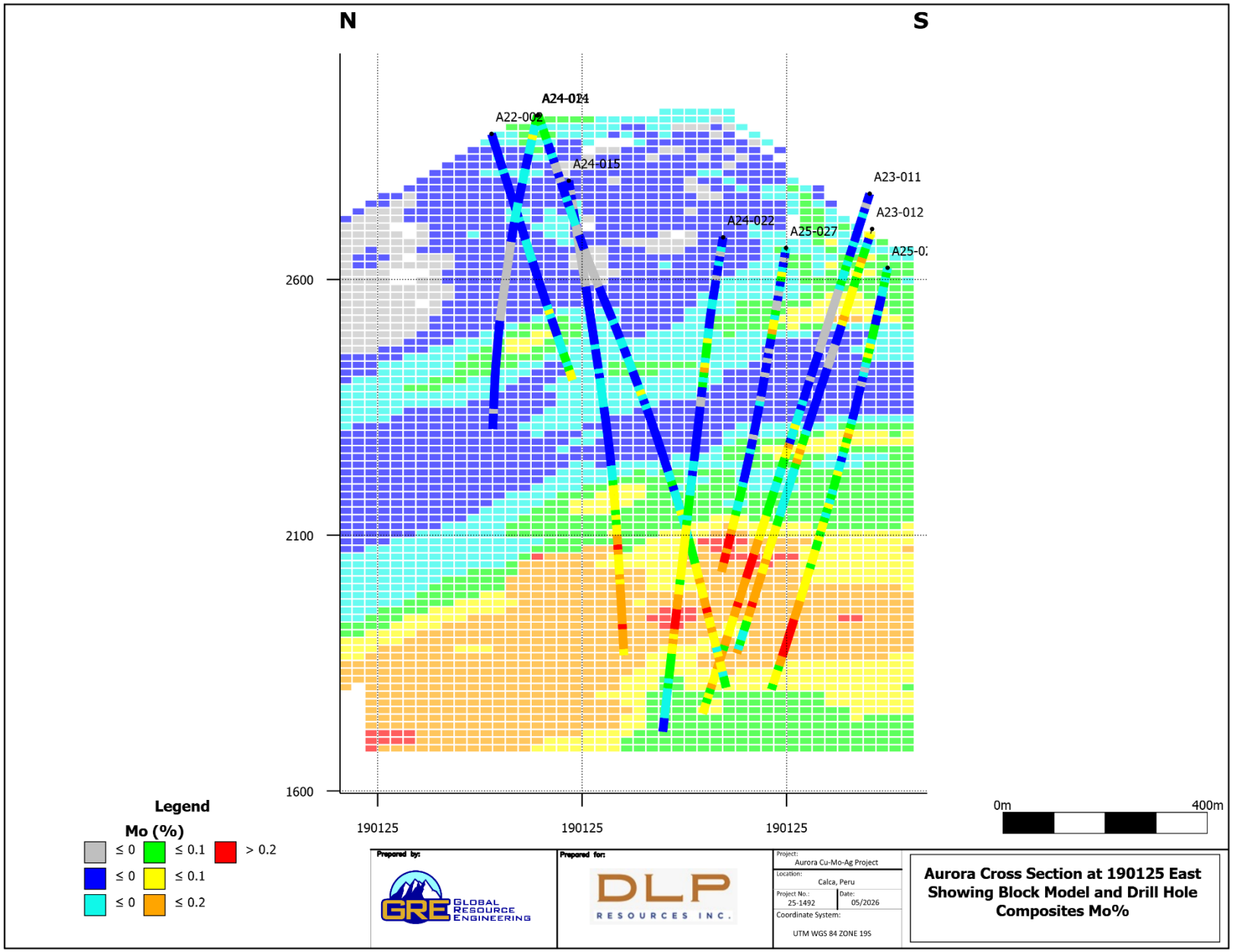


Figure 14-19: Aurora Deposit Ag Cross Section

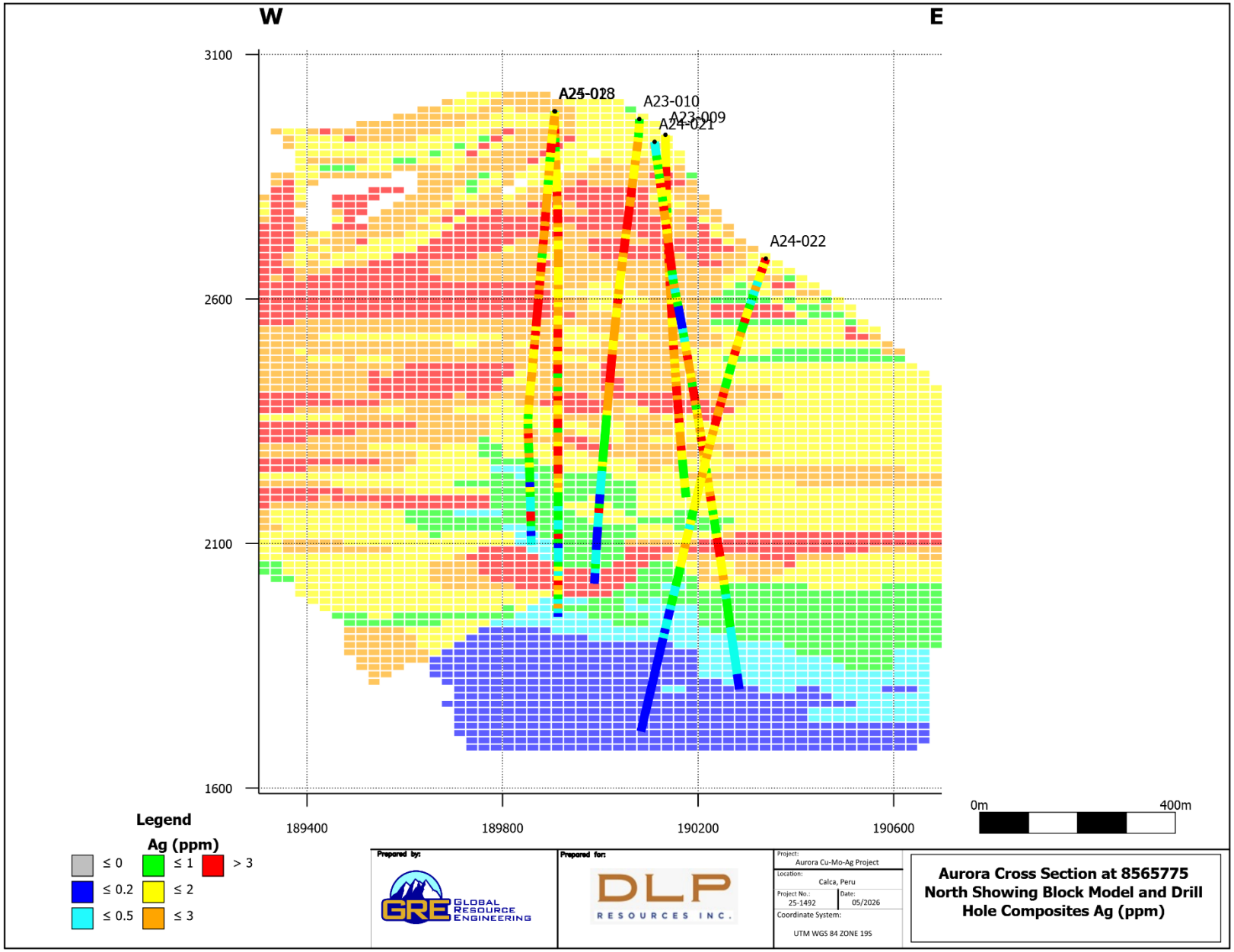
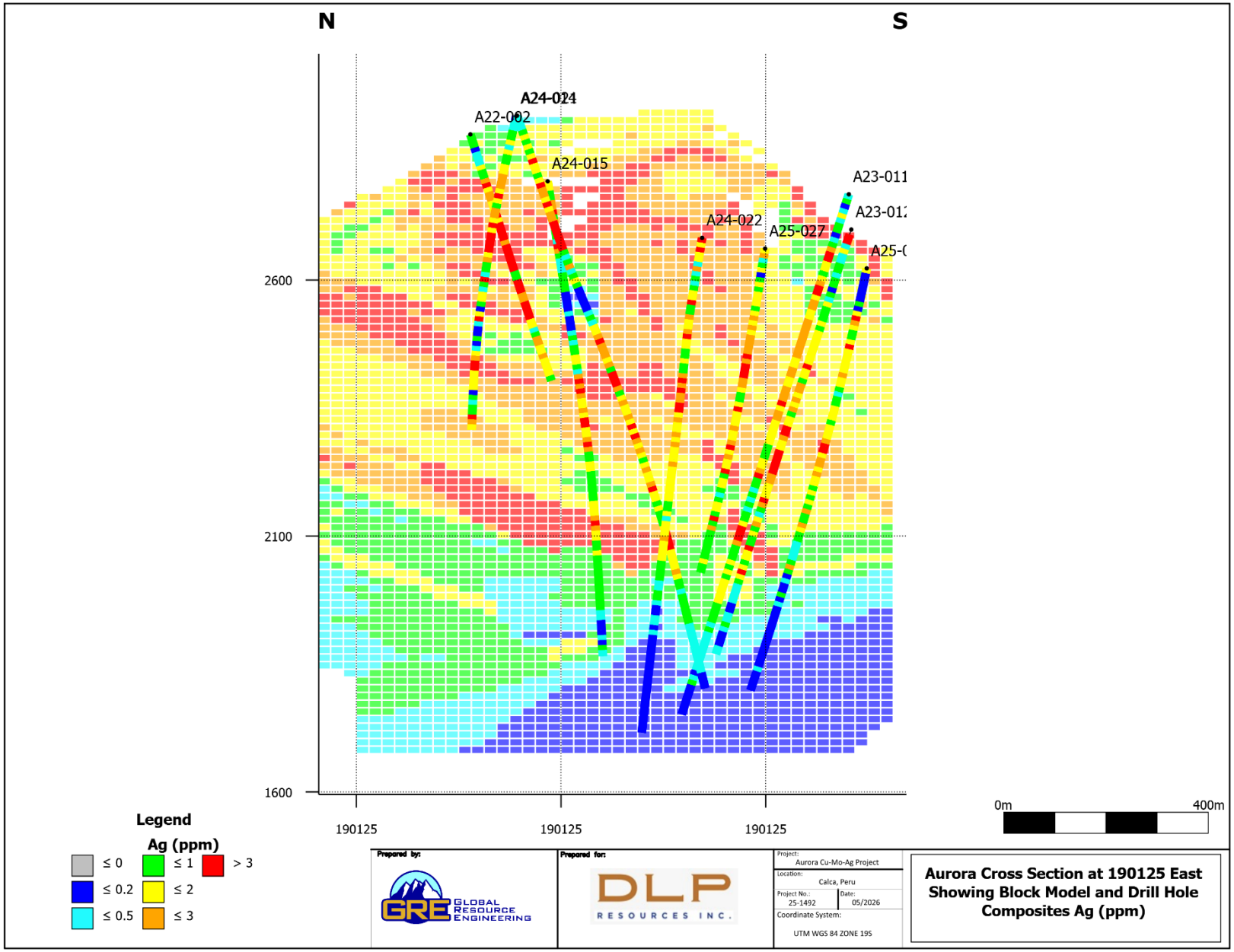


Figure 14-20: Aurora Deposit Ag Long Section



14.10.1.2 Statistical Comparison

To ensure that the grade estimations are representative of the composites they are based on and to validate the resource estimation results, the block model grade estimation statistics were analyzed. Ms. Lane of GRE compared the means, quantiles, and variance between 10-m composites, NN, ID2, and OK estimators, as shown in Table 14-7.

Table 14-7: Aurora Deposit Comparison of Composite Values to Grade Estimation Methods

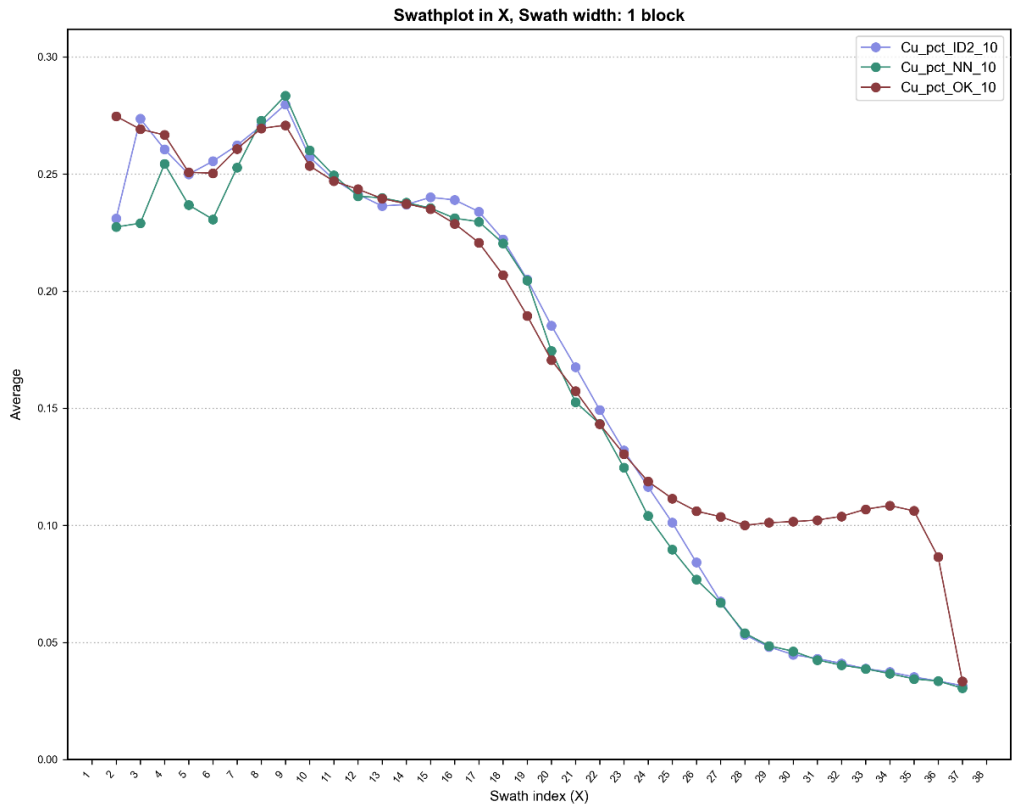
Parameter	Composites	Parameter	NN	ID2	OK
Copper					
Count	2,250	Block Count	180,436	168,539	168,842
Mean	0.167	Mean	0.162	0.151	0.148
SD	0.163	SD	0.175	0.115	0.111
CV	0.976	CV	1.087	0.766	0.747
Variance	0.027	Variance	0.031	0.013	0.012
Minimum	0.002	Minimum	0.002	0.0027	-0.0.02
Q1	0.054	Q1	0.042	0.056	0.06
Q2	0.117	Q2	0.097	0.012	0.12
Q3	0.24	Q3	0.21	0.23	0.22
Maximum	1.67	Maximum	1.0	1.12	1.11
Molybdenum					
Count	2,250	Block Count	174,510	154,818	153,518
Mean	0.050	Mean	0.055	0.055	0.054
SD	0.076	SD	0.073	0.068	0.066
CV	1.51	CV	1.34	1.24	1.23
Variance	0.006	Variance	0.005	0.005	0.004
Minimum	3.2e-05	Minimum	3.32e-05	5.75e-05	-0.006
Q1	0.002	Q1	0.003	0.007	0.007
Q2	0.013	Q2	0.023	0.028	0.027
Q3	0.069	Q3	0.071	0.070	0.071
Maximum	0.47	Maximum	0.47	0.38	0.36
Silver					
Count	2,250	Block Count	173,410	159,124	159,124
Mean	1.89	Mean	1.70	1.73	1.76
SD	3.00	SD	1.99	1.55	1.59
CV	1.59	CV	1.17	0.90	0.91
Variance	9.04	Variance	3.95	2.41	2.53
Minimum	0.021	Minimum	0.052	0.033	-1.44
Q1	0.52	Q1	0.38	0.61	0.61
Q2	1.21	Q2	0.99	1.40	1.40
Q3	2.33	Q3	2.06	2.36	2.37
Maximum	68.88	Maximum	10.8	46.66	21.48

As expected, the NN estimator generates more blocks due to the lack of restrictions of having to use multiple samples and multiple drillholes to estimate a block grade. Both ID2 and OK produced similar quantiles, means, variance, etc. The one marked difference seen between ID2 and OK is that is possible to have a negative value in Kriging as seen in the minimum value of the OK estimator.

14.10.1.3 Swath Plots

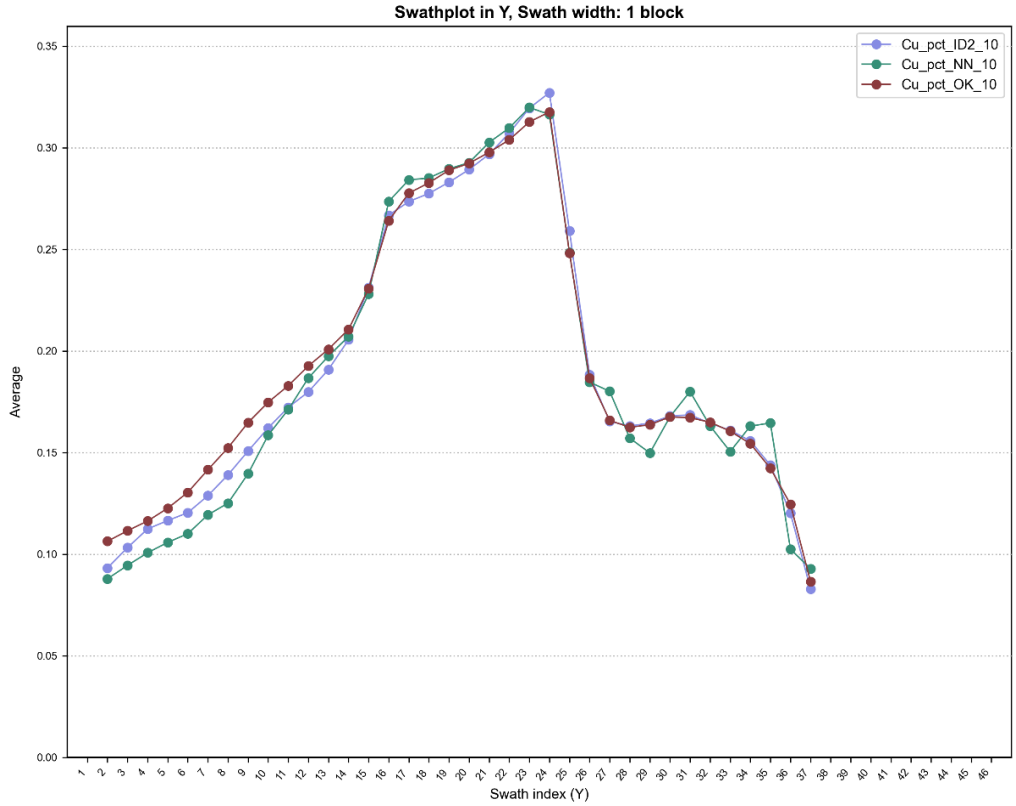
Swath plots of the three estimation methods (NN, ID2, and OK) were used to compare the results from each estimation method to the composite values and examine which method smoothed the estimated grades. As an example, swath plots are provided for the CPY Primary Cu, Mo, and Ag domains in Figure 14-21 through Figure 14-29. The swath plots show a general trend that the ID2 estimator smoothed out drastic swings in grade while not over smoothing local variability.

Figure 14-21: Aurora Deposit Swath Plot X axis, CPY Primary Domain Cu



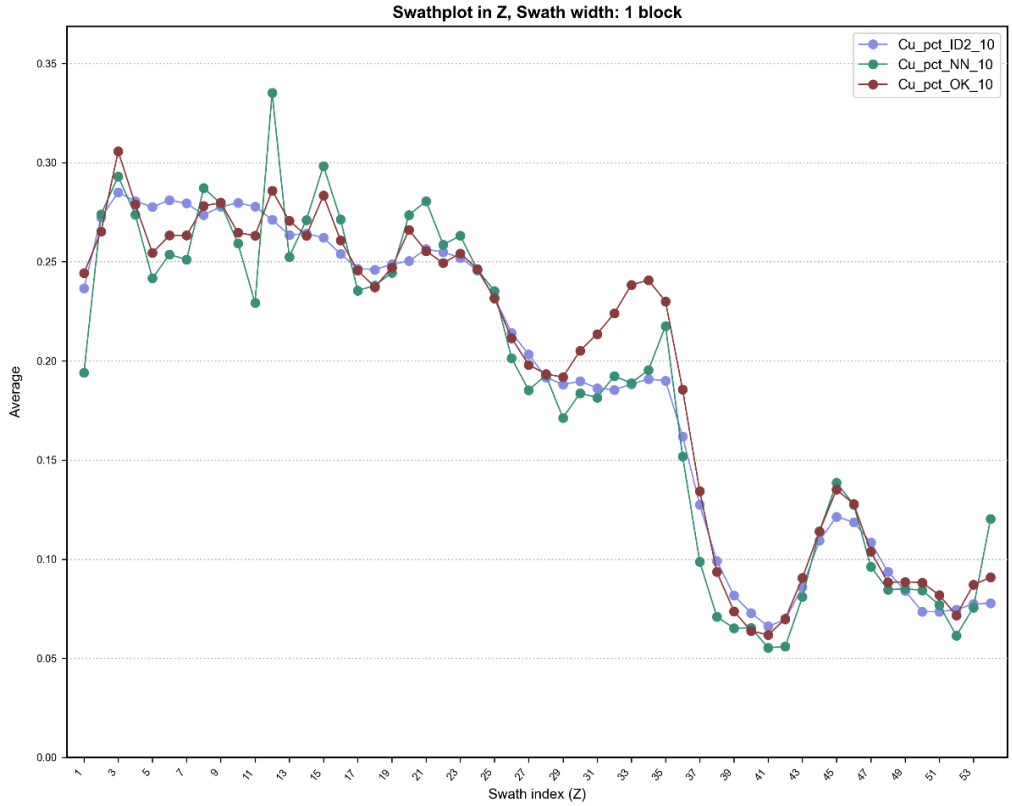
Source: GRE, 2026

Figure 14-22: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Cu



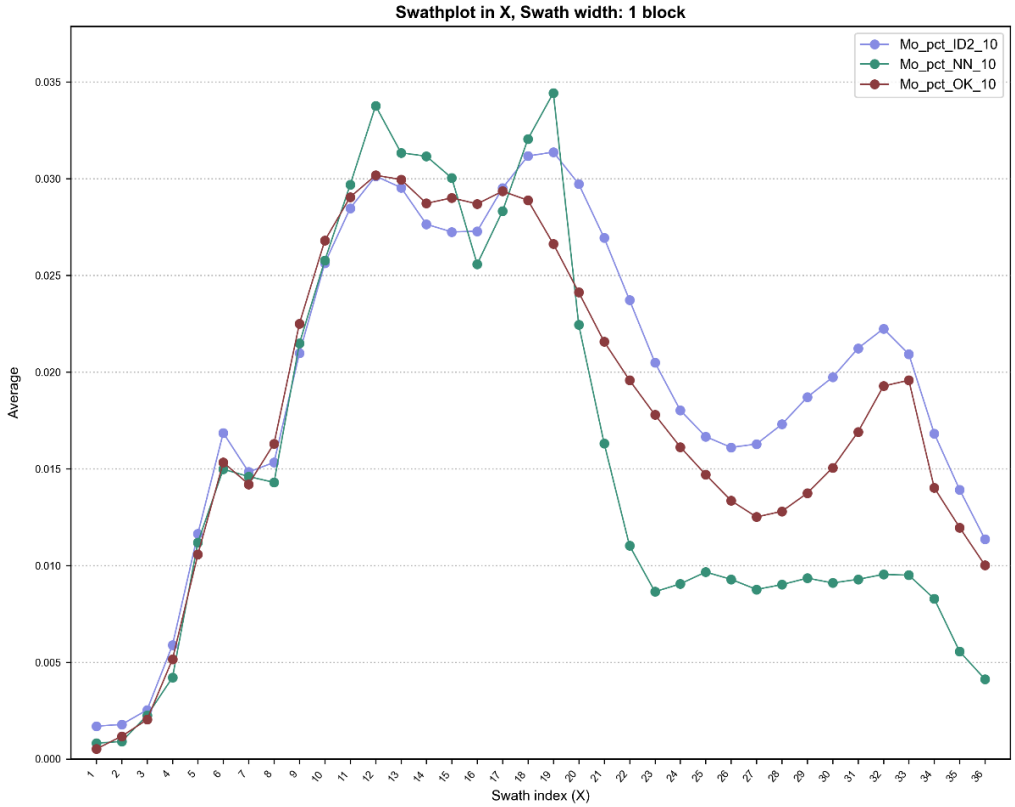
Source: GRE, 2026

Figure 14-23: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Cu



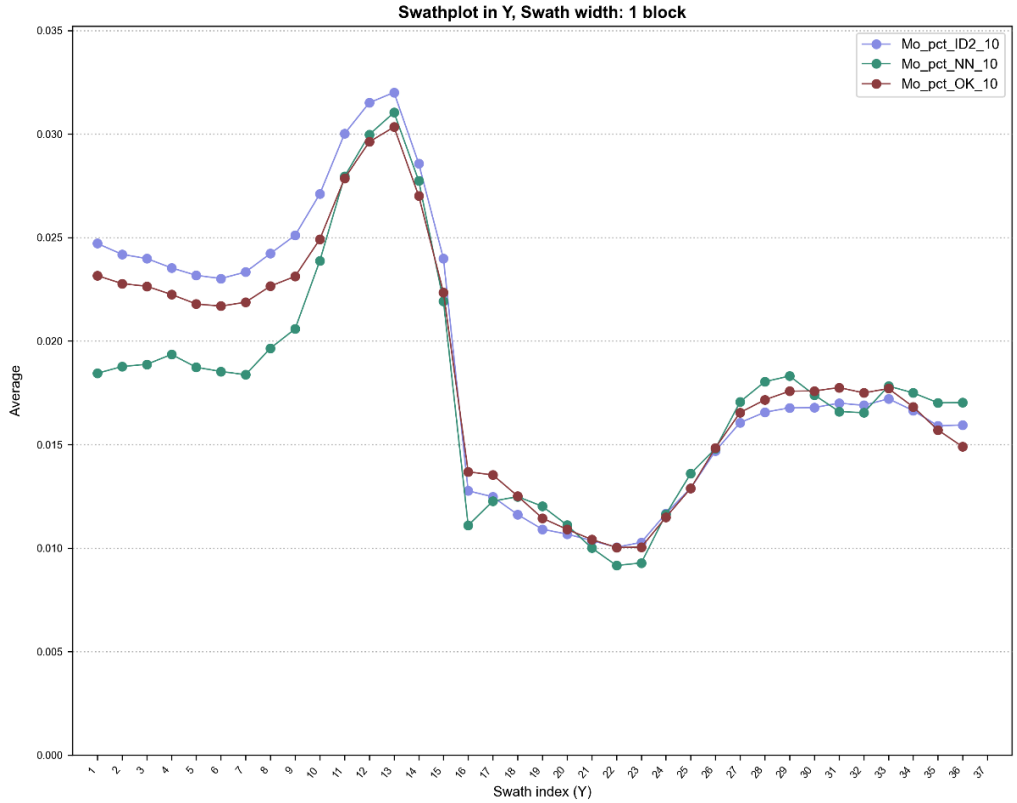
Source: GRE, 2026

Figure 14-24: Aurora Deposit Swath Plot X axis, CPY Primary Domain Mo



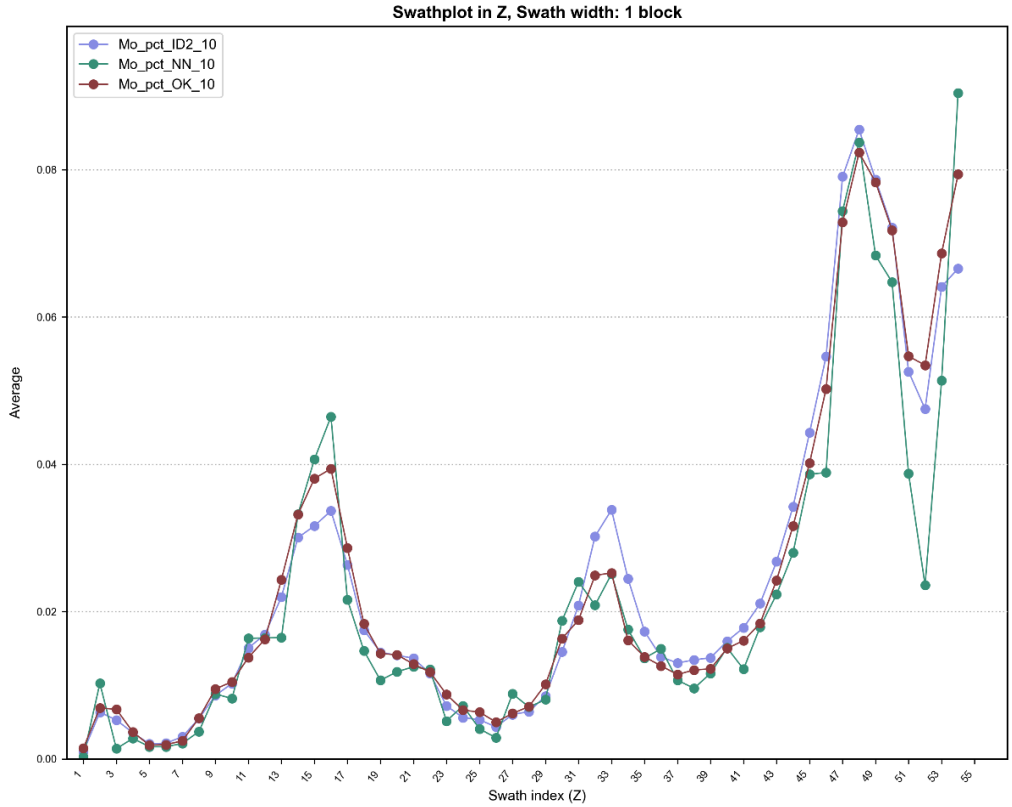
Source: GRE, 2026

Figure 14-25: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Mo



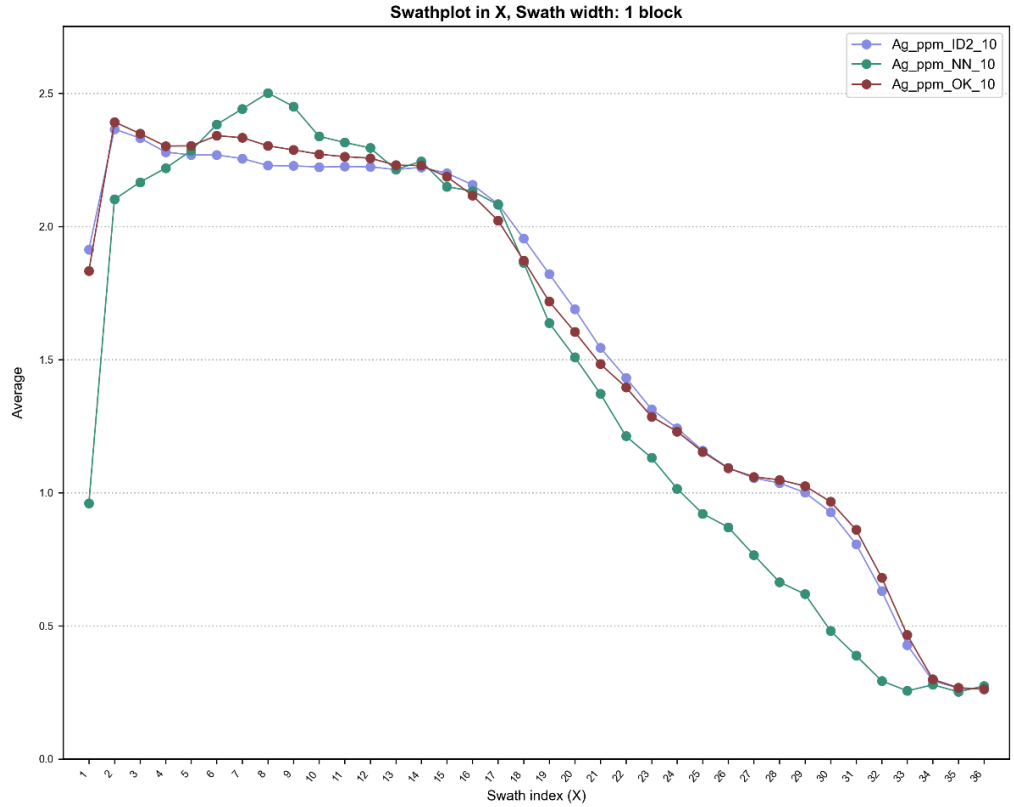
Source: GRE, 2026

Figure 14-26: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Mo



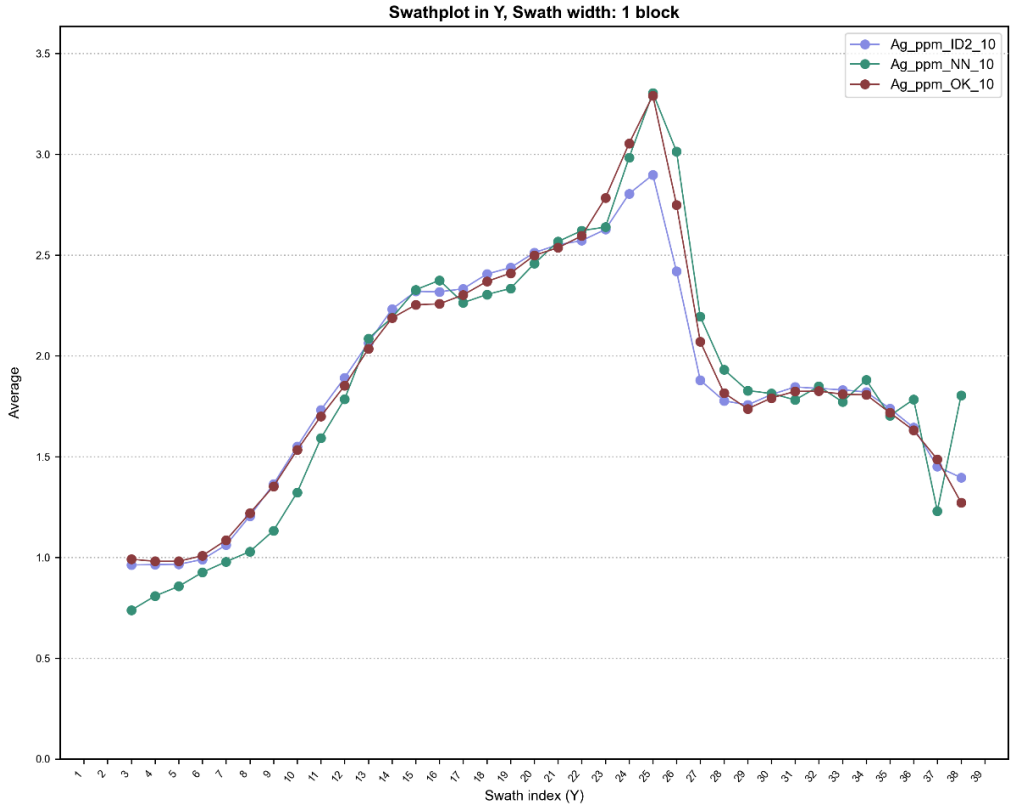
Source: GRE, 2026

Figure 14-27: Aurora Deposit Swath Plot X axis, CPY Primary Domain Ag



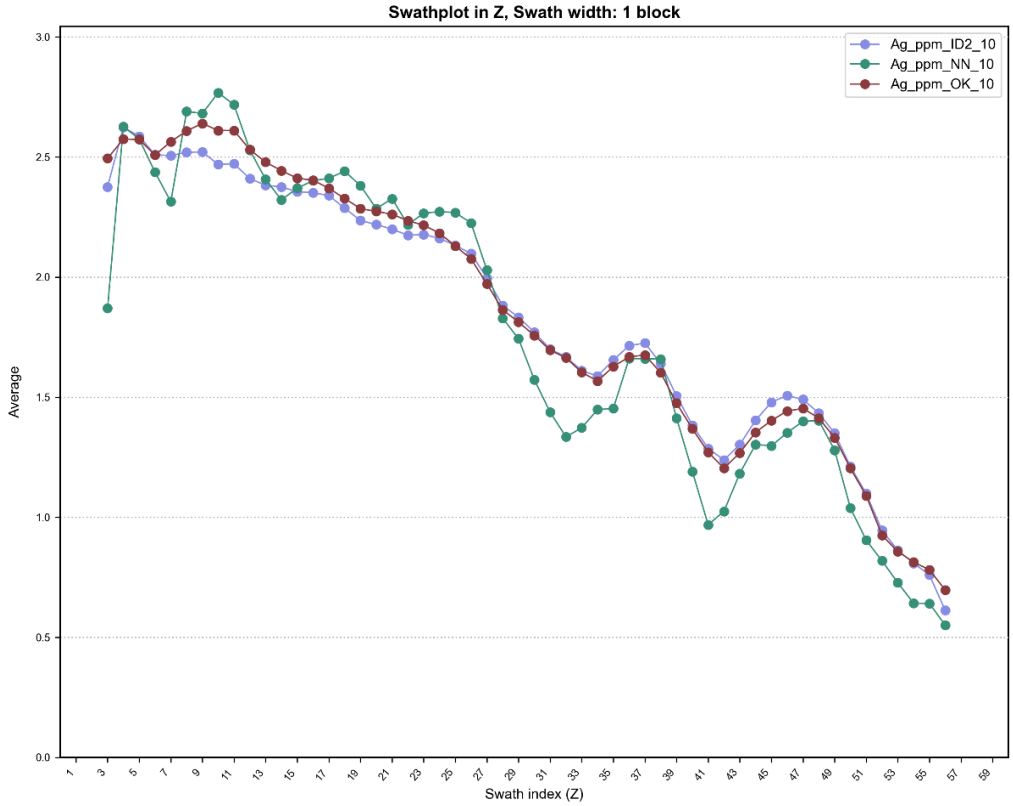
Source: GRE, 2026

Figure 14-28: Aurora Deposit Swath Plot Y axis, CPY Primary Domain Ag



Source: GRE, 2026

Figure 14-29: Aurora Deposit Swath Plot Z axis, CPY Primary Domain Ag



Source: GRE, 2026

14.11 Mineral Resource Classification

Multiple factors have been considered in the classification of the Aurora Mineral Resource estimate. Factors considered in the classification include data completeness and verification, geological continuity, statistical confidence, and drillhole spacing. Data completeness and verification as well as geological confidence are discussed in more detail below.

14.11.1 Data Completeness and Verification

The dataset that informs the Mineral Resource estimate is of acceptable quality. The assay results have been obtained through independent laboratories with appropriate quality certifications. The analytical processes are appropriate for the deposit style, and the integrity of the results is supported with appropriate QA/QC. Density measurements have been obtained at a suitable frequency and are considered fit for purpose.

14.11.2 Geological Confidence

Confidence has been established in the geological data, and continuity is present in the geological model. Observations and interpretations of lithological and mineralogical characteristics made by DLP geologists in the field through core logging activities are supported by geochemical observations.

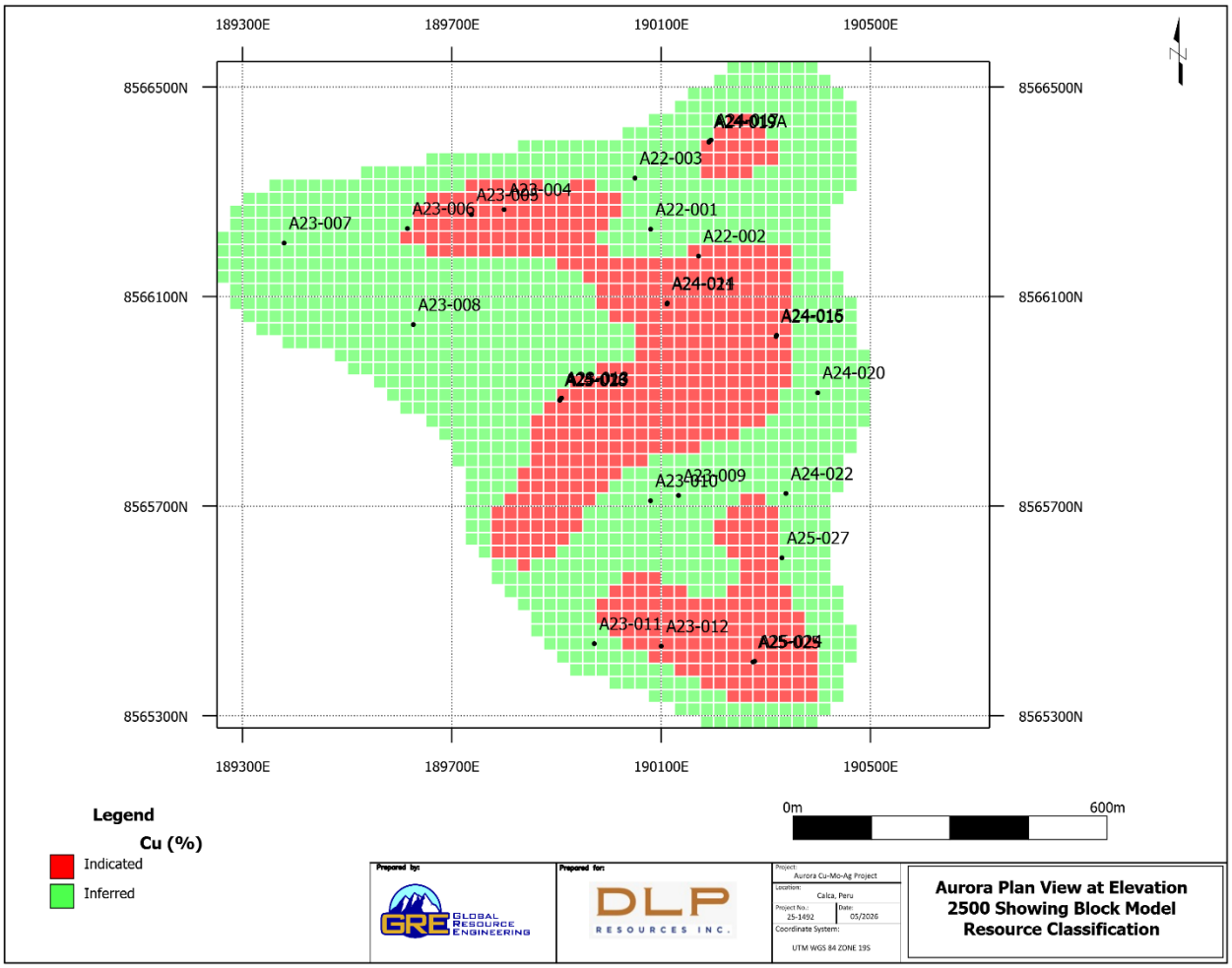
The resulting geological domains have formed a series of continuous domains which form a coherent 3D geological model. The geological domains are shown to form unique populations of grades, demonstrating the geological controls on mineralization.

14.11.3 Final Classification

Blocks within an average anisotropic distance of 110 meters of a drillhole for each Pass 1 were coded as Indicated. For the Inferred, a NN estimator was built with search ranges of 150 meters in each direction. Blocks within the 150-meter NN search were coded as Inferred. Wireframe models were then constructed around the identified Indicated and Inferred blocks to ensure continuity and completeness and to reduce spottiness.

A plan view of the estimated resource classes is shown in Figure 14-30.

Figure 14-30: Aurora Deposit Resource Class Plan View



14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) defines a mineral resource as: "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling." The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic, and other factors. Mineral resources are not mineral reserves and do not have demonstrated economic viability. Mineral reserves can only be estimated based on the results of an economic evaluation as part of a Preliminary Feasibility Study or Feasibility Study. As a result, no mineral reserves have been estimated as part of this study. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

The requirement, "reasonable prospects for eventual economic extraction," generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are

reported at a cutoff grade considering appropriate extraction scenarios and processing recoveries. To meet this requirement, Ms. Lane of GRE considered that major portions of the Aurora deposit are amenable for open pit or underground extraction.

14.12.1 Open Pit

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by an open pit, Ms. Lane of GRE constructed open pit scenarios developed from the resource block model estimate using Whittle's Lerchs-Grossman miner "Pit Optimizer" software. Reasonable mining assumptions were applied to evaluate the portions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit. The optimization parameters presented in Table 14-8 were selected based on experience and benchmarking against similar projects. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cutoff grade. Ms. Lane of GRE considers that the blocks located within the resulting conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a mineral resource.

A full breakdown of parameters used in the optimization are presented in Table 14-8.

Table 14-8: Aurora Deposit Pit Optimization Parameters

Parameter	Description	Units	Cost
Base Mining Costs	Mining Costs	$US / t mined	$ 2.00
Mineralization Costs	G&A	$US / t processed	$ 0.70
	Processing Cost	$US / t processed	$ 6.20
Copper			
Processing Parameters	Copper Metallurgical Recovery	%	Variable based on Cu grade and other factors as determined by metallurgical testing
	Copper Concentrate Grade	% (dry basis)	25.0%
Payable Copper	Payable Copper	%	96.50%
Concentrate Costs (TC/RC)	Combined Transportation	$US / dmt of con	$ 169.35
	Smelting costs	$US / dmt of con	$ 25.00
	Refining	$US / dmt of con	$ 44.251
Metal Price	Copper Price	$US / lb Cu	$ 4.37
Silver			
Processing	Metallurgical Recovery	%	Variable based on Ag grade and other factors as determined by metallurgical testing
Payable Silver	Payable Silver	%	96.00%
(TC/RC)			
	Refining	$US / dmt of con	$ 3.40
Metal Price	Silver Price	$US / oz troy Ag	$ 37.00
Molybdenum			
Processing	Metallurgical Recovery	%	Variable based on Mo grade and other factors

Parameter	Description	Units	Cost
			as determined by metallurgical testing
	Moly Concentrate Grade	% (dry basis)	55.0%
Payable Moly	Payable Moly	%	96.00%
Concentrate Costs	Combined Transportation	$US / dmt of con	$ 236.62
TC/RC	Roasting	$US / lb Mo	$ 0.08
	Refining	$US / lb Mo (recovered payable)	$ 0.08
Metal Price	Moly Price	$US / lb Mo	$ 22.10
Pit Parameters			
Slope Angle		Degrees	45
Royalty			$0

Dmt = dry metric tonne

lb = pound

It should be noted that the metal prices, recoveries, and concentrate costs used to develop optimized pit shells may differ from those used in the economic analysis, which was developed after additional detailed work on the project.

NSR values were calculated for each block in the block model using the parameters specified above. The Whittle pit optimizer was run on the NSR values.

Due to the large ratio of deposit size to block size and method of grade estimation, the grade model is fully diluted, and the resource is 100% recoverable as estimated.

The Aurora open pit mineral resource constrained by a Whittle pit shell that corresponds to a Revenue Factor (RF) of 1 and truncated at an elevation of 2835 is shown in Table 14-9. The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for eventual economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves on the project.

Table 14-9: Aurora Open Pit Mineral Resource

Class	Estimation Domain	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated	CPY Enriched	57.80	0.22	275.49	0.009	11.87	2.46	4.56	375.18
	CPY Leached	10.52	0.12	28.32	0.006	1.49	1.44	0.49	40.13
	CPY Primary	76.21	0.26	433.57	0.010	17.61	2.27	5.57	571.87
	DPY Primary	0.00				0.00	0.00	0.00	0.00
	HFL Enriched	65.00	0.33	467.83	0.018	25.12	3.23	6.75	655.61
	HFL Leached	31.98	0.15	107.06	0.023	16.48	3.05	3.13	219.94
	HFL Primary	107.52	0.21	495.03	0.021	48.92	2.33	8.07	820.33
	MPY Primary	33.56	0.24	178.02	0.059	43.92	2.64	2.85	435.15
Total Indicated		**382.58**	**0.24**	**1,985.32**	**0.020**	**165.39**	**2.55**	**31.42**	**3,118.21**
Inferred	CPY Enriched	47.53	0.22	233.68	0.010	10.46	2.09	3.19	314.90
	CPY Leached	5.22	0.12	14.06	0.008	0.86	1.31	0.22	20.42
	CPY Primary	57.79	0.25	323.23	0.015	19.69	2.21	4.11	461.08

Class	Estimation Domain	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
	DPY Primary	0.00				0.00	0.00	0.00	0.00
	HFL Enriched	109.46	0.36	874.10	0.021	51.46	4.92	17.32	1,286.62
	HFL Leached	84.38	0.17	312.26	0.018	34.00	2.76	7.49	553.38
	HFL Primary	168.60	0.23	855.34	0.019	68.84	2.64	14.32	1,336.79
	MPY Primary	64.87	0.21	293.25	0.057	81.63	2.15	4.49	764.73
Total Inferred		**537.86**	**0.25**	**2,905.93**	**0.023**	**266.95**	**2.96**	**51.14**	**4,737.91**

1) The effective date of the Mineral Resources Estimate is April 30th, 2026.
2) The Qualified Persons for the estimate are Terre Lane QP-MMSA and Hamid Samari QP-MMSA of GRE.
3) Mineral resources are not mineral reserves and are not demonstrably economically recoverable.
4) Mineral resources are reported at a $6.9 NSR cutoff, calculated using a Cu price of $4.37/lb, a Mo price of $22.10/lb, a Ag price of $37/tr.oz., using variable recovery, a slope angle of 45 degrees, 0% royalty, processing cost $6.90 per tonne (includes admin).
5) Cu Equivalent (CuEq) calculated as Cu lbs + Mo lbs * (Mo price * Mo recovery * Mo payable) / (Cu price * Cu recovery * Cu payable) + Ag oz * (Ag price * Ag recovery * Ag payable) / (Cu price * Cu recovery * Cu payable)

Table 14-10 shows the sensitivity of the open pit constrained mineral resource to cutoff grade in each estimation domain.

Table 14-10: Aurora Open Pit Mineral Resource Sensitivity to Cutoff Grade

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	Cu Eq (million lbs)
Indicated - CPY Enriched								
4	58.80	0.213	276.24	0.009	12.07	2.422	4.58	377.13
5	58.63	0.214	276.14	0.009	12.05	2.428	4.58	376.87
6	58.29	0.215	275.90	0.009	11.98	2.440	4.57	376.24
6.9	**57.80**	**0.216**	**275.49**	**0.009**	**11.87**	**2.455**	**4.56**	**375.18**
8	56.79	0.219	274.27	0.009	11.66	2.488	4.54	372.68
9	55.71	0.222	272.96	0.009	11.37	2.522	4.52	369.62
10	54.38	0.226	270.75	0.009	11.07	2.561	4.48	365.48
11	53.08	0.229	268.50	0.009	10.69	2.607	4.45	360.96
Indicated - CPY Leached								
4	13.33	0.110	32.17	0.007	1.99	1.364	0.58	47.42
5	12.26	0.115	31.01	0.006	1.65	1.389	0.55	44.18
6	11.44	0.118	29.88	0.006	1.56	1.412	0.52	42.35
6.9	**10.52**	**0.122**	**28.32**	**0.006**	**1.49**	**1.442**	**0.49**	**40.13**
8	8.71	0.129	24.77	0.006	1.24	1.486	0.42	34.72
9	6.12	0.142	19.13	0.006	0.78	1.511	0.30	25.67
10	3.80	0.160	13.39	0.005	0.45	1.589	0.19	17.32
11	2.52	0.178	9.85	0.005	0.29	1.725	0.14	12.50
Indicated - CPY Primary								
4	76.23	0.258	433.61	0.010	17.61	2.272	5.57	571.92
5	76.23	0.258	433.61	0.010	17.61	2.272	5.57	571.92
6	76.23	0.258	433.61	0.010	17.61	2.272	5.57	571.92
6.9	**76.21**	**0.258**	**433.57**	**0.010**	**17.61**	**2.272**	**5.57**	**571.87**

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	Cu Eq (million lbs)
8	75.96	0.259	433.17	0.011	17.59	2.277	5.56	571.34
9	75.56	0.260	432.44	0.011	17.57	2.284	5.55	570.39
10	75.14	0.261	431.57	0.011	17.55	2.290	5.53	569.26
11	74.72	0.261	430.57	0.011	17.52	2.296	5.52	567.99
Indicated - HFL Enriched								
4	65.18	0.326	468.13	0.017	25.12	3.222	6.75	655.96
5	65.18	0.326	468.13	0.017	25.12	3.222	6.75	655.96
6	65.13	0.326	468.05	0.017	25.12	3.224	6.75	655.88
6.9	**65.00**	**0.326**	**467.83**	**0.018**	**25.12**	**3.229**	**6.75**	**655.61**
8	64.94	0.327	467.72	0.018	25.12	3.230	6.74	655.49
9	64.76	0.327	467.31	0.018	25.11	3.236	6.74	654.99
10	64.65	0.328	467.02	0.018	25.11	3.240	6.73	654.67
11	64.55	0.328	466.75	0.018	25.11	3.242	6.73	654.32
Indicated - HFL Leached								
4	45.60	0.124	125.11	0.020	19.61	2.599	3.81	260.12
5	41.38	0.132	120.86	0.021	18.95	2.716	3.61	250.76
6	37.89	0.139	116.27	0.022	17.98	2.827	3.44	239.66
6.9	**31.98**	**0.152**	**107.06**	**0.023**	**16.48**	**3.047**	**3.13**	**219.94**
8	23.76	0.176	92.28	0.027	14.29	3.473	2.65	189.67
9	20.42	0.190	85.46	0.029	13.06	3.693	2.42	174.47
10	18.44	0.199	81.00	0.029	11.77	3.819	2.26	161.87
11	15.98	0.212	74.75	0.029	10.24	4.009	2.06	145.82
Indicated - HFL Primary								
4	127.37	0.186	520.95	0.018	49.28	2.120	8.68	853.13
5	123.17	0.190	516.61	0.018	49.24	2.159	8.55	847.49
6	115.00	0.200	506.16	0.019	49.10	2.244	8.30	834.31
6.9	**107.52**	**0.209**	**495.03**	**0.021**	**48.92**	**2.333**	**8.07**	**820.33**
8	100.28	0.218	482.56	0.022	48.69	2.425	7.82	804.68
9	95.16	0.225	472.48	0.023	48.51	2.489	7.62	792.05
10	91.67	0.230	464.85	0.024	48.36	2.530	7.46	782.37
11	88.65	0.234	457.74	0.025	48.16	2.569	7.32	773.12
Indicated - MPY Primary								
4	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
5	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
6	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
6.9	**33.56**	**0.241**	**178.02**	**0.059**	**43.92**	**2.645**	**2.85**	**435.15**
8	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
9	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
10	33.56	0.241	178.02	0.059	43.92	2.645	2.85	435.15
11	33.53	0.241	177.96	0.059	43.91	2.646	2.85	435.06
Indicated Total								
4	420.06	0.220	2,034.23	0.018	169.60	2.431	32.83	3,200.85
5	410.41	0.224	2,024.37	0.019	168.53	2.460	32.46	3,182.33
6	397.53	0.229	2,007.88	0.019	167.27	2.504	32.01	3,155.50
6.9	**382.58**	**0.235**	**1,985.32**	**0.020**	**165.39**	**2.554**	**31.42**	**3,118.21**

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	Cu Eq (million lbs)
8	363.99	0.243	1,952.79	0.020	162.51	2.614	30.59	3,063.73
9	351.28	0.249	1,927.80	0.021	160.33	2.656	29.99	3,022.33
10	341.64	0.253	1,906.60	0.021	158.23	2.687	29.51	2,986.12
11	333.02	0.257	1,886.12	0.021	155.92	2.715	29.07	2,949.77
Inferred - CPY Enriched								
4	47.80	0.222	234.03	0.010	10.49	2.077	3.19	315.41
5	47.75	0.222	233.97	0.010	10.49	2.078	3.19	315.34
6	47.65	0.223	233.81	0.010	10.48	2.082	3.19	315.16
6.9	**47.53**	**0.223**	**233.68**	**0.010**	**10.46**	**2.086**	**3.19**	**314.90**
8	47.31	0.224	233.29	0.010	10.44	2.093	3.18	314.34
9	46.87	0.225	232.43	0.010	10.38	2.107	3.17	313.10
10	46.18	0.227	231.06	0.010	10.26	2.126	3.16	310.95
11	45.42	0.229	229.46	0.010	10.10	2.148	3.14	308.33
Inferred - CPY Leached								
4	7.18	0.106	16.75	0.009	1.36	1.251	0.29	26.29
5	6.48	0.112	15.98	0.008	1.10	1.277	0.27	23.98
6	5.88	0.117	15.16	0.008	0.98	1.285	0.24	22.32
6.9	**5.22**	**0.122**	**14.06**	**0.008**	**0.86**	**1.306**	**0.22**	**20.42**
8	4.23	0.130	12.10	0.007	0.67	1.363	0.19	17.16
9	3.24	0.139	9.90	0.007	0.49	1.419	0.15	13.67
10	2.35	0.149	7.68	0.005	0.28	1.452	0.11	10.03
11	1.57	0.159	5.52	0.005	0.16	1.514	0.08	6.99
Inferred - CPY Primary								
4	57.79	0.254	323.23	0.015	19.69	2.215	4.11	461.08
5	57.79	0.254	323.23	0.015	19.69	2.215	4.11	461.08
6	57.79	0.254	323.23	0.015	19.69	2.215	4.11	461.08
6.9	**57.79**	**0.254**	**323.23**	**0.015**	**19.69**	**2.215**	**4.11**	**461.08**
8	57.62	0.254	322.94	0.015	19.68	2.219	4.11	460.69
9	57.47	0.255	322.68	0.016	19.67	2.222	4.11	460.35
10	57.32	0.255	322.37	0.016	19.66	2.226	4.10	459.94
11	56.85	0.256	321.26	0.016	19.63	2.236	4.09	458.55
Inferred - HFL Enriched								
4	110.04	0.361	875.07	0.021	51.47	4.900	17.34	1,287.79
5	110.01	0.361	875.03	0.021	51.47	4.901	17.33	1,287.75
6	109.94	0.361	874.91	0.021	51.47	4.904	17.33	1,287.61
6.9	**109.46**	**0.362**	**874.10**	**0.021**	**51.46**	**4.921**	**17.32**	**1,286.62**
8	107.99	0.366	871.27	0.022	51.42	4.971	17.26	1,283.10
9	107.42	0.367	870.03	0.022	51.40	4.989	17.23	1,281.52
10	106.68	0.369	868.30	0.022	51.36	5.012	17.19	1,279.27
11	105.97	0.371	866.56	0.022	51.30	5.034	17.15	1,276.89
Inferred - HFL Leached								
4	98.25	0.152	329.77	0.018	38.26	2.606	8.23	599.51
5	92.83	0.159	324.49	0.018	36.94	2.667	7.96	585.03
6	89.78	0.162	320.60	0.018	35.82	2.701	7.80	573.83
6.9	**84.38**	**0.168**	**312.26**	**0.018**	**34.00**	**2.760**	**7.49**	**553.38**

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	Cu Eq (million lbs)
8	76.91	0.176	298.77	0.018	31.02	2.852	7.05	520.46
9	71.18	0.183	286.54	0.018	28.67	2.940	6.73	493.12
10	65.58	0.189	273.23	0.018	26.43	3.022	6.37	465.03
11	60.03	0.196	258.95	0.018	23.49	3.081	5.95	431.64
Inferred - HFL Primary								
4	193.86	0.208	887.20	0.016	69.26	2.450	15.27	1,378.44
5	187.92	0.213	881.42	0.017	69.19	2.491	15.05	1,370.57
6	178.21	0.221	869.19	0.018	69.05	2.568	14.72	1,354.89
6.9	**168.60**	**0.230**	**855.34**	**0.019**	**68.84**	**2.642**	**14.32**	**1,336.79**
8	160.39	0.238	841.71	0.019	68.63	2.704	13.95	1,319.02
9	155.90	0.242	833.31	0.020	68.49	2.734	13.70	1,307.97
10	152.57	0.246	826.39	0.020	68.37	2.751	13.49	1,298.73
11	147.95	0.250	815.58	0.021	68.16	2.777	13.21	1,284.58
Inferred - MPY Primary								
4	64.87	0.205	293.25	0.057	81.63	2.153	4.49	764.73
5	64.87	0.205	293.25	0.057	81.63	2.153	4.49	764.73
6	64.87	0.205	293.25	0.057	81.63	2.153	4.49	764.73
6.9	**64.87**	**0.205**	**293.25**	**0.057**	**81.63**	**2.153**	**4.49**	**764.73**
8	64.87	0.205	293.25	0.057	81.63	2.153	4.49	764.73
9	64.85	0.205	293.20	0.057	81.63	2.153	4.49	764.66
10	64.77	0.205	293.06	0.057	81.62	2.155	4.49	764.43
11	64.72	0.205	292.95	0.057	81.61	2.155	4.49	764.26
Inferred Total								
4	579.80	0.232	2,959.30	0.021	272.16	2.839	52.92	4,833.25
5	567.67	0.236	2,947.37	0.022	270.51	2.872	52.41	4,808.48
6	554.11	0.240	2,930.15	0.022	269.12	2.912	51.88	4,779.61
6.9	**537.86**	**0.245**	**2,905.93**	**0.023**	**266.95**	**2.957**	**51.14**	**4,737.91**
8	519.32	0.251	2,873.32	0.023	263.48	3.008	50.23	4,679.51
9	506.92	0.255	2,848.09	0.023	260.72	3.042	49.58	4,634.40
10	495.44	0.258	2,822.09	0.024	257.96	3.071	48.91	4,588.39
11	482.51	0.262	2,790.28	0.024	254.44	3.100	48.09	4,531.24

14.12.2 Underground

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by underground extraction methods, Ms. Lane of GRE considered all blocks below the open pit described in Section 14.12.1 and with NSR values above a cutoff grade of $25 to be amenable to economic extraction by underground mining methods.

The Aurora underground mineral resource constrained by being outside the Whittle pit shell corresponding to an RF of 1 and truncated at elevation 2835 and being above and NSR cutoff of $25 is shown in Table 1-3. The reader is cautioned that the results do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves on the project.

Table 14-11: Aurora Out-of-Pit Mineral Resource

Class	Estimation Domain	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated	CPY Primary	30.24	0.25	169.24	0.05	36.53	2.44	2.37	383.07
	DPY Primary	45.02	0.05	45.38	0.13	124.28	0.34	0.50	712.72
	HFL Primary	41.21	0.24	219.94	0.04	34.09	3.18	4.21	435.42
	MPY Primary	115.79	0.08	194.03	0.19	493.14	0.74	2.74	2,848.10
Total Indicated		**232.27**	**0.12**	**628.59**	**0.134**	**688.03**	**1.32**	**9.82**	**4,379.31**
Inferred	CPY Primary	45.32	0.21	205.98	0.05	48.88	1.80	2.62	487.76
	DPY Primary	217.72	0.06	277.41	0.12	579.52	0.42	2.91	3,393.90
	HFL Primary	154.01	0.21	702.71	0.06	187.98	2.07	10.24	1,787.58
	MPY Primary	163.89	0.06	221.37	0.19	676.54	0.61	3.21	3,858.08
Total Inferred		**580.94**	**0.11**	**1,407.47**	**0.117**	**1,492.91**	**1.02**	**18.98**	**9,527.32**

1) The effective date of the Mineral Resources Estimate is April 30th, 2026.
2) The Qualified Persons for the estimate are Terre Lane QP-MMSA and Hamid Samari QP-MMSA of GRE.
3) Mineral resources are not mineral reserves and are not demonstrably economically recoverable.
4) Mineral resources are reported at a $25 NSR cutoff, calculated using a Cu price of $4.37/lb, a Mo price of $22.10/lb, a Ag price of $37/tr.oz.
5) Cu Equivalent (CuEq) calculated as Cu lbs + Mo lbs * (Mo price * Mo recovery * Mo payable) / (Cu price * Cu recovery * Cu payable) + Ag oz * (Ag price * Ag recovery * Ag payable) / (Cu price * Cu recovery * Cu payable)

Table 14-12 shows the sensitivity of the out-of-pit constrained mineral resource to cutoff grade in each estimation domain.

Table 14-12: Aurora Out-of-Pit Mineral Resource Sensitivity to Cutoff Grade

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated - CPY Primary								
10	46.28	0.210	214.14	0.044	44.93	2.162	3.22	479.58
15	44.80	0.213	210.79	0.045	44.60	2.195	3.16	474.03
20	38.41	0.232	196.08	0.049	41.17	2.329	2.88	438.73
25	**30.24**	**0.254**	**169.24**	**0.055**	**36.53**	**2.436**	**2.37**	**383.07**
30	22.72	0.259	129.79	0.066	33.27	2.501	1.83	321.92
35	15.96	0.254	89.53	0.083	29.32	2.594	1.33	256.60
40	11.14	0.241	59.27	0.103	25.27	2.665	0.95	201.73
45	7.82	0.233	40.19	0.122	21.06	2.740	0.69	158.10
Indicated - DPY Primary								
10	61.001	0.038	51.01	0.104	139.62	0.286	0.56	800.75
15	58.798	0.039	50.68	0.107	138.19	0.295	0.56	792.72
20	52.877	0.042	48.72	0.114	133.00	0.315	0.54	762.88
25	**45.023**	**0.046**	**45.38**	**0.125**	**124.28**	**0.344**	**0.50**	**712.72**
30	40.618	0.047	42.42	0.132	118.44	0.351	0.46	678.26
35	35.871	0.049	38.78	0.140	110.90	0.361	0.42	634.06
40	30.390	0.051	34.13	0.151	100.88	0.367	0.36	575.48
45	25.203	0.053	29.31	0.162	90.07	0.377	0.31	512.53

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Indicated - HFL Primary								
10	115.03	0.189	478.04	0.024	60.79	2.262	8.36	869.08
15	107.81	0.193	458.85	0.025	59.54	2.311	8.01	840.42
20	78.04	0.211	362.22	0.029	50.26	2.564	6.43	681.70
25	**41.21**	**0.242**	**219.94**	**0.038**	**34.09**	**3.180**	**4.21**	**435.42**
30	24.52	0.256	138.36	0.047	25.46	3.836	3.02	298.34
35	15.36	0.262	88.77	0.057	19.19	4.499	2.22	208.87
40	9.71	0.260	55.77	0.066	14.22	5.261	1.64	144.77
45	5.22	0.279	32.08	0.073	8.37	6.306	1.06	85.20
Indicated - MPY Primary								
10	115.81	0.076	194.13	0.193	493.15	0.736	2.74	2,848.29
15	115.81	0.076	194.13	0.193	493.15	0.736	2.74	2,848.29
20	115.81	0.076	194.13	0.193	493.15	0.736	2.74	2,848.29
25	**115.79**	**0.076**	**194.03**	**0.193**	**493.14**	**0.736**	**2.74**	**2,848.10**
30	115.64	0.076	193.43	0.193	493.02	0.735	2.73	2,846.81
35	114.63	0.075	189.90	0.195	491.86	0.727	2.68	2,836.66
40	112.95	0.074	184.52	0.197	489.44	0.718	2.61	2,817.80
45	110.31	0.073	177.89	0.199	484.60	0.711	2.52	2,784.65
Indicated Total								
10	338.12	0.126	937.32	0.099	738.49	1.369	14.88	4,997.70
15	327.22	0.127	914.45	0.102	735.48	1.376	14.47	4,955.47
20	285.14	0.127	801.14	0.114	717.58	1.373	12.59	4,731.61
25	**232.27**	**0.123**	**628.59**	**0.134**	**688.03**	**1.315**	**9.82**	**4,379.31**
30	203.49	0.112	504.00	0.149	670.19	1.229	8.04	4,145.34
35	181.82	0.102	406.99	0.162	651.27	1.137	6.65	3,936.19
40	164.19	0.092	333.69	0.174	629.82	1.054	5.56	3,739.78
45	148.56	0.085	279.48	0.184	604.11	0.958	4.58	3,540.49
Inferred - CPY Primary								
10	103.55	0.147	335.81	0.034	77.45	1.553	5.17	790.35
15	80.02	0.170	299.66	0.039	69.09	1.671	4.30	702.65
20	63.31	0.188	261.96	0.043	60.42	1.744	3.55	612.72
25	**45.32**	**0.206**	**205.98**	**0.049**	**48.88**	**1.800**	**2.62**	**487.76**
30	26.04	0.220	126.39	0.061	34.95	1.845	1.54	325.23
35	15.17	0.227	75.84	0.076	25.29	1.938	0.95	218.36
40	8.64	0.218	41.42	0.095	18.04	2.029	0.56	142.22
45	4.93	0.199	21.59	0.119	12.89	2.144	0.34	93.09
Inferred - DPY Primary								
10	283.348	0.048	301.69	0.102	638.99	0.354	3.23	3,738.13
15	273.781	0.049	297.34	0.105	633.30	0.360	3.17	3,702.98
20	238.326	0.054	286.27	0.115	602.22	0.395	3.02	3,524.86
25	**217.723**	**0.058**	**277.41**	**0.121**	**579.52**	**0.415**	**2.91**	**3,393.90**
30	193.572	0.061	260.08	0.128	547.60	0.437	2.72	3,204.67
35	173.655	0.063	241.59	0.135	516.71	0.450	2.52	3,019.67
40	155.254	0.065	223.31	0.141	483.43	0.463	2.31	2,822.12
45	139.374	0.068	207.43	0.147	450.55	0.471	2.11	2,629.17

Cutoff NSR ($)	Mass (million tonnes)	Cu Grade (%)	Cu Content (million lbs)	Mo Grade (%)	Mo Content (million lbs)	Ag Grade (ppm)	Ag Content (million t. oz.)	CuEq (million lbs)
Inferred - HFL Primary								
10	346.00	0.170	1,295.49	0.033	248.79	1.896	21.09	2,791.32
15	288.88	0.181	1,155.34	0.038	239.63	1.966	18.26	2,579.77
20	211.52	0.195	908.26	0.046	216.41	2.068	14.06	2,175.32
25	**154.01**	**0.207**	**702.71**	**0.055**	**187.98**	**2.067**	**10.24**	**1,787.58**
30	111.83	0.213	524.77	0.065	159.95	2.045	7.35	1,437.05
35	78.70	0.214	370.89	0.076	131.38	1.979	5.01	1,112.01
40	52.08	0.229	262.72	0.085	97.03	1.679	2.81	803.02
45	33.29	0.252	184.69	0.090	65.82	1.668	1.79	550.25
Inferred - MPY Primary								
10	163.89	0.061	221.37	0.187	676.54	0.609	3.21	3,858.08
15	163.89	0.061	221.37	0.187	676.54	0.609	3.21	3,858.08
20	163.89	0.061	221.37	0.187	676.54	0.609	3.21	3,858.08
25	**163.89**	**0.061**	**221.37**	**0.187**	**676.54**	**0.609**	**3.21**	**3,858.08**
30	163.69	0.061	220.73	0.187	676.34	0.607	3.19	3,856.28
35	161.40	0.060	213.61	0.189	673.52	0.594	3.08	3,833.24
40	158.66	0.059	205.05	0.191	669.51	0.585	2.98	3,802.49
45	155.51	0.057	196.32	0.194	663.80	0.579	2.90	3,762.56
Inferred Total								
10	896.79	0.109	2,154.36	0.083	1641.76	1.134	32.69	11,177.88
15	806.57	0.111	1,973.71	0.091	1618.55	1.116	28.94	10,843.49
20	677.05	0.112	1,677.86	0.104	1555.58	1.095	23.84	10,170.99
25	**580.94**	**0.110**	**1,407.47**	**0.117**	**1492.91**	**1.016**	**18.98**	**9,527.32**
30	495.13	0.104	1,131.97	0.130	1418.83	0.930	14.81	8,823.23
35	428.92	0.095	901.93	0.142	1346.90	0.837	11.55	8,183.28
40	374.63	0.089	732.51	0.154	1268.02	0.720	8.67	7,569.86
45	333.09	0.083	610.04	0.162	1193.06	0.666	7.13	7,035.08

15.0 MINERAL RESERVE ESTIMATES

"Mineral reserves" differ from "Mineral Resources" in that Mineral Reserves are known to be economically feasible for extraction. The CIM Definition Standards require the completion of a Preliminary Feasibility Study (PFS) as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. At this time, a PFS has not been completed for the Aurora Project. Therefore, reserve estimates have not been made.

16.0 MINING METHODS

A number of mining scenarios and strategies were evaluated for this PEA, including a single large open pit recovering the majority of the resource over a 35-year life and a large, multi-panel block caving mine that would recover a smaller percentage of the resource but focus on the higher-value, higher-grade molybdenum/copper ores at depth. The selected approach combines the advantages of both the surface mine, which can be started and producing economic medium grade copper mineralized material very quickly, simultaneously with the development of a large, single-panel block caving mine in the zone of highest grade, highest value mineralized material. This approach brings forward the production of higher value material, optimizing the economic value of the project. This strategy also leaves the opportunity to significantly expand the operation and to significantly extend its economic life.

The Aurora Cu-Mo-Ag Mine Project will employ conventional open pit mining techniques using hydraulic shovels and rear dump rigid frame haul trucks and underground block caving.

The mine plan is designed to deliver an average of 65,000 tonnes of mineralized material to the mill per day (tpd). Mining of the open pit will commence in year -3, during construction of the mill and other mine facilities and development for the underground. Mine scheduling was designed as follows:

- Open Pit
    - In years -3, -2, and -1, mineralized material mined from the open pit would be stockpiled for processing at a later time.
    - In Years 1 through 4, following commissioning of the mill, open pit mining would be conducted at an average rate of 131.8 thousand (k) tpd (48.1 million [M] tpa), with 65 ktpd (23.7 Mtpa) to be direct to the crusher and the remaining 66.8 ktpd (25.5 Mtpa) stockpiled.
    - In Years 5 through 18, stockpiled material from the open pit would supplement the underground mined material to reach the mill feed rate of 65,000 tpd (23.7 Mtpa).
    - There is a zone of resources between the bottom of the surface mine and the top of the projected Block Caving resources which will potentially be produced through the Block Cave. For conservatism, these resources are not included as production in this study but given that the underground draw points are in hard, stable rock, it is considered likely that some portion of these resources could continue to be produced through the Block Cave since their grade exceeds the cutoff value for underground mining.
- Underground
    - Underground development of the Block Cave mine commences simultaneously with the surface mine, but due to the longer development schedule, production from the underground mine does not commence until Year 3, with the mining of 533k tonnes.
    - In Years 4 through 7, the underground mining would ramp up to full capacity, mining 4.0 M tonnes in Year 4, 7.0 M tonnes in Year 5, 7.2 M tonnes in Year 6, and 12.0 M tonnes in Year 7.
    - In Years 8 through 17, the underground mining would be conducted at a rate of 39.5 ktpd (14.4 Mtpa).

16.1 Open Pit

The Pit was designed to operate from commencement of operations until a date approximately in year 7 when subsidence from the Block Cave below will require ceasing surface operations. The pit was designed to provide the needed amount of mineralized material to ensure a total feed of 65,000 tpd to the mill, of which some would come from the underground mining. The total open pit mineralized tonnage would be 237.6 M tonnes, averaging 96.8 Mtpd. Mineralized material produced in excess of the 65,000 tpd capacity of the plant would be stored in a stockpile and fed to the plant to supplement the underground production in order to average 65,000 tpd. The average daily waste production rate over the life of the mine would be 98.8 ktpd. Waste rock would be placed in a waste rock storage facility (WRSF).

Open pit mining would use conventional truck and hydraulic shovel mining techniques, using 227 tonne haul trucks and 22 cubic meter (m^3) hydraulic shovels.

16.1.1 Site Layout and Pit Access

Figure 16-1 shows the location of topographic features at the Aurora Project site. Given the approximate locations of the pit and WRSF, locations for a haul road from the pit to the WRSF, the primary crusher/equipment maintenance area, and ammonium nitrate fuel oil (ANFO) storage and magazines were developed, as shown on Figure 16-2.

Figure 16-1: Aurora Project Site Features

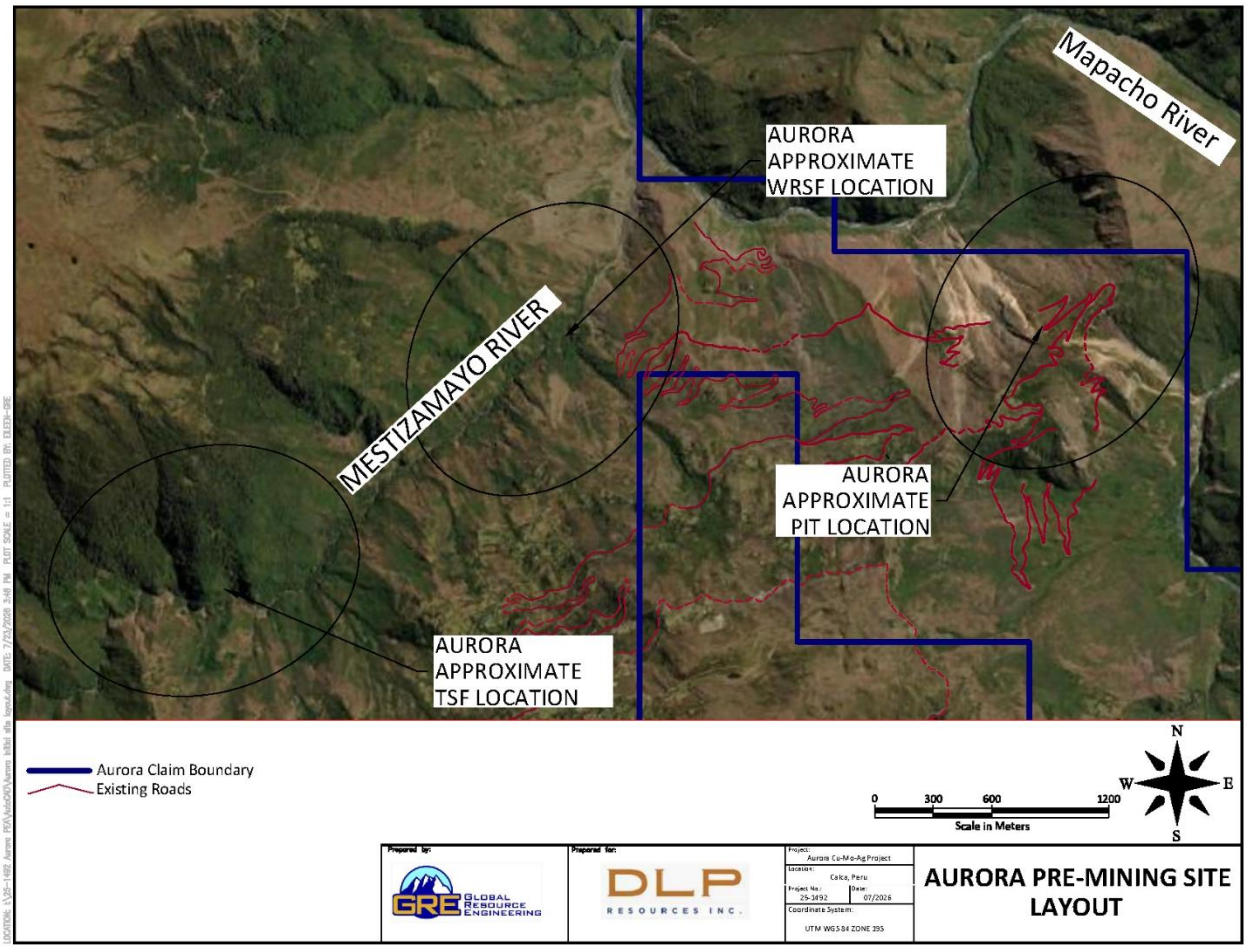
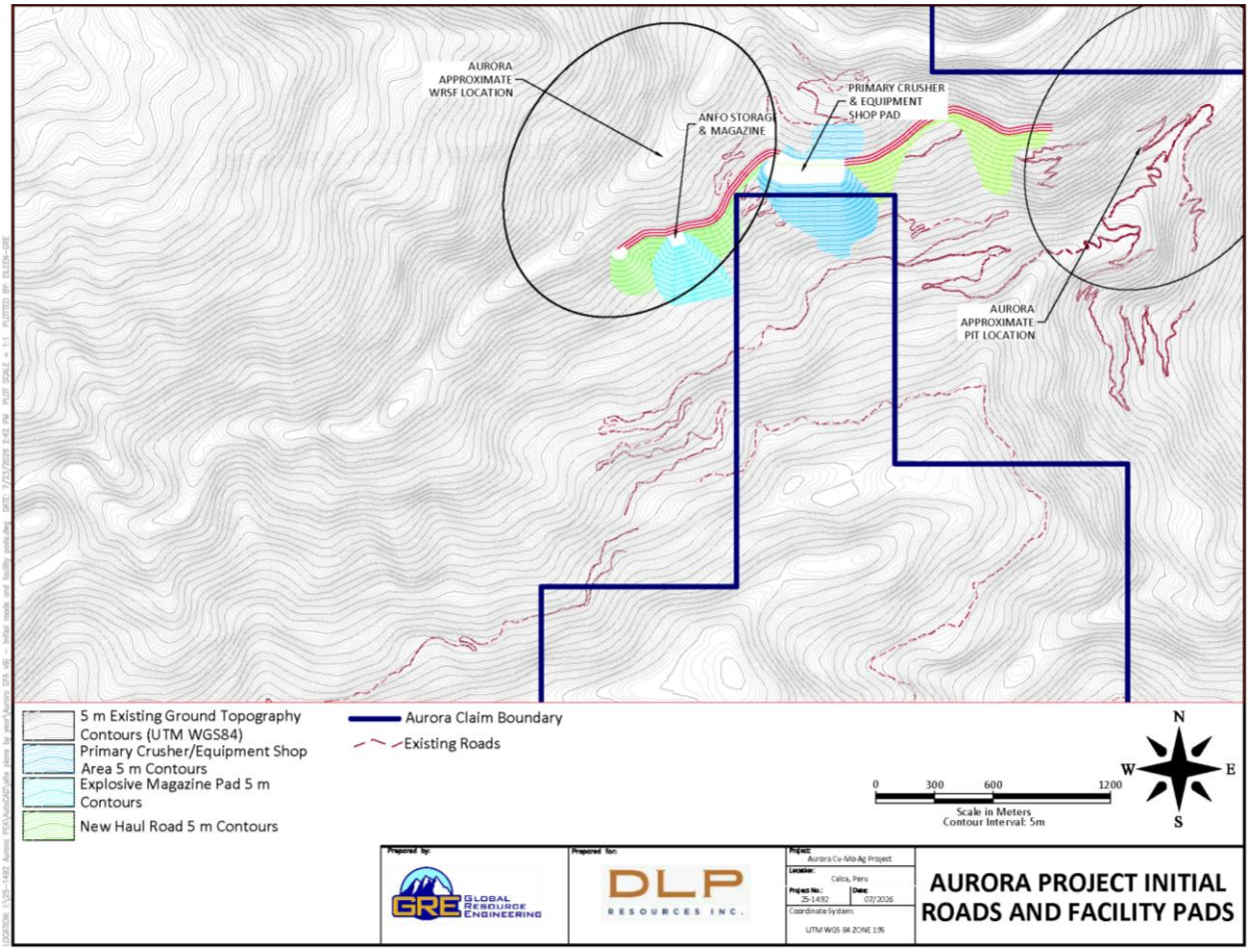


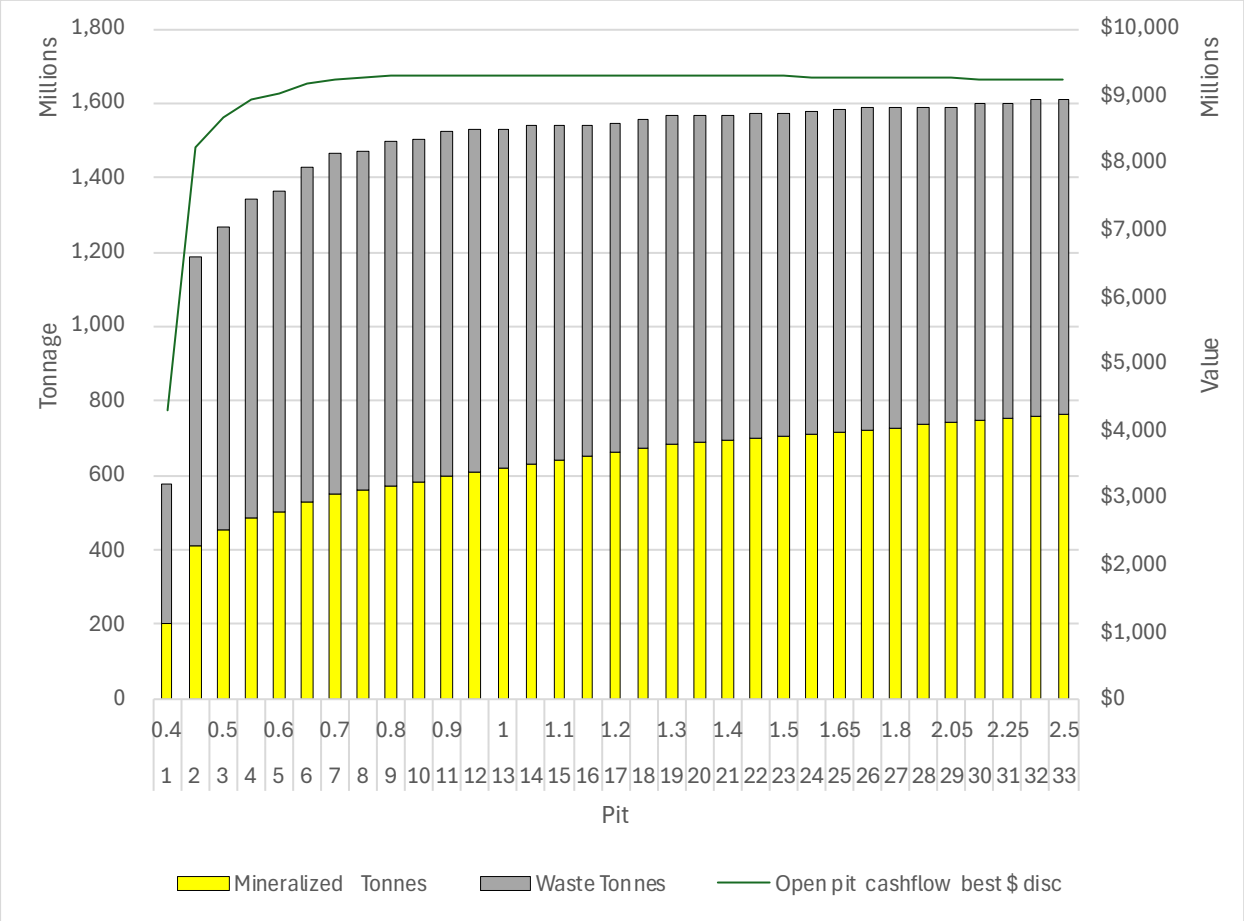
Figure 16-2: Aurora Project Initial Haul Road and Equipment Pads



16.1.2 Whittle Pit Shell Analysis

The parameters and formulae shown in Section 13.4, along with the parameters in Table 14-8, were used to generate NSR values for each block in the block model, which were then used to generate Whittle™ pseudoflow pit shells at revenue factors of 0.4 to 2.5 NSR in increments of 0.05, generating 33 pit shells. The output for the Whittle pit shells was analyzed to determine the optimal pit shell for economic extraction of the needed open pit mineral resources of 237.6 M tonnes. The Whittle output indicated 200.3 M tonnes from the revenue factor 0.4 pit, and 409.5 M tonnes from the revenue factor 0.45 pit, as shown in Figure 16-3. GRE based the pit design on the revenue factor 0.4 Whittle pit shell.

Figure 16-3: Aurora Project Open Pit Whittle Output



Source: GRE, 2026

16.1.3 Pit Design

Ms. Lane of GRE used a double bench format consisting of 15-meter vertical benches with a horizontal 16-meter catch bench. The resulting open pit parameters are listed in Table 16-1.

Table 16-1: Aurora Project Open Pit Geotechnical Design Parameters

Pit Design Parameter	Value (degrees)
Max Inter-ramp Angle Hard Rock	45
Max Bench Face Angle	68

The Whittle pit shell selected for ultimate mine design was imported into GEOVIA Surpac™ software and designed with pit slopes and benching as described above and with haul roads. Haul roads were designed with a minimum width of 30 meters and a maximum gradient of 10%. Haul ramps and roads have been designed to accommodate two-way traffic using 293-tonne haul trucks, water diversion ditches, and safety berms. Minor sections were narrowed to a single lane of 20 meters. A minimum mining width of 50 meters was used. The designed ultimate pit is illustrated in After the ultimate pit was designed, an access road to the upper benches of the pit was designed, as shown in Figure 16-5.

Figure 16-4.

After the ultimate pit was designed, an access road to the upper benches of the pit was designed, as shown in Figure 16-5.

Figure 16-4: Aurora Project Designed Ultimate Pit

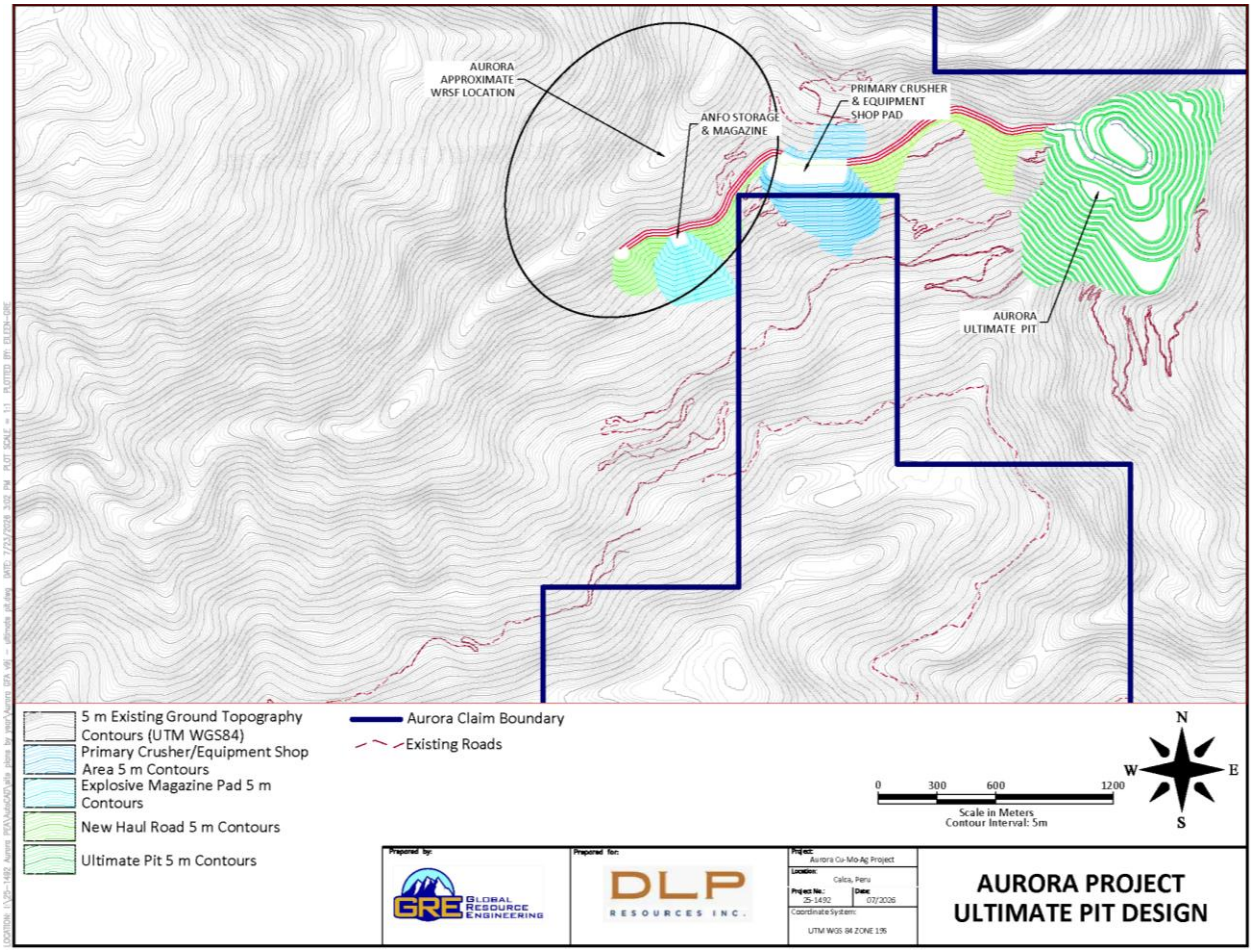
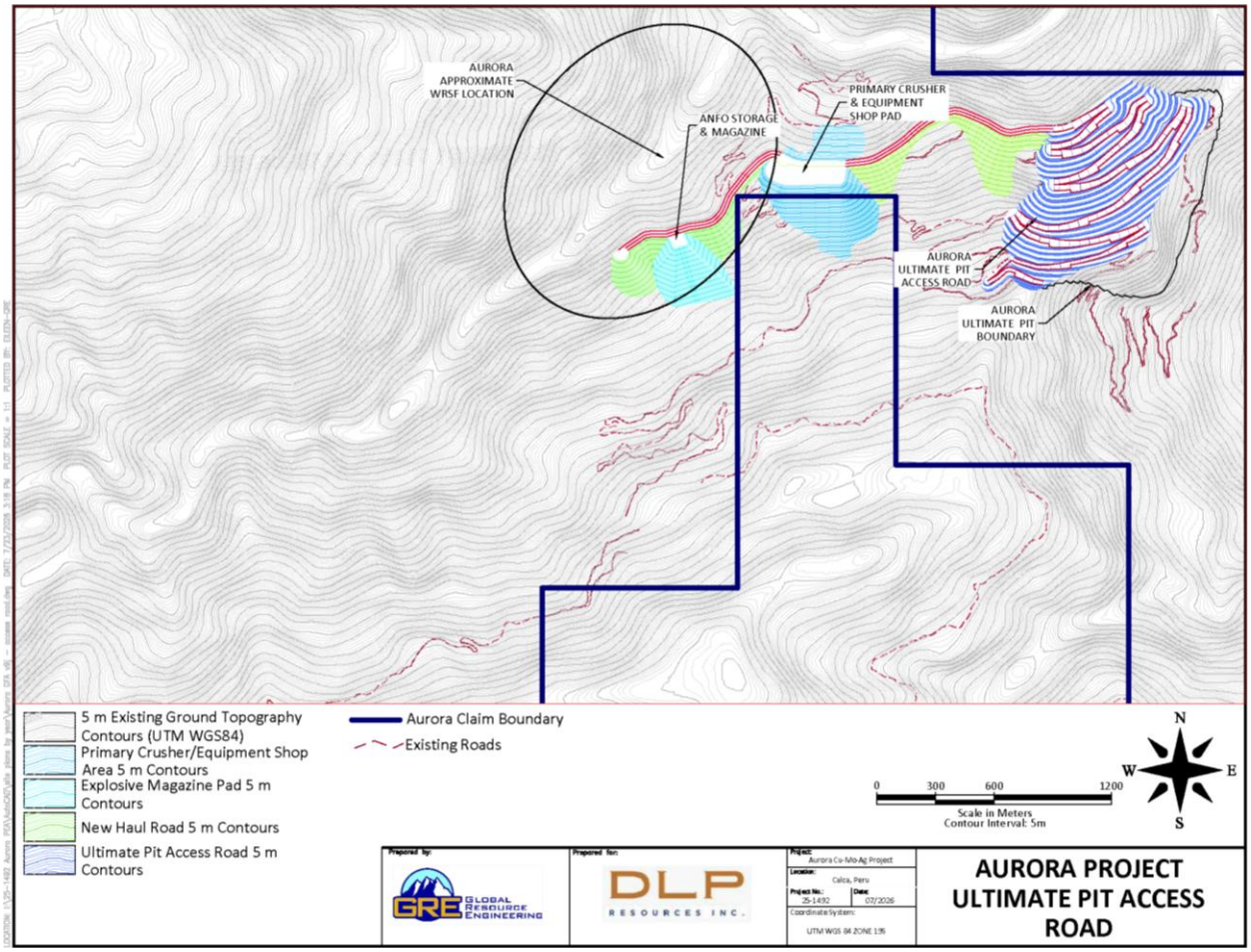


Figure 16-5: Aurora Project In-Pit Access Road



The access road would be constructed prior to the pit excavation and intersects some of the near-surface mineralization; therefore, the GRE QP designated the access road as a Phase 1 of the open pit.

16.1.4 Pit Resources

The resources for the ultimate pit and phase 1 were reported out of Surpac by bench and pit phase. A summary of the reported resources for each pit phase is shown in Table 16-2. A cutoff of $15 NSR was used.

Table 16-2: Aurora Project Pit Phase Resources

Pit Phase	Mineralized Material (million tonnes)	Waste (million tons)	Contained Cu ('000s lbs)	Contained Mo ('000s lbs)	Contained Ag ('000s oz)	Cu Grade (%)	Mo Grade (%)	Ag Grade (opt)	Stripping Ratio
1 (access road)	36.42	103.91	421,016	18,720	6,346	0.524	0.023	5.420	2.85
2 (ultimate pit)	201.23	120.59	2,793,317	114,477	41,346	0.631	0.026	6.403	0.60
Total	**237.65**	**224.5**	**3,214.33**	**133.2**	**47.69**	**0.615**	**0.025**	**6.252**	**0.95**

16.1.5 Mine Schedule

A preliminary mining schedule was generated from the pit resource estimate. Ms. Lane of GREused the following assumptions to generate the schedule:

- Mining Production Rate: variable, nominally 65,000 tpd of mineralized material
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

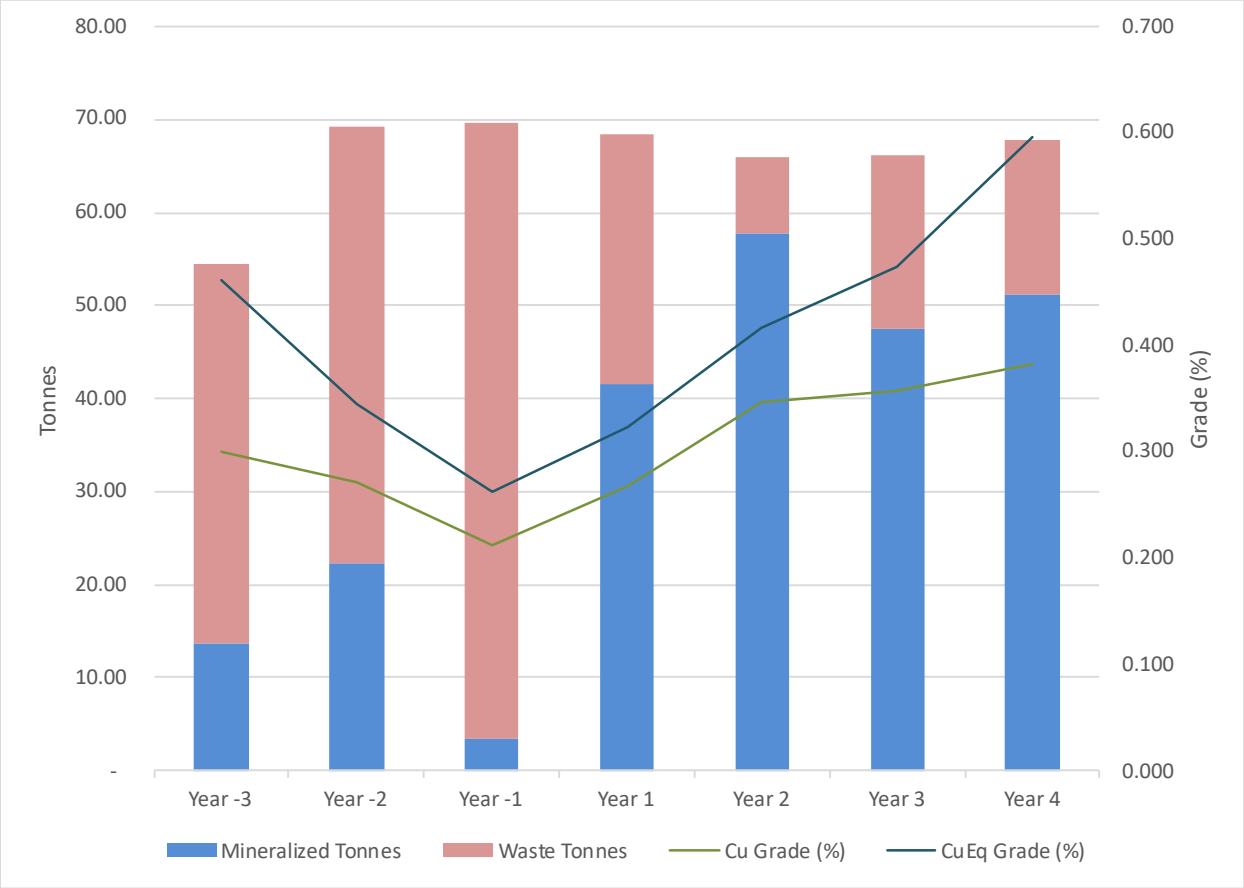
Pre-stripping of waste was included if waste occurred on a bench that had no corresponding processable material or if the tonnage of waste on a bench exceeded ten times the tonnage of processable material on that bench. The production rate for pre-strip benches was generally set to 3.2 times the mineralized material production rate, or 208,000 tpd, but varied considerably to smooth out the mining fleet where possible.

The mining schedule is summarized in Table 16-3 and illustrated in Figure 16-6.

Table 16-3: Aurora Project Mine Schedule Summary

Pit Phase	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Total
Mineralized Tonnes (millions)								
1 (access road)	13.72	22.28	0.42	-	-	-	-	**36.42**
2 (ultimate pit)	-	-	2.96	41.51	57.77	47.43	51.16	**200.84**
Total Mineralized Tonnes	**13.72**	**22.28**	**3.38**	**41.51**	**57.77**	**47.43**	**51.16**	**237.26**
Waste Tonnes (millions)								
1 (access road)	40.76	46.96	16.19	-	-	-	-	**103.91**
2 (ultimate pit)	-	-	50.13	26.89	8.22	18.78	16.57	**120.59**
Total Waste Tonnes	**40.76**	**46.96**	**66.32**	**26.89**	**8.22**	**18.78**	**16.57**	**224.50**
Cu Lbs Contained (millions)								
1 (access road)	90.83	133.40	1.98	-	-	-	-	**226.22**
2 (ultimate pit)	-	-	13.79	244.29	440.62	373.58	431.85	**1,504.13**
Total Cu Lbs Contained	**90.83**	**133.40**	**15.76**	**244.29**	**440.62**	**373.58**	**431.85**	**1,730.34**
Mo Lbs Contained (millions)								
1 (access road)	7.22	3.96	0.13	-	-	-	-	**11.31**
2 (ultimate pit)	-	-	0.19	4.26	8.08	15.31	36.90	**64.73**
Total Mo Lbs Contained	**7.22**	**3.96**	**0.32**	**4.26**	**8.08**	**15.31**	**36.90**	**76.04**
Ag Oz Contained (millions)								
1 (access road)	1.58	2.07	0.03	-	-	-	-	**3.69**
2 (ultimate pit)	-	-	0.23	3.84	6.33	5.72	7.03	**23.15**
Total Ag Oz Contained	**1.58**	**2.07**	**0.26**	**3.84**	**6.33**	**5.72**	**7.03**	**26.84**

Figure 16-6: Aurora Project Open Pit Mine Schedule Summary



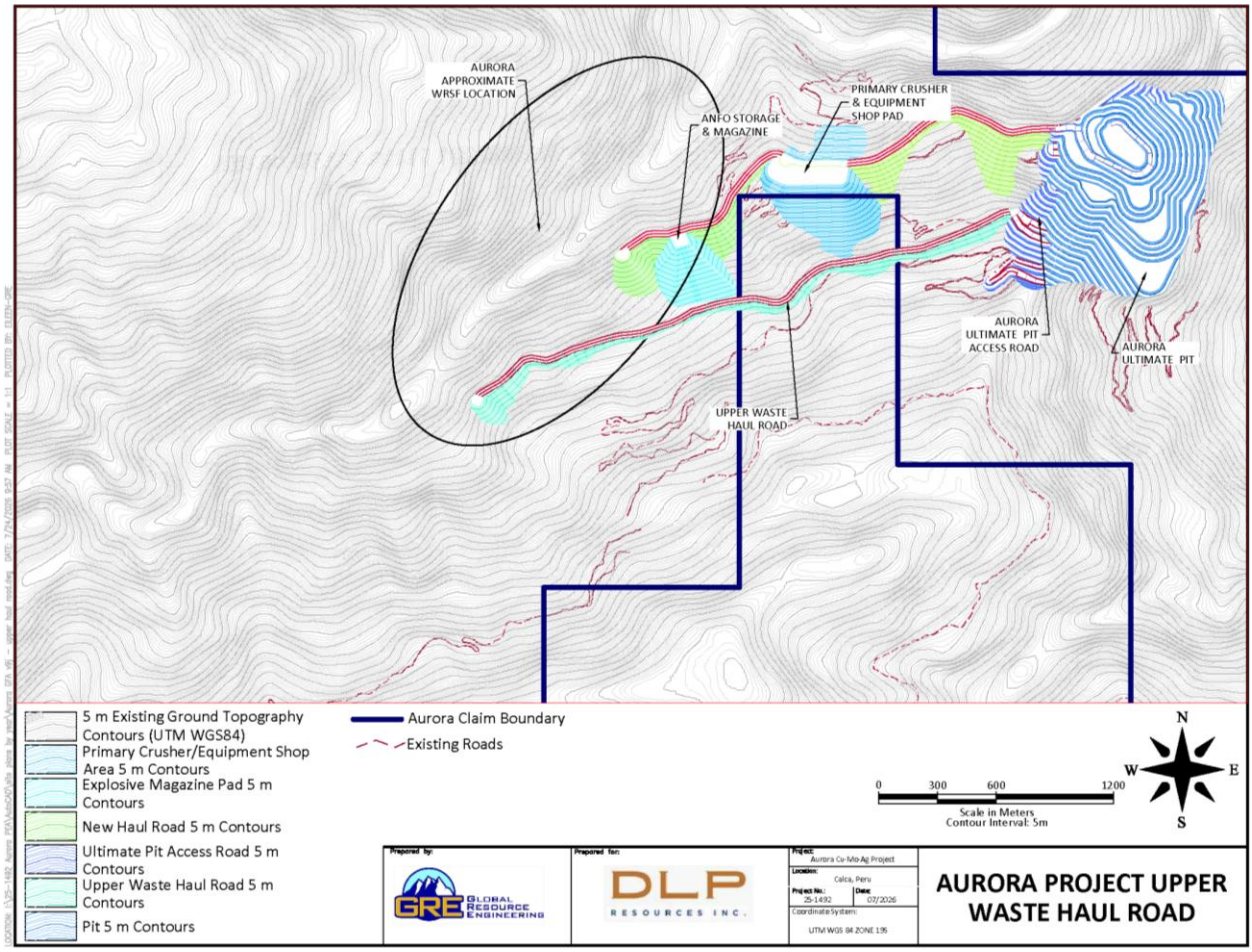
Source: GRE, 2026

16.1.6 Open Pit High Wall Access

Due to the terrain at the Aurora Project, the open pit mine extends 40 benches above the elevation 2505 pit entrance along the southwestern high wall (After the ultimate pit was designed, an access road to the upper benches of the pit was designed, as shown in Figure 16-5.

Figure 16-4 and Figure 16-5). Hauling material from the pit upper benches down to the entrance, across to the WRSF, then up the WRSF ramp would result in excessive cycle times; therefore, an upper haul road was designed to provide access to the upper pit benches. This upper haul road would be used only for waste and would lead to a relatively high elevation within the WRSF valley, where the haul trucks could end dump the waste rock into the valley. Figure 16-7 illustrates the location of the upper haul road and its access to both the pit and the WRSF.

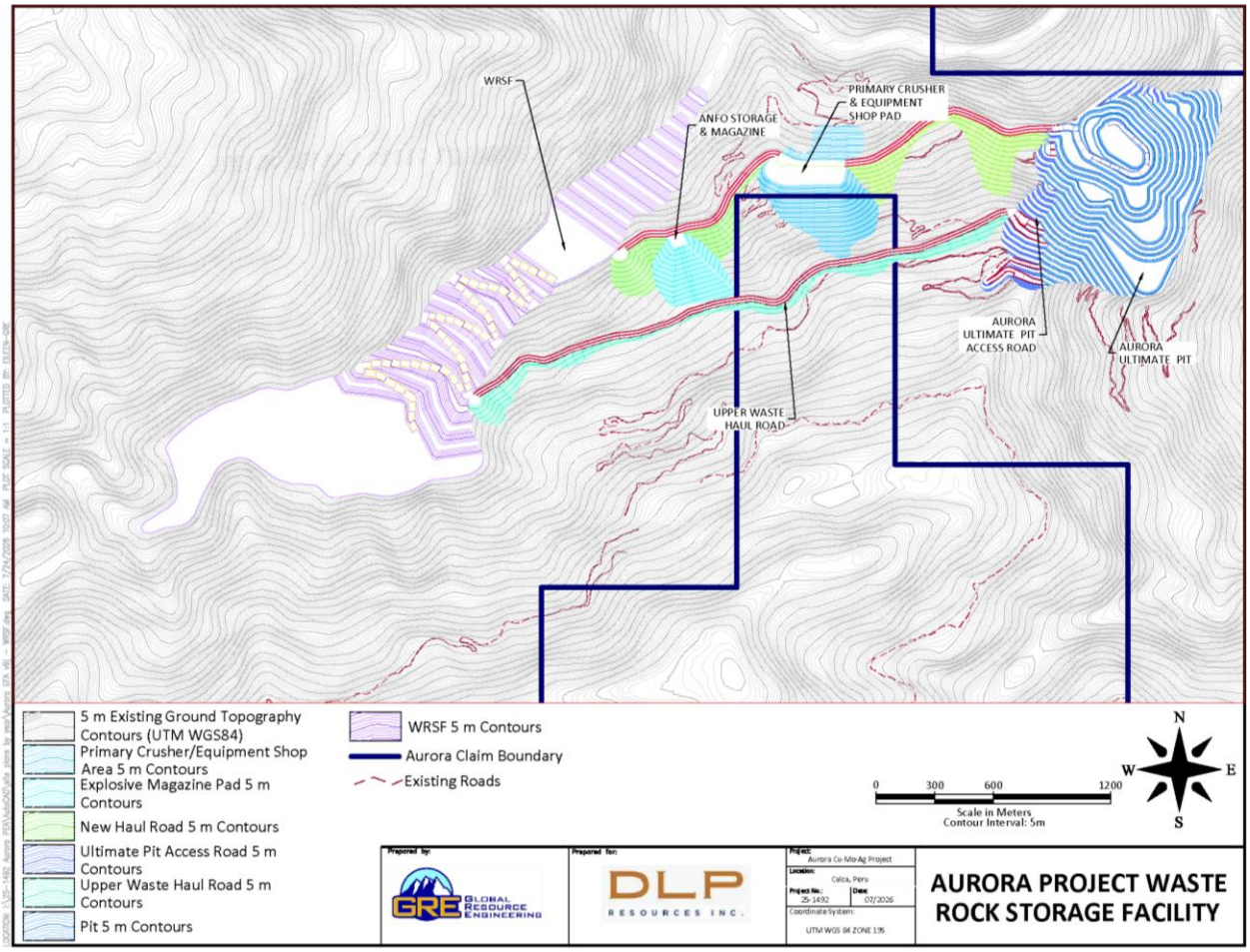
Figure 16-7: Aurora Project Upper Haul Road



16.1.7 Waste Rock Storage Facility

Waste rock from the open pit mining would be placed in a WRSF in the lower Mestizamayo valley. The WRSF would have 2H:1V sidewall slopes with 30-meter-wide benches every 30 vertical meters, resulting in an overall slope of 3H:1V. A 30-meter-wide haul road would be built up the face of the WRSF at a maximum 10% slope. The configuration of the final WRSF, accommodating 224.1 M tonnes or approximately 123.5 M m^3, is illustrated in Figure 16-8.

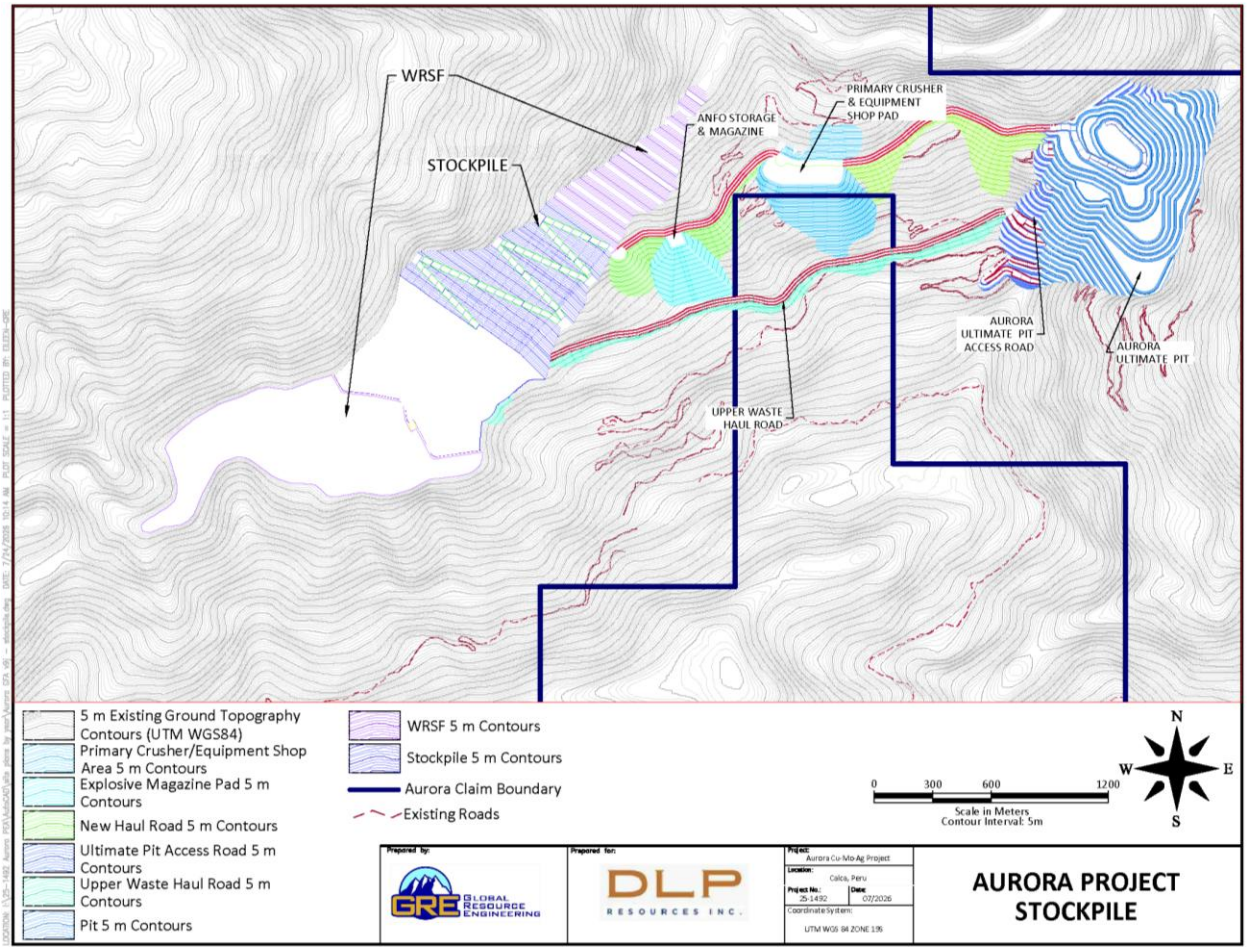
Figure 16-8: Aurora Project Waste Rock Storage Facility Final Layout



16.1.8 Stockpile

A stockpile for mineralized material that does not go direct to the primary crusher has been designed within the lower Mestizamayo valley alongside the WRSF. The stockpile would have 2H:1V sidewall slopes with a 30-meter-wide haul road at a maximum grade of 10%. The configuration of the stockpile at its peak capacity of 147.3 M tonnes or approximately 65.1 Mm^3, is illustrated in Figure 16-9.

Figure 16-9: Aurora Project Stockpile Peak Capacity Layout



16.1.9 Site Plans by Year

Aurora Project site layout plans for each year of mining operations (Years -3 through 4) are illustrated in Figure 16-10 through Figure 16-16.

Figure 16-10: Aurora Project Site Layout Year -3

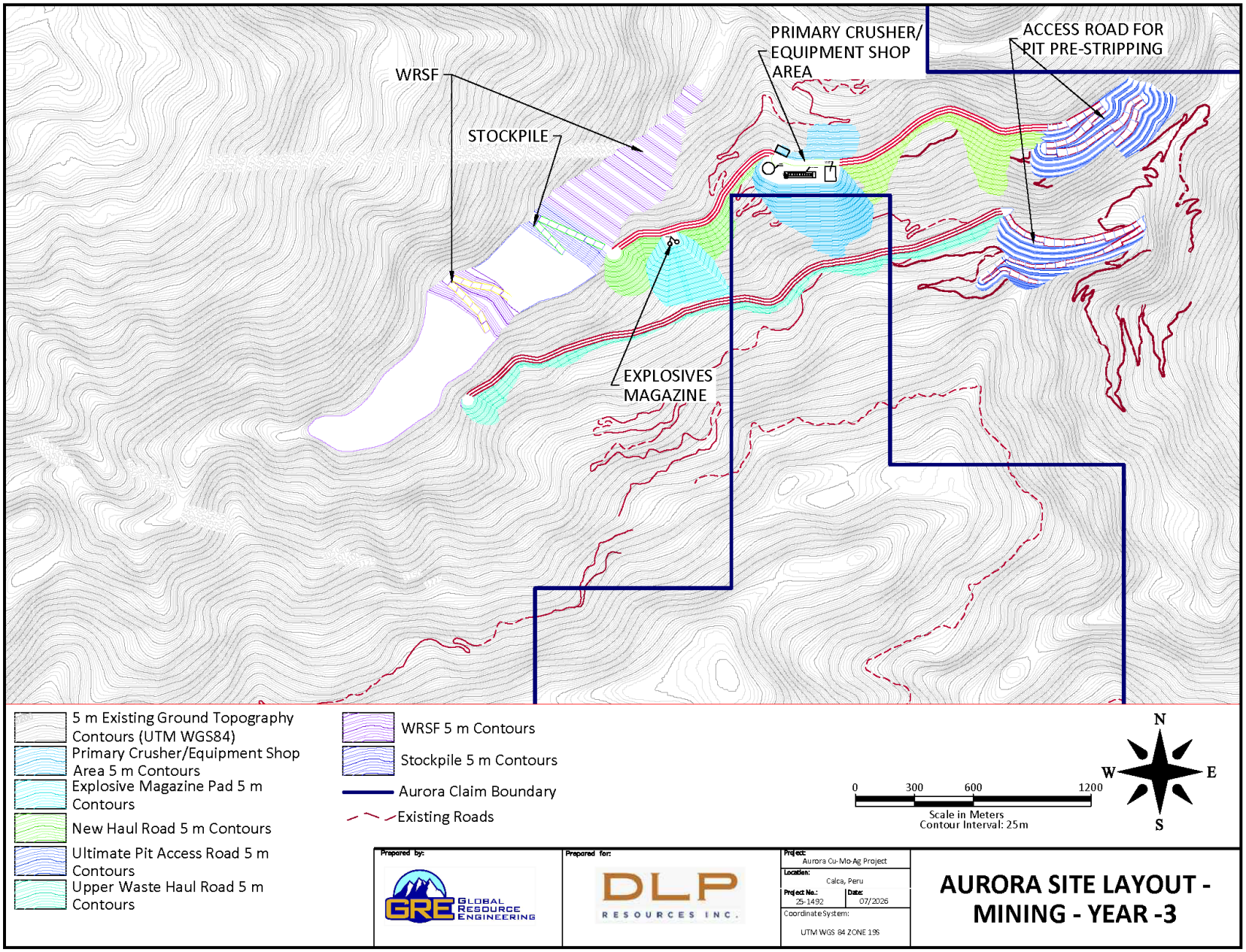


Figure 16-11: Aurora Project Site Layout Year -2

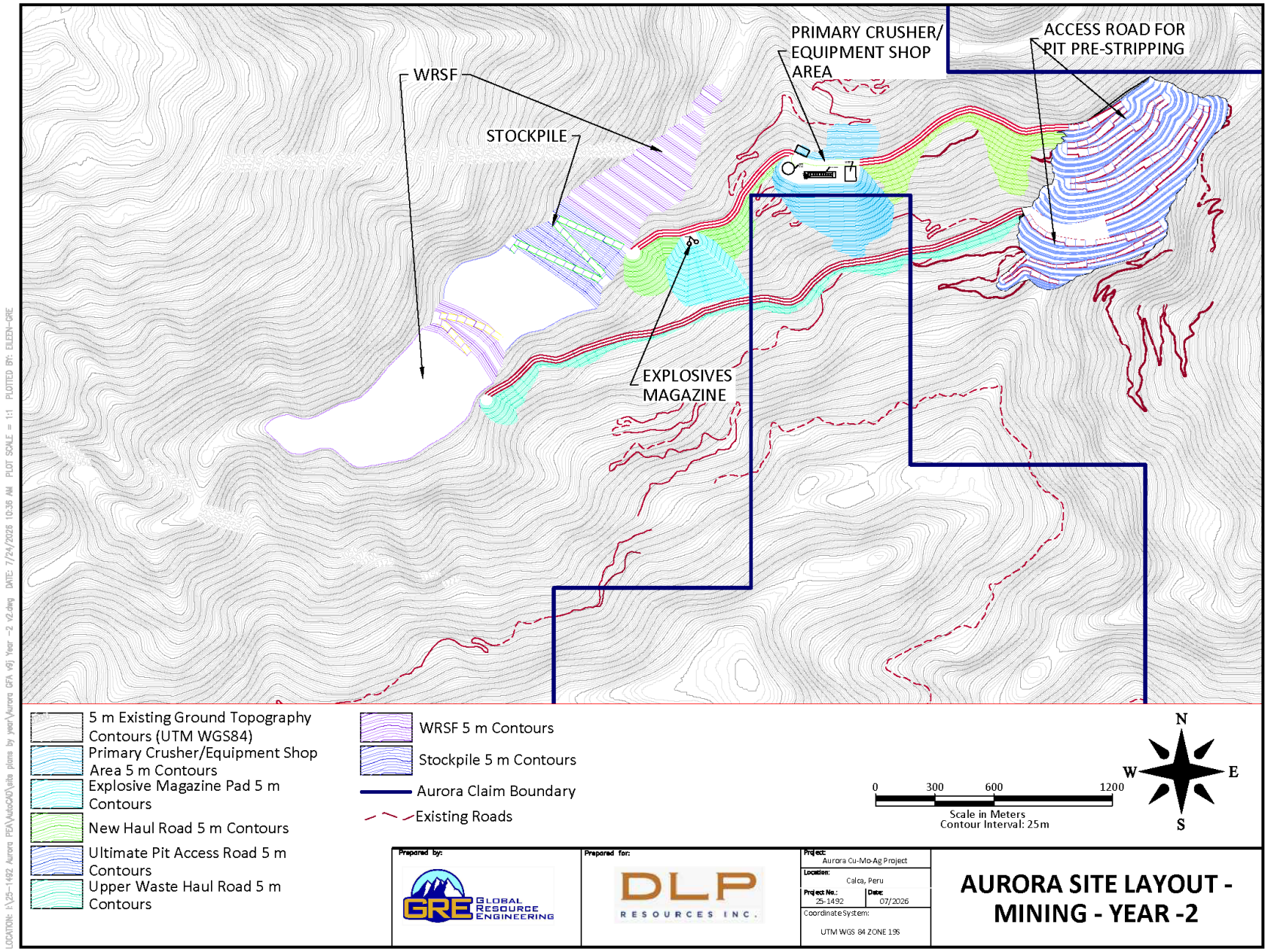


Figure 16-12: Aurora Project Site Layout Year -1

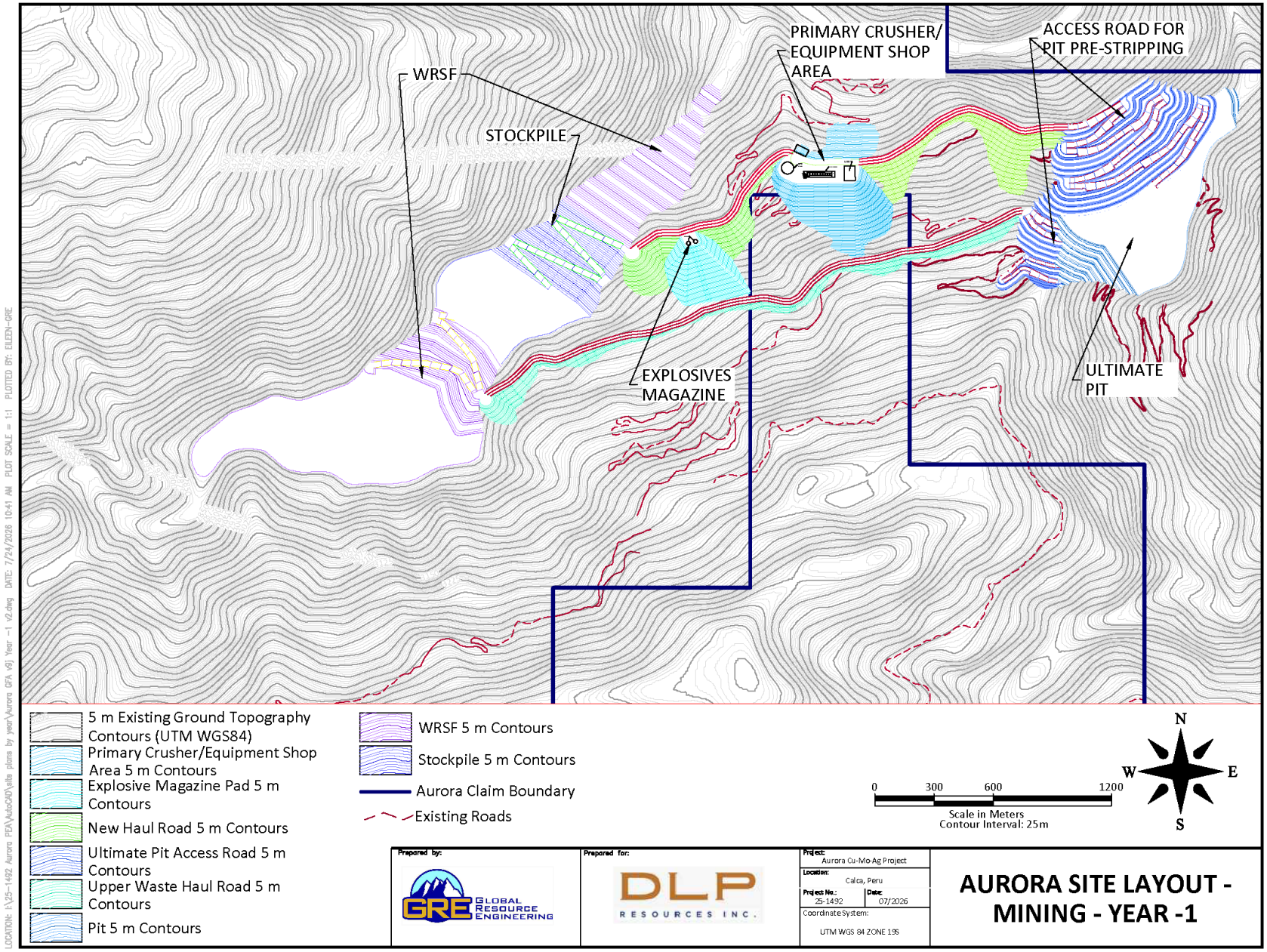


Figure 16-13: Aurora Project Site Layout Year 1

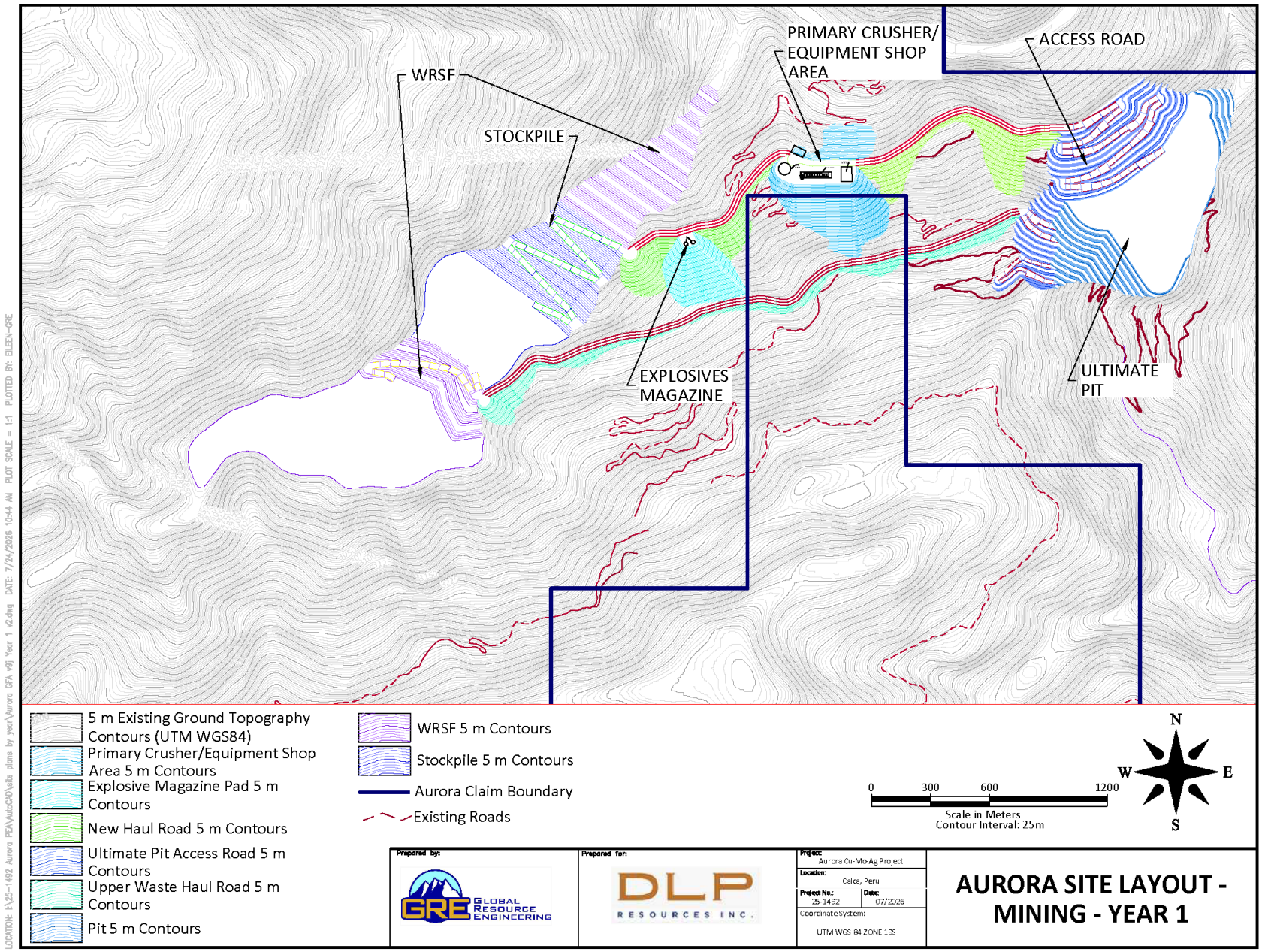


Figure 16-14: Aurora Project Site Layout Year 2

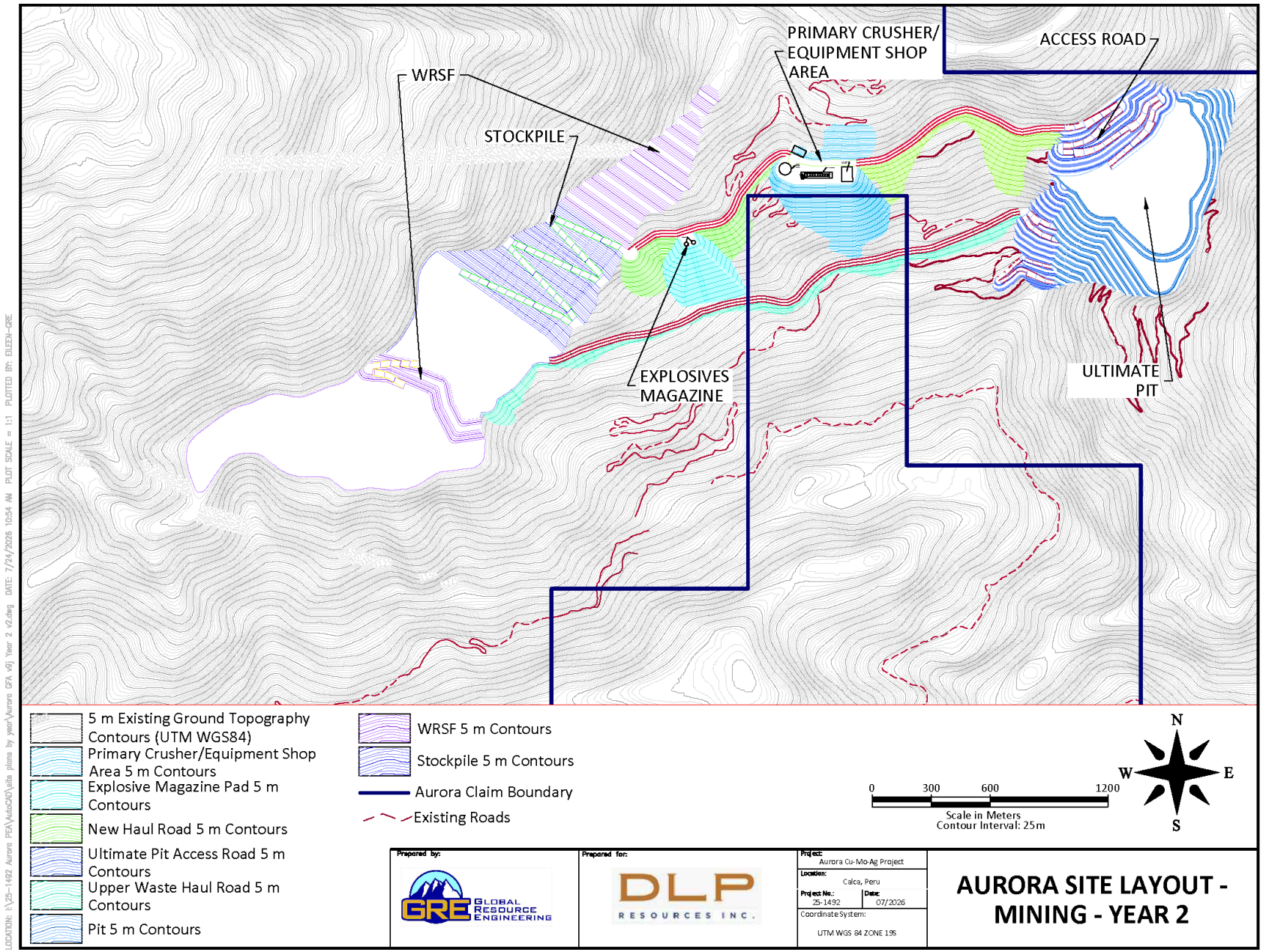


Figure 16-15: Aurora Project Site Layout Year 3

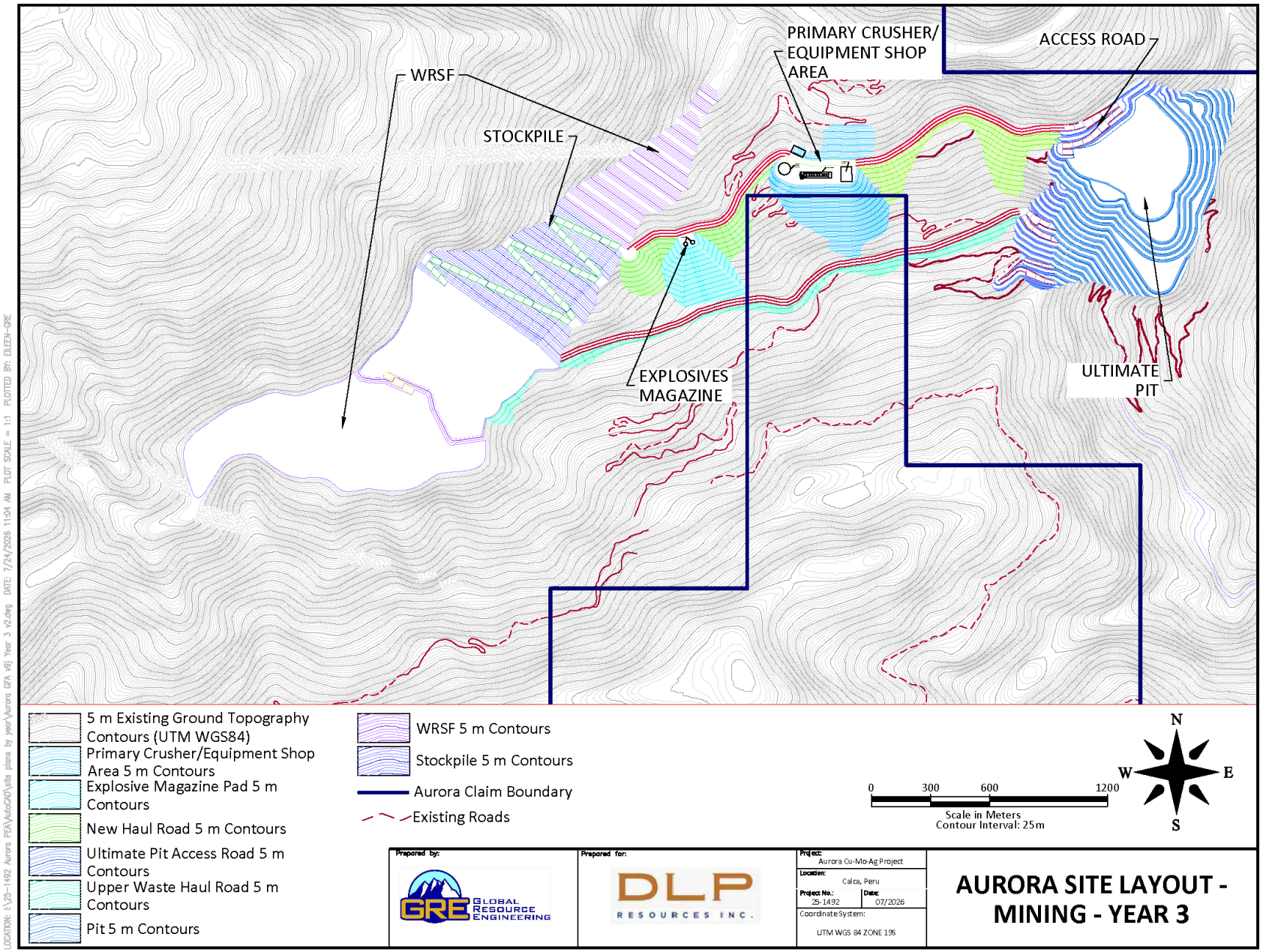
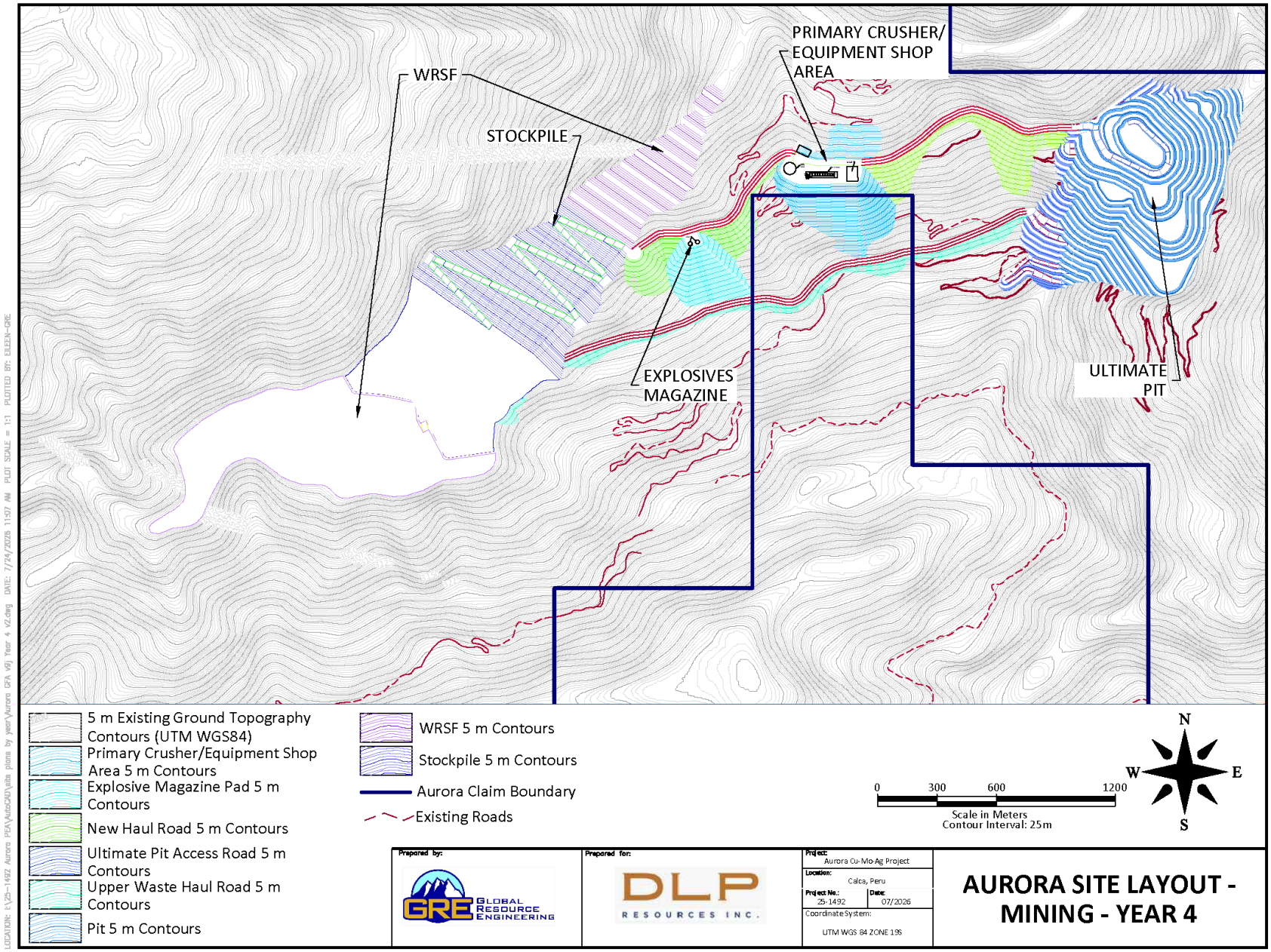


Figure 16-16: Aurora Project Site Layout Year 4



16.2 Underground

16.2.1 Geotechnical Basis of Design

16.2.1.1 Rock Mass Characterization

Data Availability

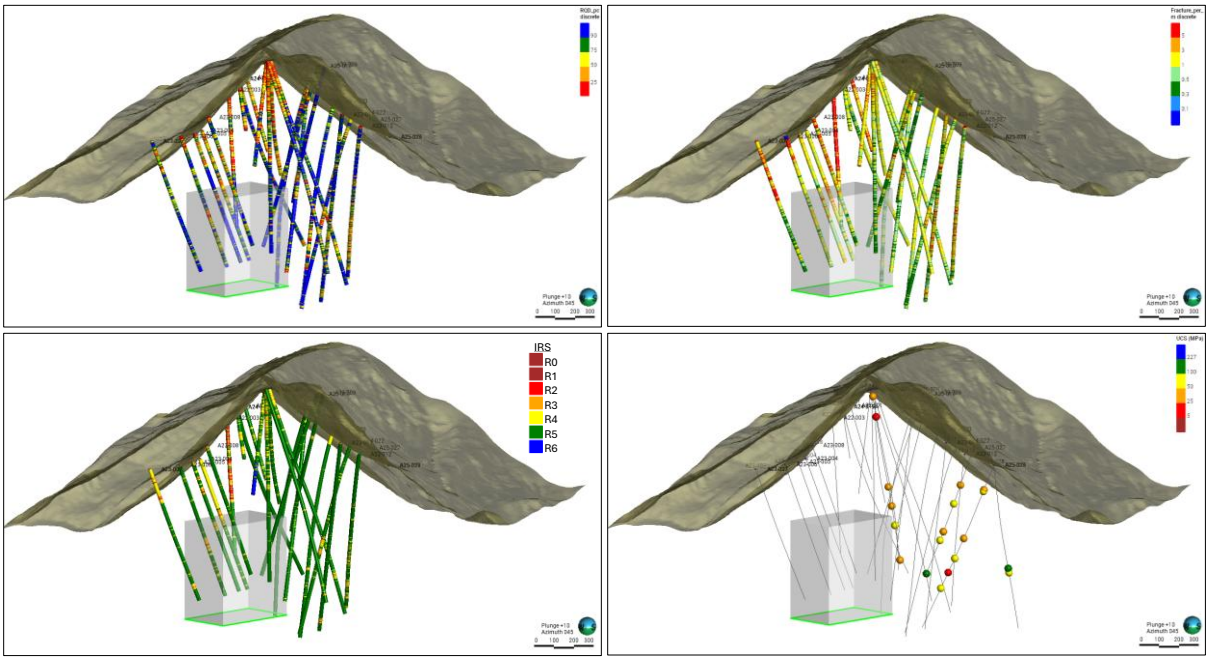
The geotechnical review was primarily based on a database comprising 23 drillholes with partial geotechnical data. As shown in Figure 16-17, the available geotechnical information only includes basic core logging parameters such as Total Core Recovery (TCR), Rock Quality Designation (RQD), Intact Rock Strength (IRS), and Fracture Frequency per meter (FF/m).

The 2025 drilling program included five drillholes with partial oriented core data (A25-023, A25-024, A25-025, A25-026, and A25-027) and a limited number of laboratory tests. These data were used to supplement the rock mass characterization and provide preliminary information on structural orientations and intact rock properties (Table 16-4).

The geotechnical database was reviewed in conjunction with drill core observations and associated core photographs to support the rock mass characterization process.

In addition, 3D geological models representing lithological units and mineralized domains were provided for the study. However, no 3D structural model was available at the time of the review, limiting the assessment of structural controls and their potential influence on rock mass behavior and caveability.

Figure 16-17: Aurora Project Oblique 3D View (Looking Northeast) of the Drillholes with Geotechnical Parameters (RQD, FF/m, IRS, and UCS)



Source: SRK, 2026
Grey box depicts the cave zone

Table 16-4: Aurora Project Underground Geotechnical Data Availability

Item	Data Available
Drillhole Database	23 drillholes with RQD, TCR, FF/m, IRS
Structural Data	Five drillholes partially oriented
Laboratory Tests	19 Uniaxial Compressive Strength tests (UCS)
3D Models	Five 3D Models: Cu_Domain, Den_Domain, Fe_Domain, Min_Zone, Mo_Domain, S_Domain

Geology

The review of the 3D models provided by DLP shows three main lithologies which include Igneous, Breccia, and Hornfels (Sandia Formation) units. During the site visit, the DLP team shared information and additional details about the geology, including the existence of two porphyries, early porphyry and intermineral porphyry, both of which are part of what has been modeled as Igneous.

During the site visit, it was possible to identify the early Porphyry and Sandia Formation lithologies through outcrop mapping (Photo 16-1).

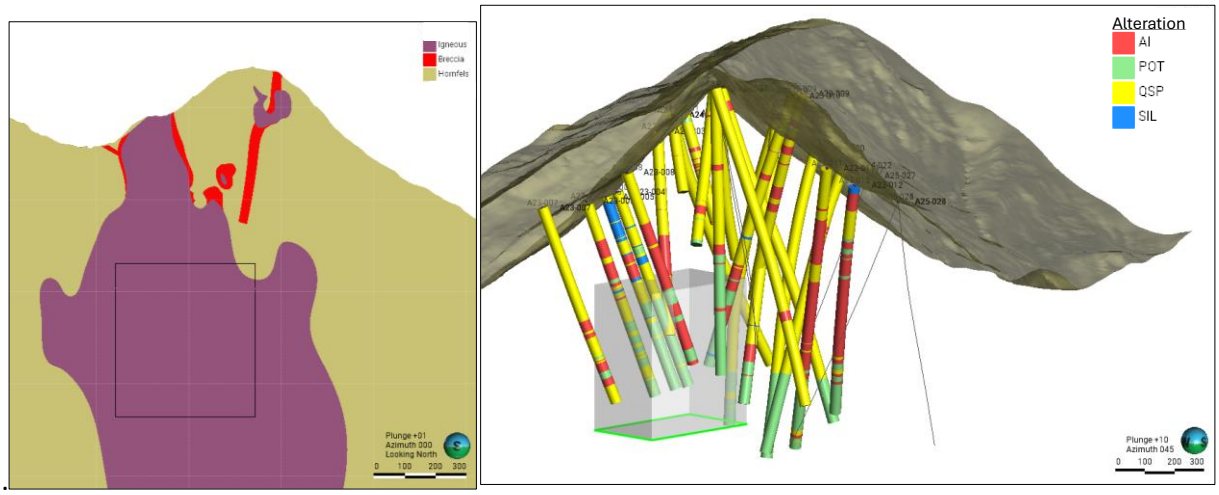
Photo 16-1: Outcrops of Early Porphyry (Left) and Hornfels – Sandia Formation (Right)

Source: SRK, 2026

As presented in Figure 16-18, most of the caving area will be within the Igneous unit, while the other units will be affected by the subsidence closer to surface.

No alteration model was provided to SRK. However, the logging database indicates distinct alteration zones within the deposit area. The principal alteration types include Potassic (POT), Quartz-Sericite-Pyrite (QSP), and Argillic (AI), while Silicification (SIL) appears more localized in shallow portions of the deposit. POT and QSP alterations are the predominant alterations within the proposed caving zone.

Figure 16-18: Aurora Project Cross-Section Showing the Main Lithologies of the 3D Model "Den_Domain Model"



Source: SRK, 2026
Black and grey boxes depict the cave zone

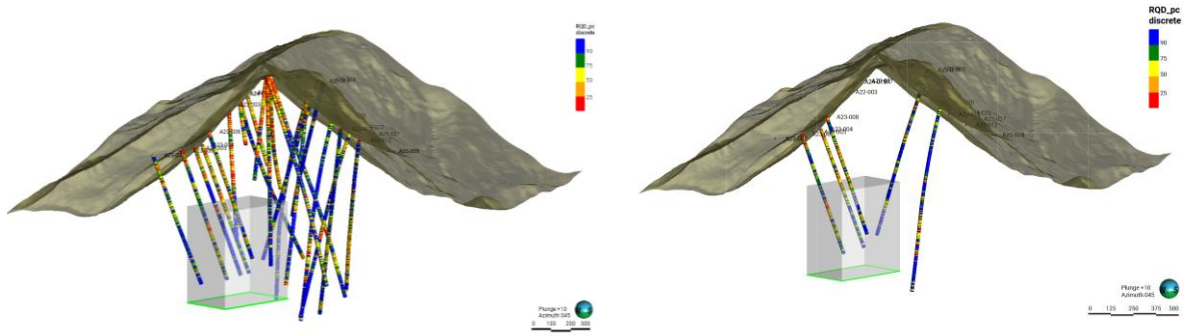
Rock Mass Characterization Through Drillcore Review

As presented in Figure 16-19, only six drillholes intersect the cave zone (gray box), and a similar number of drillholes reach the elevation of 1,845 masl (green outline), corresponding to the proposed production level discussed later in this report. Nevertheless, to support a high-level rock mass characterization review and as part of the site visit, five drillholes (A23-004, A23-006, A23-008, A23-010 and A24-020) were selected for a quick logging and comparison with the basic geotechnical parameters logged by DLP. The review was focused on intervals within the area of interest (the proposed cave zone), as well as the deepest section of Drillhole A24-020, which reaches the footprint elevation.

As all core samples had been split in half, fracture-related parameters could not be reviewed. Therefore, the observations were focused on parameters such as IRS, joint condition, infill material, and general conditions of weak zones, defect intensity, and defect strength. This information was used to develop preliminary rock mass classifications within the caving area using the In-Situ Rock Mass Rating (IRMR) system (Laubscher, et al., 2001).

Some of the main observations regarding the database and new data collected during the site visit are summarized below. All these findings were considered for the rock mass characterization and classification.

Figure 16-19: Aurora Project Drillholes within the Area of Interest



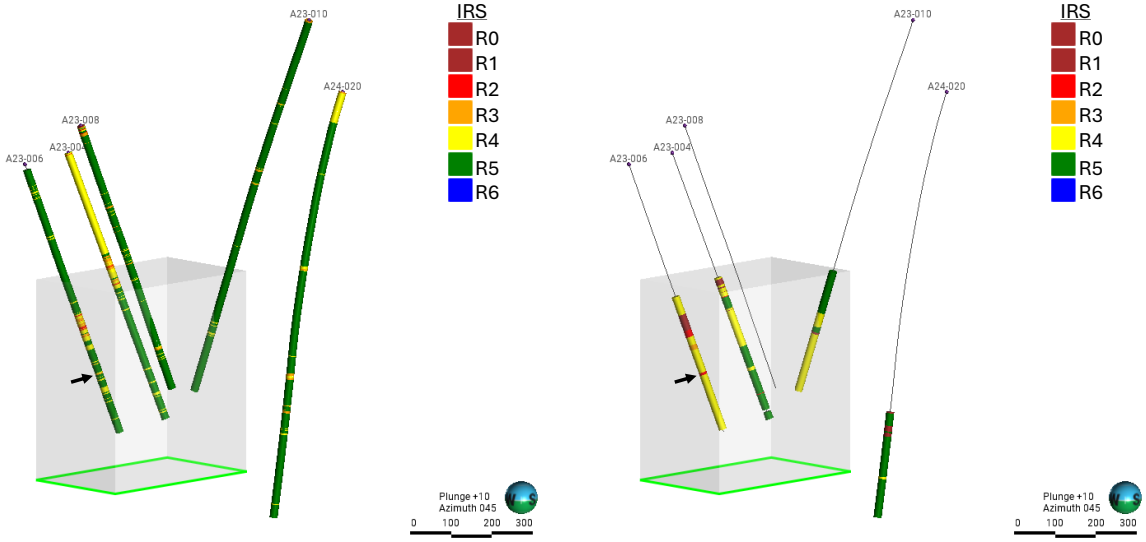
Source: SRK, 2026
Left: drillholes with basic geotechnical logging
Right: drillholes selected for quick log

Intact Rock Strength

Observations from the core review indicated that the Early Porphyry, the dominant lithology within the proposed caving area, is generally affected by both QSP and potassic alteration. Site observations indicate that this unit with QSP alteration presents an Intact Rock Strength (ISRM, 1978) ranging from R3 (Medium-Strong rock) to R4 (Strong rock), whereas potassic alteration is generally stronger, ranging from R4 (Strong Rock) to R5 (Very Strong Rock).

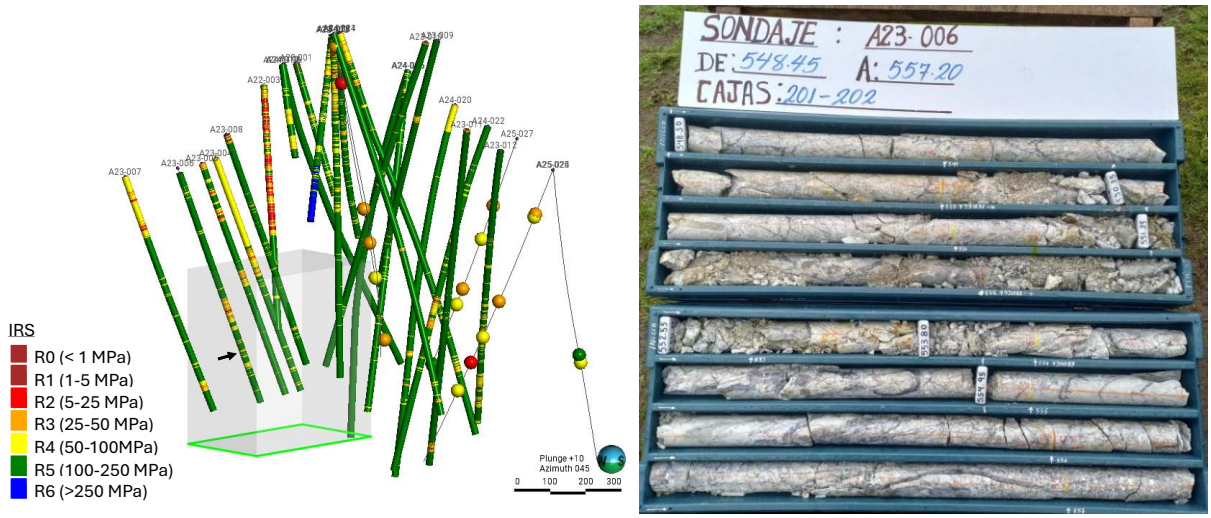
When compared with these observations, the IRS values recorded in the existing database appear to be overestimated. Figure 16-20 compares the IRS values recorded in the DLP database with those assigned during SRK's re-logging. This interpretation is further supported by the available laboratory test results, which comprise 19 UCS tests (Figure 16-21).

Figure 16-20: Aurora Project Selected Drillholes Comparing the Intact Rock Strength Values Estimated on Site



Source: SRK, 2026
Left: values logged by DLP
Right: Quick log by SRK

Figure 16-21: Aurora Project Comparison of Logged Intact Rock Strength



Source: SRK, 2026

Left: Comparison of the logged intact rock strength from DLP and the UCS test results (spheres). The UCS results are color coded according to the same strength ranges used for the IRS classification.

Right: core photo of an interval of drillhole A23-006 (pointed with an arrow) with overestimation of the IRS.

Rock Defects

Defect intensity and strength were assessed through core observations and a series of drop tests conducted at the core facility. The results indicate that the rock defects are generally of moderate to strong strength, even where defects intensity is high. Figure 16-22 presents examples of the drop tests.

Figure 16-22: Aurora Project Examples of Drop Test Results

Source: SRK, 2026

Photos show both pre- and post-test images of the samples. Note that most samples fractured into only a few core pieces, indicating that the defects are generally strong.

Joint Condition

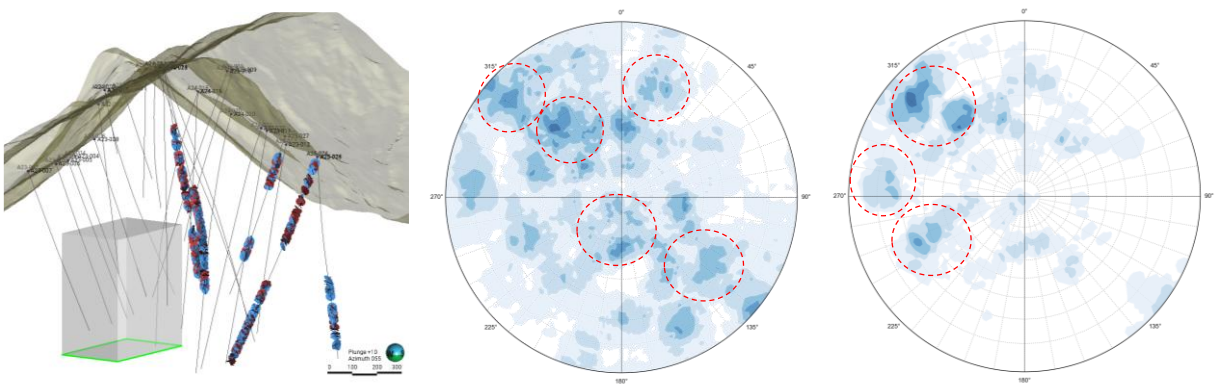
The condition of the discontinuities in the cave area varies and is related to the alteration type and mineralization zones. In general, the discontinuities vary from rough planar to rough undulated, and the infill material consist mainly in sericite-clays, pyrite, and molybdenite.

Orientation of Discontinuities

There are only five drillholes from the 2025 drilling program that include oriented core. As shown in Figure 16-23, these drillholes are located south of the proposed caving area and contain only partial orientation data, with discontinuity orientation measurements available for only a few intervals along the drillholes.

A high-level review of the available data was conducted to identify the principal structural trends within the project area. For this purpose, all available data was compiled and analyzed to determine the number of structural sets and their orientations. The stereonet presented in Figure 16-23 indicates a dominant NE-SW trending, subvertical set, while a less developed sub-horizontal set is also present. When the data was filtered by structure type and features classified as veins were excluded, the number of identified sets was reduced; however, the dominant set continued to exhibit a NE-SW trend with a steep dip to the southeast and a less developed set oriented NNE-SSW became apparent.

Figure 16-23: Aurora Project Orientation of Discontinuities

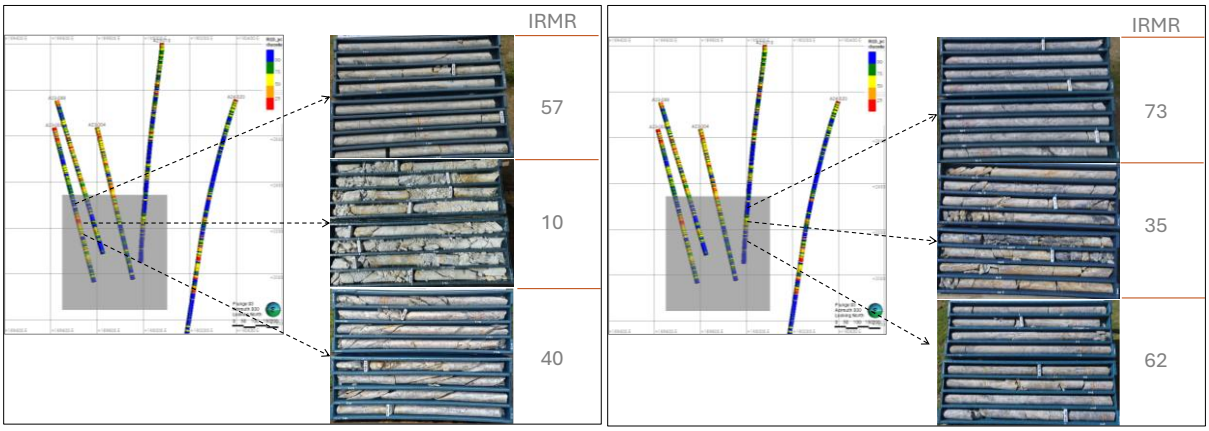


Source: SRK, 2026
Left: five drillholes with partial oriented core data showing the logged discontinuities as red and blue disks.
Middle: stereonet of all structural data combined
Right: stereonet excluding veins, both highlighting the main pole concentrations.

Rock Mass Classification

Based on the existing geotechnical logging database, which consists primarily of fracturing parameters (RQD and FF/m), supplemented by a review of core photographs and onsite observations from the quick logging, a conceptual rock mass characterization was undertaken. Representative intervals (examples in Figure 16-24) within the selected drillholes located in the cave area were used to broadly classify the rock mass into three main groups with distinct IRMR values. The main zones are grouped in Table 16-5.

Figure 16-24: Aurora Project Rock Mass Characterization Examples



Source: SRK, 2026

Examples of representative intervals from drillhole A23-006 (Left) and A23-010 (Right) showing the IRMR used for rock mass classification.

Table 16-5: Aurora Project In-situ Rock Mass Ratings

Main Zone	IRMR
Poor Rock Mass (Weak Zones)	<20
Fair Rock Mass	30 – 40
Strong Rock Mass	60 - 75

The IRMR values were converted to Mining Rock Mass Rating (MRMR) (Laubscher, et al., 2001), considering stress and joint orientation adjustment factors as the most relevant for this project. Assuming a stress adjustment factor of 1.0 and the presence of three major joint sets and an adjustment factor of 0.8. The resulting MRMR values are presented in Table 16-6.

Table 16-6: Aurora Project Mining Rock Mass Ratings

Main Zone	MRMR
Poor Rock Mass (Weak Zones)	<16
Fair Rock Mass	24 – 32
Strong Rock Mass	48 – 60

For this early stage of the project, geotechnical domains have not been defined, however, a basic numeric RQD model has been used to spatially identify the poorest rock mass quality zones. This zones in combination with the three different ground types and MRMR classes have been used for empirical caveability and subsidence assessment.

16.2.1.2 In-Situ Stress

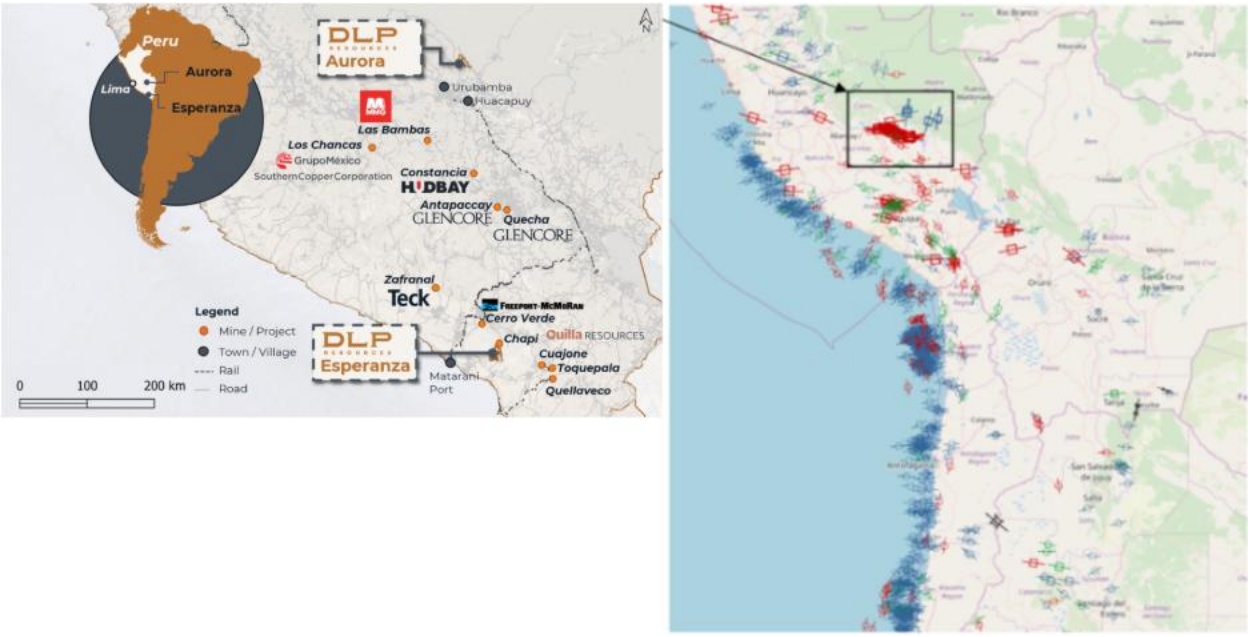
DLP's Aurora porphyry project is located near Parobamba, approximately 60 km north of Cusco (Figure 16-25), at an elevation of roughly 2,000 m to 3,000 m. The regional World Stress Map data is dominated by a pervasive thrust faulting regime at depth. In the deeper lithosphere beneath the Andes, horizontal compressive forces (σ_H) far exceed the vertical overburden, driven by the active subduction of the Nazca Plate beneath the South American Plate.

The presence of normal faulting near the Cusco region where the Aurora project is located points to a decoupling between the deep regional tectonics and the shallow crustal stress (Figure 16-26):

- In high-relief mountainous terrains, the local stress field is often partially or completely isolated from regional tectonic compression above a certain elevation. The mountain mass physically shields the upper crust from deep lateral clamping forces.
- At the elevations relevant to the upper footprint of the Aurora deposit, vertical gravity loading dominates. The topography relaxes horizontally into adjacent valleys, which manifests geologically as normal faulting.

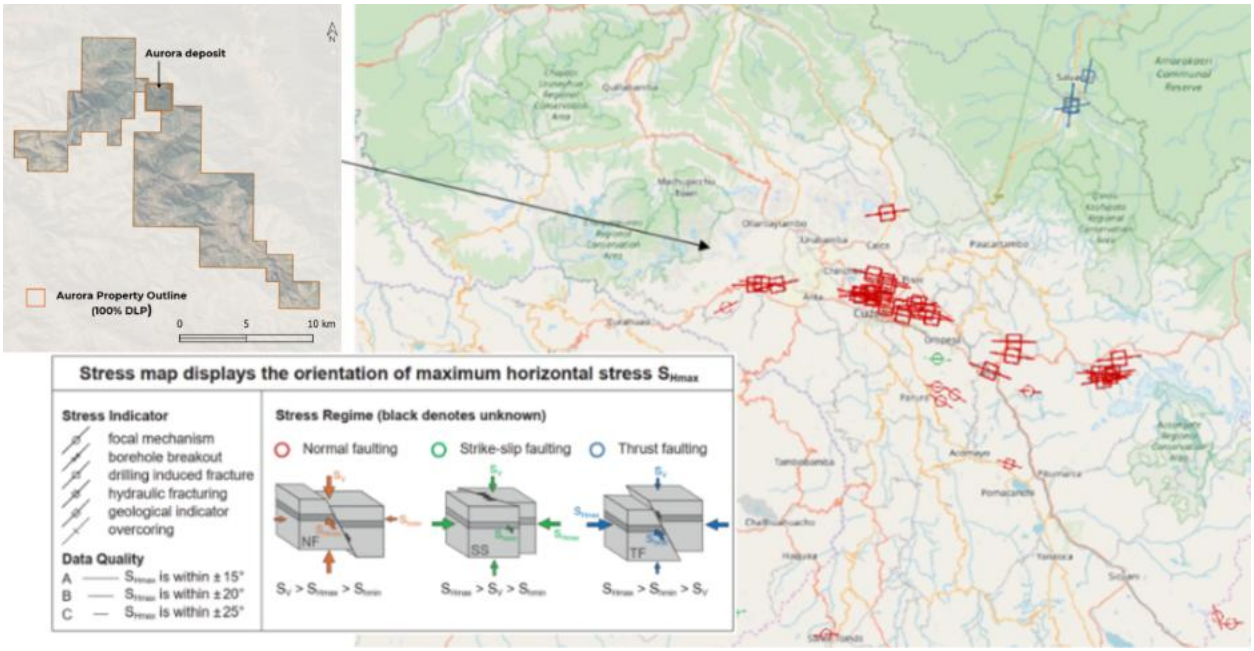
Therefore, it is assumed that the upper domain of the deposit is expected to be dominated by vertical stress (σ_v) with a low amount of horizontal confinement. As extraction levels push below the immediate topographic influence, the stress tensor will likely rotate and transition back into the regional compressive regime. This can create a heterogeneous stress profile with depth.

Figure 16-25: Regional World Stress Map along with Aurora Property and Other Mines



Source: World Stress Map, 2025

Figure 16-26: Local World Stress Map



Source: World Stress Map, 2025

16.2.1.3 Structural Model

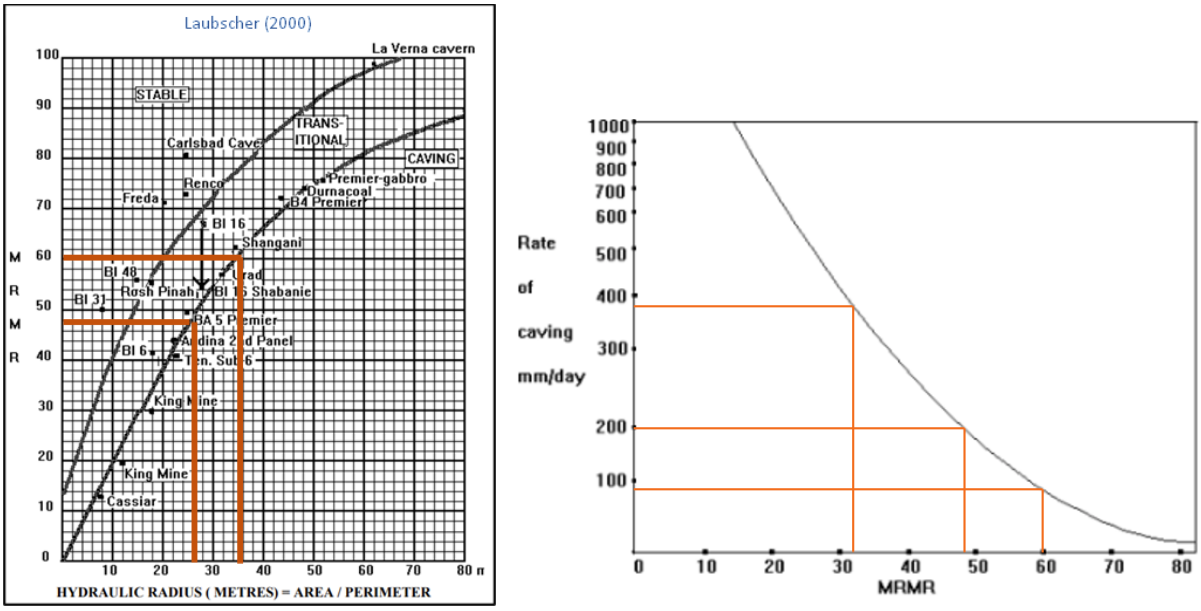
SRK understands that no structural model has been developed to date and that the available structural database consists only of five drillholes (A25-023, A25-024 A25-025, A25-026, and A25-027) from the 2025 drilling program, which include partially oriented core data. These five drillholes are located south of the cave area.

16.2.1.4 Caveability Assessment

At the current stage of the project, where geotechnical data is limited, an empirical approach to assess caveability is justified. The prediction of the minimum hydraulic radius (HR) required to initiate caving was carried out using Laubscher's caveability chart. This method, developed by Diering & Laubscher (1987) and further refined by Laubscher (1990; 1994; 2000; 2001) consists of two curves that correlate rock mass quality and the HR of the undercut footprint.

MRMR values estimated based on the available data and observation made during the site visit were used to indicate a minimum HR of 36 (Figure 16-27). This HR corresponds to an area of approximately 150 m x 150 m, which is significantly smaller than the footprint used for the mineable inventory and production schedule.

Figure 16-27: Aurora Project Caveability Assessement



Source: Laubscher, 2000

Left: Empirical charts to estimate the minimum hydraulic radius required to initiate/propagate caving in the strong rock mass

Right: estimated caving rates (Right)

When the caving HR is reached, the production ramp-up must occur gradually to ensure cave propagation without the formation of excess air gaps. Draw rates of the ore must be less than caving rates, and it is therefore important to also establish the caving rate for the rock mass. Laubscher empirical guidelines for draw rates that ensure continuous caving without forming excess air gaps are illustrated in (Figure 16-27). When the cave-back reaches the strong rock mass, caving rates are expected to be between 90 and 200 millimeters per day (mm/day). However, in the more fractured and weaker zones, a caving rate greater than 380 mm/day has been estimated.

16.2.1.5 Fragmentation Assessment

The preliminary fragmentation assessment was focused primarily on the massive hard rock mass domain, which is the predominant geotechnical domain within the planned cave zone. Relying on this competent domain ensures that the fragmentation estimates remain appropriately conservative for mine design and material handling considerations.

Block Caving Fragmentation (BCF) tool (version 4) was used to estimate the proportion of hang-ups, oversize, and fine material reporting to the drawpoints as a function of the height-of-draw (HOD). Referring to Figure 16-28, the results suggest a steady reduction in oversize material as the column matures and secondary fragmentation increases. The projected oversize percentages and frequencies (oversize per kilotonne [kt]) are summarized as follows:

- For 100 m HOD: 18.0% oversize (8.7 oversize / kt)
- For 300 m HOD: 7.3% oversize (3.5 oversize / kt)
- For 500 m HOD: 3.5% oversize (2.9 oversize / kt)

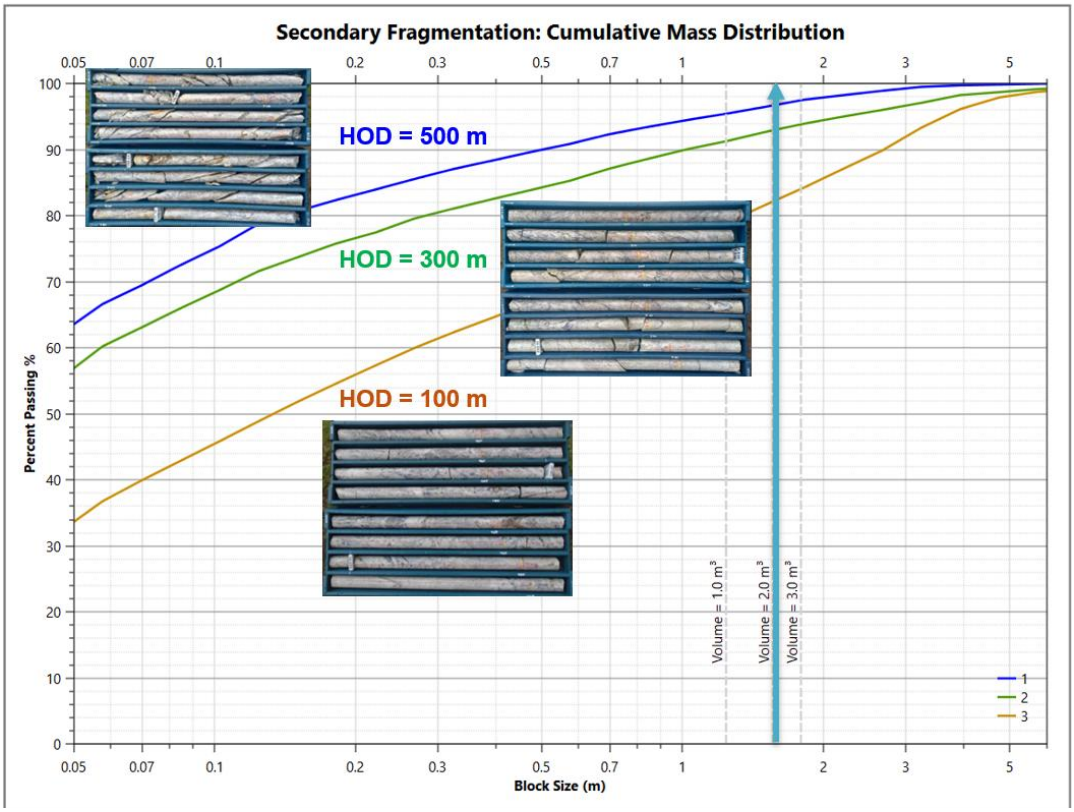
While the BCF analysis predicted zero hang-ups for this domain, based on empirical data and SRK's experience with block cave mines operating in similar hard rock masses and HODs, a frequency of 1 to 2 hang-ups per 1,000 tonnes is typical, even within mature cave footprints. Secondary breakage equipment and logistics should be planned to accommodate this empirical expectation.

Additionally, from the recent SRK site visit to core shacks, approximately 6% weak zones within the rock mass were observed. Based on SRK's experience in similar caving environments, these weak zones have the potential to contribute an additional 5% to 10% of fines, depending on their spatial distribution relative to the drawpoints. The geological controls and spatial distribution of these weak zones will be investigated in greater detail during the next level of study.

Although the current version of the software (BCF-V4) has been improved through the benchmarking of drawpoint fragmentation observations across various operations, inherent uncertainties remain. The exact impacts of the following factors on fragmentation are difficult to measure and subsequently back-analyze using the conventional BCF approach:

- Faults, shearing, and broken zones
- Draw rates and draw panel widths
- Material ingress
- Air gap formation

Figure 16-28: BCF Predicted Fragmentation Curves for Different HODs



Source: SRK 2026

16.2.1.6 Ground Support Strategy

Three support classes are defined based on rock mass quality or Ground Type as listed below.

Class 1 Ground

- For very poor to poor ground; Q less than 1, RMR89/GSI less than 30, Rock Strength R2-R3
- 2.4 m fully-grouted #7 resin rebar at 1.2 m x 1.2 m spacing on the back; double-strand 0.7" bulbed cable bolts at 6.0 m length and 2.0 m x 2.0 m spacing as required for long support; equivalent bolt pattern on walls; welded wire mesh with 50 mm shotcrete applied to back and walls to within 1.0 m of floor.

Class 2 Ground

- For poor to fair ground; 1 less than Q less than 10, 30 less than GSI/RMR89 less than 50, Rock Strength R3-R4
- 2.4 m fully-grouted #7 resin rebar at 1.5 m x 1.5 m spacing (100% coverage) on back and walls; welded wire mesh with 50 mm shotcrete applied to approximately 50% of back and walls. Rehabilitation allowance is estimated at 10% or more.

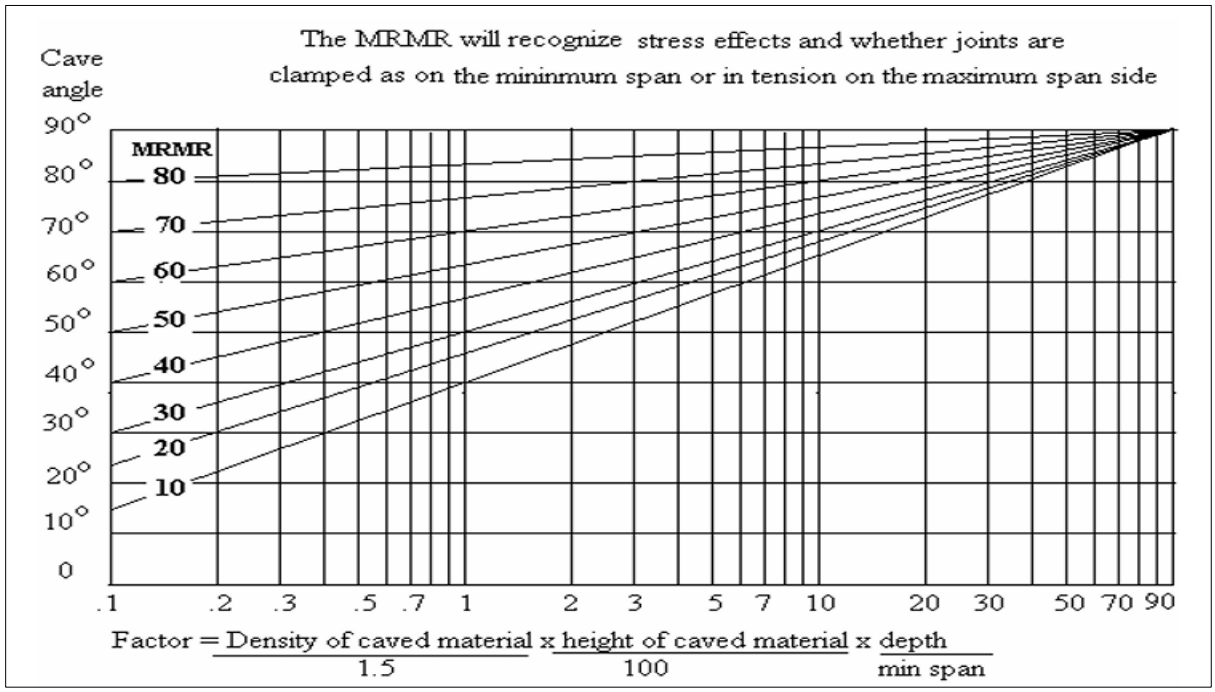
Class 3 Ground

- For fair to good ground; 10 less than Q less than 50, 50 less than GSI/RMR89 less than 80, Rock Strength R4 and above
- 2.4 m fully-grouted #7 resin rebar at 1.5 m x 1.5 m spacing on the back (100% coverage) and walls (50% coverage); welded wire mesh with 50 mm shotcrete where required. Rehabilitation allowance is estimated at 10% or more for extraction development.

16.2.1.7 Surface Subsidence

For the subsidence assessment, the empirical Laubscher MRMR approach was adopted (Figure 16-29) to predict the angle of subsidence after it reaches the surface level; this is a conservative approach for locating important supporting surface infrastructure. The production level located at 1,845 masl has been used as reference for the angle estimates. The inputs required include the density of caved material, the height of caved material, the depth below the surface, and minimum span.

Figure 16-29: Empirical Subsidence Angle Chart



Source: Laubscher, 2000

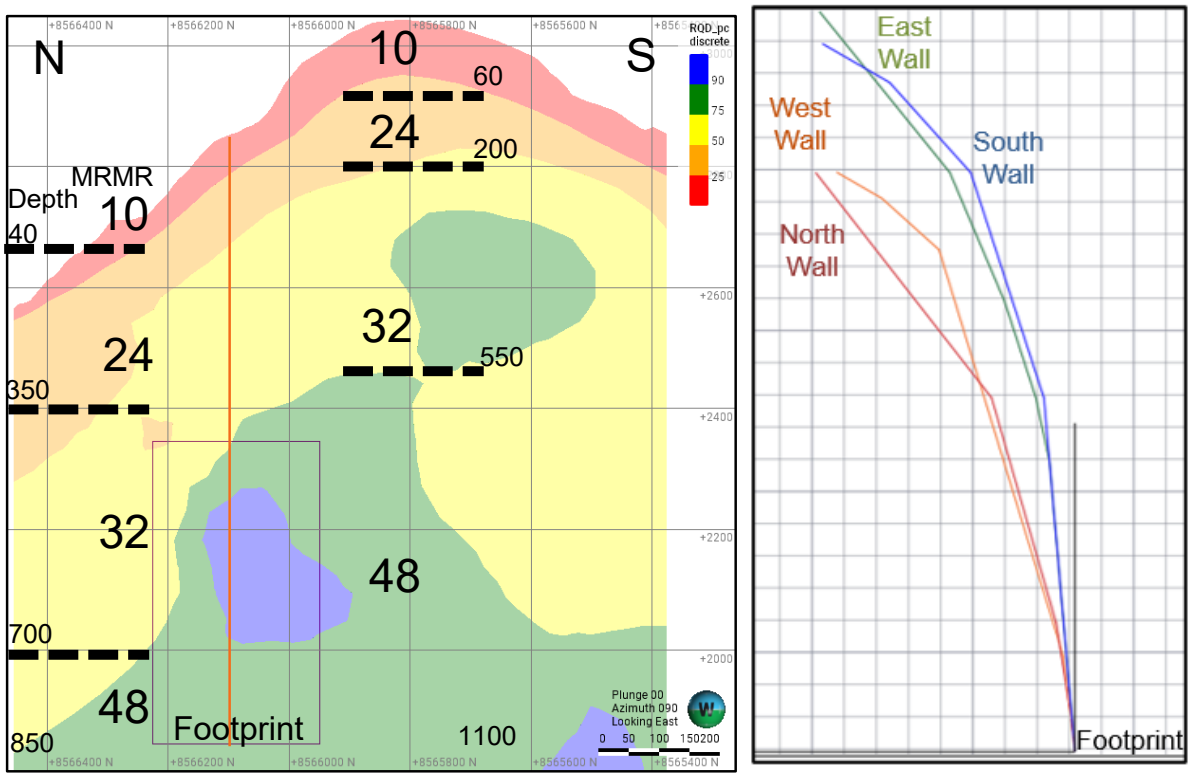
Figure 16-30 shows an example of how different zones of rock mass quality were divided above the footprint for the subsidence empirical approach. The same exercise was conducted for the four sides of the footprint (North, South, East, and West). As a result, different overall subsidence angles were estimated for each side of the cave. The flattest overall angle was estimated for the north wall (66°) while the steepest overall angle was estimated for the east wall (71°).

The main factors influencing the subsidence angles estimated for Aurora, and the resulting variations around the subsidence crater, include:

- The clearly poorer rock mass quality near surface and its thickness variation.
- An apparent stronger, more massive rock mass in the southeast area of the deposit
- The steep topographic slopes, which result in variations in the height of the overlying rock column above the production level.

It must be understood that subsidence angles have widespread variability and empirical methods provide indicative estimates only.

Figure 16-30: Aurora Project Subsidence Zoning Example



Source: SRK, 2026

Left: Example of the zoning of the rock mass column above the footprint for subsidence angle prediction.

Right: Cross section showing the obtained empirical subsidence angle for the North, South, East and West walls.

16.2.1.8 Air Blast and Mudrush Risk

A mudrush is defined as the sudden, uncontrolled ingress of wet muck into underground workings, posing a severe risk to personnel safety, equipment, and mine infrastructure. Wet muck comprises a mixture of water, fine-grained material (mud), and broken rock in proportions that facilitate fluid-like flow when mechanically disturbed by draw activities.

Advancing the project to next level of study (e.g., pre-feasibility study) will require a comprehensive geotechnical and materials testing program to quantify fines generation and flow characteristics. These results should then be benchmarked against analogous caving operations. While the inherent likelihood of a mudrush cannot be entirely eliminated, the associated risks and consequences can be effectively managed. Industry benchmarking (e.g., PT Freeport Indonesia) demonstrates that caving in high-susceptibility environments can be executed safely provided that rigorous draw control, comprehensive Standard Operating Procedures, and Trigger Action Response Plans are implemented and actively enforced.

An air blast is the rapid displacement of large volumes of air, often highly pressurized, through constrained underground excavations. This is typically triggered by a massive fall of ground or the sudden collapse of a stable cave-back into an underlying void.

Cave propagation at the Aurora project is anticipated to occur within a jointed to massive competent rock mass, presenting a baseline potential for localized hang-ups, void formation, and, in extreme scenarios,

air blasts. However, due to the proposed large undercut footprint, the overarching risk of the cave stalling, failing to propagate to the surface, or forming a massive stable overhang is considered very low. Furthermore, the likelihood of air blast events can be reduced to acceptable levels through the implementation of a strict "no void" draw management policy coupled with comprehensive cave monitoring to track the air gap. In addition, the operational consequences of potential air blasts are mitigated by ensuring that a sufficient buffer of broken muck is maintained above the drawpoints at all times.

16.2.2 Underground Mine Design

SRK has been engaged by DLP to complete the underground (UG) mine, infrastructure, and block cave design for the Aurora Project 2026 PEA.

Prior to the 2026 PEA, SRK was engaged by DLP to complete an internal block cave assessment (CAPR004408, 2026-03-25) to assess whether the Aurora deposit would be amenable to block caving, to produce a preliminary mineable inventory based on block caving, and to assess whether the block cave mining method warranted further study. The assessment determined that the geometry and of the Aurora mineralized zone makes it highly amenable to block cave mining.

The footprint finder assessment used GEOVIA's Footprint Finder module to identify a mineable inventory and prepare a simple production schedule, screening candidate footprints by discounted operating cashflow (revenue net of operating and footprint development cost). Additional rapid economic assessments were completed using the discounted cashflows generated from footprint finder and benchmark block cave design, as well as capital cost considerations. The screening-level cost assumption behind this analysis reflects SRK's benchmarking of comparable operations together with SRK's experience with similar deposits.

The preliminary whole-deposit footprint finder analysis work determined that the block model had the potential of being developed either as multi-block caves or as very large block cave footprints. Sensitivity analysis showed that larger footprints generate higher absolute value but carry materially higher capital requirements and a longer construction period, which reduces the rate of return and lengthens payback time. A single block cave footprint, targeting the highest value core of the deposit beneath the open pit, was therefore selected for capital efficiency over the higher absolute value of the larger block cave options, and forms the basis of design for this PEA.

The single-level block cave design for the Aurora mine was developed and scheduled using GEOVIA's PCBC™. The design comprises a single extraction level at elevation 1,845 m, and an undercut level on a single advanced-undercut horizon 18 m above the extraction level. The block cave has a 490-drawpoint / 263-drawbell footprint spanning 14 production drifts.

Table 14-12 provides the current Mineral Resource estimate, effective April 30, 2026.

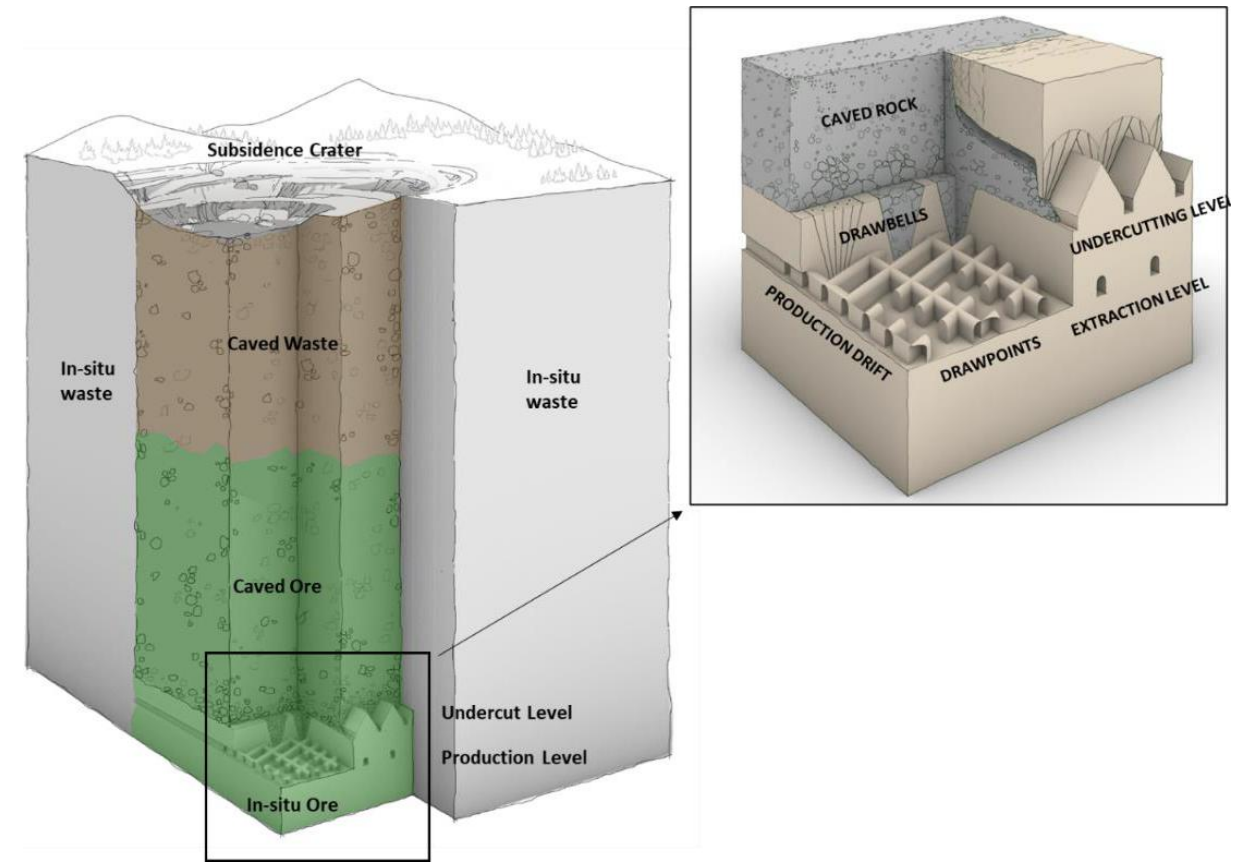
The single-level block cave design was developed using the current Mineral Resource estimate for Aurora; which includes Inferred Mineral Resources, that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves; as this is a Preliminary Economic Assessment, the is no certainty that the Preliminary Economic

Assessment will be realized, and the mine design and all tonnage/grade figures derived from it are necessarily preliminary and may change materially as the Resource model, geotechnical basis of design (Section 16.2.1), and underground infrastructure design (Section 16.2.3) are advanced.

16.2.2.1 Block Caving

Block caving is a large-scale, low-cost underground mining method in which a horizontal undercut is excavated beneath the mineralized zone to remove its vertical support, thereby initiating gravity-induced caving of the rock mass above (Figure 16-31). As the undercut advances, the overlying rock progressively fractures and collapses under its own weight and stress (assisted by blasting where the rock mass is not sufficiently pre-conditioned to cave on its own), and the broken mineralized material is drawn from an underlying extraction level through a grid of drawpoints connected to drawbells, which are funnel-shaped excavations that collect mineralized material from the caving column above and direct it to the drawpoint for loading. As broken mineralized material is extracted, the back of the cave progressively migrates upward through the mineralized zone, and the column above continues to draw down in a continuous, self-sustaining process. The material drawn from the drawpoints is initially mineralized material, but dilutive waste eventually enters the column from above it and/or from the sides.

Figure 16-31: Block Cave Mining Method Example



Source: Jakubec et al., 2023

Block caving is suited to large, thick, and continuous orebodies with rock mass conditions favorable to caving, where high sustained production rates and low unit operating costs can be achieved without the drilling-and-blasting intensity of stoping methods. A trade-off to the low operating costs is a substantial

capital investment in undercut and extraction-level development ahead of first production, and comparatively limited flexibility to alter the extraction sequence once the cave has been initiated. Aurora's selection of this method, and the trade-off studies behind it, are described in Section 16.2.2.2.

16.2.2.2 Mining Method Selection and Trade-off Studies

A mining method must be appropriate to the context of the orebody's rock mass, geometry, and geology. The choice of method depends on a standard set of factors: geology (deposit style and internal zoning); orebody size and geometry (continuity, thickness, dip); grade (value) distribution; geotechnical conditions (rock mass competency and structure); disturbances (in-situ stress, groundwater); and external constraints such as the desired production rate.

The Aurora deposit is large and continuous, with the whole-deposit footprint screening indicating that most of the block model could support a technically viable cave footprint. In considering the Aurora deposit geometry and continuity, grade distribution, and the production rate targeted for this PEA, SRK's assessment is that block caving is the appropriate mining method for the Aurora deposit, consistent with the block cave assessment described further in Section 16.0. Grade distribution is zoned, with a higher-grade core that is the basis for the block cave footprint selected for this PEA.

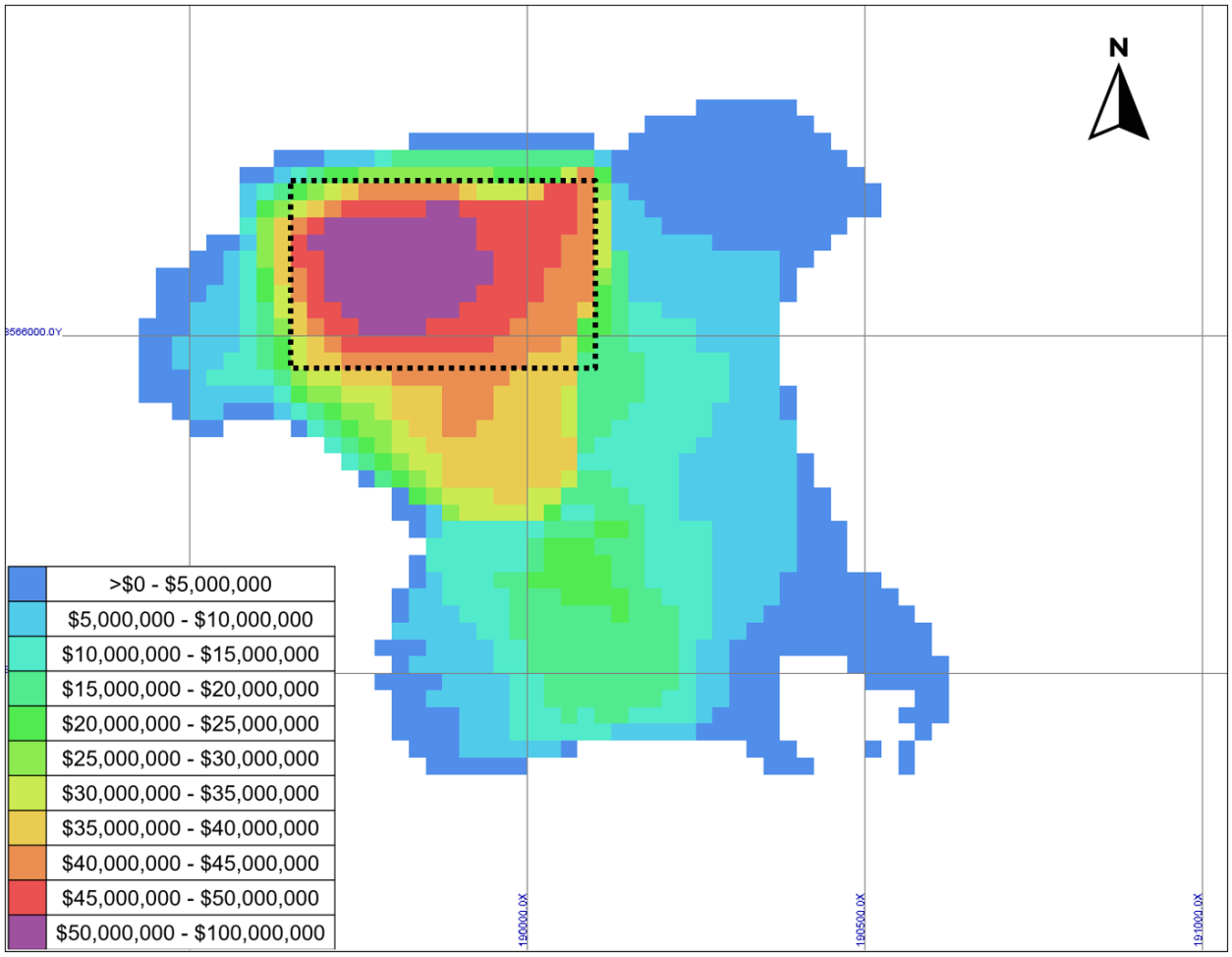
The QP agrees with the selection of the block caving mining method for the Aurora deposit.

16.2.2.3 Block Cave Footprint Selection and Production Modelling

Footprint Selection

As part of this PEA, SRK conducted a follow-up footprint finder analysis to refine the footprint against the current PEA Mineral Resource block model and is the source of the footprint described in Section 16.2.2.4. Footprint finder analysis is used as an initial assessment to identify potentially economic caving footprint elevations and extents and optimize cave footprint selection. Figure 16-32 shows the result of the updated footprint finder run at the best elevation,1,845 m. The colors correspond to the estimated footprint finder dollar value (FFDOL) of the 500 m vertical columns above the selected blocks, and the dashed line shows the footprint selected for the PEA. FFDOL reflects the estimated discounted operating cashflow of the block columsn assessed (NSR – Operating Cost – Footprint Cost).

Figure 16-32: Aurora Project Footprint Finder Results at 1,845 m Elevation (Plan View, 500 m Grid Spacing)



Source: SRK, 2026

PCBC Production Model

Following the footprint refinement using GEOVIA's GEMS Footprint Finder module, the block cave footprint and production were modeled using GEOVIA's PCBC software, designed specifically for the planning and scheduling of block cave mines. PCBC generates draw columns above each drawpoint for which properties are derived from the block model.

The PCBC model estimates the tonnes and grades of material to be extracted from each drawpoint through the application of the cutoff NSR and caving parameters. Caving parameters applied include minimum and maximum HOD, fragmentation assumptions, drawpoint geometry, and mixing characteristics. The model incorporates dilution from the top of the columns and the side walls of the cave, depending on the assumed mixing characteristics. PCBC caving and mixing parameters have been selected based on benchmarking on comparable block caves and SRK's internal experience.

Production shape and cave-back development are estimated in PCBC through the HOD time series reported for each drawpoint. The leading edge of the cave is used to screen a range of potential cave-back positions at a range of cave propagation multiples.

The process of determining the optimal mine plan and cave layouts is an iterative one and should continue to be assessed at future stages of study.

Cut-Off Grade

SRK used a NSR value, calculated by GRE (see Section 16.1.2), on the block model to apply a cut-off grade for individual drawpoint shutoff. A cut-off grade of $20 per tonne was selected for the block cave based on a combined set of initial inputs provided by DLP, and SRK internal benchmarks and experience. This mining cost was used as input for the production analysis.

Maximum Height-of-Draw (HOD)

Each drawpoint will be limited by its maximum HOD, ultimately limiting the total tonnes that can be mucked from the cave if the cutoff grade has not been triggered. A maximum HOD of 500 m was selected for the Aurora block cave based on industry performance benchmarking and SRK's extensive experience with block cave mining projects.

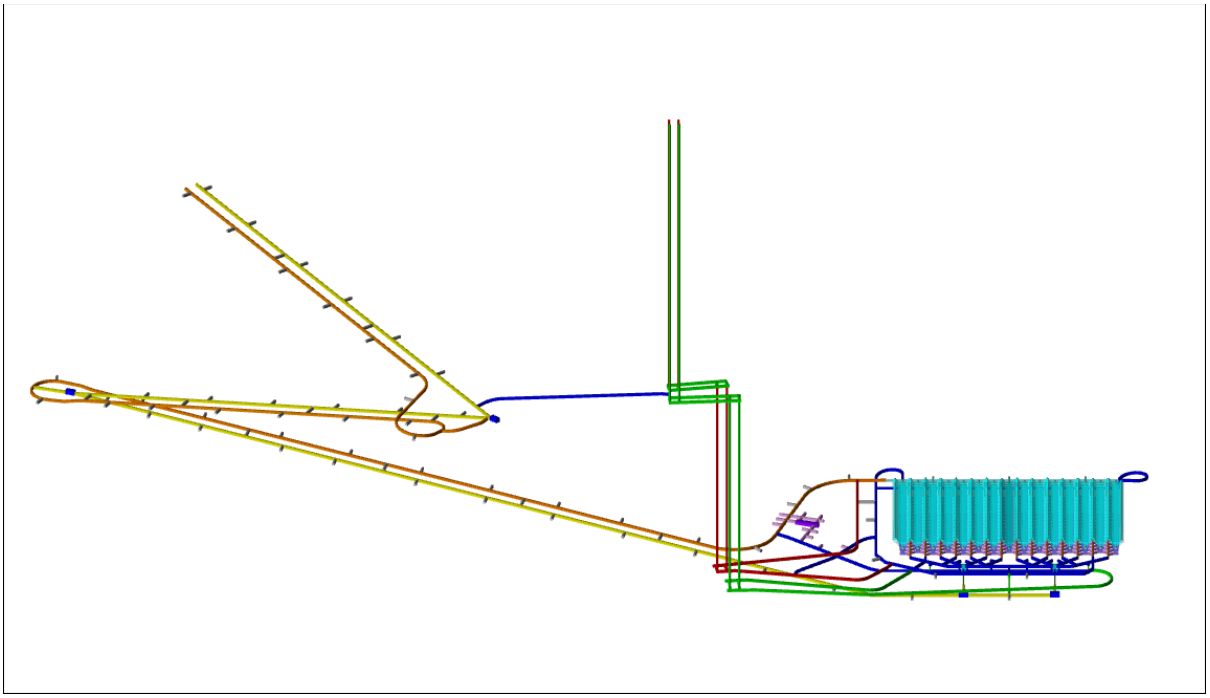
Mixing and Dilution

Block cave dilution is simulated dynamically within PCBC. It is based on the geometry of the cave, mixing parameters, and mining sequence. PCBC does not apply a fixed dilution rate; rather, mixing is applied to all columns at each step of the production model, mixing material between columns and external material from above. Taller mineralized columns, which are mined longer, subsequently incur more dilution over time. This dilution is integrated into the production model outputs.

16.2.2.4 Block Cave Design

A block cave mine design was created in Deswik®, based on the block cave production model (Figure 16-33).

Figure 16-33: Aurora Project Underground Mine Design (Isometric View, Looking North)



Source: SRK, 2026

Extraction Level

The Aurora block cave footprint (Figure 16-34) is located approximately 575 m below the ultimate pit bottom at an elevation of 1,845 m. The footprint has a total of 490 drawpoints (263 drawbells). The cave footprint measures approximately 485 m x 285 m for an area of approximately 138,000 m^2 and a total hydraulic radius of 90 m. The footprint's 263 drawbells are laid out in a herringbone bone configuration with drawbell spacings of 16 m x 32 m.

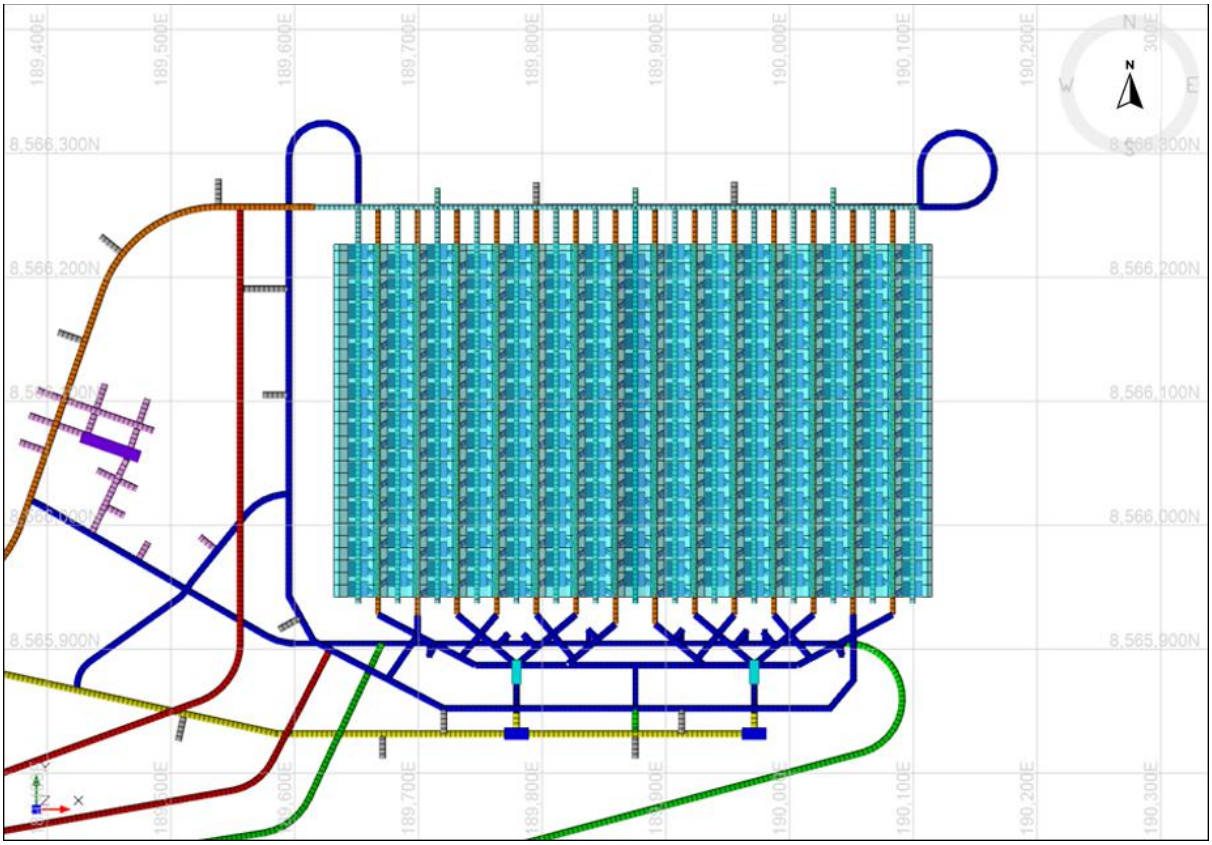
Herringbone and "El Teniente" layouts were both considered for the Aurora block cave layout, with the herringbone layout being selected as it is more favorable for a single-direction mucking configuration. Drawbell dimensions are based on consideration of fragmentation and ground conditions and benchmarking of other comparable operations. The Aurora drawbell spacing was chosen based on the Chuquicamata block caves as a comparable benchmark.

The drawbell blasts are designed at approximately 2,000 m^3 (5,400 t blasted) each (Figure 16-35).

Mineralized material is mucked from the drawpoints and transported directly to the crusher tip points south of the cave footprint.

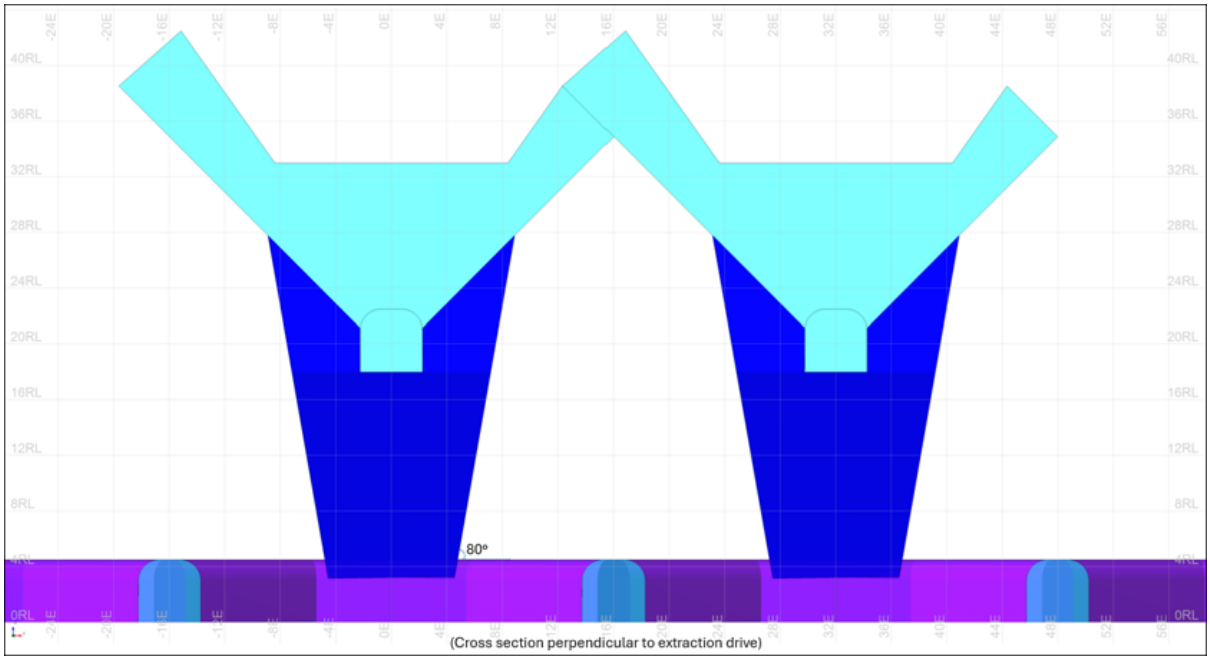
Ventilation for the extraction level is provided by fresh air raises to the south of the level. Air exhausts from the north end of the extractions drives to the footwall drive and is routed to the level's exhaust air circuit.

Figure 16-34: Aurora Project Cave Footprint Layout (Plan View, 100-m Grid Spacing)



Source: SRK, 2026

Figure 16-35: Aurora Project Typical Drawbell Layout (Section View, Looking North, 4-m Grid Spacing)



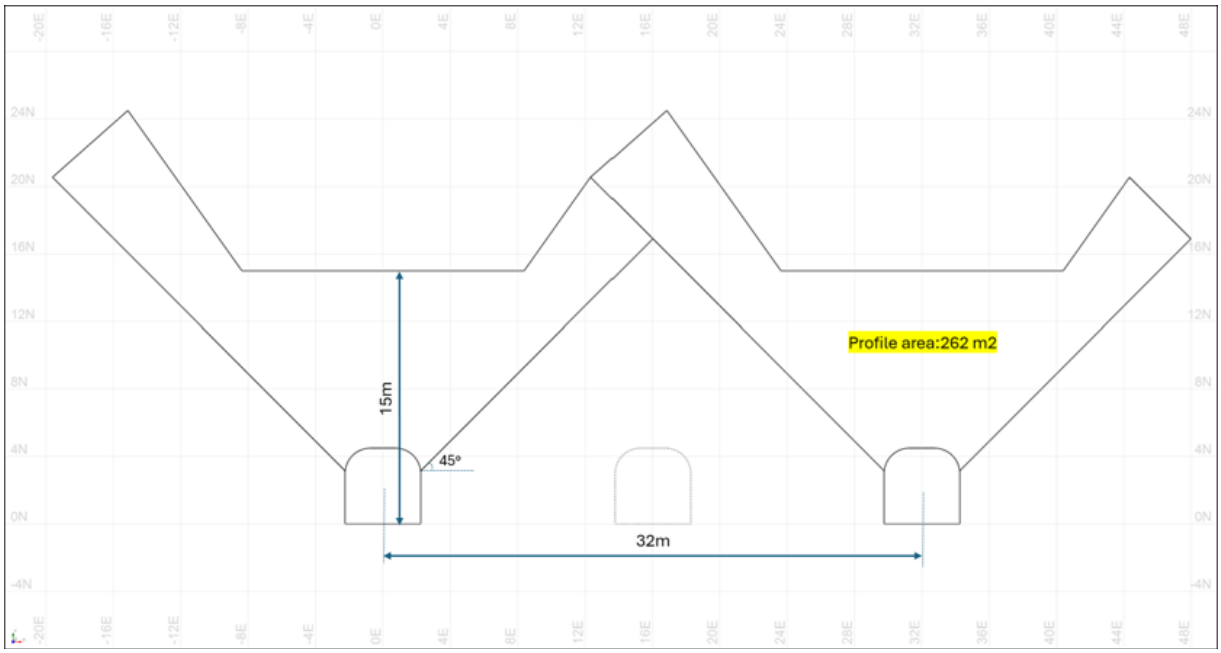
Source: SRK, 2026

Undercut Level

The undercut level is located 18 m above the extraction level and uses a single-drive advanced undercut with a high-sawtooth face profile. The advanced undercutting method has been selected for its benefits of destressing the ground in the extraction level below prior to developing the drawbells. This technique will reduce the risk of potential damage to drawpoints during undercutting and support greater longevity of the level through the block cave's production life.

Undercut drives are centered on the midpoint between its two adjacent extraction drive centerlines spaced 32 m apart, with 15 total undercut drives across the footprint. Undercut rings are drilled on a uniform 3.2-m burden along each drive, giving 1,380 rings in total, and covering approximately 138,000 m² (Figure 16-36).

Figure 16-36: Aurora Project Typical Undercut Ring Layout (Section View, Looking North, 4-m Grid Spacing)



Source: SRK, 2026

Undercutting rings designed at approximately 840 m³ (2,268 t blasted / 1,361 t muck), drilled with a ring drive offset of 30 m from the edge of the corresponding drawbell blast. Only 60% of blasted undercut tonnes will be mucked to confirm successful blasting and mitigate air blast risk.

Undercut mineralized material will be transferred to the extraction level via orepasses on the north of the level and rehandled by haul truck to the primary crushers.

Ventilation to the undercut level is provided by booster fans from the northern undercut footwall access, pulling fresh are from the mine access decline.

Material Handling Level

The footprint is designed with on-level tipping into two crushers to the south of the footprint. Mineralized material is mucked from drawpoints, trammed, and dumped directly into the crushers on-level. The

crushed material is then conveyed to surface. The bottom of the conveyor drives provides a natural low point of the underground mine and would also be used as a water collection level for the footprint.

Ventilation Level

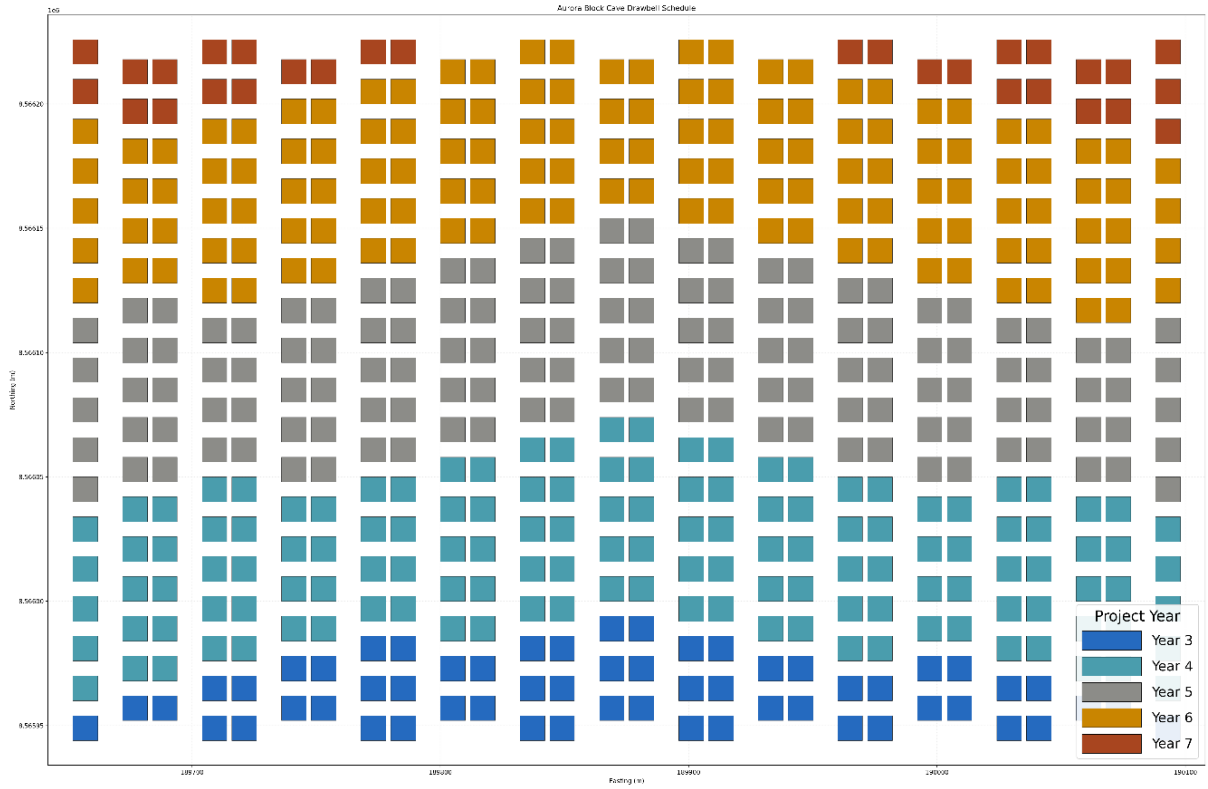
Fresh air for the underground mine is provided by the access decline, and twin fresh air raises from surface. A fresh air ventilation drive runs beneath the southern extremity of the footprint and provides fresh air to the extractions level and material handling via a series of fresh air raises.

Exhaust from the working levels is collected on the footwall drives and exhausted to surface via the conveyor decline and twin exhaust raises.

16.2.2.5 Cave Initiation Sequence

The cave initiation follows a V-chevron sequence: the front initiates at the south-center of the footprint and expands outward and northward with a 14-degree cave front. In alignment with advanced undercutting philosophy, drawbell development is completed within the stress shadow of the undercut level once it has passed over the drawbell location. The advanced undercut strategy involves a high degree of scheduling and coordination between activities on the undercut and extraction levels.

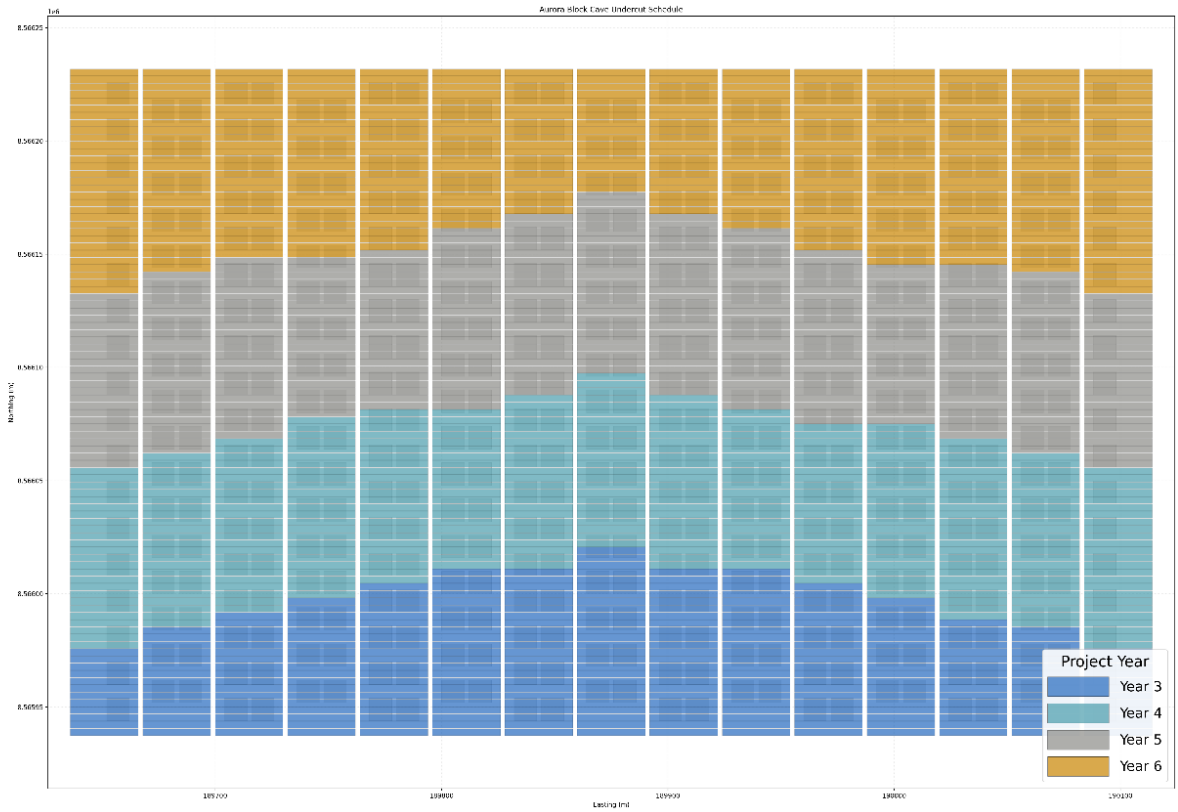
Figure 16-37: Aurora Project Drawbell Blasting Schedule (Plan View)



Source: SRK, 2026

Each of the 263 drawbells is sequenced by its distance from this expanding front (Figure 16-37). Undercut rings are scheduled to fire on a strict three-month lead ahead of the bell they release, using just-in-time scheduling, a maximum lead-lag between undercut drives of 10 m, and a steady-state minimum of one undercut ring blasted per month once a drive is active (Figure 16-38).

Figure 16-38: Aurora Project Undercut Ring Blasting Schedule (Plan View)

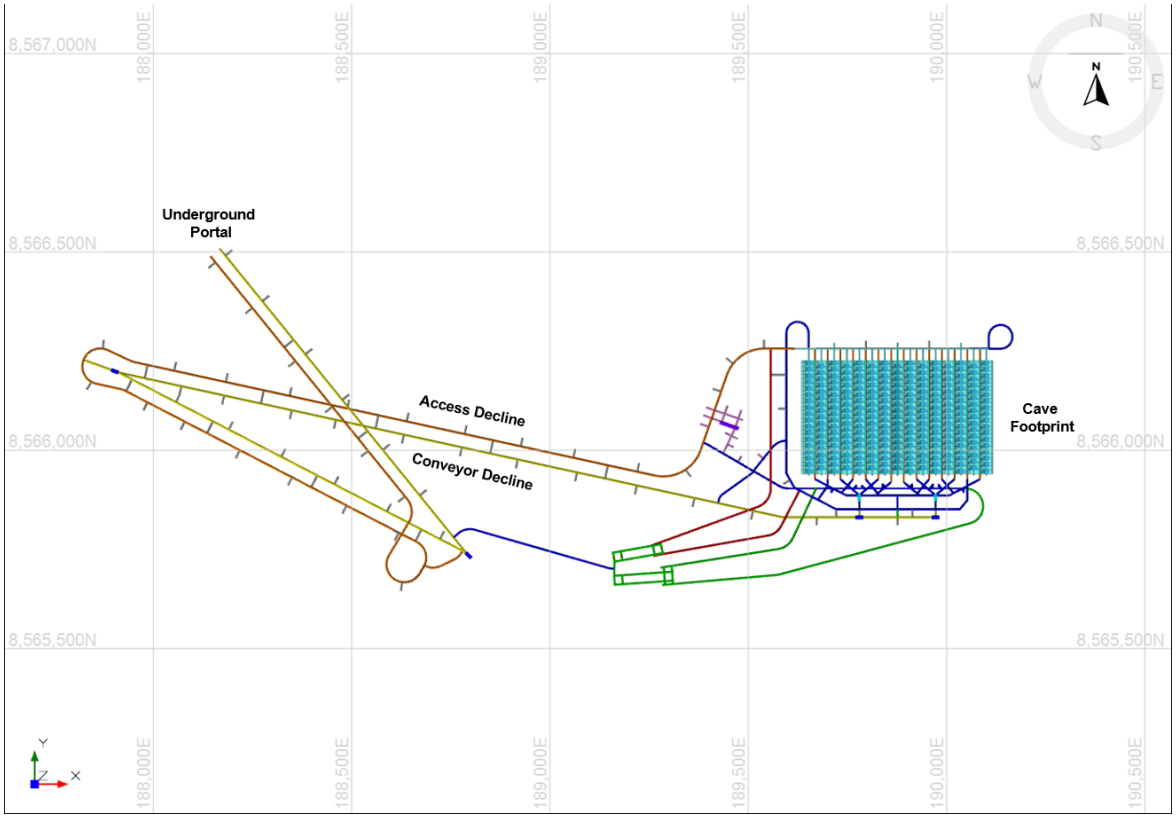


Source: SRK, 2026

16.2.2.6 Underground Access

The underground mine access, seen in Figure 16-39 is by portal, via dual declines: one access decline and one conveyor decline. The portal collars into a crest on the mountainside approximately 1,500 m west of the cave footprint, at an elevation of 2,430 m. The dual declines have a total length of approximately 4,100 m, split into four decline sections.

Figure 16-39: Aurora Project Mine Access Layout (Plan View, 500-m Grid Spacing)



Source: SRK, 2026

16.2.2.7 Development Profiles

Various development profiles were used throughout the design to take into consideration different development uses, equipment requirements and stability, and categories by function. Table 16-7 outlines the development profiles used for the mine design.

Table 16-7: Aurora Project Development Profile Summary

Description	Development Profile	Total Meters
Access Decline	5.5 m W X 5.8 m H	5,122
Conveyor Decline	5.5 m W X 6.0 m H	5,017
Footwall and Infrastructure	5.5 m W X 5.8 m H	4,692
Block Cave – Extraction Drive	4.5 m W X 4.5 m H	4,625
Block Cave - Drawbells	4.5 m W X 4.5 m H	7,788
Block Cave – Undercut Level	4.5 m W X 4.5 m H	5,290
Fresh Air Drives	5.5 m W X 5.8 m H	2,276
Return Air Drives	5.5 m W X 5.8 m H	1,186

Source: SRK, 2026

The vertical development for the main ventilation raises will be developed using raisebore methods.

16.2.3 Underground Infrastructure Design

16.2.3.1 Materials Handling System

Crusher

The selection of the crushing equipment was based primarily on the desired feed arrangement (four tipping points), Load-Haul-Dump (LHD) tramming distances, production rate, estimated caved ore fragmentation size distribution, and desired crusher output size. The use of two gyratory crushers was determined to be the best arrangement for the Aurora project.

Jaw crushers are the most common solution in underground crushing applications when the throughput is less than 1,000 tonnes per hour (tph); however, above 1,000 tph, primary gyratory crushers are more common and for production rates exceeding 2,000 tph, jaw crushers are not commonly used. Therefore, for a production rate of 40,000 tpd, a gyratory crusher is the preferred type of crusher.

To minimize the tramming distances of LHDs on the crusher tipping level, two gyratory crushers are recommended with four loading points for each crusher. As the crushers require loading from four locations, only LHDs should be used to load the crushers at the tipping locations, where material would be fed onto the support arms (aka the spider) of the crushers; an LHD operator can better control the flow of the material being fed into the crusher compared to the use of a haul truck.

Crusher selection was based on a target production rate of 40,000 tpd, and a maximum operating time of 16 hours/day was assumed.

Two FLSmidth gyratory crushers (or equivalent) are recommended for the Aurora Project (Figure 16-40). The FLSmidth 1600-2400 TSUV gyratory crushers have a power rating of 1,000 kW each and can accept a feed of up to 1,600 mm (63 inches) in size and have an adjustable open side setting range of 152 mm (6 inches) – 203 mm (8 inches). This model of crusher is well suited to the estimated crusher feed size shown in Section 16.2.1.5.

Each of the two crushers can produce approximately 3,000 tph at their slowest rate of production, which is when the open-side setting (OSS) is set to 152 mm (6 inches). At this setting, the combined crushing capacity the two crushers would be 6,000 tph and this rate will easily achieve the project's target production of 40,000 tpd.

Figure 16-40: FLSmidth TSUV (Mark V) Gyratory Crusher

Source: FLSmidth, 2026

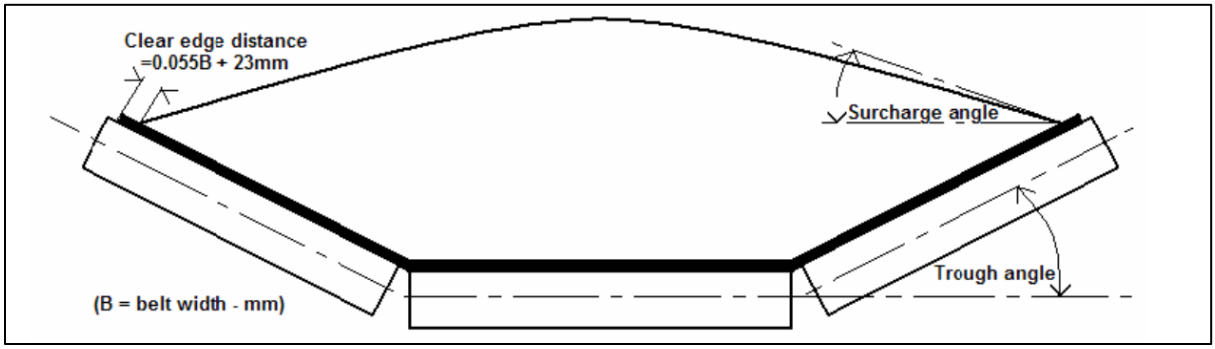
Conveyor

The selection of the conveying equipment was based primarily on the production rate, conveyance gradient, and crusher output size. For simplicity of design, a single belt was selected for the Aurora project.

A material surcharge angle of 10 degrees was assumed for this material, and a troughing angle of 35 degrees was selected for the carrying idlers (Figure 16-41). Due to the conveying distance involved from the crushers to surface, there will be several conveyor belts required, and the longest should not exceed 1,000 meters in length as a general guideline.

Conveyor Equipment Manufacturers Association (CEMA) D6 idlers are recommended for both the carrying and return directions. A 5-inch diameter roll could be reasonably used as well, but the 6-inch roll will improve maintenance demands throughout the conveyors' lives. The roll diameter can be finalized later in the design process.

Figure 16-41: Cross-section of Conveyor Load Showing Trough and Surcharge Angles



Source: Fenner Dunlop_ 2009_ Conveyor Handbook

Insitu ore density is 2.7 tonnes/cubic meter, and the crushed bulk density is estimated to be 1.95 tonnes/cubic meter.

Belt capacity calculations were undertaken, and a single belt with a width of 1,600 mm (63 inches) is recommended. Based on a speed of 2.5 meters/second, this belt will convey 4,300 tph up an inclination of 18% (10.2 degrees) and can convey the daily tonnage production rate of 40,000 tpd in 9.3 hours. The conveying system's daily maximum capacity, based on 16 hours of operation per day, is 68,800 tonnes.

This belt width is a good size for the application as the lift and length of the conveyors demands a higher strength belt, which will not as easily trough as a smaller width.

The belt width of a conveyor must accommodate both the carrying capacity and the material lump size. CEMA states that the maximum lump size should not exceed belt width over five for predominantly lumpy material. The selection of a belt width of 1,600 mm gives a belt width to maximum lump size ratio of 10.5 for when the crushers are set to an OSS of 152 mm (6 inches), and a maximum lump size ratio of 7.9 for when the crushers are set to an OSS of 203 mm (8 inches); therefore, a belt width of 1,600 mm meets CEMA's belt size to maximum lump size ratio guidance, and is therefore well suited for the maximum lump size that is to be expected.

The conveying system would consist of four conveyor lengths as shown in Table 16-8 and in Figure 16-42. A 1.1 safety factor has been included on drive power.

Table 16-8: Aurora Project Conveyor Belt and Motor Sizing

Tag	Length (m)	Width (mm)	Inclination (%)	Belt Speed (metres/s)	Drive Power* Required HP	Drive Power* Required kW	Drive Power* Chosen HP	Drive Power* Chosen kW
C1	375	1,600	0	2.2	706	527	800	597
C2	1,739	1,600	17	2.2	2,592	1,933	2,600**	1,939
C3	1,000	1,600	17	2.2	1,285	958	1,300	969
C4	1,000	1,600	17	2.2	1,285	958	1,300	969

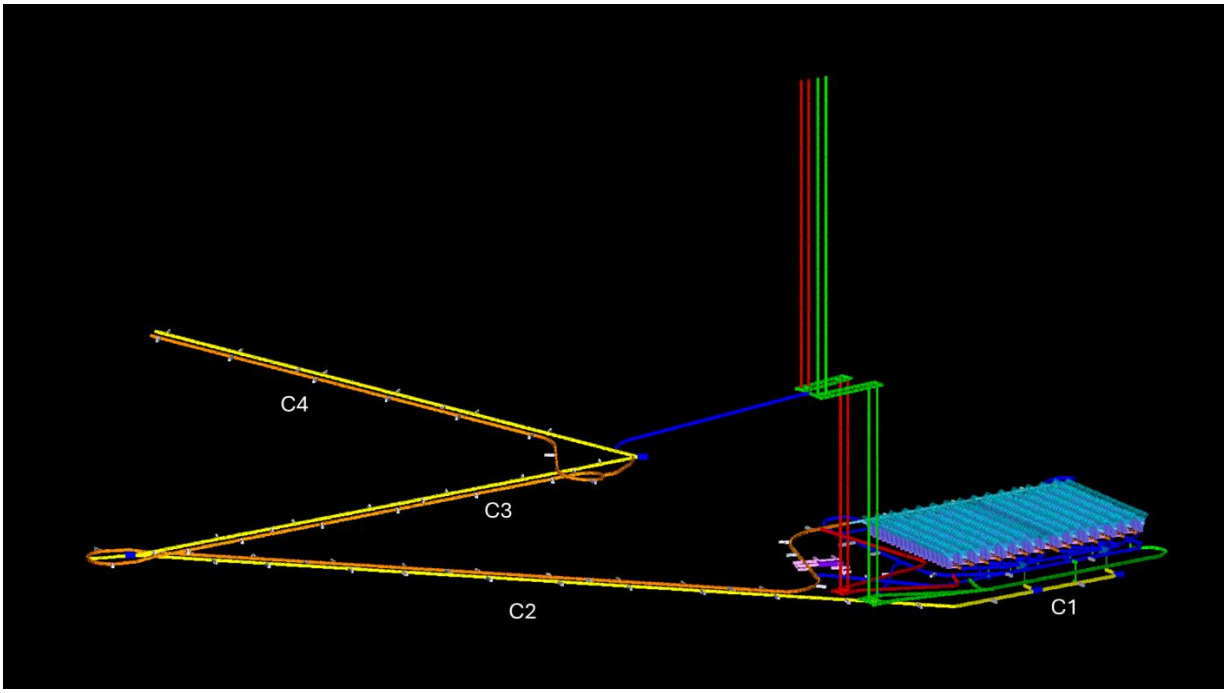
Source: SRK, 2026

Notes:

* Single motor drives.

** Future study should look at the feasibility of splitting this into 2 – 1,300 HP drives

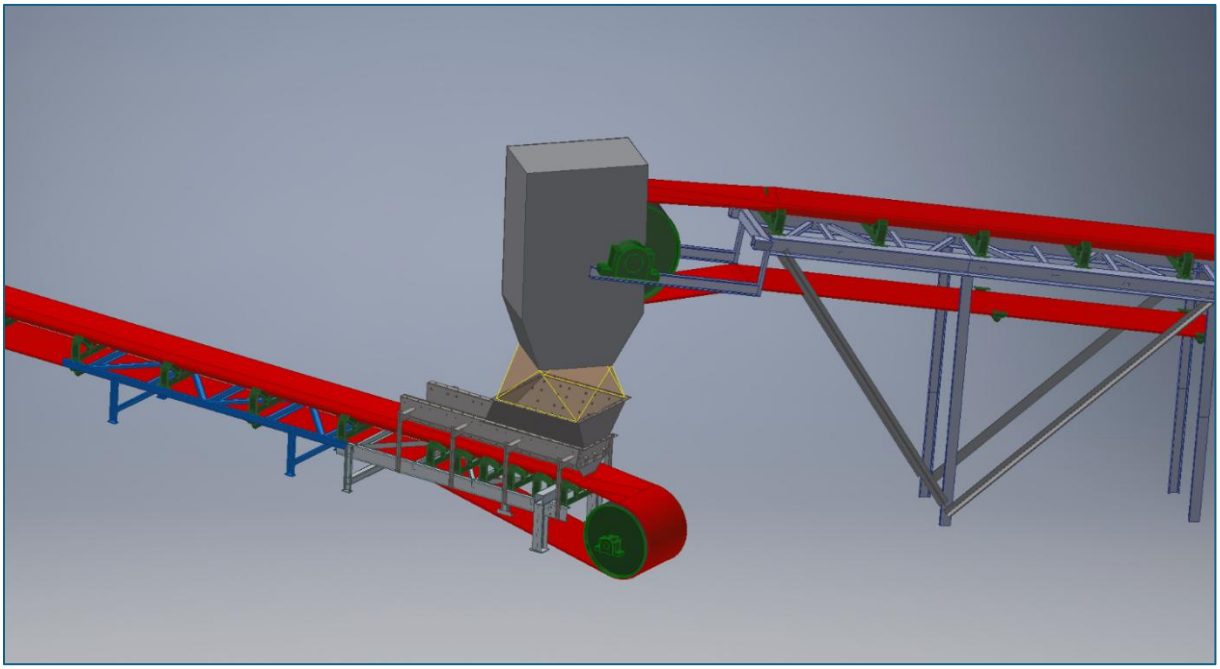
Figure 16-42: Aurora Project Conveyor Transport System (Isometric View)



Source: SRK, 2026

Figure 16-43 shows a typical transfer arrangement between two underground conveyors.

Figure 16-43: Typical Transfer Arrangement



Source: SRK, 2026

16.2.3.2 Ventilation

The ventilation system for the Aurora Mine consists of two 5-m diameter fresh air raises, two 5-m diameter exhaust raises, one service decline down casting airflow to the mine, and one conveyor decline

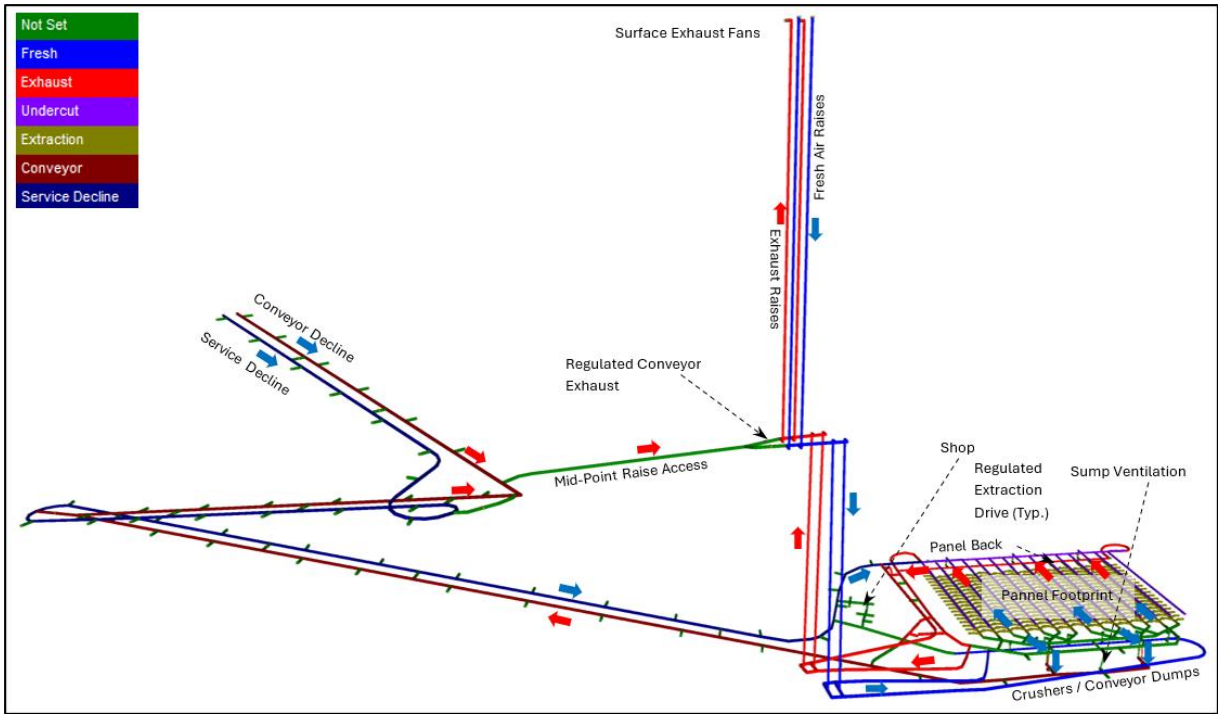
exhausting airflow from the crushers and transfers as shown in Figure 16-44. The principal exhaust fans (two parallel exhaust fans for each of two exhaust raises) are installed on surface with several smaller "booster fans" required for crusher/conveyor exhaust.

The footprint is ventilated by injecting fresh air between the crushers and the extraction drives to provide separate ventilation splits and assist in the natural control of dust. Airflow is drawn through the extraction drives toward the back of the panel and is then directed to the exhaust. The undercut is ventilated by wrapping fresh air across the back side of the footprint from which the auxiliary face fans can draw airflow to the development headings, with the exhaust continuing down the drift to merge with the extraction level exhaust. The end of each extraction drive will be regulated so that an even airflow distribution across extraction drives can be achieved.

The crushers are ventilated to the conveyor along with the sump; dust extraction/filtration systems can be used to remove the dust loading from the crusher exhaust (in addition to water sprays at the crusher and draw points). A single stage ventilation model was developed to provide input to the mine design, required fan operating points, and general ventilation layout.

It is assumed that the primary access will be through the service decline with secondary egress through the conveyor access. The conveyor will be ventilated by drawing airflow up from the crusher/sump area to the exhaust raise mid-point, and down from the surface to the exhaust raise mid-point, breaking up the conveyor route into two ventilated segments.

Figure 16-44: Aurora Project General Ventilation Circuit



Source: SRK, 2026

As the mine design progresses through the study phases based on enhanced information, the ventilation system will need to be refined for alignment with system modifications and improvements. Additional

modeling stages will need to be examined to ensure adequate ventilation during various transition periods.

Air Quantity Estimate

A generic value for diesel dilution of 0.06 cubic meter per second per kilowatt ($m^3/s/kW$) was used to determine the general airflow requirement for the mine. General utilization rates were applied to the equipment based on assumed availability and time used in the underground environment. The general calculations are shown in Table 16-6.

Table 16-9: Aurora Project General Airflow Calculation

Project Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Development and Production Equipment																				
Jumbo, 2 boom	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Emulsion Loader, Development	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Scaler	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Rockbolter	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Cablebolter	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Shotcrete Sprayer	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Production Drill	0	0	0	0	0	2	3	3	2	0	0	0	0	0	0	0	0	0	0	0
ITH Drill with V30 Reamer	0	0	0	0	0	1	2	2	2	0	0	0	0	0	0	0	0	0	0	0
Emulsion Loader, Production	0	0	0	0	0	2	2	2	2	0	0	0	0	0	0	0	0	0	0	0
Water Cannon	0	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blockholer	0	0	0	0	0	1	1	1	1	2	2	2	2	2	2	2	2	2	2	1
Mobile Rockbreaker	0	0	0	0	0	1	5	9	9	6	6	6	6	4	4	4	4	4	4	2
Hydrofracturing Equipment																				
LHD, 7.0 m^3, 14 t	0	0	0	0	0	2	4	7	10	12	15	15	15	15	15	15	15	15	12	7
LHD, 7.5 m^3, 15 t																				
LHD, 8.6 m^3, 17 t	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0	0	0	0
LHD, 10.7 m^3, 21 t																				
LH515i Onboard Automation Kit	0	0	0	0	0	2	4	7	10	12	15	15	15	15	15	15	15	15	12	7
Haulage Truck, 51 t	2	2	3	4	4	4	2	2	2	2	0	0	0	0	0	0	0	0	0	0
Auxiliary Equipment																				
Grader	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift	2	2	2	2	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	2
Scissor Lift Attachments																				
Boom Truck	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Flat Deck Truck	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel/Lube Truck	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Explosives Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Transmixer	4	4	4	4	5	4	4	4	4	4	2	2	2	2	2	2	2	2	2	2
Forklift/Telehandler	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water Sprayer	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Roller	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Bobcat	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	1	1	1	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Personnel Carrier MacLean (22 person)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Personnel Carrier Toyota	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Supervisor/Engineering Vehicle	5	10	10	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Cassettes System Prime Mover	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Cassettes Attachments																				
Weighted Dilution Airflow (m^3/s)	**176.0**	**183.2**	**209.4**	**242.8**	**261.3**	**311.8**	**305.8**	**358.2**	**394.0**	**392.5**	**295.1**	**295.1**	**295.1**	**288.5**	**288.5**	**288.5**	**288.5**	**288.5**	**249.3**	**174.1**
Diesel Contingency (25%) (m^3/s)	44.0	45.8	52.4	60.7	65.3	78.0	76.4	89.5	98.5	98.1	73.8	73.8	73.8	72.1	72.1	72.1	72.1	72.1	62.3	43.5
Footprint Production Flexibility (40% LHD) (m^3/s)	0.0	0.0	0.0	0.0	0.0	10.4	20.9	36.6	52.2	62.7	78.3	78.3	78.3	78.3	78.3	78.3	78.3	78.3	62.7	36.6
Crusher Ventilation (m^3/s)				50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0
Conveyor Exhaust (m^3/s)				50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0
Magazine (m^3/s)			25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0
Pump Chamber (m^3/s)			25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0
Shop (m^3/s)			75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0

Project Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Total (m^3/s)	**220.0**	**228.9**	**386.8**	**528.5**	**551.6**	**625.2**	**628.1**	**709.3**	**769.8**	**778.4**	**672.2**	**672.2**	**672.2**	**663.9**	**663.9**	**663.9**	**663.9**	**663.9**	**599.3**	**479.2**

Source: SRK, 2026

Ventilation Fans

The principal surface exhaust fans were sized based on a ventilation model developed for a single maximum duty time phase. The principal exhaust fan operating point is shown in Table 16-10. The anticipated number of operating fans are shown

Table 16-11. The overall power calculation for the ventilation system takes into consideration all the operating fans, and it is based on the name plate fan power number.

Table 16-10: Aurora Project Exhaust Fan Operating Point

Fan	Airflow (m^3/s)	Raise Pressure (kPa)	Installation Losses (kPa)	Full Pressure (kPa)	Efficiency (%)	Power (kW)
Exhaust Fan 1	180	2.92	0.58	3.5	85%	742
Exhaust Fan 2	180	2.92	0.58	3.5	85%	742
Exhaust Fan 3	180	2.92	0.58	3.5	85%	742
Exhaust Fan 4	180	2.92	0.58	3.5	85%	742

Source: SRK, 2026

Table 16-11: Aurora Project Ventilation Power and Operating Fan Numbers

Project Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
two stage 132 kW aux fans (long auxiliary ventilation)	2	5	5	5	5	5	5													
75 kW aux fan (medium auxiliary ventilation)			6	7	7	7	7	5	5	5	2	2	2	2	2	2	2	2	2	2
50 kW aux fan (short or conveniance auxilairy fan)			2	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	2	2
25 kW aux fan (services)			4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4
Development portal fan (150kW)	1	1	1	1	1															
Crusher fan (50kW)			2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Main Exhaust Fans (750 kW)			2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Installed Power (kW)	**678**	**1,470**	**3,720**	**5,595**	**5,595**	**5,445**	**5,445**	**3,975**	**3,975**	**3,975**	**3,750**	**3,750**	**3,750**	**3,750**	**3,750**	**3,750**	**3,750**	**3,750**	**3,450**	**3,450**

Source: SRK, 2026

16.2.3.3 Underground Power and Electrical Distribution

The major electrical power consumption in the mine would be from the following:

- Main and auxiliary ventilation fans
- Underground crushers
- Conveying system
- Mine dewatering system
- Drilling equipment
- Air compressors and maintenance shop.

Power for underground operations will be supplied from circuits on surface. It is assumed by SRK that there is sufficient capacity from the planned site power distribution system to support the underground mining loads.

The underground mine will have two 20 Mega Volt-Amperes (MVA) transformers supplied from site power sources and stepping down to 20 kV. The underground feeder circuits will be connected to high voltage switchgear, providing protection, control, and metering. Underground feeders will consist of 25 kV rated 500 kilo-circular mil armored cable and voltage rated junction boxes to facilitate development, ventilation, and dewatering loads during the initial construction phase.

Standard distribution and utilization voltages will be employed: 660 volt for low voltage motors such as fans, sump pumps, etc.; six kV for medium voltage motors (greater than 225 kW); and the main power feeders underground will be at 20 kV. Due to the loads associated with the dewatering and material handling systems and the practical limitations associated with multiple high voltage cables being installed and maintained underground, a higher transmission voltage was selected to reduce cable ampacity requirements and the total number of circuits required. Main underground feeders are sized for approximately 10 MVA capacity each.

Underground power for low voltage applications will be from movable Mine Power Centers (MPC), 20 kV to 660 Volt, 750 kVA rated. These MPCs will initially be employed for the development of the declines and once this is completed, most will be redistributed for mining loads. MPCs and electrical infrastructure to support the material handling system will be located within decline cross-cuts, thus additional excavations will not be required.

Broad assumptions have been made for the mining operational equipment. A lumped power allowance for mining activities of 10 MVA was made. This will support localized ventilation and dewatering, material conditioning equipment (crushers, rockbreakers, etc.), and miscellaneous support equipment and maintenance equipment.

The material handling conveyor system will be supplied from four dedicated feeders and associated switchgear. Each conveyor segment will use Variable Frequency Drives (VFDs). The VFDs facilitate a smooth acceleration of the belts while loaded and reduce power system transient start-up loads.

The dewatering system will be supplied by two dedicated feeders. The dewatering pump drives will be equipped with VFDs to minimize hydrodynamic effects such as water hammer and allow for a controlled start-up.

In future studies, consideration should be given to adding emergency electrical supply capacity to the underground mine to support ventilation and communications during a power interruption to the project site. Propane-based electrical generators could be used to provide sufficient power to enable life-sustaining ventilation but not permit ongoing mine operations. The ventilation would be sufficient for workers to deploy to the nearest underground refuge station and wait out the power interruption, or if deemed necessary by management, to leave the underground mine. The generators would also permit limited use of the mine's underground dewatering system, but only for normal pumping requirements. Estimating emergency power requirements was deemed by SRK to be outside the scope of this PEA study.

16.2.3.4 Underground Dewatering

The proposed block cave mining method is a subsidence mining method, meaning that as ore is mined, material above the orebody will be undermined and therefore subside. The surface subsidence means that over time, the surface water diversion features to restrict the flow of surface water (from rainfall) into the open pit mine will eventually lose their effectiveness; however, the timing for this loss has not been estimated and should be reviewed in a future study.

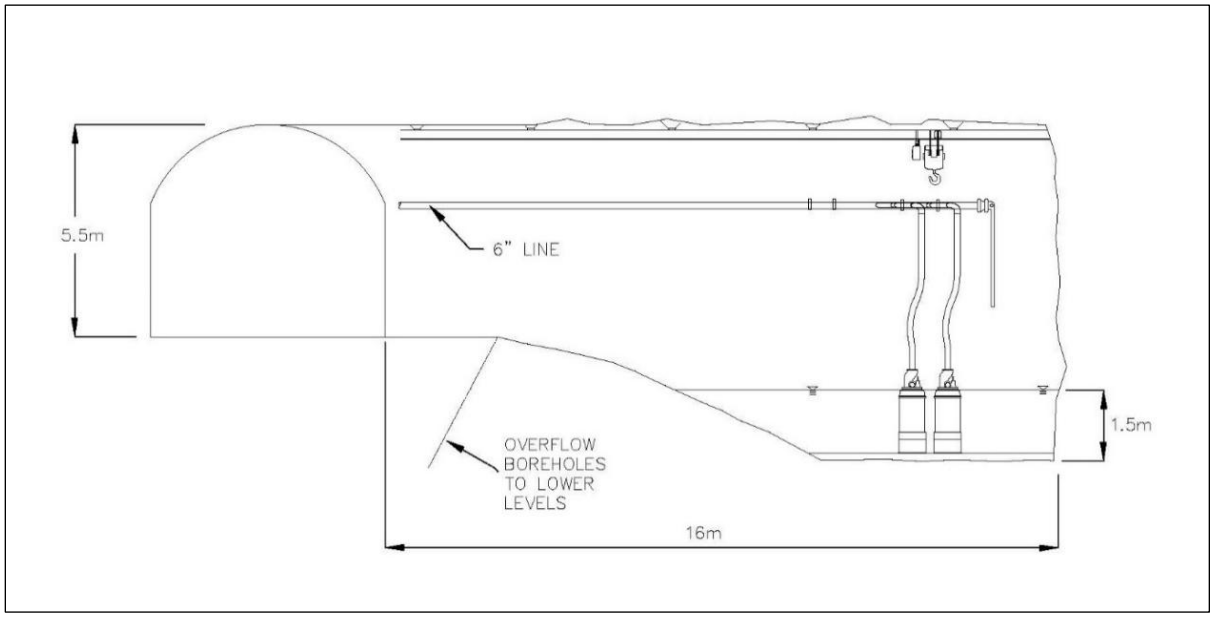
SRK has taken a conservative approach with the dewatering calculations so that the underground mine is adequately prepared for large inflows of water early in the mine life. To do otherwise risks the mine being overwhelmed with an extremely high volume of water inflow that could flood portions of the mine and impact production.

Ramp Sumps

As the access and conveyor declines are developed, dirty water sumps will be constructed on approximately 60-m vertical intervals. These sumps will each be equipped with a pair of 19-kW submersible pumps. Each pump can transfer 32 L/s of water up to the next sump along the decline via a 150-mm HDPE flanged pipe. Two pumps are placed in each of the sumps to provide redundancy. Once a decline has been fully developed, the sumps will be interconnected to the next sump below in a daisy-chain arrangement, and the collected sump water will be delivered to one of two primary dewatering pumping stations.

The dirty water sumps will use existing remucks that were developed at fixed intervals during the construction of the access and conveyor declines. These remucks will be converted into sumps with the floors slashed to have a gradient of -15%. The submersible pumps will both be hung from a monorail for ease of access and maintenance. Power will be distributed to the pumps via locally mounted switchgear, with simple float switches controlling the pumps themselves. The solids reporting to the sump will remain in the sump for clean-up via an LHD as required. The pumps will be hung at least 600 mm off the bottom of the sump. A general arrangement drawing of the dirty water sump is provided in Figure 16-45.

Figure 16-45: Aurora Project Ramp Sump Arrangement



Source: SRK, 2026
Note: dimensions shown may change in the final design

Dewatering Stations

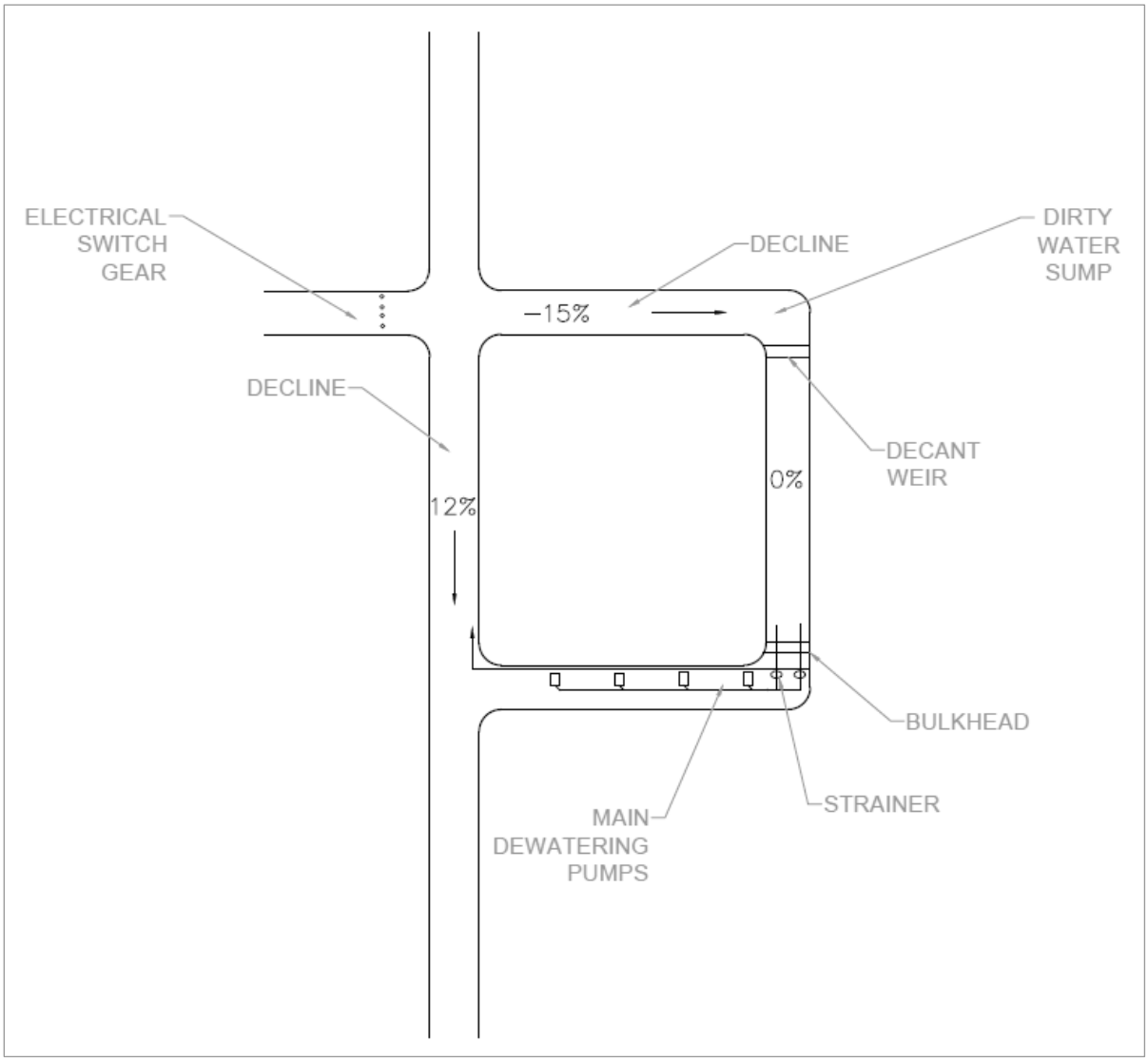
The dewatering design includes two dewatering stations that pump water out of the mine. One station is located below the crushers, and the dewatering line from this station runs up the conveyor decline. The other station is at the bottom of the access decline, and the dewatering line from this station runs up the access decline. Both lines tie into the surface water management system on surface.

Each dewatering station has six dewatering pumps sized at 63.1 l/s (1,000 gallons per minute [gpm]) capability and has a nominal power capacity of 671 kW (900 hp) respectively. Each dewatering station will have two spare pumps that will be ready for immediate use when maintenance must be performed on the other pumps or a working pump has failed. The pumps will be a stacked impeller, high efficiency type of pump and will be equipped with isolating valves at both ends and check valves on the discharge. The control system for the pumps will ensure even wear across all the pumps, even the standby pumps, as this is important to have peak flow capability when required. Since the pumps are operated in parallel, a pump with higher wear than its partners would be unable to contribute as effectively to the discharge flow rate.

Each of the two dewatering stations will consist of a sump driven down at -15% gradient. The far end of the sump will interconnect via a weir with a flat decant sump. Water will flow over the weir and into the sump leaving the solids behind. The retaining dam at the end of the sump separates it from the pump room. The pump room is flat-lying and leads back to the main decline and joins it at the same elevation.

Three independent drains mounted 300 mm off the floor of the sump will penetrate the retaining dam and are equipped with basket strainers. The three drains lead into a suction header that feeds the main dewatering pumps. Figure 16-46 shows the proposed dewatering station arrangement.

Figure 16-46: Aurora Project Dewatering Station Arrangement



Source: SRK, 2026

The pump impellers are common across the set, so that individual spare impellers can be kept in stock instead of entire pumps. Should one or more impellers fail, the spare impellers can be used to return a pump to service quickly. Other wear parts such as bearings, packing glands, couplings, and thrust plates should also be stocked.

The outlet of each pump is tied into the 200-mm discharge header, which then leads to the 300-mm schedule 80 seamless grade B welded steel pipe that comprises the main dewatering discharge line. The discharge header will be equipped with water hammer suppression bladders to protect the equipment and pipeline.

The conveyor and access declines were selected for the routing of the dewatering lines as they are existing development and therefore will not require additional CAPEX as compared to drilling dedicated boreholes for the dewatering lines. When located in the declines, the dewatering lines can be readily inspected and maintained as required over the life of the mine.

Switchgear for the pump stations will be sited in a nearby chamber located up the decline at least at the same level as the dirty water sump. Unplanned releases of water will flow down and away from the electrical switchgear. The pump chamber will be equipped with an overhead walkway, servicing area, monitoring equipment, and an area to store tools and supplies.

16.2.3.5 Maintenance Facilities

The main workshop will be located at the 1856 level with access to the main access decline. The maintenance facilities will include the following:

- Minimum of 500 square meters of service and repair area with 20-t overhead crane (space requirements based on equipment fleet size)
- Three fuel and lube bays; 2 for large equipment, 1 for small equipment (e.g., pick-up truck)
- Tire and welding bays
- Two wash bays fitted with water/oil separator
- Warehouse for parts storage
- Refuge station and offices
- Electrical substation

The workshop is ventilated from the access decline and has a dedicated connection to the return air level. Fire doors at both accesses to the workshop will be used to control ventilation during normal and emergency situations. The workshop will also be equipped with a fire suppression system.

The service area will be equipped with a stationary compressor and airlines to power air tools and provide compressed air as needed.

The surface equipment related to underground operation (haulage trucks, pickup trucks, forklift, etc.) will be serviced at the surface maintenance shop. Major rebuild work will be conducted at the surface maintenance shop.

For the underground project start up, it is assumed that the surface shop will be used during mine development, until the underground shop is established.

16.2.3.6 Compressed Air

The underground mobile drilling equipment such as jumbos, production drills, and explosives loaders would be equipped with their own compressors. No reticulated compressed air system was envisioned to be required underground.

Two portable compressors would be used to satisfy compressed air consumption for miscellaneous underground operations.

16.2.3.7 Service Water

A winterized water tank would be located on surface above the conveyor decline portal to provide water storage. The surface water storage tanks will be automatically replenished as required from the mine's existing surface water distribution system. Industrial-quality water would be distributed in 100-mm-

diameter pipelines throughout the underground workings for drilling equipment, dust suppression, and firefighting. Flexible hoses would be used to connect water pipelines to drilling equipment at working faces.

The major water consumption would be by drilling equipment such as jumbos, longhole production drills, and cable-bolters, with pump capacities of approximately 100 L/min as per the technical specification. The rockbolters would require 50 L/min of water. Water will be also used for dust suppression for each mucking LHD, crushers, and at conveyor loading points.

16.2.3.8 Fire Water

An underground water reservoir of 15,000 liters capacity that is dedicated for use in fighting fires will be located at approximately the 2,428 m elevation and will provide a source of water for fighting fires. The reservoir will us an existing remuck built during the development of the access decline and will be located close to a cross-cut linking the access and conveyor declines.

16.2.3.9 Potable Water

Potable water will be brought underground in standard water cooler sized bottles and will be placed with electric chilled water dispensers at all locations where workers gather for meals, such as at the refuge stations and in the underground shop.

16.2.3.10 Transportation of Personnel and Materials

All mine supplies and personnel would access the underground via the main access decline.

The personnel vehicles would be used to shuttle employees from surface to the underground workings via the main access decline and back during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel-powered trucks as transportation underground. Mechanics and electricians would use the maintenance service vehicles.

Two boom trucks and two flat deck trucks would be used to move supplies, drill parts, and other consumables from surface to active underground workings.

An explosives truck would be used to deliver explosives to the working faces.

Transmixer trucks would be used for shotcrete and concrete delivery to the underground.

16.2.3.11 Mine Safety

Refuge Stations

A mix of fixed refuge stations and mobile refuge stations will be utilized underground. The primary fixed 60-person refuge would be located within the underground shop complex and would provide emergency shelter for all shop personnel and mine operations personnel working in the area. Additionally, self-contained mobile refuge station chambers with 20-person capacity would be used for emergencies. These chambers would be portable for flexibility of location at the most appropriate areas of the mine and would be relocated as mining activities progress. All personnel would carry a self-contained, self-rescuer of minimum duration of 30 minutes and will be within 30 minutes of a refuge chamber or fresh air base

Emergency Egress

The main access decline provides primary access to the mine. Secondary emergency egress to the surface from the underground mine is via the conveyor decline.

Fire Prevention

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fueling stations, explosive and detonator magazines, and wherever a fire hazard exists. Every vehicle would carry at least one fire extinguisher of adequate size and proper type. All underground mobile vehicles would be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at all fresh air intakes to alert underground workers in the event of an emergency.

Conveyor Fire Risk

An underground conveyor belt carries the risk of a conveyor belt fire. With the appropriate measures in place, the risk of a conveyor fire can be safely managed. To manage the risk, the final conveyor design includes allowance for the following:

- Fire retardant belt
- Fire retardant grease and lubricants
- Ventilation controls to isolate the air in the conveyor decline in the event of a fire
- Infra-red heat detection equipment
- Fire suppression system installed in conveyor decline
- Regular inspection of the conveyor decline during operation to detect the development of faulty rollers, belt misalignment, or excessive dust build-up

Fire-rated airlock doors in the connecting development from the conveyor decline to the access decline will be required.

Mine Rescue

A mine rescue Emergency Response Plan would be developed, kept updated, and followed in an emergency. Three fully trained and equipped mine rescue teams would be established and would be available on rotation for rescue efforts. The mine rescue teams would be trained for surface and underground emergencies.

Surface and underground training facilities would be necessary for ongoing employee training and refresher training programs. Dedicated mine rescue equipment including a fire tender, ambulance, and all supporting testing and maintenance equipment for mine rescue purposes would be available, and specific underground mine rescue equipment would include self-contained breathing apparatuses.

16.2.3.12 Communications

SRK recommends installing a Wi-Fi communications system to permit radio/mobile phone communications throughout the mine. Wi-Fi fiber and ethernet cable can be premanufactured in pre-determined lengths to mitigate installation costs. Pre-manufactured communication cables would provide an ease of maintenance by having the ability to remove/add communication links from a simple plug and play setup.

In strategic areas such as refuge stations, underground shop, and at the crusher stations, Voice Over Internet Protocol (VOIP) phones would be implemented as a part of the design. The VOIP phones would rely on a direct connection back to the local communication node ethernet network switch via CAT6 cabling utilizing power-over-ethernet. In the event of a power failure, all major nodes including surface communications should have an uninterruptable power supply (UPS) designed to meet or exceed appropriate requirements to maintain communications in an emergency event and ensure seamless transition to the generator source.

Fiber Optic

The proposed topology for communications and automation would be through a primary fiber optic trunk network. Fiber interface panels would be located at key locations to provide interconnectivity to associated users. Primary user networks would be:

- Voice – VOIP phones for communications in shop spaces, refuge stations, and other strategic locations.
- Mine-wide Radio – Wi-Fi radios for mine communications that could be interfaced with an existing surface VHF radio signal.
- Business Network – provides wired and wireless connectivity to the business network for mine resources to connect to the internet, email, and other business applications while in the mine.
- Process Network – provides a communications network for programmable logic controllers (PLCs), distributed control systems , and other process specific devices. This network would be secured and access to the business network, and internet would be restricted by a firewall and intermediate demilitarized zone to protect the integrity of the control systems. Other systems on this network include geotechnical monitoring, personnel tracking, and vehicle telemetry.
- Blasting Network – provides communication to wireless portable blasting units. Blasting units would be interfaced to the central blasting station on surface.
- Security Network – provides a dedicated Virtual Local Area Network for the connection of underground cameras to a remote digital video recorder .

Leaky feeder coax cable system has been taken into consideration throughout this design process, where it can be simply replaced by the wireless network for the following but not limited reasons:

- Product(s) initial cost
- Limited wireless bandwidth (throughput)
- Radio licensing and possible interference with neighboring radio frequencies

- Limited voice and process data acquisition

The leaky feeder system has served the mining industry for many years with proven success in the past but comes with limited capabilities. With the need to keep current with technology, the proposed fiber optic system is highly recommended to mitigate installation/operating costs with the ability to be interconnected with current technology.

The proposed communications network design would use fiber optic cabling to provide communications for mining activities, voice telephony, and business data. The fiber optic system would also be designed to provide process information to the control room operators via computers and monitors in the control room, which display information from the underground network of computers, cameras, PLC panels, fan controls, pump controls, control valves, flow meters, and other process equipment.

Wireless Access points

Wireless access points (APs) for Wi-Fi coverage and provisions for additional devices such as ventilation/air quality monitoring, and other equipment as may be required would be installed. This would permit mobile telephone or tablet communications and data transfer like surface services provided in office and administrative spaces. The current proposal is to install two wireless APs at each mine power station, with one on each side of the control equipment. Additionally, wireless APs would also be installed in the shop space and refuge stations to support employees working on equipment and throughout the surface facilities (office space, substation, etc.).

16.2.3.13 Monitoring and Control Systems

The geotechnical monitoring systems would be installed to allow monitoring of cave propagation, surface subsidence, and pillar and drive deformation. A production management system would be used for monitoring of the tonnage mucked from each drawpoint to control caving draw.

A combination of 12-pair fiber optic (main trunk) distribution cables and 6-pair fiber optic (secondary branch) drop cables would be installed in parallel with the electrical reticulation system. Network communications and control racks would be installed at various electrical substation, control room, and ventilation raise locations. These control racks act to terminate the fiber optic cable, allowing breakout of communication and control services as required.

The main functions provided by the fiber optic network include:

- Data communications – process control, automation, and computer network
- Voice communications – fixed and mobile phones.

Functions are deployable in both hardwire and wireless formats.

The resulting network provides a unified communications backbone, supporting advanced mine monitoring and control.

The extraction level production LHD mucking operation would be controlled using a system such as Sandvik's AutoMine Lite system or equivalent.

The Sandvik Optimine Monitoring & Location Tracking Package, or equivalent, would be used for scheduling, task management, data collection and reporting, and location trucking.

16.2.4 Underground Life-of-Mine Schedule

16.2.4.1 Mine Development Strategy

The development and production sequence is designed and optimized to bring the block cave into production as quickly as possible to target the high-grade core of the deposit. Consideration was given to the overall mining rates required, the geotechnical conditions likely to be encountered, the ventilation, infrastructure, logistics, and equipment requirements.

Development advance rates are assumed at 125 m per heading per month in the main declines and 90 m per heading per month in all other heading types, with a 500-m per month aggregate maximum across all active headings. These constraints are based on SRK benchmarking and prior project experience. The mine design assumes a surface battery limit, with underground development waste hauled to a surface stockpile.

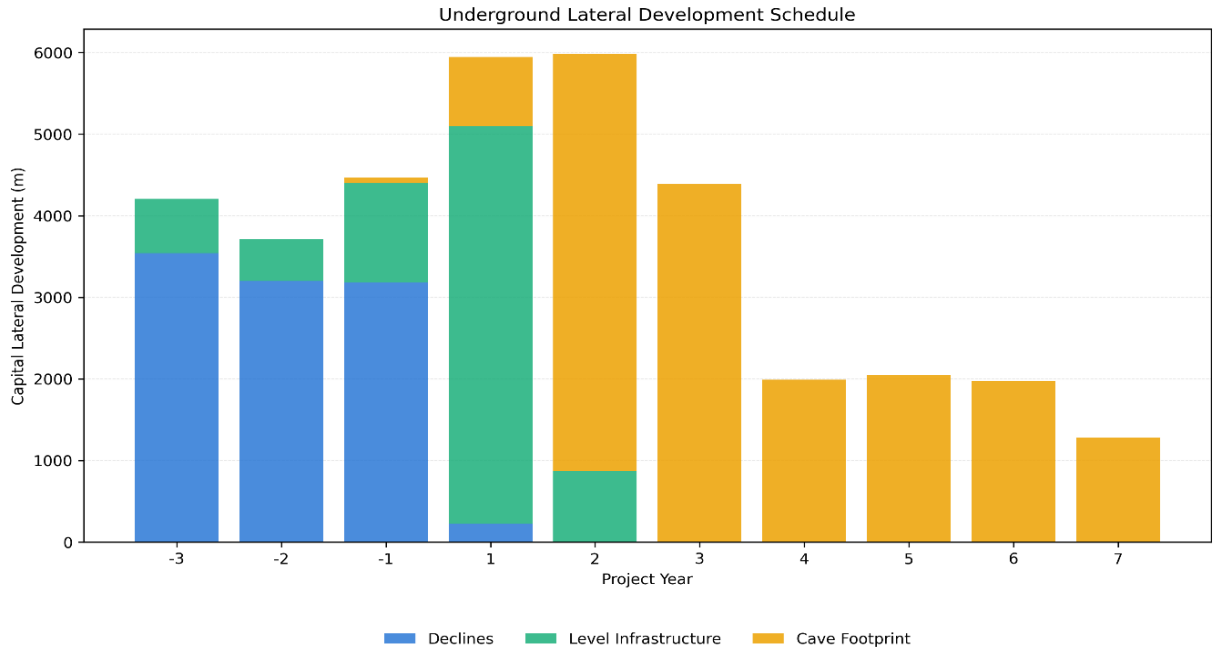
The installation of the primary ventilation fans and establishment of the first primary crusher are considered critical milestones in the initial development schedule as they are required to facilitate cave establishment. These are prioritized in the development schedule ahead of beginning cave construction.

All mine development is treated as waste and would be transported to surface via haul truck.

16.2.4.2 Development Schedule

Underground lateral development is scheduled to begin at the beginning of Project Year -3 and continue through to the completion of the underground development and cave construction in Project Year 7. Figure 16-47 shows the lateral and vertical development schedule through the Life-of-Mine (LOM). All underground development for the block cave is considered capital development.

Figure 16-47: Aurora Project Underground Lateral Development Schedule



Source: SRK, 2026

The underground mine development includes a total of 35,995 m of lateral development and 4,527 m of vertical development.

16.2.4.3 Block Cave Production Schedule

The Aurora block cave was designed for a steady-state production rate of 14.4 Mtpa (40,000 tpd). At full steady-state production, and assuming 80% drawpoint availability, the average cave draw will be approximately 102 tonnes of mineralized material per day per drawpoint.

Block Cave Establishment and Ramp-Up Strategy

The ramp-up of the block cave is dependent on the undercut and drawbell construction rates. Cave construction is planned in alignment with an advanced undercut methodology and driven primarily by bell construction rate. A construction rate of six drawbells per month, with a subsequent average undercutting rate of 30 rings per month (3.2 m/day advance and 2,880 m^2 undercut area per month).

Block cave establishment begins with the undercutting beginning in Project Year 3 (Month 3), four months ahead of the first drawbell blast in Project Year 3 (Month 7). Drawbell construction rate ramp-ups to its full rate over three months. The rate of six bells per month is maintained until completion of cave construction in Project Year 7 (Month 3).

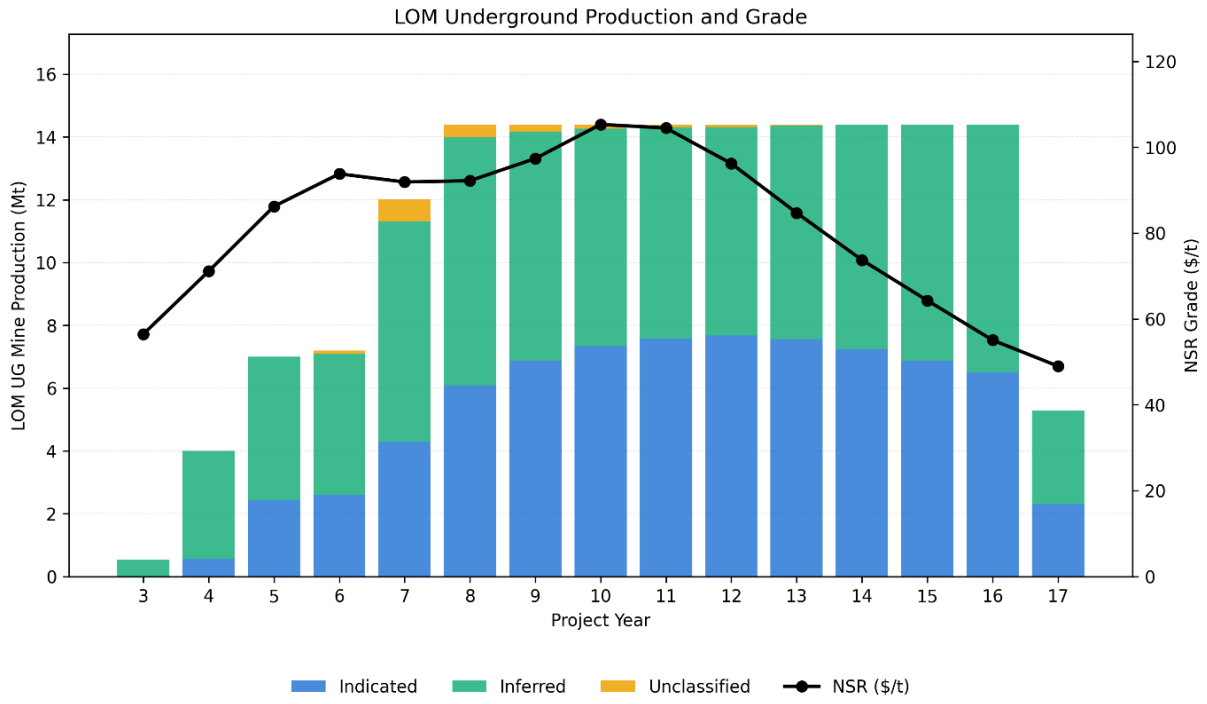
During cave construction, the block cave will ramp up to half its planned peak production rate over the first 2-years and hold this rate until cave construction is complete. Upon completion of cave construction, the cave will ramp up to its full rate of 14.4 Mtpa in Project Year 7.

Block Cave Production Schedule

The Aurora block cave is scheduled to produce at full rate from Project Year 8 through Project Year 16 and ramping down for closure in Project Year 17. A total of 165.6 Mt of mineralized material is planned to be mined from the cave with an average NSR value of $85 per tonne.

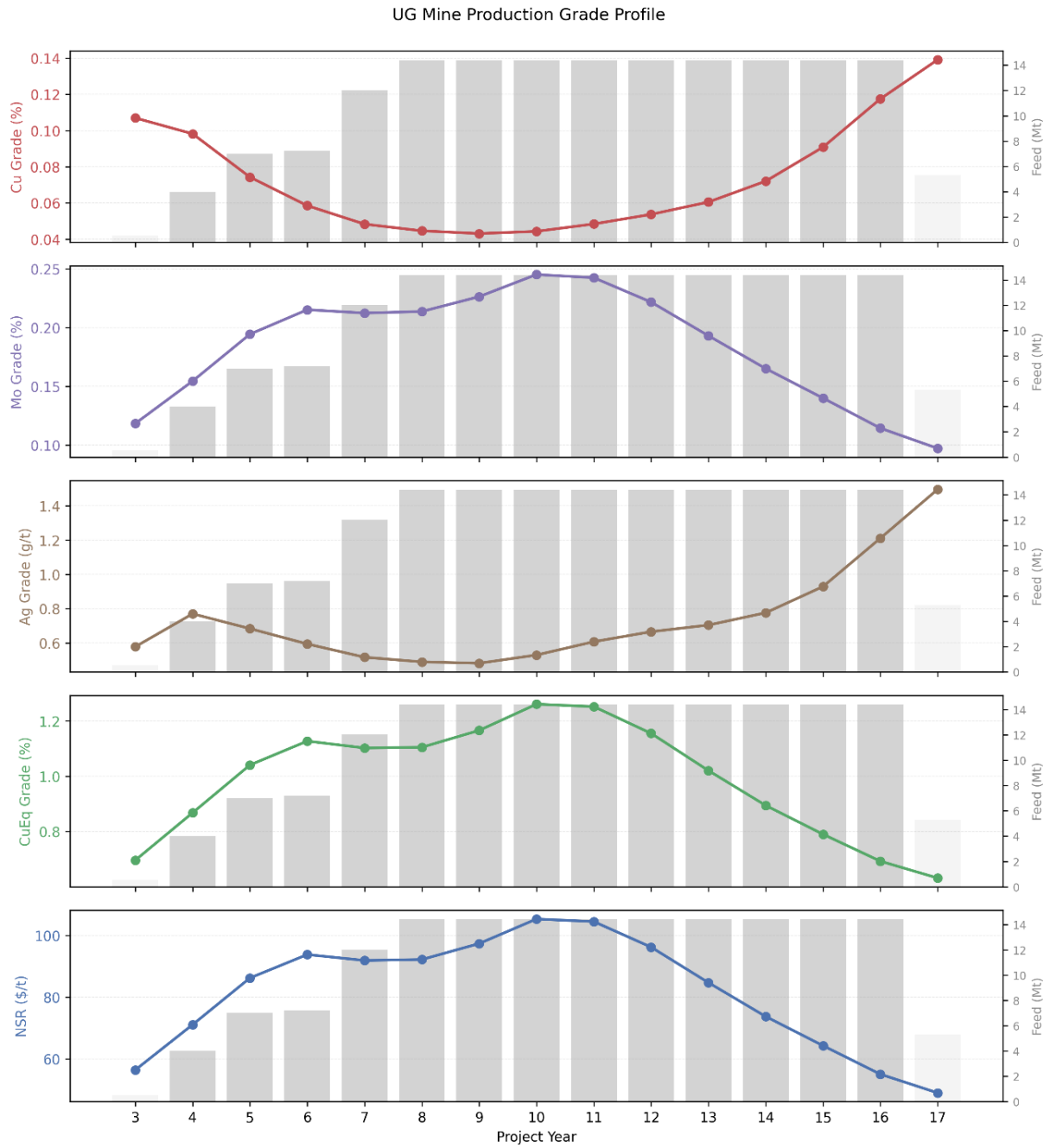
Figure 16-48 shows the LOM production, resource class distribution and NSR value, and Figure 16-49 shows the individual grade distributions through LOM.

Figure 16-48: Aurora Project Underground Cave Production Schedule



Source: SRK, 2026

Figure 16-49: Aurora Project Underground Cave Production Grade Profiles



Source: SRK, 2026

16.2.5 Underground Mobile Equipment

The size of the equipment fleet was based on the scheduled quantities of work, estimated from first principles cycle times and productivities, benchmarking, and practical experience. The selected equipment size satisfies the maximum size of excavations based on geotechnical recommendations.

The mine intends to operate two 12-hour shifts per day. This is reduced to 16.6 effective working hours per day after considering travel time, lunch breaks, pre-shift meetings, and other miscellaneous breaks.

The estimated number of operating units was converted to a fleet size by accounting for equipment mechanical availability of 80% to 90%, depending on the type of equipment.

During the initial stage of the mine development, 17-t LHDs and 51-t haul trucks would be used to transport blasted development material out of the mine. Once the materials handling system is in place, material generated during production-level development would be re-handled through the ore pass system by the production LHDs. The development crew also includes two boom jumbos, emulsion loaders, and ground support equipment (rockbolter, cablebolter, scaler).

The owner's mobile equipment fleet for the Aurora project underground mine would be completely rubber-tired and diesel-powered. The mobile equipment schedule was prepared on an annual basis. The required underground mobile equipment fleet would vary over the life of the Project. The underground mine mobile equipment list and maximum units in the equipment fleet are presented in Table 16-12.

Table 16-12: Aurora Project Underground Mine Mobile Equipment

Item	Description	Number of units (Maximum)
Development and Production Equipment		
Jumbo, 2 boom	DD422i	2
Emulsion Loader, Development	EC3	2
Scaler	RS3	2
Rockbolter	DS412i	2
Cablebolter	DS422i	2
Shotcrete Sprayer	SS5	2
Production Drill	DL422i	3
ITH Drill with V30 Reamer	DU412i	2
Emulsion Loader, Production	MC5	2
Water Cannon	WC3	1
Blockholer	BH3	2
Mobile Rockbreaker	RB3	9
LHD, 7.0 m^3, 14 t	LH514	15
LHD, 8.6 m^3, 17 t	LH617i	2
Haulage Truck, 51 t	TH551i	4
Auxiliary Equipment		
Grader	GR5	2
Scissor Lift	SL3	4
Boom Truck	BT3	2
Flat Deck Truck	DT3	2
Fuel/Lube Truck	FL3	2
Transmixer	TM3	5
Water Sprayer	WS3	2
Roller	Roller	1
Backhoe	310 P-TIER	1
Bobcat	Bobcat	1
Forklift/Telehandler	MHT	2
Mechanics Truck	Toyota Hurth	3
Personnel Carrier MacLean (22 person)	PC3	2

Item	Description	Number of units (Maximum)
Personnel Carrier Toyota	Toyota Van	3
Supervisor/Engineering Vehicle	Toyota	15
Explosives Truck	Explosives Truck	1
Cassettes System Prime Mover	CS-3	2

16.2.6 Underground Mine Personnel

Underground mining employees for the Aurora project underground mine are divided into four categories:

- Mine management and technical staff
- Mine operational management staff
- Mine operational labor
- Mine maintenance and electrical labor

An underground rotating roster was estimated by SRK based on 12-hour shift schedule and an equivalent time on and off crew rotation for most of the underground workforce. The mine roster of hourly labor was estimated on an annual basis based on the development and production schedule, mine services, and maintenance requirements for crew rotation. Hourly personnel include labor for mine development, production, services, and maintenance.

The workforce estimates have been largely based on a productivity analysis of underground activities and the physical requirements of the mine schedule. The mine roster factor of 2.32 was estimated to account for additional labor based on shift days off and annual absence from work including holidays, vacation time, sickness, training, and other duties. That factor was applied to the estimated on-site labor requirements to estimate the total mining labor roster and labor costs.

16.2.7 Comments on Mining Methods (Risks and Opportunities)

This section presents a description and discussion around the key risks and opportunities for the underground mine.

16.2.7.1 Key Underground Risks

Mining technical risks in implementing block cave mining of the Aurora orebody include:

- Delays in access development and main ventilation raises and circuit
- Difficulties in undercutting and ramp-up delays
- Stability and seismicity issues during cave development and propagation
- Airblast during cave development and propagation
- Fragmentation and material handling issues, including oversize, fines, and hang-ups
- Poor drawpoint interaction
- Overdraw, dilution, and recovery issues
- Mudrushes during cave mining

16.2.7.2 Underground Mine Opportunities

The internal footprint finder assessment study completed by SRK prior to the PEA identified potential to evaluate additional cave footprints and to expand the block cave beyond the initial underground design considered in this study. This may provide opportunities to increase mine life and underground throughput in future studies.

Geotechnical evaluation and modeling of larger development drives on the extraction level to use large loaders could provide increased block cave productivity.

Consider contractor development and cave construction to accelerate block cave establishment and production ramp-up.

16.2.7.3 Recommendations

The QP makes the following recommendations relating to mining risks and opportunities:

- Acquire comprehensive orebody knowledge through the implementation of data collection and analysis programs for rock mass, structures, and water for cave access and development.
- Undertake more work in the next level of study to evaluate rock mass caveability and understand the steps that would have to be taken if the rock mass did not cave as expected (e.g., preconditioning via hydraulic fracturing). This work will require close collaboration between the cave mining and geotechnical consultants.
- Undertake more work in the next level of study, as understanding of the resource is advanced, to evaluate additional block cave footprints and extend the underground mine life, and consider utilization of the infrastructure established for the initial block cave footprint.
- Further optimize mine design and development schedule to ensure main ventilation raises are established as quickly as possible to support footprint development and cave construction.
- Implement an undercutting strategy that minimizes damage to the rock mass due to induced abutment stress through advanced undercutting.
- Continue mine optimization and design work as the understanding of the resource is advanced. The footprint selection, draw strategy, and mine layout used in this study should be validated at the conclusion of each major resource model or geotechnical model update.
- Design the mine to reflect industry standards and the latest learnings from deep caves.
- Undertake numerical stability modeling and benchmarking.
- Develop and implement a comprehensive cave monitoring and instrumentation program prior to cave commissioning. This will enable the operator to adjust activities such as the rate and direction of undercutting and draw strategy to eliminate airblast risk and minimize potential seismic risk.
- Undertake a mudrush risk study, including an assessment of the physical properties of the caved material and the development of a surface water management strategy.
- Consider managing the subsidence zone, including water diversion, and subsidence slope management.

- Consider establishing an independent Technical Review Board to advise on compliance with industry standard block caving practices and procedures.

16.3 Mine Dewatering

The Aurora project has a complicated groundwater and dewatering system which involves the cumulative and often interacting impact of groundwater flow to tunnels, the hydrogeologic impact of block caving, and the augmented recharge within the open pit.

The block cave requires 24 kilometers of underground development (tunnels, access levels, shafts, ore shoots, and cave-pre-development) prior to extracting ore (SRK, 2026). All of these developments will produce some groundwater. Concurrently, there will be open pit development, but much of the open pit will be above the theorized groundwater level. However, the open pit will change the surface water and groundwater recharge system, resulting in a greater amount of available groundwater for the underground workings. The full conceptual groundwater flow model is discussed below.

The PEA must consider dewatering not only because of the cost of pumps and groundwater extraction, but also because dewatering of tunnels, pits, and underground development produces mine-impacted water (MIW), and MIW is likely to be poor quality (see Section 20.0), and must be treated.

16.3.1 Groundwater Dataset

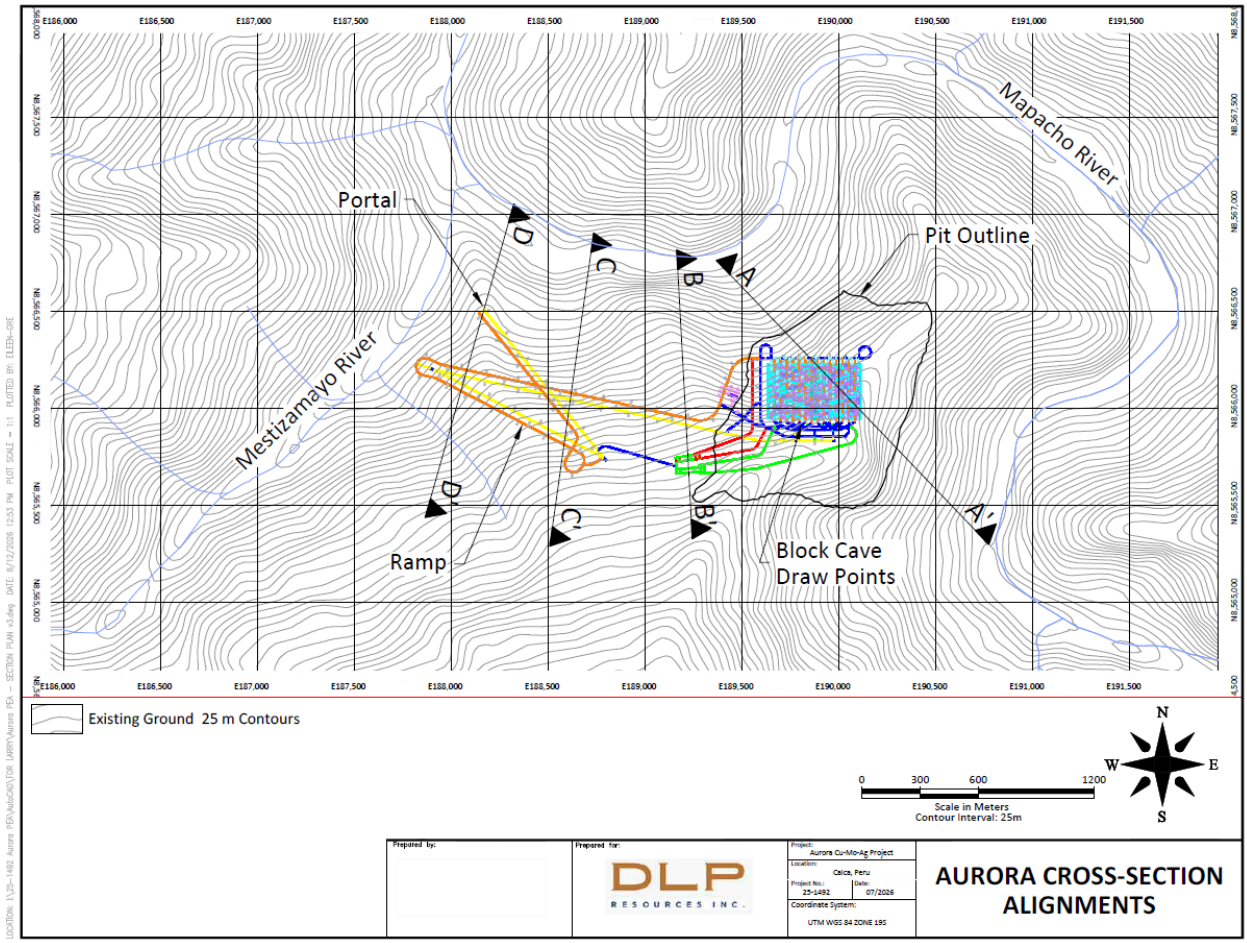
The project is in early-stage development, and there is nearly no available groundwater characterization data. As a result, groundwater and aquifer parameters must be estimated. In general, the following is assumed:

- The regional water level is slightly higher than the level of the Mapacho River. This means that the pit bottom will be dry (with respect to groundwater seepage). Many of the block cave workings will be significantly below the level of the Mapacho River, which will be the largest source of groundwater recharge in the region.
- Aquifer properties can only be estimated. GRE assumes that the bulk representative porous media equivalent conductivity of hard rock is between $1x10^{-5}$ cm/s and $1x10^{-4}$ cm/s. Block caving creates a rubble zone as the cave-in propagates upward. This “block cave zone” was simulated as broken rock with a $1x10^{-3}$ cm/s saturated hydraulic conductivity in a broad zone extending upward in a steep cone from the block cave development areas.

16.3.2 Conceptual Model

Figure 16-50 shows the location of the underground mine workings and the open pit with respect to the Mapacho River.

Figure 16-50: Mine Workings, Pit, and Mapacho River with 2D model Cross Section Location Map



Source: GRE, 2026

At PEA level, it is necessary to greatly simplify the conceptual groundwater flow model. Based on the direct site observation, GRE believes that nearly all of the dewatering water will come either from seepage from the Mapacho River reporting to the underground workings, or from direct precipitation within the pit. It is anticipated that the pit wall rock will not produce groundwater. It is also anticipated that nearly all flowpaths will be from the Mapacho to the mine, with very little groundwater coming from south to north into mine workings or the open pit. The direct precipitation on the open pit is not assessed in a groundwater model; the PCSWMM model simulates the monthly pit dewatering requirements from precipitation, and this is shown in Section 18.0.

Despite the complicated three-dimensional layout, a two-dimensional model is a more appropriate tool for a PEA-level water study. As a result, GRE selected a series of two-dimensional (2D) sections through the development and open pit area.

Figure 16-50 shows a plan view projection of the underground tunnels and underground development and an outline of the pit and ramps. Each section captures a different flow path condition between the workings and the Mapacho river.

- Section A-A': This section captures the block cave development at depth, and the open pit at higher elevations. This section has the Mapacho river on the 'A' side to the Northwest, and the Southeast side on the 'A' side as the river meanders around the ore zone. This section contains several different levels of block cave tunneling and development.
- Section B-B' covers the deepest part of the underground access tunnels to the block cave level. Flow is conceptualized as directly North to Soth from the river to the tunnels.
- Section C-C' covers several different crosses of the development tunnels at different elevations. Like with B-B', the flow is uniformly North to South.
- Section D-D' also covers a crossing area of tunnels and ramps. However, this section has generally higher elevations of underground workings.

In each case, the results of the 2D model are multiplied by the width of the section to create a "quasi-3D" flow environment. This method is akin to flow net calculation, and it is a viable modeling method as long as the groundwater flow direction is well-understood and the section is parallel to the primary direction of groundwater flow (Fetter, 2001).

16.3.3 Model Methods

GRE elected to use the Geostudio seepage and groundwater flow modeling software Seep/W to model the flownet. Seep/W has many advantages including the ability to directly-import cross sections from the mine development software to capture the topography, underground workings, and open pit. The Mapacho river is simulated as a constant head boundary, the open workings as "zero pressure" open space. Recharge from rainfall is not considered as part of the groundwater model. It is calculated within PCSWMM and added to the groundwater dewatering result for the calculation of total MIW. This simplification was selected because apart from "within the funnel" of the pit, recharge is expected to be insignificant to other flows.

The selected cross sections also work well with the four development timelines discussed in Section 18.5.2. Different time periods were simulated by activating or de-activating different tunnels or workings to simulate the mine plan. Appendix A shows the model cross sections.

Using the model results, GRE quantified the flow into the mine workings along each 2D section, and these inflow rates are multiplied by the width of the section to simulate the inflow into the 3D workings.

16.3.4 Model Results

The model results have been tabulated by Section and by mine time period. GRE ascertained the upper limit (more 10^{-4} rock mass) and the lower limit (10^{-5} cm/s rock mass) for the undisturbed rock outside of the broken rubble zone.

Table 16-13: Groundwater Modeling Results

Dewatering From	Lower Limit (L/s)	Upper Limit (L/s)	Notes
Early Mine Life			
Pit	0	0	See PCSWMM Model, Section 20.2
Tunnels	33		Lower limit only 3 L/s, not realistic for the length of tunnels

Dewatering From	Lower Limit (L/s)	Upper Limit (L/s)	Notes
Late Development			
Pit	0	0	See PCSWMM Model, Section 20.2
Tunnels	217	343	
Block Cave			
Pit	0	0	See PCSWMM Model, Section 20.2
Tunnels	32	32	Access tunnels to the Block Cave
Block Cave	240.4	411	Rubble Zone in Subsidence Area

GRE reported these dewatering numbers to both the engineering group (to determine dewatering costs) and to the water balance (to assess water management methods and strategies). As a conservative measure, the "Upper Limit" was selected for water management costing.

16.3.5 Closure Phase Dewatering

Considering that the open pit is a zone of enhanced groundwater recharge, it is assumed that there will be a slight groundwater mound under the pit within the block cave. This creates a net-positive gradient between the MIW from the pit/block cave to the nearby river. Because of water quality concerns, this mounting would create an unacceptable discharge of low-quality water to the Mapacho river, either through the underground workings, or fractures in the rock mass. As a result, the groundwater model upon closure assumes that the water level in the block cave/subsidence area will be actively managed to be a level slightly below the level of the Mapacho river (~2175 meters elevation). This results in a modest (3.3 liters per second [L/s]) pumping rate from the underground mine in perpetuity.

17.0 RECOVERY METHODS

17.1 Introduction

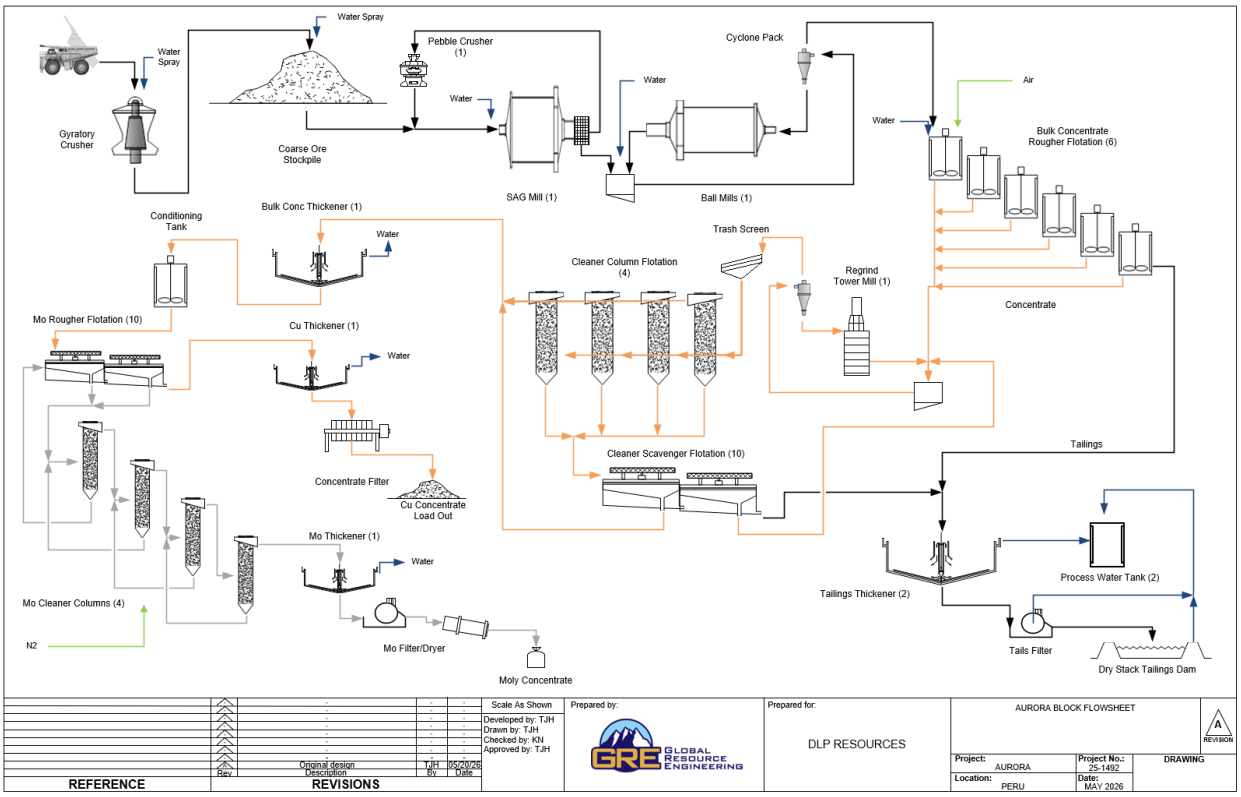
The Aurora plant is designed to treat porphyry copper ore with varying molybdenum. The main minerals of interest are chalcopyrite and molybdenite, with by-product quantities of precious metals, primarily silver. The concentrator is designed to treat 65,000 tpd with an availability of 92% for the concentrator.

The Aurora concentrator flowsheet includes:

- Crushing - primary crushing using a gyratory crusher
- Grinding – semi-autogenous mill and conventional ball mill with a pebble crusher (SABC)
- Flotation – copper and molybdenum - conventional cells for roughing and scavenging and columns for primary cleaning
- Concentrate regrinding
- Copper concentrate thickening and filtration
- Molybdenum concentrate thickening, filtration, and drying
- Tailings thickening
- Tailings filtration
- Tailings impoundments

Figure 17-1 shows a simplified process flowsheet of the Aurora concentrator.

Figure 17-1: Aurora Project Concentrator Process Flowsheet



17.2 Process Design Criteria

A process design criteria was produced for the Aurora concentrator based on the results of the test work, MetSim modeling, and in-house expertise. The target concentrator throughput is 65,000 tpd. The key elements of the process design criteria are shown in Table 17-1.

Table 17-1: Process Design Criteria

Area	Parameter	Units	Value	Reference
1.0 Site Conditions	Location	Name	Calca Peru	1
	Elevation	AMSL	3,000	1
2.0 Production Capacity	Mineable Reserve	mt	403,301,337	3
	Mine Life	yr	17.0	3
	Ore Production Rate	mtpa	23,725,000	3
		tpd	65,000	2
	Ore Grades and Recovery - Average			
	Cu Grade	%	0.222%	2
	Cu Recovery – To Cu Conc	%	85.5%	4
	Mo Grade	%	0.089%	2
	Mo Recovery – To Mo Conc	%	88.5%	4
	Ag Grade	g/t	2.37	2
	Ag Recovery – To Cu Conc	%	74.0%	4
	Metal Production - Average			
	Cu	tpa	45,093	3
	Mo	tpa	18,708	3
	Ag	oz/y	1,337,756	3

Area	Parameter	Units	Value	Reference
3.0 Operating Schedules	Operating Time			
	Day Per Year	num	365	5
	Hours Per Day	num	24	5
	Availability	%	100.0%	3
4.0 Ore Characteristics	Ore			
	Moisture	%	3%	1
	Specific Gravity		2.65	4
	Crusher Work Index (CWi) – Average	KWh/t	12.93	5
	Abrasion Index	Ai	0.3	5
	Crusher Liner Wear	kg/KWhr	0.021	5
	SAG Mill Specific Energy - Composites			
	Composito Cu	KWh/t	9.88	4
	Composito EN	KWh/t	9.12	4
	Composito HF	KWh/t	12.27	4
	Composito Mo	KWh/t	9.87	4
	Ball Mill Work Index - Composites			
	Composito Cu	KWh/t	11.71	4
	Composito EN	KWh/t	12.08	4
	Composito HF	KWh/t	12.31	4
	Composito Mo	KWh/t	10.53	4
	Regrind Ball Mill Work Index	KWh/t	20.0	5
	Concentrate Density - Typical			
	Cu Final Conc SG	t/m^3	3.2	5
	MoS2 Final Conc SG	t/m^3	3.1	5
	Rougher Conc SG	t/m^3	2.9	5
	Cleaner Conv Conc SG	t/m^3	3.0	5
	Cleaner Scav Conc SG	t/m^3	2.9	5
5.0 Crushing	Gyratory Crusher Availability	days	365	3
		hrs	20	3
	Operating Time	%	83.3%	3
	Crusher Feed	tpa	23,725,000	3
		tpd	65,000	3
	Design	tph	3,250	3
	Nominal	tph	2,708	3
	P80	mm	150.0	2
6.0 Grinding	Mill Availability	days	365	1
		hrs	22.0	1
	Operating Time	%	91.7%	1
	Primary SAG Mill			
	Design	tpd	65,000	1
	Design	tph	2,954.5	3
	Nominal	tph	2,708.3	3
	F80	um	150,000	2
	P80	um	12,700	2
	Moisture	%	3%	5
	W_a - ave	kwh/t	9.5	3

Area	Parameter	Units	Value	Reference
	Number of Mills	num	1	2
	SAG Mill Sizing			
	Specific Energy	Kwh/t	9.5	3
	Required Power at Mill	KW	28,159.9	3
	Efficiency	%	95%	2
	Net KW/Mill	kw/mill	29,642	3
		hp/mill	39,735	3
	Primary Ball Mill			
	Design	tph	2,954.5	1
	Nominal	tph	2,708.3	1
	F80	um	12,700	3
	P80	um	150	1
	Moisture	%	60%	3
	Wb	kwh/t	4.0	3
	Number of Mills		1	2
	Ball Mill Sizing			
	Specific Energy	Kwh/t	4.0	3
	Efficiency	%	95%	2
	Required Power at Mill	KW	12,578	3
	Net KW/Mill	kw/mill	12,578	3
		hp/mill	16,861	3
	Cyclone Overflow	%S	35%	2
	Cyclones			
	Number	#	14.0	3
	Dia	in	33.0	3
	Feed Solids	%	75.0	2
	Circulating Load	%	300.0	2
	Cyclone feed per cyclone	tph	844	3
	Volume per cyclone	m^3/hr	600	3
	Pressure	psi	13.3	3
	Pebble Crusher			
	Feed	%	25%	2
	Design	tph	738.6	3
	Nominal	tph	677.1	3
	Crusher Work Index Wb	kwh/t	1.3	3
	Number of Crushers	num	1.0	2
	Crusher Power	KW	926	3
		kw/crusher	926	3
		hp/crusher	1,242	3
7.0 Bulk Flotation	**Rougher Flotation Feed**			
	Solids - Design	tph	2,954.5	3
	Percent Solids	%	35%	3
	Rougher			
	Retention Time	min	16.0	3
	Banks	num	1	2
	Cells Per Bank	num	6	2
	Cell Type		500 m^3 Tank	2
	Rougher Stage Recovery			

Area	Parameter	Units	Value	Reference
	Cu Recovery	%	86%	4
	MoS2 Recovery	%	92%	4
	Ag Recovery	%	76%	4
	Regrind Mill			
	Design	tph	149.0	3
	Nominal	tph	136.6	3
	F80	um	125	3
	P80	um	45	3
	Moisture	%	65%	2
	BWi	kwh/t	20.0	2
	Number of Mills		1	2
	Ball Mill Sizing			
	Specific Energy	kwh	11.9	3
	Required Power at Mill	KW	1,776.8	3
	Net KW/Mill	KW/Mill	1,777	3
	Regrind Cyclones			
	Number	#	10.0	2
	Dia	in	20.0	2
	Cleaner Columns			
	Retention Time	min	12.0	4
	Banks	num	1.0	2
	Cells Per Bank	num	4.0	2
	Cell Type		Columns	2
	Cell Dia	m	2.5	2
	Cell Height	m	16.0	2
	Cell Volume	m^3	78.5	3
	Column Mass Pull	%	15.0%	5
	Cleaner Column Tails			
	Tails	tph	126.6	3
	Solids	%	17%	3
	Mechanical Cleaner Feed			
	Feed	tph	126.6	3
	Solids	%	17%	3
	Cleaner Conventional			
	Retention Time	min	10.0	4
	Banks	num	1.0	2
	Cells Per Bank	num	10.0	2
	Volume Per Cell	m^3	10.81	3
	Cell Type		Denver	2
	Cleaner Conv Mass Pull	%	0.5%	4
	Concentrate	tph	0.6	3
	Solids	%	15%	5
	Cleaner Conv Tails			
	Tails	tph	126.0	3
	Solids	%	17%	3
	Cleaner Scavenger			
	Retention Time	min	12.0	4
	Banks	num	1	2

Area	Parameter	Units	Value	Reference
	Cells Per Bank	num	10	2
	Volume Per Cell	m^3	12.90	3
	Cell Type		Denver	2
	Mass Pull	%	1.0%	4
	Concentrate	tph	1.3	3
	Solids	%	15%	5
	Cleaner Scavenger Tails			
	Tails	tph	124.7	3
	Solids	%	17%	3
8.0 Bulk Concentrate	Concentrate Thickener			
	Feed	tph	23.0	3
	Solids	%	15.0%	2
	Area	m^2/tpd	1.0	5
		m^2	551.6	3
	Dia	m	27.0	3
	U/F Solids	%	60%	5
9.0 Molybdenum Flotation	**Rougher Flotation Feed**			
	Solids	tph	23.5	3
	Percent Solids	%	37%	2
	Conditioning Tank			
	Retention Time	min	10.0	4
	Flow	m^3/hr	70.8	3
	Tank			
	Volume Required	m^3	11.8	3
	Volume Tank	m^3	13.9	3
	Rougher			
	Retention Time	min	30.0	4
	Banks	num	1	2
	Cells Per Bank	num	10	2
	Volume Per Cell	m^3	3.54	3
	Cell Type		Denver	2
	Mass Pull	%	22.0%	1
	Concentrate	tph	5.2	3
	Rougher Stage Recovery			
	Cu Recovery	%	1%	4
	MoS2 Recovery	%	96%	4
	Ag Recovery	%	30%	4
	Cleaner Columns 1			
	Retention Time	min	12.0	3
	Banks	num	1.0	2
	Cells Per Bank	num	1.0	2
	Volume Per Cell	m^3	7.14	3
	Cell Type		Columns	2
	Column Mass Pull	%	75.0%	5
	Concentrate	tph	4.7	3
	Cleaner Column Tails 1			
	Tails	tph	0.5	3
	Solids	%	4%	3

Area	Parameter	Units	Value	Reference
	Cleaner Columns 2			
	Retention Time	min	12.0	4
	Banks	num	1.0	2
	Cells Per Bank	num	1.0	2
	Volume Per Cell	m^3	2.93	3
	Cell Type		Columns	2
	Column Mass Pull	%	77.0%	5
	Concentrate	tph	3.6	3
	Cleaner Column Tails 2			
	Tails	tph	1.1	3
	Solids	%	20%	3
	Cleaner Columns 3			
	Retention Time	min	12.0	4
	Banks	num	1.0	2
	Cells Per Bank	num	1.0	2
	Volume Per Cell	m^3	1.94	3
	Cell Type		Columns	2
	Column Mass Pull	%	75.0%	5
	Concentrate	tph	2.7	3
	Cleaner Column Tails 3			
	Tails	tph	0.9	3
	Solids	%	20%	3
	Cleaner Columns 4			
	Retention Time	min	12.0	4
	Banks	num	1.0	2
	Cells Per Bank	num	1.0	2
	Volume Per Cell	m^3	2.34	3
	Cell Type		Columns	
	Column Mass Pull	%	75.0%	5
	Concentrate	tph	2.0	3
	Cleaner Column Tails 4			
	Tails	tph	0.7	3
	Solids	%	13%	3
10.0 Cu Concentrate	Concentrate Thickener			
	Feed	tph	18.3	3
	Solids	%	32.4%	2
	Area	m^2/tpd	1.0	5
		m^2	439.3	3
	Dia	m	24.0	3
	U/F – Solids	%	60%	5
	Concentrate Filter			
	Feed	tph	23.0	3
	Solids	%	60%	5
	Filter Area	m^2	51.1	3
11.0 Mo Concentrate	Concentrate Thickener			
	Feed	tph	2.0	3
	Solids	%	25.0%	2
	Area	m^2/tpd	1.0	5

Area	Parameter	Units	Value	Reference
		m^2	48.6	3
	Dia	m	8.0	3
	U/F – Solids	%	60%	5
	Concentrate Filter			
	Feed	tph	2.0	3
	Solids	%	60%	5
	Filter Area	m^2	4.5	3
12.0 Tailings	Tailings Thickener			
	Number of Thickeners Operating	num	2	2
	Solids	tph	1,465.8	3
	Solids	%	37.6%	3
	Area	m^2/tpd	0.224	5
		m^2	7,880.0	3
	Dia	m	100.0	3
	U/F – Solids	%	60%	5
	Tailings Filter			
	Feed	tph	2,747	3
	Solids	%	60%	5
	Filter Area/filter	m^2	1,314	3
	Filter Cake moisture	%	14.6	3

17.3 Process Description

The following are the descriptions of the primary unit processes in the Aurora concentrator. The process is typical of most copper/molybdenum porphyry concentrators utilizing primary crushing followed by a SABC comminution circuit, bulk concentrate flotation, bulk concentrate cleaner flotation and molybdenum separation flotation. The plant is designed to treat 65,000 metric tpd.

17.3.1 Primary Crushing

Run-of-Mine (ROM) ore is delivered by haul truck to a 60-inch × 89-inch gyratory primary crusher. The trucks discharge directly into the crusher, which is set in a dump pocket allowing two truck access. Alternatively, trucks can discharge to a local static stockpile for blending purposes. The crushed ore is withdrawn from a surge pocket under the crusher by an apron feeder, which discharges onto the primary conveyor. The primary conveyor is a rope design and the feed passes over a tramp metal magnet.

The primary conveyor transports the primary crushed ore to the coarse ore stockpile, which has a nominal live capacity of 35,500 tonnes. The P_{80} of the primary crushed ore is approximately 150 mm. The stockpile is equipped with a dome covering to control dust. Ore is extracted from the stockpile through a series of vibrating feeders under the pile.

17.3.2 Comminution Circuit

The comminution circuit is typical for this type of deposit consisting of a primary SABC, a primary ball mill, and a pebble crusher. The material withdrawn from the stockpile is conveyed to the semi-autogenous grinding (SAG) mill feed at a design rate of 2,955 tonnes per hour (tph). Water is added to create a slurry of ~70% solids. The comminution circuit design was based on SMC test work and review by Alex Doll

Consulting Limited (Alex G Doll Consulting Limited, 2026). The SAG mill would require 29.6 megawatts (MW) with a mill dimension of 12.2 m diameter and a length of 6.8 m.

The SAG mill discharge passes across a trommel screen, where course particles are removed and diverted to a pebble crusher. An estimated 25% of the discharge reports to the crusher at a designed rate of 730 tph. The closed side setting of the crusher is set at 12 mm and the discharge reports back to the SAG feed conveyor. The crusher is an 84-inch shorthead cone crusher with a 740 kilowatt (KW) motor.

The combined slurry from the SAG mill and ball mill reports to a common sump feeding a cyclone pack equipped with 14 x 33-inch cyclones. The cyclone underflow reports to the ball mil feed. The ball mill is sized based on a discharge product size of 150 µm and a 300% circulating load. The ball mill requires 12.6 MW and is 7.9 m in diameter and 12.2 m long. The cyclone overflow, at 2,955 tph and 35% solids, reports to the rougher flotation circuit.

17.3.3 Flotation Circuit

The flotation circuit is designed as a bulk copper and molybdenum flotation process. Subsequent differential flotation is designed to produce the final individual copper and molybdenum concentrates. The rougher flotation circuit is operated to maximize recovery of the primary sulfide minerals from the gangue. Subsequent cleaner flotation of the bulk concentrate is operated to maximize the copper concentrate grade while minimizing copper recovery losses.

The flotation reagents used include xanthate, dithiophosphate, diesel (for molybdenum recovery), and a frother. The majority of the flotation reagents are added to the ball mill feed with trim reagents added down the flotation bank where required.

Regrinding of the rougher concentrate is required to provide increased mineral liberation to allow cleaner flotation to produce high concentrate grades. The molybdenum is co-recovered to the bulk cleaner concentrate with the copper. The molybdenum flotation circuit provides the separation of the copper and molybdenum into respective salable concentrates.

17.3.3.1 Rougher Flotation

The rougher flotation circuit design consists of six (6) x 500 m^3 Tank cells configured as a single bank. The frother is added to the head of the rougher flotation cells, with supplemental reagents added as required down the bank. The rougher section is operated in open circuit, with the rougher tailings reporting directly to the final tailings. The addition of a Jameson-type cell could improve the course material recovery.

17.3.3.2 Regrind

The rougher concentrate is delivered to the regrind circuit. The regrind circuit consists of a tower mill equipped with 1.8 MW motor. The regrind circuit is a reverse configuration common for regrinding. The rougher concentrate is combined with the regrind mill discharge and pumped to a cyclone pack consisting of ten (10) x 14-inch cyclones The cyclone overflow passes over a trash screen before reporting to the cleaning circuit and the cyclone underflow reports to the regrind mill feed. The target product size (P_{80}) is 45 µm.

17.3.3.3 Cleaner Flotation

The cleaner circuit consists of four 2.5-m diameter by 16.0-m tall column flotation cells operated in parallel. The bulk copper-molybdenum concentrate is 22% to 29% Cu and 5% to 11% Mo (depending on the material blend). The column cell tails are sent to the cleaner-scavenger flotation bank. The cleaner scavenger bank comprises twenty (20) x 11 m^3 Denver-style flotation cells. The first bank of cells produces a final concentrate, and scavenger concentrate from the remaining cells is recirculated to the regrind circuit feed. The tails of the cleaner-scavenger bank report to final tailings.

17.3.3.4 Molybdenum Circuit

The bulk copper-molybdenum concentrate from the cleaner circuit is thickened and conditioned before reporting to the molybdenum Circuit. NaHS is added to the slurry to provide depression of copper and iron sulfides using redox control. Additional diesel is added as a molybdenum promoter. Nitrogen gas is utilized as the mineral carrier.

The rougher comprises of one bank of conventional Denver-style cells containing ten (10) x 3.5 m^3 cells. The cleaning stage consists of four stages of cleaning using column cells in series. The tailings from each column are fed back to the previous flotation stage in the circuit. The columns range in size based on the feed load and decrease in volume down the bank from 7 m^3 to 2.3 m^3. The target minimum final concentrate grades are 22% for copper and 60% for molybdenum.

17.3.4 Concentrate Dewatering

The molybdenum rougher tailing becomes the final copper concentrate reporting to a single 24-m diameter copper concentrate thickener. The final molybdenum product is thickened in an 8-m diameter molybdenum thickener.

The final copper concentrate is thickened to a target 60% solids and flows from the copper thickeners a concentrate stock tank. The slurry is pumped from the storage tank to the filter plant. The concentrate is filtered in a plate and frame pressure filter targeting 10% moisture and conveyed to the copper concentrate storage shed for loadout.

The molybdenum concentrate is filtered in a plate and frame pressure filter targeting 10% moisture, then dried in a rotary dryer and bagged for shipment.

17.3.5 Tails Thickening

Tailings from the rougher bank and the cleaner-scavenger bank are combined and feed two x 100-m diameter tailings thickeners. The overflow water is reclaimed and sent to the process water tanks. The thickened tails are pumped to a filter plant. Details of tailings pumping and storage are described in Section 18.0.

17.3.6 Process Water

Water for the process is primarily based on recycled streams originating from thickener overflows and filter plant filtrate. Makeup water will be derived from other site sources including pit/underground water, and surface capture. The site is designed as a closed water system.

17.3.7 Filter Plant

Thickened tailings will be pumped from the two thickener underflows to the tailings filter feed tanks prior to being pumped to sixteen (16) plate and frame filters for further dewatering. Filtered solids will be conveyed as filter cake to the dry stack storage facility. Filtrate will be recycled back to the tailings thickener and used as process water.

17.4 Risks

Key risks to the operation of the Aurora concentrator include:

- There are four metallurgical domains with varying grades for both copper and molybdenum. Copper circuits can generally respond reasonably well to changing grades, but molybdenum can be more challenging. Care needs to be taken to consider the changing grades across the mine life in the flotation circuit design.
- Process water recycle can negatively impact the flotation performance and it may be necessary to keep the molybdenum process water separate from the bulk flotation circuit because of the presence of NaHS.
- The SAG and ball mill put forward in this design are large and limited experience exists in the industry managing and maintaining mills of this size.
- Minimal test work has been conducted on solid/liquid separation and thus the thickener and filter sizing may need modification.

17.5 Opportunities

Several opportunities may exist to improve the Aurora process design:

- Optimizing the primary grind size may allow for improved copper and molybdenum recovery. This could also involve the use of alternative flotation technologies like the Jameson or Nova cells.
- Optimizing the silver that reports to the copper concentrate will improve the silver payability.
- The use of HPGR instead of a SAG mill could reduce the overall power consumption in the comminution circuit.

17.6 Recommendations

The Aurora concentrator is typical of most copper/molybdenum porphyry plants. The optimization of the process requires additional test work and value engineering:

- Review the comminution circuit design and optimize based on capital and operating costs.
- Consider alternative flotation reagents for molybdenum flotation to reduce the risk of using NaHS.
- Examine alternative flotation technologies to improve the plant performance.
- Consider alternative tailings designs to avoid the capital and operating costs of drystack tailings.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

Infrastructure for the Aurora project will include on-site and off-site infrastructure such as civil working, buildings and facilities, water and power supply systems and other miscellaneous systems that support operations. The site infrastructure will include:

- Mine facilities including truck shop, explosive storage and other miscellaneous facilities
- Process facilities including the process plant, crusher facilities, filter plant, metallurgical lab, plant workshop and warehouse
- DSTF
- Administrative and mine offices and a permanent camp

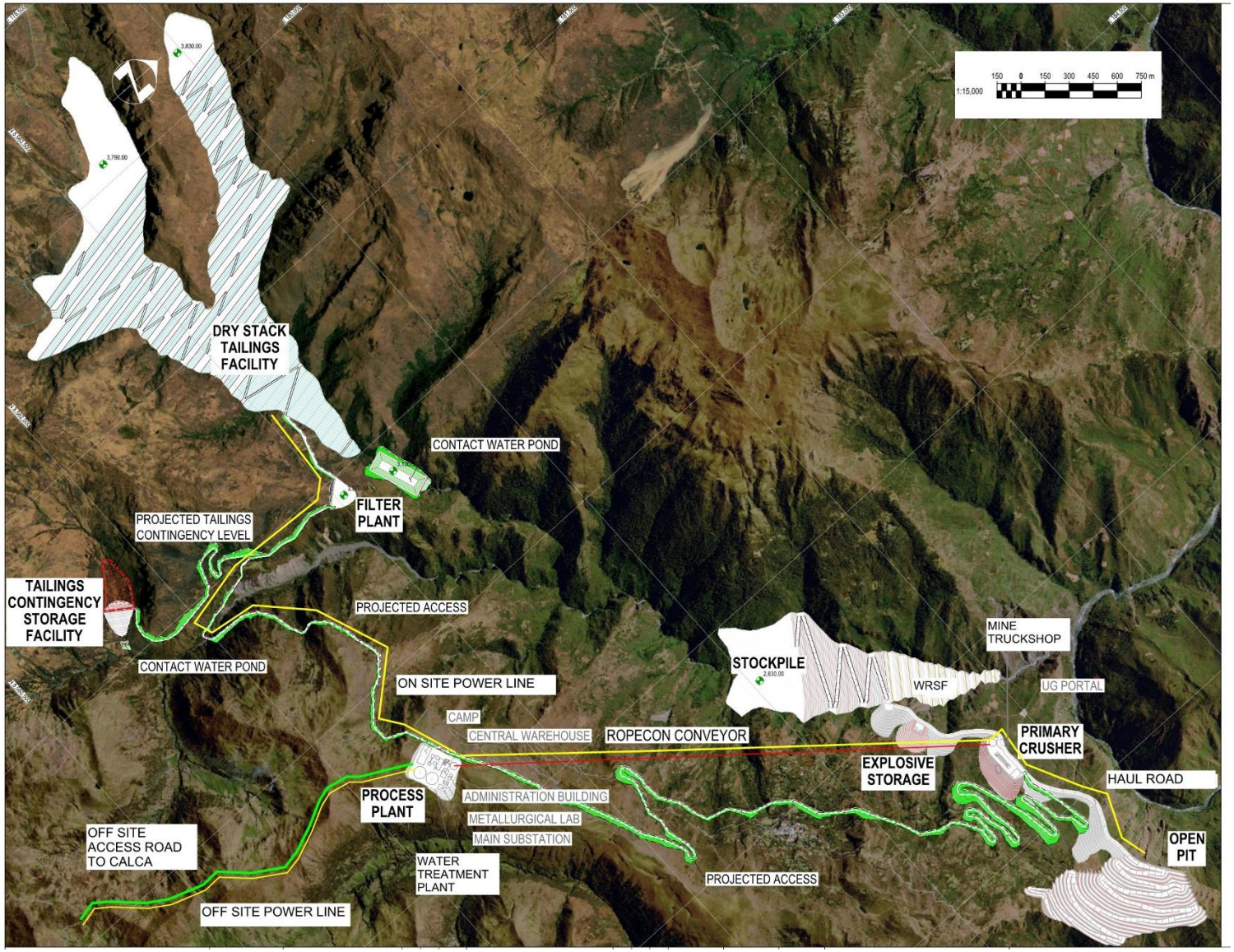
Mine and process administration facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication and sanitary systems.

Site selection was performed based on the following inputs:

- Selecting a site within the Aurora claim boundary while maximizing the use of previously disturbed land
- Locating the primary crushing and ROM pad sites near both the open pit and UG portal to minimize haulage distance
- Balancing earthwork cut and fill across the process plant, access roads, primary crusher pad and filter plant
- Locating the filter plant and DSTF in the most economic location and balancing costs for dry stack tailings handling
- Using natural changes in elevation to the greatest extent possible.

The Aurora site layout is shown in Figure 18-1.

Figure 18-1: Aurora Project Site Layout



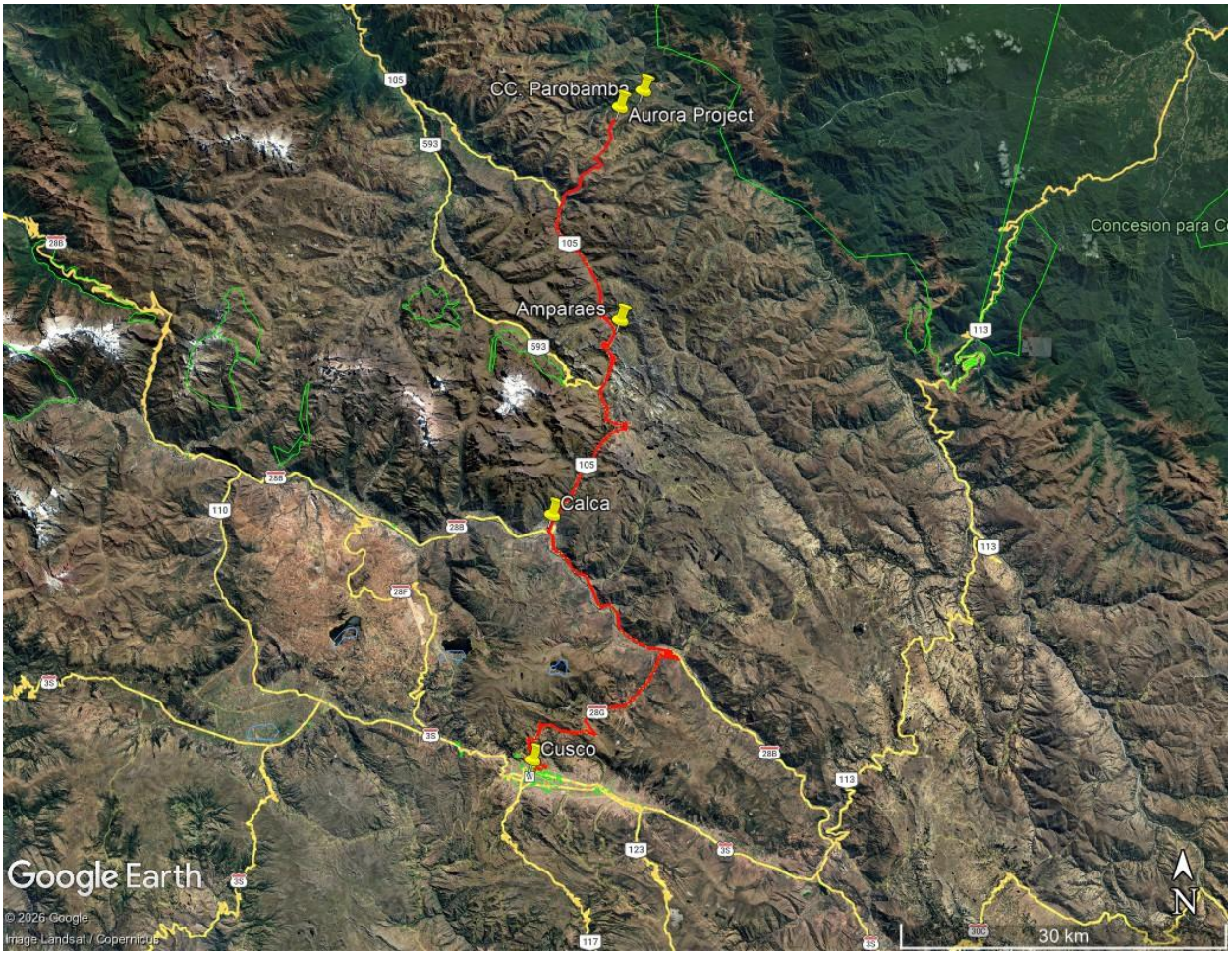
Source: Ausenco, 2026

18.2 Off Site Infrastructure

18.2.1 Site Access

The Aurora project is located approximately 120 km from the city of Cusco and 70 km from the town of Calca (Figure 18-2). Access from the city of Lima to the concession area is by air, from Lima to Cusco followed by land through the towns of Calca, Amparaes, and Parobamba. The total time required from Lima to Cusco is one hour by air and then an additional four and a half hours by road.

Figure 18-2: Road Route from Cusco to the Aurora Project.



Source: Ausenco, 2026

18.2.2 High Voltage Power Supply

Permanent electrical power is provided by a transmission line that will connect to the electrical grid in Calca. This assumes the right-of-way for transmission lines and estimated alignment and design. The 138-kV transmission line will connect to a substation at the site before being stepped down to 25 kV for distribution to different power requirements across the project site.

18.3 On-Site Infrastructure

18.3.1 Site Preparation

All infrastructure areas will be cleared of vegetation, and the topsoil will be removed before construction. Drains, safety bunds and backfilling with granular material and aggregates for road construction are all elements of the initial site development.

Site civil work includes design for the following infrastructure:

- Roads for light vehicles and heavy equipment
- Access roads
- Primary crusher and mine facility pads
- Process plant pad
- Explosives storage pad
- Filter plant pad
- DSTF area
- Tailings contingency storage facility
- Water management facilities, ditches, and drainage channels

18.3.2 On-Site Roads

The project site has unpaved roads connecting the main road to the process plant. New roads will be constructed from the process plant to the primary crusher pad, UG portal, filter plant, DSTF, and tailings contingency storage facility.

18.3.3 Accommodation

Permanent accommodations on site will be in a camp of 1,070 beds. The camp will be a modular building design with multiple levels and will include a kitchen, dining area, and recreation room. The camp will be built to accommodate workers for the construction phase of the project and will be converted thereafter for use by operations personnel.

18.3.4 Rope Conveyor

A four (4) km rope conveyor system is planned to transport primary crushed material from the primary crusher pad to the process plant (Figure 18-1).

18.3.5 Power and Electrical

The HV transmission line from the grid will be connected by distribution line to a 138kV/25kV substation on site. The substation will distribute power to various areas of the project, including the process plant, primary crusher, mining areas, filter plant, DSTF, and tailings contingency facility.

18.4 Tailings Facility

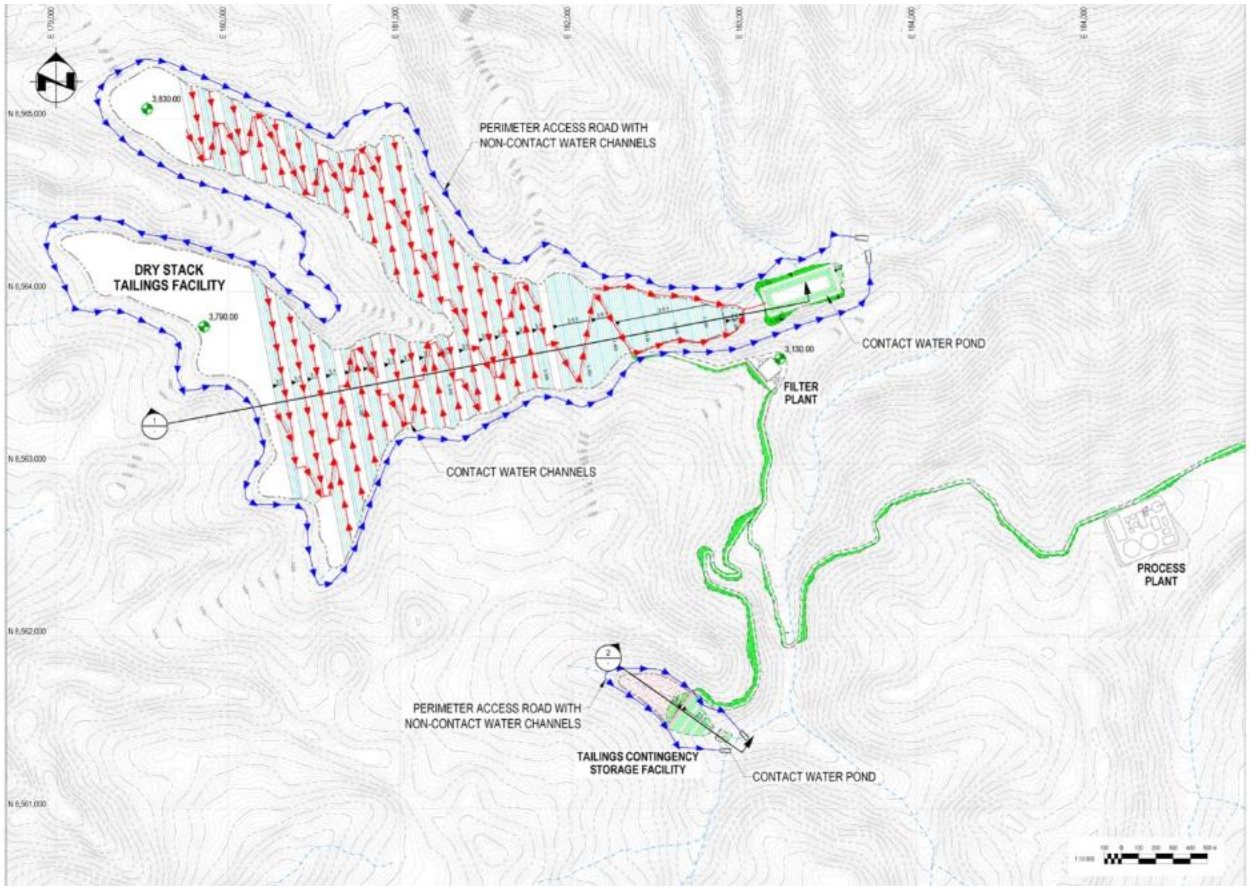
As part of the PEA, a siting and deposition technology study was conducted. Due to the steep, rugged terrain, slurry tailings storage facilities had a very low tailings-to-dam ratio; therefore, filtered tailings

were the preferred option for the study. The Dry Stack Tailings Facility (DSTF) site is located directly above the Waste Rock Storage Facility (WRSF). As part of the DSTF, a second, very small slurry-contingency tailings storage facility was designed to accept out-of-specification tailings; for example, if the filter plant is shut down for a short period slurry, tailings would be pumped to this facility.

18.4.1.1 Dry Stack Tailings Facility

Tailings from mining activities will be placed in the DSTF, located 7 km southwest of the open pit and underground portal. The DSTF will be developed as a valley fill within the same watershed as the open pit and underground portal. The filtered tailings will be stacked using a conveyor system and radial stacker, with dozers and a compactor to spread the tailings in thin lifts. The facility is designed to provide secure, permanent storage of 393.9 Mt of tailings for a 64.4 ktpd mining operation over a 19-year life-of-mine (see Figure 18-3).

Figure 18-3: Aurora Project Dry Stack Tailings Facility



Source: Ausenco, 2026

18.4.1.2 Design Basis

Both Peruvian and international guidelines were considered in the design of the DSTF, including the Canadian Dam Association (CDA) Application of Dam Safety Guidelines to Mining Dams (2014), which is regarded as one of the industry's best practices for tailings management.

In addition, Table 18-1 identifies the guidelines, codes, and standards applicable to the facility and regional requirements.

Table 18-1: DSTF Guidelines, Codes, and Standards

Autor	Year	Document
CDA	2013	Dam safety guidelines (2013 revision)
CDA	2014	Technical Bulletin: Application of CDA Dam Safety Guidelines to Mining Dams
CDA	2019	Application of Dam Safety Guidelines to Mining Dams
ICMM/UNEP/PRI	2020	Global Industry Standard on Tailings Management
Hawley, M., & Cunning, J.	2017	Guidelines for Mine Waste Dump and Stockpile Design
Ministerio de Energía y Minas (MEM)	1997	Guía Ambiental para la Estabilidad de Taludes de Depósitos de Desechos Sólidos de Mina. Dirección General de Asuntos Ambientales
Ministerio de Energía y Minas (MEM)	2023	Decreto Supremo N.° 034-2023-EM – Modificaciones al Reglamento de Seguridad y Salud Ocupacional en Minería. Anexo N40

The design criteria are shown in Table 18-2.

Table 18-2: Design Criteria for DSTF

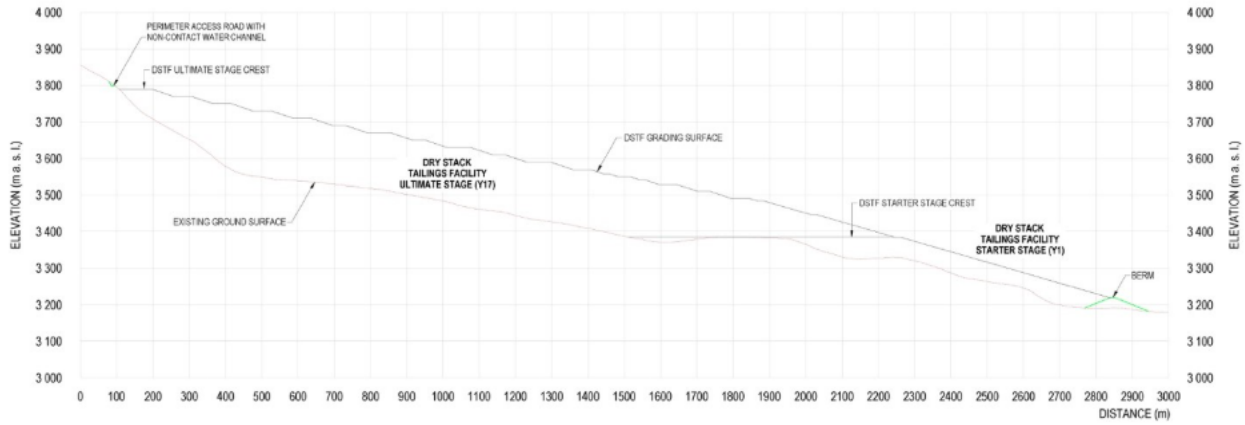
Design Criteria	Unit	Criteria
Storage Capacity Tailings	Mt	393.9
Area	Ha	347
Dry Stack Tailings Facility		
Dry Density	t/m^3	1.65
Storage Capacity	Mm^3	238.7
Project Design Life	years	19
Production Schedule	t/d	64,355
Dam Hazard Classification	unitless	Very High
Overall Slope	X:1	3.5
Short Term - Static Factor of Safety	-	1.3
Long Term - Factor of Safety	-	1.5
Pseudo-Static Factor of Safety	-	1.0
Design Seismic Event	years	5,000
Seismic Coefficient	-	0.16
Non-Contact Water Diversion Channel - Design Storm Event	years	100
Contact Water Diversion Channel – Design Storm Event	years	30
Contact Water Pond Capacity -Design Storm Event	years	100

The facility will be developed as a valley-fill structure, using conveyors and a radial stacker, dozers, and compactors to deposit, spread, and compact filtered tailings generated by mining operations and ore processing. The DSTF will be built using bottom-up construction. Clearing and grubbing and topsoil removal will be carried out in stages, ahead of stacking operations on the existing terrain. The filtered tailings will be stacked in 1-m lifts. In its final configuration, the facility will cover approximately 347 ha and store approximately 393.9 Mt of tailings (Figure 18-4).

The main design features of the DSTF are summarized below:

- The DSTF will be constructed in multiple phases to reduce capital cost and place development costs into sustaining capital.
- The permanent exterior faces for DSTF lifts are 2:1 (H:V), with an overall slope of 3.5:1 (H:V).
- DSTF will have an underdrain that discharges into the contact water pond located below the facility. The underdrains will collect infiltration through the tailings, as well as any recharge located below these facilities.
- Top surfaces of the filtered tailings will be compacted to reduce infiltration and provide a trafficable surface.
- A non-contact diversion channel will be constructed to capture and convey runoff upstream of the facility.
- Once a lift's permanent slope is completed, it will be reclaimed and closed with waste rock, topsoil, and vegetative cover to minimize sediment generation.

Figure 18-4: Aurora Project Dry Stack Tailings Facility



Source: Ausenco, 2026

18.4.1.3 Seismic Hazard Assessment

Ausenco did not conduct a site-specific seismic hazard assessment for the PEA but instead relied on local developed data. The seismic event used for the stability analysis was the 1:5,000 year return period, based on the CDA guidelines. The seismic coefficient adopted was 0.16.

18.4.1.4 Stability Analysis

As part of the DSTF design, Ausenco conducted stability and deformation analyses using Slide2v9.03 (Rocscience). A single section was analyzed within the DSTF area. Factors of safety were calculated using the following methods:

- Limit Equilibrium Method (LEM)
- Spencer Methodology

The stability analysis results showed that the safety factors were greater than 1.5 under static conditions and greater than 1 under pseudo-static conditions for each section (see Table 18-3). Therefore, the DSTF is stable under static and pseudo-static conditions for operations and post-closure.

Table 18-3: Aurora Project DSTF Stability Analysis Results

Structure	Section	Stage	Failure surface (surface type)	Factors of Safety (FoS)		
				Static Condition	Pseudo-static condition (kh= 0.16)	Post-earthquake condition
Section A						
DSTF	1-1	Final design	Global failure (circular)	2.15	1.31	1.97

18.4.1.5 Instrumentation and Monitoring

The stability of the DSTF will be monitored using vibrating-wire piezometers, inclinometers, and GPS/Global Navigation Satellite System (GNSS) sensors strategically placed within the foundation and the co-placement facility. The piezometers will monitor pore pressure within the foundation and the co-placement facility; the inclinometers will monitor the critical failure plane within the facility and the foundation; and the GPS/ GNSS sensors will monitor any movement of the pile.

These instruments will be connected to a data logging and telemetry system for continuous, real-time, unattended monitoring. Additionally, water flow and level instrumentation will be installed to monitor surface-water flows in the water collection ponds on the facility's southern side.

18.4.1.6 Closure Concept

The objective of closure for the DSTF is to ensure the long-term stability of its physical, geochemical, hydrological, and revegetation conditions. To achieve this, the closure strategy emphasizes progressive reclamation wherever practical, enabling early mitigation of project impacts, minimizing environmental exposure associated with sediment management, and reducing the scope of final closure activities. The strategy also includes implementing a robust environmental closure design aligned with best practices in the extractive industries to minimize environmental risks. Key goals include addressing and reducing post-closure public safety concerns and protecting downstream groundwater and surface water quality.

Additionally, the closure strategy aims to maintain the long-term stability and integrity of the DSTF while integrating it into the surrounding landscape. This involves restoring the site's natural appearance and supporting a return to its pre-mining land use and productivity, ensuring the area can sustain ecological functions and meet long-term land-use goals.

18.4.1.7 Construction and Operations

The DSTF, including the development phase, consists of multiple lifts. It will be developed as a valley-fill structure using mobile equipment to stack, deposit, spread, and compact mine waste materials.

18.4.1.8 Surface Water Management

The surface water management will include four components:

- Non-contact water for the DSTF in open diversion channels that run around the perimeter of the facility.

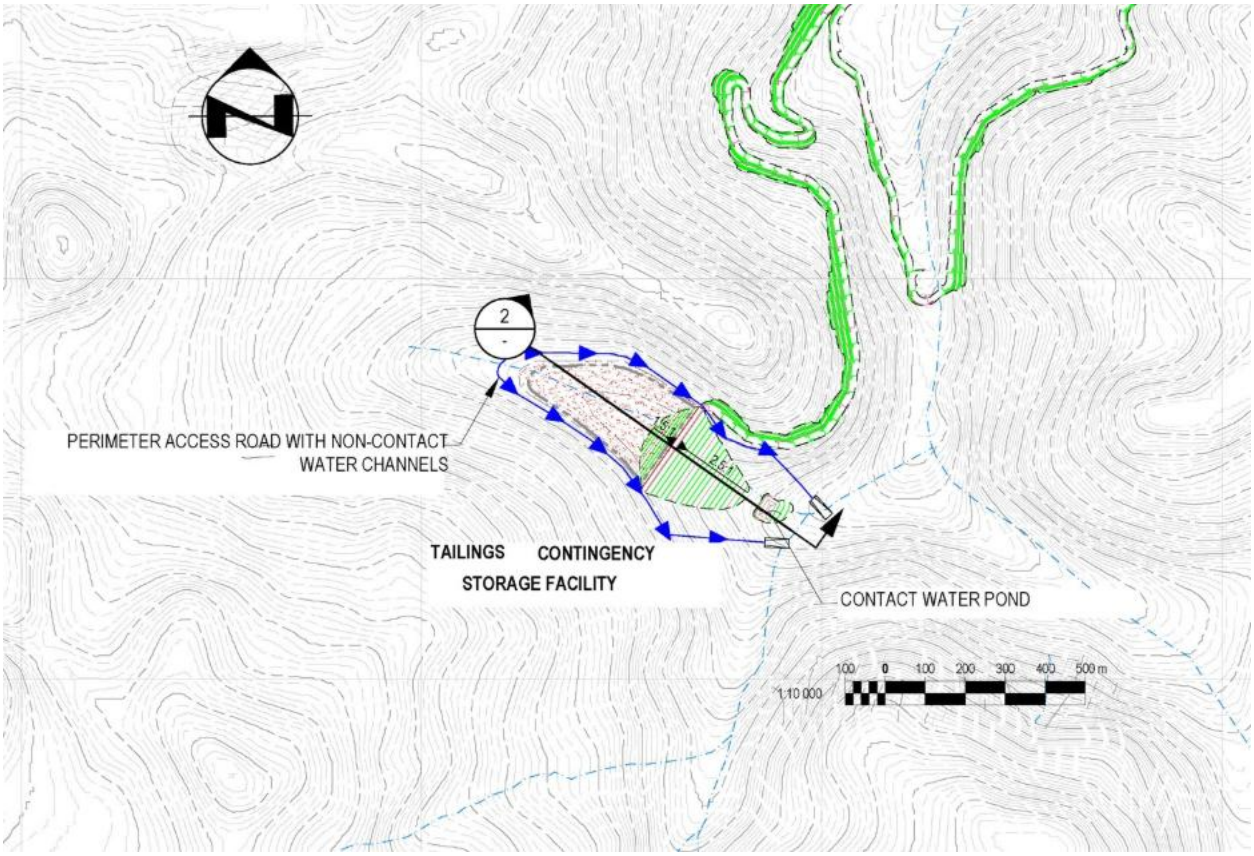
- Contact water for the DSTF in open diversion channels that run on the surface of the facility and discharge into the contact water pond.
- Non-surface water at the foundation contact with the DSTF will be captured by the facility's underdrain system and conveyed to the contact water pond.
- The non-contact surface water management for the co-placement includes permanent diversion channels designed to safely direct runoff around and away from the facility. The non-contact surface water diversion channels will be constructed in a single phase and will convey the 100-year storm event around the facility. The contact water channels within the DSTF have been designed to convey up to the 30-year storm event to the contact water pond.

Water captured in the contact water ponds will be pumped to the process plant for use as process water and treated for fresh water supply. Any surplus water will be treated and released to the environment.

18.4.1.9 Tailings Contingent Facility

As part of tailings storage management, a Tailings Contingency Storage Facility (TCSF) will be constructed to provide temporary storage for out-of-specification tailings; for example, when the filter plant shuts down or partially shuts down due to an emergency then slurry tailings will be pumped to this facility. The TCSF is located 2 km south of the DSTF. The TCSF will be developed as a valley fill with an embankment, allowing tailings to be pumped as a slurry into the facility. The TCSF will include a water reclaim system. The TCSF can temporarily store up to 1 Mm^3 of slurry tailings during the 18-year life-of-mine (see Figure 18-5).

Figure 18-5: Aurora Project Tailings Contingency Storage Facility



Source: Ausenco, 2026

18.4.1.10 Design Basis

Both Peruvian and international guidelines were considered in the design of the TCSF, including the CDA Application of Dam Safety Guidelines to Mining Dams (2014), which is regarded as one of the industry's best practices for tailings management.

The design criteria are shown in Table 18-4.

Table 18-4: Aurora Project Design Criteria for TCSF

Design Criteria	Unit	Criteria
Storage Capacity Tailings	Mt	1.35
Area	Ha	12
Dry Stack Tailings Facility		
Dry Density	t/m^3	1.35
Storage Capacity	Mm^3	1.0
Project Design Life	years	18
Dam Hazard Classification	unitless	Significant
Overall Slope	X:1	3.5
Short Term - Static Factor of Safety	-	1.3
Long Term - Factor of Safety	-	1.5
Pseudo-Static Factor of Safety	-	1.0
Design Seismic Event	years	500

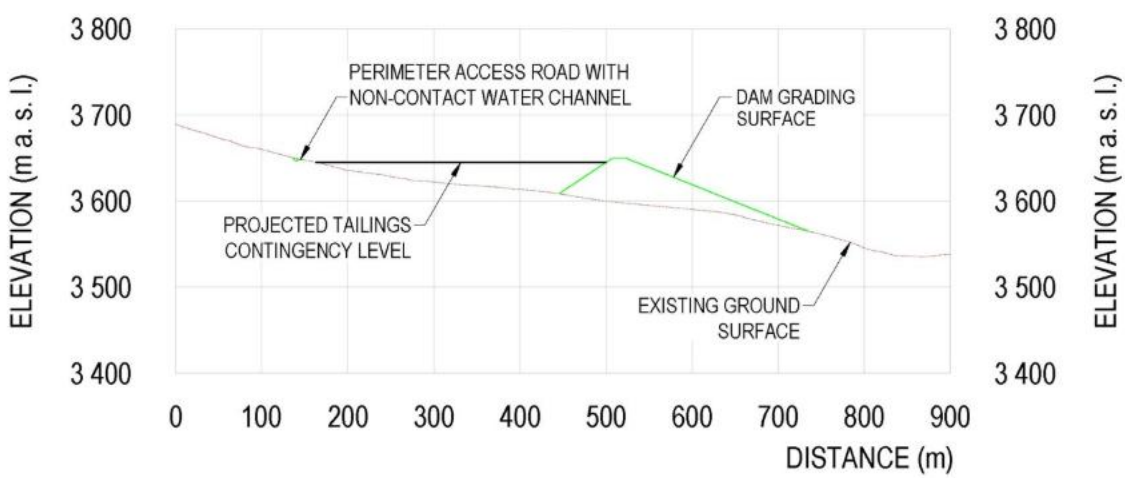
Design Criteria	Unit	Criteria
Seismic Coefficient	-	0.05
Non-Contact Water Diversion Channel - Design Storm Event	years	100
Contact Water Diversion Channel – Design Storm Event	years	30
Contact Water Pond Capacity - Design Storm Event	years	100

This facility is a slurry storage facility with an embankment. Out-of-specification tailings will be transported via a slurry pipeline and spigotted off the dam crest, with a water reclaim system at the northwest end of the facility. Depending on the volume of out-of-specification tailings stored in the facility, capacity can be increased by excavating tailings and placing them in the DSTF (see Figure 18-6).

The main design features of the TCSF are summarized below:

- The TCSF will be constructed in a single phase.
- The embankment's upstream slope will be constructed at 1.5:1 (H:V), the downstream slope at 2.5:1 (H:V), and the overall slope at 3.5:1 (H:V).
- TCSF will have an underdrain in the base of the embankment that discharges into the contact water pond located below the facility.
- The impoundment will be lined with compacted subgrade and a geomembrane liner. The upstream embankment will have a filter, compacted soil liner, and a geomembrane.
- Tailings will be transported from the filter plant to the TCSF via a slurry pipeline and spigotted off the embankment, creating a beach to the northwest. At the northwest end of the facility, a water reclaim system pumps decant water to the filter plant, and the water is then transported with filter water to the process plant for operations.
- A non-contact diversion channel will be constructed to capture and convey runoff upstream of the facility.

Figure 18-6: Aurora Project Tailing Contingency Storage Facility



Source: Ausenco, 2026

18.4.1.11 Seismic Hazard Assessment

Ausenco did not conduct a site-specific seismic hazard assessment for the PEA but instead relied on local developed data. The seismic event used for the stability analysis was the 1:500 year return period, based on the CDA guidelines. The seismic coefficient adopted was 0.05.

18.4.1.12 Stability Analysis

As part of the TCSF design, Ausenco conducted stability and deformation analyses using Slide2v9.03 (Rocscience). A single section was analyzed within the TCSF. Factors of safety were calculated using the following methods:

- LEM
- Spencer Methodology

The stability analysis results showed that the safety factors were greater than 1.5 under static conditions and greater than 1 under pseudo-static conditions for each section (see Table 18-5). Therefore, the TCSF is stable under static and pseudo-static conditions for operations and post-closure.

Table 18-5: TCSF Stability Analysis Results

Structure	Section	Failure surface (surface type)	Factors of Safety (FoS)		
			Static Condition	Pseudo-static condition (kh= 0.16)	Post-earthquake condition
Section A					
TCSF	1-1	Global failure (circular)	2.31	1.42	1.98

18.4.1.13 Instrumentation and Monitoring

The stability of the TCSF will be monitored using vibrating-wire piezometers, inclinometers, and GPS/GNSS sensors strategically placed within the foundation and the tailings storage facility. The piezometers will monitor pore pressure within the foundation and the tailings facility; the inclinometers will monitor the critical failure plane within the facility and the foundation; and the GPS/GNSS sensors will monitor any movement of the facility.

These instruments will be connected to a data logging and telemetry system for continuous, real-time, unattended monitoring. Additionally, instrumentation for water flow and level will be installed to monitor surface-contact water flows in the water collection ponds on the facility's southern side.

18.4.1.14 Closure Concept

The objective of closure for the TCSF is to ensure the long-term stability of its physical, geochemical, hydrological, and revegetation conditions. To achieve this, the closure strategy is to remove the facility at closure. This involves placing any tailings remaining in the facility in the DSTF, removing the liner and embankment, and restoring the site's natural appearance. This will support a return to its pre-mining land use and productivity, ensuring the area can sustain ecological functions and meet long-term land-use goals.

18.4.1.15 Construction and Operations

The TCSF, including the development phase, consists of a tailings containment embankment, a liner system, and a non-contact diversion channel. Out-of-specification tailings will be temporarily stored in the TCSF and placed in the DSTF after they have desiccated.

18.4.1.16 Surface Water Management

The surface water management will include four components:

- Non-contact water for the TCSF in open diversion channels that run around the perimeter of the facility.
- A water reclaim system that returns decant water to the filter plant, where it is combined with filter water and sent back to the process plant for operations.
- Non-surface water at the embankment foundation will be captured by the facility's underdrain system and conveyed to the contact water pond.
- The non-contact surface water management for the TCSF includes permanent diversion channels designed to safely direct runoff around and away from the facility. The non-contact surface water diversion channels will be constructed in a single phase and will convey the 100-year storm event around the facility. The contact water within the TCSF have been designed to hold up to the 30-year storm event in the TSF.

Water captured in the contact water ponds will be pumped to the process plant for use as process water and treated for fresh water supply. Any surplus water will be treated and released to the environment.

18.5 Water and Water Infrastructure

Water is an essential element of all mining project. Section 17.0 discusses the water requirements of the plant, this subsection discusses the water availability and the water balance.

The objectives of the water study for a PEA are limited to determining the economic impact of water management practices.

The following objectives were included as part of the design and cost estimate of water management practices:

- Determination of the availability of surface water (see Section 20.0).
- Determination of the availability of groundwater (See Section 16.0).
- Prediction of how the surface and groundwater systems change with the execution of the mining project.
- Determination if the mine has sufficient water resources to run the nameplate mill capacity in all expected climate conditions, and for all phases of mine life.
- Determination if there is excess MIW present. This is wastewater which may have degraded water quality from interaction with mining, mine wastes, or the mine process.
- Determination of MIW treatment costs, if required.

- Determination of how MIW management impacts closure liabilities.

The following section describes the general methods used to accomplish these objectives.

18.5.1 Methodology

The generalized methodology for predicting water management requirements for a metal mine is a series of industry-standard predictive models of groundwater and surface water. For this study, the following were used:

Table 18-6: Modeling Software and Their Usages in the Aurora Project

Model Name	Company	Media	Used For	Data Required	Section Discussed
IDF Curves	NA	Rainfall	Simulate Design Storms	Climate Data	4.2
PCSWIMM		Surface Water	Predicting baseline surface water flow and volume, and predicting surface water flow at various stages of mine life	Climate data, topographic map, stream channel type, soil type, vegetation type.	20.0
Seep/W	Geostudio	Groundwater	Predict the dewatering requirements of the pit, access tunnels, shafts, and block cave.	Hydrogeologic parameters, layout of tunnels, shafts, development and open pit vs time.	16.0
Goldsim	Goldsim	All Water	Integrated water balance model.	All of the above, process water balance including function of the filter press	This Section

18.5.2 Time Periods

GRE created a site-wide surface water model for the Aurora project using the following distinct timesteps in the mine life:

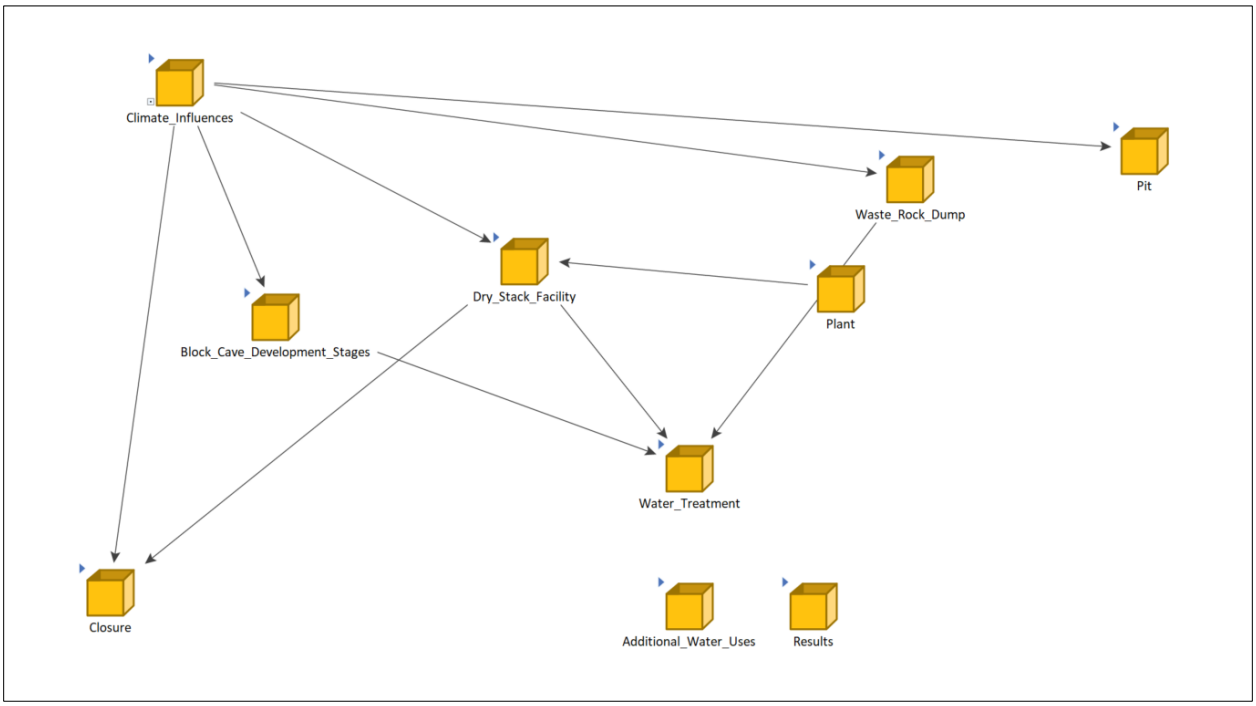
- Pre-mining (baseline conditions)
- Early mine life (open pit, no block cave, beginning underground tunnels)
- Later mine life (closed Late Development (open pit, block cave underground development)
- Block Cave (Pit with hole in bottom, full block caving)
- Closure

These four periods were simulated in all models shown in Table 18-6.

18.5.3 Water Balance

A site-wide water balance (SWWB) was completed in GoldSim, an industry-standard software tool used for mine water balances. In this case, it was used to construct a water balance model that simulates the movement, use, and storage of water over time across the mine site including in the plant, the tailings thickener the tailings filter plant, mine dust control, and other uses and transfers of water. Figure 18-7 shows a schematic of the SWWB layout with all the major components.

Figure 18-7: Aurora Project GoldSim Water Balance Outline



Source: GRE, 2026

18.5.4 Input Parameters and Model Methodology

The Goldsim model uses the following input parameters:

- Climate data (see Section 4.0)
- Results from the surface water models (see Section 20.2)
- Results from the mine dewatering groundwater model (see Section 16.0)
- Results from the plant water balance (17.3.6)
- Mine waste site facility layouts (16.1.7).
- Results from Ausenco's analysis of the tailings filtration system (18.4)

Each of the mine life time periods (see Section 18.5.2) was simulated using Dry, Average, and Wet conditions (see Table 5-1).

Throughout the mine life, the plant makeup water needs are constant because the production rate (65,000 tpd) and the post-filtration moisture content of the mine tailings, at 14.6% (gravimetric), are held constant. This results in a makeup water requirement to the plant of ~966 L/s. Because of water quality

concerns (see Section 20.0), MIW was selected as the preferential plant makeup water source, but for those same water quality reasons, was capped at no more than 50% of the total plant makeup water feed. As a result, about 483 L/s of plant water comes from MIW.

The PCSWMM surface water model and the SEEP/W groundwater model simulate the seasonal rates of MIW and Non-Contact water production, both of which are available sources of makeup water within the GoldSim Water balance The GoldSim model runs one year of possibilities and displays the results as shown in the sections below.

18.5.5 Results

All results for the water balance will be shown only for the average year of rainfall, although modelling done run for dry, average, and wet year of rainfall conditions.

18.5.5.1 Early Mine Life

During the early stages of mine life (year -3 through year 2), the pit is assumed to function as a collection pool with no bottom outlet. Underground water is limited to gradual groundwater inflows into the block cave access tunnels as they are constructed. The underground inflows are averaged to 2,758.87 cubic meters per day (m^3/day) across the 5-year span as the tunnels are developed. The total of MIW from the tunnels per year for the early mine life stage that needs to be treated sums to 1,004,000 m^3.

During early mine life, and assuming the plant is fully operational, all the MIW from the tunnels, the pit, the WRSF and DSTF can be fully consumed by the plant, and the mine does not need a water treatment facility in the early mine life stage.

The pit dewatering system must be sized to manage both monthly (and daily) rainfall and also extreme rainfall events. The pit sumps are designed to have a maximum rate of 126 L/s (2,000 gpm). This pumping rate was selected to allow for the pump-out of the 10- year storm over the course of 72 hours.

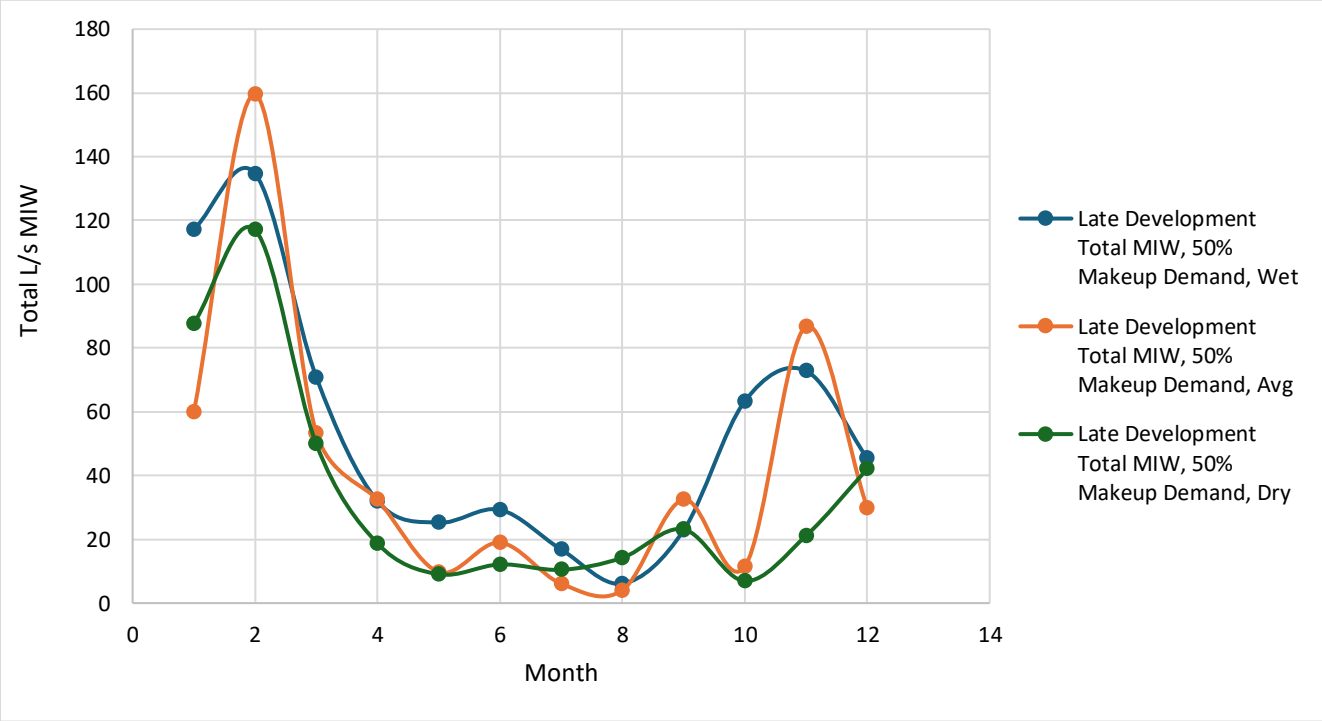
18.5.5.2 Late Development

During the late development stage of the mine (year 3 to year 7), the access tunnels have been excavated and there is further development occurring for the underground workings under the future block cave. Continuing from the early mine life, precipitation into the pit remains a source of MIW, and the size of the pumps in the pit sump remain unchanged. The underground water inflows range from 18,788.72 m^3/day to 29,646.53 m^3/day (see Section 16.3.4). The total MIW from the tunnels and the underground developments in the late development stage that needs to be treated totals to 8,815,000 m^3 per year.

Assuming 50% of the MIW (from the tunnels, underground workings, pit, WRSF, DSTF) is sent to the plant to meet its make-up water demand of 46,903 m^3/day, the remaining MIW will require treatment in a dedicated water treatment plant beginning in year 3 (at the beginning of the late development phase of the mine). See Figure 18-8 for the MIW that needs to be treated during the late development phase.

Figure 18-8 shows that there are treatment needs for all simulated climate conditions, and that the treatment requirements vary greatly throughout the year due to the pronounced wet season. Section 20.0 details the specifics of the necessary MIW treatment system.

Figure 18-8: Aurora Project MIW that Needs to be Treated During the Late Development Phase



Source: GRE, 2026

18.5.5.3 Block Cave

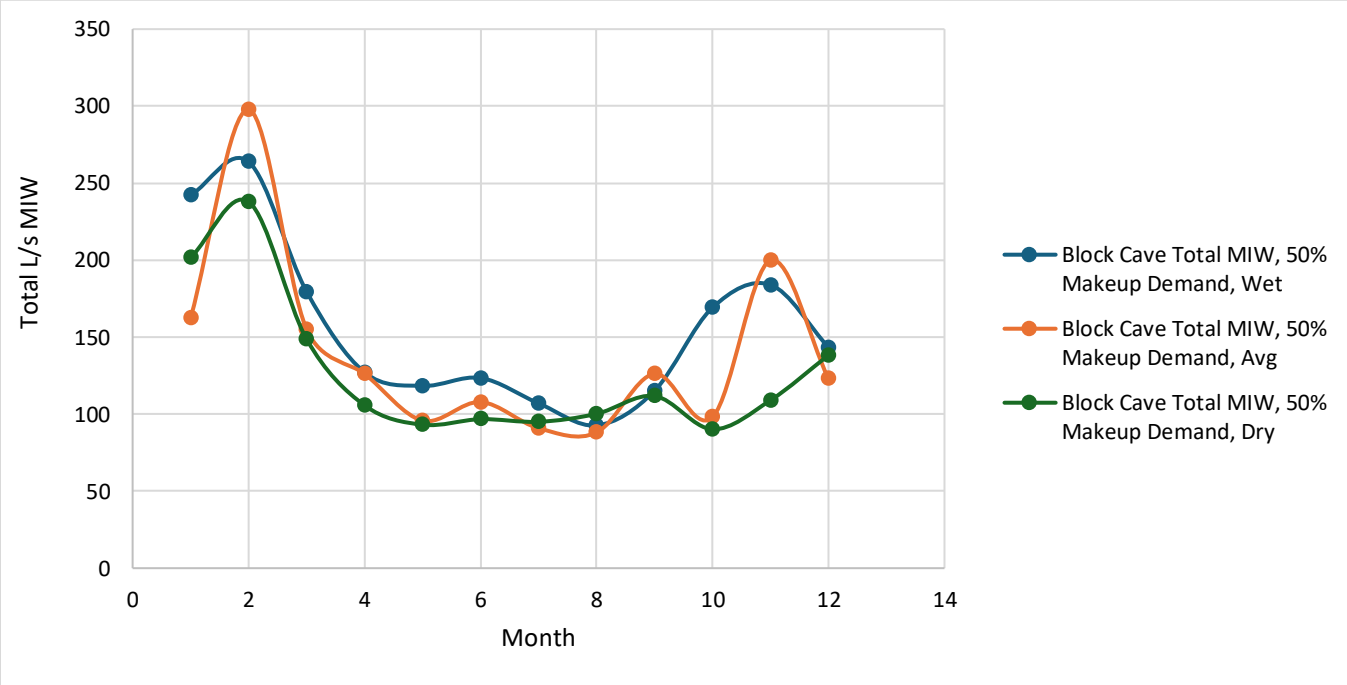
During the block cave phase of the mine life (year 7 to closure) the tunnels are completed, the underground workings are prepped, and the pit now has an open subsidence zone at the bottom that drains down into the block cave. The underground flows from the tunnels (2758.87 m^3/day) are added to the underground flows in the block cave which ranges from 20,776.17 m^3/day to 35,527.27 m^3/day (see Section 16.3.4). These underground flows are then added to the rainfall from coming into the block cave from subsidence zone below the pit. In an average year the total inflow into the block cave that needs to be handled is 11,750,000 m^3.

Assuming 50% of the MIW (from the tunnels, underground workings, pit, WRSF, DSTF) is sent to the plant to meet its make-up water demand of 46,903 m^3/day, the remaining MIW will require treatment in a dedicated water treatment plant. Below in One can see that the increase in total groundwater inflows has increased the minimum treatment requirements from as little as ~10 L /s up to 100 L/s. The maximum treatment requirements have likewise increased from 160 L/s to 300 L/s.

Figure 18-9 can be seen the MIW treatment need.

One can see that the increase in total groundwater inflows has increased the minimum treatment requirements from as little as ~10 L /s up to 100 L/s. The maximum treatment requirements have likewise increased from 160 L/s to 300 L/s.

Figure 18-9: Aurora Project MIW Needing Treatment During the Block Cave

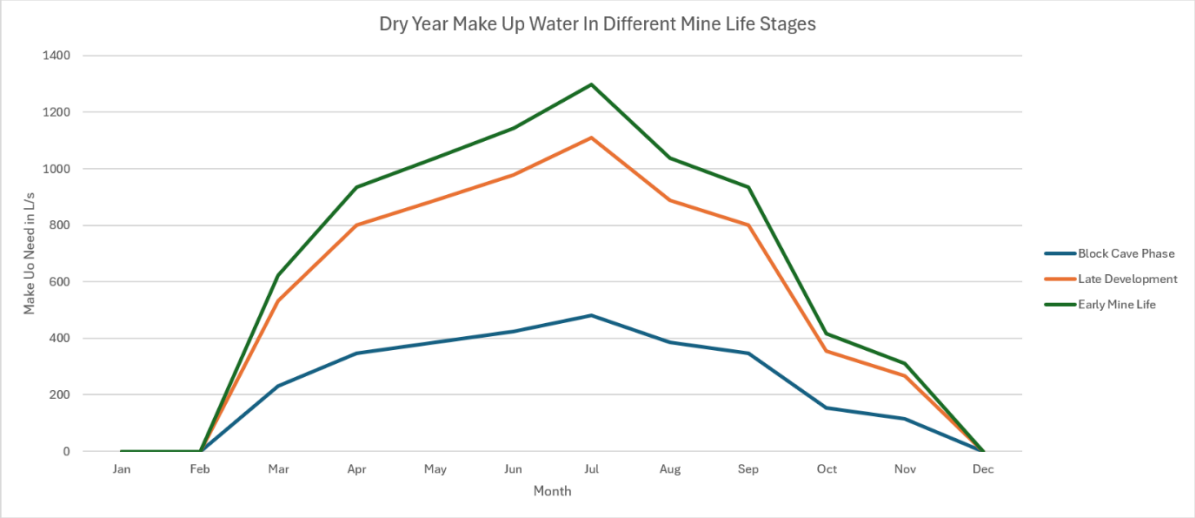


Source: GRE, 2026

18.5.6 Makeup Water Demand

During each phase of mine life, there is a makeup water demand from drainage outside of the Mestizamayo. Figure 18-10 shows the Dry Year (simulated worst-case) makeup water demand for each simulated time period.

Figure 18-10: Aurora Project Makeup Water Needed from Sources Outside the Mestizamayo Drainage Basin



Source: GRE, 2026

One can see that the makeup water is nearly zero during the wet season, but that water abstractions of up to 1200 L/s are required in early mine life, decreasing to ~400 L/s. Typically, when a mine has excess water in one season and a water deficit in another season, this issue is managed through water storage.

However, due to the topography of the site, GRE did not believe that a substantial (~1M cubic meter) water storage reservoir was feasible. The water balance showed that a reservoir of 4,000,000 cubic meters would be required to run the mine on Mestizamayo surface and groundwater were the only source.

GRE assumes that additional makeup water will be pumped from the Mapacho river and has considered that there will be a pump between the Mapacho river and the plant with a maximum yield of 1,200 L/s. Studies of the Mapacho river flow (see Section 20.0) show that this large river basin has abundant water. Dry season flows are ~12,000 L/s, and this flow would make the project's peak abstraction equal to ~10% of the river volume.

18.5.6.1 Closure

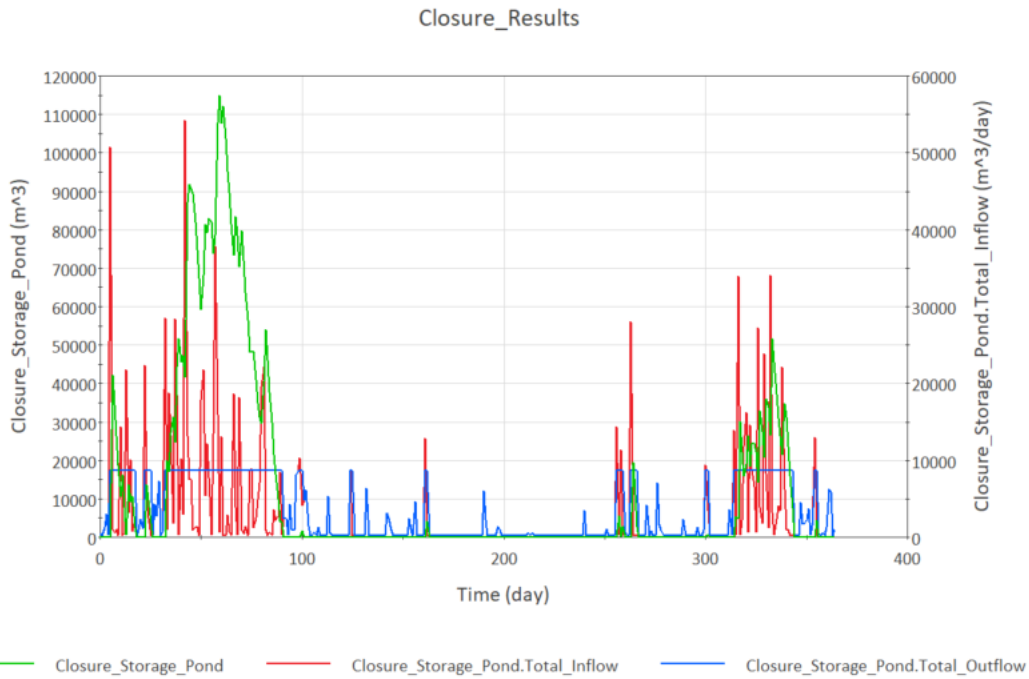
The GoldSim model was also used to model the MIW treatment needs after closure. During closure, the following changes occur:

- The DSTF and WRSF are covered with soil and revegetated (this is assumed to be an ET cover with 80% effectiveness).
- Some diversion channels will be removed, as required and where possible to re-establish natural (pre-mining) drainage patterns.
- Plant areas and other mine process areas will be reclaimed and revegetated.
- A MIW storage pond is added atop the WRSF. This is a requirement to stabilize seasonal MIW flows (see Section 18.5.5).
- The block cave subsidence zone remains and will funnel water into the underground. This makes water level control, and a modest amount of dewatering required (see Section 16.0)

In completing this balance, it was determined that there was a need for a storage surge pond. This was due to the fact that during operations, 50% of the MIW went into the plant as makeup water; however, once the plant is shut down, 100% of MIW must be treated. As a result, a storage surge pond was designed in AutoCAD on the reclaimed WRSF which has a maximum storage capacity of 269,484 m^3.

Figure 18-11 shows the results of the GoldSim simulation of the MIW water volumes in the storage pond over the course of an average year of rainfall, considering a maximum pumping rate out of the surge pond of 100 L/s.

Figure 18-11: Aurora Project Closure Storage Surge Pond Results



Source: GRE, 2026

The pumping rate of 100 L/s was chosen to ensure that the storage pond remains substantially below the surge pond capacity to be able to successfully manage the 10-year design storm event without overflowing.

18.5.7 Water Management Conclusions

The following water management infrastructure is required:

- A toe pond at the base of the WRD to capture MIW in the Mestizamayo for treatment.
- A 300 L/s high density sludge mine water treatment system to treat and neutralize MIW prior to discharge.
- Surface water diversion channels around all site infrastructure.
- Pumping systems to dewater the underground mine and the open pit.

All of these are included in the cost estimate (see Section 21.0).

18.5.7.1 Makeup Water and Changes in Water Availability

The site has a net water shortage which must be acquired from water sources outside of the Mestizamayo drainage. This PEA assumes the water will be abstracted from the Mapacho river. The Mapacho River has a dry season baseflow of ~12,000 L/s (Urano, 2022). Furthermore, the Mapacho river is not widely used for agriculture or industrial purposes. A study of the Mapacho river showed that only 12 L/s of use is currently allocated to communities (Urano, 2025).

GRE concludes that the Aurora project will not have a significant impact to local or regional water availability. In addition to the makeup water abstraction during the dry season discussed above, the

project will diminish water flow in the Mestizamayo drainage basin. The maximum decrease is approximately 253 L/s during the wet season and 158 L/s during the dry season. With both the water abstraction from the Mapacho, and the decrease in flow in the Mesitizimayo, the project is expected to have a maximum total change in water availability during the dry season of ~ 1,300 L/s. This is less than 11% of the dry season baseflow discussed above. This abstraction will not have a significant impact on ecological or anthropogenic water use.

19.0 MARKET STUDIES AND CONTRACTS

The Aurora project would produce copper concentrate and molybdenum concentrate. A long-established, active, worldwide market exists for the buying and selling of copper, molybdenum, and silver. DLP expects this to continue throughout the life of the Aurora Project. Further market studies are not deemed necessary to establish the existence of a market for the products.

DLP does not have current contracts in place for sale of copper and molybdenum concentrates; however, typical terms for nearby producing mines were used and include the following:

- Copper Concentrate:
  - Cu Payable: 96.5%
  - Smelting charge: $25/dmt
  - Refining charge: $0.025/dmt
  - Transport charge: $169.35/dmt
- Molybdenum Concentrate:
  - Mo Payable: 96%
  - Roasting charge: $0.08/dmt
  - Transport charge: $160/dmt
- Silver:
  - Payable only in the copper concentrate
  - Ag Payable: 96%
  - Ag refining charge: $0.75/oz Ag
- Copper Concentrate Solids: 90%
- Molybdenum Concentrate Solids: 97%

The base case copper price used was $4.90/lb, which incorporates the 3-year trailing average of $4.44/lb and the one-year futures price of $6.674/lb (as of 30 April 2026). GRE has provided sensitivity analysis from -25% ($3.675/lb) to +25% ($6.125/lb). The price of copper has been rising, and the GRE QP believes the $4.90/lb base case price reflects the consensus market forecast for copper.

The base case molybdenum price used was $25.40/lb, which incorporates the three-year trailing average of $22.08/lb and the one-year futures price of $30.5/lb (as of 30 April 2026). GRE has provided sensitivity analysis from -25% ($19.05/lb) to +25% ($31.75/lb). The price of molybdenum has been rising, and the GRE QP believes the $23.80/lb base case price reflects the consensus market forecast for molybdenum.

The base case silver price used was $45.30/oz, which incorporates the three-year trailing average of $37.03/oz and the one-year futures price of $78.406/oz (as of 30 April 2026). GRE has provided sensitivity analysis from -25% ($33.975/oz) to +25% ($45.625/oz). The price of silver has been rising, and the GRE QP believes the $45.30/oz base case price reflects the consensus market forecast for silver.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Conditions, and Environmental Baseline

This section provides an overview of the site conditions and provides a summary of the existing baseline data acquired by DLP. Aurora is an early-phase development and exploration project which has a less-developed baseline database than more-advanced exploration properties. The site environment, including climate, vegetation, and physiography, is covered in Section 4.1.3.

20.2 Environmental Studies

DLP has commissioned environmental studies in support of the Peruvian Environmental Impact Assessment (called the Delcaracion de Impactos Ambientales (DIA) for the exploration platforms and exploration drilling activities. Most of these were performed by Urano Consultores in Peru.

The following studies were conducted as part of the DIA:

- Climate
- Meteorology
- Air quality
- Noise
- Soil type and soil chemistry
- Hydrology
- Water quality
- Socio-Economic
- Community involvement and community relations

In addition to these, GRE performed the following studies in support of this technical report:

- Climate analysis
- Surface water availability study
- Site-wide water balance
- Environmental geochemistry study

Each of the preliminary studies is summarized below, except for the water balance, which is in Section 18.5.3. It is important to note that early-phase projects are not required to have a full set of baseline environmental studies, and that work is ongoing. Section 26.0 discusses the recommendations for advancing the environmental baseline environmental effort. Section 5.4 discusses the climate and the data used in climate analysis, which was part of the DIA and part of GRE's water analysis.

20.2.1 Baseline Air Quality

Air quality studies discovered that no baseline air quality issues exist. Ten parameters were evaluated covering the range of regulated metals, volatile components, and particulates, and no samples exceeded air quality guidelines. (Urano, 2022)

20.2.2 Baseline Noise

A noise study was performed and determined that prior to exploration activities, neither daytime nor nighttime noise exceeded Peruvian guidelines (Urano, 2022).

20.2.3 Baseline Soil and Soil Chemistry

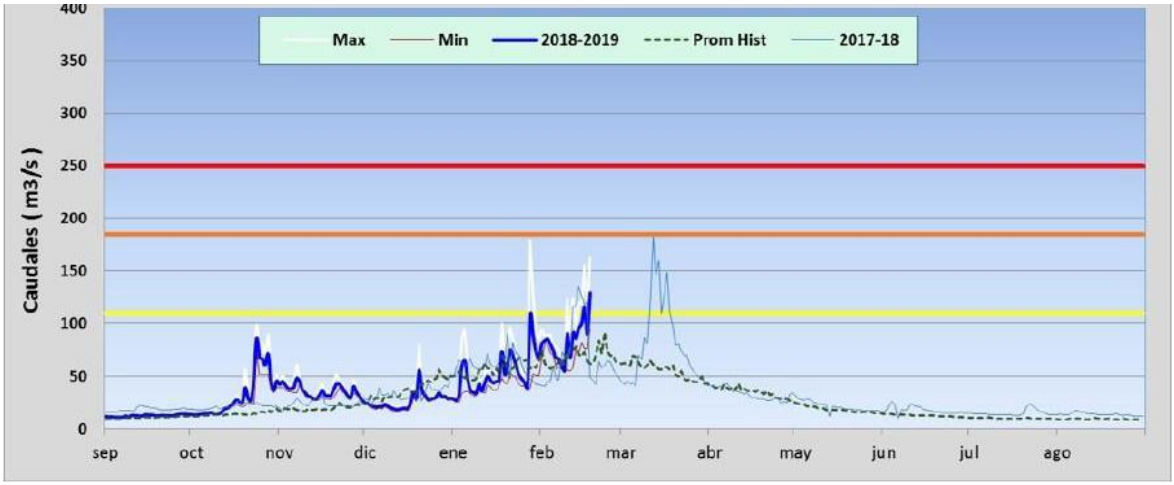
A soil study was performed, and no soils had regulated parameters in excess of regulatory guidelines (Urano, 2022).

20.2.4 Baseline Surface Water Flows (Quantity)

Urano conducted a baseline study of the flows in the Mapacho River. The Mapacho river drains 58,000 square kilometers of eastern Peru and in many places, forms the natural boundary between the Altiplano and the Amazon (Urano, 2025). The nearest gauging station for this river is at Paracartambo, located ~50 km (in a straight line) upstream from the site. By GRE's count, there are at least nine or ten tributary drainages which join the river between Paracartambo and Aurora, so the values at the gauging station can be assumed to be a conservative (low end) estimate of flows at the site. The river is capable of flows in excess of 150 m^3/s. Baseline flows are approximately 12 m^3/s (or 12,000 L/s) (Urano, 2022).

Figure 20-1 shows a hydrograph of the Mapacho flows at Paracartambo, with the maximum, minimum, historic average, and the 2018-2019 values.

Figure 20-1: Annual Hydrograph of the Rio Mapacho at Paracartambo (~50 Km Upstream from Aurora)



Source: Urano, 2022

At present, there are no flow readings of the river discharge in the Mestizamayo drainage basin, but GRE performed a surface water model (see next section) to predict flows in this drainage basin both prior to mining, and after mining impacts.

20.2.5 Surface Water Flow and Surface Water Study

Because no on-site stream gauging data exists, GRE had to predict the water availability and the water use conditions for the Aurora Project. To accomplish this objective, GRE created a surface water flow model using PCSWMM. PCSWMM is a professional stormwater and wastewater modeling software based on EPA SWMM (PCSWMM, 2026). It is used to create and compare multiple scenarios to evaluate impacts on water systems.

GRE simulated surface water flow during the following different time periods in the mine life:

- Pre-mining (baseline conditions)
- Early mine life (open pit, no block cave, beginning underground tunnels)
- Later mine life (closed Late Development (open pit, block cave underground development)
- Block Cave (Pit with hole in bottom, full block caving)
- Closure

These time periods correspond to water simulations for mine dewatering (see Section 16.3) and the Site Wide Water Balance (SWWB) (see section 18.5.3).

20.2.5.1 Water Model Without Facilities (Baseline Conditions)

The first step to understanding the surface water movements at the Aurora project was to map the water basin that will be impacted by mine facilities. The yellow lines represent the Mestizamayo River and stream channels that convey surface runoff from the surrounding mountainous terrain eventually into the Mapacho River after the outfall. The green areas delineate the sub-catchments that are a part of the drainage basin and will eventually have run off that drains down into the channels as it rains. Each sub-catchment represents a distinct hydrologic unit that contributes runoff to the drainage network in response to precipitation. Rain falling within these areas is routed overland and through tributary channels before arriving at the Mestizamayo River.

The red triangle represents the outflow of the Mestizamayo River into the Mapacho River. This is the location that is most pertinent to understand the flow because the confluence of the Mestizamayo and the Mapacho is assumed to be the ultimate Point of Compliance for Peruvian water quality regulations and requirements.

The water model without facilities was run for a dry, average, and wet rainfall year as defined by Table 5-1. The results for the outflow of the Mestizamayo River for dry, average, and wet years without facilities (baseline conditions) are as follows:

- Dry: 10,097, 600 m^3
- Average: 16,490,000 m^3
- Wet: 20,849,900 m^3

Figure 20-2 shows the hydrologic basin that the of the Meztizimayo river, which is the drainage basin which contains most of the mine infrastructure and mine waste facilities.

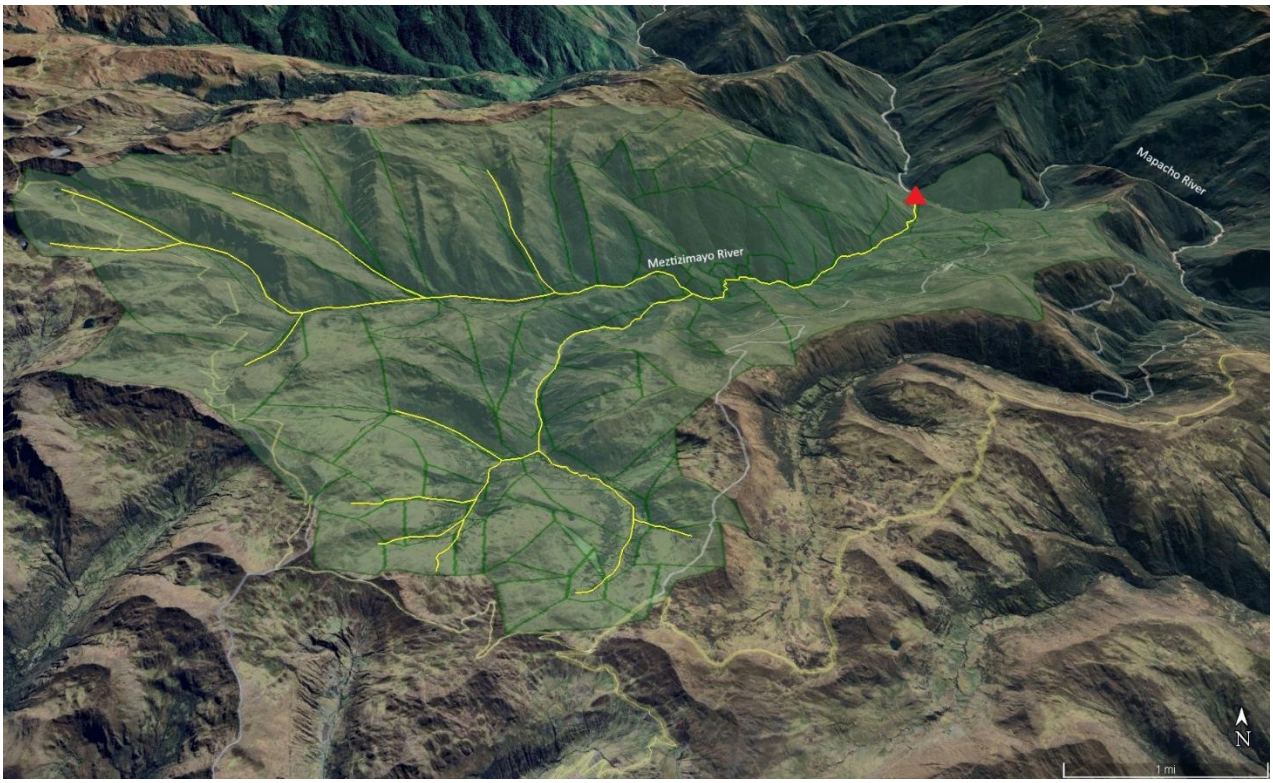
The yellow lines represent the Mestizamayo River and stream channels that convey surface runoff from the surrounding mountainous terrain eventually into the Mapacho River after the outfall. The green areas delineate the sub-catchments that are a part of the drainage basin and will eventually have run off that drains down into the channels as it rains. Each sub-catchment represents a distinct hydrologic unit that contributes runoff to the drainage network in response to precipitation. Rain falling within these areas is routed overland and through tributary channels before arriving at the Mestizamayo River.

The red triangle represents the outflow of the Mestizamayo River into the Mapacho River. This is the location that is most pertinent to understand the flow because the confluence of the Mestizamayo and the Mapacho is assumed to be the ultimate Point of Compliance for Peruvian water quality regulations and requirements.

The water model without facilities was run for a dry, average, and wet rainfall year as defined by Table 5-1. The results for the outflow of the Mestizamayo River for dry, average, and wet years without facilities (baseline conditions) are as follows:

- Dry: 10,097, 600 m^3
- Average: 16,490,000 m^3
- Wet: 20,849,900 m^3

Figure 20-2: Aurora Site Hydrologic Basin Without Facilities



Source: GRE, 2026

20.2.5.2 Early Development Mine Life Surface Water Model

After receiving the final site layout plan, a surface water model with mine facilities was created for the early mine life. This model incorporates the open pit acting as a collection pool with no hole at the bottom yet, the WRSF at 60% total capacity, and the DSTF at 50% total capacity. This model also includes the plan to divert sections of the Mestizamayo River around both the DSTF and the WRSF. A small collection pond will be created in this phase of the mine life and it will be likely located at the toe of the WRSF in the Mestizamayo drainage basin as a collection point for the treatment of MIW.

20.2.5.3 Later Development Mine Life Surface Water Model

All pertinent site facilities are added at this time, including the WRSF at 80% of its total capacity, the DSTF at 70% of its total capacity, pit, haul roads, plant, explosive storage, and primary crusher. At this stage also begins the digging of underground development beneath the block cave zone alongside the already dug access tunnels. This model also includes the plan to divert sections of the Mestizamayo River around both the DSTF and the WRSF for the purpose of keeping the amount of MIW to a minimum.

20.2.5.4 Block Cave Stage Mine Life Surface Water Model

The surface water model was modified to include the later block cave stage of the mine life facilities. This permutation of the model includes the DSTF and WRSF at maximum capacity and the pit as a subsidence zone above the block cave. It also includes the fully built haul roads, plant, explosive storage, and primary crusher.

20.2.5.5 Closure Phase Surface Water Model

For the closure phase, the following modifications were made:

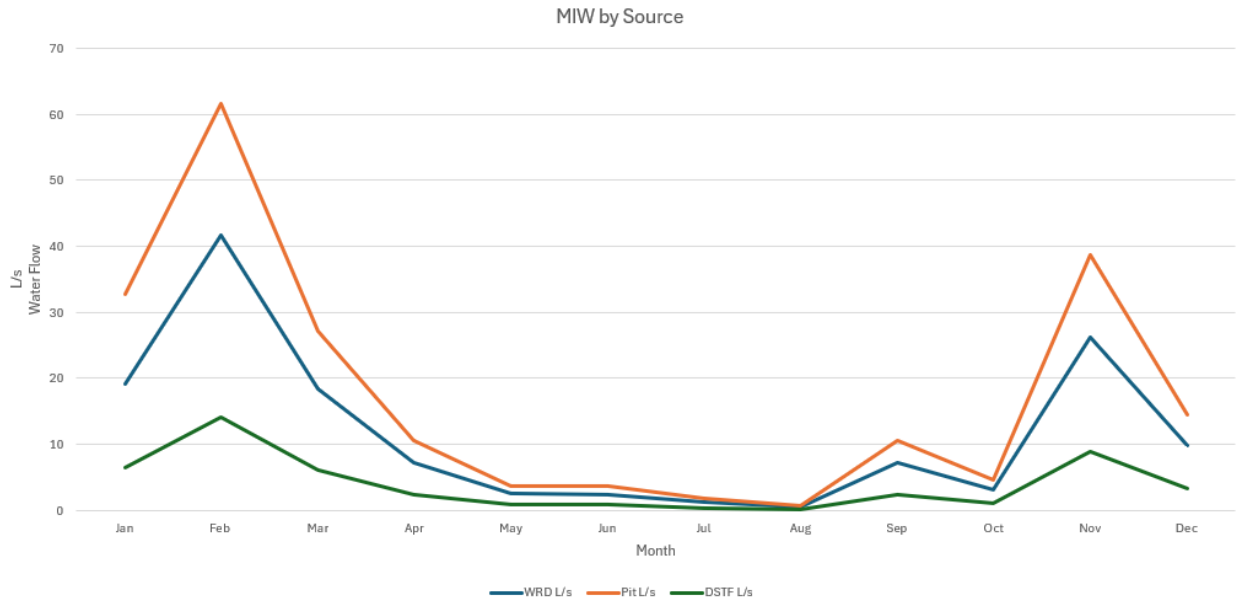
- The DSTF and WRSF are covered with soil and revegetated. Pond for MIW treatment will be built on the bench of the WRSF for water treatment equalization.
- The subsidence zone of the pit remains and will funnel water into the underground.
- Areas of disturbed ground were converted to revegetated, resulting in a change in total runoff.

The PCSWIMM model was adapted to account for this post-closure land use pattern

20.2.5.6 Model Results

The following subsection discusses the model results. The model calculated the difference between baseline non-contact water and “with project” non-contact water, but the more-relevant value for the PEA is the quantity of MIW over time, because this water must be treated (see Figure 20-3). GRE simulated the production of MIW for wet, dry, and average conditions (see Table 20-1), but only the average year MIW production graphs are shown below. Three sources of MIW are shown on the graphs, WRSF-sourced MIW (from runoff), DSTF sourced MIW, and Pit sourced MIW (from direct precipitation within the pit shell, groundwater-sourced mine dewatering is discussed separately, in Section 16.3.

Figure 20-3: Aurora Project MIW by Source for Early Mine Life Stage During an Average Year

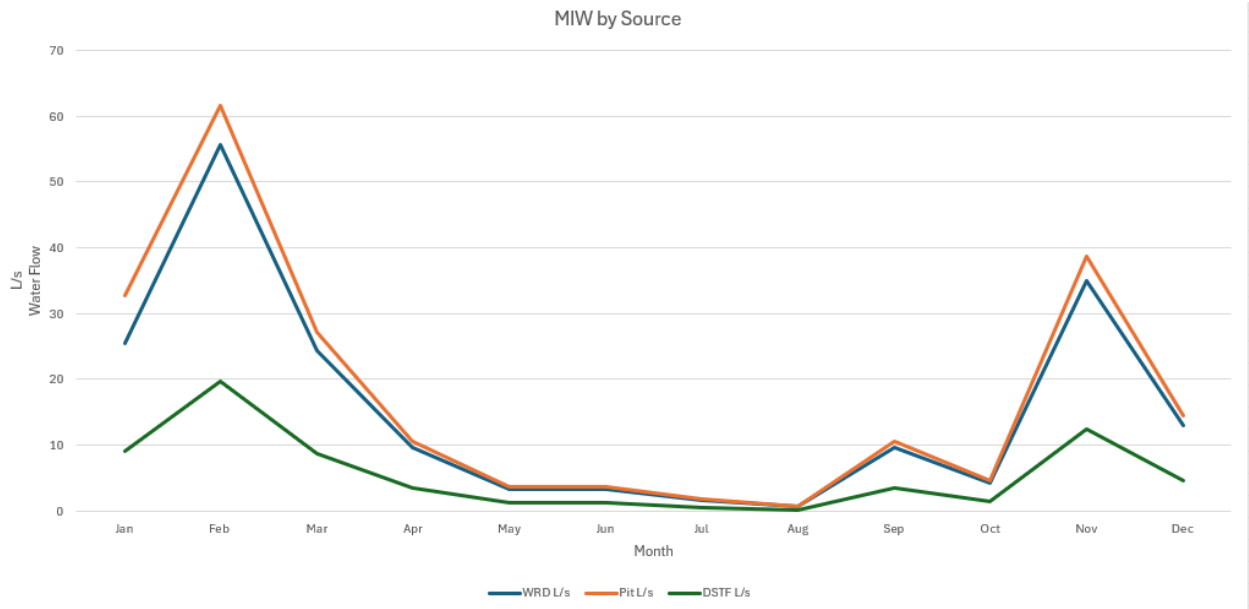


Source: GRE, 2026

As can be seen in Figure 20-3, during an average year of rainfall in the early mine life stage, the majority of the MIW continues to come from the pit with a maximum flow of 61 L/s.

During late development and an average year of development (Figure 20-4), most of the MIW comes from the pit, with the WRSF not far behind. The maximum flow comes from the pit during February and is 61 L/s.

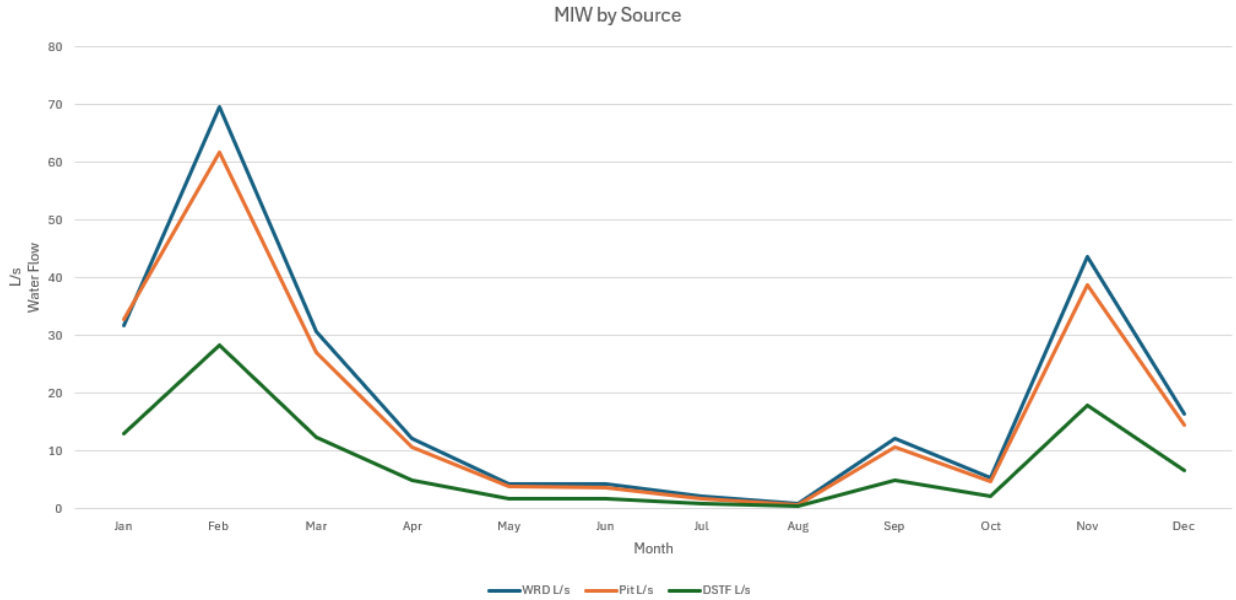
Figure 20-4: Aurora Project MIW by Source for Late Development Mine Stage During an Average Year



Source: GRE, 2026

Figure 20-5 displays a graph of the sources of MIW and their volumes over an average rainfall year during the block cave stage.

Figure 20-5: Aurora Project MIW by Source During the Block Cave Stage for an Average Rainfall Year



Source: GRE, 2026

During an average year, the maximum flow comes from the WRSF and is at 70L/s in February. The pit remains a close second for contributing to MIW during the block cave stage of the mine life.

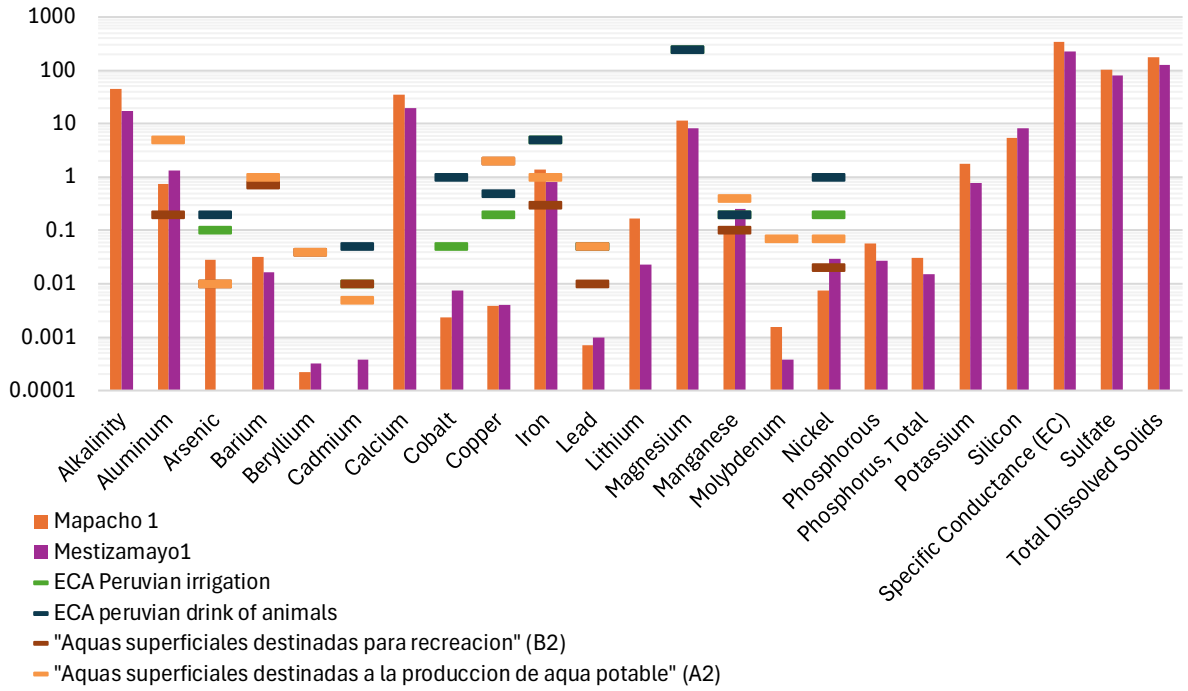
20.2.5.7 Surface Water Modeling Summary

Using the PCSWIMM model, GRE was able to simulate the available surface water in the Mestizamayo drainage over time considering mine infrastructure and waste disposal. Additionally, the model was able to determine the relative quantities of MIW and non-contact water over the life of the project. The output from this surface water modeling effort became the input parameters for subsequent analysis of the SWWB (see Section 18.5.3), and the water treatment plant design (see Section 20.3.1).

20.2.6 Baseline Surface Water Quality

During the 2025 GRE field trip, Mr. Breckenridge collected two water samples: one from the Mapacho river and one from the Mestizamayo river. Both samples were transported to the US and submitted to Enthalpy laboratory in Denver, Colorado, for total dissolved solids (TDS) by 2540C, total anions by IC300.1, total metals analysis by EPA 200.8 and total mercury by EPA 245.1. The hold time was exceeded for the TDS analysis, but the analysis was performed anyway. The results of the analysis are presented Figure 20-6 in reference to various Peruvian water quality standards (2017) (Espinoza, 2017). The background water quality of these two rivers is important in determining the potential impact that mining operations will have on water quality, as well as the amount of mine-impacted water that can be discharged.

Figure 20-6: Aurora Project Results of Water Quality Samples Collected by GRE



Source: GRE, 2026

Antimony, Bismuth, chloride, chromium, fluoride, mercury, and selenium concentrations were below detection limits for both rivers. Additionally, cadmium was below detection limits in the Mazapacho sample, and arsenic was below the detection limit in the Mestizamayo sample.

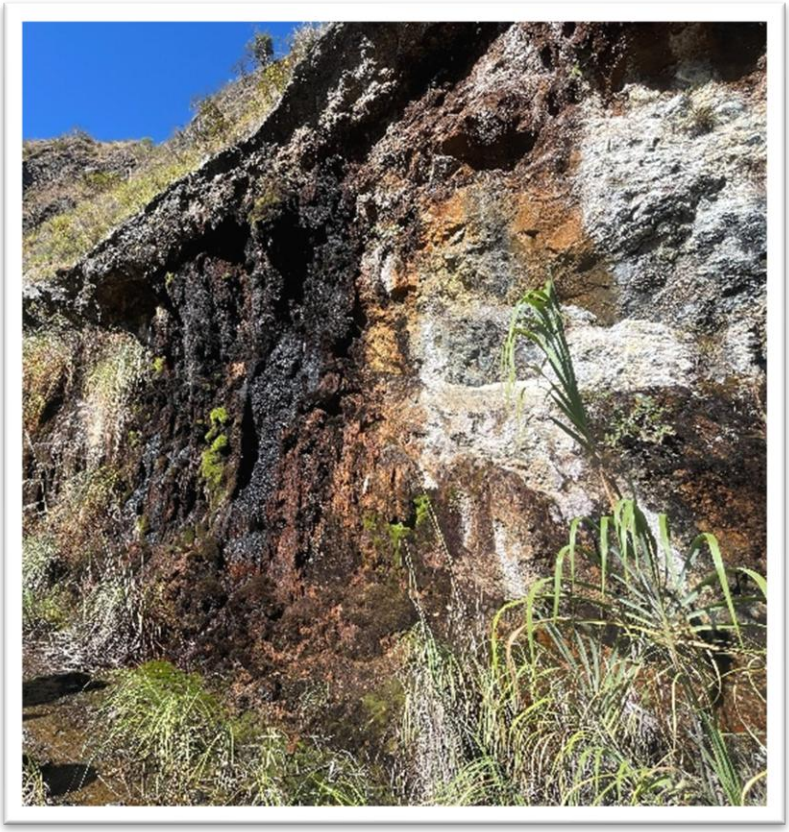
The water quality of the two river samples is generally good; water from both rivers largely meets water quality standards for the majority of elements. Aluminum, arsenic and iron in the Mapacho river are elevated above a few water quality reference values. Iron, manganese, and nickel in the Mestizamayo river are elevated above their respective water quality reference values. TDS are 178 milligrams per liter (mg/L) in the Mestizamayo and 126 mg/L in the Mapacho river, with sulfate at 80.2 mg/L and 102 mg/L respectively. Neither river showed the impact of mining-related water quality impacts even though the upper reaches of the Mapacho river contain small-scale mining operations. Urano took samples from the Mapacho river as part of the DIA and found similar water quality results (Urano, 2022).

20.2.7 Baseline Environmental Geochemistry Studies

During his field visit, Mr. Breckenridge observed evidence of acid rock drainage (ARD) at the outcrops of the basement rocks (in particular, the Sandia formation). The Sandia Formation is a slate/shale formation which includes reduced sulfur in the form of pyrite and is naturally oxidizing.

The evidence included both iron oxide staining and the formation of acidic salts where water has evaporated on the face of the outcrop, as well as a naturally occurring acidic seep. Photo 20-1 shows the seep where Mr. Breckenridge measured a water pH of 3.2, as well as the outcrop from which the seep originates.

Photo 20-1: Aurora Project Photos of Observed Onsite Acidic Seep



Source: GRE, 2026

20.2.7.1 Geochemical Sampling

GRE performed industry-standard geochemical sampling based on the protocols and recommendations in the Global Acid Rock Drainage (GARD) Guide published by the International Network for Acid Prevention (INAP, 2009). A total of 32 rock samples were collected from exploration core on Mr. Breckenridge's field visit. Samples were selected to encompass the entire range of rock types that could be encountered onsite, as well as the estimated relative abundance of each rock type in the waste material. Samples of waste rock were selected based on the best-available indication of where the future pits may be. The GRE team avoided ore-grade material while capturing rock that appeared within a future pit extent.

Selected samples were analyzed for the following industry-standard static geochemical testing program at the SGS Laboratory in Burnaby, Canada:

- Acid Base Accounting (ABA) by modified Sobek (Lawerence & Wang, 1996)
- Metals by *aqua regia* digestion and ICP-MS
- Synthetic Precipitation Leaching Procedure (SPLP) (US EPA, 1994)

20.2.7.2 Acid Base Accounting Results

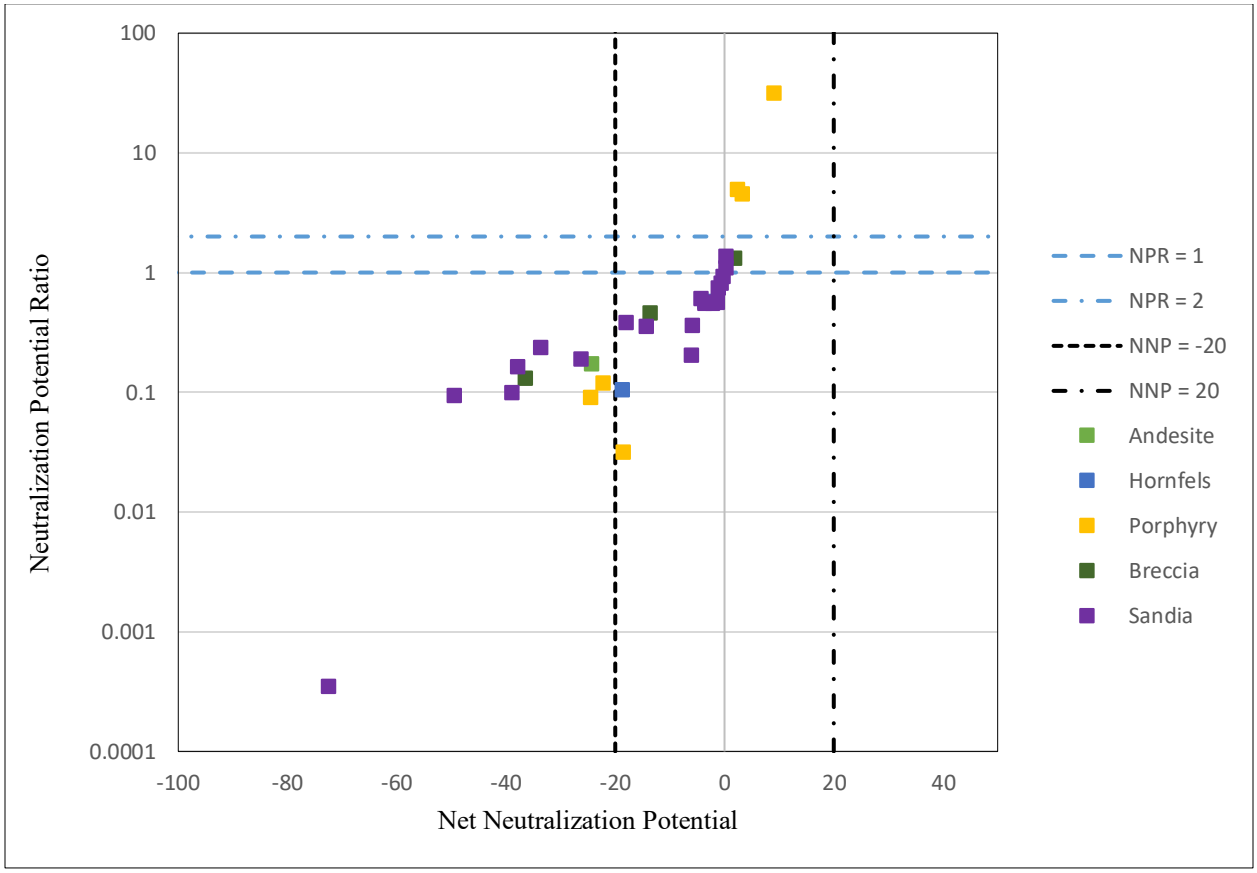
In ABA tests, the acid generating potential (AP) and neutralization potential (NP) values can be combined to derive a quantitative screening-level estimate of a material's overall acid-generating or neutralizing potential. The AP is calculated from the total amount of acid generating sulfides in the rock, whereas the NP is determined from the total amount of acid neutralizing minerals (primarily carbonates). Subtracting AP from NP (NP - AP), gives the Net Neutralization Potential (NNP), while dividing NP by AP (NP/AP) gives the Neutralization Potential Ratio (NPR). Based on the resulting values of NNP and NPR, the samples are classified as "potentially acid-generating" (PAG), "potentially acid neutralizing" (PAC) or "uncertain" according to the criteria given by the GARD Guide, (INAP, 2009). These criteria are presented in Table 20-1.

Table 20-1: Screening Guidelines for Acid Generation Potential Prediction

Material Designation:	Comparative Criteria	
	NNP ($TCaCO_3$/kT)	NPR
Potentially Acid-Generating (PAG)	< -20	< 1
Uncertain	-20 < NNP < 20	1 < NPR < 2
Potentially Acid-Consuming (PAC)	> 20	> 2

The results of the ABA testing for all sampled rock and tailings are presented below in Figure 20-7, which plots the NNP vs the NPR for each sample. Samples have been separated based on rock type. The horizontal lines and vertical lines in the figure represent the different cutoff criteria as shown in Table 20-1.

Figure 20-7: Aurora Project ABA Results, by Rock Type



Source: GRE, 2026

Of the 32 geochemistry rock samples taken for the PEA level study, 10 tested as PAG, 5 are non acid generating/uncertain, and the remaining 17 are non-reactive. No samples had neutralizing potential significantly in excess of acid generating potential, and therefore none of the samples are PAC.

20.2.7.3 Metals Analysis

Quantitative assessment of rock mass for metals and metalloids can be used to screen out metals for consideration in future water quality analysis. This is best done by comparing the concentration of metals in a rock sample, as determined by digestion with Aqua Regia (a strong acid) followed by ICP-MS analysis, with the abundance of that metal in the earth's crust. In general, if a rock type has a concentration of a metal greater than five times the crustal average, it is considered enriched for that metal. A rock enriched with a metal is at higher risk for leaching that metal under metal leaching (ML) conditions. However, if rock has a metal concentration near or below crustal averages, it can reasonably be removed from consideration as a Constituent of Concern (COC).

The analysis shows that the following metals may be COCs in future analysis of mine leachate water quality, as they show levels greater than 5 times crustal averages:

- Silver
- Arsenic
- Bismuth

- Copper
- Molybdenum
- Selenium
- Zinc

20.2.7.4 Short-Term Leaching Tests

Short-term SPLP tests were conducted on exploration rock to estimate the site-specific concentration of contaminants that may leach from waste rock and impact water quality. It is important to note that the results of the SPLP tests serve as a general indication of metal leaching potential from waste rock, and are not representative of the expected levels of onsite ML. The maximum, average, and median values of the samples for each constituent of the SPLP analysis were compared to permissible limits for the discharge of liquid effluents from mining-metallurgical activities, as well as the water quality limits in Peru for waters used for recreation (Decreto Supremo No.004-2017-MINAM, 2017) (Espinoza, 2017).

The SPLP leaching tests indicate that aluminum, arsenic, copper, iron, nickel, and zinc may be of concern in waste rock leachate. The maximum leaching concentrations for all constituents were seen in the most potentially-acid generating samples. This shows that greater ML risk exists in samples with greater PAG.

20.2.7.5 Waste Rock Characterization Conclusions

The geochemical tests performed show that 31% of the waste rock onsite is potentially acid generating due to high sulfides. Additionally, rock onsite does not appear to have significant neutralizing potential available to assist in the neutralization of generated acid on a bulk scale. SPLP tests showed that aluminum, arsenic, copper, iron, nickel, and zinc may be metals of concern for ML. ML leaching behavior is intensified in samples with greater potential for acid generation. As a result, it is necessary to conclude that the future Aurora project will have ARD and ML issues to manage and mitigate at all stages of mine life.

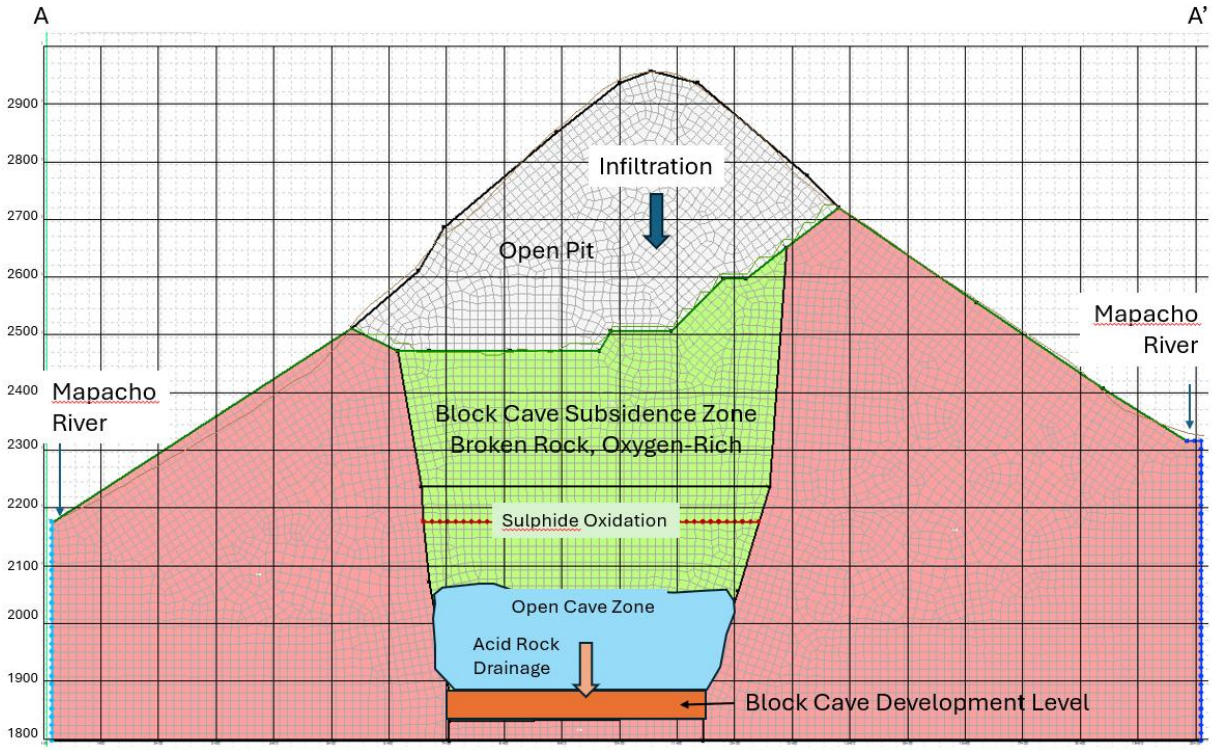
20.2.7.6 Estimates of Future Mine Influenced Water Quality

Based on the conclusions above, GRE assumed that all MIW will need to be treated. At this early phase of mine development, there is insufficient test work to predict the quality of MIW using industry-standard methods – for example, by evaluating the results of long-duration geochemical kinetic test work. However, some assumptions on future MIW water quality must be made.

During operations, water will be rapidly circulated throughout the mine process, and excess alkalinity will be available from the tailings filtrate water. As a result, it is likely that the MIW will be mild ARD rather than severe ARD. GRE has therefore selected a representative acidified SPLP water quality as the basis of operations-phase MIW. A breccia sample from A24-014, from 173 to 174 meters had a pH in the SPLP test of 3.8 with 92 mg/L total acidity. This was simplified to a pH of 3.0 and a total acidity of 100 mg/L and assumed as the best choice for MIW quality for the PEA.

However, conditions are different during closure. The block cave creates a subsidence zone above the block cave. Figure 20-8 shows a schematic of the post closure block cave hydrologic and hydrogeologic system.

Figure 20-8: Aurora Project Post Closure Block Cave Hydrologic System



Source: GRE, 2026

Due to the unstable and rubble-filled subsidence zone, there is a portion of the rock which will permit oxygen penetration, will capture rainfall in a "funnel effect", and will be inaccessible due to safety issues. This rubble zone will have sulfide oxidation and ARD production that cannot easily be mitigated. The water will report to the block cave development area, and ultimately, will fill up the underground workings and report to either the portals, or fractures in the bedrock. Because the ARD cannot be controlled, GRE assumes it will be worse quality. The selected quality for post-closure water from the block cave and glory hole is assumed to be pH of 2, and 1000 mg/L of acidity. Due to the short contact time, none of the SPLP samples had similar water quality characteristics.

20.2.8 Baseline Biological Studies

On behalf of DLP, Urano conducted a biodiversity study and a study of flora and fauna in the project vicinity as part of the DIA (Urano, 2022). Ecological regions are described in Section 5.4. This area is a transition/border zone between the altiplano and the upper amazon basin. It has evapotranspiration of approximately twice annual precipitation. Biological baseline studies covered all major plant and animal species and determined the baseline biodiversity indice (Urano, 2022).

20.3 Requirements for Waste and Tailings Disposal

As mentioned in Section 20.2.7, mitigation strategies for prevention and treatment of ARD and ML will be required for the Aurora project throughout mine life and post closure. Mitigation will be focused on the separation of MIW and Non-Contact Water to minimize MIW volume. This includes the building of diversion channels and culverts around major mine waste facilities to reduce the amount of water that contacts mine workings. Additionally, MIW volume will be reduced through the consumption of it in the

process. The SWWB (see Section 18.0) shows that 50% of the MIW can be consumed by operations, but the remainder must be treated prior to discharge. The waste rock and tailings management plan must be considered in context with the water studies presented in Section 20.2.5 and Section 18.0.

Because of the steep terrain, it will not be possible to line the mine waste facility with an unconducive liner. However, the steep valley, and the single-discharge point at the mouth of the Mestizamayo will allow for the capture of MIW runoff and seepage in a pond where the MIW can be conveyed to the Water Treatment Plant (WTP).

20.3.1 Water Treatment

A mine WTP will need to be constructed by mine year 3 to process up to 300 L/s of MIW (see Section 18.0). This treatment plant would be a High-Density Sludge active water treatment system. Sludge from the WTP will be added to the tailings stream and would ultimately be dewatered in the filter plant and placed on the DSTF – co disposed with the tailings. Section 21.0 discusses the cost estimate for the capital and operating costs for the WTP.

20.3.2 Monitoring

Onsite water will need to be monitored at all major outflows to ensure that water quality guidelines are being met. DLP will continue to monitor water quality as per the permitting requirements for the exploration phase. This is currently being done by URANO on behalf of DLP.

20.3.3 Waste Management and Closure

Upon closure (either concurrent or final), it will be necessary to create an engineered evapotranspiration mine waste cover (evapotranspiration [ET] cover) to minimize the infiltration of precipitation into the mine waste. Section 6.6.6 of the Global Acid Rock Drainage Guide (GARD Guide) published by INAP discusses engineered covers, of which evapotranspiration covers are the most common (INAP, 2009). ET covers are specialized waste covers designed to store wet-season moisture so that it can be evaporated or transpired in the dry season. Despite the 2:1 ratio of evaporation to transpiration (Urano, 2022), because of the ~1000 mm of precipitation per year, It is assumed that the site may be at the design limits for a store and release cover (Evaluation of Final Cover Performance: Field Data from the Alternative Cover Assessment Program (ACAP), 2002), and the design for the Aurora waste is intended to minimize, but not expected to eliminate, mine waste leachate and percolation. Post closure, the site will need to continue to treat any MIW.

20.4 Project Permitting Requirements

Like many jurisdictions, the mine must comply with a wide range of different governmental permits. Section 4.0 discusses the overall legal framework and shows the permits required to continue exploration. The permitting agencies are listed below in decreasing level of responsibility over mine permitting:

- MINEM, (Ministero de Energia y Minas), which manages permits related to mineral resource extraction and processing
- DGM (Directorate General de Minas), which manages some mining permits

- SENACE (Servicio Nacional de Certificación Ambiental para las Inversiones Sostenibles), which manages enviromental permits
- ANA (Authoridad National de Aguas), which manages water permits
- SERFOR (Servicio Nationcal Forestal y de Fauna Silvestre), which manages forest resources and wildlife
- MTC (Ministero de Transporte), which manages roads and transport
- OSINERGMIN (Organismo Supervisor de la Inversion en Engergia y Minera), which supervises mining investment
- SERNANP (Servicio Nacional de Áreas Naturales Protegidas por el Estado), which oversees protected areas
- DIGESA (La Dirección General de Salud Ambiental), which is responsible for environmental health (for example, permits for potable water treatment systems)
- OEFA, which monitors environmental investments
- SUCAMEC: Superintendencia Nacional de Control de Servicios de Seguridad, Armas, Municiones y Explosivos de Uso Civil, which manages the civillian use of explosives
- Ministero de Cultura, which oversees archeological resources
- DIGESA: La Dirección General de Salud Ambiental, which manages workplace safety
- Ministerio de Telecommunicaciones (MTC), which supervises telecommunications
- Local agencies, which may cover environmental issues, building safety, or a variety of other locally-supplied regulatory oversight functions

20.4.1 Permits to Advance to Feasibility Study Stage

Permits required to advance to PFS stage and the permits to execute the work plan are detailed in Table 20-2 and Table 20-3.

Table 20-2: Aurora Project Development, PFS/FS and Construction-Readiness Permits and Requirements

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
1	Physical, biological and social environmental baseline	Titleholder / registered consultant	Mandatory	PEA/PFS	Preliminary area of Influence	Environmental legislation
2	Environmental classification and/or project-specific terms of reference, where applicable	SENACE / MINEM	Conditional	PFS	Description of the Project	
3	Detailed Environmental Impact Assessment (EIA-d)	SENACE	Mandatory	PFS/FS	Well-developed project engineering	Environmental certification for the larger project
4	Public participation for the EIA-d	SENACE	Mandatory	Before and during site development	Approved Plan of Operations	
5	Binding or non-binding technical opinions	ANA, SERNANP, SERFOR	Conditional	Evaluation of DIA	Plan of Operations	
6	Global Environmental Certification	SENACE	Conditional	FS	Strategic regulatory plan	Integrated studies from the site
7	Initial Mine Closure Plan	MINEM	Mandatory	Start of operations	DIA and closure engineering	Ley 28090 y D.S. 033-2005-EM, modificados.
8	Establishment of mine-closure financial assurance	MINEM / financial entity	Mandatory	Before and during operations	Approved closure plan	Engineering cost estimate for closure
9	Certification of water availability	ANA	Conditional	PFS/FS	Water balance	License for water use, water infrastructure
10	Authorization to construct water-use works	ANA	Conditional	FS/construction	Accredited water resources are available	For all water abstraction structures
11	Water-use license	ANA	Conditional	Before operations	Approved water infrastructure	
12	Authorization to reuse treated wastewater	ANA	Conditional	Before reuse	Treatment plant design	Water quality prediction
13	Authorization to discharge treated wastewater	ANA	Conditional	Before discharge	Treatment plant	

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
14	Delineation / authorization for marginal strips and watercourse crossings	ANA	Conditional	Design/construction	Infrastructure near rivers	
15	CIRA / DAS / archaeological PEA for the final project footprint	Ministerio de Cultura	Conditional	PFS/FS	Defined area of impacts	Archeaological survey
16	PMAR for construction	Ministerio de Cultura	Conditional	Before works	Archaeological survey found cultural resources	Relocate resources or revise plans
17	Forest-clearing and flora/fauna management authorization	SERFOR	Conditional	Before construction	Defined area of impact	Required forest mitigation
18	Opinion/compatibility for protected natural areas or buffer zones	SERNANP	Conditional	PFS/EIA	Defined area of impact	Land north of the Mapacho River is a protected buffer zone. While outside the project footprint, additional discussion will likely be necessary during the EIA permitting stage.
19	Prior social consultation procedure, where applicable	Competent State entity / Ministry of Culture	Conditional	Before administrative measure	Identification of impacted communites	
20	Mining easement or access procedure where no agreement is reached	MINEM	Conditional	Before construction	Private negotiation	
21	Permits for research, collection or access to biological resources	SERFOR/ SERNANP	Conditional	Baseline surveys and ongoing monitoring	Ongoing biological surveys	
22	Electricity concession/authorization and easement	MINEM / OSINERGMIN	Conditional	PFS/FS	Calculation of electrical demand	Plan for transmission and supply
23	Permits for roads, crossings and rights-of-way	MTC and Local/regional government	Conditional	PFS/FS	Layout for right-of-way	Layout of right-of-way zones

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
24	Telecommunications authorizations	MTC	Conditional	Construction/operations	Network design	.
25	Municipal building, operating and civil-defense permits	Local agencies	Conditional	Construction/operations	Type of installation	

Table 20-3: Aurora Project Permits for Construction and Operation

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
1	Authorization for mining activities: mine plan and associated components	MINEM –	Mandatory	Before exploitation	Detailed EIA	D.S. 020-2020-EM y TUPA vigente.
2	Authorization to construct mineral-processing concession facilities	MINEM –DGM	Mandatory	Before plant construction	Concession of benefaction, EIA	Plant design
3	Grant or modification of a mineral-processing concession	MINEM – DGM	Mandatory	Development /operations	Plan of operations	
4	Authorization to commission/start operation of the mineral-processing concession	MINEM – DGM	Mandatory	Before processing	Final design	
5	Authorization to commence/recommence mining operations	MINEM – DGM	Mandatory	Before production	Mining plan	
6	General mining works authorization, where applicable	MINEM	Conditional	Before use	Mine design	For ancillary mine features
7	Mining transport concession/authorization, where applicable	MINEM	Conditional	Before system operation	Mine transport design	Applicable if ore is transported offsite to a different facility
8	Authorization/management of tailings storage facility and raises	MINEM / SENACE / OEFA	Mandatory/ conditional	Construction and operations	EIA, Tailings design	
9	Authorization/management of waste dumps, stockpiles and pads	MINEM / SENACE	Mandatory/ conditional	Construction /operations	Mine plan	
10	Water-use license and metering/payment obligations	ANA	Conditional	Operations	Source and water infrastructure	Water studies, and water use payments

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
11	Discharge or reuse authorization and water-quality control	ANA / OEFA	Conditional	Operations	Treatment plant design	Compliance with water quality regulations.
12	Explosives, magazines and handling authorizations	SUCAMEC	Mandatory if used	Construction /operations	Explosives management plan	
13	Authorization for explosives or blasting-agent plant	SUCAMEC	Conditional	Before operations	Magazine design	
14	ITSE / building-safety certificate, where applicable	Local Agencies	Conditional	Before use	Final installation	For all camps, offices, and ancillary structures
15	Health authorizations for potable water, food, waste and health services	MINSA / DIGESA / DIRESA	Conditional	Before and during operations	Water purification plan	
16	Registration and management of solid/hazardous waste and manifests	MINAM / OEFA / operadores autorizados	Ongoing obligation	Construction /operations	Hazardous material management plan	Integrated Solid Waste Management Law.
17	Authorizations for transport of hazardous materials and substances	MTC / SUCAMEC / otras	Conditional	Construction /operations	Routes, vehicles, mitigation measures	Including fuel, reagents, and explosives
18	Hydrocarbon permits for storage and direct consumer facilities	OSINERGMIN / MINEM	Conditional	Before operation	Tanks, secondary containment	
19	Electrical permits, commissioning and safety	MINEM / OSINERGMIN	Conditional	Construction /operations	Electrical system completed	
20	Occupational health and safety program, committees and supervision	MINEM / SUNAFIL / OSINERGMIN	Ongoing obligation	Throughout the stage	Staff and contractors	D.S. 024-2016-EM y modificatorias.
21	Annual Consolidated Declaration (DAC) and mining statistics	MINEM	Ongoing obligation	Annual/applicable frequency		
22	Reporting of production, incidents, accidents and emergencies	MINEM / OSINERGMIN / OEFA / SUNAFIL	Ongoing obligation	Operations		
23	Environmental monitoring and commitment reporting	OEFA / SENACE / MINEM / ANA	Ongoing obligation	Construction /operations	IGA y permisos	Water, air, noise, biodiversity, social

No.	Permit / Requirement	Authority	Classification	Timing	Main Dependency	Basis / Comment
24	Update of the Mine Closure Plan and financial assurance	MINEM	Ongoing obligation	Statutory frequency / changes	Approved Plan	Includes plan and costs
25	Emergency Preparedness and Response Plan	MINEM / OEFA / OSINERGMIN	Ongoing obligation	Before and during operations	Analysis of risks	
26	Social agreements and commitments undertaken	Comunidades / Estado / titular	Ongoing obligation	Throughout project life	Agreements with community	

Managing the permitting requires a dedicated team. Section 26.0 (Recommendations) discusses the need for a permit steering committee to manage the permitting requirements in a timely and effective manner.

20.5 Social and Community Relations

20.5.1 Description of the Community

The Aurora project is located in and near the community of Parobamba. Urano (2022) conducted a social survey of the town in 2022 as part of the DIA. The study was conducted by direct surveys of the population and through coordination with the local government. The town of Parobamaba has 705 inhabitants. Forty-two percent of the town is over sixty years old, and 75% of the town is over age 45. The lack of younger people (15 to 29 year olds are 8% of the population) is directly related to the lack of local economic opportunities and the fact that the majority of working-age people migrate to cities.

The town is relatively primitive, but most inhabitants have water and electrical services. Sixty percent of the town has sewage services. There is a small two-person health clinic in the town, and a primary and secondary school. Eighteen percent of households have a television, but 82% own and use a cell phone.

Agriculture (39%), artisan work (such as weaving) (36%), and raising livestock (20%) comprise nearly all the local economic activity. Potatoes, corn, grain, beans, and other vegetables are the most-produced farm products.

20.5.2 Community Relations

DLP has been active in the community from the beginning of exploration and has been executing an active and deliberate community relations plan defined by the legal requirements (D.S. N° 052-2010-EM). As part of this, the DIA included various community meetings and community discussions. Some of the comments by community members are paraphrased and translated below by GRE:

- We ask that you use local employment and that there be more workshops.
- We ask that the roads and walkways be respected so we can move around our animals
- We ask that there is no water contamination.
- We ask to see and understand the environmental studies.
- We ask that the mine not create contamination or dust or exploit local labor.

These interviews show that the community has a high engagement level with the project already and that they understand some of the fundamental potential impacts that mining projects have on a local community, both positive and negative.

GRE believes that DLP is doing what is necessary to accomplish the key goal of a community communication plan to (paraphrased and translated from the DIA): “Establish good relations, based on mutual trust between the population and the management of the project. Furthermore, to convey relevant information in a transparent manner with the community and to respond to their concerns and recommendations.”

DLP also built the community a football (soccer) pitch (Photo 20-2).

Photo 20-2: DLP-Constructed Soccer Pitch in Parobamba, Peru

DLP also has a signed agreement with the community for ongoing exploration work (SUNARP, 2026). This community agreement has a number of provisions, highlighted below:

- Payments to the community
- Commitments to hire locally, with a specified minimum wage
- Occasional use of the DLP Excavator for civil projects
- A scholarship program for students in town to attend secondary school
- Commitments to support local agriculture including bringing in an animal husbandry expert
- Commitments to purchase from local suppliers
- Commitments to provide internet service
- Commitments to provide for local potable water treatment
- Local surface land use rights for exploration activities

This has been signed by the community leaders and DLP and is currently in place for the next six years (until 2032).

On the QP site visit, GRE observed good relations between the town and the project. The camp is in the center of town, and many locals work in the camp and in support of exploration drilling activities.

20.6 Closure

The following subsection discusses the mine closure plan and mine closure cost estimation.

20.6.1 Regulatory Requirements

Mine closure is governed by Ley No. 28090. This law requires the following:

- Long term physical stability of the post-mining landforms
- Long term chemical stability
- Rehabilitation of affected areas
- Alternative use provisions of area or installations
- Determination of the future use of land and installations.
- A financial guarantee to ensure the execution of the mine closure plan.

Regulations require an approved mine closure plan with a cost estimate. This plan is shared with local and national regulatory agencies and with the local community. The mine must also acquire guaranteed funds for a mine closure bond. This guarantees the availability of funds to cover the direct and indirect costs of closure, post-closure, supervision, contingencies, and complementary expenses.

20.6.2 PEA-Level Closure Design Concepts

At PEA level, it is necessary to make a closure plan and cost estimate, but this estimate is preliminary and subject to future revision as the project details are better-understood.

20.6.2.1 Physical Stability

Physical stability, as defined by the regulations, includes the geotechnical stability of mine waste structures and the stability of reclamation soil covers from erosion. Physical stability is equated to geotechnical stability. At the Aurora project, this will include the physical stability of slopes on the DSTF, WRSF, or other disturbed ground. Due to the nature of block-caving, it is not possible to have geotechnically stable pit slopes, but the subsidence zone must be isolated because it is inherently geotechnically unstable. Physical stability provisions also consider erosion, which must be minimized, managed, and repaired. The erosion resistance of the ET cover (see Section 20.3) is an essential part of its function and success.

To comply with this requirement, the closure plan has:

- Mine waste structures are constructed to comply with static and seismic slope stability standards
- 3:1 H:V slopes on mine waste facilities
- Appropriate surface water management
- Erosion control best management Practices (BMPs) for the retention of soil covers
- Mine portals and shafts must be sealed

20.6.2.2 Chemical Stability

Chemical stability is defined as a facility which, with both runoff and leachate controls, does not have a risk of impacting surface or groundwater quality. Because the site has demonstrated ARD risk, chemical stability is a primary driver for the closure activities.

<u>Chemical Stability of Mine Waste Facilities</u>

The best way to manage the chemical stability of a dry stack tailings facility or a waste rock dump containing acid-generating rocks is with an ET Cover. These facilities are also called Engineered Covers in

the GARD guide (INAP, 2009). The Aurora site lies at the uppermost limit of the function of ET covers with ~1 meter of precipitation per year. It is unlikely that the ET cover at Aurora will be zero-discharge – a level attainable at many sites (Albright, 2003), but it can substantially reduce the infiltration and thereby reduce the long-term production of MIW.

Furthermore, the Mestizamayo river is in a steep valley with "gaining stream" behavior along its full length. Gaining streams are streams which have a continual and increasing discharge of groundwater into surface water along their length. Recharge from the upland areas flows through short and medium length groundwater flow paths to discharge in the stream. The presence of mine waste will decrease but not end this behavior. As a result, even if the ET cover were 100% effective, seeps, springs, and groundwater discharge would create an underflow beneath the WRSF.

Based on current testing, the Aurora site has no obvious sources of geochemically inert rock nor acid-consuming rock. Therefore, GRE must assume that the seepage from beneath the WRSF and DSTF upon closure is acidic MIW requiring treatment.

Chemical Stability of the Underground Mine and Block Cave Subsidence Zone

The underground mine is located between two meandering reaches of the Mapacho River, a major regional watercourse. Most of the workings are substantially below the river level, with the block cave tunnels ~350 m below the river. Figure 20-8 shows the configuration upon closure.

Due to the unstable and rubble-filled subsidence zone, there is a portion of the rock which will permit oxygen penetration, will capture rainfall in a "funnel effect", and will be inaccessible due to safety issues. This rubble zone will have sulfide oxidation and ARD production that cannot easily be mitigated. The water will report to the Block Cave development area, and ultimately, will fill up the underground workings and report to either the portals or fractures in the bedrock. This will create a groundwater "mound" due to the high recharge of the funnel effect and will eventually discharge laterally to the river (through rock fractures) or out of the portal (once the workings fill with water). This is an unacceptable post-closure condition because the discharging water will likely not comport with Peruvian water quality discharge standards.

As a result, the closure plan includes perpetual drainage and treatment of water from the underground mine.

20.6.2.3 Demolition

All mine facilities must be demolished, and the construction debris must be managed. Due to the remote location, GRE assumes that all facilities, buildings, and foundations must be demolished.

20.6.2.4 Reclamation

Rehabilitation of the land is part of the law. The mine must define a post-mining land use and must reclaim the disturbed ground to meet that land use. GRE has assumed wildlife habitat and livestock grazing as the post-mining land use. Agriculture, although important in the region, was not selected as the post-mining land use because the irrigation of crops decreases the effectiveness of an ET cover, whereas a healthy cover of native grasses is essential to stabilize soil and improve water uptake through plant transpiration. Reclamation will include topsoil placement and the placement of a native-seed grass mixture.

There are provisions in the law which require progressive closure of impacted areas, if possible. Closure regulation in Peru require that Aurora evaluate if existing infrastructure can be re-purposed for local uses. Frequently, this is when a camp is converted to housing, offices into commercial property, or mine workshops are converted into local workshops. Many features, such as crushing, grinding, and flotation, will not have cross-over use, and will need to be demolished.

A period of post-closure monitoring is required to determine if the objectives of the closure plan are met. This includes environmental monitoring (water, air, soil), revegetation monitoring (to ensure a diverse local grass cover, for example), and the correction of any observed erosion on reclaimed areas.

20.6.2.5 Closure Design Criteria for Earthworks

The following closure design criteria have been applied.

DSTF

Maximum slope of 3:1 H:V to ensure geotechnical stability. Because it is constructed for closure, the DSTF requires minimal post-closure regrading.

Evapotranspiration cover of 1-m will be applied. The surface of the DSTF will have surface water diversions around the perimeter and surface water drainage channels on the surface. Where necessary, channels will have appropriate erosion protection. While all of the placed ET cover topsoil will have erosion control best management practices, an extra $1/m^2 erosion protection cost has been added to sloped areas to allow for enhanced erosion control. The cover will be ripped to relieve compaction and revegetated with a native grass seed mixture.

WRSF

The WRSF has nearly the same design criteria as the DSTF. The only significant difference is that the WRSF requires $1M in regrading to meet the stability requirements. The WRSF has the same evapotranspiration cover, erosion control, and revegetation criteria.

Pit

The pit cannot be reclaimed because of the subsidence zone area. Upon closure, a high fence will be constructed to keep out people and livestock.

20.6.3 Water Management

The closure design includes a surface water management plan which minimizes the contact between natural water and reclaimed facilities. Because long-term treatment is required, and because the seasonal production of MIW is uneven, GRE has included a treatment plant surge pond in the closure plan. This lined pond will be located atop the WRSF in the lower reaches of the Mestizamayo drainage basin. It will store wet-season MIW for dry season water treatment, thus reducing the total overall capacity required of the treatment plant. The pond at the toe of the DSTF will serve as the collection point for the water from the DSTF and the WRSF. The underground workings will also be pumped during closure. All MIW from the Mestizamayo and the underground mine will be treated prior to discharge.

20.6.4 Water Treatment

The closure plan will include the ongoing function of the water treatment plant. This plant is required during operations and will continue the same function upon closure. It is the largest part of the mine infrastructure that will remain post-closure. Sludge from the WTP will be dumped into the subsidence area of the open pit, where there are no foreseeable capacity limits, and where the addition of ferric iron will assist in immobilizing arsenic and other regulated metals.

20.6.5 Post-Closure Monitoring

Under Peruvian regulations, there is no fixed period for post-closure monitoring, maintenance, or management. Post closure activities should continue for as long as required to meet closure objectives. GRE therefore recommends a 5-year period for the monitoring of revegetation, erosion control, and site wide water conditions. However, influent and effluent monitoring will be required for the post-closure water treatment as long as it is running (which is expected to be in perpetuity). The Net Present Value discount factor makes post-closure monitoring past 20 years a negligible cost, and as a result, a 20-year period was selected.

20.6.6 Unit Rate Estimates

GRE has created a cost estimate of all closure activities. GRE utilized the best-available unit rates for key reclamation tasks based on the following (in order of preference):

- Actual unit rates taken directly from observed costs at Aurora
- Actual unit rates taken directly from similar operations in Peru
- Actual unit rates taken from other operations within Latin America
- Calculated unit rates from first-principals based on engineering calculations
- US-based industry costing guidelines like InfoMine or RS Means
- Values taken from interviews with professionals in Latin America with reclamation experience
- GRE's professional judgement

GRE has worked with the Aurora mine to acquire the best-possible unit rates for key activities. GRE's effort in acquiring unit rates was proportional to each unit rate's potential impact to the total cost. For example, large items like revegetation had more engineering rigor than small items like the cost of small portal plugs.

20.6.7 Closure Cost Estimate

Section 21.0 discusses the closure cost estimate and the timeline for closure expenses.

21.0 CAPITAL AND OPERATING COSTS

21.1 Basis of Estimate and Accuracy Classification

The capital cost estimate meets the requirements for a PEA, consistent with NI 43-101 reporting requirements and International Association for the Advancement of Cost Engineering (AACE) cost estimating guidelines for a Class 5 estimate. The estimate accuracy range of ± 30% is defined by the level of project definition, the amount of engineering inputs, the time available to prepare the estimate, and the amount of project cost data available.

21.2 Capital Cost Estimate

The capital cost estimate has been prepared for the PEA under the assumption of processing of open pit mined mineralized material at a design rate of approximately 65,000 tpd. Project costs were estimated using GRE, Ausenco, and SRK in-house data, cost data from Infomine (2025), manufacturer equipment quotes, and experience of senior staff. The estimate assumes that the Project will be operated by the owner with purchased equipment.

The initial capital costs are incurred in the years prior to processing, i.e., Years -3 to -1. The QPs expect there will be five to seven years of continued exploration, engineering, and permitting prior to a production decision.

Initial capital costs are defined as all capital costs until processing starts. This includes labor and development costs in the pre-production years. Sustaining capital is defined as the capital costs incurred in the periods after processing begins.

All capital costs cited in this Report are referenced in US dollars with an effective date of April 30, 2026.

Capital cost estimates were prepared based on current and expected long-term pricing assumptions and to a PEA level of ±35% level of accuracy.

The capital costs are summarized in Table 21-1**.**

Table 21-1: Aurora Project Capital Cost Summary ($millions)

Item	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Total
OP Mine Capitalized Opex	$93.85	$117.29	$118.56	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$329.70**
OP Mine Equipment Capex	$197.66	$3.81	$10.95	$0.03	$0.03	$0.03	$0.03	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$212.53**
OP Mine Equipment Salvage	$0.00	$0.00	($11.18)	($0.35)	$0.00	$0.00	($42.58)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**($54.11)**
UG Mine Capitalized Opex	$17.48	$20.83	$23.60	$28.41	$35.38	$51.55	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$177.24**
UG Capex	$69.29	$39.07	$32.18	$49.04	$69.65	$99.28	$120.03	$116.52	$107.45	$47.44	$7.78	$15.12	$23.82	$12.83	$16.85	$10.74	$14.98	$5.04	$3.96	$3.96	$0.00	$0.00	$0.00	**$865.01**
Plant Capex	$50.21	$225.93	$225.93	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$502.06**
Tailings Capex	$66.42	$132.84	$132.84	$0.00	$25.36	$0.00	$32.92	$0.00	$0.00	$0.00	$35.58	$0.00	$0.00	$0.00	$31.59	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$457.55**
Infrastructure Capex - Ausenco	$43.83	$87.65	$87.65	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$219.13**
Infrastructure Capex – GRE	$57.65	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$57.65**
Indirects Capex	$27.53	$55.05	$55.05	$0.00	$0.27	$0.00	$0.35	$0.00	$0.00	$0.00	$0.38	$0.00	$0.00	$0.00	$0.34	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$138.97**
Closure and Water Treatment	$4.00	$0.00	$0.00	$0.00	$35.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$14.91	$14.91	**$68.81**
Owner's Costs	$12.67	$12.67	$23.17	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	$1.29	($33.71)	$0.00	**$38.00**
G&A Capitalized Opex	$3.50	$7.01	$17.52	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$28.04**
Working Capital	$0.00	$0.00	$0.00	$24.45	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	($24.45)	$0.00	**$0.00**
Sustaining Capital	$0.00	$0.00	$0.00	$0.00	$22.36	$0.69	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	**$23.06**
Contingency	$98.79	$137.21	$136.27	$12.36	$31.65	$24.62	$31.40	$28.27	$32.52	$11.00	$1.08	$2.92	$5.09	$2.35	$3.35	$1.82	$2.88	$0.40	$0.13	$0.13	$0.13	($0.39)	$2.98	**$566.97**
Total Capital Costs	**$742.87**	**$839.36**	**$852.53**	**$116.19**	**$221.00**	**$177.47**	**$143.43**	**$146.08**	**$141.26**	**$59.73**	**$46.11**	**$19.32**	**$30.20**	**$16.47**	**$53.42**	**$13.85**	**$19.15**	**$6.73**	**$5.38**	**$5.38**	**$1.42**	**($43.65)**	**$17.89**	**$3,630.62**

21.2.1 Open Pit Mining

Open pit mining capital costs include production and support equipment, which is assumed to be owner purchased, mining-related office equipment, initial tire purchase, and initial spare and maintenance parts purchase. The open pit mining capital costs for the life of the project are shown in Table 21-2.

Table 21-2: Aurora Project Open Pit Mining Capital Cost Summary ($millions)

Item	Quantity Required	Capital Costs ($millions)
Equipment		
Production Equipment		
Hydraulic Shovel CAT 6040	3	$25.44
Haul Truck CAT 793F	21	$97.41
Bulldozer CAT D10T	5	$10.92
Blast Hole Drill	2	$7.56
Support Equipment		
Loader CAT 993K	2	$5.81
Loader CAT 992K	1	$2.30
Bulldozer CAT D9T	3	$5.33
Bulldozer CAT D6T	1	$0.80
Wheel Dozer CAT 854K	2	$6.90
Water Truck CAT 777G	3	$5.83
ANFO Truck	2	$0.80
Fuel/Lube Truck	2	$2.21
Mechanic Truck	2	$0.75
Grader CAT 24M3	2	$5.26
Compactor CAT CP-56B	2	$0.69
Blast Hole Drill	2	$7.56
Backhoe CAT 416	1	$0.13
Crane	2	$2.74
4x4 Pickup	20	$1.53
Dewatering Pump	10	$4.84
Generator	1	$0.03
Light Plants	20	$0.95
Other Mining-Related Equipment		
Surveying Equipment	2	$0.08
Computers	20	$0.06
Operations Software	1	$0.09
Planning Software	1	$0.14
Geology Software	1	$0.09
Maintenance Software	1	$0.17
Dispatch System	1	$0.10
Plotter	1	$0.01
Radios	20	$0.01
First Fills/Spare Parts		
Tires	278	$6.86
Parts		$9.15
Total Mining Equipment Capital Costs		**$212.53**

21.2.2 Underground Mining

The capital cost estimate for underground mining was undertaken by SRK, including:

- Underground mine development
- Underground block cave construction
- Underground mine infrastructure
- Underground mobile equipment

The underground mining capital costs were estimated using a combination of first-principles calculations, quotations from mining contractors and equipment vendors, SRK's relevant project experience, and factored costs. The costs were developed for an Owner operating scenario with contractor use only for vertical development.

The mining capital cost estimate was based on the following:

- Preliminary project development plan
- Mining equipment list
- Factored quotes for the major mobile equipment obtained from equipment manufacturers
- Factored quotes from mining contractors (e.g., vertical development)
- SRK in-house database

Lateral development costs were estimated from first principles. The raiseboring quotes for vertical development include fixed costs for mobilization to the project site and demobilization, drill setup and teardown, reamer installation, and variable costs, which include pilot hole drilling and reaming to the final size of a raise. Additional fixed costs were added for raise collar and drilling platform construction on the surface and underground excavation for the raisebore chambers and reamers.

Block cave footprint construction cost estimates were based on a combination of first principles for undercut and drawbell drilling, blasting, mucking and hauling, and SRK internal benchmarking and experience with drawbell and footprint construction costs.

Underground infrastructure costs were based on the preliminary estimated amount of required underground infrastructure and indicative costs for underground infrastructure in similar projects. The underground mine infrastructure costs include equipment purchase and installation.

Mobile equipment costs were developed from estimated fleet requirements and factored vendor budgetary quotations. Mobile equipment fleet was estimated as described in Section 16.2.5. The mobile equipment schedule was prepared on an annual basis. The required underground mobile equipment fleet will vary over the life of the project.

The underground block cave mining capital costs for the life of the project are shown in Table 21-3.

Table 21-3: Aurora Project Underground Block Cave Mining Capital Cost Summary ($millions)

Item	Capital Costs ($millions)
Surface Infrastructure and Contractor Indirects	
Contractor Mobilization and Indirects	$3.08
Raises Collar Construction	$2.00
Decline Portal Construction	$5.00
Underground Mine Development	
Lateral Capital Development	$150.05
Mass Excavations	$8.42
Drawpoint Construction	$323.07
Vertical Development	$29.22
Mine Infrastructure	
Mine Ventilation System	$21.30
Primary Crushers	$27.30
Conveyor System	$29.41
Underground Water Management System	$11.90
Electrical Power Distribution and Control	$34.18
Maintenance Shop	$5.83
Safety	$2.65
Mine Technology & Engineering Support Equipment	$5.33
Underground Mobile Equipment Purchase	
Development and Production Equipment	
Jumbo, 2 boom	$3.26
Eumulsion Loader, Development	$1.32
Scaler	$2.13
Rockbolter	$6.69
Cablebolter	$4.34
Shotcrete Sprayer	$1.74
Production Drill	$5.16
ITH Drill with V30 Reamer	$2.83
Emulsion Loader, Production	$2.15
Water Cannon	$3.05
Blockholer	$6.47
Mobile Rockbreaker	$24.73
Hydrofracturing Equipment	$0.00
LHD, 7.0 m3, 14 t	$78.08
LHD, 8.6 m3, 17 t	$4.82
LH514 Onboard Automation Kit	$7.76
Haulage Truck, 51 t	$7.61
Auxiliary Equipment	
Grader	$5.37
Scissor Lift	$5.29
Scissor Lift Attachments	$0.34
Boom Truck	$3.18
Flat Deck Truck	$2.71
Fuel/Lube Truck	$4.28

Item	Capital Costs ($millions)
Explosives Truck	$1.30
Transmixer	$5.82
Forklift/Telehandler	$0.90
Water Sprayer	$1.79
Roller	$0.99
Backhoe	$0.86
Bobcat	$0.83
Mechanics Truck	$0.55
Personnel Carrier MacLean (22 person)	$2.42
Personnel Carrier Toyota	$0.36
Supervisor/Engineering Vehicle	$2.06
Cassettes System Prime Mover	$2.68
Cassettes Attachments	$2.42
Subtotal UG Mining Capital Costs	$206.28
Capitalized OPEX for Initial Capital Period	$177.24
Grand Total Underground Project Capital Costs	**$1,042.26**

21.2.3 Process Plant

The concentrator is assumed to require 24 to 36 months for construction with capital payments being made as equipment is ordered and delivered. The concentrator capital costs are shown in Table 21-4.

Table 21-4: Aurora Project Concentrator Capital Cost Summary ($million)

Item	Capital Costs ($millions)
Equipment	$314.81
Spares and First Fill	$17.24
Engineering & Design	$89.48
Construction Management	$65.08
Taxes/Insurance	$15.46
Total Concentrator Capital Costs	**$502.06**

21.2.4 Dry Stack Tailings Facility

The Aurora Project will include placement of filtered tailings in a dry stack tailings facility (DSTF). Costs for the DSTF include the cost of the filter plant, tailings slurry line, and water reclaim line. Costs also include earthworks to initially build the DSTF and associated water management infrastructure. The DSTF capital costs are shown in Table 21-5.

Table 21-5: Aurora Project Dry Stack Tailings Facility Capital Cost Summary ($millions)

Item	Capital Costs ($millions)
Tailings Disposal & Reclaim Water System	$7.25
Filter Plant	$210.92
Dry Tailings Deposition	$87.85

Item	Capital Costs ($millions)
Non-Contact Water Management Infrastructure	$14.30
Contact Water Management Infrastructure	$11.78
Total Tailings Capital Costs	**$332.10**

21.2.5 Site Infrastructure

In support of the major installation construction contracts, engineering for the project was completed to a PEA-level of definition. Bulk material quantities were derived for earthworks, including the entire project site, process plant, primary crusher pad, explosive storage pad, filter plant pad and all access roads and were priced from other benchmark projects.

The site infrastructure also covers the cost of the rope conveyor system, the permanent camp, administrative facilities, mine shop, fuel bay, magazines, and water supply.

The capital costs are shown in Table 21-6.

Table 21-6: Aurora Project Site Infrastructure Capital Cost Summary ($millions)

Item	Capital Costs ($millions)
General Site Development	$63.96
Permanent Camp	$13.22
Rope Conveyors	$65.67
Main Access Road	$8.40
Total Site Infrastructure Capital Costs	**$151.24**

Note: Total may not add up due to rounding

21.2.6 Power Distribution

The power requirements for the project were developed for the process plant, filter plant, and mining operations. The cost of the high voltage power supply, substation and cable routing were estimated based on benchmarked South American projects. The capital costs are shown in Table 21-7.

Table 21-7: Aurora Project Power Supply Capital Cost Summary ($millions)

Item	Capital Costs ($millions)
Power Distribution	$46.62
Power Supply	$21.27
Total Power Supply Capital Costs	**$67.89**

Note: Total may not add up due to rounding

21.2.7 Facility Infrastructure

All buildings and associated infrastructure installed on a permanent or semi-permanent basis are considered infrastructure. They include material and installation costs. These costs are incurred in Years -3, -2, and -1. Each item's capital cost was estimated based on knowledge of nearby mine operations or senior engineers' experience. Table 21-8 shows total estimated costs for each infrastructure item.

Table 21-8: Aurora Project Facility Infrastructure Capital Cost Summary ($millions)

Item	Capital Costs ($millions)
Office	$1.28
Warehouse	$6.00
Mine Shop	$16.50
Fuel Bay	$2.50
Wash Bay	$4.88
Cap Magazine and ANFO Storage Bin	$1.50
Water Supply	$25.00
Total Facility Infrastructure Capital Costs	**$57.65**

Note: Total may not add up due to rounding

21.2.8 Water Treatment Plant and Closure

Section 18.5 and Section 20.0 discuss the need to treat MIW coming from the leachate of the DSTF, WRSF, open pit, and underground workings. GRE calculated the CapEx for the WTP including tanks, mixers, lime dosing systems, and pumps using the same methodology that was applied to the processing plant.

For the WTP OpEx, GRE considered reagent consumption (lime), labor, energy, and the cost of sludge management. Sludge management during operations will involve blending the sludge into the tailings feed on the way to the filter presses. WTP sludge will be dewatered along with the tailings and placed in the DSTF. During post-closure, sludge will not be filtered, instead, it will be dumped into the open pit subsidence zone.

Aurora Project closure costs were calculated using the existing end-of-mine fleet and the until rates discussed elsewhere in this section. The largest costs pertain to the placement of the ET covers on the DSTF and WRSF, as well as the ongoing water treatment of MIW coming from the block cave subsidence area and from the seepage under the DSTF and WRSF. Closure costs have been applied appropriately in the closure timeline, and the value listed in Table 21-9 is a total cost, but not a NPV-adjusted total cost.

Table 21-9 shows total estimated costs for water treatment plant construction and closure.

Table 21-9: Aurora Project Water Treatment Plant and Closure Cost Summary ($millions)

Item	Capital Cost ($millions)
Permitting	$2.00
Environmental Monitoring Network	$2.00
Water Treatment Plant	$35.00
Closure	$70.00
Total Water Treatment Plant and Closure Capital Costs	**$109.00**

21.2.9 Indirect Costs

Indirect capital costs are calculated as a percentage of the direct costs for the infrastructure and tailings. The indirect capital costs are summarized in Table 21-10 and described below.

Project indirect costs include the following:

- Temporary construction facilities and services
- Accommodation and catering during construction
- Contractor distributable costs
- Vendor reps and assistance
- Equipment spares
- First fills and initial charges
- Project delivery
- Contingency.

Table 21-10: Aurora Project Indirect Costs Summary ($millions)

Item	Capital Cost ($millions)
Indirects	
Field Indirects	$22.05
Accommodation and Catering - During Construction	$26.47
Miscellaneous Distributable Cost	$5.51
Vendor Representatives	$4.99
Spares - Commissioning & Initial	$7.48
First Fill and Initial Charges	$4.99
Project Delivery/EPCM	
EPCM Expenses	$11.02
Engineering & Procurement	$27.56
Construction & Project Management	$27.56
Total Indirects	**$137.64**

21.2.10 Owner's Costs

Owner's costs include include training, project management, bonding, etc. The total Owner's costs are summarized in Table 21-11.

Table 21-11: Aurora Project Owner's Costs

Item	Capital Cost ($millions)
Startup Training	$3.00
Project Management	$1.00
Drilling and Met Testing	$10.00
Feasibility Study	$6.00
Construction Insurance	$8.00
Commissioning and Start-up	$10.00
Reclamation Bond*	$10.50
Total Cost	**$48.50**

* Bond costs are recovered at the end of the project.

21.2.11 Sustaining and Development Capital

Sustaining costs include equipment replacement costs, totaling $32 million, which are incurred throughout the life of the project, and DSTF sustaining capital, totaling $151.6 million, which are incurred in years 2, 4, 8, and 12.

21.2.12 Working Capital

Working capital is the necessary cash on hand for the next period's operating cost. The estimated total is $24.75 million. This cost is recovered at the end of production.

21.2.13 Contingency

Contingency varying from 10% to 25% was applied on all capital items, totaling $566.97 million.

21.3 Operating Cost Estimate

The operating costs assume owner operation. Operating costs are summarized in Table 21-12.

Table 21-12: Aurora Project Operating Cost Summary

Item	Total Operating Costs (million$)	Unit Operating Cost ($/tonne mined)[1]	Unit Operating Cost ($/tonne processed)[2]
OP Mine Opex	$796	$1.72	$1.08
UG Opex	$1,142	$6.90	$2.35
Process Opex	$2,220		$5.51
Tailings Opex	$718		$1.78
Water Treatment Opex	$29		$0.07
G&A Opex	$429		$1.07
Contingency	$480		$1.19
Total	**$5,814**		**$13.05**

[1] - Total mine operating costs (including capitalized pre-production operating costs) per tonne mined

[2] - Operating costs less capitalized pre-production costs per total tonnes milled

21.3.1 Open Pit Mining

Open pit mining operating costs include production and support equipment hourly costs, blasting costs, and labor.

Production and support equipment operating costs were estimated from InfoMine (2025) data and include overhaul parts, maintenance parts, diesel fuel, diesel exhaust fluid, electric power (where applicable), lube, tires, and wear parts. In lieu of the standard $0.71/liter diesel and $0.0886 dollars per kilowatt-hour ($/kWhr) unit rates for diesel and electricity, respectively, included in the InfoMine data, the GRE QP substituted site-specific rates of $1.2/liter for diesel and $0.11/kWhr for electricity.

- Mining production equipment hours were estimated from equipment productivity estimates, including haul truck cycle time analyses, and the number of tonnes required to be moved. It was assumed that all mining will be owner-operated.

- Mining support equipment hours were calculated using the number of shifts that each piece of equipment was estimated to operate each day, the number of pieces of equipment, and the operating hours per day. Operating hours per day were calculated assuming 90% utilization, 85% availability, two twelve-hour shifts per day, and one-hour fixed delay per shift.

Blasting costs include ANFO, caps, primers, ore control testing, and miscellaneous yearly costs. Unit rates were obtained from InfoMine (2025), with the exception of ANFO. Because the price of ammonium nitrate is substantially lower in South America than in North America, the GRE QP used a unit rate of ½ the InfoMine rate.

Labor includes hourly rate employees and salaried employees.

- Hourly rate employees include equipment operators, haul truck drivers, laborers, mechanics, blasters, and helpers. Hourly rates were obtained from online searches and comparison with other operating mines in Peru. All hourly rate employees were assumed to be local, Peruvian labor. To estimate numbers of hourly rate employees needed, the GRE QP estimated the number that would be required in North America and then doubled that number. The burden rate for Peruvian labor was assumed to be 100%.
- Salaried employees include superintendents, foremen, clerks, engineers, geologists, surveyors, and modelers. These employees were all assumed to be expatriate (originating from North America or other non-Peruvian locales). The burden rate for expatriate employees was assumed to be 40%.

The open pit mining operating costs are summarized in Table 21-13.

Table 21-13: Aurora Project Open Pit Mining Operating Cost Summary (million $)

Item	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Total
Equipment Operation								
OP Mine Production Equipment	$45.26	$65.49	$67.25	$65.33	$64.69	$64.13	$64.74	**$436.90**
OP Mine Support Equipment	$25.78	$25.78	$25.78	$25.78	$25.78	$25.78	$25.30	**$179.95**
OP Mine Equipment Contingency	$7.10	$9.13	$9.30	$9.11	$9.05	$8.99	$9.00	**$61.68**
Total OP Mining Equipment Operating Costs with Contingency	**$78.14**	**$100.40**	**$102.33**	**$100.22**	**$99.51**	**$98.90**	**$99.03**	**$678.53**
Blasting								
ANFO	$5.98	$7.72	$7.28	$8.13	$8.30	$8.06	$8.31	**$53.76**
Caps	$0.11	$0.15	$0.14	$0.15	$0.16	$0.15	$0.16	**$1.01**
Primers	$0.13	$0.16	$0.15	$0.17	$0.18	$0.17	$0.18	**$1.14**
Ore Control Testing	$2.72	$3.46	$3.48	$3.42	$3.30	$3.31	$3.39	**$23.09**
Miscellaneous	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$0.98	**$6.98**
Blasting Contingency	$0.99	$1.25	$1.21	$1.29	$1.29	$1.27	$1.30	**$8.60**
Total Blasting Costs with Contingency	**$10.93**	**$13.74**	**$13.26**	**$14.16**	**$14.23**	**$13.96**	**$14.31**	**$94.59**

Item	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Total
Labor								
OP Mine Hourly Labor	$11.65	$12.30	$12.25	$12.25	$12.25	$12.01	$12.02	**$84.73**
OP Mine Salaried Labor	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.20	**$8.57**
OP Mine Labor Contingency	$1.29	$1.35	$1.35	$1.35	$1.35	$1.32	$1.32	**$9.33**
Total OP Mine Labor Costs with Contingency	**$14.16**	**$14.89**	**$14.82**	**$14.82**	**$14.82**	**$14.57**	**$14.55**	**$102.63**
Total OP Mine Operating Costs with Contingency	**$103.24**	**$129.02**	**$130.41**	**$129.20**	**$128.56**	**$127.42**	**$127.89**	**$875.75**
Capitalized OP Opex for Initial Capital with Contingency	($103.24)	($129.02)	($130.41)	$0.00	$0.00	$0.00	$0.00	**($362.67)**
OP Mining Opex with Contingency (less capital allocation)	**$0.00**	**$0.00**	**$0.00**	**$129.20**	**$128.56**	**$127.42**	**$127.89**	**$513.08**

21.3.2 Underground Block Cave Mining

The operating cost estimate for underground mining was undertaken by SRK. The underground mining operating costs generated were based on an annual LOM schedule for full owner operating scenario. The operating costs have been estimated as a combination of first-principles calculations, experience, and factored costs. Productivity, equipment operating hours, labor and supply requirements and costs were calculated for each mining activity.

Production mucking cost includes manual mucking from the block cave drawpoints direct to the on-level crusher tip by 14-t LHDs.

Secondary breaking cost estimates were based on the assumed number of hang-ups and oversize at drawpoints and amount of oversize material to be broken to pass through the grizzly openings. Based on the geotechnical assessment, approximately 8.5% of the rock blasted would require secondary blasting. Secondary breakage activities require one water cannon, two blockholers and nine mobile rock breakers to manage hang-ups and drawpoint oversize. A stationary rockbreaker will be installed on the top of each crusher station.

Mine services and maintenance costs include operating costs of ancillary mobile equipment, road maintenance, mine safety, power reticulation, controls, and communication, mine technology, general mine services, and miscellaneous costs.

Annual underground infrastructure operating costs were based on the preliminary estimated amount of required underground infrastructure and indicative costs of underground infrastructure for similar projects. Where applicable, the underground infrastructure operating costs include power, replacement parts, and related shop supplies.

Crushing cost includes cost per tonne of underground crushing applied to the ore tonnage to be conveyed from underground. Operating costs per year for each conveyor were estimated based on the conveyor type, belt width and rating, length, lift, and power requirements.

Ventilation and dewatering costs were estimated based on estimated power consumption and allowance for equipment maintenance.

An allowance for mine rehabilitation was made to account for drawpoint and general mine rehabilitation including installation of bulkheads to isolate mined-out production levels.

It was assumed that geotechnical and definition drilling will be performed by a drilling contractor. The geotechnical and definition drilling costs were estimated based on the assumptions made for the geotechnical and definition drilling requirements and the contractor's cost per meter of drilling.

Mine labor costs were estimated as separate cost items based on the annual labor rates benchmarked by SRK and an underground rotating roster estimated by SRK based on 12-hour shift schedule and equivalent time on and off crew rotation.

Material costs were based on estimated supply consumption and prices of consumables obtained from the suppliers. Maintenance consumables, such as parts, tires, fuel, and lube were included in equipment operating costs. Costs for consumables are included in the mine development, production, services, and other underground mining costs.

The underground block cave mining operating costs are summarized in Table 21-14.

21.3.3 Concentrator Plant

The processing operating costs include labor, reagents and consumables, and power. Table 21-15 shows the estimated labor, consumable, and power costs. The total unit rate for processing is $6.04 per tonne of material processed.

21.3.4 Dry Stack Tailings Facility

The DSTF operating cost estimate is based on a 65,000 tpd mill, consisting of tailings handling, filter plant operation, and stacking dry stack. The tailings operating costs include labor, consumables, power, equipment maintenance, and dry stacking equipment. The operating costs are estimated at $1.94 per tonne of material processed.

The DSTF operating costs are summarized in Table 21-16.

21.3.5 Water Treatment

Water treatment operating costs include a mine WTP constructed by mine year 3 to process up to 300 L/s of MIW (see Section 18.0). This treatment plant would be a High-Density Sludge active water treatment system. Sludge from the WTP will be added to the tailings stream and would ultimately be dewatered in the filter plant and placed on the DSTF – co disposed with the tailings.

The water treatment operating costs are summarized in Table 21-17.

Table 21-14: Aurora Project Underground Block Cave Mining Operating Cost Summary (million $)

Item	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Total
Production Mucking	$0.00	$0.00	$0.00	$0.00	$0.00	$0.52	$3.92	$6.87	$7.06	$11.79	$14.12	$14.12	$14.12	$14.12	$14.12	$14.12	$14.12	$14.12	$14.12	$5.19	**$162.47**
Secondary Breaking	$0.00	$0.00	$0.00	$0.00	$0.00	$0.54	$4.01	$7.02	$7.23	$6.05	$7.25	$7.25	$7.25	$4.69	$4.69	$4.69	$4.69	$4.69	$4.69	$1.73	**$76.47**
Crushing & Conveying	$0.00	$0.00	$0.00	$0.00	$0.00	$9.77	$9.78	$9.77	$9.77	$9.77	$9.78	$9.77	$9.77	$9.77	$9.78	$9.77	$9.77	$9.77	$9.78	$9.77	**$146.59**
Ventilation	$1.15	$1.92	$4.09	$5.89	$5.89	$5.75	$5.76	$4.33	$4.33	$4.33	$4.12	$4.11	$4.11	$4.11	$4.12	$4.11	$4.11	$4.11	$3.83	$3.82	**$84.03**
Dewatering	$0.45	$0.90	$1.63	$2.08	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	$2.80	**$49.86**
Electrical	$0.16	$0.31	$0.47	$0.62	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	$0.78	**$13.98**
Mine Services	$5.07	$5.85	$6.29	$7.47	$10.93	$11.46	$11.48	$11.46	$11.46	$11.46	$10.46	$10.44	$10.44	$10.44	$10.46	$10.44	$10.44	$10.44	$10.46	$10.44	**$197.41**
Mine Rehabilitation	$0.00	$0.00	$0.00	$0.00	$0.00	$0.20	$0.30	$0.50	$1.00	$1.30	$1.60	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	$2.00	**$22.90**
Definition and Geotechnical Drilling	$0.52	$1.21	$0.50	$0.29	$1.06	$0.50	$0.75	$0.75	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	**$17.58**
Mine Management and Technical Staff	$6.55	$6.75	$6.60	$7.78	$7.94	$10.67	$11.03	$11.03	$11.03	$11.03	$9.64	$9.64	$9.64	$9.64	$9.64	$9.64	$9.64	$9.64	$9.64	$9.64	**$186.74**
Mine Operating and Maintenance Labor	$3.58	$3.89	$4.02	$4.27	$5.99	$8.57	$9.96	$11.03	$11.57	$11.53	$11.69	$11.69	$11.69	$11.00	$11.00	$11.00	$11.00	$11.00	$10.54	$9.32	**$184.37**
Contingency	$1.75	$2.08	$2.36	$2.84	$3.54	$5.15	$6.06	$6.63	$6.80	$7.18	$7.32	$7.36	$7.36	$7.04	$7.04	$7.04	$7.04	$7.04	$6.96	$5.65	**$114.24**
Total UG Mining Operating Costs with Contingency	**$19.23**	**$22.92**	**$25.96**	**$31.25**	**$38.92**	**$56.70**	**$66.64**	**$72.97**	**$74.82**	**$79.01**	**$80.57**	**$80.96**	**$80.96**	**$77.39**	**$77.44**	**$77.39**	**$77.39**	**$77.39**	**$76.61**	**$62.14**	**$1,256.64**
Capitalized UG Opex for Initial Capital with Contingency	-19.23	-22.92	-25.96	-31.25	-38.92	-56.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**($194.97)**
UG Mining Opex (less capital allocation) with Contingency	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$66.64**	**$72.97**	**$74.82**	**$79.01**	**$80.57**	**$80.96**	**$80.96**	**$77.39**	**$77.44**	**$77.39**	**$77.39**	**$77.39**	**$76.61**	**$62.14**	**$1,061.67**

Table 21-15: Aurora Project Concentrator Operating Cost Summary (million $)

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Total
Labor	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$4.91	$0.90	**$84.40**
Reagents and Consumables	$62.25	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$77.82	$14.20	**$1,321.55**
Power	$32.27	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$40.33	$7.36	**$684.95**
Contingency	$9.94	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$12.31	$2.25	**$209.09**
Total Plant Opex with Contingency	**$109.38**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$123.06**	**$22.46**	**$2,100.84**
Rehandle with Contingency	$0.00	$0.00	$0.00	$0.00	$15.64	$15.45	$10.95	$8.72	$8.72	$8.72	$8.72	$8.72	$8.72	$8.72	$8.72	$8.72	$17.23	$4.05	**$141.78**
Total Process Opex with Contingency	**$109.38**	**$123.06**	**$123.06**	**$123.06**	**$138.70**	**$138.51**	**$134.01**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$131.78**	**$140.29**	**$26.51**	**$2,242.62**

Table 21-16: Aurora Project DSTF Operating Cost Summary (million $)

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Total
Power	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$19.17	$3.50	**$329.36**
Consumables	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$2.22	$0.40	**$38.19**
Maintenance	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$6.92	$1.11	**$118.68**
Labor	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.81	$0.15	**$13.96**
DSTF	$12.56	$12.56	$12.56	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$12.70	$2.29	**$217.73**
Contingency	$4.17	$4.17	$4.17	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$4.18	$0.74	**$71.79**
Total Tailings Opex with Contingency	**$45.85**	**$45.85**	**$45.85**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$46.00**	**$8.19**	**$789.72**

Table 21-17: Aurora Project Water Treatment Operating Cost Summary (million $)

Item	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Total
Water Treatment	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.83	$0.83	$0.83	$0.83	$0.83	$2.44	$2.44	$2.44	$2.44	$2.44	$2.44	$2.44	$2.44	$2.44	$2.44	**$28.55**
Contingency	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.08	$0.08	$0.08	$0.08	$0.08	$0.24	$0.24	$0.24	$0.24	$0.24	$0.24	$0.24	$0.24	$0.24	$0.24	**$2.86**
Total Water Treatment Opex with Contingency	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$0.91**	**$0.91**	**$0.91**	**$0.91**	**$0.91**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$2.69**	**$31.41**

21.3.6 General and Administrative

General and administrative (G&A) costs include both salaried and hourly labor, supplies, office equipment, and anticipated regular expenses. The G&A operating costs average $23.36 million per year and are ramped up in Years -3, -2, and -1 at 15%, 30%, and 75%, respectively.

21.3.7 Contingency

A 10% contingency was applied to operating costs, totaling 479.02 million, as shown in Table 21-13 through Table 21-17.

22.0 ECONOMIC ANALYSIS

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this PEA will be realized.

22.1.1 Assumptions

Ms. Lane of GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Copper price of $4.90/lb, based on using a weighted average of the 3-year trailing average copper price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $4.44/lb + 40% x 1-year futures price of $6.674/lb
- Molybdenum price of $25.40/lb, based on using a weighted average of the 3-year trailing average molybdenum price and the 1-year futures price, calculated as: 80% x 3-year trailing average price of $22.08/lb + 20% x 1-year futures price of $30.50/lb
- Silver price of $45.30/oz, based on using a weighted average of the 3-year trailing average silver price and the 1-year futures price, calculated as: 60% x 3-year trailing average price of $37.03/oz + 40% x 1-year futures price of $78.406/oz
- The Aurora plant is designed to treat porphyry copper ore with varying molybdenum. The Aurora concentrator flowsheet includes:
    - Crushing - primary crushing using a gyratory crusher
    - Grinding – semi-autogenous mill and conventional ball mill with a pebble crusher (SABC)
    - Flotation – copper and molybdenum - conventional cells for roughing and scavenging and columns for primary cleaning
    - Concentrate regrinding
    - Copper concentrate thickening and filtration
    - Molybdenum concentrate thickening, filtration, and drying
    - Tailings thickening
    - Tailings filtration
    - Tailings impoundments
- Variable mineral recoveries calculated from the equations provided in Section 13.4 for five domains: Cu, Mo, HF, EN, and LE. Overall, mineral recoveries averaged 85.5% for copper, 88.5% for molybdenum, and 74.0% for silver.
- NSR values calculated using the following inputs:
    - Copper 96.5% payable; molybdenum 96% payable; silver 96% payable
    - $25/dmt Cu smelting
    - $0.025/dmt Cu refining
    - $169.35/dmt Cu transport

  - $0.08/dmt Mo roasting
  - $160/dmt Mo transport
  - $0.75/oz Ag refining
  - 90% Cu concentrate solids, 97% Mo concentrate solids
- All costs input to the model are in US dollars
- Sales and use taxes are not included in the model
- Taxes, depreciation, amortization, and deductions as described below.
- Project completion is in Year 18, with closure costs incurred in Years 19 through 24. Post-closure costs would be incurred in Years 20 and beyond. The model extends only to year 20.

22.1.2 Taxes

Note: Ms. Lane is not an expert in Peruvian taxes and relied on information provided by DLP and obtained from on-line searches of tax codes and information from nearby projects to generate a tax model for the project. The calculations are based on the tax regime as of the date of this 2026 PEA. The tax calculations should be considered approximations because actual tax estimates involve complex calculations that can be accurately determined only during operations.

- Mining Royalty: Mining Royalty was applied to operating income on a sliding scale with marginal rates ranging from 1% to 12%
- Special Mining Tax (SMT): SMT was applied to operating income on a sliding scale with marginal rates ranging from 2% to 8.4%
- Depreciation:
  - Per Ernst & Young (2025), mining and processing equipment was depreciated at a rate of 20% (i.e., a 5-year straight line depreciation)
  - Per Ernst & Young (2025), pre-operative development costs were deducted in the year incurred
- OSINERGMIN Fee: OSINERGMIN Fee was applied to NSR at a rate of 0.14%
- OEFA Fee: OEFA Fee was applied to NSR at a rate of 0.10%
- Miner's Retirement Fund: Miner's Retirement Fund was applied to net income after net operating loss at a rate of 0.5%
- Tax Rate: a tax rate of 31.5% was applied to taxable income

22.2 Results

Ms. Lane of GRE considered the following key economic parameters to determine the best scenario: Net Present Value (NPV), Internal Rate of Return (IRR), payback period, mine life, and initial capital cost. Table 22-1 summarizes the results of the economic model.

Table 22-2 presents the key economic results for the project.

22.3 Sensitivity Analyses

Ms. Lane of GRE evaluated the after-tax NPV@8% and IRR sensitivity to changes in capital costs, operating costs, copper price, copper grade, molybdenum price, molybdenum grade, silver price, and silver grade. The results indicate that the after-tax NPV@8% and IRR are most sensitive to molybdenum price and grade, then capital costs, then copper price and grade, with moderate sensitivity to operating cost, and minimal sensitivity to silver price and grade (NPV@8% shown in Figure 22-1; IRR shown in Figure 22-2).

22.4 Life of Mine Cumulative Free Cash Flow

The LOM cumulative free cash flow is illustrated in Figure 22-3

22.5 Conclusions of Economic Model

The project economics shown in the PEA are favorable, providing positive NPV values at varying metal prices, metal grades, capital costs, and operating costs.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this PEA will be realized.

Table 22-1: Aurora Project Economic Model Summary

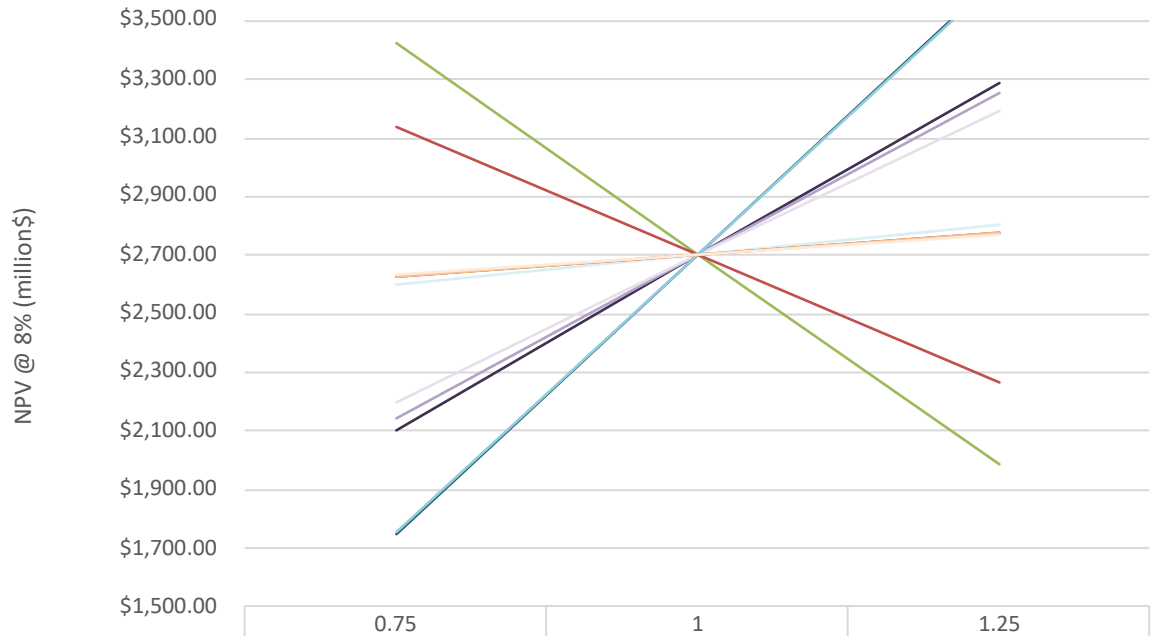
Item	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
Mine Production																								
Aboveground																								
Total Mineralized Material (M t)	**237.26**	13.72	22.28	3.38	41.51	57.77	47.43	51.16																
Cu Contained (M lbs)	**1,730.34**	90.83	133.40	15.76	244.29	440.62	373.58	431.85																
Mo Contained (M lbs)	**76.04**	7.22	3.96	0.32	4.26	8.08	15.31	36.90																
Ag Contained (M oz)	**26.84**	1.58	2.07	0.26	3.84	6.33	5.72	7.03																
Cu Grade (%)	**0.3308**	0.3004	0.2715	0.2116	0.2669	0.3459	0.3572	0.3829																
Mo Grade (%)	**0.0145**	0.0239	0.0081	0.0043	0.0047	0.0063	0.0146	0.0327																
Ag Grade (g/t)	**3.5186**	3.5852	2.8941	2.4223	2.8783	3.4064	3.7514	4.2755																
Total Tonnes (M t)	**461.77**	54.48	69.25	69.70	68.40	66.00	66.21	67.73																
Underground																								
Total Mineralized Material (M t)	**165.65**						0.53	4.00	7.00	7.20	12.02	14.40	14.40	14.40	14.40	14.40	14.40	14.40	14.40	14.40	5.29			
Cu Contained (M lbs)	**244.45**						1.29	8.67	11.47	9.48	13.57	14.59	13.93	14.24	15.49	17.15	19.27	22.90	28.85	37.31	16.24			
Mo Contained (M lbs)	**716.32**						1.41	13.63	30.00	34.69	60.25	70.11	73.09	78.61	77.56	70.86	61.56	52.49	44.44	36.30	11.34			
Ag Contained (M oz)	**3.85**						0.01	0.10	0.15	0.14	0.21	0.23	0.23	0.25	0.28	0.31	0.33	0.36	0.43	0.56	0.25			
Cu Grade (%)	**0.0669**						0.1094	0.0983	0.0743	0.0597	0.0512	0.0460	0.0439	0.0449	0.0488	0.0540	0.0607	0.0721	0.0909	0.1175	0.1392			
Mo Grade (%)	**0.1962**						0.1194	0.1545	0.1944	0.2185	0.2274	0.2208	0.2302	0.2476	0.2443	0.2232	0.1939	0.1653	0.1400	0.1144	0.0971			
Ag Grade (g/t)	**0.7224**						0.5911	0.7704	0.6837	0.6033	0.5483	0.5046	0.4905	0.5351	0.6114	0.6688	0.7061	0.7766	0.9290	1.2104	1.4949			
Plant Production																								
Tonnes Processed (M t)	**407.89**				16.61	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	23.73	11.69		
Cu Concentrate Produced (M t)	**3.06**				0.10	0.28	0.29	0.28	0.20	0.20	0.15	0.13	0.12	0.13	0.13	0.13	0.13	0.14	0.15	0.16	0.23	0.13		
Mo Concentrate Produced (M t)	**0.58**				0.00	0.00	0.01	0.02	0.03	0.03	0.05	0.05	0.06	0.06	0.06	0.05	0.05	0.04	0.03	0.03	0.01	0.00		
Recovered Metals																								
Cu (M lbs)	**1,688.16**				57.00	153.16	159.66	153.14	112.34	109.18	83.26	69.55	68.89	69.08	70.05	71.35	72.98	75.70	80.01	86.26	125.27	71.29		
Mo (M lbs)	**701.40**				0.91	2.56	6.99	23.02	30.73	35.00	56.92	65.28	68.01	72.99	72.04	65.98	57.58	49.38	42.09	34.74	14.45	2.72		
Ag (M oz)	**22.70**				0.84	2.07	2.23	2.15	1.52	1.49	1.11	0.92	0.91	0.92	0.94	0.95	0.96	0.97	0.99	1.04	1.69	1.00		
Revenue																								
Cu Revenue	**$7,982.44**				$269.50	$724.21	$754.95	$724.12	$531.18	$516.27	$393.69	$328.85	$325.77	$326.64	$331.25	$337.37	$345.06	$357.96	$378.33	$407.87	$592.35	$337.08		
Mo Revenue	**$17,102.90**				$22.24	$62.32	$170.38	$561.42	$749.43	$853.55	$1,387.92	$1,591.77	$1,658.27	$1,779.90	$1,756.51	$1,608.82	$1,404.14	$1,204.16	$1,026.35	$847.08	$352.33	$66.31		
Ag Revenue	**$987.14**				$36.65	$90.15	$97.11	$93.35	$66.25	$64.86	$48.34	$39.82	$39.58	$39.95	$40.66	$41.22	$41.56	$42.08	$43.11	$45.32	$73.54	$43.55		
Cu Transport	**-$576.34**				-$19.46	-$52.29	-$54.51	-$52.28	-$38.35	-$37.28	-$28.42	-$23.74	-$23.52	-$23.58	-$23.92	-$24.36	-$24.91	-$25.84	-$27.32	-$29.45	-$42.77	-$24.34		
Cu Refining	**-$42.20**				-$1.42	-$3.83	-$3.99	-$3.83	-$2.81	-$2.73	-$2.08	-$1.74	-$1.72	-$1.73	-$1.75	-$1.78	-$1.82	-$1.89	-$2.00	-$2.16	-$3.13	-$1.78		
Cu Smelting	**-$76.57**				-$2.59	-$6.95	-$7.24	-$6.95	-$5.10	-$4.95	-$3.78	-$3.15	-$3.12	-$3.13	-$3.18	-$3.24	-$3.31	-$3.43	-$3.63	-$3.91	-$5.68	-$3.23		
Mo Transport	**-$95.41**				-$0.12	-$0.35	-$0.95	-$3.13	-$4.18	-$4.76	-$7.74	-$8.88	-$9.25	-$9.93	-$9.80	-$8.98	-$7.83	-$6.72	-$5.73	-$4.73	-$1.97	-$0.37		
Mo Roasting	**-$56.11**				-$0.07	-$0.20	-$0.56	-$1.84	-$2.46	-$2.80	-$4.55	-$5.22	-$5.44	-$5.84	-$5.76	-$5.28	-$4.61	-$3.95	-$3.37	-$2.78	-$1.16	-$0.22		
Ag Refining	**-$16.34**				-$0.61	-$1.49	-$1.61	-$1.55	-$1.10	-$1.07	-$0.80	-$0.66	-$0.66	-$0.66	-$0.67	-$0.68	-$0.69	-$0.70	-$0.71	-$0.75	-$1.22	-$0.72		
Royalty																								
Net Revenue (M $)	**$25,209.48**				$304.12	$811.57	$953.59	$1,309.30	$1,292.87	$1,381.09	$1,782.57	$1,917.05	$1,979.90	$2,101.61	$2,083.34	$1,943.10	$1,747.59	$1,561.66	$1,405.04	$1,256.50	$962.30	$416.28		
Operating Costs																								
OP Mine Opex (M $)	**($796.13)**	($93.85)	($117.29)	($118.56)	($117.45)	($116.87)	($115.84)	($116.26)																
UG Opex (M $)	**($1,142.40)**	($17.48)	($20.83)	($23.60)	($28.41)	($35.38)	($51.55)	($60.58)	($66.33)	($68.02)	($71.83)	($73.24)	($73.60)	($73.60)	($70.36)	($70.40)	($70.36)	($70.36)	($70.36)	($69.65)	($56.49)			
Process Opex (M $)	**($2,219.79)**				($99.43)	($123.06)	($123.06)	($123.06)	($137.28)	($137.11)	($133.01)	($130.99)	($130.99)	($130.99)	($130.99)	($130.99)	($130.99)	($130.99)	($130.99)	($130.99)	($138.73)	($26.14)		
Tailings Opex (M $)	**($717.93)**				($41.68)	($41.68)	($41.68)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($41.82)	($7.45)		
Water Treatment Opex (M $)	**($28.55)**							($0.83)	($0.83)	($0.83)	($0.83)	($0.83)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)	($2.44)		
G&A Opex (M $)	**($429.47)**	($3.50)	($7.01)	($17.52)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($23.36)	($4.26)		
Contingency (M $)	**($479.93)**	($0.00)			($28.19)	($30.50)	($30.39)	($36.59)	($26.96)	($27.11)	($27.09)	($27.02)	($27.22)	($27.22)	($26.90)	($26.90)	($26.90)	($26.90)	($26.90)	($26.83)	($26.28)	($4.03)		
Total Operating Costs (M $)	($5,814.20)	($114.83)	($145.14)	($159.68)	($338.53)	($370.86)	($385.88)	($402.51)	($296.58)	($298.25)	($297.94)	($297.26)	($299.43)	($299.43)	($295.86)	($295.91)	($295.86)	($295.86)	($295.86)	($295.09)	($289.12)	($44.32)		
OP Mine Capitalized Opex (M $)	**$362.67**	$103.24	$129.02	$130.41																				
UG Mine Capitalized Opex (M $)	**$194.97**	$19.23	$22.92	$25.96	$31.25	$38.92	$56.70																	
Total Production Cost	**($5,256.56)**	**$7.63**	**$6.80**	**($3.31)**	**($307.28)**	**($331.94)**	**($329.18)**	**($402.51)**	**($296.58)**	**($298.25)**	**($297.94)**	**($297.26)**	**($299.43)**	**($299.43)**	**($295.86)**	**($295.91)**	**($295.86)**	**($295.86)**	**($295.86)**	**($295.09)**	**($289.12)**	**($44.32)**	**$0.00**	**$0.00**
Operating Income (M $)	**$19,952.92**	**$7.63**	**$6.80**	**($3.31)**	**$189.25**	**$479.64**	**$624.40**	**$906.80**	**$1,003.96**	**$1,090.43**	**$1,490.01**	**$1,624.07**	**$1,684.75**	**$1,806.45**	**$1,791.76**	**$1,651.47**	**$1,456.01**	**$1,270.08**	**$1,113.46**	**$965.70**	**$681.63**	**$111.93**	**$0.00**	**$0.00**

Item	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
Royalties and Fees																								
Regulatory (OSINERGMIN & OEFA Fees) (M $)	**($60.54)**				($1.19)	($1.95)	($2.29)	($3.15)	($3.12)	($3.34)	($4.29)	($4.61)	($4.76)	($5.06)	($5.01)	($4.68)	($4.21)	($3.76)	($3.38)	($3.03)	($2.33)	($0.38)		
Modified Mining Royalty (MMR) (M $)	**($1,051.63)**				($4.97)	($8.12)	($9.54)	($26.15)	($40.76)	($46.78)	($84.62)	($94.32)	($102.89)	($111.30)	($110.41)	($96.71)	($86.55)	($73.37)	($62.94)	($52.87)	($34.09)	($5.24)		
Special Mining Tax (SMT) (M $)	**($817.99)**						($8.32)	($23.92)	($34.16)	($38.68)	($66.10)	($73.29)	($79.34)	($85.61)	($84.91)	($75.02)	($67.19)	($57.40)	($49.61)	($42.12)	($27.98)	($4.34)		
Total Royalties and Fees (M $)	**($1,930.16)**	**$0.00**	**$0.00**	**$0.00**	**($6.16)**	**($10.07)**	**($20.14)**	**($53.22)**	**($78.04)**	**($88.80)**	**($155.02)**	**($172.22)**	**($187.00)**	**($201.96)**	**($200.34)**	**($176.41)**	**($157.94)**	**($134.53)**	**($115.94)**	**($98.02)**	**($64.41)**	**($9.96)**	**$0.00**	**$0.00**
Depreciation	**($3,656.40)**	($355.83)	($621.86)	($705.06)	($364.60)	($452.05)	($319.21)	($224.92)	($154.24)	($150.84)	($57.02)	($55.23)	($18.69)	($20.89)	($20.69)	($51.42)	($20.23)	($20.23)	($15.54)	($13.32)	($10.10)	($4.44)	$43.67	($17.89)
Net Income after Depreciation and Royalty (M $)	**$14,366.36**	**($348.20)**	**($615.05)**	**($708.36)**	**($181.51)**	**$17.52**	**$285.05**	**$628.66**	**$771.67**	**$850.79**	**$1,277.97**	**$1,396.62**	**$1,479.06**	**$1,583.60**	**$1,570.73**	**$1,423.64**	**$1,277.84**	**$1,115.32**	**$981.99**	**$854.36**	**$607.13**	**$97.54**	**$43.67**	**($17.89)**
Net Operating Loss																								
Net Operating Loss Applied (M $)	**($1,853.12)**					($17.52)	($285.05)	($628.66)	($771.67)	($150.22)														
Net Income After NOL (M $)	**$12,539.02**	**($348.20)**	**($615.05)**	**($708.36)**	**($181.51)**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$700.57**	**$1,277.97**	**$1,396.62**	**$1,479.06**	**$1,583.60**	**$1,570.73**	**$1,423.64**	**$1,277.84**	**$1,115.32**	**$981.99**	**$854.36**	**$607.13**	**$97.54**	**$43.67**	**($17.89)**
Other Expenses																								
Profit Sharing (M $)	**($1,152.80)**									($56.05)	($102.24)	($111.73)	($118.32)	($126.69)	($125.66)	($113.89)	($102.23)	($89.23)	($78.56)	($68.35)	($48.57)	($7.80)	($3.49)	
Miner's Retirement Fund (MRF) (M $)	**($72.05)**									($3.50)	($6.39)	($6.98)	($7.40)	($7.92)	($7.85)	($7.12)	($6.39)	($5.58)	($4.91)	($4.27)	($3.04)	($0.49)	($0.22)	
Net Income Before Taxes (M $)	**$11,314.17**	**($348.20)**	**($615.05)**	**($708.36)**	**($181.51)**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$641.02**	**$1,169.34**	**$1,277.91**	**$1,353.34**	**$1,449.00**	**$1,437.22**	**$1,302.63**	**$1,169.23**	**$1,020.51**	**$898.52**	**$781.74**	**$555.53**	**$89.25**	**$39.96**	**($17.89)**
Taxes Federal																								
Total Income Tax (M $)	**($4,153.33)**									($201.92)	($368.34)	($402.54)	($426.30)	($456.43)	($452.72)	($410.33)	($368.31)	($321.46)	($283.03)	($246.25)	($174.99)	($28.11)	($12.59)	
Net Income After Taxes (M $)	**$7,160.84**	**($348.20)**	**($615.05)**	**($708.36)**	**($181.51)**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$439.10**	**$801.00**	**$875.37**	**$927.04**	**$992.56**	**$984.49**	**$892.30**	**$800.92**	**$699.05**	**$615.49**	**$535.49**	**$380.53**	**$61.13**	**$27.37**	**($17.89)**
Capital Costs																								
OP Mine Capitalized Opex (M $)	**($329.70)**	($93.85)	($117.29)	($118.56)																				
OP Mine Equipment Costs (M $)	**($212.53)**	($197.66)	($3.81)	($10.95)	($0.03)	($0.03)	($0.03)	($0.03)																
OP Mine Equipment Salvage (M $)	**$54.11**			$11.18	$0.35			$42.58																
UG Mine Capitalized Opex (M $)	**($177.24)**	($17.48)	($20.83)	($23.60)	($28.41)	($35.38)	($51.55)																	
UG Capex (M $)	**($865.01)**	($69.29)	($39.07)	($32.18)	($49.04)	($69.65)	($99.28)	($120.03)	($116.52)	($107.45)	($47.44)	($7.78)	($15.12)	($23.82)	($12.83)	($16.85)	($10.74)	($14.98)	($5.04)	($3.96)	($3.96)			
Process Capital Costs (M $)	**($502.06)**	($50.21)	($225.93)	($225.93)																				
Tailings Capital Costs (M $)	**($457.55)**	($66.42)	($132.84)	($132.84)		($25.36)		($32.92)				($35.58)				($31.59)								
Infrastructure Capital Costs (M $)	**($219.13)**	($43.83)	($87.65)	($87.65)																				
Indirects Capital Costs (M $)	**($138.97)**	($27.53)	($55.05)	($55.05)		($0.27)		($0.35)				($0.38)				($0.34)								
Closure & Water Treatment (M $)	**($68.81)**	($4.00)				($35.00)																	($14.91)	($14.91)
Owner's Costs (M $)	**($38.00)**	($12.67)	($12.67)	($23.17)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	($1.29)	$33.71	
Sustaining Capital (M $)	**($23.06)**					($22.36)	($0.69)																	
Working Capital (M $)	**($0.00)**				($24.75)																		$24.75	
Contingency (M $)	**($566.97)**	($98.79)	($137.21)	($136.27)	($12.36)	($31.65)	($24.62)	($31.40)	($28.27)	($32.52)	($11.00)	($1.08)	($2.92)	($5.09)	($2.35)	($3.35)	($1.82)	($2.88)	($0.40)	($0.13)	($0.13)	($0.13)	$0.39	($2.98)
Total Capital Costs (M $)	**($3,572.97)**	**($685.22)**	**($839.36)**	**($852.53)**	**($115.52)**	**($221.00)**	**($177.47)**	**($143.43)**	**($146.08)**	**($141.26)**	**($59.73)**	**($46.11)**	**($19.32)**	**($30.20)**	**($16.47)**	**($53.42)**	**($13.85)**	**($19.15)**	**($6.73)**	**($5.38)**	**($5.38)**	**($1.42)**	**$43.94**	**($17.89)**
Add Back Depreciation	**$3,630.62**	$355.83	$621.86	$705.06	$364.60	$452.05	$319.21	$224.92	$154.24	$150.84	$57.02	$55.23	$18.69	$20.89	$20.69	$51.42	$20.23	$20.23	$15.54	$13.32	$10.10	$4.44	($43.67)	$17.89
Add Back NOL	**$1,853.12**					$17.52	$285.05	$628.66	$771.67	$150.22														
Net Cashflow pre-tax (M $)	**$13,224.94**	($677.59)	($832.56)	($855.83)	$67.57	$248.57	$426.79	$710.15	$779.84	$800.83	$1,166.63	$1,287.02	$1,352.70	$1,439.69	$1,441.44	$1,300.63	$1,175.60	$1,021.60	$907.33	$789.68	$560.24	$92.27	$40.23	($17.89)
Net Cashflow after tax (M $)	**$9,071.61**	($677.59)	($832.56)	($855.83)	$67.57	$248.57	$426.79	$710.15	$779.84	$598.91	$798.29	$884.48	$926.40	$983.25	$988.72	$890.30	$807.29	$700.13	$624.30	$543.43	$385.25	$64.16	$27.64	($17.89)

Table 22-2: Aurora Project Key Economic Results

Item	Value
After-Tax NPV @8% (million $)	$2,703
IRR	18.5%
Initial Capital (million $)	$2,377
Payback Period (years)	5.2
C1 Cash Cost Cu ($/lb)	$0.90
C1 Cash Cost Mo ($/lb)	$7.83

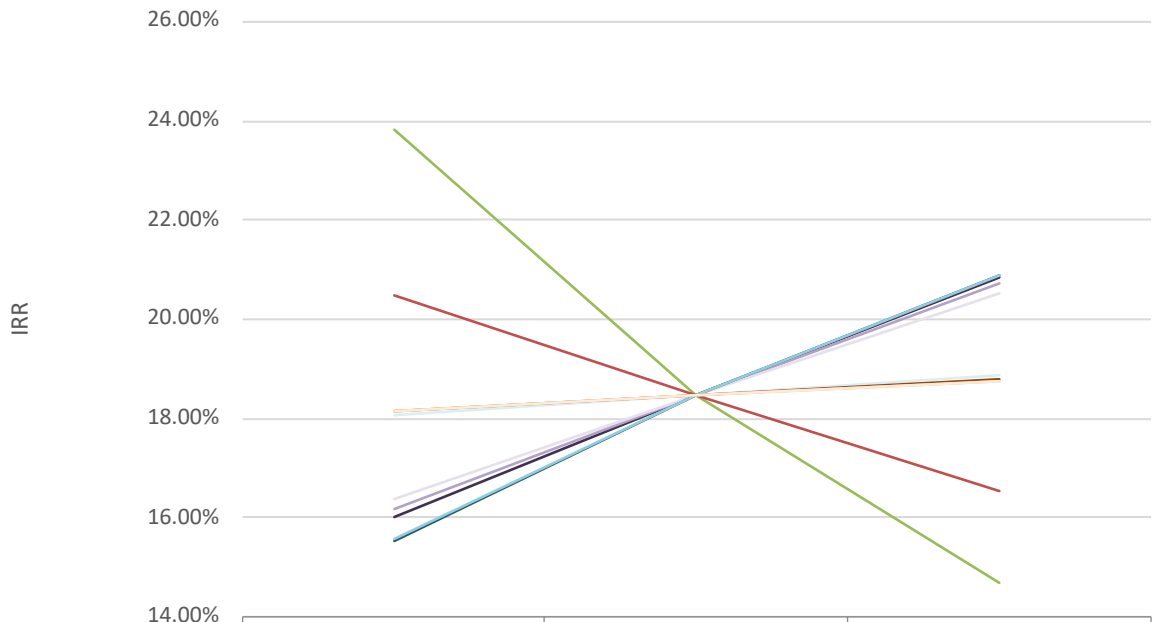
Figure 22-1: Aurora Project Post-Tax NPV@8% Sensitivities



	0.75	1	1.25
Capital Costs	$3,422.38	$2,702.99	$1,988.19
Operating Costs	$3,135.67	$2,702.99	$2,263.02
Cu Price	$2,101.64	$2,702.99	$3,286.07
Cu Grade	$2,140.31	$2,702.99	$3,256.13
Cu Recovery	$2,200.44	$2,702.99	$3,191.12
Mo Price	$1,745.08	$2,702.99	$3,655.71
Mo Grade	$1,753.58	$2,702.99	$3,647.29
Mo Recovery	$2,600.99	$2,702.99	$2,804.91
Ag Price	$2,627.91	$2,702.99	$2,778.05
Ag Grade	$2,629.15	$2,702.99	$2,776.81
Ag Recovery	$2,633.18	$2,702.99	$2,772.78

Source: GRE, 2026

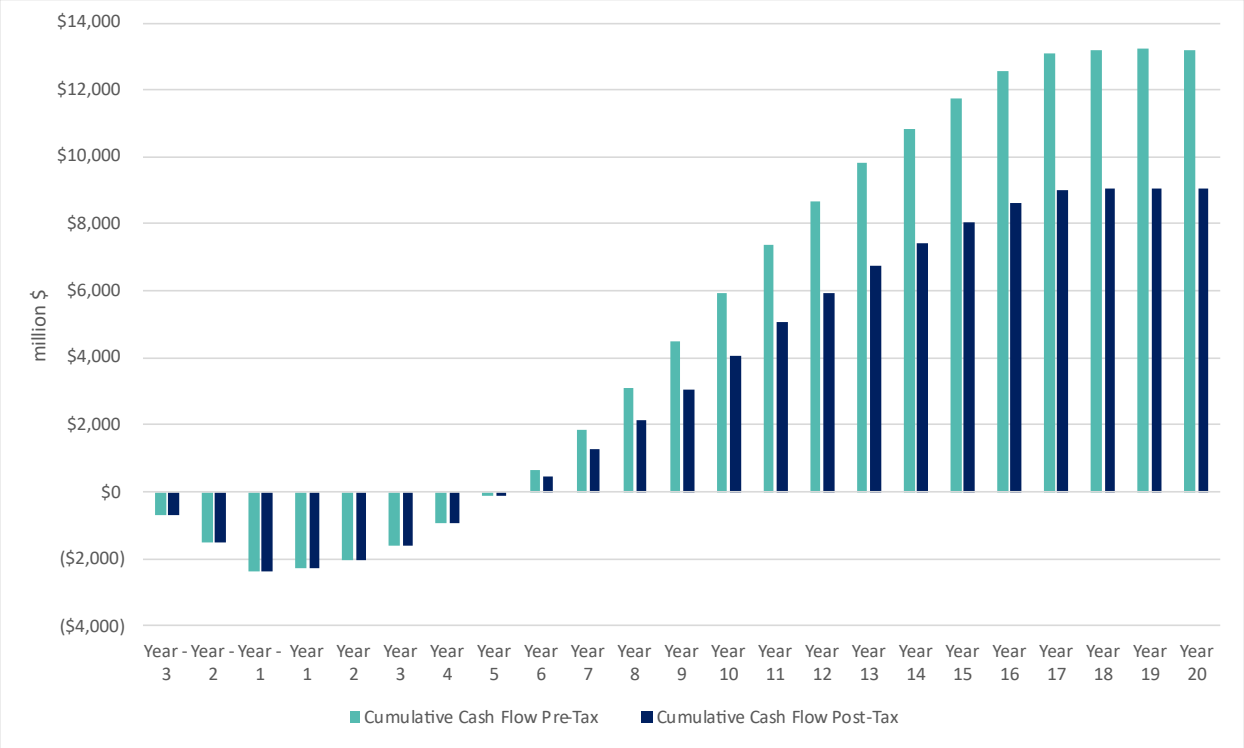
Figure 22-2: Aurora Project Post-Tax IRR Sensitivities



	0.75	1	1.25
Capital Costs	23.80%	18.46%	14.69%
Operating Costs	20.49%	18.46%	16.51%
Cu Price	16.01%	18.46%	20.84%
Cu Grade	16.16%	18.46%	20.72%
Cu Recovery	16.35%	18.46%	20.52%
Mo Price	15.54%	18.46%	20.89%
Mo Grade	15.57%	18.46%	20.87%
Mo Recovery	18.05%	18.46%	18.86%
Ag Price	18.14%	18.46%	18.77%
Ag Grade	18.15%	18.46%	18.76%
Ag Recovery	18.16%	18.46%	18.75%

Source: GRE, 2026

Figure 22-3: Aurora Project Cumulative Free Cash Flow



Source: GRE, 2026

23.0 ADJACENT PROPERTIES

The QP is not aware of any adjacent properties with a similar type of mineralization.

24.0 OTHER RELEVANT DATA AND INFORMATION

The QP is not aware of any additional information or explanation that is necessary to make the Technical Report understandable and not misleading.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 Geology and Mineralization

The Aurora Project is located in southern Peru within the Central Andean metallogenic belt, where long-lived subduction of the Nazca Plate beneath the South American Plate has resulted in extensive magmatic activity and the development of numerous porphyry copper deposits. The regional geology comprises Lower Paleozoic sedimentary and metasedimentary rocks of the San José Group, Sandia Formation, and Ananea Formation, which have been intruded by granodioritic to quartz monzonite plutons and andesitic to dioritic subvolcanic intrusions. The Property is underlain by the Sandia Formation, which is intruded by the multiphase Aurora Porphyry consisting of Early, Intermineral, and Late Intermineral porphyry phases.

The Aurora deposit is interpreted as a Cu-Mo porphyry deposit. The deposit is characterized by multiple intrusive phases, well-developed hydrothermal alteration, stockwork quartz-sulfide veining, hydrothermal brecciation, and disseminated and vein-hosted Cu-Mo mineralization. Mineralization developed through two principal hydrothermal stages comprising an earlier Mo-Cu porphyry stage associated with the emplacement of granodioritic to quartz monzonitic porphyry intrusions, followed by younger structurally controlled Cu-Ag vein mineralization. The observed lithological units, intrusive phases, hydrothermal alteration assemblages, mineralization, and structural controls are consistent with the interpreted Cu-Mo porphyry deposit model.

Understanding the Aurora deposit setting, lithologies, mineralization, and the geological, structural, and hydrothermal alteration controls on mineralization is sufficient to support the estimation of Mineral Resources.

Based on the independent database audits, QA/QC reviews, site inspections, geological verification activities, and independent check assay results, the GRE QP is of the opinion that the Aurora Project databases are sufficiently accurate, reliable, and suitable for use in Mineral Resource Estimation.

25.2 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

Exploration programs completed on the Aurora Property, including geological mapping, surface geochemical sampling, geophysical surveys, and diamond drilling, have successfully defined the extent of the Cu-Mo porphyry system and significantly improved the understanding of the geological setting, hydrothermal alteration, structural controls, and mineralization. The exploration data provide an adequate basis for geological modelling and Mineral Resource estimation.

The drilling methods employed by DLP during the 2022 through 2025 exploration programs are considered appropriate and consistent with accepted industry practice for the estimation of Mineral Resources. Drillhole collar surveys, downhole surveys, core recovery, geological logging, sampling procedures, and density determinations are of sufficient quality to support geological modelling and Mineral Resource estimation.

The sample preparation, analytical methods, sample security, and chain-of-custody procedures implemented by DLP are considered appropriate and consistent with accepted industry practice. The

analytical methods employed by ALS Global are suitable for the style of mineralization encountered at the Aurora Project and provide analytical data of sufficient quality to support Mineral Resource estimation.

DLP implemented comprehensive QA/QC programs during the 2022 through 2025 drilling campaigns, including the insertion of CRMs, blank samples, duplicate samples, and independent check assays. Review of the QA/QC data indicates that no material sample contamination, analytical bias, or precision issues were identified that would adversely affect the Mineral Resource Estimate. Although the overall QA/QC insertion frequency for most drilling campaigns was below current industry best practice, the QA/QC results demonstrate that the analytical database is suitable to support Mineral Resource Estimation. Future drilling programs should implement a minimum QA/QC insertion protocol consisting of at least one blank sample, one CRM, and one duplicate sample for every 20 routine samples submitted for analysis.

The data verification program confirmed that the geological, drilling, sampling, analytical, and QA/QC data collected by DLP between 2022 and 2025 adequately support the geological interpretations and constitute a database of sufficient quality to support Mineral Resource Estimation.

25.3 Metallurgy

The Aurora deposit primary copper species is chalcopyrite, and the primary molybdenum species is molybdenite, there is minor enargite in the enriched zone. Small zones of breccia are observed in the contacts between the porphyry and hornfels units. Test work was conducted on all zones to examine grindability and flotation response.

The flotation flowsheet tested is typical of copper/molybdenum deposits consisting of a bulk rougher flotation of copper and molybdenum, concentrate cleaning followed by downstream molybdenum separation. In general, the flotation performance of the deposit is good, with high metal recoveries to individual copper and molybdenum concentrates. The concentrates, as developed from test work, showed no penalty elements above typical smelter thresholds.

The samples selected were representative of the deposit in terms of spatial, domain, and grade distributions. Additional test work will be required prior to advancing the project to the next study level.

25.4 Resource Estimation

The Aurora Project contains a large developing copper and molybdenum resource with minor silver and gold credits.

The Mineral Resource is open in many directions, especially the deeper high-grade molybdenum-bearing domain that is open horizontally and to depth.

A large surface exploration and potentially an underground exploration program will be required to define the extent and tenor of the mineralization.

The Aurora Project has the potential to produce copper and molybdenum mineralization from both open pit and underground mining methods. Following continued exploration drill programs, optimization and integration of the two mining methods is needed.

25.5 Open Pit Mining

Geotechnical and hydrological data is currently limited. Additional study in both areas is needed. However, the available information was sufficient to support a preliminary open pit design. The pit would produce 237.26 Mt of mineralized material with 1,730 k lbs contained Cu, 76.0 k lbs contained Mo, and 26.8 k oz contained Ag, grading 0.331% Cu, 0.014% Mo, and 3.52 g/t Ag. The pit is mineable using conventional open pit mining techniques with 227 tonnes haul trucks and 22 m^3 hydraulic shovels.

25.6 Underground Mining

25.6.1 Underground Geotechnical

Although a limited and incomplete geotechnical and structural database is available at this stage of the project, the available information, together with core photo review and site observations, was sufficient to support a preliminary rock mass assessment and the application of empirical methods related to the caving process.

The empirical caveability assessment indicates a minimum hydraulic radius (HR) requirement of 36, corresponding to an area of approximately 150 m × 150 m, which is significantly smaller than the footprint used for the mineable inventory and production schedule. Similarly, the empirical subsidence assessment indicates subsidence angles ranging from 66° to 72°.

The geotechnical assessments should be updated as additional geotechnical and structural data become available.

25.6.2 Underground Mining Plan

Based on design and study, SRK offers the following conclusions:

- Work undertaken in this study represents a preliminary economic assessment study standard of work. This document is intended to support an investment decision and/or to guide further study and is not intended as a design issued for construction. The QP recommends additional engineering be undertaken prior to construction.
- Mining is planned to be a block caving mining method, and all horizontal development will be undertaken using conventional drill and blast practices.
- The mine design is based on proven industry standards and productivities achieved at other similar block cave mines around the world.
- A target production rate of 14.4 Mtpa (40,000 tonnes per day) was used for the block cave.
- Access to the Aurora underground mine is expected to be via twin declines commencing from a portal located on the mountainside.
- Vertical development for the main ventilation raises will be excavated using raisebore methods.
- The extraction level has direct on-level tipping into the crushers. Ore will be mined via LHD to the crusher tipping point. The crushed ore will be transferred to the main decline conveyor via transfer conveyors.

- In addition to the access and conveyor declines, the PEA design includes four raises for ventilation (two exhaust raises and two intake raises).
- The costs were developed for an Owner-operating scenario with a contractor used for vertical development only.
- Although specific equipment manufacturers and equipment models are described in this report, no final equipment selection has been made. The equipment listed is indicative of the type and size of equipment planned and was used as the basis for the capital cost estimate. The equipment selected for this study are conventional to block cave operations.

25.6.3 Underground Mining Capital and Operating Cost

Capital and operating cost estimates were completed to an industry standard level for a preliminary economic assessment, and the cost details are shown in Section 21.0. The estimate was prepared at a PEA target level of accuracy of ±30%.

- Underground mine development costs were estimated from first-principles and compared to the SRK internal database of contractor quotes for development.
- A contingency has been added to the capital cost estimate as part of the overall project capital cost aggregation process.
- A significant portion of the initial capital cost for the underground portion of the project is mobile equipment purchases.
- Operating costs for the block cave were estimated from first-principles and are within industry benchmarking ranges for similar operations.
- Further details on the mine capital and operating cost estimates are contained in the PEA mine cost model that has been supplied by SRK to DLP accompanying this PEA report.

25.7 Recovery

The flowsheet selected for the Aurora project was developed based on the available test work and is typical of most copper porphyry deposits containing molybdenum. The ore is subjected to primary crushing using a gyratory crusher followed by SAG and ball milling in an SABC circuit. Flotation consists of a rougher stage using large tank cells followed by concentrate cleaning in a series of Jameson or similar cells. Molybdenum separation is conducted in a second flotation circuit consisting of molybdenum rougher flotation and four stages of cleaner flotation also employing Jameson type cells. Copper and molybdenum concentrates are thickened and filtered, with the molybdenum concentrate undergoing drying in a rotary drier prior to shipment.

The life-of-mine average recoveries for copper, silver, and molybdenum are 85.5%, 88.5% and 74.0%, respectively.

25.8 Infrastructure

25.8.1 Overview

The Aurora Project includes on site infrastructure such as civil, structural and earthworks development, site facilities and buildings, on-site roads, water management facilities and site electrical power facilitites. Off-site infrastructure includes access roads, fresh water supply and power supply. The site infrastructure will include:

- Mine facilities include truck shop, explosive storage and other miscellaneous facilities
- Process facilities including the process plant, crusher facilities, filter plant, metallurgical lab, plant workshop and warehouse
- Dry stack tailings facility and tailings contingency storage facility
- Administrative and mine offices and a permanent camp

The property site can be accessed from the city of Lima to the concession area by air, from Lima to Cusco followed by land through the towns of Calca, Amparaes, and Parobamba. The total time required from Lima to Cusco is one hour by air and then an additional four and a half hours by road from Cusco to site.

25.8.2 Power Supply

Permanent electrical power is provided by a transmission line that will connect to the electrical grid in Calca. The 138-kV transmission line will connect to a substation at the site before being stepped down to 25 kV for distribution to different power requirements across the project site.

25.8.3 Tailings Management

The tailings management facilities were designed in accordance with Canadian Dam Association (CDA) Dam Safety Guidelines (2013) and Application of Dam Safety Guidelines to Mining Dam (2019) to a PEA level. A number of filtered tailings currently are being designed at the feasibility stage with greater storage capacities, depths, terrain, and rain fall conditions than those anticipated for the Aurora DSTF, demonstrating that the project is technically feasible. However, the final facility configuration may evolve as additional engineering studies, site investigations, and optimization work are completed.

Filtered tailings from mining activities will be placed in the DSTF, located 7 km southwest of the open pit and underground portal. The facility is designed to provide secure, permanent storage of 393.9 Mt of tailings for a 64.4 ktpd mining operation over an 18-year life-of-mine. As part of the DSTF, a second, very small slurry-contingency tailings storage facility was designed to accept out-of-specification tailings; for example, if the filter plant is shut down or partially shut down for emergencies or unscheduled repairs for short period, slurry tailings will be diverted to the contingency facility. Water captured in contact water ponds will be pumped to the process plant for use as process water and treated for fresh water supply. Any surplus water will be treated and released to the environment.

25.8.4 Water Supply and Water Management

The site has sufficient water to process the ore, but it must take a modest amount (~10% maximum) from the dry-season base flow of the Mapacho River. This abstraction will not impact local or regional water users, nor will it pose an ecological risk. In addition, the peak abstraction timeline for the project is very

short and lasts only during the few years of open pit operations. Once the block cave is fully developed, the water abstraction drops to ~4% of river baseflow, which may not be measurable.

The site has ARD and ML risk resulting from the oxidation of sulfide minerals in the ore, tailings, waste rock, pit wall rock, and rock in the subsidence zone of the block-cave. As a result, MIW will require active treatment to normalize the pH and to remove regulated metals from the water stream. Provisions exist in the plan to treat this water during operations, and post closure.

25.9 Market Studies and Contracts

Further market studies and contracts will need to be undertaken for future work on the project.

25.10 Environmental Issues

The site has all relevant permits to continue exploration and development. The site has an understanding of the future permitting required to advance the project through construction and production and is taking appropriate early measures to acquire these permits. Therefore, there is no serious permitting issue which might prevent development of the project.

The project is located adjacent to but not within a buffer zone for the Manu National Park. No project features cross the Mapacho River, which forms the southern boundary for the buffer zone.

25.11 Economics

The project economics shown in the PEA are favorable, providing positive NPV values at varying metal prices, metal grades, capital costs, and operating costs. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this PEA will be realized.

25.12 Risks and Opportunities

25.12.1 Underground Mining

Mining technical risks in implementing block cave mining of the Aurora orebody include:

- Delays in access development and main ventilation raises and circuit
- Difficulties in undercutting and ramp-up delays
- Stability and seismicity issues during cave development and propagation
- Airblast during cave development and propagation
- Fragmentation and material handling issues, including oversize, fines, and hang-ups
- Poor drawpoint interaction
- Overdraw, dilution, and recovery issues
- Mudrushes during cave mining

Several opportunities may exist to improve the Aurora underground mine plan and production:

- The internal footprint finder assessment study completed by SRK prior to the PEA identified potential to evaluate additional cave footprints and to expand the block cave underground beyond the initial underground design considered in this study. This may provide opportunities to increase mine life and underground throughput in future studies.
- Geotechnical evaluation and modeling of larger development drives on the extraction level to use large loaders could provide increased block cave productivity.
- Consider contractor development and cave construction to accelerate block cave establishment and production ramp-up.

25.12.2 Concentrator

Key risks to the operation of the Aurora concentrator include:

- There are four metallurgical domains with varying grades for both copper and molybdenum. Copper circuits can generally respond reasonably well to changing grades, but molybdenum can be more challenging. Care needs to be taken to consider the changing grades across the mine life in the flotation circuit design.
- Process water recycle can negatively impact the flotation performance and it may be necessary to keep the molybdenum process water separate from the bulk flotation circuit because of the presence of NaHS.
- The SAG and ball mill put forward in this design are large and limited experience exists in the industry managing and maintaining mills of this size.
- Minimal test work has been conducted on solid/liquid separation and thus the thickener and filter sizing may need modification.

Several opportunities may exist to improve the Aurora process design:

- Optimizing the primary grind size may allow for improved copper and molybdenum recovery. This could also involve the use of alternative flotation technologies like the Jameson or Nova cells.
- Optimizing the silver that reports to the copper concentrate will improve the silver payability.
- The use of HPGR instead of a SAG mill could reduce the overall power consumption in the comminution circuit.

25.12.3 Infrastructure

There has been no geotechnical investigation to date, therefore foundation conditions are unknown at this time and may require adjustments to the assumed foundation condition that could add cost to the project.

25.12.4 Tailings Management Risks

There has been no geotechnical or hydrological investigation to date in the area of the two tailings facilities, which could have an impact on the design of the facilities that could add cost to the project if the assumed conditions are determined to be different.

The project is located in a moderate seismic zone; there has been no tailings geotechnical test work to date. If the parameters are different than those assumed, there could be both static and dynamic stability issues which could affect the design and add costs to the project.

25.12.5 Water Risks

There are few water risks on site. GRE believes that the abstraction from the Mapacho will be tolerable to the regulator agencies, and that even with climate change, the water supply for operations will be secure.

25.12.6 Environmental Risks

There is a risk that the ARD and ML predictions in the PEA will not be accurate. Insufficient geochemical exists to conduct state-of-the-practice geochemical prediction of the quality of MIW and insufficient hydrogeologic data exists to create a more-accurate prediction of dewatering contributions to MIW. Costs for water treatment may be higher or lower than predicted based on the difference between the preliminary predictions of MIW quantity and quality in this study, and that which will actually be observed by operations.

Regulators are often hesitant about perpetual treatment obligations in a closure plan. There is no apparent viable alternative to perpetual treatment based on the post-mining confirmation of the block cave subsidence zone and the proximity of the workings to the Mapacho river.

25.12.7 Socal Risks

The current mine plan has a major haul road running near the town of Parobamba. This may not be suitable for the community, and some relocation may be required. Community relocation is a difficult task and may have unforeseen risks and costs.

25.13 Opportunities

The Authors have identified the following opportunities for the Aurora Project:

- The mineral resource is open in most directions and will certainly grow with continued exploration.
- Optimization of underground material to open pit material blend and plant throughput/capacity has upside potential.
- Analysis of cutoff grade vs. stockpile cutover grade may identify strategies that improve project economics.
- Optimization of phase design and production scheduling may improve project economics.
- There is potential to continue open pit mining in areas distant from the underground block caving operation.
- Other underground mining methods may be able to extract portions of the resource distant from the block cave operation.
- Mine equipment fleet, haul road layout, and maintenance shop location optimizations may improve project economics.

26.0 RECOMMENDATIONS

The QPs have prepared recommendations within their respective areas of expertise. The following recommendations are organized by technical discipline and reflect the opinions and recommendations of the respective QPs responsible for geology, mineral resource estimation, mining, metallurgy, and other applicable disciplines.

Overall, the Authors recommend advancing the project towards a Pre-Feasibility Study, which will require infill drilling, geotechnical studies, additional metallurgy, and evaluation of future open pit and underground mining options.

26.1 Geology

The QP recommends that DLP continue systematic exploration of the Aurora Project to improve the geological understanding of the deposit, better define the geometry, continuity, and controls of the copper-molybdenum-silver mineralization, and evaluate the lateral and vertical extensions of the known mineralized system. The recommended geological work includes the following:

- Continue diamond drilling to infill and to expand the known mineralized zones along strike and at depth, evaluate the continuity of mineralization, support future Mineral Resource classification upgrades where warranted by increased drilling density and geological confidence, and test additional exploration targets identified through geological, geochemical, and geophysical interpretation.
- Continue detailed geological mapping to improve the interpretation of lithological contacts, intrusive phases, hydrothermal alteration, and mineralization.
- Complete detailed structural mapping and structural analysis to better define the controls on intrusive emplacement and mineralization, improve geological interpretations, and optimize future drill targeting.
- Continue systematic surface rock and soil geochemical sampling in areas with limited previous coverage and around newly identified exploration targets.
- Continue refinement of the three-dimensional geological model by integrating geological, structural, alteration, geochemical, geophysical, and drilling data as additional information becomes available.
- Conduct additional petrographic and mineralogical studies to improve understanding of mineralization, alteration zoning, and the distribution of copper-, molybdenum-, and silver-bearing minerals.
- Continue collecting oriented core data, where practical, to further define structural controls on mineralization and improve future geological modeling and drill planning.
- Carry out airborne geophysical surveys (e.g., magnetics, radiometrics, magnetotellurics) to identify new exploration targets, refine structural and lithological interpretations, and support property-scale exploration.

The QP considers the Aurora Project sufficiently prospective to justify continued systematic exploration. The exploration results obtained to date support continued drilling and associated geological studies to

improve the geological understanding of the deposit, better define the geometry, continuity, and controls of the porphyry copper-molybdenum-silver mineralization, and evaluate the lateral and vertical extensions of the known mineralized system.

The estimated costs of the recommended exploration work program are summarized in Table 26-3. The budget is based on DLP's planned exploration activities for 2026-2027 and includes estimated costs for diamond drilling, geological mapping, geochemical sampling, assaying, geological modeling, metallurgical test work, environmental studies, project management, and other supporting technical activities.

26.2 Metallurgy

The following recommendations have been provided to aid in the advancement of the project to the next study level.

- Conduct additional comminution testing with increased sample sets and include the rod mill work index. HPGR may be applicable for this project, and the RMWi is one variable that helps identify its suitability.
- Conduct additional primary grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Conduct additional regrind grind optimization examining all relevant metallurgical domains. Confirm these with mineralogy liberation analysis and flotation testing.
- Additional testing examining reagent suits, particularly for the molybdenum cleaning phase to depress fluorine.
- Examine the use of Jameson or Nova flotation cells to assist in fluorine rejection in the molybdenum circuit.
- Additional testing of high-grade molybdenum materials particularly from future underground areas.
- Additional variability testing, including a wide range of distributions: spatial, domains, and grades.
- Impact of material weathering of flotation performance given the potential long duration stockpile storage envisioned in the production plan.
- Conduct additional test work to define the mechanism for fluorine entrainment.
- Conduct solid/liquid separation testing to develop design criteria for thickening and filtration.
- Conduct TML testing to define the maximum permissible moisture content of the concentrates for sea shipment.
- The estimated cost of this work is $550,000, excluding drilling and sample procurement.

26.3 Resource

The QP makes the following recommendation: continue expansion and definition (conversion of inferred resources to M&I categories) of both the open pit and underground resources through exploration drilling.

26.4 Open Pit Mining

The QP makes the following recommendations:

- Conduct a geotechnical drilling program and analysis to determine appropriate pit slope parameters.
- Determine the phreatic water from drilling, perform packer tests where needed, and model potential pit inflow.
- Optimize mine design and material handling plans.

26.5 Underground Mining

26.5.1 Geotechnical

The QP makes the following recommendations relating to geotechnical risks and opportunities:

- Acquire comprehensive orebody knowledge through additional drilling and the implementation of data collection and analysis programs for rock mass, structures, and water for cave access and development.
- Undertake a dedicated drilling program targeting the proposed caving area and extending to the planned production level elevation. The program should incorporate comprehensive geotechnical investigations, including detailed geotechnical logging, oriented core, and acoustic or optical televiewer surveys to improve the understanding of the deposit's structural framework. The resulting data should be used to develop a robust three-dimensional geotechnical and structural model to support future cave design, caveability assessments, and geotechnical analyses.
- Conduct a dedicated drilling program targeting the area of interest and extending to the proposed production level elevation. These drillholes should be used to support a detailed geotechnical logging program and should include oriented core and televiewer surveys to improve the structural understanding of the deposit. This additional information should allow the development of a robust 3D geotechnical and structural model.
- Undertake more work in the next level of study to evaluate rock mass caveability and understand the steps that would have to be taken if the rock mass did not cave as expected (e.g., preconditioning via hydraulic fracturing). This work will require close collaboration between the cave mining and geotechnical consultants.
- Implement an undercutting strategy that minimizes damage to the rock mass due to induced abutment stress through advanced undercutting.
- Undertake numerical stability modelling and benchmarking.
- Develop and implement a comprehensive cave monitoring and instrumentation program prior to cave commissioning. This will enable the operator to adjust activities such as the rate and direction of undercutting and draw strategy to eliminate airblast risk and minimize potential seismic risk.
- Undertake a mudrush risk study, including an assessment of the physical properties of the caved material and the development of a surface water management strategy.
- Consider managing the subsidence zone, including water diversion, and subsidence slope management.
- Consider establishing an independent Technical Review Board to advise on compliance with industry standard block caving practices and procedures.

- Consider adding hydrogeologic characterization and monitoring to underground geotechnical drilling. This would include adding packer (lugeon) tests, oriented core, and vibrating wire piezometer installation to geotechnical boreholes.

26.5.2 Underground Mining

The QP makes the following recommendations relating to mining risks and opportunities:

- Undertake more work in the next level of study, as understanding of the resource is advanced, to evaluate additional block cave footprints and extend the underground mine life, and consider utilization of the infrastructure established for the initial block cave footprint.
- Further optimize mine design and development schedule to ensure main ventilation raises are established as quickly as possible to support footprint development and cave construction.
- Continue mine optimization and design work as the understanding of the resource is advanced. The footprint selection, draw strategy, and mine layout used in this study should be validated at the conclusion of each major resource model or geotechnical model update.
- Design the mine to reflect industry standards and the latest learnings from deep caves.

26.6 Concentrator

The Aurora concentrator is typical of most copper/molybdenum porphyry plants. The optimization of the process requires additional test work and value engineering:

- Review the comminution circuit design and optimize based on capital and operating costs.
- Consider alternative flotation reagents for molybdenum flotation to reduce the risk of using NaHS.
- Examine alternative flotation technologies to improve the plant performance.
- Consider alternative tailings designs to avoid the capital and operating costs of drystack tailings.

26.7 Infrastructure

26.7.1 Infrastructure Geotechnical

Geotechnical investigations are required within the footprint of each planned structure. Table 26-1 specifies the number of field investigations required for each footprint. These investigations should be strategically located within the footprint, applying engineering judgment and taking into account the site's topography and design characteristics.

Table 26-1: Aurora Project Summary of Field Investigations Quantity

Facility	Quantity		
	Borehole	Test Pit	Geophysics Lines
DSTF	9	20	5
TCSF	4	6	2
Process Plant	2	4	2
Filter Plant	2	4	2
Rope-conveyor	5	5	5
Stockpile	4	6	

Within the framework of the proposed geotechnical investigation program, the extracted core samples shall undergo both physical property testing and specialized tests, as determined by engineering judgment.

Samples must be transported using ASTM International (ASTM) methods to ensure proper handling and consistency. Tailings samples should be obtained from a pilot plant to reflect actual processing conditions. All samples must be representative of the deposit to provide reliable inputs for the prefeasibility level for both geotechnical geochemical studies. The required laboratory tests are illustrated in Table 26-2.

Table 26-2: Aurora Project Summary of Infrastructure Geotechnical Laboratory Tests

Test	Material	ASTM
Soil		
Water Content	Foundation and tailings	ASTM D2216
Atterberg Limits	Foundation and tailings	ASTM D4319
Particle Size Distribution	Foundation and tailings	ASTM C136
Specific Gravity	Foundation and tailings	ASTM D854
One-dimensional consolidation	Foundation and tailings	ASTM D2435
Consolidated isotropic undrained (CIU)Triaxial	Foundation and tailings	ASTM D2850
Rheological tests	Tailings	
Acid-Base Accounting	Foundation and tailings	ASTM E1915 & D2492
Rock		
Unconfined compressive strength of rock core specimens	Foundation	ASTM D7012
Point load strength test	Foundation	ASTM D5731
Specific gravity of rock	Foundation	ASTM C97
Triaxial compression test of rock specimens	Foundation	ASTM D7012
Acid-Base Accounting	Foundation and tailings	ASTM E1915 & D2492

Finally, the development of both factual and Interpretative reports based on data gathered from the geotechnical field and laboratory programs are required.

The estimated cost for this work is $1.4 million.

26.7.2 DSTF and TCSF Analyses and Studies

Due to the conceptual nature of this study and the limited information available at the time of writing, several assumptions have been made regarding the layout, material takeoffs, and construction of the proposed DSTF and TCSF. Geotechnical properties of construction materials will be required to support slope stability analyses and other geotechnical assessments needed to confirm that the DSTF and TCSF can be constructed as designed. Additionally, future geochemical analysis of the tailings and waste rock will be required to determine whether considerations for acid generation and metal leaching should be incorporated into the design of these facilities.

Additional studies and data collection will be required to advance project development beyond the conceptual stage. Some—though not necessarily all—of the current data gaps that should be addressed in future studies include the following:

- Perform deterministic and probabilistic local seismic hazard studies to develop design seismic parameters for infrastructure.
- Collection of hydrological information through site-specific climate and hydrological studies to support the design of ponds, diversion channels, and spillways, the development of the DSTF and TCSF water balances, and alignment with the site-wide water balance.
- Collection of hydrogeological information from desktop studies and site investigations to improve understanding of subsurface flow regimes and their implications for facility design and performance.
- Confirm geochemical characterization of tailings and waste rock from additional waste characterization studies.
- Develop seepage predictions and seepage control measures for the DSTF and TCSF based on geotechnical field and laboratory programs.
- Update the tailings deposition strategy to optimize material handling.
- The stability model should be reviewed and updated, as required, with consideration of the final deposition and stacking plan using updated data on the material properties of the tailings, construction material and the foundations for the DSTF and TCSF.
- Perform a liquefaction assessment of foundations, considering updated information on material properties for DSTF
- Perform dam breach analysis for TCSF and DSTF runout analysis.
- Solicit budgetary quotes for earthworks and geosynthetics (i.e., geomembrane, geotextile, and piping) to get more accurate pricing for the next cost estimate.
- Develop material take-offs and cost estimates (i.e., capital, sustaining capital, and operating costs) for DSTF and TCSF.

The additional information obtained and the assumptions made in this study can be verified or updated to advance the project to the next level of design. The cost of implementing the above recommendations for geotechnical engineering design is estimated at $0.6 million.

26.7.3 Water Infrastructure

Water studies must be advanced. The existing water balance requires revision after more environmental data has been collected. Section 26.8.1 discusses the necessary surface water and groundwater flow data needed to advance the surface water flow models and the site wide water balance.

26.8 Environment and Permitting

26.8.1 Baseline Data Collection

Baseline data collection for at least three years prior to mining is strongly recommended. The following environmental data must be collected:

- Surface water flow readings: As mentioned in Section 18.5, the project must take advantage of nearly all the water in the Mestizamayo drainage basin and must abstract water from the Mapacho river. In order to acquire water rights, it is necessary to build a multi-year river flow measurement database.
- Meteorology: The current meteorological station is ~38 km away, and on-site meteorological data is required. Additionally, because the site covers two different physiographic regions (see Section 4.1.3) an upper-elevation and a lower-elevation meteorological station are recommended. The upper-elevation station should have a snow-water equivalent rain gauge.
- Environmental Geochemistry: Due to the ARD and ML risk, the site should execute a wide-spread and intensive ARD and ML characterization program involving static sampling, and kinetic (long duration) geochemical testing. Testwork should cover underground development rock (from block cave access tunnels), open pit rock, pit wall rock, and mine tailings. The mine tailings geochemical test work must include an assessment of tailings variability – in particular the geochemical difference between deeper high-grade block cave tailings, and shallow lower-grade open pit tailings.
- Groundwater: No groundwater data exists. It is necessary to collect hydrogeologic data in geotechnical boreholes for he pit walls and for the underground workings. This should be done with packer tests (lugeon-style injection tests), and vibrating wire piezometer installation. Shallow groundwater wells should be installed at strategic locations around the site. These wells should be aquifer tested, and the water levels should be monitored seasonally for at least a calendar year.
- Water Quality: Few water quality samples exist. Site-wide baseline surface water quality is required from rivers, seeps, springs, and groundwater wells. Samples should be collected from seeps and springs in the Sandia formation to see of naturally-occurring ARD conditions (see Figure 20.3) are rare, or widespread. Total Suspended Solids (TSS) sampling should be performed to understand sediment transport dynamics.
- Air and Noise: Air quality samples and noise background should be collected as per the requirements of Peruvian Law.
- Social Monitoring: The site should continue a social monitoring program to better understand the local community.

26.8.2 Environmental Studies

For a Pre-Feasibility study, the following major studies will be required:

- Geochemical predictive models of future ARD and ML for the pit, underground workings, DSTF, and WRSF. These must be transient, site-wide, and robust enough to determine the water quality of all MIW streams over time and space within the project.
- Predictive groundwater flow models of the dewatering vs time for the open pit, block cave development tunnels, and for the block cave.
- Revised surface water models and water balance models based on, and calibrated to, the surface water flows measured in the environmental monitoring work scope.
- Treatability test work for the MIW water treatment plant

- Air quality models for the project
- Noise models
- Sediment production and sediment management models
- Evapotranspiration cover studies to optimize the design.

26.8.3 Permitting

The Aurora project should continue permitting activities. At this stage, the continuation of the permitting for exploration, the baseline data collection, and the studies discussed above are the most-prudent measures to advance permitting because full-scale permit acquisition requires the data and studies above.

26.8.4 Closure

Studies above will improve the closure plan and the closure cost model. Once completed, the closure plan and closure cost estimate must be revised.

26.9 Total Cost for Recommended Program

The estimated costs of the recommended exploration and infill work program to advance the project to Pre-Feasibility Study are summarized in Table 26-3. The budget is based on DLP's planned exploration activities for 2027 and includes estimated costs for diamond drilling, geological mapping, geochemical sampling, assaying, geological modeling, metallurgical test work, environmental studies, project management, and other supporting technical activities.

Table 26-3: Estimated Costs to Complete the Aurora Project 2-Year Program Leading to PFS

Description	$USD
Diamond Exploration & Infill Drilling (100,000 m)	$50,000,000
Camp and Infrastructure	$5,000,000
Geological Mapping & Sampling	$1,500,000
Geophysical Survey (100 sq km)	$1,000,000
Metallurgical Test work	$3,000,000
Geotechnical Testing and Studies	$4,000,000
DSTF and TCSF Data Acquisition	$1,500,000
Environmental or Baseline Studies	$3,000,000
Permitting	$3,000,000
Community, Land & Legal Cost	$3,000,000
Pre-Feasibility Study	$5,000,000
Total	**$80,000,000**

27.0 REFERENCES

Alex G Doll Consulting Limited. 2026. *Grinding circuit design for the Aurora project (revised).* 2026.

ALS Minerals. 2013. *Re-Os Age Dating of Molybdenite for ALS Minerals.* 2013.

AMC. 2025. *NI 43-101 Technical Report on the Aurora Co-Mo-Ag Property in the Calca Province, Peru.* 2025.

C. H. Plenge & CIA. S.A. 2024. *Aurora Porphyry Copper -Molybdenum Project Composites M2 Metallurgical Scoping Study.* 2024.

—. 2024. *Metallurgical Investigation No.18739.* 2024.

—. 2026. *Metallurgical Investigation No.18808.* 2026.

CAMP. 2024. *TIMA Characterization of the DLP.* 2024.

CDA. 2014. *Application of Dam Sfety Guidelines to Mining Dams.* 2014.

CIM. 2014. *Definition Standards for Mineral Resources and Mineral Reserves.* s.l. : CIM Standing Committe on Reserve Definitions, 2014.

Conventional Cover Performance from the ACAP Field Program. **Albright, William. 2003.** Denver, CO : US EPA, 2003. Alternative Cover Assessment Program (ACAP) Workshop, Presentation Slides.

2017. Decreto Supremo No.004-2017-MINAM. *Aprueban Estándares de Calidad Ambiental (ECA) para Agua y establecen Disposiciones Complementarias.* [Online] June 7, 2017. https://www.minam.gob.pe/wp-content/uploads/2017/06/DS-004-2017-MINAM.pdf.

Diering, J. H. and Laubscher, D. H. 1987. Practical Approach to the Numerical Stress Analysis of Mass Mining. *Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry.* 1987, Vol. 96, pp. A179-A188.

Eagle Engineering. 2023. *AMICS ANALYSIS.* 2023.

Ernst & Young. 2025. *Peru's Mining & Metals Investment Guide, 2025/2026.* 2025.

Espinoza, Pavel Aquino. 2017. *Water Quality in Peru: challenges and contributions for sustainable management in residual waters.* Lima : Derecho, Ambiente y Recursos Naturales (DAR), 2017.

Evaluation of Final Cover Performance: Field Data from the Alternative Cover Assessment Program (ACAP). **Benson, Craig H., et al. 2002.** Tucson, Arizona : s.n., 2002. Waste Management Conference.

Fenner Dunlop. 2009. *Conveyor Handbook.* s.l. : Fenner Dunlop Belting, 2009.

Fetter, C.W. 2001. *Applied Hydrogeology, Fourth Edition.* Upper Saddle River, New Jersey : Prentice Hall, 2001.

Global Acid Rock Drainage Guide. **INAP. 2009.** s.l. : Inernational Network for Acid Prevention, 2009, International Network on Acid Prevention, Vol. www.gardguide.com.

Heidbach, O., et al. 2025. World Sress Map 2025. [Online] 2025. https://doi.org/10.5880/WSM.2025.002.

INAP. 2009. Global Acid Rock Drainage Guide. 2009.

InfoMine. 2025. *Mining Cost Service.* 2025.

Jakubec, J. 2023. Block Caving. [ed.] P. Darling. *SME Underground Mining Handbook (Ch. 11, p. 224).* 2023, ISBN 978-0-87335-484-4.

Laubacher, G. 1978. *Geologie de la Cordillre Orientale et de l'Altiplano au Nord et Nord-ouest du Lac Titcaca (Perou).* 1978.

Laubscher, D. H. 1990. A Geomechanics Classification System for the Rating of Rock Mass in Mine Design. *Journal of the South African Institute of Mining and Metallurgey.* 90(10), 1990, pp. 257-273.

Laubscher, D. H. and Jakubec, J. 2001. The MRMR Rock Mass Classification for Jointed Rock Masses. [ed.] W. A. Hustulid and R. L. Bullock. *Underground Mining Methods: Engineering Fundamentals and International Case Studies.* 2001, ISBN 978-0-87335-193-5, pp. 475-481.

Laubscher, D. H. 2000. Block Cave Manual. *Prepared for the International Caving Study, JKMRC and Itasca Consulting Group Inc., Brisbane.* 2000.

—. 1994. Cave Mining - the State of the Art. *Journal of the South African Institute of Mining and Metallurgy.* 94(10), 1994, pp. 279-293.

Lawerence, Richard and Wang, Ying. 1996. *Determination of Neutralization Potential for Acid Rock Drainage Prediction.* Vancouver, BC : Environment Canada, 1996.

Ministerio de Energia y Minas. 2005. *Guia Para la Elaboracion de Planes de Cierre De Minas.* Lima, Peru : Republica del Peru, 2005.

PCSWMM. 2026. ChiWater - PCSWMM. *Advanced modeling software for stormwater, wastewater, watershed and water distribution systems.* [Online] 2026. https://www.pcswmm.com/.

Salas, A. G., Chavez, V. A. and Cuadros, P. J. 1999. *Geologia De Los Cuadrangulos de Quebreda Honda Y Parobamba.* 1999.

SRK. 2026. *Block Cave Design for the Aurora Project.* Vancouver, BC, Canada : SRK Consulting, 2026.

SUNARP. 2026. *Inscripcion de Comunidades Campesinas y Nativas, Comunidad de Parobamba.* Zona Registral No. X - Sede Cusco : Superintendiente Nacional de los Registors Publicos, 2026. No. Partida: 02087063.

The University of Queensland. 2025. *SMC Test Report.* 2025.

Urano. 2022. *Declaracion de Impacto Ambiental (DIA) for the Aurora Exploration Project.* Cusco, Peru : Urano Group Consultin, 2022.

—. 2025. *Hydrologic Report of the Rio Mapacho.* Lima, Peru : Urano Consultores Ambientales, 2025.

US EPA. 1994. Synthetic Precipitation Leaching Procedure. 1994.

CERTIFICATE OF QUALIFIED PERSON

I, Terre A. Lane, of 17301 W. Colfax Ave., Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Aurora Project, Calca Province, Peru" with an effective date of April 30, 2026, and an issue date of October 15, 2026 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP, and I am a Registered Member of the Society for Mining, Metallurgy, and Exploration
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this PEA is as the resource estimator, mine planner, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in geology, structural geology, mineralogy, Mineral Resource estimation in university, and have taken several short courses in geostatistics subsequently.
5. I have worked in geology, managed geologic teams, created lithological and structural models, and I have been involved in or conducted the estimation of resources for several hundred projects at locations in North America, Central America, South America, Africa, Australian/New Zealand, India, China, Russia and Europe using nearly all estimation techniques.
6. I have estimated resources and created mine plans for several copper/molybdenum porphyry/skarn deposits including Vizcachitas, Henderson, Climax, Mount Emmons, North Ore Shoot (Bingham Canyon), and others and have overseen the resource estimate and mine design of many other similar porphyry deposits.
7. I have created or overseen the development of mine plans for several hundred open pit and underground projects and operating mines.
8. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
11. I have not visited the Aurora property.
12. I am responsible for Sections 1.7, 1.8, 1.9, 1.9.1, 1.11, 1.13.3, 1.13.3, 2, 3, 14.1, 15, 16.1, 19, 21.1, 21.2.1, 21.2.7, 21.2.9, 21.2.10, 21.2.11, 21.2.12, 21.2.13, 21.3.1, 21.3.6, 21.3.7, 22, 23, 24, 25.3, 25.4, 25.9, 26.3, 26.4, 26.9, and 27 of the Technical Report.
13. I am independent of DLP as described in section 1.5 by National Instrument 43-101.
14. I have not previously worked on the Aurora Project.
15. I have read National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.

16. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Terre A. Lane

"Terre A. Lane"

Mining Engineer

Global Resource Engineering, Ltd.

Golden, Colorado

Date of Signing: October 15, 2026

CERTIFICATE OF QUALIFIED PERSON

I, J. Todd Harvey, of 17301 W. Colfax Ave., Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Aurora Project, Calca Province, Peru" with an effective date of April 30, 2026, and issue date of October 15, 2026 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am currently employed as Principal Process Engineer by Global Resource Engineering, Ltd.
2. I graduated with Ph.D. in Mining Engineering from the Queen's University at Kingston in 1994, a Master's degree in Mining Engineering from the Queen's University at Kingston in 1990 and a Bachelors degree in Mining Engineering in 1988 all with a specialization in mineral processing. I also hold a degree in Metallurgical Engineering and Computer Science from Ryerson University in Toronto Canada graduating in 1986 as well as an MBA from the University of New Brunswick in Saint John Canada graduating in 2001.
3. I have worked as a Process Engineer for over 35 years since my graduation from university. My relevant experience includes process due diligence/competent persons evaluations of developmental phase and operational phase mines throughout the world, including mines in the USA, Canada, Kazakhstan, Brazil, Mexico, and Africa to name a few. I have a wide range of experience in multiple mineral fields including precious metal processing and base metals such as copper, lead, and zinc.
4. I am a Registered Member (No. 04144120) of the Society for Mining, Metallurgy & Exploration Inc. (SME). I am also a member of the Association for Mineral Exploration (AME), Minerals Engineering Journal Review Board, and the Journal of Hydrometallurgy Review Board.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
6. I have not visited the Aurora Property.
7. I am responsible for Sections 1.6, 1.13.2, 13, 17 (except for 17.3.7), 21.2.3, 21.3.3, 25.3, 26.2, and 26.6.
8. I am independent of DLP as described in section 1.5 by National Instrument 43-101.
9. I have not previously worked on the Aurora Project.
10. I have read National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

J. Todd Harvey

"J. Todd Harvey"

Metallurgist

Global Resource Engineering, Ltd.

Golden, Colorado

Date of Signing: October 15, 2026

CERTIFICATE OF QUALIFIED PERSON

I, Hamid Samari, PhD, of 17301 W. Colfax Ave., Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report, Aurora Project, Calca Province, Peru" with an effective date of April 30, 2026, and an issue date of October 15, 2026 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am currently employed as a principal geologist by Global Resource Engineering, Ltd.
2. I am a graduate of Azad University, Sciences and Research Branch, Tehran and received a PhD in Geology-Tectonics in 2000 and I am a graduate of Beheshti University, Tehran and received a MS in Geology-Tectonics in 1995 MS and I am a graduate of Beheshti University, Tehran and received a BS in Geology in 1991
3. I am a Qualified Professional in the United States from the Mining and Metallurgical Society of America (MMSA) with special expertise in Geology, with membership number 01519QP.
4. I have practiced in the areas of geology, mining, and civil engineering for over 25 years. I have worked for Azad University, Mahallat branch, as an assistant professor and head of the geology department for 19 years; for Tamavan Consulting Engineers as a senior geologist for 12 years; and for Global Resource Engineering for nearly 9 years. I have worked on geologic reports and resource statements for silver and gold deposits in the United States and Latin America. My relevant work includes field exploration, structural mapping, surface and underground sampling, QA/QC, and preparation of NI 43-101 technical reports for silver, gold, and polymetallic deposits, including epithermal silver deposits in Peru, gold deposits in Nevada and Utah, and polymetallic deposits elsewhere in the Western Hemisphere.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
6. I visited the Aurora property on June 2 and 3, 2026, for two days and conducted a personal inspection of the Aurora Project and associated core storage facilities.
7. I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.13.1, 1.14, 5, 6, 7, 8, 9, 10, 11, 12 (except for 12.6.1 and 12.6.2), 25.1, 25.2, and 26.1.
8. I am independent of DLP as described in section 1.5 by National Instrument 43-101.
9. I have not previously worked on the Aurora Project.
10. I have read National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Hamid Samari, PhD

"Hamid Samari"

Geologist

Global Resource Engineering, Ltd.

Golden, Colorado

Date of Signing: October 15, 2026

CERTIFICATE OF QUALIFIED PERSON

I, J. Larry Breckenridge, P.E., of 17301 W. Colfax Ave., Suite 400, Golden, Colorado, 80401, the co-author of the report entitled "NI 43-101 Technical Report, Aurora Project, Calca Province, Peru" with an effective date of April 30, 2026, and an issue date of October 15, 2026 (the "Technical Report"), DO HEREBY CERTIFY THAT:

1. I am currently employed as principal environmental engineer by Global Resource Engineering, Ltd.
2. I am a graduate of Dartmouth College with a degree in Engineering Modified with Environmental Science (BA) and from the Colorado School of Mines with a Masters' degree in Environmental Engineering.
3. I am a Qualified Person under NI 43-101 because I am a registered Environmental Engineer in the State of Colorado, USA, No. 38048.
4. I have practiced in the areas of water management, geochemistry, and environmental management exclusively for precious and base metals projects for over 25 years. I have worked with Global Resource Engineering in my same role for the last 13 years. I have participated in the permitting process for numerous mines in the United States and in Latin America. I have evaluated geochemical risk and permitting requirements for precious metals projects and also performed water availability studies.
5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Aurora property from August 10th 2025 until August 13th 2025. While on site, I inspected site core, took geochemical samples, and evaluated all mine waste management sites.
7. I am responsible for Sections 1.1, 1.12, 1.13.7, 1.13.8, 4, 5, 18.5, 20, 21.2.8, 21.3.5, 25.7.3, 25.8, 25.10.4, 25.10.5, 25.10.5, 26.7.3, and 26.8.
8. I am independent of DLP as described in section 1.5 by National Instrument 43-101.
9. I have not previously worked on the Aurora Project.
10. I have read National Instrument 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

J. Larry Breckenrdige

"J Larry Breckenridge"

Principal Environmental Engineer
Global Resource Engineering, Ltd.
Golden, Colorado
Date of Signing: October 15, 2026

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project, Calca Province, Peru" prepared for DLP Resources Inc. (the "Issuer") dated October 15, 2026, with an effective date of April 30, 2026 (the "Technical Report").

I, Carl Kottmeier, *PEng, MBA*, do hereby certify that:

1 I am a Principal Consultant & Practice Leader with SRK Consulting (Canada) Inc. ("SRK") with an office at Suite #2600 – 320 Granville Street, Vancouver, British Columba, Canada, V6C 1S9.

2 I am a graduate with a B.A.Sc. (Applied Science) degree in Engineering from the University of British Columbia in 1989 and an MBA degree from the University of British Columbia in 2003. I have practiced my profession continuously since 1989. I have worked on mining projects in North and South America, Africa and in Europe, and I have extensive experience with base metal and precious metal mining projects such as the Aurora project. My relevant experience includes responsibilities in operations, maintenance, mine design, mine engineering, management, contracting and construction activities.

3 I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia, license # 18702.

4 I have not visited the Aurora project site.

5 I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to 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 as defined in Section 1.5 of NI 43-101.

7 I am a co-author of the Technical Report, responsible for Section 16.2.3 (excluding 16.2.3.2), as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, and References, and I accept professional responsibility for those sections of the Technical Report.

8 I have had prior involvement with the subject property.

9 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report not misleading.

10 I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 15th day of October 2026 in Vancouver, British Columbia, Canada.

"original signed"

Carl Kottmeier, *PEng, MBA*
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project, Calca Province, Peru" prepared for DLP Resources Inc. (the "Issuer") dated October 15, 2026, with an effective date of April 30, 2026 (the "Technical Report").

I, Jarek Jakubec, *CEng, FIMMM*, do hereby certify that:

1 I am a Corporate Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at Suite #2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2 I am a graduate of the Mining University in Ostrava, Czech Republic with a MSc. in Mining Geology (1984). I have practiced my profession continuously since 1984 and I have 40 years' experience in mining. I have been involved in project management, mine design, due diligence studies, geological and geotechnical modeling around the world. I have direct operational experience from caving mines in Canada and have been involved in caving or sublevel caving mines studies in Canada, the United States, Chile, South Africa, Australia, Indonesia, Papua New Guinea, China, Kazakhstan and Mongolia. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101.

3 I am a registered Chartered Engineer (No. 509147) and Fellow of the Institute of Materials, Minerals and Mining in the United Kingdom.

4 I have not visited the Aurora project site.

5 I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to 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 as defined in Section 1.5 of NI 43-101.

7 I am a co-author of the Technical Report, responsible for Sections 1.13.5.1, 16.2.1, 25.6.1, 25.10.1, and 26.5.1, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, and References, and I accept professional responsibility for those sections of the Technical Report.

8 I have had prior involvement with the subject property.

9 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report not misleading.

10 I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 15th day of October 2026 in Vancouver, British Columbia, Canada.

"original signed"

Jarek Jakubec, *CEng, FIMMM*
Corporate Consultant (Mining & Geology)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project, Calca Province, Peru" prepared for DLP Resources Inc. (the "Issuer") dated October 15, 2026, with an effective date of April 30, 2026 (the "Technical Report").

I, Joshua Parsons, *PEng, BEng,* do hereby certify that:

1 I am a Senior Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at Suite #2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

3 I am a graduate of Dalhousie University where I obtained a Bachelor of Engineering degree in Mineral Resource Engineering. Aside from the time spent studying at Dalhousie University, I have practiced my profession continuously since 2014. My relevant experience includes 11 years' experience working at an operating block cave mine where I was responsible for cave management, mine planning, mine design, mining studies, drill and blast, long-range planning, budgeting, annual reporting, and technical team supervision. I also have 9 years' experience in block cave mineral reserve estimation. I have managed technical projects through the full mine life cycle and have extensive experience leading multidisciplinary technical teams and supporting underground mining operations.

4 I am a Professional Engineer licensed with Engineers and Geoscientists British Columbia (EGBC), Registration Number 50228.

5 I have not visited the Aurora project site.

6 I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7 I am independent of the issuer as defined in Section 1.5 of NI 43-101.

8 I am a co-author of the Technical Report, responsible for Sections 1.9.2, 1.15.5.2, 16.2 (except for 16.2.1 and 16.2.3), 25.6.2, 25.6.3, and 26.5.2, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, and References, and I accept professional responsibility for those sections of the Technical Report.

9 I have had prior involvement with the subject property.

10 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report not misleading.

11 I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 15th day of October 2026 in Kamloops, British Columbia, Canada.

"Original Signed"

Joshua Parsons, *PEng, BEng*
Senior Consultant (Mining)
SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project, Calca Province, Peru" prepared for DLP Resources Inc. (the "Issuer") dated October 15, 2026, with an effective date of April 30, 2026 (the "Technical Report").

I, Brian Prosser, *PE, BSc*, do hereby certify that:

1. I am a Principal Consultant with SRK Consulting (US), Inc. with an office at 1625 Shaw Avenue, Suite #103, Clovis, California, 93611, USA.
2. I am a graduate of Virginia Poly Technic Institute and State University where I obtained a Bachelor of Science degree in Mining Engineering. Aside from the time spent studying at Virginia Poly Technic Institute and State University, I have practiced my profession continuously since 1994. My relevant experience includes the design and evaluation of ventilation designs for subsurface mining, tunneling, and repositories.
3. I am a Professional Engineer licensed with the State of Nevada Board of Professional Engineers and Land Surveyors License Number 15465.
4. I have not visited the Aurora project site.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to 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 as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 16.2.3.2, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Capital and Operating Costs, Conclusions and Recommendations, Risks and Opportunities, and References, and I accept professional responsibility for those sections of the Technical Report.
8. I have had no prior involvement with the subject property.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portions of the Technical Report not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 15th day of October 2026 in Clovis, California, USA.

"original signed"

Brian Prosser, *PE, BSc*
Principal Consultant (Mine Ventilation)
SRK Consulting (US), Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Scott C Elfen, P.E. & P. Eng., of 12th Floor, 1050 W. Pender Street, Vancouver, British Columbia, V6E 3S7, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project Calca Province, Peru" with an effective date of April 30, 2026, and an issue date of October 15, 2026, do hereby certify that:

1. I am employed as the SME & Global Technical Leader for Mine Waste Solution with Ausenco Engineering Canada.
2. I graduated from the University of California, Davis, California, with a Bachelor of Science degree in Civil Engineering in 1991.
3. I am a professional engineer registered with the State of California and Idaho in the United States of America (No. C56527) by exam since 1996 and (No. 3961670) by reciprocity since 2024, respectively. I am also a professional engineer registered within the province of British Columbia, Canada (No. 64064) by Exam since 2025.
4. I have practised my profession continuously for 32 years, with experience in the development, design, construction, and operations of mine waste storage facilities, including waste rock and tailings storage facilities ranging from slurry to dry-stack, as well as surface water management and water balances focused on precious and base metals, both domestically and internationally. I have developed geotechnical design parameters for pit slope design, plant foundation design, and other supporting infrastructure, as well as surface water management design for infrastructure and water balances. Examples of projects I have worked on include Skeena's Eskay Creek Project PEA, PFS, and FS; O3 Mining's Marban Project PEA and PFS; and First Mining Gold's Springpole PEA and PFS, Lumina Gold's Cangrejos PEA, PFS, and FS.
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 virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the project site from June 01st, 2026 to June 04, 2026.
7. I am responsible for Sections 1.10, 1.13.6, 2.3.2, 12.6.4, 17.3.7, 18.1, 18.2.1, 18.3.1-18.3.4, 18.4, 21.2.4, 21.2.5, 21.3.4, 25.8.1, 25.8.3, 25.12.4, and 26.7 of the Technical Report.
8. I am independent of DLP Resources as independence is defined in Section 1.5 of NI 43-101.
9. I have not had prior involvement with the Aurora Project.
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 that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Scott C Elfen, P.E. & P.Eng.

"Signed and sealed"
SME & Global Technical Leader for Mine Waste Solution
Ausenco Engineering Canada ULC.
Vancouver, British Columbia
Date of Signing: October 15, 2026

CERTIFICATE OF QUALIFIED PERSON

I, Kevin Murray, P. Eng., of 12th Floor, 1050 W. Pender Street, Vancouver British Columbia, V6E 3S7, the co-author of the report entitled "Preliminary Economic Assessment NI 43-101 Technical Report Aurora Cu-Mo-Ag Mine Project Calca Province, Peru" with an effective date of April 30, 2026, and an issue date of October 15, 2026, do hereby certify that:

1. I am employed as a Senior Mineral Processing Engineer with Ausenco Engineering Canada (Ausenco).
2. I graduated from the University of New Brunswick with a Bachelor of Science degree in Chemical Engineering in 1995.
3. I am a member in good standing of Engineers and Geoscientists British Columbia (No. 32350), Northwest Territories Association of Professional Engineers and Geoscientists (No. L4940) and Association of Professional Engineers and Geoscientists of Saskatchewan (No. 82404).
4. I have practiced my profession continuously for 26 years. I have been directly involved in all levels of engineering studies form preliminary economic assessments (PEAs) to feasibility studies. I have led preliminary test work design, test work analysis, power infrastructure design and flowsheet development as well as involvement in detailed design and commissioning. I have also developed operating cost estimates and contributed to and reviewed capital cost estimates. I have been involved with copper-gold studies including Almos's (formerly Argonaut) Magino project and Lumina's Cangrejos project.
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 virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the project site.
7. I am responsible for Sections 1.10.1, 18.2.2, 18.3.5, 21.2.6, and 25.8.2 of the Technical Report.
8. I am independent of DLP Resources as independence is defined in Section 1.5 of NI 43-101.
9. I have not had prior involvement with the Aurora Project.
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 that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Kevin Murray, P. Eng

"Signed and sealed"
Senior Process Engineer
Ausenco Engineering Canada ULC.
Vancouver, British Columbia
Date of Signing: October 15, 2026