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Feasibility Study

Yukon, Canada

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DATE AND SIGNATURES PAGE

The effective date of this report is 13 June 2021. The issue date of this report is 08 August 2022. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

CASINO PROJECT
FORM 43-101F1 TECHNICAL REPORT
FEASIBILITY STUDY

TABLE OF CONTENTS

SECTION PAGE

DATE AND SIGNATURES PAGE I
TABLE OF CONTENTS II
LIST OF FIGURES AND ILLUSTRATIONS XI
LIST OF TABLES XVI
LIST OF APPENDICES XX
1 SUMMARY 1
1.1 KEY DATA 1
1.2 PROPERTY DESCRIPTION AND OWNERSHIP 1
1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 2
1.4 HISTORY 2
1.5 GEOLOGICAL SETTING AND MINERALIZATION 5
1.6 DEPOSIT TYPE 6
1.7 EXPLORATION STATUS 6
1.8 EXPLORATION PROCEDURES 7
1.9 MINERAL RESOURCE ESTIMATE 8
1.10 MINERAL RESERVE ESTIMATES 10
1.11 MINING METHODS 12
1.12 METALLURGICAL TESTING 13
1.13 RECOVERY METHODS 13
1.14 INFRASTRUCTURE 14
1.14.1 Access 14
1.14.2 Water 14
1.14.3 LNG Receiving, Storage and Distribution Facilities 14
1.14.4 Power Generation 14
1.14.5 Power Distribution 15
1.14.6 Tailings Management Facility 15
1.14.7 Heap Leach Facility 15
1.15 CAPITAL COSTS 16
1.16 OPERATING COSTS 16
1.17 ECONOMICS 17
1.18 ADJACENT PROPERTIES 18

1.19 CONCLUSIONS AND RECOMMENDATIONS 18
2 INTRODUCTION 21
2.1 ISSUER AND PURPOSE OF ISSUE 21
2.2 SOURCES OF INFORMATION 21
2.3 PERSONAL INSPECTIONS 21
2.4 UNITS AND ABBREVIATIONS 22
3 RELIANCE ON OTHER EXPERTS 25
3.1 METALLURGY AND PROCESS ENGINEERING 25
3.2 TRANSPORTATION 26
4 PROPERTY DESCRIPTION AND LOCATION 27
4.1 LOCATION 27
4.2 LAND POSITION AND STATUS 27
4.2.1 Property Description 27
4.2.2 Environmental 28
4.2.3 Mineral Tenure 28
4.2.4 Ownership and Agreements 28
4.2.5 Agreements and Royalties 29
4.2.6 Placer Claims 29
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 31
5.1 ACCESSIBILITY 31
5.2 PHYSIOGRAPHY 31
5.3 CLIMATE 31
5.4 WATER RIGHTS 32
5.5 POWER AVAILABILITY 32
5.6 SURFACE RIGHTS 32
6 HISTORY 33
6.1 HISTORY OF CANADIAN CREEK PROPERTY 35
7 GEOLOGICAL SETTING AND MINERALIZATION 37
7.1 REGIONAL GEOLOGY 37
7.2 PROPERTY GEOLOGY 42
7.3 MINERALIZATION 45
7.3.1 Hydrothermal Porphyry Alteration 45
7.3.2 Metallurgical Zoning 46
7.3.3 Hypogene Mineralization (HYP) 47
7.3.4 Structurally Hosted Gold Mineralization 49
8 DEPOSIT TYPES 50

9 EXPLORATION 51
9.1 EXPLORATION PROCEDURES PRIOR TO 2021 51
9.2 EXPLORATION IN 2021 53
9.2.1 Soil Sampling 53
9.2.2 Enersoft "AI" Analysis 61
10 DRILLING 64
10.1 1992-1994 DRILLING PROGRAM 64
10.2 2008 TO 2012 DRILLING 64
10.3 2013 DRILLING 65
10.4 2019 DRILLING 65
10.5 2020 DRILLING 66
10.6 2021 DRILLING 67
10.7 CANADIAN CREEK DRILLING SUMMARY 78
10.8 SENSITIVITY DATA PHOTOGRAMMETRY 78
10.9 COLLAR COORDINATES 78
10.10 SPERRY SUN SURVEYS 79
10.11 LIGHT-LOG SURVEY SYSTEM 79
10.12 ACID DIP TESTS 79
11 SAMPLE PREPARATION, ANALYSES AND SECURITY 80
11.1 SAMPLING METHOD AND APPROACH 80
11.1.1 Core Processing 80
11.1.2 Core Sampling 81
11.1.3 2021 Soil Sampling 83
11.1.4 Enersoft "GeologicAI" Scanning of Core 83
11.2 SAMPLE PREPARATION 84
11.2.1 2019 Sample Preparation 84
11.2.2 2020 Sample Preparation 84
11.2.3 2021 Sample Preparation 84
11.3 ASSAY ANALYSIS 85
11.3.1 Assay Analysis, 1992-1994 85
11.3.2 Assay Analysis, 2008-2020 85
11.3.3 Assay Analysis 2021 87
11.4 SECURITY 88
11.5 QUALITY ASSURANCE (QA) AND QUALITY CONTROL (QC) 89
11.5.1 Reference Material "Standards." 92
11.5.2 Blanks 116
11.5.3 Field Duplicates, 2020 125

11.5.4 Opinion of Qualified Person 147
12 DATA VERIFICATION 148
12.1 DATA ENTRY 148
12.1.1 1992-1994 148
12.1.2 2008-2012 149
12.1.3 2013-2021 149
12.2 DATA VERIFICATION 150
12.2.1 1992-1994 150
12.2.2 2008-2013 150
12.2.3 2019 151
12.2.4 2020 151
12.2.5 2021 152
12.3 VERIFICATION ERRORS 152
12.3.1 1992-1994 153
12.3.2 2008-2012 153
12.3.3 2019 153
12.3.4 2020 154
12.3.5 2021 154
12.4 OPINION OF QUALIFIED PERSON 154
13 MINERAL PROCESSING AND METALLURGICAL TESTING 155
13.1 METALLURGICAL SAMPLES 155
13.1.1 Data Verification 155
13.2 COMMINUTION TESTING 156
13.3 FLOTATION 158
13.3.1 2008 G&T Metallurgical Work 158
13.3.2 2009-2011 G&T Metallurgical Work 159
13.3.3 2011-2012 G&T Metallurgical Work 160
13.3.4 2022 ALS Metallurgy Program 161
13.4 DEWATERING TESTS 165
13.5 LEACHING TESTS 165
13.5.1 Kappes, Cassiday and Associates (KCA) 165
13.5.2 SGS E&S Engineering Solutions Inc. 165
13.5.3 SGS Canada 166
13.6 SART COPPER RECOVERY 167
13.7 HYDRODYNAMIC CHARACTERIZATION 168
13.7.1 2015 HydroGeoSense 168
13.7.2 2022 HydroGeoSense 169
13.8 DETERMINATION OF RECOVERIES, REAGENT, AND OTHER CONSUMABLE CONSUMPTIONS 171
14 MINERAL RESOURCE ESTIMATES 174

14.1 MINERAL RESOURCE 174
14.2 SENSITIVITY TO NSR CUTOFF 176
14.3 MINERAL RESOURCE PARAMETERS 177
14.3.1 Metal Prices 177
14.3.2 Cost and Recovery Estimates 177
14.3.3 NSR Calculations 178
14.3.4 Slope Angles 181
14.4 ADDITIONAL INFORMATION 181
14.5 DESCRIPTION OF THE BLOCK MODEL 183
14.5.1 General 183
14.5.2 Drilling Data 183
14.5.3 Geologic Controls 185
14.5.4 Cap Grades and Compositing 190
14.5.5 Descriptive Statistics 191
14.5.6 Variogram Analysis 195
14.5.7 Block Grade Estimation 202
14.5.8 Bulk Density 208
14.5.9 Resource Classifications 208
14.5.10 Comparison of 2022 and 2020 Mineral Resource 214
15 MINERAL RESERVE ESTIMATES 218
15.1 MINERAL RESERVE 218
15.2 ECONOMIC PARAMETERS 220
15.2.1 Commodity Prices 220
15.2.2 Cost and Recovery Estimates 220
15.2.3 NSR Calculation 222
16 MINING METHODS 224
16.1 OPERATING PARAMETERS AND CRITERIA 224
16.2 SLOPE ANGLES 224
16.3 MINING PHASES 228
16.4 MINE PRODUCTION SCHEDULE 234
16.5 WASTE MANAGEMENT 239
16.6 MINING EQUIPMENT 242
17 RECOVERY METHODS 243
17.1 PROCESS DESCRIPTION 243
17.2 SULPHIDE ORE PROCESS PLANT DESCRIPTION 243
17.2.1 Process Design Criteria and Major Equipment 243
17.2.2 Crushing and Coarse Ore Stockpile 248
17.2.3 Grinding and Classification 248
17.2.4 Flotation 249
17.2.5 Concentrate Dewatering and Storage 251

17.2.6 Pyrite Concentrate Deposition 251
17.2.7 Tailings Dewatering 251
17.2.8 Reagents and Consumables 252
17.2.9 Water System 252
17.3 POWER 253
17.4 AIR SERVICE 253
17.5 PROCESS CONTROL PHILOSOPHY – SULPHIDE PLANT 253
17.6 QUALITY CONTROL 254
17.7 METALLURGICAL PERFORMANCE PROJECTION 254
17.8 PROCESS DESCRIPTION 254
17.9 OXIDE ORE PROCESS PLANT DESCRIPTION 254
17.9.1 Process Design Criteria and Major Equipment 254
17.9.2 Crushing, Conveying, and Stacking 258
17.9.3 Heap Leaching 259
17.9.4 Carbon ADR Plant/SART 259
17.9.5 Reagents and Consumables 263
17.9.6 Water System 263
17.10 POWER 263
17.11 AIR SUPPLY 264
17.12 PROCESS CONTROL PHILOSOPHY – OXIDE PLANT 264
17.13 QUALITY CONTROL 264
17.14 METALLURGICAL PERFORMANCE PROJECTION 264
18 PROJECT INFRASTRUCTURE 265
18.1 ACCESS ROADS 265
18.1.1 Mine Site Access Road 265
18.1.2 Service Roads 266
18.2 PORT FACILITIES 267
18.3 SITE LAYOUT AND ANCILLARY FACILITIES 267
18.3.1 General 267
18.3.2 Truck Shop 269
18.3.3 Residence Camp 269
18.3.4 Operational Support Facilities 269
18.3.5 Guard Shed/Scale House 271
18.3.6 Airstrip 271
18.4 PROCESS BUILDINGS 271
18.4.1 Crushing Plant 271
18.4.2 Gold Recovery & SART Building 271
18.4.3 Grinding 272
18.4.4 Flotation/Reagent Storage & Mixing 272

18.5 WATER SUPPLY AND DISTRIBUTION 272
18.5.1 Fresh Water Supply 272
18.5.2 Fire Water 272
18.5.3 Potable Water 272
18.5.4 Water Supply for the Leach, ADM and SART Facilities 272
18.5.5 Process Water Supply 272
18.6 POWER GENERATION AND DISTRIBUTION 273
18.6.1 LNG Receiving, Storage and Distribution Facilities 273
18.6.2 Power Generation 273
18.6.3 Power Distribution 273
18.7 TAILINGS MANAGEMENT FACILITY 274
18.7.1 Design Basis 274
18.7.2 Tailings Characteristics 275
18.7.3 Storage Requirements 276
18.7.4 Hazard Classification 276
18.7.5 Facility Design 277
18.7.6 Water Balance 280
18.7.7 Instrumentation and Monitoring 281
18.7.8 Closure and Reclamation 281
18.8 HEAP LEACH FACILITY 282
18.8.1 Design Basis 282
18.8.2 Storage Requirements 284
18.8.3 Facility Design 284
18.8.4 Water Balance 287
18.8.5 Instrumentation and Monitoring 287
18.8.6 Closure and Reclamation 287
18.9 WASTEWATER DISPOSAL 288
18.10 COMMUNICATIONS 288
19 MARKET STUDIES AND CONTRACTS 289
20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 290
20.1 ENVIRONMENTAL & SOCIAL STUDIES 290
20.1.1 Biophysical Setting 294
20.1.2 Social and Community Setting 296
20.1.3 Environmental Disclosure 297
20.2 WASTE MANAGEMENT AND WATER MANAGEMENT 297
20.2.1 Geochemical Characterization 298
20.2.2 Waste Rock and Tailings Management 298
20.2.3 Water Management 299
20.3 PERMITTING 300
20.3.1 Existing Assessments and Permits 301
20.3.2 Licensing 301
20.3.3 Environmental and Socio-Economic Assessment Process 307

20.4 FIRST NATIONS AND COMMUNITY ENGAGEMENT 307
20.5 MINE CLOSURE AND RECLAMATION 308
20.5.1 Closure Plan, Cost Estimate and Financial Assurance 308
21 CAPITAL AND OPERATING COSTS 311
21.1 CAPITAL COST 311
21.1.1 Initial Capital Cost 311
21.1.2 Basis of Process Plant and Infrastructure Capital Cost Estimate 311
21.1.3 Direct Costs 313
21.1.4 Process Plant and Infrastructure Indirect Costs 315
21.1.5 Mine Capital Costs 316
21.1.6 Owner's Costs 317
21.2 SUSTAINING CAPITAL COSTS 317
21.3 OPERATING COSTS 317
21.3.1 Process Plant Operating & Maintenance Costs 319
21.3.2 General Administration 322
21.3.3 Mine Operating Costs 324
22 ECONOMIC ANALYSIS 327
22.1 MINE PRODUCTION STATISTICS 327
22.2 PLANT PRODUCTION STATISTICS 327
22.3 CAPITAL EXPENDITURE 328
22.3.1 Initial Capital 328
22.3.2 Sustaining Capital 328
22.3.3 Working Capital 328
22.3.4 Salvage Value 328
22.4 REVENUE 328
22.5 TOTAL CASH OPERATING COST 329
22.6 TOTAL CASH PRODUCTION COST 329
22.6.1 Royalty 329
22.6.2 Taxes 329
22.6.3 Reclamation & Closure 330
22.7 TOTAL PRODUCTION COST 330
22.8 PROJECT FINANCING 330
22.9 NET INCOME AFTER TAX 330
22.10 NPV AND IRR 330
23 ADJACENT PROPERTIES 337
24 OTHER RELEVANT DATA AND INFORMATION 339
24.1 PROJECT EXECUTION PLAN 339
24.1.1 Focus 339

24.1.2 Engineering 339
24.1.3 Procurement 339
24.1.4 Project Services 340
24.1.5 Construction Management 340
24.1.6 Contracting 340
24.1.7 Labor 341
24.1.8 Construction Completion and Turn-Over Procedure 341
24.1.9 Quality Plan 341
24.1.10 Health and Safety Plan 341
24.1.11 Camp Transition 341
24.1.12 Project Schedule 341

25 INTERPRETATION AND CONCLUSIONS 343
25.1 MINING 343
25.2 PROCESS 343
25.3 TAILINGS MANAGEMENT FACILITY 344
25.4 EXPLORATION INTERPRETATIONS 345
25.5 EXPLORATION CONCLUSIONS 346

26 RECOMMENDATIONS 347
26.1 METALLURGY AND MINERAL PROCESSING 347
26.2 TAILINGS MANAGEMENT FACILITY 347
26.3 HEAP LEACH FACILITY 348
26.4 ADDITIONAL FACILITIES 348
26.5 EXPLORATION 349

27 REFERENCES 350

LIST OF FIGURES AND ILLUSTRATIONS

FIGURE **DESCRIPTION** **PAGE**

Figure 1-1: Casino Property Location 20
Figure 4-1: Project Road Access Map 30
Figure 7-1: Regional Geology, Casino project area 39
Figure 7-2: Legend, Regional Geology, Casino Property area 40
Figure 7-3: Local Geology, Casino Property area 41
Figure 7-4: Regional Structures Overlain on Recent Aeromagnetic Survey 42
Figure 7-5: Property Geology (R. Johnson, 2018) 43
Figure 7-6: Geology of the Casino Deposit 44
Figure 7-7: Casino Property Geology - Cross Section 48
Figure 9-1: Copper and Gold in soil results (Johnson, 2018) 52
Figure 9-2: Layout of 2021 Soil Sampling Program 54
Figure 9-3: Cu assay values from on-site XRF analysis (Rio Tinto) 55
Figure 9-4: Cu assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting) 56
Figure 9-5: Mo assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting) 57
Figure 9-6: Au assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting) 58
Figure 9-7: Ag assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting) 59
Figure 9-8: As assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting) 60
Figure 9-9: "GeologicAI" Scanning Apparatus (Enersoft Inc.) 62
Figure 9-10: An example of Hyperspectral Scanning results, 2021 Drilling, Casino project 63
Figure 10-1: Casino Property Drilling Pre-1992 71
Figure 10-2: Casino Property Drilling 1992 to 1994 72
Figure 10-3: Casino Property Drilling from 2008 to 2012 73
Figure 10-4: Casino Property Drilling, from 2013 – 2019 74
Figure 10-5: Detail, Casino Property Drilling, 2013-2019 75
Figure 10-6: Casino Property Drilling, 2020 76
Figure 10-7: 2021 Diamond Drill Locations, 2021 Program (Image by H. Seeley, Wolfbear Geological) 77
Figure 11-1: Flow-cart for drill core processing and quality control procedures, 1992 and 1993 programs 90
Figure 11-2: Casino Drill Core Processing and Quality Control Procedures for 1994 91
Figure 11-3: Sample Standard CDN-CM-4 Gold Assay Results 93
Figure 11-4: Standard Sample CDN-CM-4, Copper Assay Results 93
Figure 11-5: Sample Standard CDN-CM-4 Molybdenum Assay Results 94

Figure 11-6: Sample Standard CDN-CM-7 Gold Assay Results 95
Figure 11-7: Sample Standard CDN-CM-7, Copper Assay Results 95
Figure 11-8: Sample Standard CDN-CM-7, Molybdenum Assay Results 96
Figure 11-9: Gold Assay Results, Sample Standard CU-185 97
Figure 11-10: Silver Assay results, Sample Standard CU-185 97
Figure 11-11: Copper Assay results, Sample Standard CU-185 98
Figure 11-12: Molybdenum Assay results, Sample Standard CU-185 98
Figure 11-13: Gold Assay results, Sample Standard CU-188 99
Figure 11-14: Silver Assay Results, Sample Standard CU-188 99
Figure 11-15: Copper Assay Results, Sample Standard CU-188 100
Figure 11-16: Molybdenum Assay results, Sample Standard CU-188 100
Figure 11-17: Gold assay results, Standard CU-188 102
Figure 11-18: Silver assay results, Standard CU-188 103
Figure 11-19: Copper assay results, Standard CU-188 103
Figure 11-20: Molybdenum assay results, Standard CU-188 104
Figure 11-21: Gold assay results, Standard CU-190 104
Figure 11-22: Silver assay results, Standard CU-190 105
Figure 11-23: Copper assay results, Standard CU-190 105
Figure 11-24: Molybdenum assay results, Standard CU-190 106
Figure 11-25: Gold assay results, OREAS 151a 108
Figure 11-26: Copper assay results, OREAS 151a 108
Figure 11-27: Molybdenum assay results, OREAS 151a 109
Figure 11-28: Gold assay results, OREAS 502c 110
Figure 11-29: Silver assay results, OREAS 502c 110
Figure 11-30: Copper assay results, OREAS 502c 111
Figure 11-31: Molybdenum assay results, OREAS 502c 111
Figure 11-32: Gold assay results, OREAS 506 112
Figure 11-33: Silver assay results, OREAS 506 113
Figure 11-34: Copper assay results, OREAS 506 113
Figure 11-35: Molybdenum assay results, OREAS 506 114
Figure 11-36: Gold assay results, OREAS 905 115
Figure 11-37: Copper assay results, OREAS 905 115
Figure 11-38: Gold Assay results, Blank Material 117
Figure 11-39: Silver Assay results, Blank Material 117

Figure 11-40: Copper Assay Results, Blank Material....118
Figure 11-41: Molybdenum Assay Results, Blank Material....118
Figure 11-42: Gold Assay results, Blank Material....119
Figure 11-43: Silver Assay Results, Blank Material....120
Figure 11-44: Copper Assay Results, Blank Material....120
Figure 11-45: Molybdenum Assay Results, Blank Material....121
Figure 11-46: Gold assay results, Blank material....122
Figure 11-47: Silver assay results, Blank material....123
Figure 11-48: Copper assay results, Blank material....123
Figure 11-49: Molybdenum assay results, Blank material....124
Figure 11-50: Plot of ALS Chemex Au assay vs Acme Labs Au assay for Field Duplicate Samples (2008, 2009 and 2010 samples)....126
Figure 11-51: Plot of ALS Chemex Ag analyses vs. Acme Labs Ag Analysis for Field Duplicate Samples (2008, 2009, 2010 samples)....127
Figure 11-52: Plot of ALS Chemex Cu Assay vs. Acme Labs Cu Assay for Field Duplicate Samples (2008, 2009 and 2010 Data)....128
Figure 11-53: Plot of ALS Chemex Mo assay vs Acme Labs Mo assay for Field Duplicate Samples (2008, 2009 and 2010 Data)....129
Figure 11-54: Comparison Plot between Original Gold Values and Duplicate Gold Values....130
Figure 11-55: Comparison Plot between Original Silver Values and Duplicate Silver Values....131
Figure 11-56: Comparison Plot between Original Copper Values and Duplicate Copper Values....131
Figure 11-57: Comparison Plot between Original Copper Values and Duplicate Copper Values....132
Figure 11-58: Comparison Plot between Gold Values from ALS Global and Gold Values from SGS....133
Figure 11-59: Comparison Plot between Silver Values from ALS Global and Silver Values from SGS....134
Figure 11-60: Comparison Plot between Copper Values from ALS Global and Copper Values from SGS....134
Figure 11-61: Comparison Plot between Molybdenum Values from ALS Global and Molybdenum Values from SGS....135
Figure 11-62: Comparison between Original and Duplicate Gold Values....136
Figure 11-63: Comparison between Original and Duplicate Silver Values....137
Figure 11-64: Comparison between Original and Duplicate Copper Values....137
Figure 11-65: Comparison between Original and Duplicate Molybdenum Values....138
Figure 11-66: Comparison of Gold Values from ALS Global and Bureau Veritas....139
Figure 11-67: Correlation between Gold Values from ALS Global and Bureau Veritas....140
Figure 11-68: Comparison between Silver Values from ALS Global and Bureau Veritas....140
Figure 11-69: Correlation between Silver Values from ALS Global and Bureau Veritas....141

Figure 11-70: Comparison between Copper Values from ALS Global and Bureau Veritas 141
Figure 11-71: Correlation between Copper Values from ALS Global and Bureau Veritas 142
Figure 11-72: Comparison of Molybdenum Values from ALS Global and Bureau Veritas 142
Figure 11-73: Correlation between Molybdenum Values from ALS Global and Bureau Veritas 143
Figure 11-74: Comparison of Original versus Duplicate Gold Values 145
Figure 11-75: Comparison of Original versus Duplicate Copper values 145
Figure 11-76: Comparison of Original versus Duplicate Silver values 146
Figure 11-77: Comparison of Original versus Duplicate Molybdenum values 146
Figure 14-1: Floating Cone Shell for Mineral Resource (IMC, 2022) 182
Figure 14-2: Hole Location Map (IMC, 2022) 184
Figure 14-3: Oxidation Domains on East-West Section 6,958,600N 186
Figure 14-4: Oxidation Domains on North-South Section 611,165E 187
Figure 14-5: Rock Types on East-West Section 6,958,600N 189
Figure 14-6: Probability Plot of Total Copper Composites by Oxidation Type 193
Figure 14-7: Probability Plot of Gold Composites by Oxidation Type (IMC, 2022) 194
Figure 14-8: Total Copper Variogram – Supergene Oxide (IMC, 2022) 196
Figure 14-9: Total Copper Variogram – Supergene Sulphide (IMC, 2022) 197
Figure 14-10: Total Copper Variogram – Hypogene Sulphide – Global (IMC, 2022) 198
Figure 14-11: Total Copper Variogram – Hypogene Sulphide – East-West (IMC, 2022) 199
Figure 14-12: Total Copper Variogram – Hypogene Sulphide – North-South (IMC, 2022) 200
Figure 14-13: Gold Variogram (IMC, 2022) 201
Figure 14-14: Total Copper Grades on East-West Cross Section 6,958,600N 204
Figure 14-15: Total Copper Grades on North-South Cross Section 611,165E 205
Figure 14-16: Gold Grades on East-West Cross Section 6,958,600N 206
Figure 14-17: Gold Grades on North-South Cross Section 611,165E 207
Figure 14-18: Probability Plot of Average Distance to Nearest 3 Holes – Supergene Sulphide (Source: IMC, 2022) 210
Figure 14-19: Probability Plot of Average Distance to Nearest 3 Holes – Hypogene Sulphide (Source: IMC, 2022) 211
Figure 14-20: Resource Classification on East-West Cross Section 6,958,600N (Source: IMC, 2022) 212
Figure 14-21: Resource Classification on North-South Cross Section 611,065E (Source: IMC, 2022) 213
Figure 16-1: Open Pit Design Sectors (Knight-Piésold, 2012) 226
Figure 16-2: Final Pit Design (IMC, 2021) 227
Figure 16-3: Mining Phase 1 (IMC, 2022) 229
Figure 16-4: Mining Phase 2 (IMC, 2022) 230
Figure 16-5: Mining Phase 3 (IMC, 2022) 231

Figure 16-6: Mining Phase 4 (IMC, 2022) 232
Figure 16-7: Mining Phase 5 (IMC, 2022) 233
Figure 16-8: Maximum Extent of Waste Storage Areas and Stockpiles (IMC, 2022) 240
Figure 16-9: Year-1 Showing Temporary Leach and ROM Ore Stockpiles (IMC, 2022) 241
Figure 17-1: Simplified Sulphide Process Flowsheet 247
Figure 17-2: Simplified Oxide Process Flowsheet 257
Figure 18-1: Proposed New Access Road 266
Figure 18-2: Overall Site Plan 268
Figure 18-3: Plant Site Plan 270
Figure 18-4: Tailings Conceptual Mass Balance 276
Figure 18-5: Tailings Management Facility Starter Embankment – Conceptual Section (KP, 2022a) 277
Figure 18-6: Tailings Management Facility Main Embankment – Conceptual Section (KP, 2022a) 278
Figure 18-7: Main Embankment Construction – Conceptual Schematic (KP, 2022a) 279
Figure 18-8: Heap Leach Facility General Arrangement (KP, 2022b - in progress) 283
Figure 18-9: Heap Leach Facility Cross-Section (KP, 2022b - in progress) 284
Figure 20-1: Regional Project Area 291
Figure 20-2: General Arrangement End of Year 1 292
Figure 20-3: General Arrangement End of Year 27 293
Figure 23-1: Adjacent properties, Casino Property Area (as of March 8, 2022) 338

LIST OF TABLES

TABLE	DESCRIPTION	PAGE
Table 1-1:	Mineral Resource for Mill Material at C$ 6.11 NSR Cutoff	9
Table 1-2:	Mineral Resource for Leach Material at C$ 6.61 NSR Cutoff	9
Table 1-3:	Mineral Resource for Copper, Gold, and Silver (Mill and Leach)	9
Table 1-4:	Mineral Reserve	11
Table 1-5:	Capital Cost Summary	16
Table 1-6:	Mill Operating Costs Per Tonne	16
Table 1-7:	Heap Leach Operating Costs	16
Table 1-8:	Mining Operating Costs	17
Table 1-9:	Financial Results Summary	17
Table 1-10:	Copper and Gold Price Sensitivity	18
Table 2-1:	Dates of Site Visits and Areas of Responsibility	21
Table 2-2:	Abbreviations Used in this Document	22
Table 5-1:	Mean Monthly Temperature and Precipitation Values	32
Table 7-1:	Stratigraphic Column	38
Table 7-2:	Leached Cap & Supergene Minerals	47
Table 10-1:	Summary of drill targets in 2020	67
Table 10-2:	2021 Drill Collar data	68
Table 10-3:	Significant Intercepts, 2021 Diamond Drilling Program	68
Table 10-4:	Summary of Canadian Creek Drilling	78
Table 11-1:	Casino 2021 Drill-Hole Requirements by Campaign	82
Table 11-2:	2019 Standard reference Material from WCM Minerals	96
Table 11-3:	Performance of Standard CU-185 during 2019 Drill Program Sampling	101
Table 11-4:	Performance of Standard CU-188 during 2019 Drill Program Sampling	101
Table 11-5:	Reference material "Standards", utilized in 2021 (OREAS)	101
Table 11-6:	Performance of Standard CU-188 during 2020 drill program	106
Table 11-7:	Performance of Standard CU-190 during 2020 drill program	107
Table 11-8:	Reference material "Standards", utilized in 2021 (OREAS)	107
Table 11-9:	Summary of Results for OREAS 151a	109
Table 11-10:	Summary of results for OREAS 502c	112
Table 11-11:	Summary of results for OREAS 506	114
Table 11-12:	Summary of Results for OREAS 905	116

Table 11-13: Performance of Blank Material during 2019 Drill Program Sampling 119
Table 11-14: Performance of Blank Material during 2020 Drill Program 122
Table 11-15: Performance of Blank material during 2021 Drill program 124
Table 11-16: Summary of Duplicate (Core) Pair Performance during 2019 Drill Program Sampling 132
Table 11-17: Summary of Check (Pulps) Pair performance during 2019 Drill Program Sampling 135
Table 11-18: Summary of Dupluicate (core) Pair Performance during 2020 Drill Program 138
Table 11-19: Summary of Check (Pulps) Pair Performance during 2020 Drill Program 144
Table 11-20: Summary of Duplicate (core) Pair Performance during 2021 Drilling program 144
Table 13-1: Summary of Comminution Results 156
Table 13-2: Summary of G&T SAG Mill Comminution (SMC) Test Results 156
Table 13-3: Summary of SAG Design Results and Crushed Bond Test Results 157
Table 13-4: Predicted Production Rate 157
Table 13-5: G&T Flotation Composite Assays 158
Table 13-6: 2010 Supergene Sulphide Composite Locked Cycle Test Results 159
Table 13-7: Flowsheet Development Locked Cycle Test Results 160
Table 13-8: Hypogene Composites 160
Table 13-9: Locked Cycle Test Results 161
Table 13-10: Copper/Molybdenum Separation Cleaner Test 161
Table 13-11: Supergene Locked Cycle Recoveries to Concentrate 162
Table 13-12: Cleaner Circuit Recoveries for Locked Cycle Test Results 163
Table 13-13: Predicted Recoveries to Copper/Molybdenum Concentrate 163
Table 13-14: Copper Concentrate Chemistry 164
Table 13-15: Molybdenum Concentrate Chemistry 164
Table 13-16: Extractions and Reagent Consumptions from Open Cycle and Locked Cycle Cyanidation 165
Table 13-17: Extractions and Reagent Consumptions from Column Tests Investigating Lithology 166
Table 13-18: Comparison of Recovery 166
Table 13-19: Comparison of Head Grade 167
Table 13-20: SART Results 168
Table 13-21: Comparison of Head Grade 168
Table 13-22: Solids Specific Gravity 168
Table 13-23: Comparison of Head Grade 169
Table 13-24: Solids Specific Gravity 170
Table 13-25: Dry Bulk Density (t/m^3) 170
Table 13-26: Saturation Moisture (%) 170

Table 13-27: Drain Down Moisture (%) 170
Table 13-28: Flotation Operational Parameters 172
Table 13-29: Heap Leach Operational Parameters 173
Table 14-1: Mineral Resource for Mill Material at C$6.11 NSR Cutoff 175
Table 14-2: Mineral Resource for Leach material at C$6.61 NSR Cutoff 175
Table 14-3: Mineral Resource for Copper, Gold, and Silver (Mill and Leach) 175
Table 14-4: Mineral Resource – Mill Material by Various NSR Cut-offs (C$) 176
Table 14-5: Mineral Resource – Leach Material by Various NSR Cut-offs (C$) 177
Table 14-6: Economic Parameters for Mineral Resource (C$) 180
Table 14-7: Casino Drilling by Date and Company 183
Table 14-8: Oxidation Zone Types 185
Table 14-9: Model Rock Types 188
Table 14-10: Cap Grades and Number of Assays Capped 190
Table 14-11: Summary Statistics of Assays 191
Table 14-12:Summary Statistics of 7.5 m Composites 192
Table 14-13: Statistics of Specific Gravity Measurement by Oxidation Zone 208
Table 14-14: Comparison of Economic Parameters for 2020 versus 2022 Mineral Resource 215
Table 14-15: Comparison of 2022 and 2020 Mineral Resource – Mill Material 215
Table 14-16: Comparison of 2022 and 2020 Mineral Resource – Leach Material 217
Table 15-1: Mineral Reserve 219
Table 15-2: Economic Parameters for Mine Design (C$) 221
Table 16-1: Recommended Slope Angles (Knight-Piésold, 2012) 225
Table 16-2: Proposed Plant Production Schedule 236
Table 16-3: Mine Production Schedule 237
Table 16-4: Production Schedule for Leach Material 238
Table 16-5: Mining Equipment Requirements 242
Table 17-1: Sulphide Process Design Criteria 244
Table 17-2: Major Process Equipment 246
Table 17-3: Bulk Flotation Cells 250
Table 17-4: Moly Flotation Cells 251
Table 17-5: Process Consumables 252
Table 17-6: Metallurgical Performance Estimate 254
Table 17-7: Oxide Process Design Criteria 255
Table 17-8: Major Process Equipment 256

Table 17-9: Process Consumables 263
Table 17-10: Metallurgical Performance Estimate 264
Table 20-1: Summary of applicable Legislation, Regulations and Regulatory Approvals for the Casino Mine 302
Table 20-2: Proposed mine closure phases and activities for the Project 309
Table 20-3: Proposed reclamation and closure of mine components 309
Table 21-1: Initial Capital Cost Summary 311
Table 21-2: Burdened Labor Rates 313
Table 21-3: Mining Capital – Mine Equipment and Mine Development (C$ x 1000) 316
Table 21-4: Mine Development Direct Costs Plus Contingency (C$ x 1000) 317
Table 21-5: LOM Sustaining Capital Costs 317
Table 21-6: Mining Cost 318
Table 21-7: Concentrator Cost 318
Table 21-8: Heap Leach Cost 318
Table 21-9: Operating Cost – Mine Site Cost Summary 319
Table 21-10: Concentrator Cost by Cost Element 319
Table 21-11: Heap Leach Cost by Cost Element 320
Table 21-12: Power Cost Summary 321
Table 21-13: Reagents and Consumables 321
Table 21-14: Operating Cost – Concentrator Cost Summary – Typical Year of Operation 323
Table 21-15: Operating Cost – Heap Leach Cost Summary – Typical Year of Operation 324
Table 21-16: Summary of Total and Unit Mining Costs 325
Table 21-17: Mine Operating Cost by Cost Centre 325
Table 21-18: Mine Operating Cost by Consumables versus Labor 325
Table 21-19: Mine Parts and Consumables Costs 326
Table 22-1: Life of Mine Mineralized Material, Waste Quantities and Mineralized Material Grade 327
Table 22-2: Metal Prices Used in Economic Analysis 328
Table 22-3: Sensitivity Analysis (After tax, CAD$M) 331
Table 22-4: Metal Price Sensitivity 332
Table 22-5: Project Cash Flow 333
Table 22-6: Financial Model 334
Table 24-1: Overview Schedule 342

LIST OF APPENDICES

APPENDIX	DESCRIPTION
A	Feasibility Study Contributors and Professional Qualifications
	• Certificate of Qualified Person ("QP")
B	List of Claims

1 SUMMARY

This Report was prepared for Casino Mining Corporation (CMC), a wholly owned subsidiary of Western Copper and Gold Corporation (Western) as well as for Western itself by M3 Engineering & Technology Corporation (M3) in association with Independent Mining Consultants (IMC), Knight Piésold Ltd. (KP), Aurora Geosciences, and Hemmera.

The purpose of this report is to provide a feasibility study on the Casino Property. This report conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

1.1 KEY DATA

The key details about this project are as follows:

1. Casino is primarily a copper and gold project that is expected to process 120,000 dry tonnes of ore per day (t/d) or 43.8 million dry tonnes per year (t/y). Metals to be recovered are copper (Cu), gold (Au), molybdenum (Mo), and silver (Ag).

2. Based on the economic analysis, the Property will produce the following over the life of the mine from the concentrator and heap leach facility:

    a. Gold – 6.95 million ounces
    b. Silver – 36.09 million ounces
    c. Copper – 4.27 billion pounds
    d. Molybdenum – 346 million pounds

3. The process will include a conventional single-line SAG mill circuit (Semi-Autogenous Ball Mill Crusher, or SABC) followed by conventional flotation to produce concentrate for sale. In addition to the concentrator, there will be a separate carbon-in-column facility to recover precious metals from heap leached oxide ore. Gold and silver bullion (doré) produced will be shipped by truck to metal refineries.

4. The Property will require the construction of a power plant and will generate its own electrical power using LNG to fuel the generator drivers.

5. The Property has several routes of access, including by the Yukon River, by aircraft, winter roads, and existing trails. A network of paved highways provides access to the region from the Port of Skagway, Whitehorse, and northern British Columbia. Paved roads to the Property currently exist up to Carmacks. A new, all weather, gravel road will be constructed by the project to connect Casino to Carmacks via the existing Freegold Road. The new access road will, in general, follow the existing Casino Trail that will be upgraded to support trucking from Carmacks to Casino.

6. Fresh water will be sourced from the Yukon River.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/10), in west central Yukon, in the northwest trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse.

To the west, Newmont is developing the Coffee Project. To the north and to the west, White Gold Corp. has a large number of claims and is actively exploring them. Approximately 100 km to the east, Minto Metals Corp. operates the Minto Mine, which produces copper concentrate.

The project is located on Crown land administered by the Yukon Government and is primarily within the Selkirk First Nation traditional territory. The Tr'ondek Hwechin traditional territory lies to the north and the proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south. The White River First Nation and Kluane First Nation are also potentially impacted by the project. The Casino Property lies within the Whitehorse Mining District and consists of 1,136 full and partial Quartz Claims and 55 Placer Claims acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 21,276.61 ha. The total area covered by Casino Placer Claims is 490.32 ha. Casino Mining Corp. (CMC) is the registered owner of all claims, although certain portions of the Casino property remain subject to royalty agreements. The claims covering the Casino property are discussed further in Section 4 of this document.

Figure 1-1, at the end of this section, shows the site's location in Yukon Territory as well as other points of interest relevant to this Report.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Casino Mine is located in Central Yukon, roughly 150 km northwest of Carmacks, at approximately N62° 44' 25", W138° 49' 32". Current site access is by small aircraft using the existing 760 m airstrip, by winter road and from the Yukon River.

Either road or barge service will provide early access for construction equipment, camp construction and initial equipment. A barge landing area at Britannia Creek and the Yukon River is currently in service.

The project plan includes a new airstrip. The project also plans a new 132 km year-round access road from the end of the Freegold Road, presently extending 70 km northwest of the village of Carmacks.

The climate at the Casino Project area can generally be described as continental and cold. Winters are long, cold, and dry, with snow generally on the ground from late September through mid-May. Summers are short, mild, and wet, with the greatest monthly precipitation falling in July. Average daytime temperature in winter reaches a maximum of -13 degrees Celsius in January, dropping to -22 degrees Celsius overnight. On average, the daytime temperatures in July reach a maximum of 20 degrees Celsius, with overnight lows of 7.7 degrees Celsius. The mean annual precipitation for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow.

1.4 HISTORY

The first documented work in the present Casino Property area comprised the working of placer claims on Canadian Creek, recorded in 1911 by J. Britton and C. Brown. A study by D.D. Cairnes, of the Geological Survey of Canada (GSC) in 1917, recognized huebnerite ($MnWO_4$) in the heavy-mineral concentrates, and also that gold and tungsten mineralization was derived from an intrusive complex on Patton Hill. The total placer gold production is unknown, although from 1980-1985 placer mining yielded 1,615 troy ounces of gold (Au).

The first recorded bedrock mineral discovery occurred in 1936 when J. Meloy and A. Brown located silver-lead-zinc (Ag-Pb-Zn) veins approximately 3 km south of the Canadian Creek placer workings. Over the next several years the Bomber and Helicopter vein systems were explored by hand trenches and pits. In 1943, the Helicopter claims were staked followed by staking of the Bomber and Airport claims in 1947.

Lead-silver mineralization was the focus of exploration on the property until 1968. Noranda optioned the property in 1948 and Rio Tinto re-optioned the property in 1963. During this time trenching, mapping, and sampling were conducted.

L. Proctor purchased the claims in 1963 and formed Casino Silver Mines Limited to develop the silver-rich veins. The veins were explored and developed intermittently by underground and surface workings from 1965 to 1980. In total, 372.5 tonnes of argentiferous galena, assaying 3,689 grams/tonne (g/t) Ag, 17.1 g/t Au, 48.3% Pb, 5% Zn, 1.5% Cu, and 0.02% bismuth (Bi), were shipped to the Trail, British Columbia smelter.

Based on the recognition of porphyry copper potential, the Brynelsen Group acquired Casino Silver Mines Limited and, from 1968 to 1973, exploration for a porphyry target was directed jointly by Brameda, Quintana, and Teck Corporation. Exploration included extensive soil sampling and geophysical surveys and trenching programs, eventually leading to the discovery of the Casino deposit in 1969. From 1969 to 1973, various parties, including Brameda Resources, Quintana Minerals and Teck Corporation, completed diamond drilling programs on the property.

Archer, Cathro & Associates (1981) Ltd. (Archer Cathro) optioned the property in 1991 and assigned the option to Big Creek Resources Ltd. In 1992, a program consisting of 21 HQ holes totaling 4,729 m systematically assessed the gold potential in the core of the deposit for the first time. In 1992, Pacific Sentinel Gold Corp. (PSG) acquired the property and commenced a major exploration program. The 1993 program included surface mapping and 50,316 m of HQ and NQ-sized drilling in 127 holes. All but one of the 1992 drill holes were deepened in 1993. PSG drilled an additional 108 diamond drill holes totalling 18,085 metres in 1994, completing the delineation drilling commenced in 1993. PSG also performed metallurgical, geotechnical, and environmental work which was used in a scoping study in 1995. This study envisioned a large-scale open pit mine and a conventional flotation concentrator that would produce a copper-gold concentrate for sale to Pacific Rim smelters.

First Trimark Resources and CRS Copper Resources subsequently obtained the property and, using the Pacific Sentinel Gold data, published a Qualifying Report in 2003 to bring the resource estimate into compliance with National Instrument 43-101 requirements. The two firms combined to form Lumina Copper Corporation in 2004 and issued an update of the Qualifying Report later that year.

In November 2006, Western Copper Corporation acquired Lumina Copper Corporation and the Casino Deposit. In the fall of 2011, Western Copper Corporation spun out all other assets except the Casino Deposit and changed its name to Western Copper and Gold Corporation (Western). Western also created a wholly owned subsidiary, the “Casino Mining Corporation” (CMC).

In 2007, Western conducted an evaluation of the Bomber Vein System and the southern slope of Patton Hill by VLF-EM and Horizontal Loop EM geophysical surveying and soil geochemistry. Environmental baseline studies were also initiated in 2007. In 2008, Western reclaimed the old camp site and constructed a new exploration camp next to the Casino airstrip. Western drilled a camp water well and two exploration holes totaling 1,163 m, to obtain fresh core samples for metallurgical and waste characterization tests. Both exploration holes twinned PSG holes to confirm historical Cu, Mo, and Au grades. Later that year, M3 Engineering produced a pre-feasibility study for Western.

In 2009, Western completed 22.5 km of Direct Current Resistivity and Induced Polarization (DC/IP) surveying and Magnetotelluric Tensor Resistivity (MT) surveying using the “Titan” system of Quantec Geosciences Ltd. The company also drilled 10,943 m in 37 diamond drill holes, of which 27 were infill holes along the north slope of Patton Hill designed to convert inferred resource and non-defined material to the measured and indicated categories. Drilling identified supergene Cu-Mo mineralization in this area. The remaining 10 holes, totaling 4,327 m, were drilled to test geophysical targets.

In 2010, infill and delineation drilling continued, with most of the drilling done to the north and west of the deposit. Drilling also defined hypogene mineralization at the southern end of the deposit. The company also drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit, and several holes for hydrogeological studies. The geotechnical drilling continued in 2011 (41 holes, 3,163 m) and 2012 (6 holes, 228 m). This work culminated in the publishing of a pre-feasibility study in 2011 and a feasibility study in 2013.

In 2019, Western carried out a program of infill drilling, comprising 13,590 m in 72 holes. This program was designed to upgrade mineralization in the inferred resource category located along the margin of the deposit to the indicated category.

In 2020, CMC completed a diamond drilling program comprising 12,007.54 m in 49 holes, targeting three main areas: the Gold, Northern Porphyry and Casino West zones. Drilling at the Gold Zone was designed to test for higher grade mineralization along the south and west boundaries of the deposit. Northern Porphyry zone drilling targeted potential northern extensions of the deposit. Drilling at the Casino West zone was designed to test for continuation of the deposit along the south flank of Canadian Creek.

In 2021, CMC completed a diamond drilling program comprising 6,074.97 m in 22 holes. Of these, 16, comprising 5 resource confirmation holes, 3 metallurgical testing holes, and 8 for geotechnical analysis were drilled within the Casino resource boundaries. An additional six exploration holes were drilled outside of the deposit resource area, and seven short geotechnical holes were drilled in the proposed heap leach, tailings management facility and processing facility areas.

In July 2021 Western completed a Preliminary Economic Assessment (PEA) report, incorporating data from drilling from 1992 through 2019. The PEA recommended advancement to a Feasibility Study to determine mineral reserves for the deposit.

In mid-2019, Western acquired the Canadian Creek property, adjacent to the west of the Casino property, from Cariboo Rose Resources Ltd., leading to the issuance of a new Mineral Resource Statement in late 2020. Exploration on the Canadian Creek property dates from 1992 when Archer Cathro staked the Ana Claims. In 1993, Eastfield Resources Ltd. acquired these claims, expanded the Ana Claim block, and explored the expanded property by soil geochemical sampling, trenching, and drilling, (Johnston, 2018). This work was directed towards exploration for additional porphyry deposits. The 1993 program was followed by extensive field programs in 1996, 1997 and 1999, comprising Induced Polarization (IP) surveying, road construction, and trenching on the Ana, Koffee, Maya and Ice claims. In 2000, Eastfield on the Ana undertook another drill campaign, Koffee Bowl, and the newly acquired Casino “B” claims immediately east of the Casino deposit. The Casino “B” holes confirmed the presence of auriferous mineralization discovered in 1994 by PSG. Modest exploration programs were conducted in 2003, 2004, and 2005, mostly over the Casino “B” area. In 2007, a five-hole core drill program at Casino “B” targeted gold and copper soil anomalies and ground magnetic “high” features.

In 2009, following discovery of gold on Underworld Resources’ nearby White Gold property, a major exploration program at Canadian Creek targeted gold potential outside of previous areas of porphyry copper exploration. Soil surveying revealed areas returning > 15 ppb Au, associated with anomalous As, Bi, and Sb (antimony) values, extending more than four kilometers ENE from the Casino deposit. The IP surveys showed numerous strong chargeability highs, many coinciding with the gold-in-soil anomalies, which were subsequently tested with 10 core holes. The holes intersected clay-altered structures with sheeted pyrite veins, and narrow, structurally controlled clay-altered structures with pyrite and quartz-carbonate veins. With few exceptions gold grades were < 1 gpt, and widths were less than 3 m.

In 2011, additional soil sampling and ground geophysical surveying and trenching were completed. The soil sampling completed coverage of the entire Canadian Creek property, whereas a limited-extent IP survey identified two zones of

> 20 mv/V of chargeability. The trenching program identified several areas with anomalous gold values, including 2,890 ppb and 4,400 ppb Au.

In 2016, Cariboo Rose, which had by then acquired the property from Eastfield, completed a modest program of trenching, prospecting, and in-fill soil sampling. Trenching work at the Ana portion of the Canadian Creek property returned locally anomalous Au, widely spread anomalous As, Bi and Sb, and locally high Ag values, generally confined to narrow structures.

Cariboo Rose's 2017 exploration program comprised surface work at the Kana and Malt West gold targets and a reverse circulation (RC) drill program that tested a variety of gold targets across the property. A total of 2,151.27 m of RC drilling in 24 holes was completed. This work confirmed gold and silver mineralization to be limited to structures less than 3 m wide, rarely traceable over more than 100 m.

1.5 GEOLOGICAL SETTING AND MINERALIZATION

The geological setting of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous-age (72-74 Ma), east-west elongated porphyry stock called the Patton Porphyry, which intrudes mid-Cretaceous granitoids of the Dawson Range Batholith and Paleozoic schists and gneisses of the Wolverine Creek suite of the Yukon Tanana Terrane (YTT). Intrusion of the Patton Porphyry caused brecciation of both the earlier intrusive rocks and surrounding country rocks along the northern, southern, and eastern contacts of the stock. Brecciation is best developed in the eastern end of the stock where the brecciated zone is up to 400 m wide in plan view. To the west, along the north and south contacts, the breccias narrow gradually to less than 100 m. The overall dimensions of the intrusive complex are approximately 1.8 by 1.0 km.

The main body of the Patton Porphyry is a relatively small, mineralized stock measuring approximately 300 m by 800 m, surrounded by a potassically-altered intrusion breccia in contact with rocks of the Dawson Range Batholith. Elsewhere, the Patton Porphyry forms discontinuous dykes ranging from less than one to tens of metres in width, cutting both the Patton Porphyry Plug and the Dawson Range Batholith. The overall composition of the Patton Porphyry is rhyodacite, with phenocrysts of a dacitic composition within a quartz latite matrix. The porphyry commonly includes abundant distinct phenocrysts of plagioclase and lesser phenocrysts of biotite, hornblende, quartz, and opaque minerals.

The Intrusion Breccia comprises granodiorite, diorite, and fragments of Paleozoic meta-igneous and metasedimentary rocks, in a fine-grained Patton Porphyry matrix. It may have formed along the margins, in part by the stoping of blocks of wall rock. An abundance of Dawson Range granitoid inclusions occurs prominently at the southern contact of the main plug, whereas abundances of Wolverine Creek metamorphic rocks increase along the northern contact, and bleached diorite fragment abundance increases along the eastern contact of the main plug. Strong potassic and phyllic alteration locally destroys primary textures.

Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Higher grades occur in the breccias and gradually decrease outbound from the contact zone, both towards the centre of the stock and outward into the granitoids and schists. Several metallogenic settings were identified as follows:

- **Leached Cap Mineralization (CAP)** – This oxidized gold-bearing zone is copper-depleted due to weathering processes and has a lower specific gravity relative to the underlying zones. Weathering has resulted in significant clay alteration, and is most intense at surface, decreasing with depth.

- **Supergene Oxide Mineralization (SOX)** – This zone is enriched in copper oxide and hydrous copper carbonate minerals, with trace molybdenite. It generally occurs as a thin layer above the supergene sulphide

zone. Where present, the supergene oxide zone averages 10 m in thickness, and may contain chalcanthite, malachite and brocanthite with minor azurite, tenorite, cuprite, and neotocite.

- **Supergene Sulphide Mineralization (SUS)** – Supergene copper mineralization occurs in a zone of sulphide mineral enrichment derived from leaching of copper-bearing mineralization from the overlying Leached Cap. The zone, located below the leached cap and above the hypogene zone, extends to 200 m of depth, with an average thickness of 60 m. Grades of the supergene sulphide zone vary widely, but are highest in fractured and highly pyritic zones, due to their ability to promote chalcocite precipitation. Copper grades in the Supergene Sulphide zone are almost double those in the Hypogene.

- **Hypogene Mineralization (HYP)** – Hypogene mineralization occurs as disseminated mineralization, stock-work veins and breccias throughout the various alteration zones below the Supergene zone. Significant Cu-Mo mineralization is related to the potassically-altered breccia surrounding the core Patton Porphyry, and in the adjacent phyllically-altered host rocks of the Dawson Range Batholith. The breccias surrounding the Patton Porphyry are host to the highest Cu values on the property.

1.6 DEPOSIT TYPE

The Casino deposit is best classified as a Calc-Alkalic Porphyry type deposit associated with a tonalite intrusive stock. Primary Cu, Au and Mo mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Higher Cu-Au grades occur in the breccias and gradually decrease outwards away from the contact zone both towards the centre of the stock and outward into the granitoids and schists. A general zoning of the primary sulphides occurs, with chalcopyrite and molybdenite occurring in the central tonalite and breccias, grading outward into pyrite-dominated mineralization in the surrounding granitoids and schists. Alteration accompanying the sulphide mineralization comprises an earlier phase of potassic (K) alteration and a later overprinting of phyllic alteration. The potassic alteration typically comprises secondary biotite and K-feldspar as pervasive replacement and veins. Quartz stockwork zones and anhydrite veinlets also occur. Phyllic alteration consists of sericite and vein and replacement-style silicification.

The Casino deposit is unusual amongst Canadian porphyry copper deposits in having a well-developed enriched blanket of secondary copper mineralization similar to that found in deposits in Chile and the southwestern United States, such as the Escondida and Morenci deposits. Unlike other Canadian porphyry deposits, the Casino deposit's enriched copper blanket was not eroded by glacial action. At Casino, weathering during the Tertiary Period leached the copper from the upper 70 m of the deposit, forming the leached cap, and re-deposited it lower in the deposit, forming the supergene enrichment zones. This created a layer-like sequence consisting of an upper leached zone up to 70 m thick, where all sulphide minerals have been oxidized and copper removed, resulting in a bleached, limonitic leached cap containing residual gold. Beneath the leached cap is a zone up to 100 m thick of secondary copper sulphide mineralization, primarily chalcocite and minor covellite, and including thin, discontinuous units of supergene copper oxide mineralization directly underlying the leached cap. The copper grades of the enriched, blanket-like zone can be up to twice that of the underlying unweathered hypogene zone of primary copper mineralization, the latter comprising pyrite, chalcopyrite and lesser molybdenite. The hypogene copper mineralization is persistent at depth, extending more than 600 m below surface, and beyond the deepest drill holes.

1.7 EXPLORATION STATUS

In 2009, Quantec Geoscience Limited of Toronto, Ontario performed Titan-24 DC/IP surveying, as well as an MT survey over the entire grid. MT surveys provide high resolution and deep penetration (to 1 km), and the Titan DC/IP survey provides reasonable depth coverage to 750 m.

In 2010, all of Pacific Sentinel's historic drill core stored at the Casino Property was re-logged to provide data for the new lithological and alteration models.

In 2011 Western focused on geotechnical, metallurgical and baseline environmental studies, but also drilled several exploration holes, prior to changing its name to Western Copper and Gold Corp (Western), and creating its wholly owned subsidiary, Casino Mining Corporation (CMC) late that year. In 2011, the program involved 41 drill holes for a total of 3,163.26 m. In 2012, CMC continued with the geotechnical and metallurgical drilling; six holes (228.07 m) were drilled for metallurgical sampling.

During the 2019 field season, Western focused on exploration drilling for the primary purpose of updating the resource base of the Casino Project. A total of 13,594.63 m in 72 holes were drilled.

During the 2020 field season, Western completed a diamond drilling program of 12,008 m in 49 core holes. The program focused on identification of high-grade gold intercepts in the "Gold Zone," as well as expansion of the main deposit to the north and west. Results are included in this Feasibility Study.

During the 2021 field season, a total of 6,074.97 metres in 22 core holes was completed. The assay values were not used in the determination of the updated resource described in this report. Four categories of diamond drilling were employed, as follows:

Resource Confirmation Drilling: 5 holes for 1,483 m.
Metallurgical Drilling: 3 holes for 1,001 m.
Geotechnical Drilling (Deposit area): 8 holes for 1,957 m.
Exploration Drilling: 6 holes for 1,634 m.

The 2021 program also included the drilling of seven geotechnical holes testing ground conditions at the proposed Tailings Management Facility, Heap Leach facility and Mineral Processing site. Roughly 40% of core from 1992 to 2012, all of the 2021 core, and much of the 2020 core underwent scanning by the GeologicAI instrument of Enersoft Inc.

Also in 2021, an extensive B-horizon soil sampling program covering areas north, east, and south of the Casino deposit was completed, leading to onsite identification from on-site XRF results of three targets, which were subsequently drilled. Three further geochemical targets were identified from lab assay results.

1.8 EXPLORATION PROCEDURES

Exploration on the property over its history included prospecting, geological mapping, multi-element soil geochemistry, magnetic and IP surveys, trenching and drilling. Targeting of early drilling on the Casino Deposit was based mainly on coincident Cu-Mo soil anomalies. Since 1993, with the exception of a Titan TM Survey, exploration centered on the Casino deposit comprised drilling on a grid pattern using a core drill with NQ and NTW widths, with a smaller number of holes drilled with HQ diameter core. The 2021 drilling program utilized PQ-sized coring gear for the metallurgical holes, and HTW gear for the resource confirmation, geotechnical and exploration holes. These were reduced to NTW-sized core when drilling conditions became challenging.

On the recently acquired Canadian Creek Property, exploration to 2017 comprised grid soil, ground magnetic and IP surveys to generate trenching and drilling targets. Initially the focus was to locate porphyry copper mineralization. After 2016, the focus changed to exploration for gold mineralization similar to that discovered at nearby Coffee Creek.

Soil sampling west of the Casino Deposit was done from the mid-1990s through to 2011. The soil results show a coincident Cu-Au anomaly at the 50 ppm Cu and 15 ppb Au threshold levels respectively, extending westward for approximately 3 km from the Casino Deposit. This anomaly has been tested by 16 core holes.

Ground magnetic surveys with a line spacing of 100 m were undertaken over the Canadian Creek property in 2011 and in 2017. IP surveys were carried out in 1993, 1996, 2009 and 2011. The surveys in the 1990s used a pole-dipole array with an a-spacing of 75 m and an n 1 to 4 depth profile. The 2009 survey was a pole-dipole survey using an a-spacing of 25 m and an n 1 to 6 depth profile, and the 2011 pole dipole survey used an a-spacing of 25 m and an n 1 to 8 depth profile. In general, the surveys used small "n" spacings and have a limited depth profile. The surveys identified a number of high chargeability anomalies which remain to be tested.

The 2021 B-horizon soil sampling was conducted using a 200-metre station spacing and a 200-metre line spacing, resulting in evenly spaced sample locations in all cardinal directions throughout the surveyed area.

Drilling procedures for resource confirmation and exploration holes were the same as for 2020 drilling, utilizing HTW-sized equipment. Geotechnical drilling in the deposit area utilized HTW-sized split tube ("Triple-tube") coring steel. The metallurgical holes utilized PQ-sized equipment for a more representative sample for testing.

The 2021 program also included re-analysis of much of the historic core, as well as 2020 and 2021 core, by the heli-portable GeologicAI unit of Enersoft Inc. The unit performed hyperspectral, LiDAR, XRF and high-resolution photography on drill core. Roughly 40% of the 1992 – 2012 core, all of the 2021 and some of the 2020 core underwent analysis by the GeologicAI unit.

1.9 MINERAL RESOURCE ESTIMATE

The Mineral Resource for the Casino Project includes Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resource amenable to heap-leach recovery methods (leach material). Also, the Mineral Resource is reported inclusive of the Mineral Reserve presented in the next section. Table 1-1 presents the Mineral Resource for mill material. Mill material includes the supergene oxide (SOX), supergene sulphide (SUS), and hypogene sulphide (HYP) mineral zones. Measured and Indicated Mineral Resources amount to 2.26 billion tonnes at 0.15% total copper, 0.18 g/t gold, 0.016% molybdenum, and 1.4 g/t silver and contained metal amounts to 7.45 billion pounds of copper, 12.9 million ounces gold, 791.2 million pounds of moly and 103.1 million ounces of silver. Inferred Mineral Resource is an additional 1.37 billion tonnes at 0.10% total copper, 0.14 g/t gold, 0.009% moly and 1.1 g/t silver and contained metal amounts to 3.03 billion pounds of copper, 6.1 million ounces of gold, 286.0 million pounds moly and 50.5 million ounces of silver for the Inferred Mineral Resource in mill material.

Table 1-2 presents the Mineral Resource for leach material. Leach material is oxide dominant leach cap (CAP or LC) mineralization. The emphasis of leaching is the recovery of gold in the leach cap. Copper grades in the leach cap are low, but it is expected some metal will be recovered. Measured and Indicated Mineral Resources amount to 231.7 million tonnes at 0.04% total copper, 0.25 g/t gold and 1.9 g/t silver and contained metal amounts to 196.9 million pounds of copper, 1.88 million ounces gold and 14.1 million ounces of silver. Inferred Mineral Resource is an additional 40.9 million tonnes at 0.05% total copper, 0.20 g/t gold and 1.4 g/t silver and contained metal amounts to 46.9 million pounds of copper, 270,000 ounces of gold and 1.9 million ounces of silver for the Inferred Mineral Resource in leach material.

Table 1-3 presents the Mineral Resource for combined mill and leach material for copper, gold, and silver. Measured and Indicated Mineral Resources amount to 2.49 billion tonnes at 0.14% total copper, 0.18 g/t gold, and 1.5 g/t silver. Contained metal amounts to 7.64 billion pounds copper, 14.8 million ounces gold, and 117.2 million ounces of silver for Measured and Indicated Mineral Resources. Inferred Mineral Resource is an additional 1.41 billion tonnes at 0.10% total copper, 0.14 g/t gold and 1.2 g/t silver. Contained metal amounts to 3.08 billion pounds of copper, 6.3 million ounces of gold and 52.3 million ounces of silver for the Inferred Mineral Resource. The Mineral Resource for molybdenum is as shown with mill material since it will not be recovered for leach material.

The Mineral Resources are based on a block model developed by IMC during December 2021. This updated model incorporated the 2020 Western drilling and updated geologic models.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a floating cone pit shell to demonstrate "reasonable prospects for eventual economic extraction" to meet the definition of Mineral Resources in NI 43-101.

Table 1-1: Mineral Resource for Mill Material at C$ 6.11 NSR Cutoff

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq %	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Measured	144.9	40.09	0.30	0.38	0.024	2.1	0.64	953	1.8	75.2	9.6
Indicated	2,114.2	20.34	0.14	0.16	0.015	1.4	0.29	6,493	11.1	716.0	93.5
M+I	**2,259.0**	**21.60**	**0.15**	**0.18**	**0.016**	**1.4**	**0.31**	**7,446**	**12.9**	**791.2**	**103.1**
Inferred	1,371.5	15.41	0.10	0.14	0.009	1.1	0.21	3,029	6.1	286.0	50.5

Table 1-2: Mineral Resource for Leach Material at C$ 6.61 NSR Cutoff

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	43.3	23.79	0.05	0.44	2.7	0.47	51.5	0.62	3.7
Indicated	188.4	11.47	0.04	0.21	1.7	0.23	145.4	1.27	10.4
M+I	**231.7**	**13.77**	**0.04**	**0.25**	**1.9**	**0.27**	**196.9**	**1.88**	**14.1**
Inferred	40.9	11.33	0.05	0.20	1.4	0.22	46.9	0.27	1.9

Table 1-3: Mineral Resource for Copper, Gold, and Silver (Mill and Leach)

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	188.2	36.34	0.24	0.40	2.2	1,005.0	2.4	13.3
Indicated	2,302.6	19.61	0.13	0.17	1.4	6,638.1	12.4	103.9
M+I	**2,490.7**	**20.88**	**0.14**	**0.18**	**1.5**	**7,643.1**	**14.8**	**117.2**
Inferred	1,412.5	15.30	0.10	0.14	1.2	3,075.5	6.3	52.3

Notes:

1. The Mineral Resources have an effective date of 29 April 2022, and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources for leach material are based on prices of US$3.50/lb copper, US$1650/oz gold, and US$22/oz silver.
5. Mineral Resources for mill material are based on prices of US$3.50/lb copper, US$1650/oz gold, US$22/oz silver, and US$12.00/lb molybdenum.
6. Mineral Resources are based on NSR Cutoff of C$6.61/t for leach material and C$6.11/t for mill material.
7. NSR value for leach material is as follows:
   NSR (C$/t) = $15.21 x copper (%) + $50.51 x gold (g/t) + $0.210 x silver (g/t), based on copper recovery of 18%, gold recovery of 80%, and silver recovery of 26%.
8. NSR value for hypogene sulphide mill material is:
   NSR (C$/t) = $73.81 x copper (%) + $41.16 x gold (g/t) + $213.78 x moly (%) + $0.386 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 50% silver, and 78.6% molybdenum.
9. NSR value for supergene (SOX and SUS) mill material is:
   NSR (C$/t) = $80.06 x recoverable copper (%) + $43.03 x gold (g/t) + $142.11 x moly (%) + $0.464 x silver (g/t), based on recoveries of 69% gold, 60% silver, and 52.3% molybdenum. Recoverable copper = 0.94 x (total copper – soluble copper).
10. Table 14-6 accompanies this Mineral Resource and shows all relevant parameters.

11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. AuEq and CuEq values are based on prices of US$3.50/lb copper, US$1650/oz gold, US$22/oz silver, and US$12.00/lb molybdenum, and account for all metal recoveries and smelting/refining charges.
13. The Mineral Resource is reported inclusive of the Mineral Reserve.

1.10 MINERAL RESERVE ESTIMATES

Table 1-4 presents the Mineral Reserve estimate for the Casino Project. It can be seen that there are Mineral Reserves amenable to milling and Mineral Reserves amenable to heap leaching. The Proven and Probable Mineral Reserves amenable to milling amount to 1.22 billion tonnes at 0.19% total copper, 0.22 g/t gold, 0.021% molybdenum and 1.7 g/t silver. The Proven and Probable Mineral Reserve amenable to heap leaching amounts to 209.6 million tonnes at 0.26 g/t gold, 0.036% copper and 1.9 g/t silver. The effective date of this Mineral Reserve estimate is June 13, 2022. The low-grade stockpile portion of the Mineral Reserve is economic, but lower grade, material that will be stockpiled and processed at the end of open-pit operations. The Mineral Reserve estimate is also based on an exchange rate of US$ 0.80 = C$ 1.00." or if you prefer, C$ 1.25 = US$ 1.00.

The Mineral Reserve estimate is based on an open pit mine plan and mine production schedule developed by IMC. The Mineral Reserve estimate is based on commodity prices of US$ 3.25 per pound copper, US$1550 per ounce gold, US$ 12.00 per pound molybdenum and US$22.00 per ounce silver. Measured Mineral Resource in the mine production schedule was converted to Proven Mineral Reserve and Indicated Mineral Resource in the schedule was converted to Probable Mineral Reserve.

The Mineral Reserves are classified in accordance with the “CIM Definition Standards – For Mineral Resources and Mineral Reserves” adopted by the CIM Council (as amended, the “CIM Definition Standards”) in accordance with the requirements of NI 43-101. Mineral Reserve estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained. The project is in a jurisdiction friendly to mining.

IMC does not believe that there are significant risks to the Mineral Reserve estimate based on metallurgical or infrastructure factors or environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. There has been a significant amount of metallurgical testing, however recoveries lower than forecast would result in loss of revenue for the project. Other risks to the Mineral Reserve estimate are related to economic parameters such as prices lower than forecast or costs higher than the current estimates. The impact of these is modeled in the sensitivity study with the economic analysis in Section 22.

All of the mineralization comprised in the Mineral Reserve estimate with respect to the Casino Project is contained on mineral titles controlled by Western Copper and Gold.

Table 1-4: Mineral Reserve

Mineral Reserve (Milling):	Tonnes Mt	NSR (C$/t)	Tot Cu (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Proven Mineral Reserve	140.1	38.50	0.31	0.39	0.024	2.1	0.67	944	1.8	74.9	9.4
Mill Ore	124.2	41.20	0.32	0.43	0.027	2.2	0.72	885	1.7	72.6	8.8
Low Grade Stockpile	16.0	17.54	0.17	0.15	0.007	1.1	0.29	59	0.1	2.3	0.6
Probable Mineral Reserve	1,076.9	23.68	0.17	0.19	0.021	1.6	0.36	4,135	6.7	497.1	55.5
Mill Ore	825.1	26.15	0.19	0.21	0.024	1.7	0.40	3,484	5.6	430.9	45.9
Low Grade Stockpile	251.9	15.57	0.12	0.14	0.012	1.2	0.24	651	1.1	66.2	9.6
Proven/Probable Reserve	**1,217.1**	**25.38**	**0.19**	**0.22**	**0.021**	**1.7**	**0.40**	**5,079**	**8.5**	**571.9**	**64.9**
Mill Ore	949.2	28.12	0.21	0.24	0.024	1.8	0.44	4,369	7.3	503.5	54.7
Low Grade Stockpile	267.8	15.69	0.12	0.14	0.012	1.2	0.25	710	1.2	68.5	10.2
Mineral Reserve (Heap Leach):	Tonnes Mt	NSR (C$/t)	Gold (g/t)	Tot Cu (%)	Moly (%)	Silver (g/t)	AuEq (g/t)	Gold (Moz)	Copper (Mlbs)	Moly (Mlbs)	Silver (Moz)
Proven Mineral Reserve	42.9	22.52	0.45	0.055	n.a.	2.7	0.47	0.62	51.8	n.a.	3.7
Probable Mineral Reserve	166.8	11.14	0.22	0.031	n.a.	1.8	0.23	1.17	113.5	n.a.	9.4
Proven/Probable Leach Reserve	**209.6**	**13.47**	**0.26**	**0.036**	**n.a.**	**1.9**	**0.28**	**1.78**	**165.3**	**n.a.**	**13.1**

Notes:

1. The Mineral Reserve estimate has an effective date of 13 June 2022 and was prepared using the CIM Definition Standards (10 May 2014).
2. Columns may not sum exactly due to rounding.
3. Mineral Reserves are based on commodity prices of US$3.25/lb Cu, US$1550/oz Au, US$12.00/lb Mo, and US$22.00/oz Ag.
4. Mineral Reserves amenable to milling are based on NSR cut-offs that vary by time period to balance mine and plant production capacities (see Section 16). They range from a low of $6.11/t to a high of $25.00/t.
5. NSR value for supergene (SOX and SUS) mill material is NSR (C$/t) = $73.63 x recoverable copper (%) + $40.41 x gold (g/t) + $142.11 x moly (%) + 0.464 x silver (g/t), based on recoveries of 69% gold, 52.3% molybdenum and 60% silver. Recoverable copper = 0.94 x (total copper – soluble copper).
6. NSR value for hypogene (HYP) mill material is NSR (C$/t) = $67.88 x copper (%) + $38.66 x gold (g/t) + $213.78 x moly (%) + $0.386 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 78.6% molybdenum, and 50% silver.
7. Mineral Reserves amenable to heap leaching are based on an NSR cut-off of $6.61/t.
8. NSR value for leach material is NSR (C$/t) = $14.05 x copper (%) + $47.44 x gold (g/t) + $0.210 x silver (g/t), based on recoveries of 18% copper, 80% gold, and 26% silver.
9. AuEq and CuEq values are based on prices of US$ 3.25/lb Cu, US$ 1550/oz Au, US$ 12.00/lb Mo, and US$ 22.00/oz Ag, and account for all metal recoveries and smelting/refining charges.
10. The NSR calculations also account for smelter/refinery treatment charges and payables.
11. Table 15-2 accompanies this Mineral Reserve estimate and shows all relevant parameters.

1.11 MINING METHODS

This Feasibility Study (FS) is based on a conventional open pit mine plan. Mine operations will consist of drilling large diameter blast holes (31 cm), blasting with a bulk emulsion, and loading into large off-road trucks with cable shovels and a hydraulic shovel. Mineral reserves amenable to processing will be delivered to the primary crusher or various stockpiles. Waste rock will be placed inside the limits of the tailings management facility (TMF). There will be a fleet of track dozers, rubber-tired dozers, motor graders and water trucks to maintain the working areas of the pit, stockpiles, and haul roads.

The following general parameters guided the development of the mining plan:

- Mill material is limited to about 1.2 billion tonnes, CMC elected to limit the capacity of the TMF to be comparable to the concept and overall physical characteristics of the TMF design favored in the Best Available Tailings Technology Study (BATT study).
- Total mine waste to be co-disposed with tailings is limited to about 600 million tonnes,
- Mill capacity is a nominal 120,000 tonnes per day (t/d), but actual plant throughput for the schedule is based on hardness of the various material types, and usually exceeds 120,000 t/d.

Based on the mining plan developed for this study, the commercial life of the project is 27 years after an approximate 3-year pre-production period. Total mill ore is 1.22 billion tonnes at 0.189% copper, 0.217 g/t gold, 0.0213% molybdenum, and 1.66 g/t silver. Only measured and indicated mineral resource is included in the mine production schedule and converted to proven and probable mineral reserve.

In addition to the potential mill ore, there is mineral reserve mined from the leach cap zone that is amenable to processing by crushing and heap leaching. This amounts to 209.6 million tonnes at 0.265 g/t gold, 1.95 g/t silver, and 0.036% total copper.

Total waste in the mine plan amounts to 611.3 million tonnes. The waste material by material type is as follows:

- 58.5 million tonnes of overburden.
- 144.6 million tonnes of leach cap material.
- 33.2 million tonnes of supergene oxide material.
- 125.1 million tonnes of supergene sulphide material.
- 249.8 million tonnes of hypogene material.

The overburden is placed in the overburden stockpile in Canadian Creek, north of the pit. The remaining waste is disposed in the tailing management facility in three facilities for mine waste: 1) the North Waste area which contains 248.4 million tonnes, 2) the Divider Dam which contains 134.4 million tonnes, and 3) the West Waste storage area which contains 164.6 million tonnes. About 5 million tonnes of mine waste will be used in the Starter Dam for the TMF embankment. The material will be placed by trucks and dozers.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 209.6 million tonnes of spent, non-reactive material, assuming all the potential leach material is processed.

- A low-grade stockpile southeast of the pit that has the capacity for 161.8 million tonnes, and a low-grade stockpile east of the pit that contains 106.1 million tonnes, both which will be processed at the end of the mine life.

- There will also be supergene oxide (SOX) stockpile south of the pit to store mining phase 1 SOX. It will be reclaimed during mining Years 4 through 13. The maximum size of this facility is estimated at 35.3 million tonnes. The SOX stockpile and the leach pad overlap by a small amount, but the SOX stockpile will be reclaimed before the leach pad gets to its final limits.

- There will be two stockpiles for leach ore. Leach ore mined during preproduction, 33.3 million tonnes, will be stockpiled in a temporary stockpile west of mining phase 1, but within the final pit limits. This material will be reclaimed and processed early in Year 7 a couple of years before waste stripping commences in that area. A larger facility for leach ore storage is located east of the pit. This is expected to reach a maximum size of 79.2 million tonnes during Year 11 and will be reclaimed by the end of Year 21.

1.12 METALLURGICAL TESTING

Flotation testing by ALS Metallurgy from 2008 to 2012 indicated that copper concentrate grades of 28% copper could be routinely achieved at good copper recoveries with a primary grind size of 80% passing 200 µm and a regrind of 80% passing 25 µm. Gold and silver will be recovered with the copper concentrate. Molybdenum will be recovered to a molybdenum concentrate in a separate flotation circuit.

The average metal recoveries expected from mill processing following the planned mill feed schedule are noted below:

- Copper recovery to copper concentrate, percent 86
- Gold recovery to copper concentrate, percent 67
- Silver recovery to copper concentrate, percent 53
- Molybdenum recovery to molybdenum concentrate, percent 71

Column leach test work completed in 2021 by SGS Canada on the oxide cap ore crushed to minus 3.8 cm (1.5 inch) showed that good recoveries of gold and acceptable cyanide consumptions could be obtained by integrating the cyanide heap leach with the SART process. Metallurgical results obtained in 2021 on samples tested by SGS Canada indicated that gold recovery from the heap leach could be increased by crushing the ore going to the heap leach to minus 1.9 cm (3/8 inch). Hydrodynamic characterization testing indicated that agglomeration will not be required with the finer crush size. A three-stage crushing circuit has been incorporated into this feasibility study.

The metal recoveries expected from oxide cap heap leach processing are based on:

- Gold recovery, percent 80
- Silver recovery, percent 26
- Copper recovery to SART precipitate, percent 18

1.13 RECOVERY METHODS

A mine plan was developed to supply mill ore to a conventional copper sulphide flotation plant with the capacity to process mill ore at a nominal rate of 120,000 t/d, or 43.8 million tonnes per year (Mt/y). Actual annual throughput will vary depending on the mill ore hardness encountered during the period. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year.

Both sulphide copper-molybdenum mill ore and oxide gold leach ore will be processed. Copper-molybdenum mill ore will be transported from the mine to the concentrator facility and oxide gold leach ore will be transported from the mine to a crushing facility ahead of a heap leaching and gold recovery facility.

Copper-molybdenum mill ore will be processed by crushing, grinding, and flotation to produce copper and molybdenum sulphide mineral concentrates. Copper concentrate will be loaded into highway haul trucks and transported to the Port of Skagway for ocean shipment to market. Molybdenum concentrate will be bagged and loaded onto highway haul trucks for shipment to market.

Oxide gold ore will be crushed and leached with an aqueous leach solution. Gold in the enriched (or pregnant) leach solution will be recovered using carbon absorption technology to produce gold doré bars. The enriched leach solution will also be treated to recover copper and cyanide and produce a copper sulphide precipitate. The copper sulphide precipitate will be bagged and loaded onto highway haul trucks for shipment to market. Recovery methods are discussed more in depth in Section 17.

1.14 INFRASTRUCTURE

1.14.1 Access

The region is serviced by paved all-weather roads connecting the towns of Carmacks and Whitehorse in the Yukon with the Port of Skagway Alaska. With the completion of the 132 km Casino access road, the project will have an all-weather access route through Carmacks to Whitehorse (approx. 380 km) and to the Port of Skagway (550 km). The Port of Skagway has existing facilities to store and load-out concentrates as well as facilities to receive bulk commodity shipments, fuels, and connection to the Alaska Marine Highway. The Port of Skagway is developing plans to expand these facilities to better serve the expanding mining activity in the Yukon and Alaska.

The City of Whitehorse is the government, financial and commercial hub of the Yukon with numerous business and service entities to support the project and represents a major resource to staff the project. Whitehorse has an international airport and provides commercial passenger and freight services for the region.

A new airstrip will be constructed at the mine to accommodate appropriately sized aircraft. The existing airstrip will be razed in preparation for grading for process facilities.

1.14.2 Water

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above-ground insulated 762 mm (30”) diameter by 17.4 km long pipeline with four pump stations to the 22,000 m^3 capacity freshwater pond near the concentrator. The design capacity of the freshwater collection and transfer system will be 2,500 m^3/hr with a maximum of 3,650 m^3/hr with all pumps running.

1.14.3 LNG Receiving, Storage and Distribution Facilities

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large 10,000 m^3 site-fabricated storage tank to provide fuel for the power plant. An LNG receiving station is provided to unload the LNG tankers and transfer the LNG into the storage facility. An LNG vaporization facility is provided to convert the LNG into gas at a suitable supply pressure to operate the power generation equipment.

1.14.4 Power Generation

Electrical power generation for the Project will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed in the vicinity of the main workforce housing complex to provide power to the camp, for construction activities, and to oxide crushing, conveying and heap leach facilities that go into operation before the main power plant is operational.

The Supplementary Power Plant will consist of three 2,250-kilowatt (kW) diesel internal combustion engines (ICE). Two of the generators will remain at the Workforce Housing complex and the third will be relocated to the Sand Cyclone (Area 640) facility to provide standby/emergency power to this area after the concentrator start-up.

A Main Power Plant will be constructed at the Casino main mill and concentrator complex to supply the electrical energy required for operations throughout the mine site. The primary electrical power generation will be provided by three Gas Turbine driven generators (two Single Fuel Gas Turbines, one Dual Fuel Gas Turbine) and a steam generator, operating in combined cycle mode (CCGT) with a total installed capacity of approximately 200 megawatts (MW). The nominal running load to the mine and concentrator complex is about 130 MW. Three diesel ICE driven generators will provide another 6.75 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines will be fueled by natural gas (supplied as liquefied natural gas, or LNG). One of the three will have Dual Fuel capabilities - LNG and Diesel.

1.14.5 Power Distribution

The 34.5 kV distribution systems will radiate from a 34.5 kV switchgear line-up with feeders to the SAG mill, Ball Mill No. #1, Ball Mill No. #2, and feeders to the mill and flotation areas in cable tray using insulated copper conductors. Overhead line feeder circuits with aluminum conductor steel reinforced (ACSR) will be provided for the tailings reclaim water, fresh water from the Yukon River, crushing/conveying and SART/ADR, camp site and two feeders to the pit loop.

Electric power utilization voltages will be 4,160 volts for motors 300 horsepower (hp) and above, 575 volts for three-phase motors 250 hp and below. For lighting, small loads and building services 600/347 or 208/120 volts will be the utilization voltage.

1.14.6 Tailings Management Facility

A single Tailings Management Facility (TMF) will be constructed south of the open pit for storage of tailings and potentially reactive waste rock generated from mining. The TMF will store approximately 805 Mt of tailings and 615 Mt of potentially reactive waste rock and overburden materials. The TMF embankments will be constructed using a combination of local borrow and cyclone underflow sand produced from Non-Acid Generating (NAG) tailings. A total of approximately 491 Mt of NAG tailings will be used for dam construction. The TMF will be constructed with centerline raises of the dam, to a final crest elevation of El. 1000 m. See Figure 18-6 that provides a schematic of the dam dimensions.

1.14.7 Heap Leach Facility

A Heap Leach Facility (HLF) will be constructed on a southeast facing hill-slope, approximately one kilometer south of the Open Pit. The HLF operations will commence during pre-production stripping of the Open Pit. The HLF has a design capacity of 210 million tonnes (Mt) of leach cap material. The heap leach pad will be stacked with ore and leached from Year -2 through Year 22 of mine operations. The ore will be stacked at a nominal rate of approximately 9.1 Mt per year.

The ore will be stacked on a prepared pad, with a composite liner system to maximize leachate collection and minimize seepage losses. A double composite liner system will be constructed within the lower portion of the HLF and this area will function as an in-heap water management pond. The double liner system will include a leak detection and recovery system (LDRS) to intercept and collect potential leakage through the upper liner. The in-heap water management pond area will be impounded by a confining embankment, constructed from mine waste rock material.

The HLF will be developed in stages by loading in successive lifts, upslope from the base platform developed within the in-heap water management pond area, behind the confining embankment. The HLF will be developed by stacking

ore in eight-meter lifts to establish a final overall slope of 2.5H:1V. Intermittent wider benches will be constructed to limit the vertical height of the HLF to a maximum of approximately 120 m.

1.15 CAPITAL COSTS

Total initial capital investment in the Project is estimated to be $3.62 billion, which represents the total direct and indirect cost for the complete development of the Project, including associated infrastructure and power plant. Table 1-5 shows how the initial capital is distributed between the various components, including $751 million for sustaining costs.

Table 1-5: Capital Cost Summary

Cost Item	Total (C$M)
Process Plant and Infrastructure	
Project Directs including freight	2,116
Project Indirects	431
Contingency	369
Subtotal	**2,916**
Mining	
Mine Equipment	433
Mine Preproduction	228
Subtotal	**661**
Owner's Costs	41
Total Initial Capital Costs	**3,617**
Sustaining Capital	751
Total Life of Mine Capital Costs	**4,369**

1.16 OPERATING COSTS

Operating costs for the milling operation were calculated per tonne of ore processed through the mill over the life of mine as shown in Table 1-6.

Table 1-6: Mill Operating Costs Per Tonne

Category	LOM (C$/t)
Milling	$6.42
General & Administrative	$0.46
Total	**$6.88**

Heap leach operating costs were calculated per tonne of ore processed through the heap leach over the life of the heap leach as shown in Table 1-7.

Table 1-7: Heap Leach Operating Costs

Category	LOM (C$/t)
Heap Leach Operation	$1.93
ADR/SART	$4.80
Total	**$6.73**

Mining costs were calculated to average $2.30 per tonne of material moved and $3.65 per tonne of ore.

Table 1-8: Mining Operating Costs

Category	(C$/t)
Cost per tonne material (material moved)	$2.30
Cost per tonne mill feed (mill + heap material)	$3.65
Cost per tonne mill feed	$4.28

The combined mining and milling costs are $11.16 per tonne ore milled for the life of mine, which compares favorably to the life-of-mine net smelter return of $29.08 per tonne at Base Case metal prices.

1.17 ECONOMICS

This economic analysis is based on proven and probable mineral reserves. The Study indicates that the potential economic returns from the Project justify its further development and securing the required permits and licenses for operation. The financial results of the Study were developed under commodity prices that were based on analyst projections of long-term metal prices and C$:US$ exchange rate ("Base Case" prices). Note that an exchange rate of C$:US$ of 0.80 was used for the capital cost estimation for all metal price scenarios. Table 1-9 summarizes the financial results:

Table 1-9: Financial Results Summary

Category and Units	Base Case
Copper (US$/lb)	US$3.60
Molybdenum (US$/lb)	US$14.00
Gold (US$/oz)	US$1,700
Silver (US$/oz)	US$22.00
Exchange Rate (C$:US$)	0.80
NPV pre-tax (5% discount, C$M)	$5,768
NPV pre-tax (8% discount, C$M)	$3,473
IRR pre-tax (100% equity)	21.2%
NPV after-tax (5% discount, C$M)	$4,059
NPV after-tax (8% discount, C$M)	$2,334
IRR after-tax (100% equity)	18.1%
LOM pre-tax free cash flow (C$M)	$13,713
LOM after-tax free cash flow (C$M)	$10,019
Payback period (years)	3.3
Net Smelter Return (C$/t milled)	$29.08
Copper Cash Cost* (C$/lb)	($1.00)

*C1 cash costs, net of by-product credits.

The financial results of the Study are significantly influenced by copper and gold prices, as shown in Table 1-10.

Table 1-10: Copper and Gold Price Sensitivity

Copper Price (US$/lb)*	$3.00	$3.50	$3.60	$4.00	$4.50	$5.00
NPV pre-tax (8%) (C$ 000s)	$2,547,382	$3,318,938	**$3,473,249**	$4,090,494	$4,862,051	$5,633,607
NPV after-tax (8%) (C$ 000s)	$1,654,597	$2,221,387	**$2,334,396**	$2,786,432	$3,351,478	$3,916,523
IRR pre-tax	18.2%	20.7%	**21.2%**	23.0%	25.3%	27.4%
IRR after-tax	15.5%	17.7%	**18.1%**	19.7%	21.6%	23.5%
Payback (years)	3.8	3.4	**3.3**	3.0	2.8	2.6
Gold Price (US$/oz)*	**$1,300**	**$1,500**	**$1,700**	**$1,850**	**$2,050**	**$2,200**
NPV pre-tax (8%) (C$ 000s)	$2,411,886	$2,942,568	**$3,473,249**	$3,871,260	$4,401,942	$4,799,953
NPV after-tax (8%) (C$ 000s)	$1,551,049	$1,944,312	**$2,334,396**	$2,626,958	$3,017,042	$3,309,604
IRR pre-tax	17.5%	19.4%	**21.2%**	22.5%	24.2%	25.5%
IRR after-tax	14.9%	16.5%	**18.1%**	19.2%	20.7%	21.8%
Payback (years)	4.0	3.6	**3.3**	3.1	2.9	2.8

*All other metal prices except those noted are the same as the Base Case.

1.18 ADJACENT PROPERTIES

Several quartz mineral claim blocks and placer claims registered to other owners are staked adjacent to and in the general vicinity of CMC's claim block. Some of the placer claims on Canadian and Britannia Creeks overlap the Casino claims in the area of the pit. These placer claims along the upper part of Canadian creek are located within the projected pit shell and are worked by their owners on a seasonal basis with small heavy equipment. The northwestern boundary of the Casino property adjoins the Coffee Creek project of Newmont Mining. The property hosts a structurally controlled gold deposit in metamorphic rocks of the Yukon Tanana terrane and granitoids of mid Cretaceous age. The mineralization is associated with quartz carbonate and illite alteration and is best described as an orogenic deposit. The project is at a pre-feasibility stage of development.

The northeastern boundary of the Casino property abuts the "Betty and Hayes" property held by White Gold Corp. This property abuts the northern boundary of the narrow eastern extension of the Casino property. At this time, the property has undergone fairly early stages of exploration for similar orogenic-style gold mineralization to that within the Coffee Creek property.

Part of the eastern extension is also directly surrounded by the Idaho claim block held by Atac Resources Ltd.

1.19 CONCLUSIONS AND RECOMMENDATIONS

The economic results of the Study demonstrate that the project has positive economics and warrants development. Standard industry practices, equipment and processes were used in this study. The project is based on conventional open pit mining and typical, well understood, processing methods. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could affect the reliability or confidence in the project based on the data and information made available.

Based on the results of this study, it is recommended that the project advance into the execution planning phase and an application for environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act be prepared to continue the permitting process.

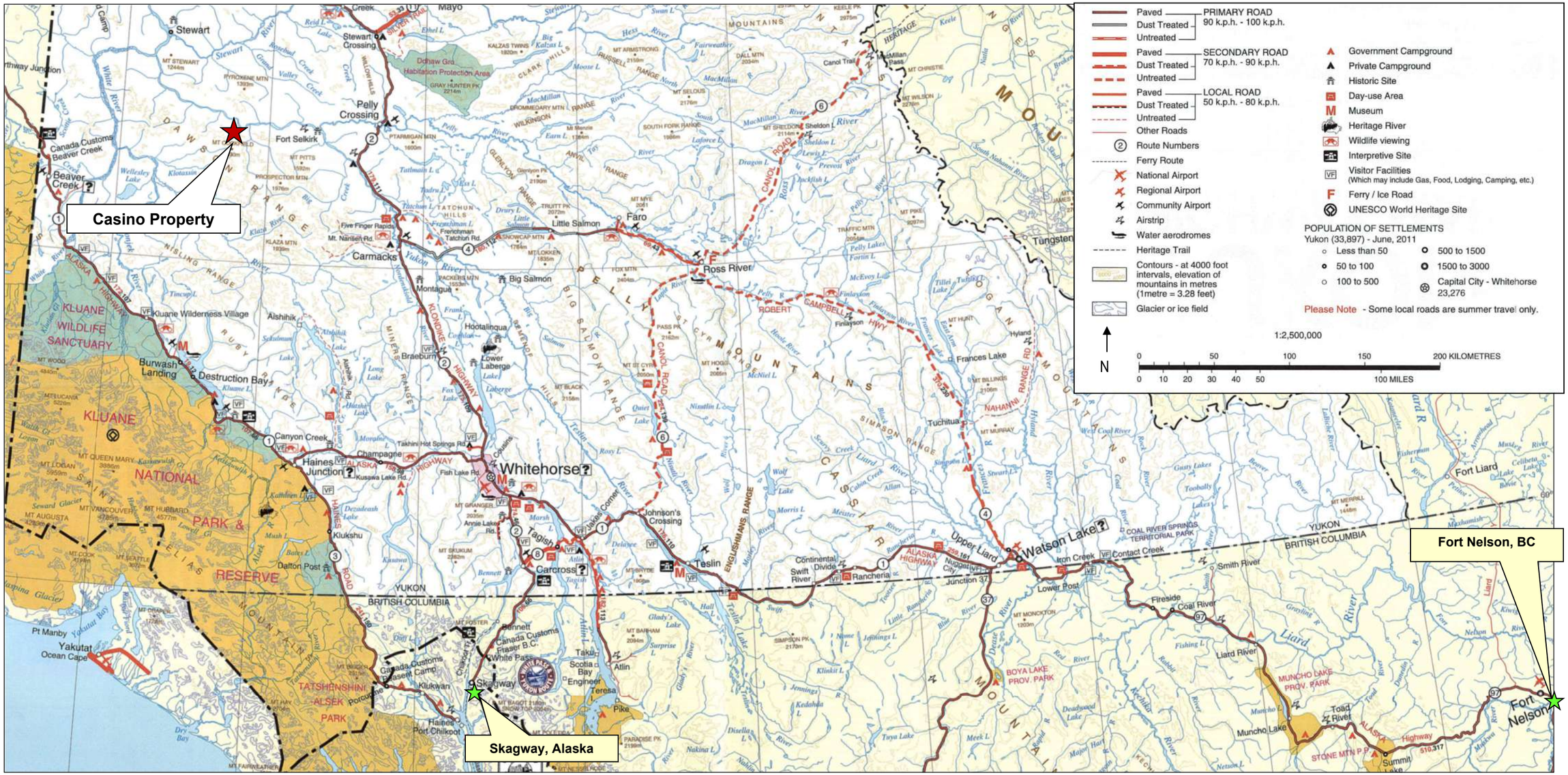
The 2019, 2020, and 2021 programs comprised 39,372.91 m of diamond drilling on the Casino and Canadian Creek properties, effectively delineating the extent of the Casino deposit. The 2020 program results indicate the previously identified Canadian zone does not have significant mineral potential, and that there are no discrete "Gold" and "North

Porphyry" zones. Results of drilling, inclusive of 2020, indicate the presence of a "Deposit Core" of higher-grade material in the east-central deposit area, both within the leached cap and underlying sulphide mineralized zones.

The 2021 drilling included several holes east of, and topographically lower than, Patton Hill. One of these returned a high-grade interval that coincides with the surface trace of the Casino fault, indicating the fault trace may represent a target for higher-grade mineralization. Farther east, one exploration hole revealed an interval having geochemical signatures, including anomalous Au-Ag values, indicative of "Bonanza-style" veining, although of lower grades than typical Bonanza-style zones. The 2021 drilling results are not incorporated into this feasibility study.

The 2021 soil sampling program identified three anomalies (A through C) from on-site XRF analysis. Three more anomalies (D, E and F) were identified from lab analysis. Of these, Anomaly F has a geochemical signature most indicative of porphyry-style Cu-Mo-Ag-Au mineralization.

The remaining undrilled exploration holes proposed for 2021 are recommended to undergo drill testing, as well as further drilling along the trace of the Casino Fault, particularly to the southeast. Additional drill holes targeting the surface strike projection of the "Bonanza" zone farther east are also recommended. Detailed B-horizon soil sampling, at a 100-metre line spacing and 50-metre station spacing, are recommended for soil anomalies D, E and F.



(Source: Yukon Highway Map, Yukoninfo.com)

Figure 1-1: Casino Property Location

2 INTRODUCTION

2.1 ISSUER AND PURPOSE OF ISSUE

This Report was prepared for Casino Mining Corporation (CMC), a wholly owned subsidiary of Western Copper and Gold Corporation (Western) as well as for Western itself, by M3 Engineering & Technology Corporation (M3) in association with Independent Mining Consultants (IMC), GeoSpark Consulting Inc., Knight Piésold Ltd. (KP), and Aurora Geosciences.

The purpose of this report is to provide an Economic Assessment (Feasibility Study, FS) on the Casino property. The estimate of mineral resources and mineral reserves contained in this report conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserve definitions (May 2014) referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

The effective date of the mineral resource was April 29, 2022, and the effective date of the mineral reserve was June 13, 2022.

2.2 SOURCES OF INFORMATION

The main sources of information for this Feasibility Study include the drillhole database provided to IMC in digital form. Also, various geologic solids that were reviewed by IMC and incorporated into the resource model. A geotechnical report by Knight-Piésold with slope angle recommendations was also used for the resource cone shell and pit design.

2.3 PERSONAL INSPECTIONS

A summary of the Qualified Persons (QPs) responsible for the content of this report is shown in Table 2-1.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Daniel Roth	M3 Engineering & Technology Corporation	PE, P.Eng.	August 6, 2021	Sections 1,1.1-1.4, 1.14.1, 1.15, 1.16, 1.17, 1.19, 2, 3, 4, 5, 18, 18.1-18.4, 18.9, 18.10, 19, 21 (except 21.1.5, 21.3.1 and 21.3.3), 22, 24, 26, 26.4 and corresponding section 27
Michael G. Hester	Independent Mining Consultants, Inc.	F Aus IMM	September 7, 2021	Sections 1.9, 14 and corresponding section 27
John M. Marek	Independent Mining Consultants, Inc.	P. Eng	September 7, 2021	Sections 1.10, 1.11, 15, 16, 21.1.5, 21.3.3, 25.1 and corresponding section 27
Laurie Tahija	M3 Engineering & Technology Corporation	MMSA-QP	N/A	Sections 1.12, 1.13, 13, 17, 21.3.1, 25.2, 26.1 and corresponding section 27
Carl Schulze	Aurora Geosciences	P. Geo.	September 9-26, 2020	Sections 1.4-1.8, 1.18, 1.19, 6, 7, 8, 9, 10, 11, 12, 23, 25.4, 25.5, 26.5 and corresponding section 27
Daniel Friedman	Knight Piésold Ltd.	P.Eng.	N/A	Sections 1.14.6, 1.14.7, 18.7, 18.8, 25.3, 26.2, 26.3 and corresponding section 27
Pat Dugan	M3 Engineering & Technology Corporation	PE	N/A	Sections 1.14.2-1.14.5, 18.5, 18.6 and corresponding section 27
Scott Weston	Hemmera Envirochem Inc.	P. Geo.	N/A	Section 20 and corresponding section 27

2.4 UNITS AND ABBREVIATIONS

This report generally uses the SI (metric) system of units, including metric tonnes. The term "tonne" rather than "ton" is commonly used to denote a metric ton and is used throughout the report. Unless otherwise specified, currency is in Canadian dollars ($ or C$). Units and abbreviations used are listed in Table 2-2.

Table 2-2: Abbreviations Used in this Document

Units	Abbreviation
Above mean sea level	ASL
Alaska Industrial Development Authority	AIDA
ALS Global	ALS
Aluminum	Al
Aluminum conductor steel reinforced	ACSR
Amperes	A
Antimony	Sb
Argillic	ARG
Arsenic	As
Associated Engineering	AE
Barium	Ba
Beryllium	Be
Bismuth	Bi
British Colombia	BC or B.C.
B-Train Double	BTD
Cadmium	Cd
Calcium	Ca
Canadian dollars	$, C$, or CAD$
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Carbon-in-column	CIC
Cariboo Rose Resources Ltd.	Cariboo Rose
Casino Mining Corporation	CMC
Central Nervous System (i.e., chemicals that affect it)	CNS
Chromium	Cr
Cobalt	Co
Combined cycle mode in gas turbines	CCGT
Copper	Cu
Copper equivalent	CuEq
CRS Copper Resources Corp.	CRS
Cubic metres	m^3
Cubic metres per hour	m^3/h
Current density	A/m^2
Dawson Range Batholith / Granodiorite	WR, WRGD
Degrees Celsius	ºC
Density	t/m^3
Direct Current Resistivity and Induced Polarization	DC/IP
Dollars per ounce	$/oz
Dollars per pound	$/lb
Dollars per tonne	$/t
Effective Grinding Length	EGL

Units	Abbreviation
Eighty percent passing	K80, P80
Electrowinning	EW
Engineering, Procurement and Construction Management	EPCM
Foot (feet)	ft
G&T Metallurgical Services	G&T
Gallium	Ga
Gas turbine	GT
General & Administrative	G&A
Gold	Au
Grams per litre	g/L or g/l
Grams per tonne	g/t
Greater than	>
Heap leach facility	HLF
Hectare(s)	ha
Horsepower	hp
Hour	h
Hour(s) per kilotonne	h/kt
Huebnerite	$MnWO_4$
Hypogene sulphide	HYP
Inch	"
Independent Mining Consultants	IMC
Induced Polarization	IP
Induced polarization	IP
Inductively Coupled Plasma-Atomic Absorption Spectroscopy	ICP-AAS
Inductively Coupled Plasma-Atomic Absorption Spectroscopy	ICP-AAS
Inductively Coupled Plasma-Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma-Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma-Emission Spectroscopy	ICP-ES
Internal Combustion Engine	ICE
Internal rate of return	IRR
Intrusive Breccia	IX
Inverse distance with a power weight of 2	ID2
Inverse distance with a power weight of 3	ID3
Iron	Fe
Kilo (1,000)	k
Kilogram(s)	kg
Kilogram(s) per tonne	kg/t
Kilometer	km

Units	Abbreviation
Kilopounds	klbs
Kilotonnes	ktonnes, kt
Kilotonnes per year	kt/y
Kilovolt(s)	kV
Kilowatt	kW
Kilowatt-hour	kWh
Kilowatt-hour per tonne	kWh/t
Knight Piésold Ltd.	KP
Lanthanum	La
Last-in-first-out	LIFO
Lead	Pb
Leak detection and recovery system	LDRS
Less than	<
Linear Low-Density Polyethylene	LLDPE
Liquefied natural gas	LNG
Litres	L, l
Litres per hour per square meter	$L/h/m^2$
Litres per second	L/s, l/s
Long term price	LTP
M3 Engineering & Technology Corporation	M3
Magnesium	Mg
Magnetotelluric Tensor Resistivity	MT
Manganese	Mn
Manganese	Mn
Mass Emission-Inductively Coupled Plasma Spectroscopy (ICP-MS)	ICP-MS
Material Takeoff	MTO
Mean annual precipitation	MAP
Mega (1,000,000)	M
Megawatt	MW
Mercury	Hg
Methyl Isobutyl Carbinol	MIBC
Metre(s)	m
Metric Tonne (1000 kg)	t, mt, or tonne
Metric tonne per day	t/d
Metric tonne per year	t/y
Micrometer or micron	µm
Milligrams per litre	mg/L
Millimeter(s)	mm
Million	M
Million Canadian dollars	C$M
Million cubic metres	Mm^3
Million dollars	$M
Million ounces	Moz
Million pounds	Mlbs
Million tonnes	Mt
Million tonnes per year	Mt/y
Million years ago	Ma
Molybdenite or Molybdenum	Mo or Moly
National Instrument 43-101	NI 43-101
Nearest neighbor	NN

Units	Abbreviation
Net present value	NPV
Net profits interest	NPI
Net smelter return royalty	NSR
Nickel	Ni
Non-Acid Generating	NAG
Non-Government Organizations	NGOs
Notice to Proceed	NTP
Ordinary kriging	OK
Ounce(s)	oz
Overburden	OVB
Oxide Dominant Leach Cap, or Leached Cap Mineralization	CAP or LC
Paleozoic schists and gneisses	YM
Parts per billion	ppb
Parts per million	ppm
Patton Porphyry	PP
Percent	%
Phosphorus	P
Post-mineralization explosive breccia	MX
Potassium	K
Potassium amyl xanthate	PAX
Potentially-Acid Generating	PAG
Pound	lb
Pounds	lbs
Power of hydrogen (measure of acidity)	pH
Preliminary Economic Assessment	PEA
Qualified Person	QP
Quality Assurance and Quality Control	QA/QC
Reclamation and closure plan	RCP
Reverse Circulation	RC
Rock Quality Designation	RQD
Run of Mine	ROM
Scandium	Sc
Semi-autogenous grinding	SAG
SGS Canada Inc.	SGS
Silver	Ag
SMC	SAG Mill Comminution
Snow water equivalent	SWE
Sodium	Na
Sodium Cyanide	NaCN
Sodium Hydrosulfide	NaSH
Specific gravity	S.G.
Square metres	m^2
Steam Turbine	ST
Strontium	Sr
Sulphidization, Acidification, Recycling and Thickening	SART
Supergene oxide	SOX
Supergene sulphide	SUS
Tailing Management Facility	TMF

Units	Abbreviation
Temperature Celsius	°C
Temperature Fahrenheit	°F
Thallium	Tl
Thousand troy ounces	koz
Titanium	Ti
Tonnage factor or specific volume	m³/tonne or m³/t
Tonnes per day	t/d
Tonnes per year	t/y
Tungsten	W
Uranium	U
US dollars	US$
Vanadium	V
Volt	V
Weak Acid Soluble	WAS
Western Copper and Gold Corporation	Western
Year	y, yr
Yukon Environmental and Socioeconomic Assessment Act	YESAA
Yukon Environmental and Socio-economic Assessment Board	YESAB
Yukon First Nations	UFA
Yukon Geological Survey	YGS
Yukon-Tanana terrane	YTT
Zinc	Zn

3 RELIANCE ON OTHER EXPERTS

In cases where the study authors have relied on contributions of other qualified persons, the conclusions and recommendations are exclusively the qualified persons' own. The results and opinions outlined in this report that are dependent on information provided by qualified persons outside the employ of M3 are assumed to be current, accurate and complete as of the date of this report.

Information received from other experts has been reviewed for factual errors by CMC and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

M3 relied upon Western Copper and Gold Corporation for project ownership data. M3 did not verify ownership or any underlying agreements. Mining is a risky business. The risk must be borne by the Owner. M3 does not assume any liability other than performing this technical study to normal professional standards.

The following sections describe additional information that this report relies upon beyond that which was provided by the QPs listed in Section 2.2.

3.1 METALLURGY AND PROCESS ENGINEERING

Outside reports that were relied upon included the following:

- ALS Metallurgy (formally G&T Metallurgical Services) of Kamloops, BC, performed numerous metallurgical testing to advance the flotation process design. Tom Shouldice was the official contact. International Metallurgical and Environmental and CMC managed and oversaw this work with input from FLSmidth.
- Starkey and Associates of Oakville, Ontario, performed a grinding circuit study. John Starkey is the official contact for Starkey and Associates. CMC managed and oversaw this work with input from FLSmidth.
- SGS Lakefield Research Limited of Lakefield, Ontario, performed a grinding circuit study. Carlos Lozano was the official contact for SGS Lakefield. CMC managed and oversaw SGS's work.
- FLSmidth, 2012. Casino Project Circuit Design Basis – Update on test work and Mill Sizing/Selection. Unpublished Company Report prepared for Western Copper and Gold Corporation. FLSmidth Salt Lake City, Inc. June 2012.
- SGS E&S Engineering Solutions (formerly METCON Research) of Tucson, AZ, USA performed metallurgical testing to advance design of the gold heap leach. Rodrigo Carneiro was the official contact. CMC managed and oversaw SGS's work.
- Carl Schulze relied on the Yukon Mining website for information necessary for the "Adjacent Properties" section (Section 23) of the Government of Yukon at: https://yukon.ca/en/mining, specifically at: https://yukon.ca/en/science-and-natural-resources/mining/find-maps-and-records-mining-claims-and-tenure.

M3 staff and consultants reviewed and evaluated metallurgical results from the tests listed above. In addition to supervising the effort, M3's Laurie Tahija also reviewed and approved design criteria, flow sheets and equipment lists for the metallurgical processes.

3.2 TRANSPORTATION

Associated Engineering (B.C.) Ltd. assisted by Lauga & Associates Consulting, Ltd. performed updates of the studies of transportation options including selection and design of the access road route. Associated Engineers and Lauga also prepared a report on port facility options.

The transportation costs for concentrates and bulk commodities used in the estimate are based on information from various sources that include: Seaspan Marine, Arrow Transport, Lynden and Braemar. Western Copper performed research for and obtained costs for some of the transportation costs. M3 also performed research for and obtained costs for some of the transportation costs and also evaluated the information received by Western Copper for the rest of the transportation costs.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/09, 10 and 15), in west central Yukon, in the north-westerly trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse. Figure 1-1 in Section 1 is a map showing the location of the Casino property in relation to the Yukon, British Columbia, and the Northwest Territories (Source: Yukon Highway Map, Yukoninfo.com). The property covers a total area of 13,124 ha.

The Yukon has a population of approximately 40,800 people. Whitehorse is the nearest commercial and population centre to the project property, with a population of approximately 30,000 people. Projected land access to Whitehorse would be 380 km via the Village of Carmacks. No human settlements can be described as "local." The Village of Carmacks is located about 150 km ESE, and the settlement of Pelly Crossing is about 115 km ENE. Beaver Creek, a village on the Alaskan Highway, is located about 112 km WSW. Fairbanks, Alaska is 500 km WNW.

The Arctic Circle is 430 km to the north. The Yukon River flows about 16 km north of the site. Yukon Highway 1, the Alaskan Highway, is about 110 km west at the nearest point. Yukon Highway 2, the Klondike Highway, is about 100 km to the east at the nearest point. No year-round roads reach the property.

The international border and Alaska are about 111 km to the west at the nearest point. British Columbia is south approximately 300 km. The closest port is Skagway, Alaska.

Exploration and mining projects in the area include the following:

- To the west, Newmont is developing the Coffee project. The project is currently at the pre-feasibility stage and is undergoing environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act (YESAA). They are also active with exploration on their project.
- To the north and to the west, White Gold Corp. has a large number of claims and is actively exploring them.
- Approximately 100 km to the east, Minto Explorations Ltd. operates the Minto Mine, which produces copper-silver-gold concentrate that is shipped through the port of Skagway.

The project is located on Crown land administered by the Yukon Government and is primarily within the Selkirk First Nation traditional territory. The Tr'ondek Hwechin traditional territory lies to the north and the proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south. The White River First Nation and Kluane First Nation are also potentially impacted by the project.

4.2 LAND POSITION AND STATUS

4.2.1 Property Description

The Dawson Range forms a series of well-rounded ridges and hills that reach a maximum elevation of 1,675 m above mean sea level (ASL). The ridges rise above the Yukon Plateau, a peneplain at approximately 1,200 m ASL, which is deeply incised by the mature drainage of the Yukon River watershed.

The characteristic terrain consists of rounded, rolling topography with moderate to deeply incised valleys. Major drainage channels extend below 1,000 m elevation. Most of the project lies between the 650 m elevation at Dip Creek and an elevation of 1,400 m at Patton Hill. The most notable local physical feature is the Yukon River which flows to the west about 16 km north of the project site.

The mean annual temperature for the Casino Project area is estimated to be -2.7°C, with minimum and maximum monthly temperatures of -18.1°C and 11.1°C occurring in January and July, respectively. The mean monthly temperature values are presented in Table 5-1 in Section 5. The Mean Annual Precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow.

Characteristic wildlife in the region includes caribou, grizzly and black bear, Dall sheep, moose, beaver, fox, wolf, hare, raven, rock and willow ptarmigan, and golden eagle.

The tops of hills and ridges are sparsely covered by tundra and buckbrush, with boreal forest covering valley floors and slopes below 1,200 m of elevation. Vegetation consists of black and white spruce forests with aspen and occasional lodgepole pine. Black spruce and paper birch prevail on permafrost slopes. Balsam poplar is common along floodplains. Scrub birch and willow "buckbrush" form extensive stands in subalpine sections from valley bottoms to well above the tree line.

4.2.2 Environmental

See Section 20 for a list of permits either obtained or in progress. No environmental liabilities are expected to impact the Project.

4.2.3 Mineral Tenure

The Casino Property lies within the Whitehorse Mining District and consists of a total of 1,136 full and partial Quartz Claims, and 55 Placer Claims acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 21,288 ha. The total area covered by Casino Placer Claims is 490.34 ha. The 825 quartz claims (of a total of 1,136 claims) comprise the initial Casino Property and 311 claims comprise the Canadian Creek Property acquired in 2019. The claims are registered in the name of, and are 100%-owned by, Casino Mining Corp. (CMC), a wholly owned subsidiary of Western Copper and Gold Corporation (Western). A list of claims is provided in Appendix B.

The historical claims held by prior owners of the project and transferred as part of 2006 Western Copper's plan of arrangement with Lumina Resources Corp. ("Lumina") consist of 83 Casino "A" claims covering an area of 1,154 ha, 23 claims in the "JOE" block covering an area of 323.63 ha and 55 Casino "B" claims covering an area of 929.93 ha, 9 claims of which were repurchased from Cariboo Rose Resources Ltd. ("Cariboo Rose") in November 2016 pursuant to an early exercise of 2002 Casino B option agreement. Forty-six of the Casino "B" claims were reacquired in July 2019 pursuant to the Canadian Creek Property Purchase Agreement, described in Section 4.2.4 in more detail. The Casino Deposit lies entirely on the Casino "A" claims.

CMC has significantly expanded the area of its mineral property by the staking and acquisition of mineral claims. The 188 VIK mineral claims, covering an area of 3,440 ha, were staked in June 2007 by CRS Copper Resources Corp ("CRS"), a predecessor of CMC. In June 2008, an additional 94 "CC" claims covering an area of 1,930 ha, 8 BL claims covering area of 157.24 ha, and 63 "BRIT" claims covering an area of 1,218 ha, were staked by CRS. In October 2009, CRS staked 136 AXS mineral claims, covering an area of 2,763 ha. In May of 2010, CRS staked an additional 63 AXS claims, covering an area of 1,254 ha. In 2011, CRS staked 18 FLY claims covering 327 ha. In May 2016, 87 PAL claims were staked by CMC, covering 1,818.18 ha. In July 2019, CMC acquired additional 311 mineral claims from Cariboo Rose that comprise the Canadian Creek Property and covering area of 6,001.47 ha. In September 2019, CMC staked 53 CAS19 claims covering an area of 759.88 ha.

4.2.4 Ownership and Agreements

CMC is a successor in title to the Casino Property pursuant to the Plan of Arrangement completed on October 17, 2011.

CRS, a predecessor of CMC, acquired the Casino A, B and JOE claims, comprising the historical Casino property, on August 9, 2007, by exercising its option pursuant to a Letter Agreement dated July 15, 2002 ("2002 Option") with Great Basin Gold Ltd. ("Great Basin"). The Casino deposit lies entirely on the Casino A claims.

On December 21, 2012, CMC entered into the Net Smelter Returns Royalty Agreement (the "NSR Royalty Agreement") with 8248567 Canada Ltd. ("8248567 Canada"), whereby the 2.75% Net Smelter Return Royalty ("NSR") was established on all Casino claims excluding fifty-five (55) Casino B Claims. As consideration for purchasing the 2.75% NSR, 8248567 Canada cancelled the existing 5% NPR (except on Casino B Claims).

On November 2, 2016, pursuant to the Early Exercise and Purchase Agreement (the "Early Exercise and Purchase Agreement"), Cariboo Rose exercised its right to acquire fifty-five (55) Casino B Claims, as described in the option agreement dated May 2, 2000 (the "Casino B Option Agreement") between Cariboo Rose and CMC (a successor to title by virtue of 2002 Option). As part of the Early Exercise and Purchase, CMC reacquired nine (9) Casino B Claims (the "Nine Casino B Claims"). Forty-six (46) Casino B Claims (the "Forty-Six Casino B Claims") were transferred to Cariboo Rose and became part of the Canadian Creek Property owned by Cariboo Rose.

On August 28, 2019, CMC and Cariboo Rose completed the Canadian Creek Property Purchase Agreement (the "Canadian Creek Property Purchase Agreement"), whereby Forty-Six Casino B Claims were reacquired as part of the Canadian Creek Property consisting of a total of 311 mineral claims.

4.2.5 Agreements and Royalties

Certain portions of the Casino property remain subject to the 2.75% NSR in favor of Osisko Gold Royalties Ltd. ("Osisko Gold") pursuant to the Royalty Assignment and Assumption Agreement dated July 31, 2017, when 8248567 Canada assigned to Osisko Gold all of its rights, title, and interest in the 2.75% NSR.

Certain royalty interests originated from certain historical agreements are deemed to have been dissolved pursuant to the insolvency proceedings and subsequent corporate dissolution of an original royalty holder given that there is no evidence that these royalty interests have ever been assigned, transferred, or sold to a third party.

4.2.6 Placer Claims

In the summer of 2010, Western staked a 5-mile Placer Lease along Casino Creek and a 3-mile Placer Lease along Britannia Creek. In 2011, these leases were converted to claims. In 2014, 30 placer claims on Britannia Creek were dropped and presently, Western, through CMC, owns 55 placer claims on Casino Creek.

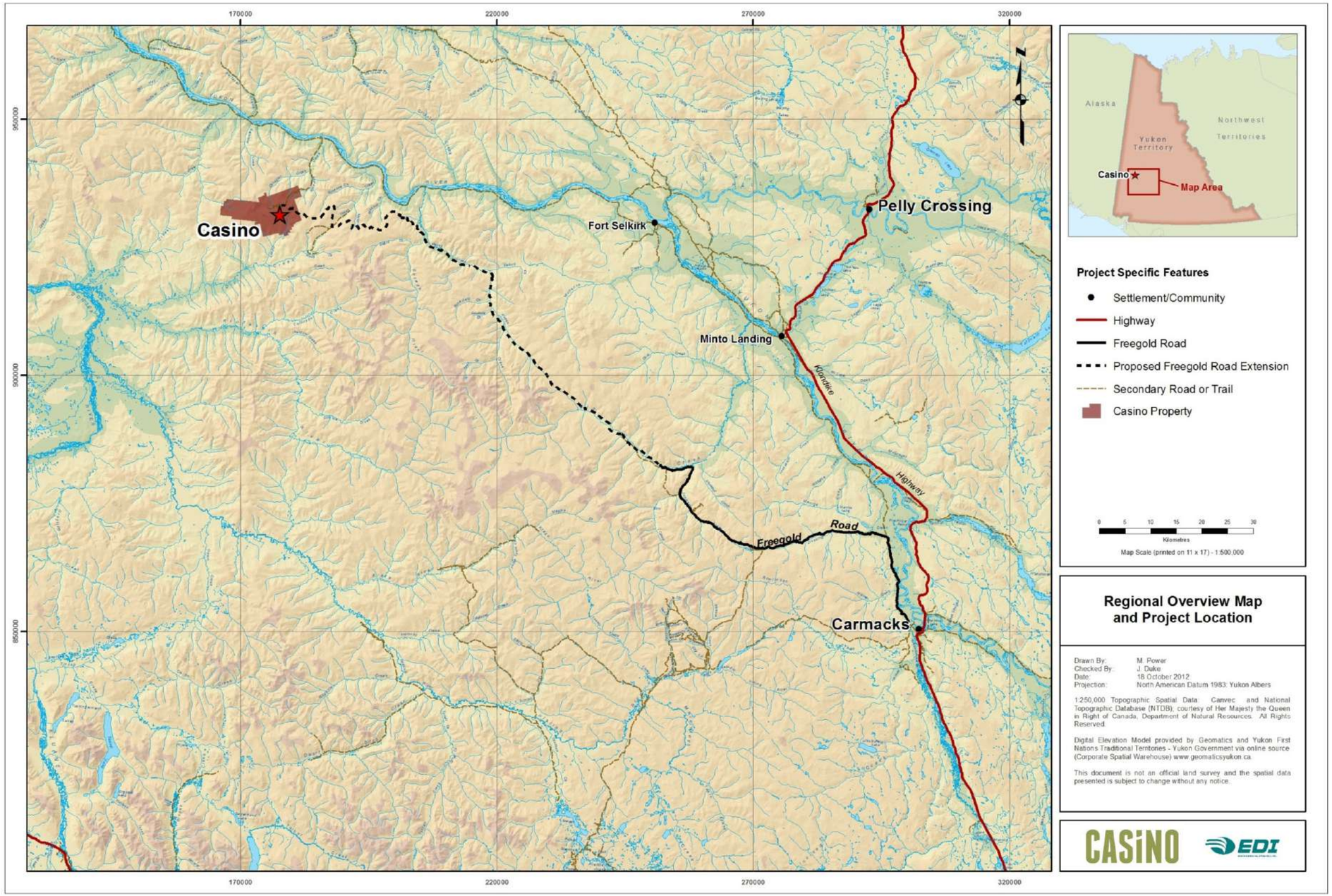


Figure 4-1: Project Road Access Map

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Casino Mine is located in Central Yukon, at approximately N62° 44' 25", W138° 49' 32" roughly 150 km due northwest of Carmacks. Current site access is by small aircraft using the existing 760 m airstrip, by winter road from the west, and from a seasonally accessible road extending from a barge landing at the Yukon River.

The barge landing area at Britannia Creek and the Yukon River was prepared in 2010 and the lower 10 km of the 23 km access road from the landing to the site was realigned.

5.2 PHYSIOGRAPHY

The Casino property is located in the Dawson Range, a north-westerly trending belt of well-rounded ridges and hills that reach a maximum elevation of about 1,675 m. The hills rise above the Yukon Plateau, at about 1,250 m and deeply incised by mature dendritic drainages. Although the Dawson Range escaped Pleistocene continental glaciation, minor alpine glaciation has produced a few small cirques and terminal moraines.

The deposit area is situated on a small divide. The northern part of the property drains to Canadian Creek and Britannia Creek into the Yukon River. The southern part of the property flows southward via Casino Creek to Dip Creek to the Donjek River and northward to the Yukon River.

Outcrop is rare on the property. Soil development is variable ranging from coarse talus and immature soil horizons at higher elevations to a more mature soil profile and thick organic accumulations on the valley floors.

5.3 CLIMATE

The climate in the Dawson Range is subarctic. Permafrost is widespread on north-facing slopes, and discontinuous on south-facing slopes. CMC installed an automated weather station at the site in 2009 and collected a certain amount of data.

The climate at the Casino Project area can generally be described as continental and cold. Winters are long, cold, and dry, with snow generally on the ground from late September through mid-May. Summers are short, mild, and wet, with the greatest monthly precipitation falling in July. The climate and hydrology at the Project site have been assessed based on both short-term site data and longer-term regional data. Site data are available from a program operated from 1993 to 1995 and from the current program that was initiated in 2008.

The mean annual temperature for the Casino Project area is estimated to be -2.7°C, with mean minimum and maximum monthly temperatures of -18.1°C and 11.1°C occurring in January and July, respectively. The mean annual precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow. The mean monthly temperatures and precipitation are presented in Table 5-1.

Table 5-1: Mean Monthly Temperature and Precipitation Values

	Parameter	
Month	Precipitation (mm)	Temperature (°C)
Jan	25	-18.1
Feb	19	-14.2
Mar	16	-8.2
Apr	15	-0.1
May	42	5.7
Jun	74	9.8
July	103	11.1
Aug	65	9.1
Sept	49	4.4
Oct	35	-3.3
Nov	31	-12.7
Dec	26	-16.5
Annual	500	-2.7

The estimated average annual lake evaporation is 308 mm, based on climate data collected at site and used in conjunction with long-term regional climate data.

Based on the estimated MAP of 500 mm and a rain/snow ratio of 0.65/0.35, the annual snowfall value for Casino was estimated to be 175 mm. This is generally consistent with the 140 mm mean annual maximum snowpack value (snow water equivalent, SWE) recorded in the Project area at the Casino Creek snow course station (09CD-SC01) operated by the Yukon Department of Environment (1977-2009), Water Resources Branch.

Based on the complete years of snowpack data, the average monthly snowmelt distribution for the Casino Project area was estimated to be 40% in April and 60% in May, although there is considerable variation from year to year.

5.4 WATER RIGHTS

It is assumed that water rights can be obtained for withdrawal of water from the Yukon River.

5.5 POWER AVAILABILITY

There is no utility power available to serve the site. The Project will need to generate its own power.

5.6 SURFACE RIGHTS

CMC has sufficient rights and available land at the Project site for a mine, tailing storage areas, waste disposal areas, heap leach pad areas and process plant areas.

6 HISTORY

The first documented work on the Casino Property was the working of placer claims in the area of the Casino Deposit in April 1911, following a placer gold discovery on Canadian Creek by J. Britton and C. Brown. A study by D.D. Cairnes, of the Geological Survey of Canada in 1917, recognized huebnerite ($MnWO_4$) in the heavy-mineral concentrates of the placer workings and also that the gold and tungsten mineralization was derived from an intrusive complex on Patton Hill. During the Second World War, a small amount of tungsten was recovered from placer workings. The total placer gold production from the area of the property is unknown, but during the period 1980-1985 placer mining yielded about 50 kg (1,615 troy ounces) of gold.

The first mineral claims at Casino were staked by N. Hansen in 1917; however, the first recorded bedrock mineral discovery occurred in 1936 when J. Meloy and A. Brown located silver-lead-zinc veins approximately 3 km south of the Canadian Creek placer workings. Over the next several years the Bomber and Helicopter vein systems were explored by hand trenches and pits. In 1943, the Helicopter claims were staked and in 1947 the Bomber and Airport groups were staked.

Lead-silver mineralization remained the focus of exploration on the property until 1968. Noranda Exploration Co Ltd. optioned the property in 1948 and Rio Tinto optioned it again in 1963. During this time trenching, mapping, and sampling were conducted.

L. Proctor purchased the claims in 1963 and formed Casino Silver Mines Limited to develop the silver-rich veins. The silver-bearing veins were explored and developed intermittently by underground and surface workings from 1965 to 1980. In total, 372.5 tonnes of hand-cobbled argentiferous galena, assaying 3,689 g/t silver (Ag), 17.1 g/t gold (Au), 48.3% lead (Pb), 5% zinc (Zn), 1.5% copper (Cu) and 0.02% bismuth (Bi), were shipped to the smelter at Trail, British Columbia.

In 1963, B. Hestor first recognized that the area had potential for a porphyry copper deposit, but his observations did not become generally known. In 1967, the porphyry potential was recognized again, this time by A. Archer and separately by G. Harper. Based on the recognition of porphyry copper potential, the Brynelsen Group acquired Casino Silver Mines Limited, and from 1968 to 1973 exploration was directed jointly by Brameda Resources (Brameda), Quintana Minerals (Quintana), and Teck Corporation towards a porphyry target. Exploration, including extensive soil sampling surveys, geophysical surveys, and trenching programs, eventually led to the discovery of the Casino deposit in 1969.

From 1969 to 1973, various parties, including Brameda Resources, Quintana Minerals and Teck Corporation, conducted drilling on the property. During this period 5,328 m of reverse circulation drilling in 35 holes, and 12,547 m of diamond drilling in 56 holes, were completed.

Archer, Cathro & Associates (1981) Ltd. (Archer Cathro) optioned the property in 1991 and assigned the option to Big Creek Resources Ltd. In 1992, a program consisting of 21 HQ (63.5 mm diameter) holes totaling 4,729 m systematically assessed the gold potential in the core area of the deposit for the first time.

In 1992, Pacific Sentinel Gold Corp. (PSG) acquired the property from Archer Cathro and commenced a major exploration program. The 1993 program included surface mapping and 50,316 m of HQ (63.5 mm diameter) and NQ (47.6 mm diameter) drilling in 127 holes. All but one of the twenty-one 1992 drill holes were deepened in 1993.

PSG drilled an additional 108 drill holes totaling 18,085 m in 1994. This program completed the delineation drilling commenced in 1993. PSG also performed metallurgical, geotechnical, and environmental work which was used in a scoping study in 1995. The scoping study envisioned a large-scale open pit mine and a conventional flotation concentrator that would produce a copper-gold concentrate for sale to Pacific Rim smelters.

First Trimark Resources and CRS Copper Resources obtained the property and, using the PSG data, published a Qualifying Report on the property in 2003 to bring the resource estimate into compliance with National Instrument 43-101 requirements. The two firms combined to form Lumina Copper Corporation in 2004. An update of the Qualifying Report was issued in 2004.

Western Copper Corporation acquired Lumina Copper Corporation, and therefore the Casino Deposit, in November 2006. In the fall of 2011, Western Copper Corporation spun out all other assets except the Casino Deposit and changed its name to Western Copper and Gold Corporation (Western).

In 2007, Western conducted an evaluation of the Bomber Vein System and the southern slope of Patton Hill by VLF-EM, Horizontal Loop EM, and soil geochemical surveying. Environmental baseline studies were also initiated in 2007.

In 2008, Western reclaimed the old camp site, constructed a new exploration camp (the present camp) next to the Casino airstrip and drilled three holes (the camp water well and two exploration diamond drill holes) totaling 1,163 m. The main purpose of the drilling was to obtain fresh core samples for the metallurgical and waste characterization tests. Both exploration holes twinned PSG's holes to confirm historical copper, gold, and molybdenum grades. Later that year, M3 Engineering & Technology Corporation produced a pre-feasibility study for Western Copper and Gold Corporation.

In 2009, Western completed 22.5 km of DC/IP surveying and MT surveying using the Titan system developed by Quantec Geosciences Ltd. As well, the company drilled 10,943 m in 37 diamond drill holes, of which 27 holes were infill holes drilled to upgrade the previously designated Inferred Resource and non-defined material to the Measured and Indicated resource categories. Infill drilling covered the north slope of Patton Hill that was mapped as a "Latite Plug" on PSG maps. The drilling also identified supergene Cu and Mo mineralization in this area. The remaining 10 holes, totaling 4,327 m, were drilled to test geophysical targets.

In 2010, Western, under the direction of the Casino Mining Corporation (CMC), a wholly owned subsidiary of Western, completed infill and delineation drilling mostly to the north and west of the deposit, as outlined by PSG. The drilling program also defined hypogene mineralization at the southern end of the deposit. In addition, the company drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit, and several other holes for hydrogeological studies. The geotechnical drilling continued in 2011 (41 holes, 3,163 m) and 2012 (6 holes, 228 m). This work culminated in the publishing of a pre-feasibility study in 2011 and a feasibility study in 2013.

In 2019, CMC carried out a program of infill drilling comprising 13,590 m in 72 holes. The program was designed to convert mineralization located along the margin of the deposit from the Inferred Resource category to the Indicated Resource category.

In 2020, CMC completed a diamond drilling program comprising 12,007.54 m in 49 holes, targeting three main areas: the Gold, Northern Porphyry and Casino West zones. Drilling at the Gold Zone was designed to test for higher grade mineralization along the south and west boundaries of the deposit. Drilling at the Casino West zone was designed to test for continuation of the deposit along the south flank of the Canadian Creek valley. Also, three holes targeted the Ana Zone, about 4 km west of the main deposit.

A breakdown of drilling by Western and CMC from 2010 to the end of 2019 is as follows:

- 173 exploration holes for 39,372.91 m.
- 11 combined hydrogeological and geological holes for 1,689.58 m.
- 53 geotechnical holes in the proposed tailings embankment, heap leach pad, plant site, waste rock storage site, airstrip, access road and water well areas, for 3,786.54 m.
- 5 holes for 1,570.63 m for the metallurgical sample.

The total meterage drilled by Western and CMC from 2008 to the end of 2020 is 58,646.91 m.

In 2021, CMC completed a diamond drill program, developed with input from Rio Tinto, comprising 6,074.97 m in 22 holes, including 16 within the Casino deposit resource boundaries. Within the resource area, drilling comprised 5 resource confirmation holes, 3 metallurgical testing holes, and 8 holes for geotechnical analysis. All holes returned copper-equivalent values that confirmed or, in some cases, exceeded values from previous drilling. Higher-grade values returned from the eastern margins of the deposit resource area may mark the trace of the Casino Fault.

An additional 6 exploration holes were drilled outside of the deposit area. Although the majority did not return significant mineralized intercepts, low-grade Au-Ag mineralization in one hole may mark another structural feature.

The 2021 program also included seven short geotechnical holes, comprising two at the proposed tailings management facility, four at the proposed heap leach site, and one at the proposed processing facility. The program also included on-site hyperspectral scanning of 48,673 m of core from 1992 -1994, 2008 - 2012, the 2021 core and some 2020 core.

In July 2021, Western and CMC completed a Preliminary Economic Assessment (PEA) report, incorporating data from drilling from 1992 through 2019. The PEA recommended advancement to a Feasibility Study to determine the mineral reserves for the deposit. Results from the 2020 program were excluded from the PEA but are included in this Feasibility Study.

6.1 HISTORY OF CANADIAN CREEK PROPERTY

In mid-2019, CMC acquired the adjacent Canadian Creek property from Cariboo Rose Resources Ltd (Cariboo Rose).

Exploration on the Canadian Creek property dates from 1992 when Archer Cathro & Associates staked the Ana Claims. In 1993, Eastfield Resources Ltd. (Eastfield) acquired the Ana Claims, expanded the Ana Claim block, and explored the expanded property with soil grids, trenching and drilling (Johnston, 2018). This work was directed at the discovery of additional porphyry deposits. The 1993 program was followed by extensive field programs in 1996, 1997, and 1999 consisting of induced polarization (IP) surveying, road construction and trenching on the Ana, Koffee, Maya and Ice claims. In 2000, another drill campaign was undertaken by Eastfield on the Ana, Koffee Bowl, and the newly acquired Casino "B" claims located immediately to the west of the Casino deposit. The Casino "B" holes confirmed the existence of gold mineralization first discovered here in 1994 by PSG, which encountered 55.17 m averaging 0.71 g/t gold in hole 94-319. Modest exploration programs were conducted in 2003, 2004 and 2005, mostly over the Casino "B" area. In 2007, a five-hole core drilling program at Casino "B" targeted gold and copper-in-soil anomalies and ground magnetic high features.

The discovery in 2009 of gold mineralization on Underworld Resources' White Gold property sparked new interest in gold exploration in the Yukon. This led to the implementation of a major exploration program at Canadian Creek. This was directed at the gold potential of the property, including areas some distance from previous work, and focusing on porphyry copper mineralization.

A soil survey revealed extensive areas returning greater than 15 ppb gold in soils with associated anomalous values in arsenic (As), bismuth (Bi) and antimony (Sb). The anomalous area extends for over 4 km in an east-northeast direction. The induced polarization (IP) surveys revealed numerous strong chargeability highs, many of which coincide with the gold-in-soil anomalies.

Ten diamond drill holes were completed within the new grid. Results include numerous intervals of anomalous gold values, commonly associated with elevated As, Sb and Bi. The mineralization is hosted in both granodiorite and gneiss country rock, commonly in clay-altered structures, sheeted pyrite veins and/or quartz-carbonate veins. With few exceptions, gold grades are less than 1 gram per tonne (g/t) and widths are less than 3 m.

Resampling of old trenches in other parts of the property was undertaken to verify significant historical gold results. In trench Tr-2, excavated in 1993 and located in the Ana Pass area, a grab sample of a tourmaline-pyrite-quartz altered intrusive rock returned 2,516 ppb gold. In the Casino "B" area, trench 9076-C averaged 376 ppb gold over 50 m, including a 10 m interval of 927 ppb.

In 2011, additional soil sampling, ground geophysical surveying and trenching were completed. The soil sampling completed the coverage of the entire Canadian Creek property and increased the known extent of the arsenic anomalies. A limited-extent induced polarization survey identified two zones of chargeability with values greater than 20 mv/V. The trenching program identified several areas with anomalous gold values, ranging from sub-detection level up to 2,890 and 4,400 ppb Au.

As a follow up on the 2011 program, a modest 2016 program of trenching, prospecting and in-fill soil sampling was carried out by Cariboo Rose, which had acquired the property from Eastfield. Trenching conducted in three areas of the Ana portion of the Canadian Creek property returned locally anomalous Au, and widely spread anomalous As, Bi, Sb and locally high Ag values, generally confined to narrow structures.

Cariboo Rose's 2017 exploration program consisted of surface work directed at the Kana and Malt West gold targets and a reverse circulation (RC) drill program that tested a variety of gold targets across the property. A total of 2,151.27 m of reverse circulation (RC) drilling was conducted in 24 holes. This work confirmed that zones of gold and silver mineralization to be limited to narrow (less than 3 m wide) structures rarely traceable over more than 100 m.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Casino deposit occurs within the Yukon-Tanana terrane (YTT), a northwest-southeast trending accreted terrane comprising Neoproterozoic to Upper Cretaceous metaigneous and metasedimentary rocks abutting the southwest side of the Tintina Fault Zone northeast of the property. This was previously described as an overlapping zone of the Yukon Cataclastic Terrane to the north and the Yukon Crystalline Terrane to the south (Templeman-Kluit, 1976). An elongate band of ultramafic rocks, 1 km north of the Casino deposit, may occur along a major tectonic suture. The YTT in this area has undergone emplacement of the 104 Ma Dawson Range Batholith, part of the Whitehorse Intrusive Suite. The Dawson Range Batholith extends WNW for about 300 km, roughly parallel to the regional orientation of strata comprising the YTT, also known as the Yukon Metamorphic Complex.

The YTT is dominated by Paleozoic rocks with scattered intrusions of the Coffee Creek Suite that are petrographically distinct from the Dawson Range Batholith. The YTT in the Dawson Range area is comprised of metasedimentary rocks of the Proterozoic to Devonian Snowcap assemblage, rocks of the Devono-Mississippian Wolverine Creek Metamorphic Suite, (Johnston, 1995) and rocks of the Permian Sulphur Creek assemblage (website, Yukon Geological Survey, 2020). Snowcap assemblage rocks comprise quartzites, pelites, psammites and marble (YGS, 2020). Stratigraphy of the Wolverine Creek Suite comprises sedimentary and igneous protoliths (Tempelman-Kluit, 1974; Payne et al., 1987). These metasedimentary rocks consist mainly of quartz-feldspar-mica schist and gneiss, quartzite, and micaceous quartzite, while the meta-igneous unit includes biotite-hornblende-feldspar gneiss and other orthogneisses, as well as hornblende amphibolite (Selby & Nesbit, 1997).

During the mid-Cretaceous period, Wolverine Creek suite rocks in this area were intruded by the Dawson Range Batholith, subsequently intruded by the Casino Intrusive Suite (Selby et al., 1999). The Dawson Range Batholith has incorporated scattered roof–pendants and blocks of the YTT, particularly Snowcap Assemblage and Wolverine Creek Suite rocks. The Dawson Range Batholith is the main country rock of the Casino Property and is represented by a relatively homogeneous, medium- to coarse-grained, hornblende-bearing, potassic quartz diorite to granodiorite, and lesser fine- to medium-grained diorite and quartz monzonitic veins, dykes, and plugs (Tempelman-Kluit, 1974).

The Casino Intrusions, also called the Casino Plutonic Suite, have been described as a suite of quartz monzonite stocks up to 18 km across (Hart and Selby, 1998) trending west-northwest parallel to the Big Creek Lineament and its northwestern extension. Mapping by Tempelman-Kluit (1974), and successively by Payne et al. (1987), associates this Casino Plutonic Suite with the mid-Cretaceous Dawson Range Batholith. Subsequently, Johnston (1995) grouped the intrusions with the late-Cretaceous Prospector Mountain Plutonic Suite, based largely on field relationships that show stocks of the Casino Plutonic Suite cutting the Dawson Range Batholith. Subsequent age determination by Mortensen and Hart (1998), as well as geochemistry provided by Selby et al. (1999), re-evaluated the Casino Intrusions as mid-Cretaceous fractionated magmas of the Dawson Range Batholith. Recent field relationships have proven that the ‘quartz monzonites’ of the Casino property, once thought to be separate intrusions, are intensely altered and recrystallized diorites of the Dawson Range Batholith.

During late Cretaceous time, stocks and apophyses of the Prospector Mountain Plutonic Suite were emplaced into the Dawson Range Batholith (Johnston, 1995; Selby et al, 1999). In the Casino area, this suite is represented by the 72.4 Ma Patton Porphyry intrusions, occurring as small, biotite-bearing, feldspar-porphyritic, hypabyssal rhyodacite to dacite intrusions near the centre of the deposit, and as discontinuous centimeter- to metre-wide dikes northwest of the property. Here, early phases of the Patton Porphyry have undergone multiple phases of brecciation, resulting in a mineralized intrusive breccia. Later, unaltered dykes of similar lithology cut surrounding hydrothermally altered and mineralized rocks (Payne et al., 1987) suggesting there are multiple phases of this unit (Bower, 1995; Selby and Creaser, 2001). The interpreted multi-phased emplacement of the Patton Porphyry was supported by detailed re-

logging of select core by Rio Tinto personnel. Hydrothermal alteration and mineralization occur in, and adjacent to, some of these late Cretaceous intrusions.

The regional geology is illustrated in Figure 7-1, with the accompanying legend shown in Figure 7-2. Figure 7-3 summarizes the local geologic setting of the Casino property area. Table 7-1 summarizes the stratigraphy and isotopic ages of the area. All isotopic dates are based on U-Pb ratios in zircons analyzed by J.R. Mortensen.

Table 7-1: Stratigraphic Column

	Geological Unit	Isotopic Age
Late Cretaceous	PROSPECTOR MOUNTAIN PLUTONIC SUITE:	
	Intrusive Breccia (Diatreme) *Heterolithic; fine-grained matrix; angular clastic*	
	Heterolithic Intrusion Breccia *Heterolithic; Patton porphyry/potassic matrix; autobrecciated fragments*	
	Patton Porphyry: Rhyodacitic to dacitic intrusion *Plagioclase-Biotite Porphyry; K-feldspar +/- Qz megacrystic porphyry*	72.4 +/-0.5 Ma
mid-Cretaceous	DAWSON RANGE BATHOLITH:	
	Granodiorite *biotite-hornblende granodiorite*	104.0 +/-0.5 Ma
	Diorite *Hornblende-Biorite-Quartz Diorite; hornblende-biotite diorite*	104.0 +/-0.5 Ma
Devono-Mississippian	WOLVERINE CREEK METAMORPHIC SUITE:	
	Meta-sedimentary *Micaceous Quartzite*	
	Meta-igneous *Qtz-Bi-Plagioclase-Microcline Gneiss; K-Feldspar-Quartz-Biotite Gneiss; Amphibolite*	
Proterooic-Devonian	SNOWCAP ASSEMBLAGE	
	Metasedimentary: Quartzite, psammites, pelites, marble	

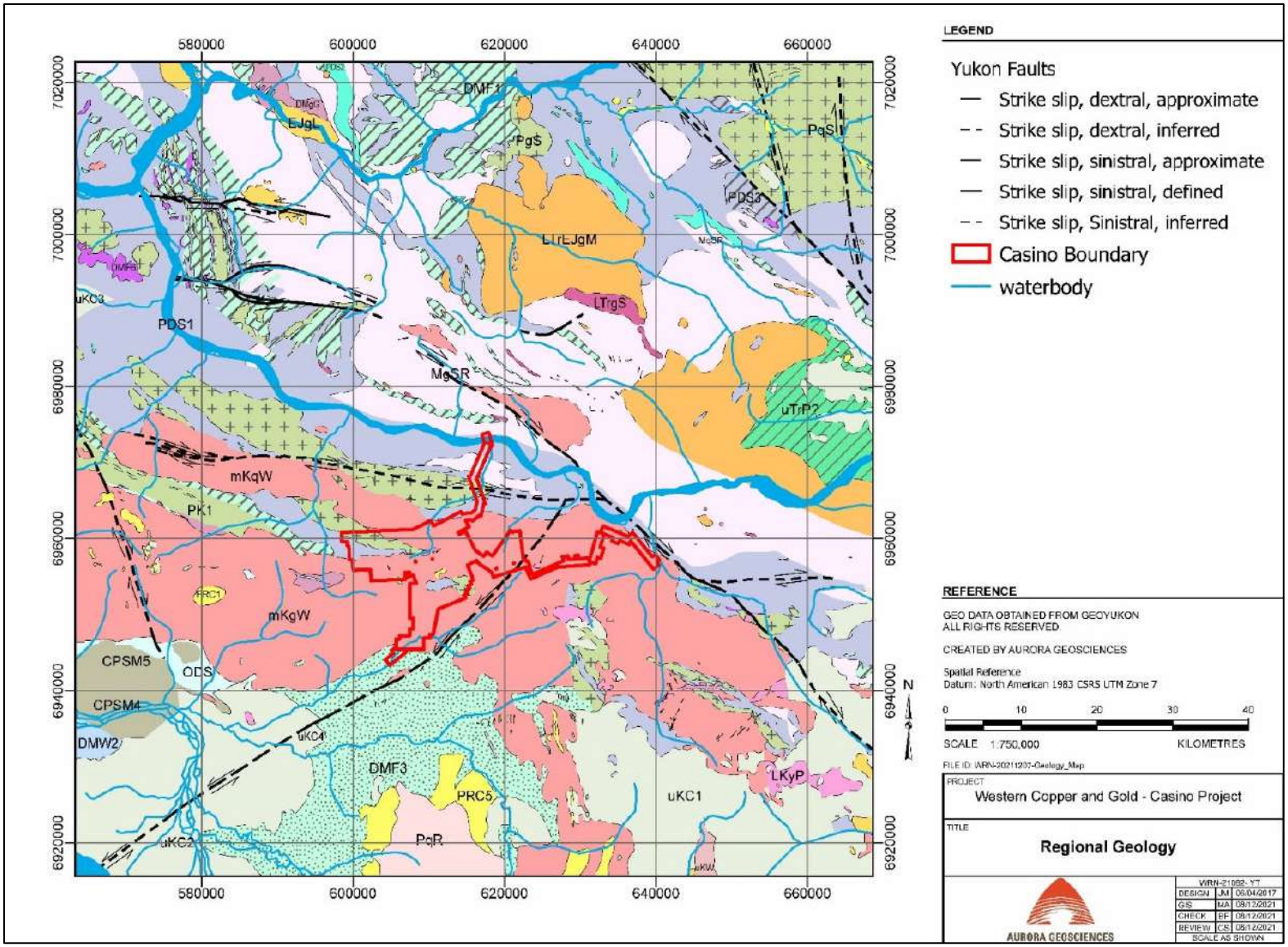


Figure 7-1: Regional Geology, Casino project area

Figure 7-2: Legend, Regional Geology, Casino Property area

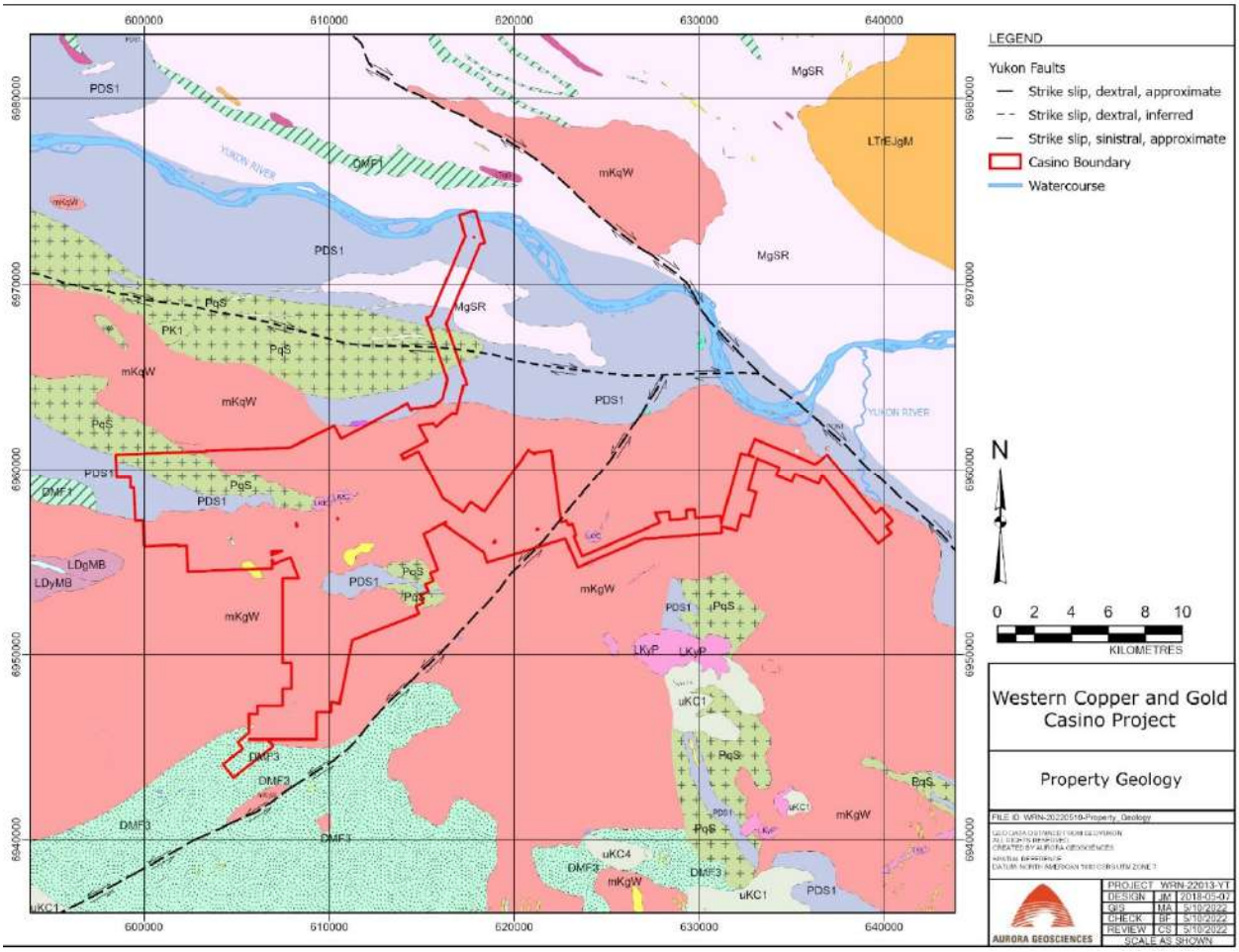


Figure 7-3: Local Geology, Casino Property area

The Casino Property is sandwiched between parallel west-northwest-trending faults that form contacts between rocks of the Wolverine Creek Metamorphic Suite and the Dawson Range Batholith. In Figure 7-4, the fault farthest to the northeast is an extension of the Big Creek Fault, interpreted as having undergone dextral offsetting of 20 to 45 km. A parallel fault, 8 km to the southwest, forms the southwest boundary of a sliver of Wolverine Creek Metamorphic Suite rocks and contains outcroppings of ultramafic rocks similar to those occurring along the Big Creek Fault.

The Casino Property is bounded to the southeast by a northeast-trending regional structure known as the Dip Creek Fault, which has a left lateral (sinistral) displacement. The left-lateral displacement of stratigraphy along the Yukon River east of the Casino Property reflects sinistral movement along this fault. The east-trending Minto-Battle Fault is also sinistrally offset by the Dip Creek Fault (Johnston, 1999). The dextrally offset Minto-Battle fault lies east of the Casino Property on the opposite side of Dip Creek, with its offset extension lying south and southwest of the Casino Property.

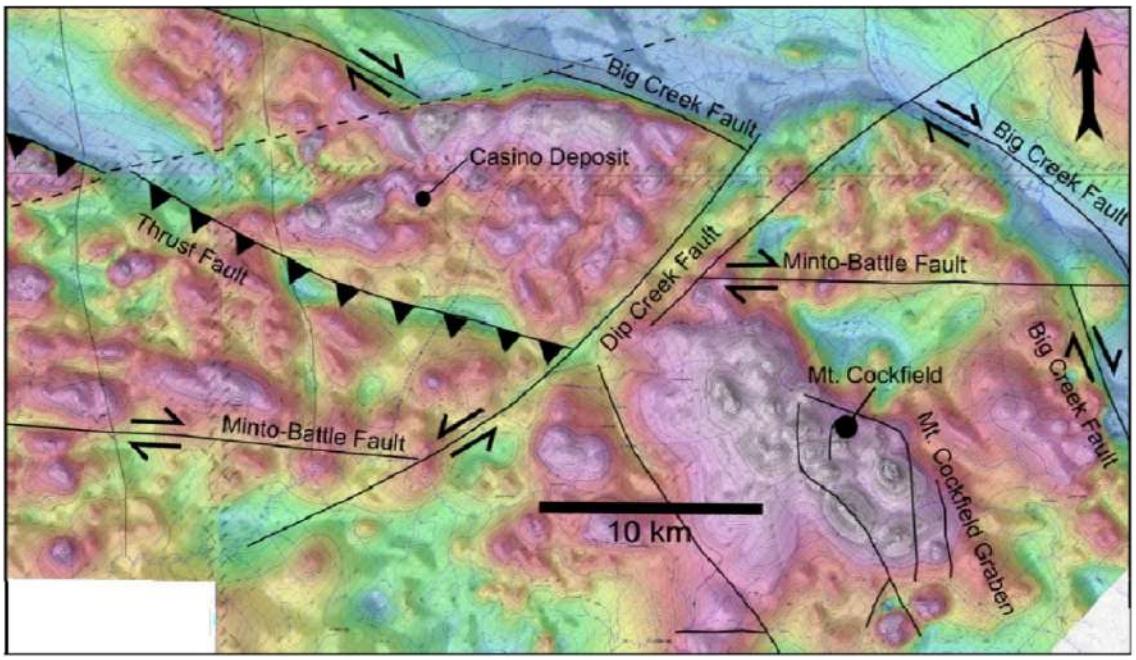


Figure 7-4: Regional Structures Overlain on Recent Aeromagnetic Survey

7.2 PROPERTY GEOLOGY

The geological setting of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous (72.4 Ma), east-west trending elongate dacite porphyry stock, called the "Patton Porphyry" (PP), a member of the Casino Suite which intrudes mid-Cretaceous granitoids of the Dawson Range Batholith (WRGD) and Paleozoic schists and gneisses (YM) of the YTT (Figure 7-6). Emplacement of the Patton Porphyry dacite stock into the older rocks resulted in brecciation of both the intrusive rocks and the surrounding country rocks along the northern, southern, and eastern contacts of the stock. Brecciation is best developed in the eastern boundary of the stock where the breccia zone can be up to 400 m wide in plan view. To the west, along the north and south contacts, the breccias narrow gradually to less than 100 m. Drilling along the western end of the dacite stock has revealed a late, post-mineralization explosive breccia (MX) that has obliterated the Patton Porphyry stock and any related contact breccia in this area. The late explosive breccia, likely a diatreme forms an elliptical body over 300 m in width. It also occurs as narrow east – west trending dykes extending into the dacite stock and surrounding granitoids and metamorphic rocks. The Patton Porphyry, intrusive breccias and late explosive breccias comprise the Casino Intrusive Complex, measuring 1.8 km by 1.0 km.

Patton Porphyry dykes extend west of the deposit for several kilometres (Figure 7-6). Locally, these dykes are associated with breccia zones developed along their margins, and are locally mineralized with pyrite, chalcopyrite and molybdenite as disseminations, veins, and fracture fillings.

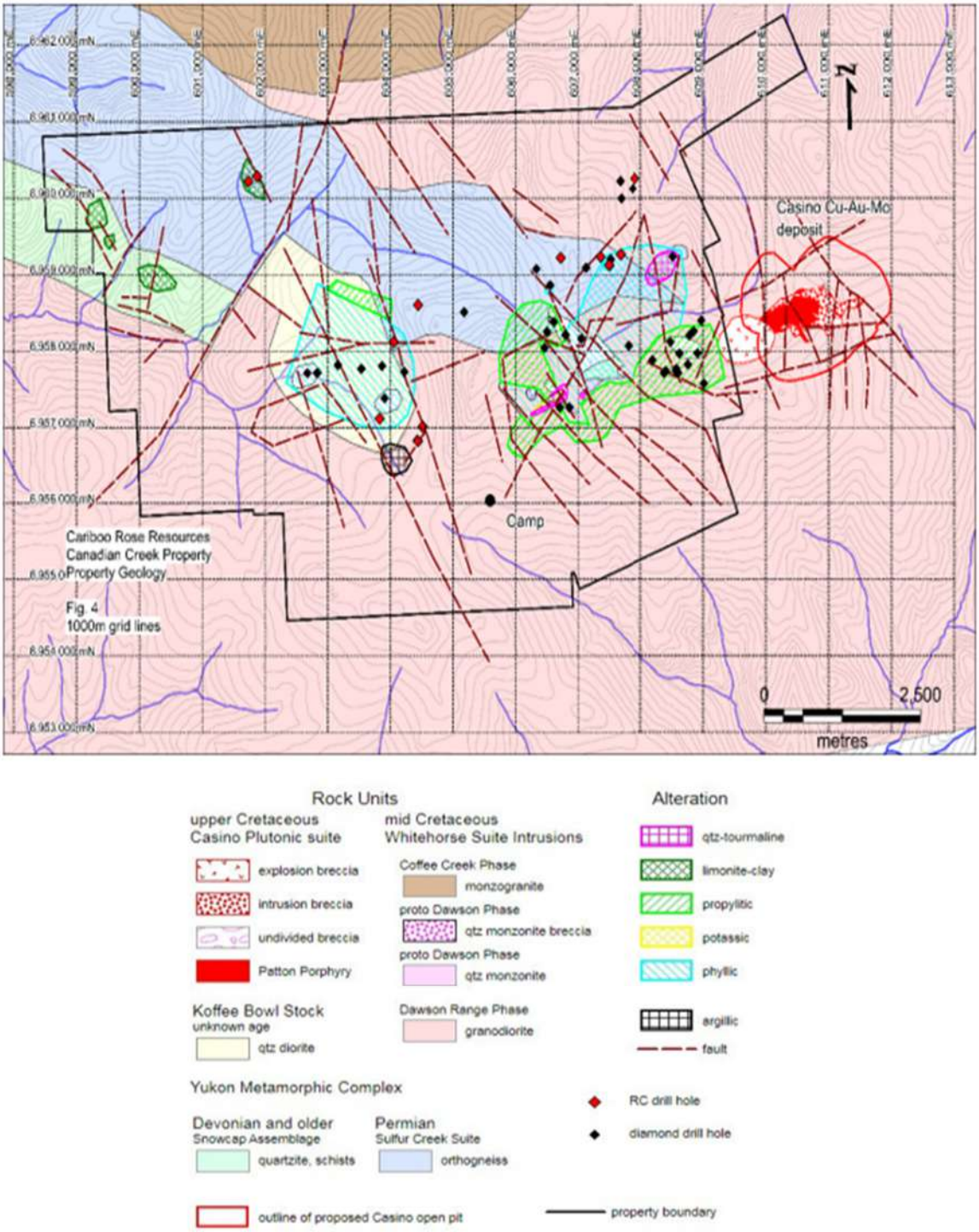


Figure 7-5: Property Geology (R. Johnson, 2018)

On the northwest side of the Casino intrusive complex, a swarm of Patton Porphyry dykes and related breccias have been identified. As of 2013, the dyke swarm was speculated to represent the upper emanation of a buried satellite stock of the main Patton Porphyry stock. This hypothesis has not been substantiated to 2021. Also, the "Northern Porphyry Zone" tested in 2020 did not establish the presence of a buried stock.

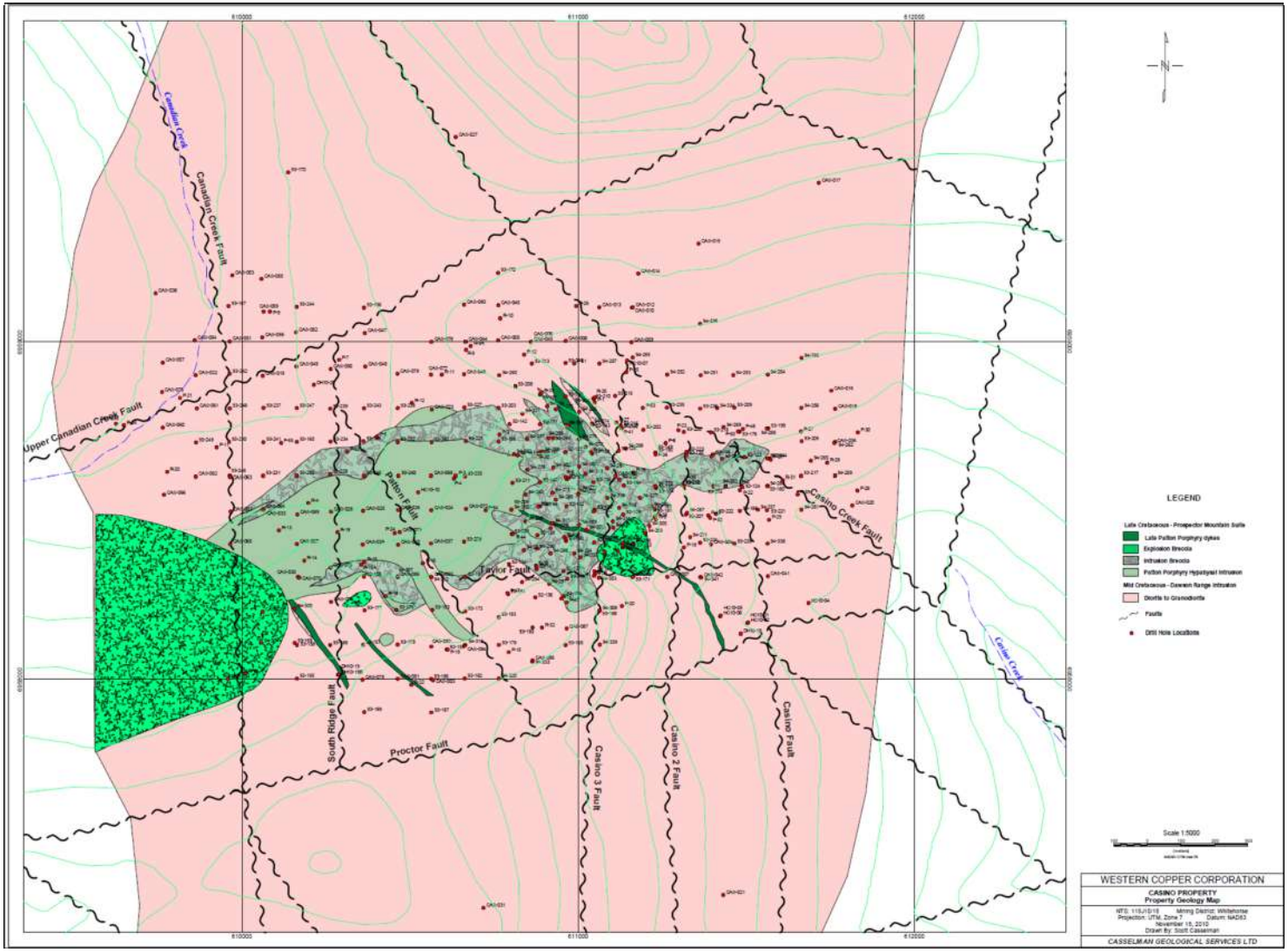


Figure 7-6: Geology of the Casino Deposit

7.3 MINERALIZATION

7.3.1 Hydrothermal Porphyry Alteration

Crystallization and exsolution of hydrothermal fluids from Patton Porphyry (PP) magmas produced porphyry style Cu-Mo-Au mineralization. Therefore, the Patton Porphyry, and associated Intrusive Breccia (IX), is genetically related to the Cu-Mo-Au mineralization of the deposit.

Hydrothermal alteration at the Casino property consists of a potassic core centered on and around the main Patton Porphyry body, in turn bordered by a contemporaneous, strongly developed and fracture controlled phyllic zone, a weakly developed propylitic zone, and a secondary discontinuous argillic overprint. Mineralized stockwork veins and breccias within the Casino Property are closely associated with the hydrothermal alteration. Potassic alteration minerals include texturally destructive K-feldspar, biotite, magnetite, and quartz, with lesser hematite, purple anhydrite, and gypsum. Biotite is mainly felted and pseudomorphic after hornblende. Locally, magnetite forms braided veinlets. In drill core, potassic alteration is represented by dark brown to black biotite alteration and/or pink potassium feldspar (K-spar) alteration.

The texturally destructive phyllic zone is found peripheral to, and locally overprinting, the potassic alteration zone. It has a distinctive 'bleached' appearance and is locally structurally controlled. Phyllic alteration minerals include quartz, pyrite, sericite, muscovite (after biotite), and abundant tourmaline, as well as minor hematite and/or magnetite towards the potassic zone. Quartz and sericite are typically alteration minerals after potassic and plagioclase feldspars. Biotite alters to muscovite or titanite, and hornblende alters to chlorite, calcite, quartz, and biotite. Tourmaline forms radiating disseminations and veinlets. Sulphide content is typically high, with pyrite ranging from 5-10% throughout, as disseminated blebs or cores to quartz-bearing "D" veins.

Where intense phyllic alteration overprints potassic alteration, relict textures are destroyed and minerals are recrystallized, commonly to equal portions of quartz, plagioclase, and K-feldspar, and including up to 10 percent biotite, trace apatite and titanite. Strongly zoned plagioclase and locally kinked biotite form subhedral laths, surrounded by K-feldspar, locally strained quartz, and biotite. The overall colour is pale pink.

Propylitic alteration is rare on surface but forms a wide halo around the deposit in gradational contact with the inner potassic alteration. Alteration minerals include epidote, chlorite, and calcite, with lesser carbonate, clay, sericite, pyrite, and albite. Hornblende and biotite are completely chloritized, whereas feldspars are relatively fresh, and textures are generally well-preserved.

Secondary argillic alteration is closely associated with the supergene zone and may appear locally as patches or pockets within the potassic and phyllic alteration zones. It is poorly developed, appears bleached or pale green, contains abundant clays (kaolinite, montmorillonite) and local chlorite and/or carbonate. In drill core, this unit may be recognized by distinctive "pock-marks" along the surface of the core.

The Casino deposit is not typical of Andean/Arizonan porphyry copper deposits in which mineralization is characterized by a dense stockwork of veinlets. Within the Casino deposit, mineralization occurs both as fine disseminations *sensu stricto* and in veinlets, with the highest concentrations occurring as disseminations in the matrix of the Intrusive Breccia (IX) zones. Notably, there is an absence of early, high-temperature "A-veins." In a typical Andean/Arizonan porphyry, the onset of mineralization propagates through a magma "mush" resulting in multiple veinlet-forming events as the magma cools. The final pulses permeate a cooled, fractured intrusive body, resulting in a dense stockwork of multiple superimposed veinlet formational events, each with a distinctive mineralogy and formation. However, at the Casino deposit, the Patton Porphyry had apparently already cooled prior to the main mineralizing event. The brecciation events allowed for subsequent fluid movement into fractures within both the Patton Porphyry and Dawson Range intrusive rocks, but the highest grades generally occur in the IX breccia (Williams, 2021); the overall vein density is relatively

low. Re-interpretation by Rio Tinto concluded that multiple events of brecciation occurred throughout the emplacement history of various pulses of Patton Porphyry, resulting in mineralizing events both within the "Early Mineral Units" and "Intermediate Mineral Units."

7.3.2 Metallurgical Zoning

The Casino deposit is unusual among Canadian porphyry deposits as it has a substantially preserved outcropping gold-bearing oxidized "Leached Cap," an upper well-developed copper-gold enriched "Supergene Zone" and a lower copper-gold bearing "Hypogene Zone". The Supergene Zone is comprised of the Supergene Oxide (SOX) zone and the more extensive Supergene Sulphide (SUS) zone. Table 7-2 summarizes the main minerals identified in the Leached Cap and Supergene zones.

7.3.2.1 Leached Cap Mineralization (CAP)

The Leached Cap is copper-depleted due to leaching processes and has a lower specific gravity relative to the supergene zones. It averages 70 m in thickness and is characterized by boxwork textures filled with jarosite, limonite, goethite, and hematite. This weathering has destroyed rock textures and has replaced most primary minerals with clay. The resulting rock is pale gray to cream in colour and is friable to the touch, and the clay is commonly stained yellow, orange, and/or brown by iron oxides. The weathering is most intense at the surface and decreases with depth. Whereas the copper minerals were leached and copper in solution mobilized downward until deposited along a redox zone to form the supergene enriched blanket, gold was significantly less mobile, resulting in significant gold grades remaining in the Leached Cap.

7.3.2.2 Supergene Sulphide Mineralization (SUS)

Supergene copper mineralization occurs in a weathered zone up to 200 m deep, located below the Leached Cap and above the Hypogene zone. It has an average thickness of 60 m and is positively correlated with high-grade hypogene mineralization, high permeability, and phyllic and/or outer potassic alteration. Grades of the Supergene sulphide zone vary widely but are highest in fractured and strongly pyritic zones, given the elevated permeabilities and high leaching capacity of these zones. Thus, secondary enrichment zones are thickest along contacts of the potassic and phyllic alteration halos. Grain borders and fractures in chalcopyrite, bornite and tetrahedrite may be altered to chalcocite, digenite, and/or covellite. Chalcocite also locally coats pyrite grains and clusters, and locally extends along fractures deep into the hypogene zone. Molybdenite is largely unaffected by supergene processes, other than local alteration to ferrimolybdite.

In drill-core, the SUS zone is commonly broken, with decreasing clay alteration and weathering with depth and is 'stained' dark blue to gray.

7.3.2.3 Supergene Oxide Mineralization (SOX)

The poorly defined Supergene Oxide zone (SOX) is copper-enriched and hosts trace molybdenite. It occurs as a few perched bodies within the Leached Cap, likely due to more recent fluctuations in the water table. This zone is thought to be related to present-day topography and is best developed where oxidation of earlier secondary copper sulphides occurs above the water table, typically on well drained slopes. Where present, the Supergene Oxide zone averages 10 m in thickness, locally contains chalcanthite, malachite and brocanthite, with minor azurite, tenorite, cuprite and neotocite. Its contact is gradational into the more well-defined Supergene Sulphide zone.

Table 7-2: Leached Cap & Supergene Minerals

Zone	Minerals Present	Average Thickness
Leached Cap	jarosite, goethite, hematite, ferrimolybdite	70 m
Supergene Oxide	chalcanthite, brochantite, malachite, azurite, tenorite, cuprite, neotocite, copper WAD native copper, copper-bearing goethite	10 m
Supergene Sulphide	digenite, chalcocite, minor covellite, bornite, copper-bearing goethite	60 m

7.3.3 Hypogene Mineralization (HYP)

Mineralization of the Casino Cu-Au-Mo deposit occurs mainly in the steeply plunging, in-situ contact breccias surrounding the Patton Porphyry intrusive plug. It was formed by crystallization and exsolution of hydrothermal fluids from late Cretaceous magmas of the Casino Plutonic Suite. The breccia forms an ovoid band around the main porphyry body with dimensions up to 250 m and has an interior zone of potassic alteration surrounded by discontinuous phyllic alteration, typical of some porphyry deposits.

Hypogene mineralization occurs throughout the various alteration zones of the Casino Porphyry deposit, hosted by low-density veinlets, and disseminated in the matrix of breccias. Field relationships show that the onset of potassic alteration is represented by mineralized quartz veins within the phyllically-altered zones, which cut those of the potassically-altered zones. Rhenium-osmium (Re-Os) age dating showed that the timing of the potassic and phyllic alteration are contemporaneous at around 74.4 ± 0.28 Ma. Significant Cu-Mo mineralization is related to the potassically-altered breccias surrounding the core Patton Porphyry, as well as in the adjacent phyllically-altered host rocks of the Dawson Range Batholith.

Mineralization in the potassic zone comprises mainly finely disseminated pyrite, chalcopyrite and molybdenite, as well as trace sphalerite and bornite. The phyllic alteration zones have increased gold, copper, molybdenite and tungsten values concentrated within disseminations and veinlets of pyrite, chalcopyrite and/or molybdenite along the inner part of the pyrite halo. The pyrite halo occurs within the phyllic alteration zone along the potassic-phyllic contact and discontinuously surrounds the main breccia body. It is host to the highest copper values on the property.

Chalcopyrite commonly occurs in veinlets, as disseminations, and as irregular patches. Within breccia zones and granodiorite west of the Casino Fault, disseminated chalcopyrite is more abundant than vein and veinlet-style chalcopyrite, whereas to the east of the fault, chalcopyrite is controlled by brittle deformation and occurs in fractures and open space fillings. Pyrite to chalcopyrite ratios range from less than 2:1 in the core of the deposit, to greater than 20:1 in the outer phyllic zones. Locally, coarse-grained bornite and tetrahedrite are intergrown with chalcopyrite.

Molybdenite is not generally intergrown with other sulphides and occurs as selvages in early, high-temperature, quartz veinlets with potassic-alteration halos and as discrete flakes and disseminations.

Native gold occurs as free grains (50 to 70 microns) in quartz and as inclusions in pyrite and/or chalcopyrite grains (1 to 15 microns). High-grade smoky quartz veins with numerous specks of visible gold have been reported but are rare, and do not contribute significantly to the gold inventory of the deposit.

Overall, the copper, gold, and molybdenum grades are not sympathetic to each other suggesting that, at least in part, each metal is distinguished by its own hydrothermal event.

Late-stage, commonly vuggy, polymetallic veins, similar to the Bomber Vein, extend along roughly parallel, steeply dipping fractures trending 150° to 170°. Metallic mineralogy includes abundant sphalerite and galena, with less abundant tetrahedrite, chalcopyrite (commonly intergrown with tetrahedrite), and bismuth-bearing minerals, and are geochemically anomalous in any or all of Ag, As, Bi, Cu, Cd, Mn, Pb, Sb, Zn and locally W.

In drill-core, the hypogene zone is unweathered and unoxidized. Figure 7-7 is a cross section of the deposit showing the main metallurgical zones.

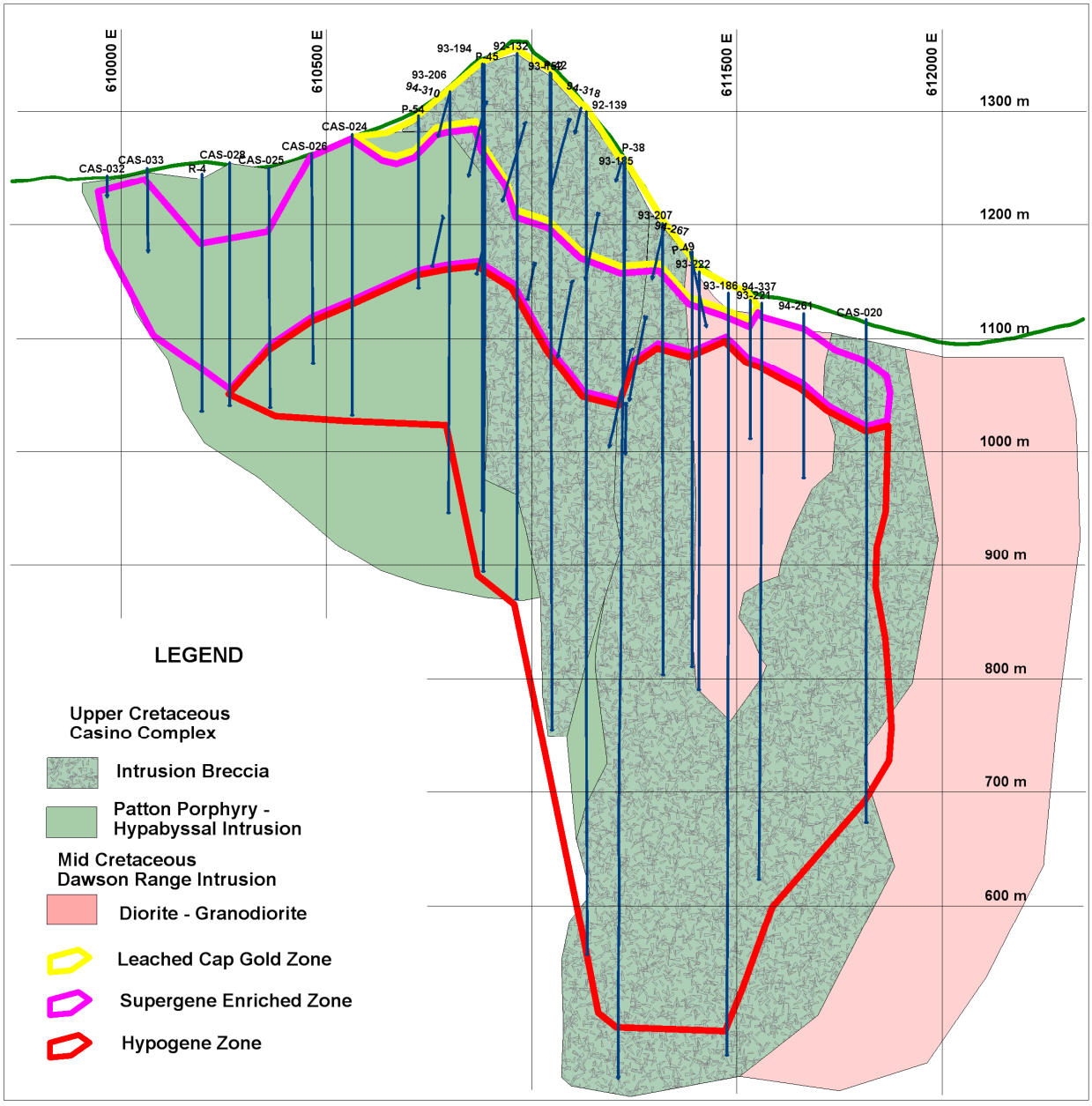


Figure 7-7: Casino Property Geology - Cross Section

7.3.4 Structurally Hosted Gold Mineralization

Structurally controlled gold mineralization within the Canadian Creek portion of the Casino property occurs mostly in the northwestern part of the property. Drilling in 2009 and 2017 discovered widespread anomalous gold mineralization associated with clay altered-shears, sheeted pyrite veins and quartz-carbonate veins hosted in both intrusive and metamorphic rocks. To date, the identified structures are generally less than 3 m thick and of short strike length. Gold is accompanied by silver, arsenic, antimony, molybdenum, barium, and bismuth.

8 DEPOSIT TYPES

The Casino deposit is best classified as a calc-alkalic porphyry deposit associated with a dacitic intrusive stock (the Patton Porphyry). Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids in the matrix of the contact breccias as well as fractures in wall rocks. Higher grades occur in the contact breccias, and grades gradually decrease away from the contact zone, both towards the centre of the stock and outward into the host granitoids and schists. A general zoning of the primary sulphides occurs, with chalcopyrite and molybdenite occurring in the core dacite and breccias, grading outward into pyrite-dominated mineralization in the surrounding granitoids and schists. Alteration accompanying the sulphide mineralization consists of an earlier phase of potassic alteration and a later overprinting of phyllic alteration. The potassic alteration typically comprises secondary biotite and K-feldspar as pervasive replacement and includes veins and stockworks of quartz and anhydrite veinlets. Phyllic alteration consists of replacements and veinlet-style sericite and silicification.

The Casino Copper deposit is unusual amongst Canadian porphyry copper deposits in having a well-developed enriched secondary supergene blanket of copper mineralization. This is similar to those found in deposits in Chile, including the Escondita deposit, and the Morenci Deposit in the southwest United States. Unlike other porphyry deposits in Canada, the Casino deposit's enriched supergene copper blanket was not eroded by the glacial action of ice sheets during the last ice age. At Casino, weathering during the Tertiary Period leached most of the copper from the upper 70 m of the deposit and re-deposited it lower in the deposit, forming the Supergene Sulphide zone. This resulted in a layer-like sequence comprising an upper leached zone, where all sulphide minerals have been oxidized and copper mostly removed leaving a bleached, iron-oxide leached cap containing residual gold. Beneath the leached cap is a zone up to 100 m thick of secondary copper mineralization, consisting primarily of chalcocite and minor covellite. The copper grades of the enriched, blanket-like zone are up to twice that of the underlying, unweathered hypogene zone hosting primary copper mineralization. Primary mineralization consists of pyrite, chalcopyrite and lesser molybdenite. The primary copper mineralization is persistent at depth, extending to more than 600 m of vertical depth, which is beneath the deepest drill holes completed to date.

The Casino deposit also differs from typical Andean/Arizonan porphyry copper deposits, in which mineralization is characterized by a dense stockwork of veinlets. Within the Casino deposit, mineralization occurs both as fine disseminations and within a low density of veinlets, with the highest concentrations occurring within the matrix of the Intrusive Breccia zones surrounding the central Patton Porphyry intrusion. In a typical Andean/Arizonan porphyry, the onset of mineralization propagates through a magma "mush" resulting in multiple veinlet-forming events as the magma cools. The final pulses permeate a cooled, fractured intrusive body, resulting in a dense stockwork of multiple superimposed veinlet-formational events, each with a distinctive mineralogy. However, at the Casino deposit, the Patton Porphyry had apparently already cooled prior to the main mineralizing event, the latter manifesting as the IX breccia containing clasts of both the Patton Porphyry and the Dawson Range Batholith. The brecciation event allowed for subsequent fluid movement into both sets of intrusive rocks but resulted in the highest grades occurring in the IX breccia. Re-interpretation of select drill core concluded that multiple events of brecciation occurred throughout the emplacement history of various phases of Patton Porphyry resulting in superposed mineralizing events during the Patton Porphyry emplacement history.

9 EXPLORATION

9.1 EXPLORATION PROCEDURES PRIOR TO 2021

The history of exploration on the property includes prospecting, geological mapping, multi-element soil geochemical sampling, magnetic and Induced Polarization (IP) surveying, trenching and diamond and reverse-circulation drilling. Drilling will be discussed in Section 10.

A Titan TM Geophysical survey was carried out by Quantec Geoscience Limited of Toronto, Ontario in 2009, to search for potential deeply buried porphyry mineralization beneath or peripheral to the Casino deposit. The survey utilized Titan-24 Galvanic Direct Current Resistivity and Induced Polarization (DC/IP) surveys as well as a Magnetotelluric Tensor Resistivity (MT) survey over the entire grid. Magnetotelluric Resistivity surveys result in high resolution and deep penetration (to 1 km), whereas the Titan DC Resistivity & Induced Polarization surveys provide reasonable depth coverage to 750 m. The survey grid, covering a 2.4 km by 2.4 km area, was centered on the Casino deposit. The grid comprised nine 2.4 km long lines, spaced 300 metres apart, at an azimuth of approximately 064°, perpendicular to the Casino Creek Fault. Results of the Titan survey were used by Quantec to identify a series of drill targets within the survey grid and adjacent to the known mineralization. A total of 10 holes, comprising 4,327 m, were drilled to test geophysical targets. With the exception of several distal Pb-Zn veins and arsenopyrite-rich veins intercepted during this drilling, no porphyry copper mineralization was found.

To the west of the Casino deposit, on the recently acquired Canadian Creek Property, exploration utilized grid soil sampling, ground magnetic and IP surveys to generate targets for trenching and drilling. Initially, the focus of the geochemical and geophysical surveys was to locate porphyry copper mineralization. Subsequent to 2016, the focus of this work switched to the identification of gold mineralization similar to that discovered at the nearby Coffee Creek project (Johnston, 2018).

Soil geochemical sampling surveys, to the west of the Casino Deposit, were carried out from the mid 1990s through to 2011. The soil surveys targeted mainly B horizon soils, but due to local talus cover and permafrost, sampling of the B horizon was not always possible. Soil samples underwent multi-element and gold analysis, mostly at Acme Analytical Labs Vancouver, using ICP methods and fire assay with atomic absorption finish for gold. The historical soil grids utilized a 200-metre line spacing and sampling spacings that ranged from 25 m to 75 m. Locally, infill sampling to define Au and As anomalies from the original grids was done at a 100-metre line spacing and a 25-metre station spacing. Soil geochemical results show a coincident Cu-Au anomaly at the 50 ppm Cu and 15 ppb Au threshold levels respectively, extending approximately 3 km west from the western limits of the Casino deposit. Copper values are shown in Figure 9-1. This coincident Cu-Au anomaly has been tested by 16 core holes. The holes closest to the Casino Deposit revealed moderate potassic alteration and strong propylitic alteration. The four closest holes intersected zones of leached cap or incipient leaching, weak supergene enrichment, and hypogene copper-gold-molybdenum mineralization typical of the outer edges of a porphyry copper-gold-molybdenum deposit. Copper grades are in the 300 - 700 ppm (0.03% to 0.07%) range, gold grades range from 0.1 g/t to 0.3 g/t, and molybdenum values range from 20 - 40 ppm (0.002% to 0.004%). There is a progressive eastward increase in Cu, Au, and Mo grades in the Casino B drill holes towards the Casino deposit. Results from these holes define the western limits of the Casino deposit system.

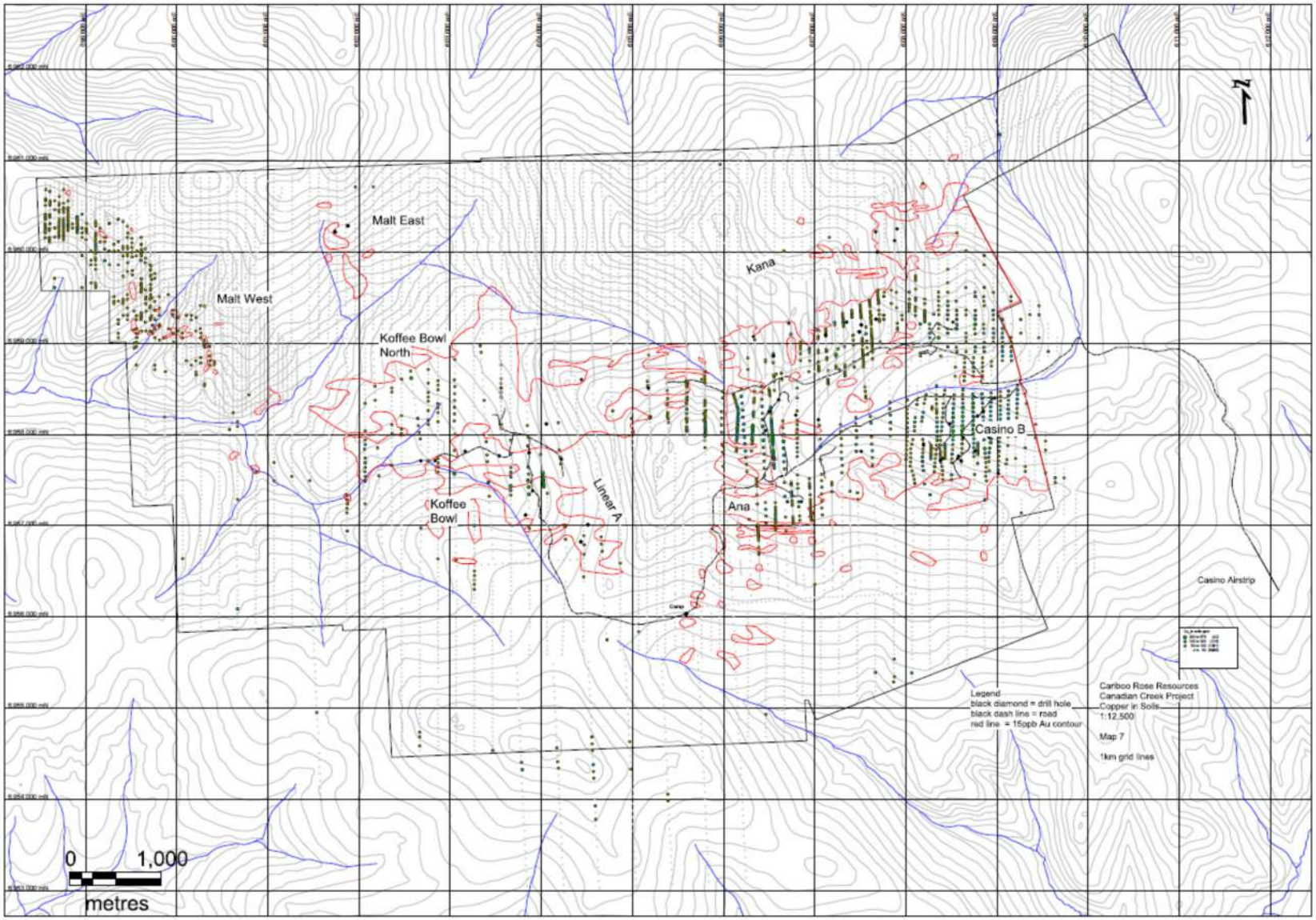


Figure 9-1: Copper and Gold in soil results (Johnson, 2018)

Elsewhere, the soil results identified a number of areas of anomalous Au, As and Bi. These anomalies were explored further with trenching, core drilling and reverse circulation drilling. This work identified scattered narrow zones of gold mineralization associated with clay-altered shears, sheeted pyrite veins and quartz-carbonate veins, hosted both in intrusive and metamorphic rocks. With few exceptions, gold grades in the structures are sub-1.0 g/t (1,000 ppb). The structures identified to date are mainly less than 3 m thick and of short strike length.

Ground magnetic surveying at a line spacing of 100 m was undertaken over the Canadian Creek portion in 2011 and 2017. The surveys detected a number of lineaments, oriented mostly northwest-southeast, though none obviously align with the soil geochemical anomalies. A plot of the un-levelled magnetic survey results of the property is shown in Figure 9-2. The ground magnetic data shows a trend of magnetic high features extending from the Casino Deposit through the Ana Zone area to the Koffee Bowl area. This west-southwest trend follows the trend of Patton Porphyry dykes extending from the main intrusive complex.

Induced polarization (IP) surveys were carried out in 1993, 1996, 2009 and 2011. The 1993 and 1996 surveys used a pole-dipole array with a spacing of 75 m and an N1 to N4 depth profile. The 2009 survey was a pole-dipole survey using an a spacing of 25 m and an N1 to N6 depth profile. The 2011 pole dipole survey used a spacing of 25 m and an N1 to N8 profile. The surveys mainly utilized small "n" spacings resulting in limited depth profiles. The survey identified a number of high chargeability anomalies which remain to be tested. A compilation of the results is shown in Figure 9-3.

9.2 EXPLORATION IN 2021

The 2021 program comprised regularly spaced reconnaissance soil geochemical sampling, re-analysis of select core from 1992 through 2020 and all 2021 core utilizing "Enersoft Inc's "GeologiAI" (Artificial Intelligence) instrument, and completion of a diamond drilling program comprising 6,074.97 metres in 22 holes, including 8 geotechnical holes for 1,957 metres. "Televiewer" and piezometer surveying was also done on the geotechnical holes. Seven additional geotechnical holes were drilled to test ground conditions for proposed construction of the Tailings Management Facility (TMF), the heap-leach facility and mineral processing facilities. Drilling will be discussed in Section 10.1.6.

Camp and logistical management, including fixed wing and heli-support, was contracted to High Altitude Construction of Whitehorse, Yukon. Kluane Diamond Drilling of Whitehorse, Yukon, completed the main diamond drilling program. For the eight geotechnical holes, televiewer surveying was done subsequent to core drilling by DGI Geoscience Inc. of Toronto, Ontario. This was followed by piezometer surveying done by TetraTech EBA Inc. based in Whitehorse, Yukon. Geotechnical holes targeting proposed milling, heap leaching and tailings facilities were completed by Kluane Drilling and supervised by Knight-Piesold Consulting of Vancouver, British Columbia.

9.2.1 Soil Sampling

A reconnaissance-style soil geochemical sampling program was completed in 2021. This program involved B-Horizon soil sampling on an evenly spaced 200-metre spaced grid, covering previously unsampled areas east and south of the Casino deposit. A total of 2,502 samples were obtained (Figure 9-2). All soil samples underwent preliminary X-ray Diffraction (XRF) analysis utilizing a portable unit encased within a protective chamber to prevent radiation leakage. Preliminary XRF analysis was done for Cu, Mo, As, Pb, Sr and for Cu: Zn and Cu: Mo ratios. The XRF results were considered sufficiently reliable to determine anomalous zones per element.

By mid-August, three Cu anomalies were identified from XRF results (Figure 9-3), and were considered robust enough to warrant diamond drilling, which was done later in 2021 (Section 10.1.6). The sampling program was still underway at the time of XRF data compilation, resulting in incomplete data coverage, compared to that for assay results. Three further anomalies, D, E and F, were identified from lab analytical results.

Figure 9-4 through Figure 9-8 show ranges of analytical results for Cu, Mo, Au, Ag, and As, respectively. Assay results for Cu from soil geochemical sampling confirmed XRF results identifying Anomalies A and C. Anomaly B was less well defined from assay results (Figure 9-4). Assay results also indicated two other areas of elevated Cu values: Anomaly D, at the extreme northeastern tip of the survey; and Anomaly E, along the northwest side of Canadian Creek at the northwestern survey margin (Figure 9-4).

High Cu values for Anomaly A show a strong correlation with Au, Ag and As, but not Cu, indicating this is unlikely to represent a porphyry-style target. The Cu-Au-Ag-As signature is more representative of outlying bonanza vein or epithermal-style mineralization, or structurally controlled mineralization, which tend to be less Cu-enriched. The anomaly extends downslope to the southwest, with highest values of all elements occurring towards the base of slope.

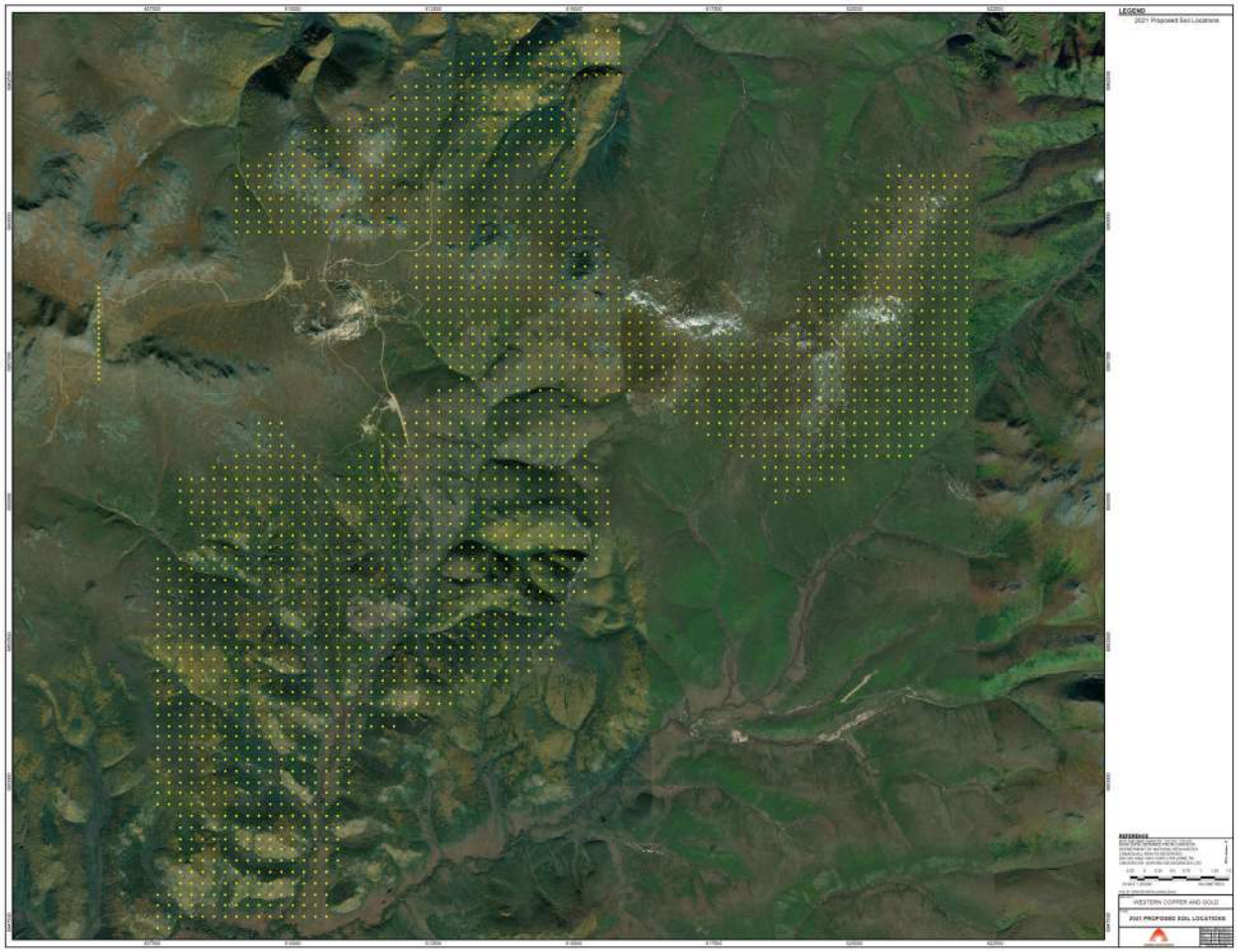


Figure 9-2: Layout of 2021 Soil Sampling Program

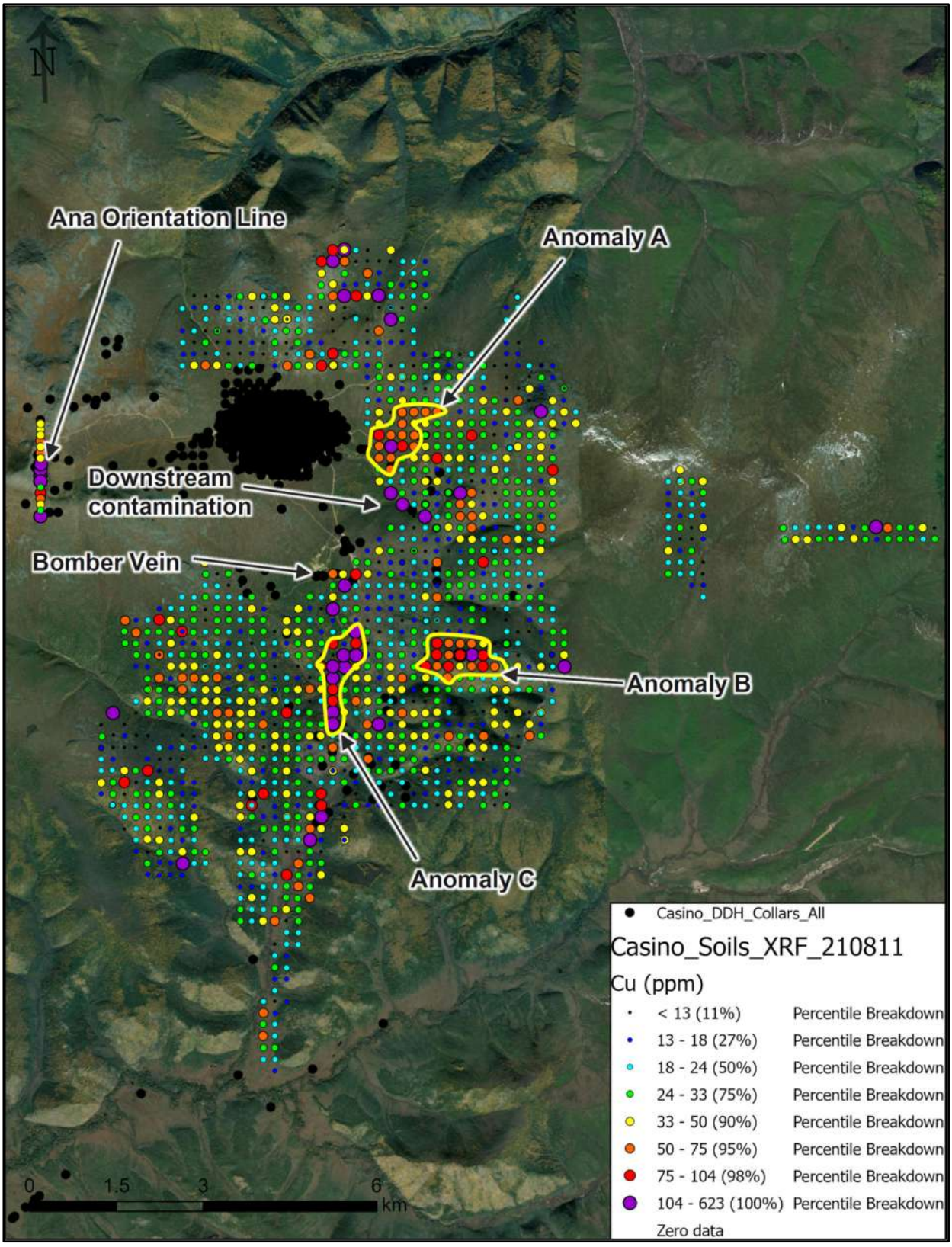


Figure 9-3: Cu assay values from on-site XRF analysis (Rio Tinto)

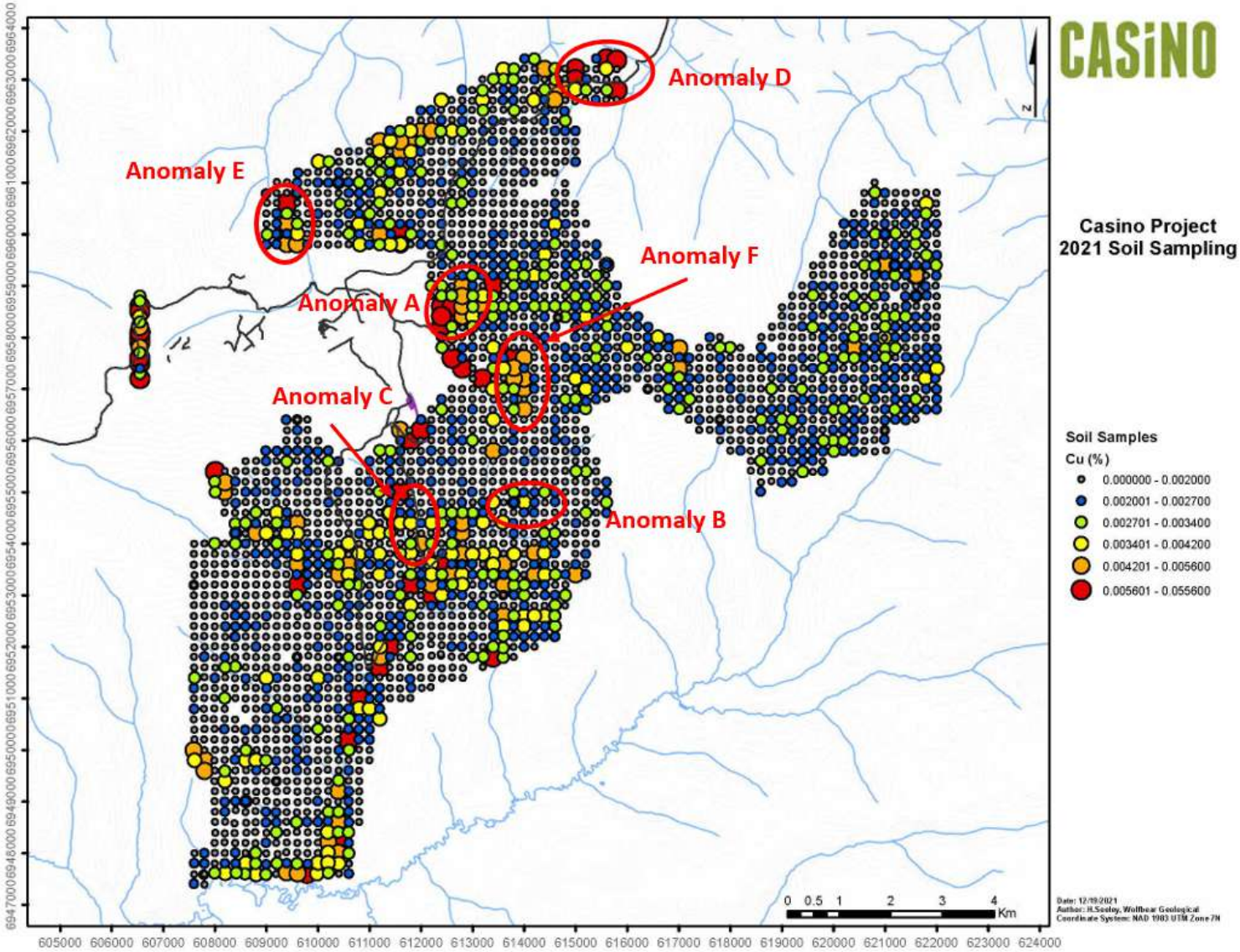


Figure 9-4: Cu assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting)

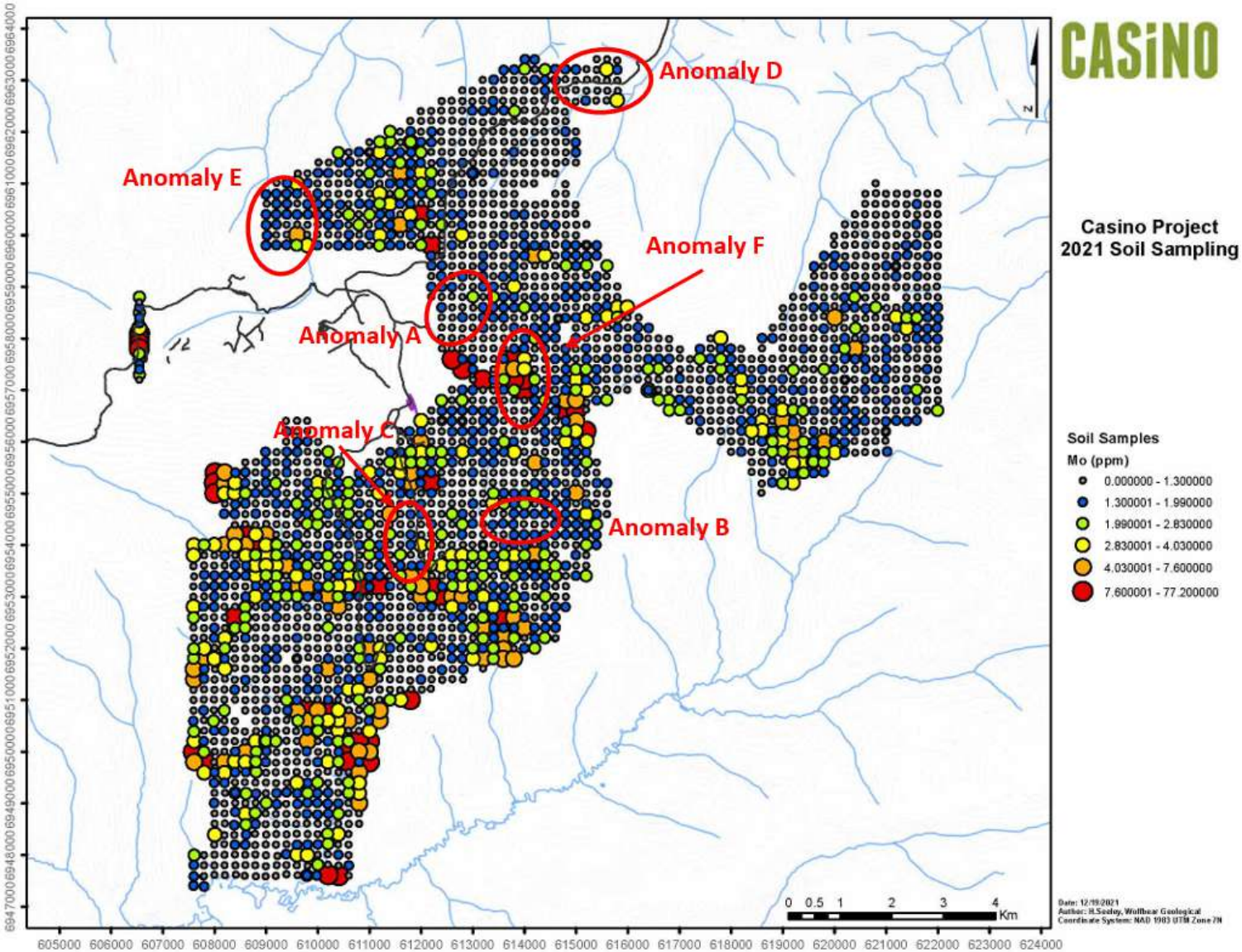


Figure 9-5: Mo assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting)

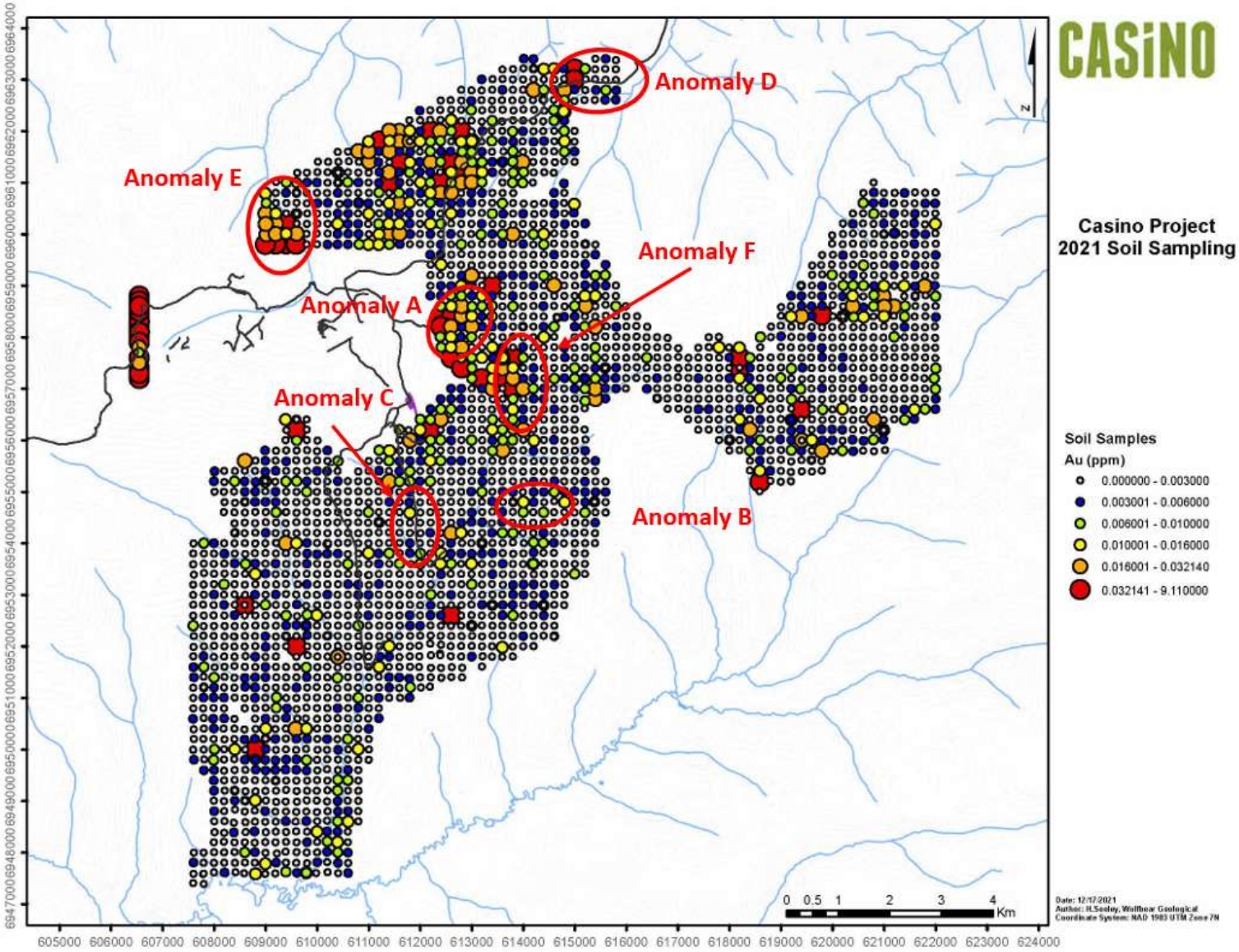


Figure 9-6: Au assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting)

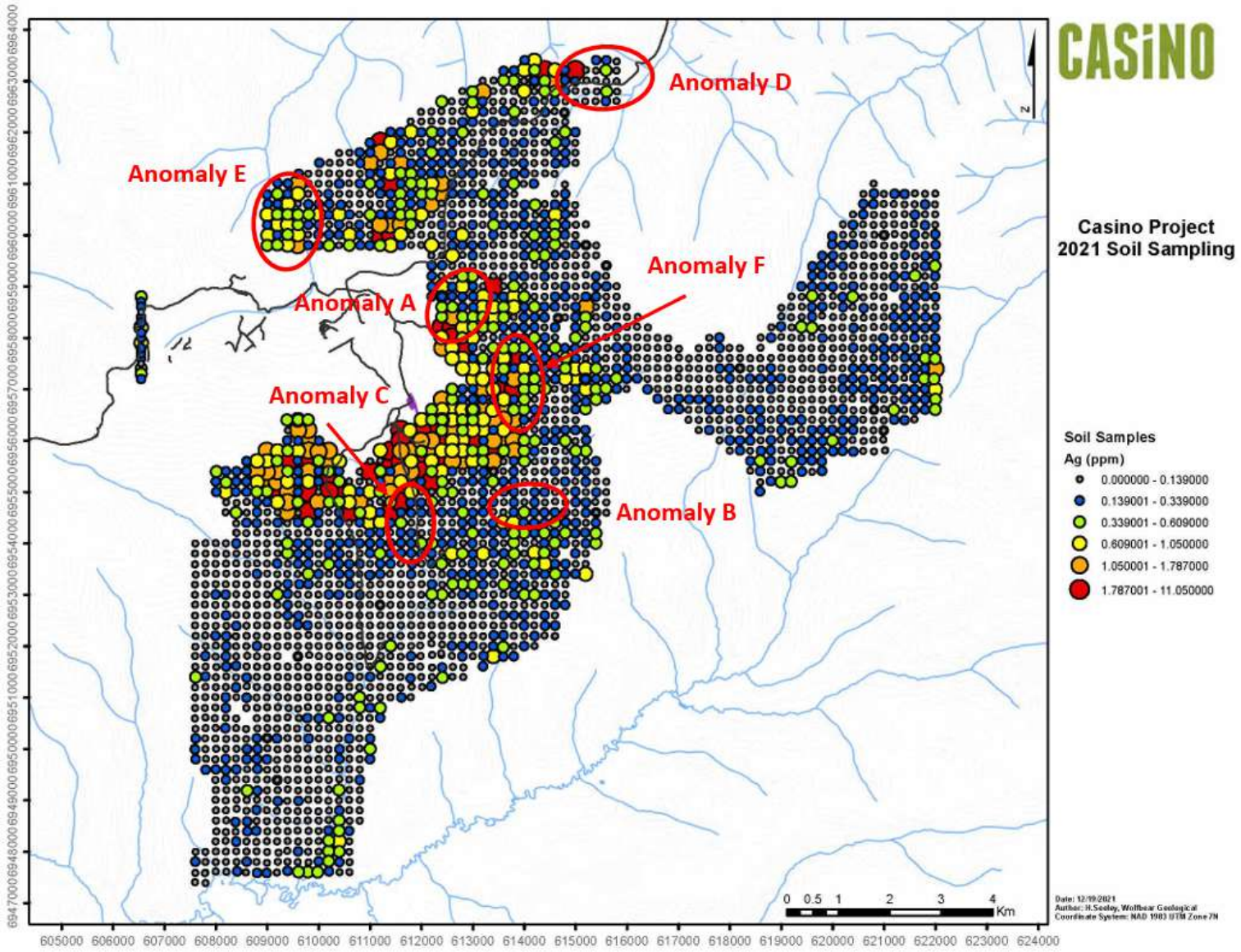


Figure 9-7: Ag assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting)

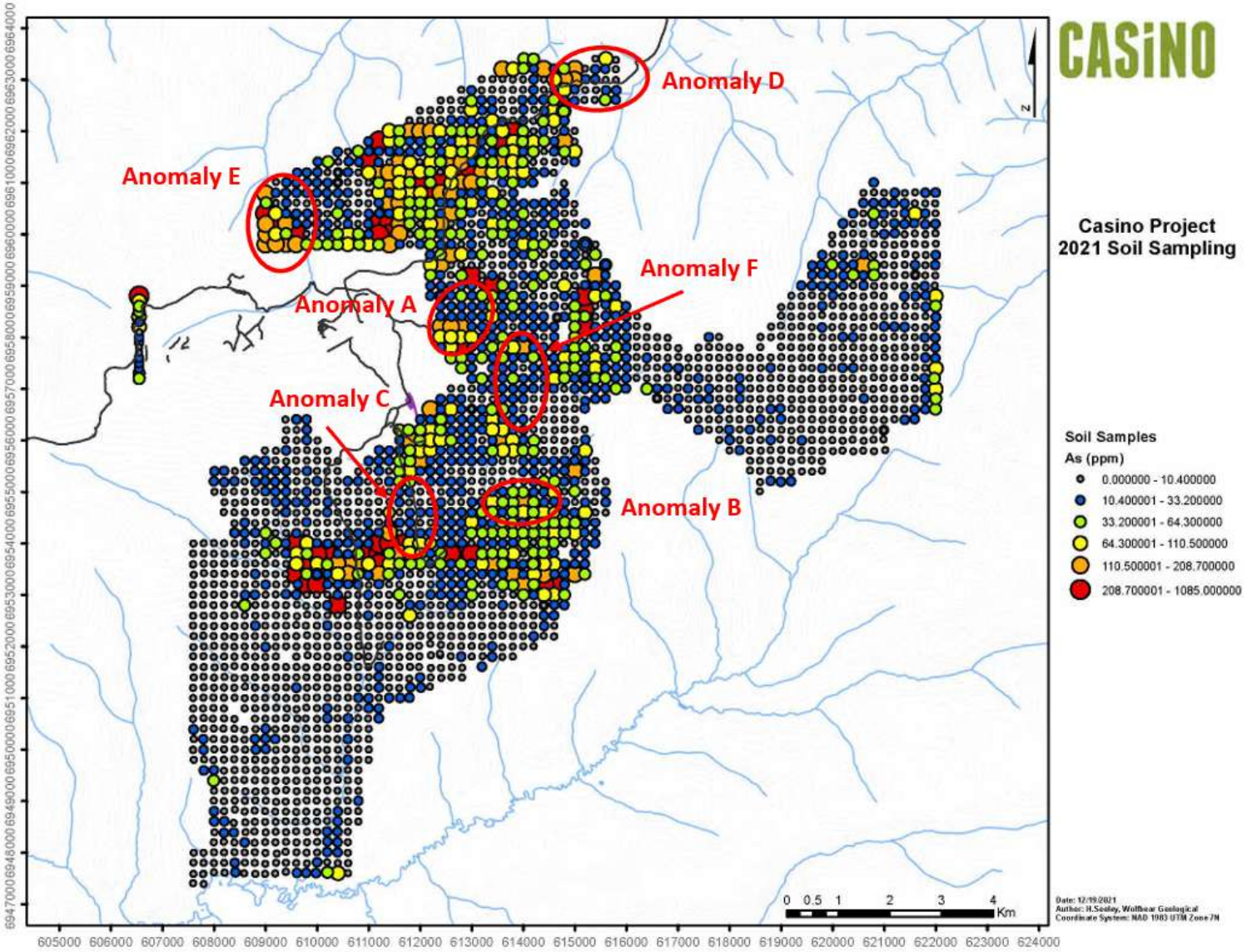


Figure 9-8: As assay ranges, 2021 soil sampling program, Casino Project (Image: Wolfbear Consulting)

At Anomaly B, laboratory analytical results for Cu (Figure 9-4) are more subdued compared to those from XRF analysis (Figure 9-4). Elevated Cu results show a weak correlation with Au and Ag values, and essentially no correlation with Mo. The anomaly covers the northern portion of an extensive As anomaly (Figure 9-8), which may represent a broad area of enrichment, rather than one specifically associated with the Cu anomaly.

At Anomaly C, lab analytical results for Cu from its southern portion supported XRF results, although less so in its northern portion. Elevated Cu results show a strong correlation with As, a moderate correlation with Mo, and little to no correlation with Au and Ag. The elevated Cu, Mo and As values form part of a continuous east-west trending band of Cu, Mo and As enrichment, with a weak Ag and negligible Au correlation. The strongly defined Cu values from XRF analysis in its northern part are less pronounced from lab analysis. Anomaly C may cover a portion of the continuous Cu-Mo-As enriched band, rather than a discreet anomaly.

At Anomaly D, Cu values in the 98th percentile in its the western part are associated with 98th percentile Au (Figure 9-6) and Ag (Figure 9-7) values, elevated values of As (Figure 9-8) but near-background values of Mo (Figure 9-5). Elevated As values represent the northeast limit of sampling of a more extensive trend, extending northeast from the deposit area. Portions of this trend also show notable Au and Ag enrichment, although this is not significantly enriched in Cu and Mo. The Au-Ag-As trend is roughly parallel to local drainage and topographic features, which in turn may be marking structural features.

Anomaly E comprises moderately elevated Cu values (Figure 9-4) from lab analysis, supporting on-site XRF results. The anomaly also comprises high Au, Ag and As values (Figure 9-6 to Figure 9-8). The anomaly covers the west flank of the Canadian Creek valley, on the opposite side of the stream from the Casino deposit. Casino Creek marks a sharp eastern terminus for anomalous metal values, indicating an upslope source to the west. The west boundary of the anomalous area is also the western limit of surveying marking the boundary of the Casino block. However, the Cu-Au-Ag anomaly likely extends farther west onto the Canadian Creek block, now 100% held by Western.

Anomaly F, southeast of Anomaly A, shows moderately elevated Cu values (Figure 9-4) with a strong correlation to Mo and Au values (Figure 9-5 and Figure 9-6). Figure 9-7 indicates a moderate correlation with Ag as well, although this is partially masked by a halo of elevated Ag values surrounding the Bomber Vein area. No significant correlation occurs with As. The geochemical signature, centered on a topographically elevated area, is the most indicative of a second potential porphyry centre, compared to Anomalies A through E.

The east-west trending As anomaly is not precisely marked by a similar Mo-Cu anomaly, but the aerial extent and locations are similar. High As values show only a very weak correlation with Ag, and a very weak and inconclusive association with Au.

9.2.2 Enersoft "AI" Analysis

In 2021, "GeologicAI" scanning instrumentation by Enersoft Inc. of Calgary, Alberta (Figure 9-9), was utilized on site to scan select core from 1992 through 2020 and all 2021 core. Scanning involved chemical XRF analysis, hyperspectral analysis for alteration and mineralization, including sulphide mineral, identification, "Light Detection and Ranging" (LiDAR) textural mapping, and high-resolution photography (Figure 9-10). The purpose was to establish consistency of scanned data across all drilling programs from 1992 through 2021. A total of 48,673 metres of core were scanned in 2021, comprising slightly over 40% of the total amount drilled from 1992 through 2020.

Figure 9-9: "GeologicAI" Scanning Apparatus (Enersoft Inc.)

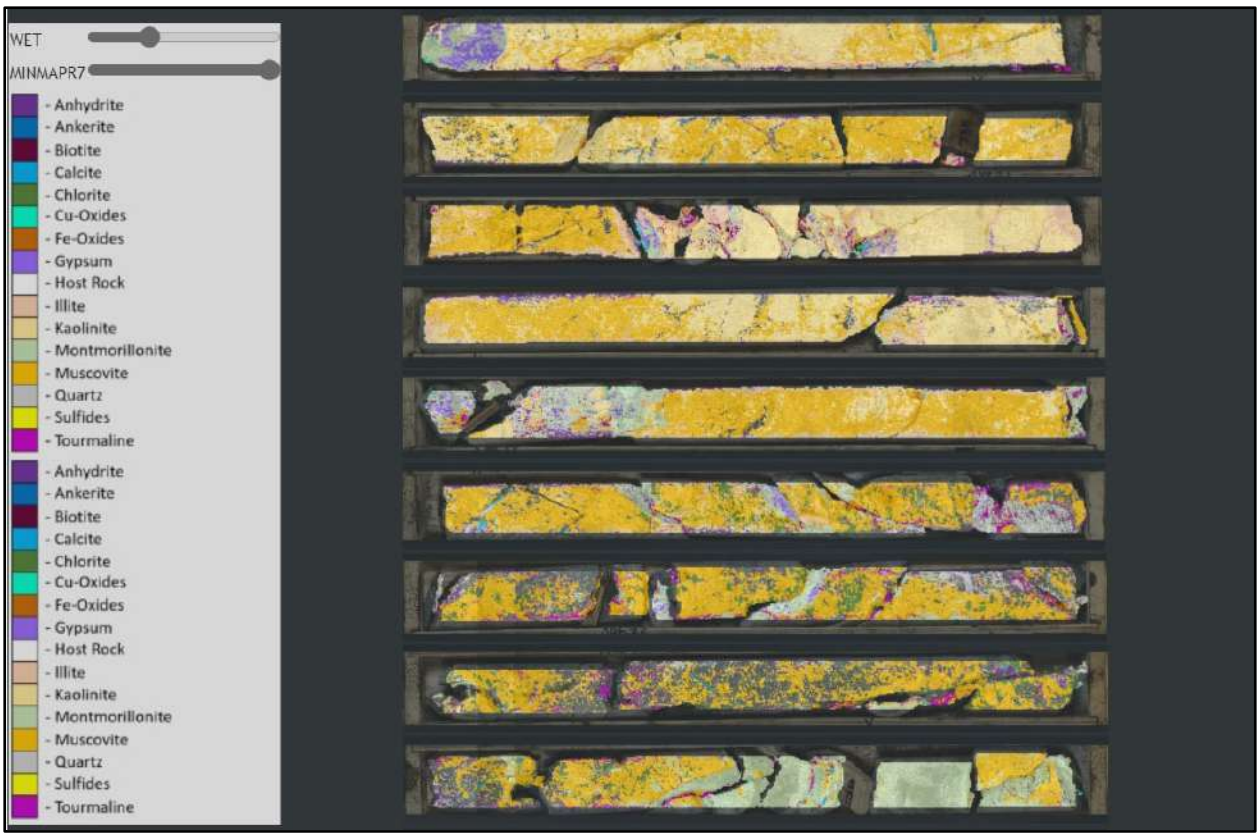


Figure 9-10: An example of Hyperspectral Scanning results, 2021 Drilling, Casino project

10 DRILLING

Targets of early diamond and reverse-circulation drilling on the Casino Deposit were based mainly on coincident copper and molybdenum-in-soil anomalies. Since 1993, with the exception of a Titan TM Survey, exploration in the vicinity of the Casino deposit has comprised drilling on a grid pattern using a core drill of NQ and NTW thickness, with a smaller number of holes drilled with HQ diameter core. The earlier soil sampling and geophysical survey anomalies, in the vicinity of the Casino Deposit, have all been tested by drilling and shown to be caused by porphyry copper-related mineralization.

The following sections describe the various drilling programs developed on the Casino Property.

10.1 1992-1994 DRILLING PROGRAM

Drilling prior to 1992 (Figure 10-1) comprised reverse circulation drilling and NQ-diameter diamond drilling. There is little documentation that specifically focused on this early drilling, its specifications, or challenges. Drilling following the acquisition of Casino Silver Mines Ltd. by Archer Cathro and Associates, then by PSG, is well documented.

The drilling campaigns from 1992 through 1994, (Figure 10-2) were contracted to E. Caron Drilling Ltd. of Whitehorse, Yukon. Up to six diamond drills were utilized. The 1994 drilling program fulfilled a variety of purposes: infill and delineation drilling, and geotechnical, structural, and waste rock characterization. Infill drilling involved a program of angle and vertical holes designed to outline and define more fully the Leached Cap (Oxide Gold zone) and Supergene copper zones. Delineation drilling to the north, northeast, east and southeast outlined the extent of the deposit area. Four oriented angle holes were drilled into the deposit area for geotechnical information, primarily rock strength and structural characteristics, and for geological information regarding vein-set orientations. Five vertical holes targeted peripheral areas of the deposit for waste rock characterization studies. Seven vertical holes were drilled into the peripheral areas of the deposit for geotechnical information. Eighteen vertical holes were drilled outside the deposit area for geotechnical and geological information regarding potential site development.

The combined drilling from 1992 through 1994 comprised 71,437.59 m of NQ and HQ core in 236 holes. Core recoveries were consistently in the 80% to 90% range in the Leached Cap and Supergene zones and 90% to 100% in the Hypogene zone.

The 1992 to 1994 collar co-ordinates (northing, easting, and elevation) were surveyed using a total station Nikon C-100 system. Surveying of the 1992 and 1993 drill holes was done by Lamerton & Associates. The 1994 holes were surveyed by Z. Peter, Surveyor from Burnaby, B.C.

Drilling can be carried out at Casino from mid-May through September under summertime conditions, although some heating of water lines may be necessary from early September onwards. Drilling can also occur in March and April, and October and early November, although these would entail wintertime conditions and the necessity of winter drilling equipment and heating of water lines. The use of a water supply truck would be necessary during very cold weather conditions, due to freezing of water lines. Three reliable water supply sites exist on the property.

10.2 2008 TO 2012 DRILLING

From 2008 through 2012 Western Copper and Gold Corp. (Western) continued the drilling pattern established by PSG, utilizing mainly a vertical drill hole orientation and a nominal 100 m grid spacing (Figure 10-3). Later in this series of programs, Western drilled a series of inclined holes in the northern part of the deposit. Several inclined holes were also drilled in the western deposit area to establish contacts with the post-mineral explosion breccia (MX) and to confirm orientation of the interpreted N-S structure.

The 2008 to 2012 drilling programs were contracted to Kluane Drilling Ltd. of Whitehorse, Yukon, utilizing up to three hydraulic diamond drills. Water for the drilling was pumped from the Canadian Creek bend, at the location of the old placer camp (which has undergone renewed placer mining commencing in 2019), and from Casino Creek.

Drilling was carried out from March through November, although drilling from March to early May, and October and November required winter-adapted drilling equipment. The main challenges during the winter drilling were water supply, due to the low water level in both creeks, and the freezing of long water lines.

All drilling was done using "thin wall" drill rods. Holes CAS-001 to CAS-007 utilized HTW-size rods (core diameter 70.92 mm) and the remainder of the drilling was done utilizing primarily NTW core size (core diameter 56.00 mm). Deeper holes were collared using HTW rods and reduced to NTW rods typically from 200 m to 300 m of depth. In a few cases, holes were reduced further to BTW core size (core diameter 42.00 mm). Core recoveries in the Leached Cap and Supergene zones were consistently in the 80% to 90% range and 90% to 100% in the Hypogene zone.

Down-hole orientation surveying was performed by Yukon Engineering Services of Whitehorse, Yukon, using an Icefield Tools MI3 Multishot Digital Borehole Survey Tool for holes CAS-002 to CAS-076. For holes CAS-077 to CAS-092, as well as the geotechnical and hydrogeological holes, a Reflex Instruments downhole survey instrument was used.

A 2010 geostatistical study indicated that the 100-metre spacing was sufficient for delineation of supergene mineralization, but only marginally sufficient for delineation of hypogene copper mineralization.

10.3 2013 DRILLING

Drilling during the 2013 field season was contracted to Kluane Drilling Ltd. of Whitehorse, Yukon. Up to two hydraulic diamond drills were used for this program.

Drilling in 2013 was primarily for water wells and hydrogeological purposes. Each hole was fully logged by core loggers, but no samples were taken. Eleven holes (MW13-01D/S through MW13-06D/S) were drilled throughout June and another fifteen (DH13-01 through DH13-12) were completed during August. See Figure 10-4 and Figure 10-5 for detailed locations of drilling.

The 2013 drill collars were surveyed by CAP Engineering from Whitehorse. CAP used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. These results were reported in UTM NAD83, Zone 7 coordinates.

No diamond drilling was completed on the property from 2014 through to the end of 2018.

10.4 2019 DRILLING

Between May and October of 2019, Kluane Drilling Ltd. of Whitehorse, Yukon, completed a 13,590-metre diamond drilling program in 72 core holes (DH 19-01 through DH 19-53, CRD 19-54 through CRD 19-59 and DH 19-60 through DH 19-69) on the Casino Property, using up to two hydraulic diamond drills. The purpose of the 2019 drill program was to infill the previous drill hole spacing, in order to upgrade the resource estimate for the deposit.

All drill holes in 2019 were of NTW core size (core diameter 56.00 mm) with the exception of several holes in difficult ground that were collared with HTW core size (core diameter 70.92 mm) and reduced to NTW when drilling conditions improved. Water was pumped from the Canadian Creek bend, from Casino Creek, and from several small ponds in the property area. Core recoveries were consistently in the 75% to 80% range within the Leached Cap, 80% to 90% within the Supergene zones and 90% to 100% within the Hypogene zone. Down-hole orientation surveying was

performed using a DeviShot Magnetic Multishot survey tool. Each drillhole was surveyed on 30-50 m increments by the Kluane drilling team.

All holes were logged, sampled, and photographed by geologists on site before samples were sent to ALS Global (ALS) in Whitehorse for analysis. A total of 20% of those pulps from ALS were randomly selected and sent on to SGS Canada Inc. (SGS) in Vancouver for Quality Assurance/Quality Control (QA/QC) check analysis.

CAP Engineering, of Whitehorse, Yukon was on site for 2 days in late August to survey the drill hole collars. A team of two people used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. Figure 10-4 and Figure 10-5 show detailed locations of the drill holes.

10.5 2020 DRILLING

Between June and September of 2020, Kluane Drilling Ltd. of Whitehorse, Yukon, drilled a total of 12,008 metres in 49 core holes (DDH20-01 through DDH20-49) on the Casino Property, utilizing up to three hydraulic drill rigs.

The majority of holes in 2020 comprised NTW core with the exception of some holes in difficult ground that were collared with HTW core and reduced to NTW when drilling conditions improved. Several NTW core holes were reduced to BTW core (where difficult rock conditions were encountered.

Core recoveries were consistently in the 75% to 90% range within the Leached Cap, 80% to 100% within the Supergene zones and 90% to 100% within the Hypogene zone. Down-hole orientation surveying was performed using a DeviShot Magnetic Multishot survey tool. Each drillhole was surveyed on 30 m to 50 m increments by the Kluane drilling team.

CAP Engineering, of Whitehorse, Yukon surveyed the drill hole collars, using a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. These results were reported in UTM NAD83, Zone 7 coordinates. See Figure 10-6 for a detailed location of the drill holes.

By the end of the 2019 program, three major zones were interpreted: 1) the “Gold Zone”, an arcuate zone along the southern and western deposit boundaries and hosting the majority of previous short high-grade gold intercepts; 2) the “North Porphyry Zone”, extending north and northwest of the main deposit; and 3) the “Canadian Zone” immediately west of the deposit. The 2020 program tested the continuity of porphyry mineralization at the “North Zone”, attempted to confirm the roughly east-west oriented “Gold Zone”, and investigated the continuity of the “Canadian Zone” .

All holes were logged, sampled, and photographed by geologists at the camp site before samples were sent to ALS Global (ALS) in Whitehorse for analysis, with 5% of those pulps from ALS randomly selected and sent on to Bureau Veritas Canada Inc. (BV) in Vancouver for QA/QC check analysis.

Drilling results in 2020 indicate the eastern area of the “Gold Zone” hosts an area of higher-grade Cu-Au mineralization, now termed the “Deposit Core” (Table 10-1) (Williams, 2021). Several holes returned CuEq values over widths significantly exceeding deposit averages. However, no further high-grade intercepts were returned from the “Gold Zone,” which is no longer considered a discrete mineralized horizon. High-grade intercepts were also determined not to be significant contributors to the mineral resource base at the Casino deposit.

At the Canadian Zone, Cu and Au values somewhat increase towards the northern limit marked by Canadian Creek. The Canadian Zone has been re-named the “Casino West” zone. Results from the North Porphyry Zone show a progressive northward decrease in Cu and Au values, essentially negating potential for a second porphyry centre in this area.

Table 10-1 lists the location per zone of the 2020 holes. Results of 2020 drilling are incorporated into this Feasibility Study.

Table 10-1: Summary of drill targets in 2020

Sector	2020 Drill holes
System Core (4 holes)	DH20-05, -08, -27, -45
"Gold Zone" (14 holes)	DH20-01, -02, -10, -12, -13, -17, -18, -20, -22, -24, -25, -29, -32, -35
Northern Porphyry (13 holes)	DH20-03, -04, -05, -07, -09, -11, -15, -42, -44, -45, -47, -48, -49
Western Sector (7 holes)	DH20-33, -34, -37, -38, -39, -41, -43
Casino West (Canadian) (8 holes)	DH20-14, -16, -19, -21, -23, -26, -28, -30
Ana Zone (3 holes)	DH20-31, -36, -40

Source: Williams, 2021

10.6 2021 DRILLING

Between June and September of 2021, Kluane Drilling Ltd. of Whitehorse, Yukon, drilled a total of 6,074.97 metres in 22 core holes (DDH20-01 through DDH20-49) on the Casino Property utilizing up to three hydraulic drill rigs. Water for the drilling was pumped from the Canadian Creek bend, from Casino Creek, and from several small ponds in the property area.

In 2021, four categories of diamond drilling were employed, as follows:

Resource Confirmation Drilling: 5 holes for 1,483 m
Metallurgical Drilling: 3 holes for 1,001 m
Geotechnical Drilling (Deposit area): 8 holes for 1,957 m
Exploration Drilling: 6 holes for 1,634 m

The majority of the Resource Confirmation, Geotechnical and Exploration holes were collared using HTW core (core diameter: 70.92 m), with some holes reduced to NTW core (core diameter 56.00 m) depending on rock conditions. Metallurgical drilling utilized PQ core (core diameter 85.00 m) to facilitate a larger sample size for metallurgical testing. The Metallurgical, Geotechnical and Resource Confirmation holes were collared mainly at the Deposit Core or along the eastern, northern, and southern periphery of the Casino deposit. Drilling results were not incorporated into this feasibility study.

All 2021 core underwent geotechnical and geological logging as well as scanning by the Enersoft "GeologicAI" instrument on site. The "Geotechnical" holes were drilled using "Split Tube" (also known as "Triple Tube") coring equipment, and underwent "Televiewer" surveying, which included down-hole optical, acoustical, and gamma-ray surveying. This was followed by piezometer surveying, designed to determine groundwater conditions per hole. To facilitate the piezometer, the holes were grouted, the instruments were lowered to their respective depths, and the holes were then cemented, sealing the piezometers in place. Televiewer surveying was done by DGI Geoscience Inc. of Toronto, Ontario, and piezometer surveying was done by TetraTech EBA Inc. based in Whitehorse, Yukon.

Core recoveries were consistently in the 75% to 90% range within the Leached Cap, 80% to 100% within the Supergene zones and 90% to 100% within the Hypogene zone. Down-hole orientation surveying was performed using a DeviShot Magnetic Multishot survey tool. Each drillhole was surveyed on 30 m to 50 m increments by the Kluane drilling team.

Challenger Geomatics, of Whitehorse, Yukon was on site for 2 days in September 2021 to survey the drill hole collars. A team of two people used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. See Table 10-2 and Figure 10-7 for a detailed location of the drill holes, and Table 10-3 for significant intercepts.

Table 10-2: 2021 Drill Collar data

Drill hole	Proposed hole	UTM East	UTM North	Elevation	Azimuth	Dip	Final Depth (m)
DDH 21-01	CAS21_RES_A	611065	6958785	1238	225	-60	352.04
DDH 21-02	CAS21_RES_C	611100	6958375	1288	315	-60	350.22
DDH 21-03	CAS21_RES_B	611220	6958506	1253	195	-60	228.60
DDH 21-04	CAS21_MET_A	610940	6958340	1358	35	-60	350.52
DDH 21-05	CAS21_RES_D	610890	6958220	1391	240	-60	300.23
DDH 21-06	CAS21_MET_B	610971	6958746	1251	215	-60	324.61
DDH 21-07	CAS21_MET_C	611430	6958705	1178	170	-55	326.14
DDH 21-08	CAS21_RES_E	611520	6958670	1169	60	-60	252.98
DDH 21-09	CAS21_GTH_E	611510	6958620	1162	100	-60	225.55
DDH 21-10	CAS21_GTH_F	611430	6958530	1165	220	-55	202.69
DDH 21-11	CAS21_GTH_C	610860	6958310	1368	240	-60	251.46
DDH 21-12	CAS21_EXP_K	611200	6959465	1323	45	-75	326.14
DDH 21-13	CAS21_GTH_A	611020	6958450	1349	140	-60	300.23
DDH 21-14	CAS21_EXP_H	612100	6958660	1136	225	-60	350.52
DDH 21-15	CAS21_GTH_B	610850	6958650	1320	260	-60	225.55
DDH-21-16	CAS21_EXP_F	612350	6958250	1089	215	-60	300.23
DDH-21-17	CAS21_GTH_D	611145	6958640	1256	55	-60	300.23
DDH-21-18	CAS21_EXP_N	612700	6958500	1171	225	-60	254.51
DDH-21-19	CAS21_EXP_O	613900	6954800	1060	315	-60	251.46
DDH-21-20	CAS21_GTH_G	610860	6958310	1368	240	-60	248.41
DDH-21-21	CAS21_EXP_P	611700	6954700	910	135	-60	150.88
DDH-21-22	CAS21_GTH-H	611430	6958530	1165	220	-55	202.99

Table 10-3: Significant Intercepts, 2021 Diamond Drilling Program

Hole	Zone[3]	From (m)	To (m)	Width[2] (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	CuEq[1] (%)
DDH21-01	All	0.50	352.04	351.54	0.17	0.21	0.99	0.027	0.45
	Cap	0.50	57.25	56.75	0.07	0.15	0.98	0.035	0.33
	SUP	57.25	177.98	120.73	0.28	0.26	1.11	0.038	0.65
	HYP	177.98	352.04	174.06	0.13	0.20	0.91	0.017	0.36
DDH21-02	All	5.20	350.22	345.02	0.20	0.45	3.06	0.038	0.74
	CAP	5.20	109.93	104.73	0.01	0.47	3.92	0.010	0.46
	Including	88.10	91.10	3.00	0.00	5.20	9.23	0.033	4.36
	SUP	109.93	217.61	107.68	0.34	0.44	2.35	0.051	0.91
	HYP	217.61	350.22	132.61	0.24	0.44	2.96	0.050	0.82
DDH21-03	All	0.00	228.60	228.60	0.10	0.21	2.36	0.026	0.40
	CAP	0.00	115.54	115.54	0.01	0.20	1.39	0.039	0.34
	SUP	115.54	228.60	113.06	0.19	0.22	3.34	0.012	0.45

Hole	Zone[3]	From (m)	To (m)	Width[2] (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	CuEq[1] (%)
DDH21-04	All	3.05	350.52	347.47	0.26	0.55	4.03	0.031	0.86
	CAP	3.05	115.91	112.86	0.05	0.61	3.67	0.020	0.65
	SUP	115.91	301.41	185.50	0.39	0.56	4.87	0.033	1.01
	HYP	301.41	350.52	49.11	0.23	0.42	1.66	0.052	0.80
DDH21-05	All	3.05	300.23	297.18	0.16	0.37	1.98	0.013	0.53
	CAP	3.05	82.00	78.95	0.04	0.49	2.23	0.008	0.48
	SUP	82.00	159.82	77.82	0.21	0.38	2.21	0.015	0.59
	HYP	159.82	300.23	140.41	0.21	0.30	1.72	0.015	0.53
DDH21-06	All	0.00	324.61	324.61	0.27	0.45	2.03	0.026	0.74
	CAP	0.00	39.62	39.62	0.11	0.19	0.77	0.007	0.30
	SUP	39.62	73.41	33.79	0.21	0.34	1.24	0.015	0.55
	HYP	73.41	324.61	251.20	0.30	0.50	2.33	0.030	0.84
	Including	115.41	151.41	36.00	0.71	0.86	2.17	0.035	1.55
DDH21-07	All	18.29	326.14	307.85	0.42	0.52	2.14	0.030	0.98
	CAP	18.29	36.58	18.29	0.05	0.53	2.82	0.019	0.58
	SUP	36.58	326.14	289.56	0.45	0.52	2.10	0.031	1.01
	Including	117.72	123.72	6.00	1.14	1.38	4.17	0.046	2.46
	HYP	145.72	326.14	180.42	0.38	0.47	2.06	0.032	0.90
DDH21-08	All	1.52	256.03	254.51	0.17	0.22	1.22	0.005	0.38
	CAP	1.52	18.57	17.05	0.06	0.12	1.02	0.004	0.18
	HYP	18.57	256.03	237.46	0.17	0.21	1.21	0.005	0.37
DDH21-09	All	0.00	225.55	225.55	0.52	0.54	3.04	0.008	1.01
	CAP	0.00	10.62	10.62	0.05	0.18	1.11	0.005	0.23
	SUP	10.62	52.42	41.80	1.60	1.61	8.92	0.018	3.04
	Including	10.62	76.42	65.80	1.32	1.33	7.09	0.019	2.53
	Including	25.67	52.42	26.75	2.18	1.92	10.80	0.020	3.90
	Including	34.67	35.67	1.00	3.27	4.20	112.00	0.018	7.74
	HYP	52.42	225.55	173.13	0.29	0.30	1.73	0.006	0.56
DDH21-10	All	13.76	123.6	109.84	0.18	0.20	0.91	0.004	0.36
	CAP	13.76	18.49	4.73	0.18	0.24	1.30	0.004	0.40
	HYP	18.49	123.60	105.11	0.18	0.20	0.89	0.004	0.36
DDH21-11	All	1.52	251.46	249.94	0.17	0.44	2.44	0.012	0.59
	CAP	1.52	76.52	75.00	0.04	0.50	2.89	0.013	0.52
	SUP	76.52	142.52	66.00	0.24	0.48	2.04	0.012	0.69
	Including	97.52	115.52	18.00	0.28	0.69	2.29	0.019	0.92
	HYP	142.5	251.46	108.94	0.23	0.36	2.36	0.012	0.59
DDH21-12	n/a	162.80	168.80	6.00	0.09	0.26	3.70	0.000	0.34
DDH21-13	All	0.00	300.23	300.23	0.19	0.54	4.09	0.010	0.70
	CAP	0.00	152.00	152.00	0.01	0.48	3.90	0.013	0.49
	HYP	152.00	300.23	148.23	0.37	0.59	4.29	0.008	0.92
	Including	179.00	200.00	21.00	0.23	1.51	5.86	0.005	1.50
	Including	207.65	223.35	15.70	0.99	1.31	8.68	0.017	2.18
	Including	216.65	218.79	2.14	2.29	5.88	23.90	0.016	7.26
DDH21-14	n/a	114.04	120.04	6.00	0.07	0.18	0.90	0.000	0.22
	n/a	141.04	150.04	9.00	0.05	0.18	1.56	0.000	0.20
	n/a	191.00	203.00	12.00	0.04	0.96	0.20	0.000	0.81
	n/a	269.00	323.35	54.35	0.02	0.86	5.45	0.000	0.78
DDH21-15	All	4.57	225.55	220.98	0.19	0.33	2.36	0.015	0.54
	CAP	4.57	60.62	56.05	0.09	0.59	3.00	0.014	0.64
	SUP	60.62	167.59	106.97	0.25	0.26	1.93	0.011	0.52
	HYP	167.59	225.55	57.96	0.18	0.21	2.54	0.024	0.46
DDH21-16	n/a	149.5	150.5	1.00	0.01	0.20	18.75	0.000	0.34

Hole	Zone[3]	From (m)	To (m)	Width[2] (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	CuEq[1] (%)
DDH21-17	All	0.00	300.23	300.23	0.24	0.28	1.38	0.021	0.57
	CAP	0.00	21.00	21.00	0.15	0.40	2.12	0.017	0.56
	SUP	21.00	36.00	15.00	0.51	0.55	2.49	0.028	1.09
	CAP	36.00	94.00	58.00	0.14	0.37	1.88	0.017	0.53
	SUP	94.00	154.00	60.00	0.46	0.31	1.18	0.029	0.84
	HYP	154.00	300.23	146.23	0.18	0.19	1.04	0.020	0.42
DDH21-18	n/a	191.05	206.05	15.00	0.05	0.28	2.04	0.000	0.29
DDH21-19	n/a	No significant intervals							
DDH21-20	All	0.00	248.41	248.41	0.16	0.45	2.15	0.011	0.58
	CAP	0.00	75	75.00	0.04	0.48	2.95	0.012	0.50
	SUP	75.00	143.05	68.05	0.25	0.57	2.24	0.011	0.77
	Including	101.05	122.05	21.00	0.33	0.84	3.04	0.017	1.10
	HYP	143.05	248.41	105.36	0.19	0.35	1.53	0.009	0.52
DDH21-21	n/a	No significant intervals							
DDH21-22	SUP	13.72	118	104.28	0.19	0.16	0.92	0.007	0.35

[1]CuEq Metal Prices: US$2.75/lb copper, US$1,500/oz gold, US$11/lb molybdenum, US$18/oz silver with no adjustment for metallurgical recovery.
[2]Widths are core length, not true width of mineralized intersection
[3]Zone refers to oxidation zone. LC designates material from the "Leached Cap" zone and SUL to material from the "Sulphide" zone comprised of the supergene and hypogene zones.

The 2021 program comprised 16 holes within the Casino deposit resource boundaries, centered on and extending somewhat outbound of the "Deposit Core" area. The deposit-area drilling comprised 5 resource confirmation holes, 3 metallurgical testing holes, and 8 holes for geotechnical analysis. An additional 6 exploration holes were drilled outside of the deposit area. All holes within the deposit area returned values that confirmed and, in some cases, exceeded previous and historical drilling results (Figure 10-4).

At Casino, higher grades are hosted mostly by intrusive breccias and, in the Deposit Core, also by Patton Porphyry intrusive rock. Holes DDH21-07, -08 and -09, collared east of, and at a significantly lower elevation than, the deposit core indicate that higher grades occur in brecciated zones east of the deposit core. These high-grade intervals may be influenced by proximity to the Casino Fault, potentially providing an additional exploration target for the Casino project.

Significant CuEq values have now been established for the various metallurgical (MET) zones comprising the Casino deposit (Table 10-4). Analysis of results within respective zones continue to support pre-2021 results showing that the supergene enrichment zones host somewhat higher CuEq values than the underlying hypogene zone. Significantly higher supergene to hypogene ratios in DDH21-09 may reflect structurally controlled mineralization along the Casino fault.

The exploration holes returned low to negligible metal values. The exception is Hole DDH21-14, which returned a 12.00-metre interval from 191.00 m to 203.00 m grading 0.963 g/t Au, 0.20 g/t Ag and 0.0417% Cu for a CuEq value of 0.8104% Cu. This was followed by a 54.35-metre intercept from 269.00 m to 323.35 m grading 0.861 g/t Au, 5.45 g/t Ag and 0.0198% Cu for a CuEq value of 0.7812%. Values for Mo are negligible for both holes. The geochemical signature of the lower zone, showing very strongly anomalous As and Sb values, and somewhat elevated Bi, Pb and Zn values, is quite distinct from that of supergene or hypogene mineralization within the deposit. The geochemical signature of mineralization in DDH21-14 is more indicative of hydrothermally derived "bonanza-style" mineralization, although the Au and Ag grades are considerably lower than for many bonanza-style zones.

Widths within the deposit are not necessarily true widths, particularly in outlying areas, although holes in the central deposit area closely approximate true widths due to deposit geometry. The true widths of mineralized intervals within the six exploration holes are unknown.

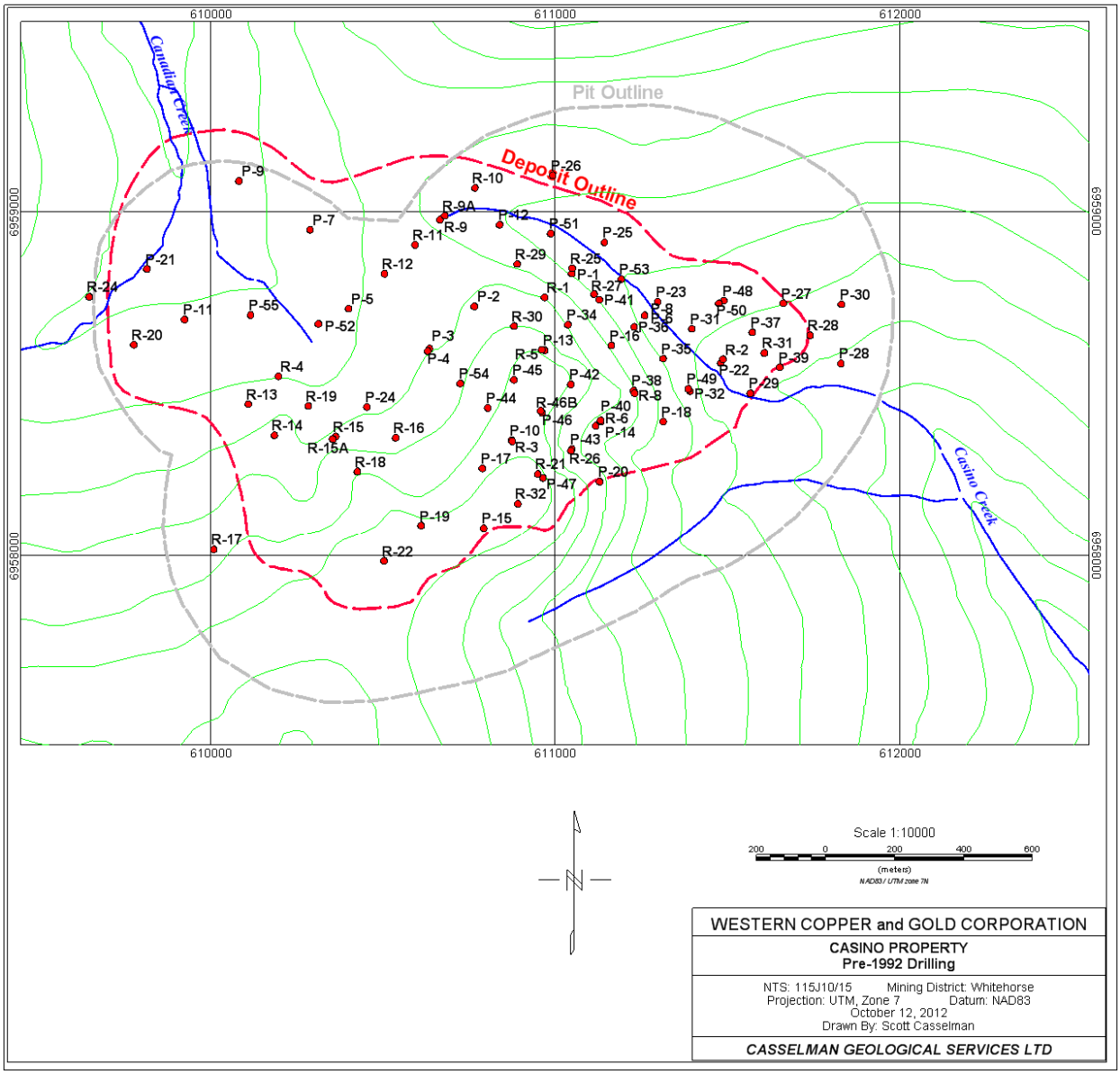


Figure 10-1: Casino Property Drilling Pre-1992

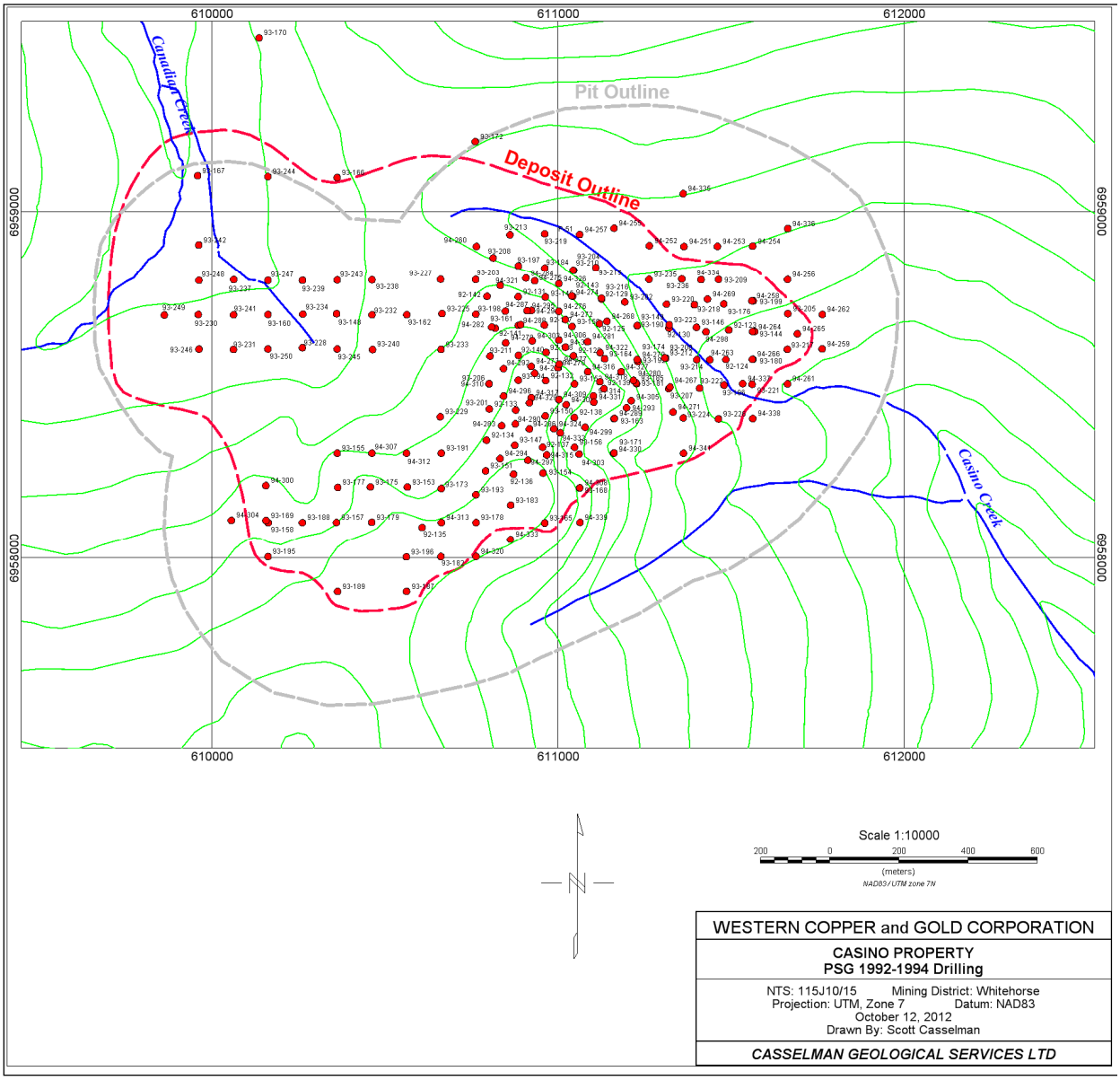


Figure 10-2: Casino Property Drilling 1992 to 1994

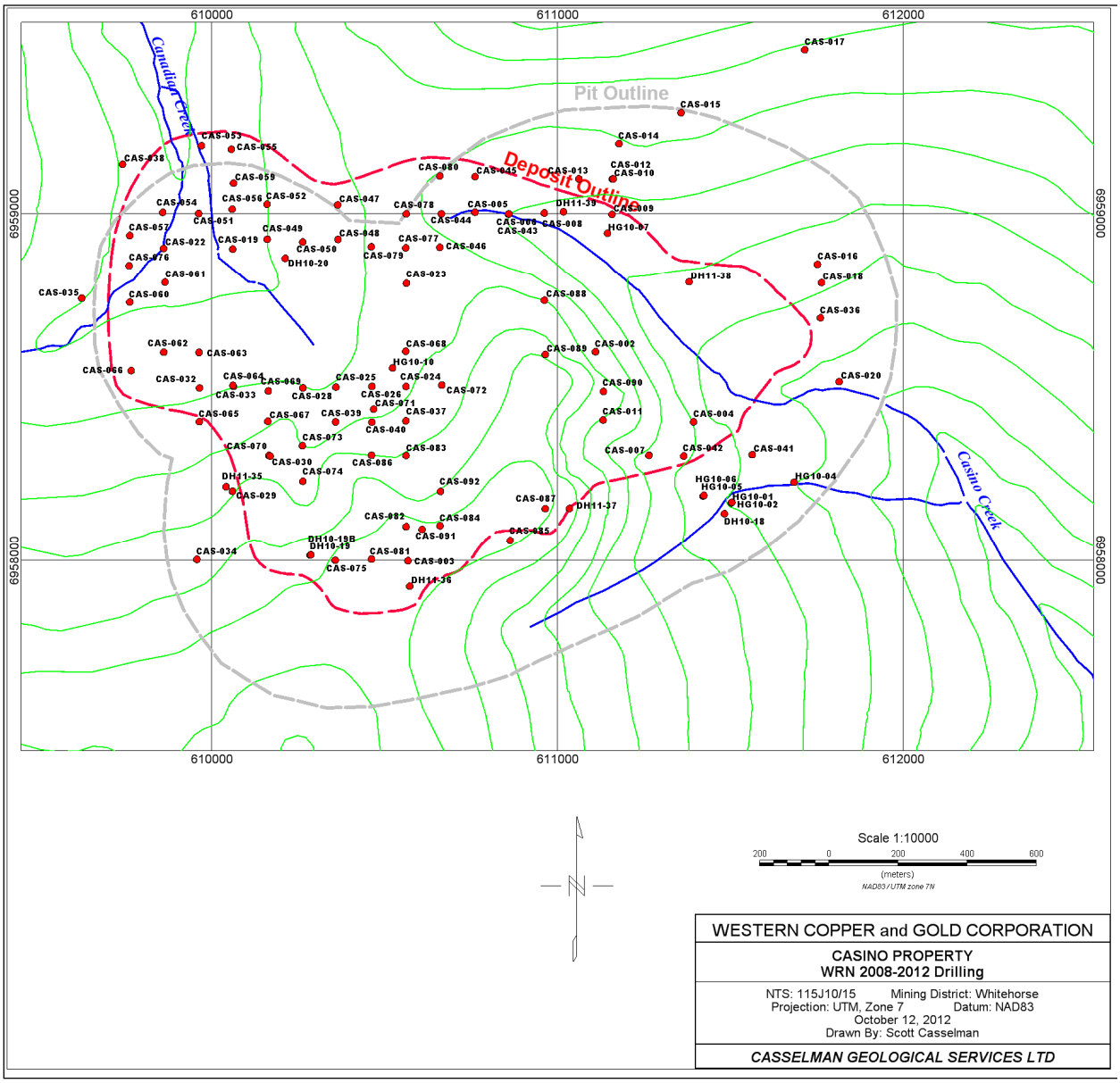


Figure 10-3: Casino Property Drilling from 2008 to 2012

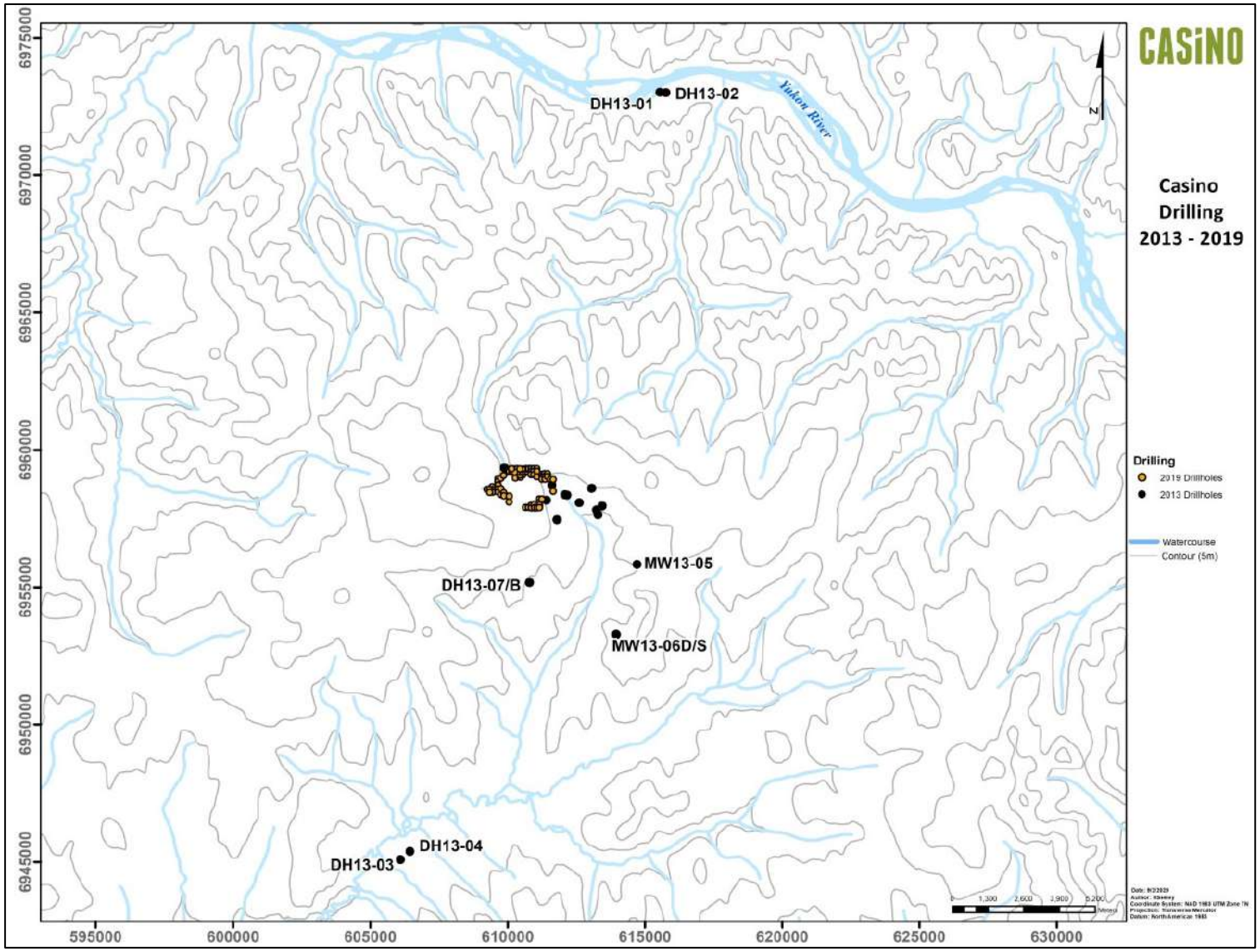


Figure 10-4: Casino Property Drilling, from 2013 – 2019

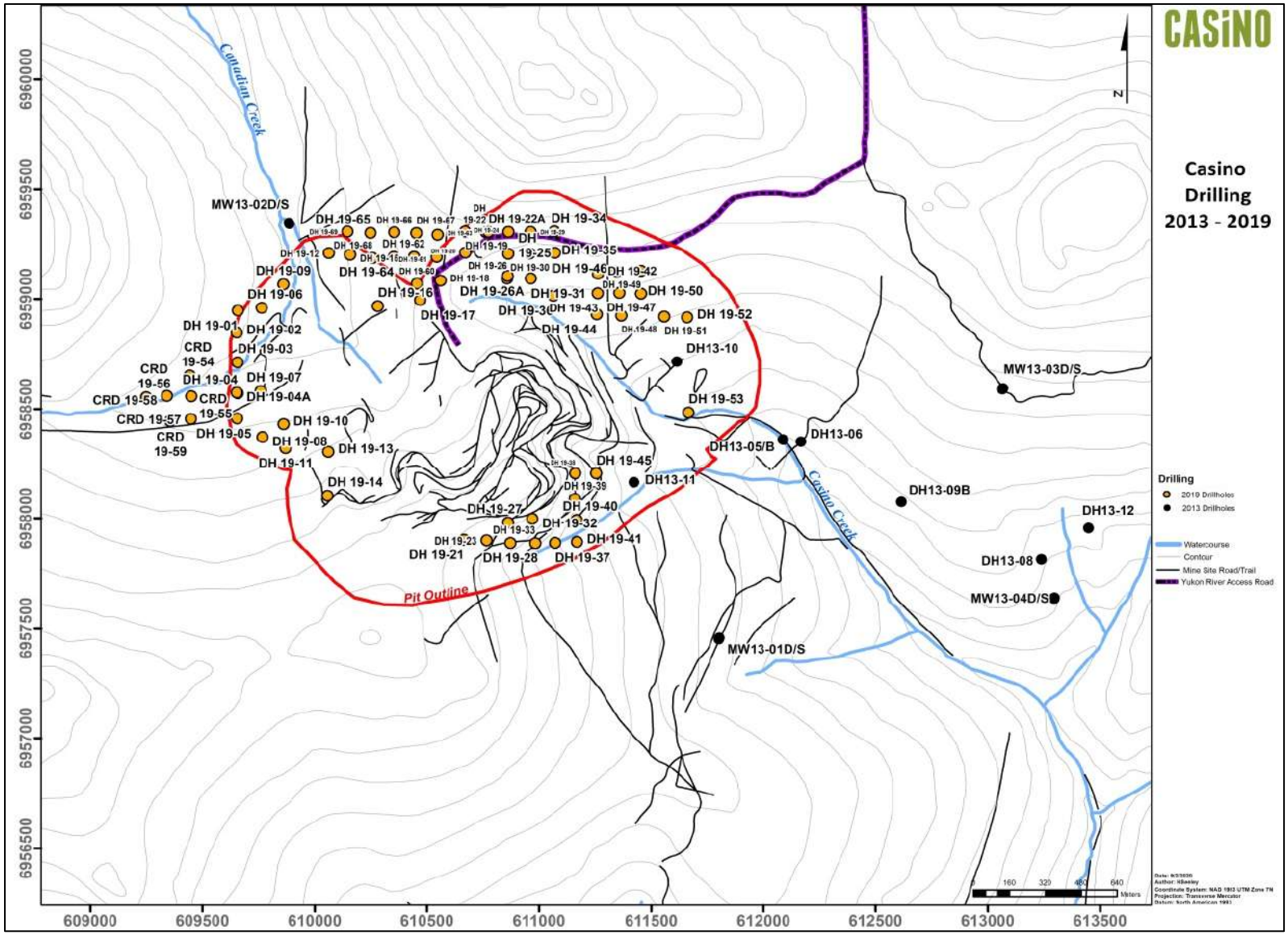


Figure 10-5: Detail, Casino Property Drilling, 2013-2019

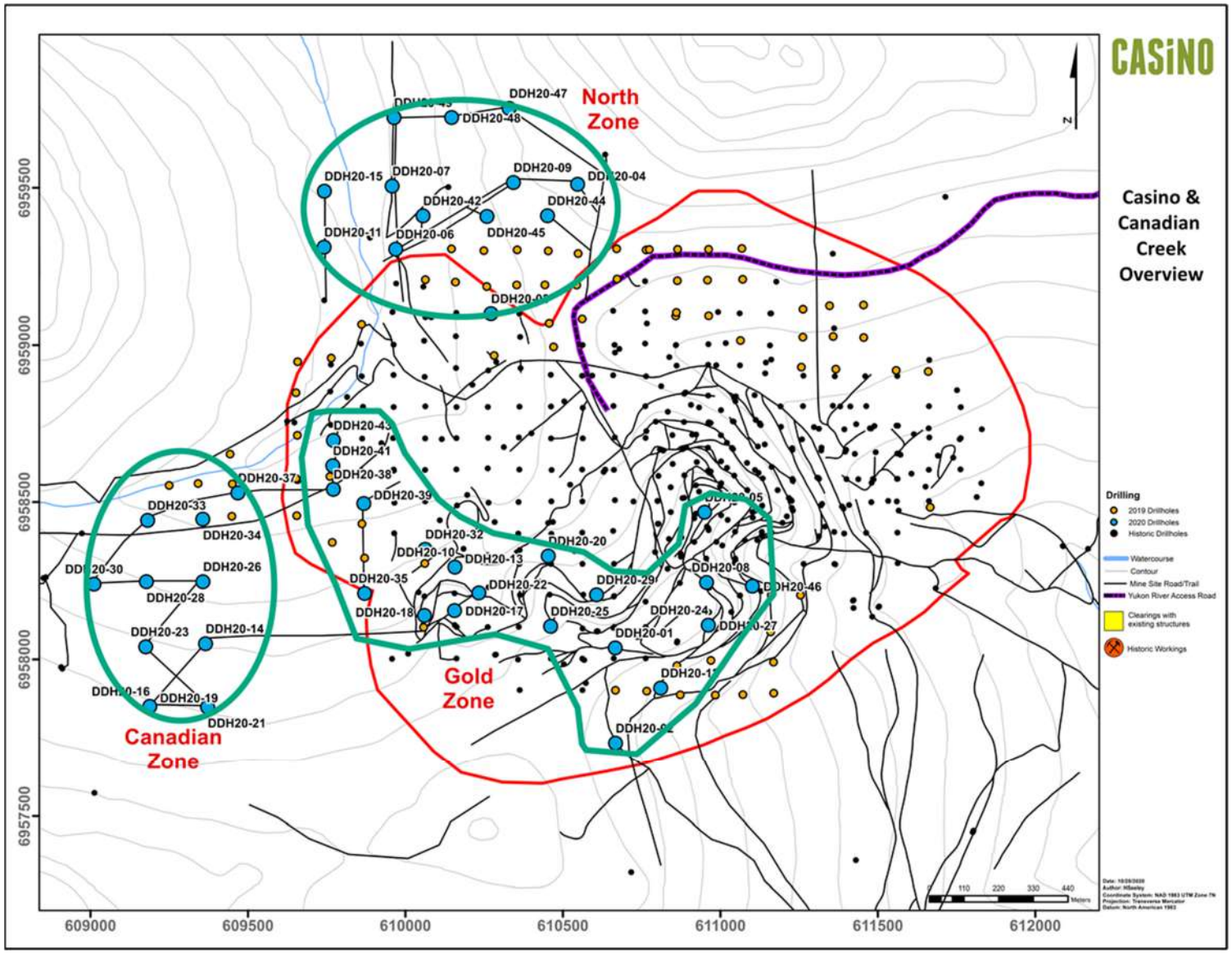


Figure 10-6: Casino Property Drilling, 2020

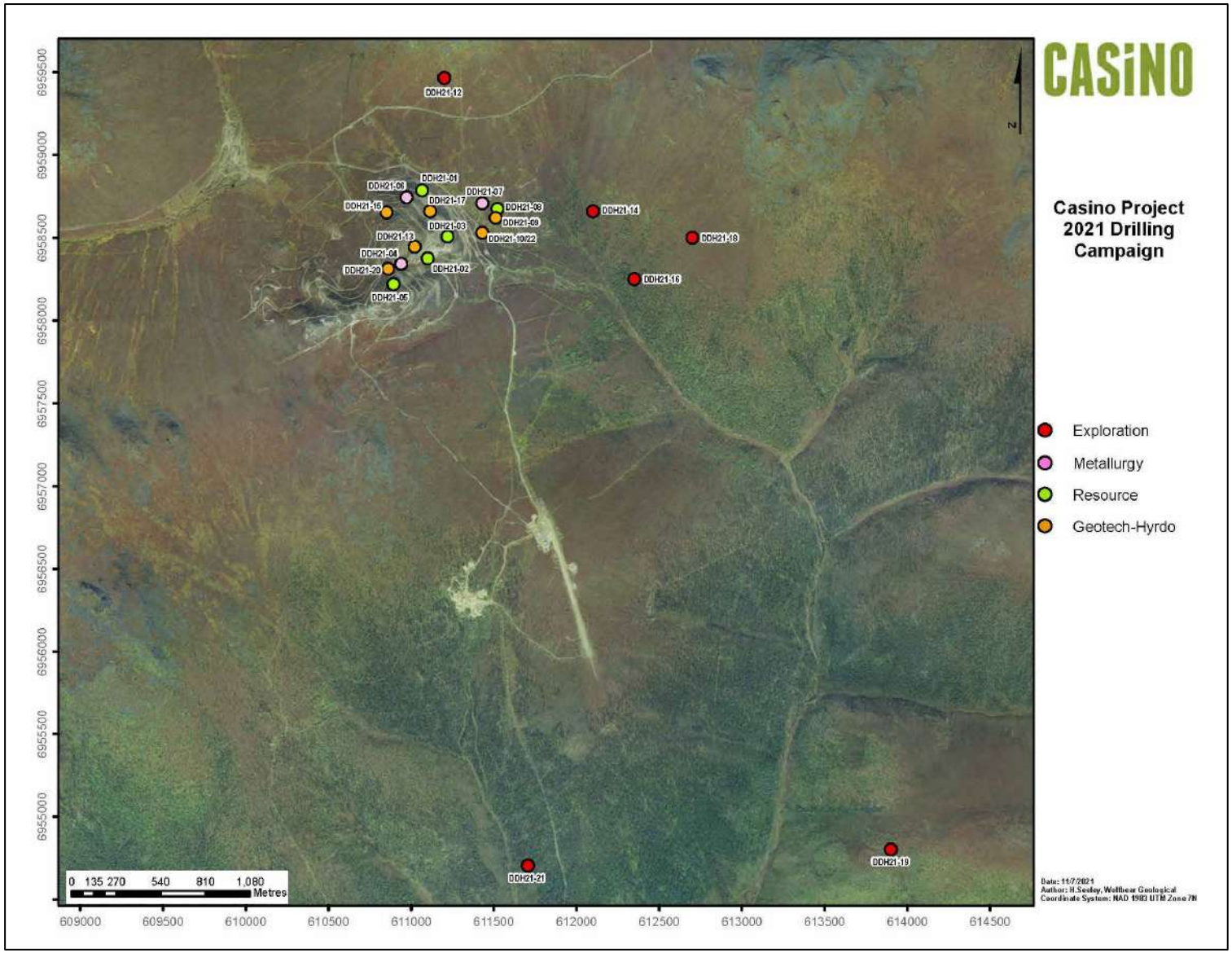


Figure 10-7: 2021 Diamond Drill Locations, 2021 Program (Image by H. Seeley, Wolfbear Geological)

10.7 CANADIAN CREEK DRILLING SUMMARY

Following acquisition of the Canadian Creek property by Western in 2019, all drilling data was transferred from Cariboo Rose Resources Ltd. and is summarised in Table 10-4. Since 1992, when exploration first began on the Canadian Creek property, soil sampling, trenching, geophysical surveys, and drilling have focused on several areas of interest. A full history of the Canadian Creek property can be found in Section 6 of this report.

Table 10-4 summarizes drilling at Canadian Creek from 1970 to 2017.

Table 10-4: Summary of Canadian Creek Drilling

Year	Drilling Summary (# holes)	Area	Type of Drilling
1970	2	Casino B	Diamond Drilling
1993	7	Ana, Koffee	Diamond Drilling
1994	4	Casino B	Diamond Drilling
2000	11	Ana, Casino B, Koffee	Diamond Drilling
2007	5	Casino B	Diamond Drilling
2009	10	Kana	Diamond Drilling
2017	24	Various	Reverse Circulation Drilling

10.8 SENSITIVITY DATA PHOTOGRAMMETRY

In April 1993, McElhanney Consulting Services Ltd. of Vancouver, BC, produced a map of the Casino area based on 1985 air photos provided by the Department of Energy, Mines and Resources.

New aerial photography was conducted in July 1993, by Lamerton & Associates of Whitehorse. The area was mapped by Eagle Mapping Services Ltd. of Port Coquitlam, BC. Eagle Mapping utilized two government UTM co-ordinates systems, NAD83 and WGS84, in the derivation of the deposit grid co-ordinates at photo target station #11. The following transformation parameters were used to convert from UTM coordinates to Property Grid:

ROTATION:	-0° 00' 05"
SCALE:	1.000453652
TRANSLATION:	-6703701.92 N
	-499861.96 E
ELEVATION SHIFT:	-8.32 m

The contours on McElhanney and Eagle Mapping Services maps compare to within approximately 5 m and commonly closer. Generally, Eagle Mapping contours are smoother, having more gradual changes in direction.

10.9 COLLAR COORDINATES

The 1992 to 1994 collar co-ordinates (northing, easting, and elevation) were surveyed using a total station Nikon C-100 system. Surveying of the 1992 and 1993 drill holes was undertaken by Lamerton & Associates. The 1994 holes were surveyed by Z. Peter, Surveyor from Burnaby, B.C. Note: All of Pacific Sentinel's collar coordinates were surveyed in local grid coordinates.

The 2008-2012 drill collars were surveyed by Yukon Engineering Services from Whitehorse. The survey was completed using Differential GPS units and the results are reported in UTM NAD83, Zone 7 coordinates. Yukon Engineering also re-surveyed 28 of Pacific Sentinel's 1992 – 1994 drill hole collars for comparison purposes. Those were entered into the data base with their new UTM NAD83 collar coordinates.

The 2013 and 2019 drill collars were surveyed by CAP Engineering from Whitehorse. CAP used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. These results were reported in UTM NAD83, Zone 7 coordinates.

10.10 SPERRY SUN SURVEYS

During the 1993 drilling program, all drill holes, including 1992 holes deepened in 1993, underwent down-hole surveying utilizing a Sperry Sun magnetic compass tool to determine azimuth and dip. In the 1994 drilling program, only angle holes were surveyed by Sperry Sun. Tests were normally performed every 152 m (500 ft) down-hole on vertical holes, and every 76 m (250 ft) down-hole on angle holes. In 1994, when the program comprised shallower angle holes, Sperry Sun tests were taken at the bottom and mid-points of the holes.

The Sperry Sun surveys averaged a dip reading of 89.03° from 123 vertical holes that were either drilled or deepened in 1993. As the average deviation was less than one degree, it was decided not to survey the vertical holes of the 1994 program.

10.11 LIGHT-LOG SURVEY SYSTEM

A Light-Log directional drill hole survey system, developed by H.J. Otte & Co., was used for sixteen angle holes at Casino, starting at hole 94-285 and continuing for most angle holes through the remainder of the 1994 drilling program. This system recorded, on film, the bending of the unit caused by the natural curvature in the drill hole. The instrument's timer activated the camera and advanced the film at pre-set time intervals, allowing the instrument to be lowered (normally every 3 m) into place between pictures. Upon completion of the survey, the film was developed. The values observed on the film were converted by a computer program to provide co-ordinates, dip, and azimuth at every three-metre interval downhole.

10.12 ACID DIP TESTS

In the 1994 program, acid dip tests were performed in the vertical holes while Sperry Sun surveys were continued in the angle holes.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section summarizes the 2019 and 2021 sampling and assaying protocols that have been utilized at the Casino Project site.

11.1 SAMPLING METHOD AND APPROACH

11.1.1 Core Processing

In 2019, core was placed into wooden core boxes directly upon emptying of the core tube at the drill site. A wooden block marked with the depth, both in feet and metres, was placed in the core box upon completion of each drill run. Under good ground conditions, each run comprises 10 ft (3.05 m) of core. Core boxes were stored at the core logging facility adjacent to the Casino Airstrip. As core came in from the rig, each hole was stacked separately and clearly labelled on core racks outside of the core shack. When the core was ready to be logged, it was laid out in sequence on elevated tables in the core shack.

Core logging commenced with geotechnical logging. Core boxes were labelled with black felt tip pens and embossed steel tags containing hole number, box number and interval of core within the box. Geotechnical data, including core recovery, rock quality designation (RQD), hardness and natural breaks, were recorded for each drill run, as marked by the wooden core run blocks. In 2019 this information was recorded on paper by the geologist or geotechnical logger, supervised by the lead geologist. Logging of the geotechnical data followed codes and formats outlined in a project-specific manual prepared by Knight Piésold. In 2020 and 2021 all data was entered directly into a "Geospark" database.

In 2019 the geologist recorded key geologic information, including lithology, zone, mineralization, and alteration. The data was entered onto paper. The codes and logging forms followed, as closely as possible, the format used by Western during the 2010-2012 drilling programs. The lithology codes, copper mineralization zone codes and alteration codes utilized in the 2013 to 2020 drilling programs were all initially developed by Pacific Sentinel and modified by Western.

Core was photographed after the geological logging was completed and after the sample intervals were marked.

The core processing protocol for 2020 was similar to that for 2019. Core boxes were laid out outside of the core shacks, then cleaned utilizing spray bottles and/or brushes in preparation for geological summary logging. All core then underwent calculation of box intervals, core recoveries and "Rock Quality Designation" (RQD), prior to more detailed geotechnical logging comprising core hardness, nature of core fractures and "breaks" and type of fracture-filling material, if any. The core was then logged for lithology, alteration, mineralization, colour, structural characteristics and metallurgical ("Met") zones. Select intervals were also analyzed by an XRF device for Au, Cu, Mo, and Ag. Sample intervals were also laid out, and were typically 3.0 m in length, although shorter if a lithological contact or significant structural zone was encountered. Following logging, all core was digitally photographed, utilizing the same location to minimize effects of varying light conditions. Two boxes were photographed at once, together with a "white board" showing the box numbers and core intervals.

All 2020 and 2021 data were entered into a Geospark database, uploaded every evening to a "Share Point" portal managed by Wolfbear Geological Consulting (Wolfbear). All data were subsequently managed by Wolfbear and paired with analytical results upon their receipt.

The core preparation and geotechnical protocols for 2021 were essentially the same as for 2020. The core logging protocol was considerably modified, based on development of a significantly more detailed table of lithological codes and descriptions by Rio Tinto at the onset of the program. Summary logs with less detail than those of 2020 were prepared either during or directly following detailed core logging, to ensure consistency of lithological and alteration characteristics with the detailed logging. All data, including the expanded lithological codes, was entered into a

Geospark database, again uploaded every evening to a "Share Point" portal, managed by Wolfbear. Core sample intervals were the same as for 2019 and 2020.

11.1.2 Core Sampling

Sampling and analytical protocols in use prior to the 1992 – 1994 PSG diamond drill programs are not well documented. In June 1992, core from 22 previous holes was re-sampled by Archer Cathro. The new assay results for all metals were compared to the originals. The results indicated 14 holes (64%) had essentially identical results, while five holes (23%) had higher re-assays and three (13%) were inconclusive. When results of the old holes were compared with those of new holes drilled in the same locations, the results were similar to the re-sampling tests. Archer Cathro surmised that the higher gold values in the new holes were due to a combination of improved drilling techniques that resulted in better core recovery, and advanced laboratory techniques that provided lower analytical detection limits.

The PSG core sampling followed rigorous procedures that were well documented and standardized throughout the drilling programs. In the 1992, 1993 and 1994 programs, exploration targets were sampled by HQ (63.5 mm diameter) core drilling; occasionally this was reduced to NQ (47.6 mm). The boxed core samples were transported by truck less than 5 km to a core logging facility adjacent to the Casino Airstrip for geotechnical logging, sample selection, quality control designation, and sampling by PSG personnel. The average core recovery for all PSG core was 94%, with Hypogene core averaging 96%, Supergene averaging 92% and the Leached Cap (Oxide Gold zone) averaging 89%. Sample intervals were marked on the core by the geologist mainly at 3-metre intervals or at geological contacts. Core intervals were sampled by mechanical splitting. The remaining half core was returned to the boxes and stored in racks at the site. The average 3.0-metre lengthwise half-split sample provided 10 to 15 kg of material, which was transported by charter aircraft (primarily DC-3) directly from the core sampling facility to Whitehorse and then by commercial air freight to Vancouver for delivery to the sample preparation laboratory.

In 2008, all samples were split using a conventional core splitter. In 2009, about 150 samples were split with a core splitter at the beginning of the program. From then on, in 2009 through 2012, all samples were cut with a core saw. All samples were split or cut on site and placed in individually labelled plastic sample bags with the unique sample number selected by the geologist logging the hole. The core samples were split lengthwise with half of the core placed in the sample bag, and the other half returned to the core box. The samples were sent to ALS Chemex Labs in North Vancouver for analysis.

In 2013 no core was sampled, but all other core logging protocols were followed as per 2012.

The 2019 drill program followed the protocols established in 2012. Core was split in half lengthwise with a core saw and half of the core was placed, with a sample tag, in plastic bags with the corresponding sample ID noted on the outside of the bag. The sample bags were sealed with cable ties. Metal tags marking the sample intervals (in metres) and with the sample ID matching the tag book were added at the applicable locations in the core boxes. The remaining half of the core was placed back into the core box, stacked outside the core cutting shack and then moved to the core storage yard where each hole was stored either in stacks, securely covered by tarps and labelled as per hole, or directly within the core racks. Bagged and labelled samples were then placed in larger white rice bags, each labelled with a unique batch letter and the address of the receiving lab and sealed with a cable tie. A running list of each batch was maintained in Excel spreadsheet form, including the samples per bag and the dates they were sent out by plane to ALS Global in Whitehorse.

In 2020, the core was sawn lengthwise utilizing rock saws for an even cut, so that half of the core was sent to the laboratory, and the other half remained in the core box. The sample was placed in a pre-labelled poly bag, with a tag having a unique assay number placed within it, and the same number written with a "Sharpie" indelible marker on the bag. All samples were sealed with a cable tie ("Zap Strap"), and placed into rice bags, with the sample numbers, address of the receiving lab, and contact information for written on the bags. Each rice bag was sealed with a

specifically numbered security tag. Shipments comprised 20 rice bags unless it was for the final series of bags beyond an even multiple of 20. A Sample Shipment Form listing each bag number (per batch), its weight (in lbs.), number of samples and sample IDs within each rice bag, and total shipment weight was included in Bag 20. Samples were shipped in batches of 20 by fixed wing aircraft, ensuring that no partial batches were shipped. The samples were flown to the Alkan base at the Whitehorse airport, and picked up directly by Small's Expediting of Whitehorse, Yukon, which delivered them to the ALS Geochemistry lab in Whitehorse, Yukon. The ALS lab performed both sample preparation and analysis.

In 2021, a core sampling regime was instituted, depending on which of the four drill campaigns (Resource confirmation, Metallurgical, Geotechnical or Exploration) the hole belonged to. Resource ("RES") holes, of HTW size, underwent lengthwise splitting by core saw, with one half sent for "super trace element" ICP-MS analysis and for 30-gram fire assay analysis, and the other half sent for metallurgical testing. The exception was for duplicate samples (one sample per 30 regular stream samples), in which the remaining half-core split was sent for the same analysis as the main sample stream. The metallurgical ("MET") samples, of PQ size, were sawn in half lengthwise, with one half sent for metallurgical analysis. The remaining half was again sawn lengthwise, resulting in "quartered" core. One quarter was sent for regular ICP-ES and fire assay analysis, and the other quarter remained in the core box, which was stored in covered core racks on site. Again, duplicate sampling involved removal of the final quarter core for the interval sampled. Geotechnical samples were sawn lengthwise, with one-half sent for analysis, and the other half retained in the core box. Duplicate geotechnical samples involved the entire remaining half-core for the sample interval. Exploration samples were treated the same as geotechnical samples, except that duplicate samples were of "quarter core" rather than "half core." Maximum sample length was 3.0 m for all samples, except 2.0 m for metallurgical samples; minimum length was 1.0 m for all drilling campaigns. All core boxes containing remaining resource, geotechnical and exploration drillhole core are stored on-site in covered core racks. Table 11-1 lists the various drilling and assaying protocols per drill campaign.

Table 11-1: Casino 2021 Drill-Hole Requirements by Campaign

Hole Type	Resource	Metallurgical	Geotechnical	Exploration
No. of holes	5	3	8	6
Total Metres	1,483	1,001	1,957	1,634
Core Diameter	HTW	PQ	HTW	HTW/NTW
Assay Samples	1/2 Core	1/4 Core	1/2 Core	1/2 Core
Assay Duplicates	1/2 Core	1/4 Core	1/2 Core	1/4 Core
Met Samples	1/2 Core	1/2 Core	None	None
Min. Sample Interval	1.0 m	1.0 m	1.0 m	1.0 m
Max. Sample Interval	3.0 m	2.0 m	3.0 m	3.0 m

The chain of custody protocol for the 2021 resource confirmation, geotechnical, exploration and metallurgical samples selected for regular analysis was essentially the same as for 2020, except that a sample batch comprised an entire hole, rather than 20 rice bags. These samples were flown out by Alkan Air and held in safe storage at the compound of Smalls Expediting until the complete batch was compiled. The batch was then delivered directly to the ALS Geochemistry lab in Whitehorse. However, the half-core samples selected for metallurgical analysis from the "MET" and Resource Confirmation holes remained in the core boxes, which were palletized, covered in shrink-wrap, and loaded as palletized units onto the airplane. These were stored at the Smalls Expediting compound until all samples for all three metallurgical holes were received. These were then transported overland by MYCAN transport to ALS Metallurgy in Kamloops, British Columbia.

In 2008, 422 drill core samples were collected and shipped; in 2009, 3,832 drill core samples were shipped; in 2010, 4,768 drill core samples were shipped; in 2011, 387 drill core samples were shipped; and in 2012, 533 drill core samples were shipped. In 2013, no samples were collected. In 2019, 4,939 core samples were collected, shipped, and analyzed. In 2020, 4,069 samples were collected, shipped, and analyzed. In 2021, 2,295 core samples (excluding QC samples) were collected, shipped, and analyzed. All core is currently stored within covered core racks, which are continually constructed during the field programs.

11.1.3 2021 Soil Sampling

In 2021, a reconnaissance-style B-horizon soil geochemical sampling program on a 200-metre square grid was completed across the property areas located north, east, and south of the Casino deposit. Samples were collected using 1.2-metre-long hand augers, and placed in cloth bags, together with a sample tag with a unique sample number supplied by the lab. This number was also written on both sides of the bag, which was tied with the attached draw string. All samples were described in the field utilizing a printed spreadsheet with the following parameters: Sample name, sample location (UTM NAD 83, Zone 7), surface vegetation, nature, and steepness of terrain, colour, depth of sample, horizon sampled, depth within horizon, moisture content, percent gravel, percent sand, percent silts and clays, percent organics and percent of angular fragments where gravel was encountered. Soil sampling tools were cleaned following completion of each sample, to eliminate potential for contamination. At each site, a photograph of the sampled material was taken.

All samples were brought back to camp, where they were untied and homogenized with a clean spoon prior to undergoing preliminary XRF analysis, to determine whether any sampled areas warrant immediate follow-up exploration. Following this, samples were sealed with a cable tie, and placed in rice bags labelled with the sample ID's per bag, name, and address of receiving lab, and name of the provider. Each rice bag contained 25 samples, and each was sealed with a security tag having a unique ID number. Samples were shipped in batches of 12 rice bags per batch, labelled with the batch ID and the bag number per batch. Soil shipment data were recorded in a spreadsheet and added to the final bag in each batch, and a copy was kept on file. Soil samples were shipped out in complete batches by Alkan Air to their Whitehorse base of operations, where they were picked up by Smalls Expediting personnel and delivered directly to the ALS Geochemistry lab in Whitehorse. A total of 2,502 samples, excluding QC samples, were taken in 2021.

11.1.4 Enersoft "GeologicAI" Scanning of Core

Although no samples were taken during this process, the core movement and preparation protocol warrants description. A tent, open on one side, with several core tables, was set up adjacent to one of the core logging shacks. A schedule of drill holes selected for scanning was established. Core boxes, in order from top to bottom of each hole, were transported from the core racks to the layout tables. Here, the core was cleared of leaves and small organic debris and cleaned using large spray bottles. Following this, the core was moved into the core shack in final preparation for actual scanning, including the laying flat of meterage blocks (face up) to identify depths. If excess core boxes were moved from the core racks to the tent, they were stacked in sequential order prior to moving into the tent.

The GeologicAI apparatus scanned two core boxes simultaneously, requiring about 12 minutes to do so. Scanning comprised chemical analysis by XRF, hyperspectral analysis to map alteration and mineral assemblages, including sulphide mineralization, LIDAR (Light Detection and Ranging) textural mapping, and high-resolution photography, used for, but not limited to, RQD determination. Following scanning, the core was stacked on a separate layout area, in preparation for return to the core racks.

The entire extent of selected holes underwent scanning, including all 2021 holes. A total of 48,673 m of core, representing slightly over 40% of the total amount, were scanned in 2021.

11.2 SAMPLE PREPARATION

11.2.1 2019 Sample Preparation

All original samples in 2019 were sent to ALS Global Labs in Whitehorse for analysis. The standard analytical request for all samples was for preparation by procedure Prep-31A. This process involved logging the sample into the tracking system, weighing, drying, and crushing the entire sample so that more than 70% passed through a 2 mm screen. A 250-gram split of the crushed material was then collected by riffle splitter and pulverized so that more than 85% passed through a 75-micron (µm) screen. The resultant pulp was analyzed by the ALS lab in Whitehorse.

Sample "standards," provided by CDN Resource Labs and inserted in the sample stream at site, arrived at ALS Global in pulp form and went straight to analysis. Blank samples inserted into the sample stream at site arrived as rock and went through the same preparation and analytical processes as the core samples. Duplicate samples were sent to ALS in separate batches, arriving at a later date than the original samples. These also underwent the same preparation and analytical processes as the original core samples.

Check pulp samples were sent from ALS in Whitehorse to SGS Canada Inc. (SGS) in Burnaby, British Columbia (BC). At SGS, the pulps were checked for weight and fineness before a full geochemical assay was run. This involved logging the sample into the tracking system (confirming the samples received matched the electronic list of samples sent by Western staff), weighing, and then checking that the pulps were of appropriate fineness.

11.2.2 2020 Sample Preparation

The standard analytical request for all samples was for preparation by procedure Prep-31A. This process involved logging the sample into the tracking system, weighing, drying, and crushing the entire sample so that more than 70% of the material could pass through a 2 mm screen. A 250-gram split of the crushed material was then collected by riffle splitter and was pulverized so that a minimum of 85% of the material could pass through a 75-micron screen. The resultant pulp was then sent for analysis within the ALS lab in Whitehorse.

"Standard" reference material (SRM), provided by CDN Resource Labs and inserted in the sample stream at site, arrived at ALS Global in pulp form and went straight to analysis. Blank samples, comprising dolostone available at gardening centres, arrived as rock and went through the same process as the core samples. Duplicate samples were sent to ALS in separate batches, arriving at a later date than the original samples and then undergoing the same process as the original core samples.

"Check" pulp samples were sent from the Whitehorse ALS lab to the Bureau Veritas Minerals lab in Vancouver, BC. At Bureau Veritas, the pulps were checked for weight and fineness before a full geochemical assay was run. This involved logging the sample into the tracking system and checking to confirm that the samples received matched the electronic list of samples sent by Western Copper staff. Samples were then weighed and checked to ensure the pulp was of appropriate fineness.

11.2.3 2021 Sample Preparation

The standard analytical request for non-metallurgical 2021 resource confirmation, geotechnical and exploration core was essentially the same as that for 2020 samples. This process involved logging the sample into the tracking system, weighing, drying, and crushing the entire sample so that more than 70% of the material could pass through a 2 mm screen. A 250-gram split of the crushed material was then collected by riffle splitter and was pulverized so that a minimum of 85% of the material could pass through a 75-micron screen. The resultant pulp was then sent for analysis within the ALS lab in Whitehorse.

"Standard" reference material, provided by OREAS North America Inc. and inserted in the sample stream at site, arrived at ALS Global in pulp form and went straight to analysis. Blank samples, comprising dolostone available at gardening centres, arrived as rock and went through the same process as the core samples. However, in 2021 duplicate samples were included in the main sample stream, rather than as a separate batch.

The protocol for "check" pulp samples was the same as per the 2020 program.

11.3 ASSAY ANALYSIS

11.3.1 Assay Analysis, 1992-1994

The analytical procedures utilized prior to 1992 are unknown. All 1992 - 1994 regular, main sample-stream samples, 1992 - 1993 selected duplicate samples and 1994 random half-core duplicate samples were analyzed by Chemex Labs. Immediately prior to selecting each pulp's analytical aliquot, each pulp sample was passed through a 20-mesh screen to eliminate lumps of agglomerated clay minerals. Gold (Au) was analyzed by fire assay with atomic absorption finish. Silver (Ag) values were reported in g/t and Cu and MoS_2 values were reported as percentages. Chemex also performed 32-element ICP analysis for: Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sc, Sr, Ti, Tl, U, V, W and Zn. Mineral Environments (Min-En) Laboratories, of North Vancouver, BC, analyzed the selected duplicate samples from 1992 and 1993, and random duplicate samples from 1994. Gold was analyzed by fire assay and reported in g/t. Values for Cu and MoS_2 were reported as percentages.

11.3.2 Assay Analysis, 2008-2020

11.3.2.1 Gold Analysis

Samples for the 2008 – 2012 and 2019 programs at ALS Global were analyzed for gold, using a 30-gram sample of the pulp, by fire assay and atomic absorption (AA) finish (Analytical code Au-AA23) to a 0.005 ppm detection limit. Results were reported in parts per million (ppm). At SGS gold assays were run by using a 30-gram sample of the pulp with fire assay and AAS finish to a 5-ppb detection limit, according to procedure GE_FAA30V5. Results were reported in parts per billion (ppb). Note that 5 ppb = 0.005 ppm, equivalent to g/tonne.

In 2020, following preparation of the sample at the Whitehorse ALS lab, a full ICP suite was run on the resultant pulps at the Vancouver ALS lab using four-acid digestion with Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES) finish. Gold analyses were run using a 30-gram sub-sample by fire assay and atomic absorption spectroscopy (AAS) finish (procedure code Au-AA23) to a 0.005 ppm detection limit. Results were reported in parts per million (ppm), equivalent to grams/tonne (g/t). Ore grade analysis for gold was run by fire assay and gravimetric finish. At Bureau Veritas, where the check assays were done, gold assays were run using a 30-gram sample of the pulp with fire assay and AAS finish to a 0.005 ppm detection limit according to procedure FA530. Results were reported in parts per million (ppm).

11.3.2.2 Copper, Molybdenum and Silver Assay

Samples that returned over-limit values for copper, molybdenum, or silver in the ICP analysis were assayed by analytical process OG62 at ALS Global. This process involved four-acid digestion and analysis by ICP-AES or Inductively Coupled Plasma-Atomic Absorption Spectroscopy (ICP-AAS). Results were reported in percent (%). At SGS a similar process was followed for any over-limit values for copper, molybdenum or silver involving sodium peroxide fusion with ICP-AES, using analytical method GO ICP90Q.

These analytical processes were employed by Western in 2019, as well as from 2008 through 2012. In 2020, the same analytical processes were employed at ALS Global, and a similar process was employed at Bureau Veritas for any

over-limit results for copper, molybdenum, or silver. This process at Bureau Veritas comprised a four-acid digestion (0.5 g/ 200 ml) with AAS finish using the MA404 method. Results were reported in percent (%).

11.3.2.3 ICP Analysis

Samples sent to ALS Global from 2008 - 2012 and 2019 were analyzed for multiple elements, including copper, molybdenum, and silver by analytical process ME-ICP61. This process involved a four acid "Near Total" digestion of 1.0 grams of sample pulp with Mass Emission-Inductively Coupled Plasma Spectroscopy (ICP-MS) for the analysis. This process returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), Ga (ppm), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Ni (ppm), P (ppm), Pb (ppm), S (%), Sb (ppm), Sc (ppm), Sr (ppm), Th (ppm), Ti (%), Tl (ppm), U (ppm), V (ppm), W (ppm), and Zn (ppm).

Samples sent to SGS were analyzed for 56 elements, including copper, molybdenum, and silver, by analytical method GO_ICP90Q100. This process involved a mineralized material grade sodium peroxide fusion with ICP-AES. This process returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Ce (ppm), Co (ppm), Cr (%), Cs (ppm), Cu (ppm), Dy (ppm), Er (ppm), Eu (ppm), Fe (%), Ga (ppm), Gd (ppm), Ge (ppm), Hf (ppm), Ho (ppm), In (ppm), K (%), La (ppm), Li (%), Lu (ppm), Mg (%), Mn (ppm), Mo (ppm), Nb (ppm), Nd (ppm), Ni (ppm), P (%), Pb (ppm), Pr (ppm), Rb (ppm), Sb (ppm), Sc (ppm), Si (%), Sm (ppm), Sn (ppm), Sr (ppm), Ta (ppm), Tb (ppm), Th (ppm), Ti (%), Tl (ppm), Tm (ppm), U (ppm), V (ppm), W (ppm), Y (ppm), Yb (ppm), Zn (ppm), and Zr (ppm).

In 2020, samples sent to ALS Global also underwent the same analytical processes. However, "check" pulp samples were sent to the Bureau Veritas Commodities lab in Vancouver, British Columbia, where they underwent analysis for 35 elements, including copper, molybdenum, and silver by analytical method MA300. This process involved trace analysis using a multi-acid digestion and ICP-ES finish, providing results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Nb (ppm), Ni (ppm), P (%), Pb (ppm), S (%), Sb (ppm), Sc (ppm), Sn (ppm), Sr (ppm), Th (ppm), Ti (%), U (ppm), V (ppm), W (ppm), Y (ppm), Zn (ppm), and Zr (ppm).

11.3.2.4 Acid-soluble Copper Analysis

In 2008 and 2009, following receipt of the copper analyses, samples were selected for "non-sulphide" or "acid soluble" copper analysis. The criterion for "non-sulphide" selection was any sample that contained >100 ppm (0.0100%) Cu in the Leached Cap, Supergene Zone, or top 50 m of the Hypogene Zone. A list of these samples was presented to ALS Chemex, which then retrieved the pulps and analyzed them by 5% sulphuric acid leach and AAS finish (procedure Cu-AA05).

In 2010 to 2012, selected samples for "acid soluble" copper analysis were identified by the core logging geologist and the request for this analysis was submitted when the samples were originally sent to the lab. The samples identified by the geologist were generally located from the top of the hole to the top 50 m of the hypogene zone. On a few occasions, after receiving the geochemical results, additional samples were identified for "non-sulphide" copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

In 2019, when the initial ICP-MS assays were returned from ALS Global on the original samples, another group of sample pulps were sent for further assay by procedure code Cu-AA05 to identify non-sulphide copper. All pulps returning > 100 ppm copper within the Leached Cap, Supergene Oxide, Supergene Sulphide, and the initial 50 m of the Hypogene zone were pulled by ALS Global for this analysis. The processes for 2020 and 2021 were the same. The pulps were assayed by analytical method Cu-AA05a (3% sulfuric acid leach and AAS) to test for non-sulphide copper.

11.3.2.5 Cyanide Soluble Copper Analysis

In 2010, a large number of samples from the 2008, 2009 and 2010 programs were identified for cyanide-soluble copper analyses. These samples were selected to aid with identification of the Supergene Sulphide - Hypogene metallurgical boundary. The selected samples were analyzed by cyanide leach with AAS finish (ALS Chemex procedure Cu-AA17a). For samples that had already been received and processed at the lab, ALS Chemex retrieved the pulps and analyzed this material. For samples not yet sent to the lab, the geologist identified the Supergene Sulphide - Hypogene boundary visually, and samples 30 m on either side of the boundary were identified for cyanide leach copper analysis. On a few occasions, after receiving the geochemical results, additional samples were identified for cyanide soluble copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

In 2019, the senior geologist used the core logging results to choose samples for cyanide-soluble copper analysis using method Cu-AA17a at ALS Global; all sample pulps from 30 m on either side of the Supergene Sulphide - Hypogene boundary were sent for this type of assay. The process utilized in 2020 was the same, using analytical method Cu-AA17, which was identical to method Cu-AA17a for 2019.

In 2021, the senior geologist used the core logging results to choose samples for cyanide-soluble copper analysis using method Cu-AA17a at ALS Global. These samples were selected to aid with identification of the Supergene Sulphide - Hypogene metallurgical boundary. The geologist identified the Supergene Sulphide - Hypogene boundary visually, and samples 30 m on either side of the boundary were identified for cyanide leach copper analysis. For samples that had already been received and processed at the lab, ALS Global retrieved the pulps and analyzed this material. On a few occasions, after receiving the geochemical results, additional samples were identified for cyanide soluble copper analyses and ALS Global was requested to pull these sample pulps and perform the analysis. The selected samples were analyzed by cyanide leach with AAS finish (Procedure Cu-AA17a).

The Vancouver facility of ALS Minerals has been assessed by the Standards Council of Canada (SCC) and was determined to conform to requirements of ISO/IEC 17025:2017, and therefore recognized as an Accredited Testing Laboratory. Similarly, Bureau Veritas Metals, Minerals and Environmental also meets the requirements of ISO/IEC 17025 and ISO 9001. The Vancouver lab of SGS Canada also has ISO/IEC 17025 accreditation, and the Delta, BC lab has ISO 9001 accreditation.

11.3.3 Assay Analysis 2021

Analytical Techniques in 2021 varied somewhat, depending on the drilling campaign (Resource Confirmation, Metallurgical, Geotechnical or Exploration) employed. All samples were submitted to the ALS Geochemistry prep lab in Whitehorse, and the pulps were then sent to their analytical lab in North Vancouver.

The procedures described here are for the analytical “regular samples” of Resource Confirmation holes DDH21-01 through DDH21-03, and DDH21-05. All samples underwent four-acid digestion followed by “super trace element” ICP-MS analysis of a 0.25-gram sample for Ag, Al, As. Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, In, K, la, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn, and Zr. Also, a 30-gram sample underwent fire assay analysis with “Inductively coupled plasma atomic emission spectroscopy” (ICP-AES) finish, providing a detection range of 0.001 g/t to 10.0 g/t Au. Overlimit values for Cu and Mo underwent further analysis using a 0.4-gram sample undergoing analytical procedure OG62. Overlimit values for Au and Ag underwent reanalysis of a 30-gram sample by fire assay with gravimetric finish, using analytical codes Au-GRA 21 (detection range: 0.05 - 10,000 ppm) and Ag-GRA21 (detection range: 5 – 10,000 ppm) respectively. All samples also underwent XRF analysis (analytical code: pXRF-30RT) for Cr, Nb, Si, Ta, Ti, Y, and Zr. A 0.1-gram subsample of each also underwent analysis for total carbon (analytical code: C-IR07), with an analytical range of 0.01% to 50.0%.

The analytical protocol for the geochemical analytical portions of the Metallurgical holes (DDH21-04, 21-06 and 2-07), Geotechnical holes (DDH 21-09 through DDH 21-11, DDH21-13, 21-15, 21-20 and 21-22) and the final Resource Confirmation hole (DDH21-08) was the same as for the other Resource Confirmation holes, except that the XRF analysis for Cr, Nb, Si, Ta, Ti, Y, Zr (code: pXRF-30RT) was excluded.

The analytical procedure for the Exploration holes (DDH 21-12, 21-14, 21-16 through 21-19 and 21-21) was the same as above, except that analysis for total carbon (code: C-IR07) was also excluded.

11.4 SECURITY

During the pre-1992 drilling campaigns at Casino the rigours of "chain of custody" were not as stringent as presently required. The remoteness of the Casino site provided a large degree of security as air traffic into the project was closely monitored. Further, the Casino gold grades were low and any metal contamination or grade enhancement would be quickly and easily identified.

Adequate sample handling procedures were in place during the 1992 - 1994 PSG programs. Geologists supervised the sampling process and the samples were kept in a secure impoundment prior to shipping. The best vigilance on the samples was the attention to results, and in that regard, PSG maintained a thorough QA/QC program. Samples were shipped in rice bags with uniquely numbered, non-re-sealable security tags. Each sample shipment was transported from the Casino Property via air to Whitehorse. The samples were received at the airport by the project expediter and shipped to the appropriate lab from there. In 2008 and early 2009, all shipments were sent by Byers Transport to the ALS Chemex lab in North Vancouver. Later in 2009 and early 2010, samples for ALS Chemex were shipped by Byers Transport to the ALS Chemex preparation facility in Terrace, BC, where they were crushed and pulverized. The pulps were then shipped by ALS Chemex to North Vancouver for analysis. In May of 2010, ALS Chemex opened a preparation facility in Whitehorse. From then on, all samples were delivered to the Whitehorse preparation lab by the project expediter. The samples were logged in, crushed, and pulverized in Whitehorse and the pulps were shipped to North Vancouver for analysis.

By 2019, ALS Chemex had changed its name to ALS Global, and installed an analytical lab in Whitehorse so that samples could be both prepared and analyzed there. This eliminated the problems that can occur with transport of pulps. Samples were shipped from the Casino site by Alkan Air to their base in Whitehorse where Small's Expediting of Whitehorse, Yukon picked them up upon arrival and delivered them directly to the Whitehorse lab of ALS Global. Rice bags were organized in batches of 20, with unique identifiers written on each bag and sealed with a uniquely numbered non-resealable security strap. Each 20th bag contained the sample submittal form and a list of all the samples that should be included in that particular batch. Upon receipt, ALS would confirm via email with the project manager/senior geologist the exact samples received.

The same security regimen was in place in 2020. Upon landing at the Alkan airbase, the samples were picked up by Small's Expediting of Whitehorse, Yukon, and driven to the ALS lab in Whitehorse.

The same security regimen was in place in 2021 as in previous campaigns for regular stream geochemical samples, except that each sample batch comprised all samples for a particular hole. Rice bags were labelled with unique identifiers written on each bag and sealed with a uniquely numbered non-resealable security strap. The first bag of each shipment contained the sample submittal form and a list of all the samples that should be included in that particular batch. Upon landing at the Alkan airbase, the samples were picked up by Small's Expediting of Whitehorse, Yukon, and driven to the ALS lab in Whitehorse. Upon receipt, ALS would confirm via email with the project manager/senior geologist the exact samples received.

If a shipment was received with a broken security tag, the lab would notify the project manager to determine if the shipment had been tampered with, or if the tag was accidentally damaged during shipping. Any broken sample bags

were also brought to the attention of the project manager. In 2020 and 2021, the project manager was not notified of any broken security tags or sample bags.

The security regime for the 2021 Metallurgical samples and the half-core Resource Confirmation samples destined for metallurgical testing varied somewhat from the regular stream samples. These samples remained in place in the core boxes, which were palletized, strapped, and wrapped in plastic "shrink-wrap" to prevent tampering. The pallets were loaded onto the airplane, flown to the Alkan facilities in Whitehorse where they were picked up by Smalls Expediting. The pallets were held in secure facilities owned by Smalls Expediting until the entire shipment was received. The shipment was transported by MYCAN Transport to the ALS Metallurgy lab in Kamloops, British Columbia.

11.5 QUALITY ASSURANCE (QA) AND QUALITY CONTROL (QC)

Exploration sampling and analysis prior to 1992 were not subjected to the rigors required of modern regulatory requirements, but work conducted by major companies, such as Quintana and Teck Corporation, followed industry standard best practices of the time.

However, details of the sampling and analytical methodology are unknown. Moreover, analytical quality, particularly with respect to the determination of gold in the sub-1.0 g/t range, has improved considerably since the pre-1992 work was done. It is for these reasons that the assay results from these old holes were not used in this study.

During the 1993 and 1994 Pacific Sentinel Gold drilling programs, standards, reject duplicates, and half-core replicates were assayed at regular intervals in order to check the security of the samples, as well as the quality and accuracy of the laboratory analyses. Further, in-house laboratory standards, duplicates and blanks were also run and reported as normal assays on certificates.

Figure 11-1 is a flow-chart showing the processing of drill core and quality control procedures for the 1992 and 1993 programs, and Figure 11-2 is a similar chart for the 1994 program.

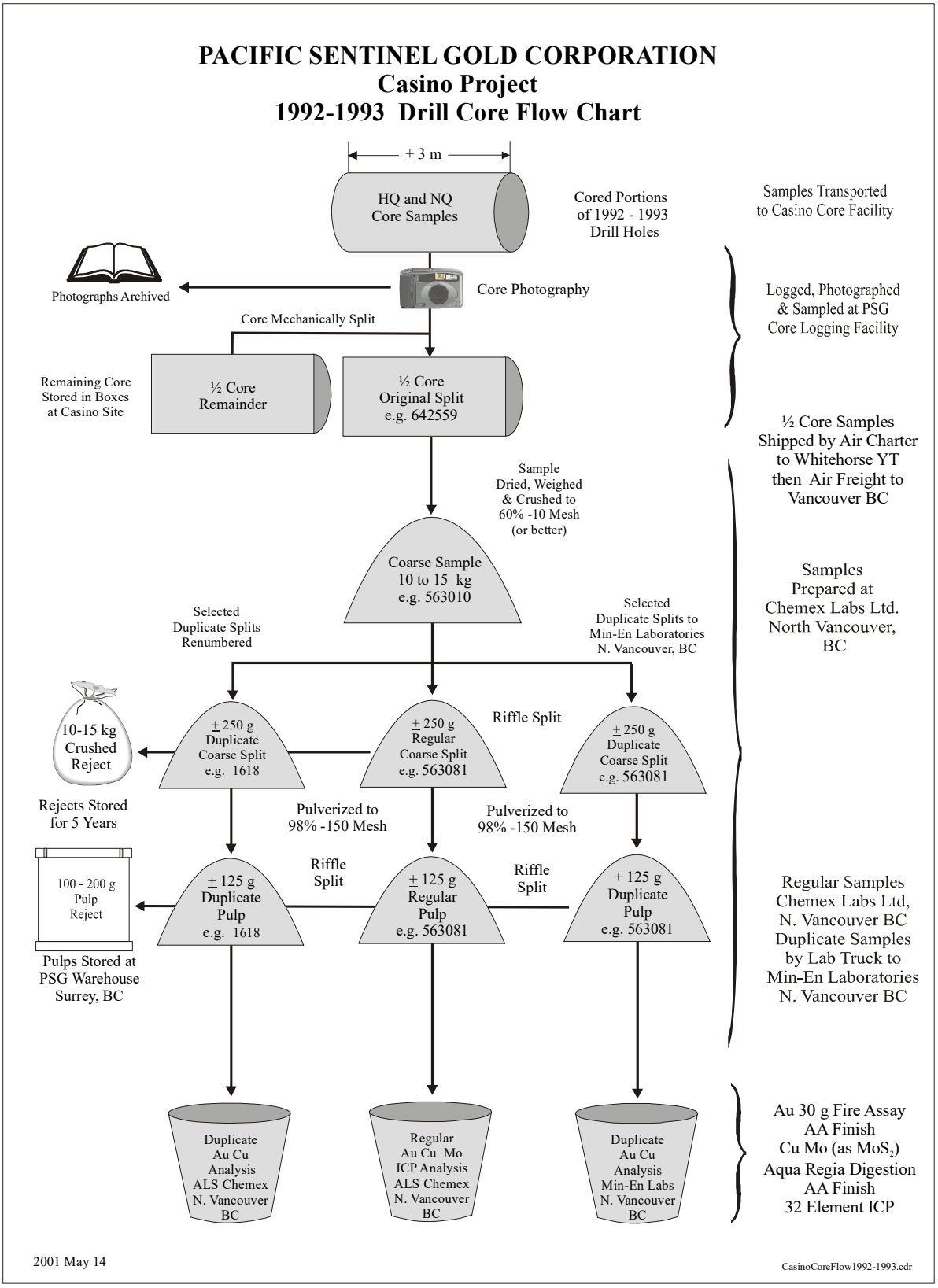


Figure 11-1: Flow-cart for drill core processing and quality control procedures, 1992 and 1993 programs

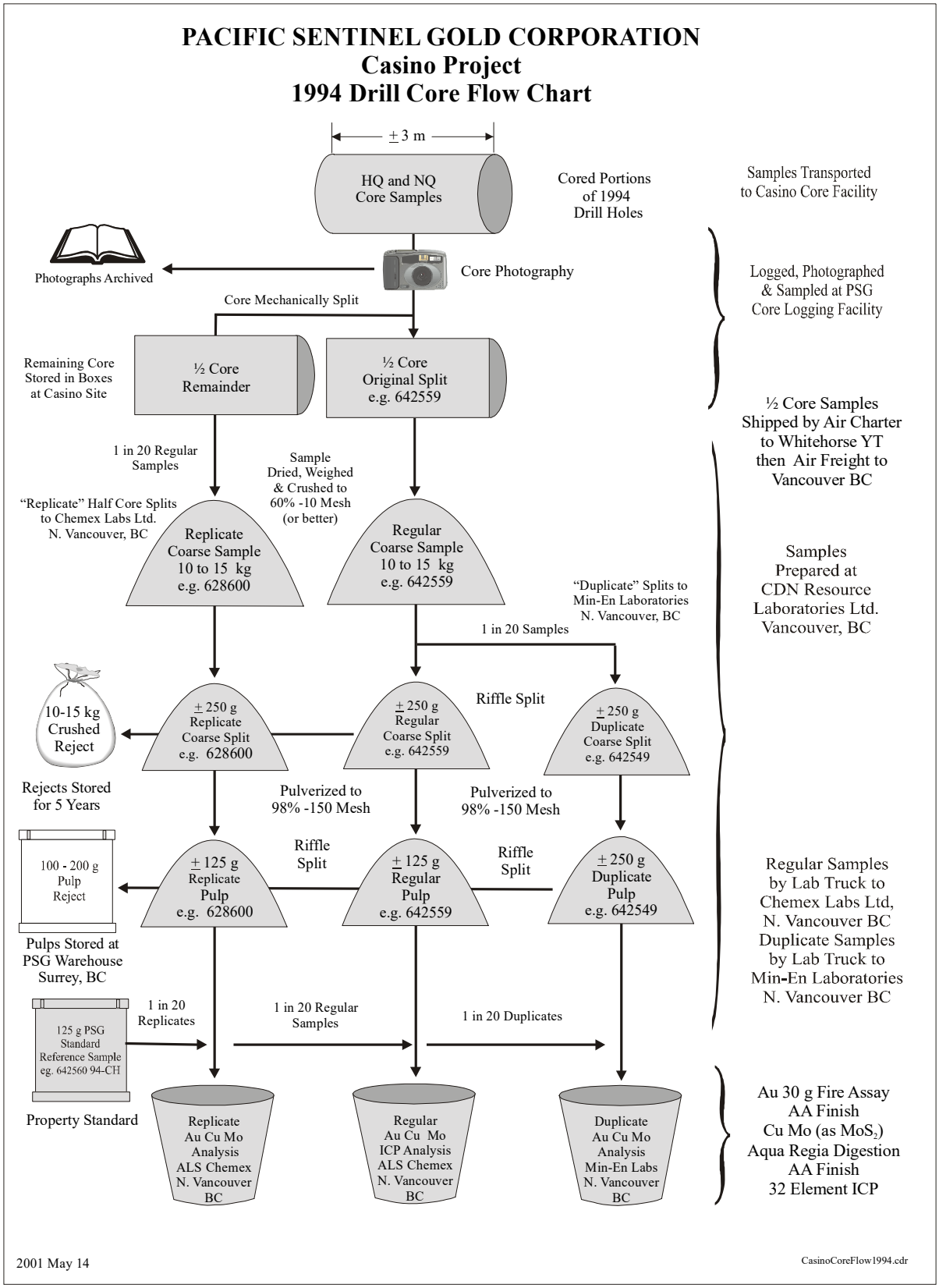


Figure 11-2: Casino Drill Core Processing and Quality Control Procedures for 1994

During the 2008 through 2012 drilling programs, reference material "standards" of known metal content, "blanks", with background metal values, and half-core duplicates were assayed at regular intervals in order to check the security of the samples, as well as the quality and accuracy of the laboratory analyses. The standards and blanks were prepared by CDN Resource Laboratories Ltd. of Delta, BC.

In 2019 and 2020, standards, quarter-core duplicates and blanks were assayed at regular intervals within the sample stream by the primary lab, ALS Global. One of each (standard, blank, duplicate) were randomly inserted within every 20 core samples. The standards were prepared by WCM Minerals in Burnaby, BC. Blank material was comprised of dolomite pebbles commonly sold as garden stone. The same protocol was followed in 2021, for all samples undergoing regular fire assay and ICP analyses.

11.5.1 Reference Material "Standards."

11.5.1.1 Reference Material "Standards," 2008 through 2010

The standard samples used in 2008, 2009 and 2010 were prepared by CDN Resource Laboratories Ltd. of Delta, BC. The "Standard Reference Material" ("SRM" or "Standard") was a gold-copper-molybdenum standard, CDN-CM-4. It was certified by Duncan Sanderson, Licensed BC Assayer with independent certification by Dr. Barry Smee, Ph.D., geochemist. Round-robin assaying for the standard was performed at 12 independent laboratories. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of the standard values as:

- Gold: 1.18 + 0.12 g/t
- Copper: 0.508 + 0.025 %
- Molybdenum: 0.032 + 0.004 %

In 2008, 8 "standard" samples were submitted at regular intervals with the sample shipments; in 2009, 81 standard samples were submitted; and in 2010, 86 standard samples were submitted (approximately 1 per 50 core samples). ALS Chemex analyzed the standards along with the drill core samples by gold, copper, and molybdenum assay, as well as multi-element ICP as described above.

The results from sample standard CDN-CM-4 for 2008, 2009 and 2010, for gold, copper and molybdenum analyses are plotted in Figure 11-3 to Figure 11-5 below.

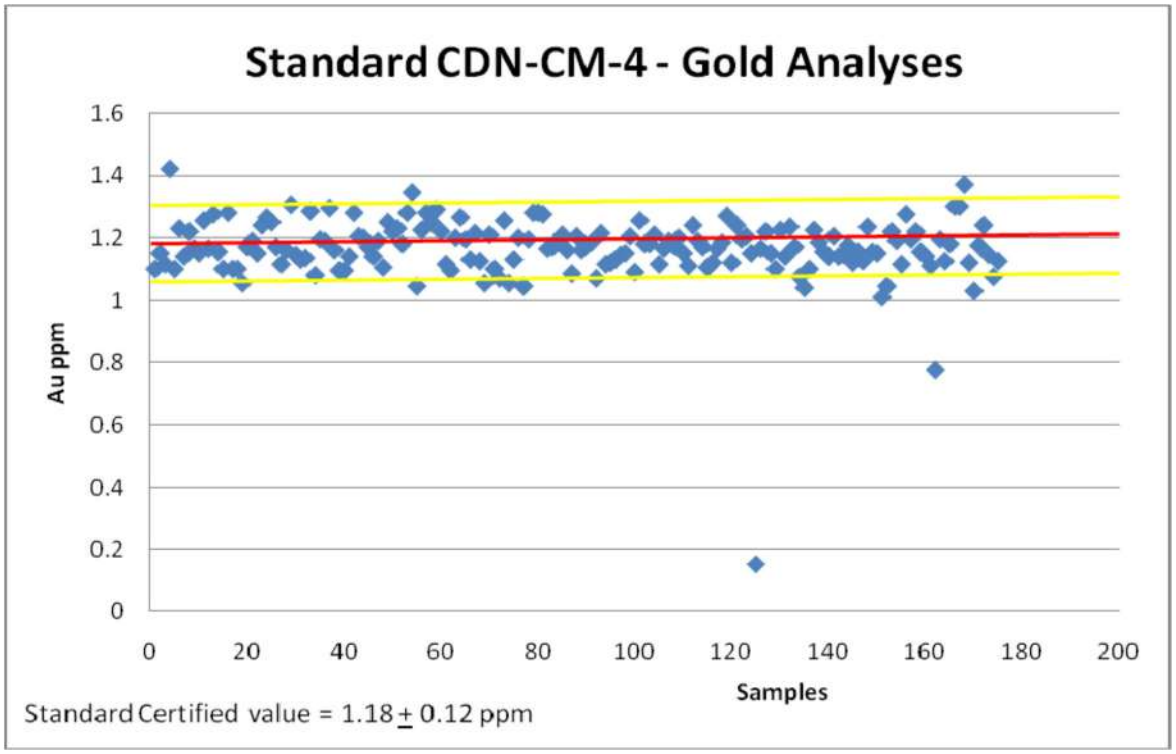


Figure 11-3: Sample Standard CDN-CM-4 Gold Assay Results

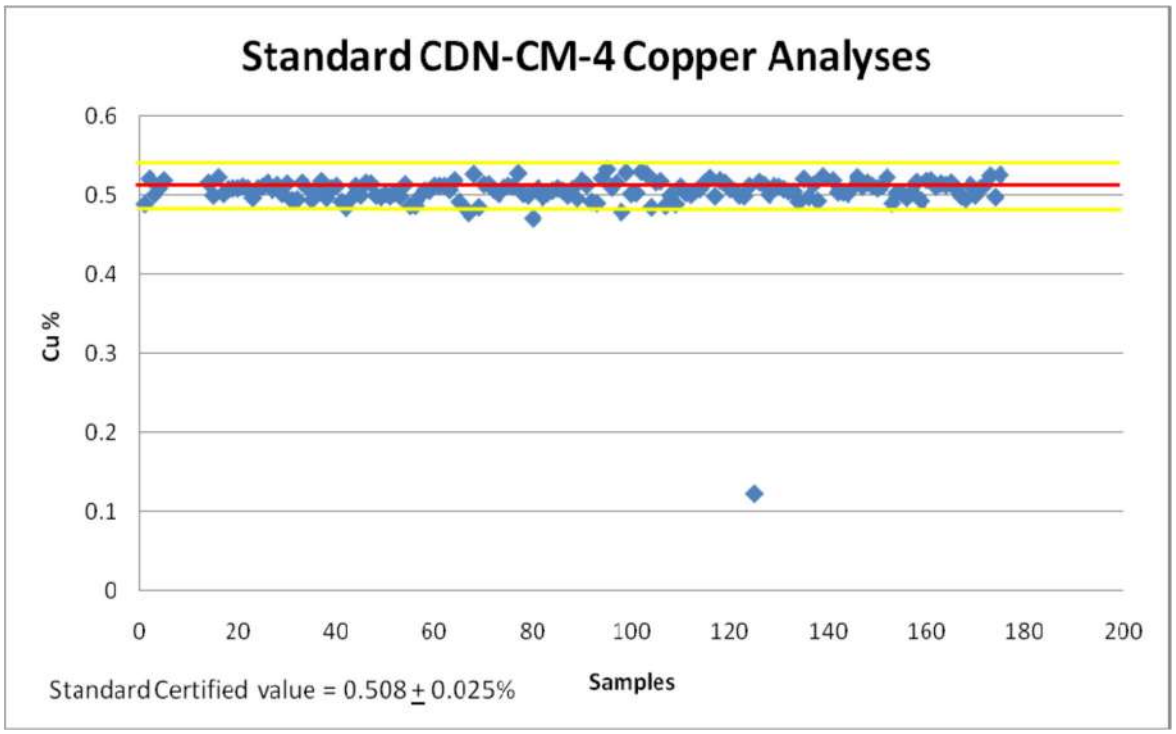


Figure 11-4: Standard Sample CDN-CM-4, Copper Assay Results

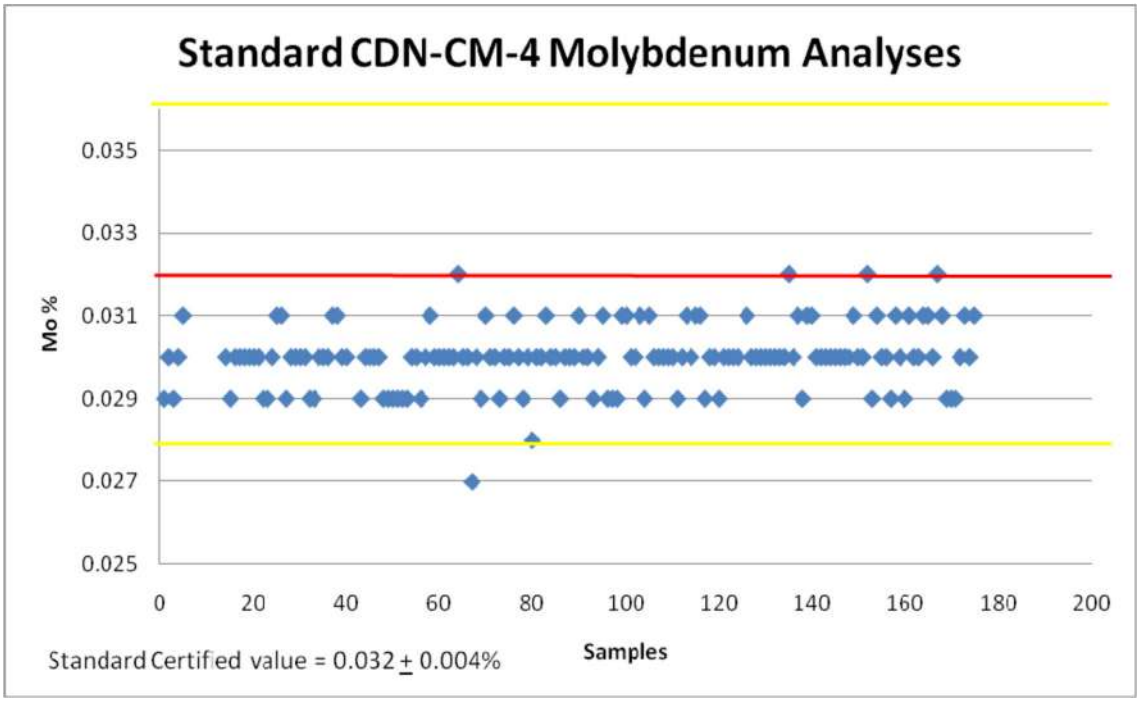


Figure 11-5: Sample Standard CDN-CM-4 Molybdenum Assay Results

These three plots demonstrate that with very few exceptions (9 exceptions for gold, 2 for copper, and one for molybdenum), the values plot within the acceptable range of the certified standard. The plots also demonstrate that there is a reasonable spread of values within the recommended value range of 2 standard deviations as provided by CDN Resource Laboratories Ltd. There does not appear to be any systematic bias.

Later in 2010, a second sample standard (CDN-CM-7) was purchased from CDN Resource Laboratories Ltd. as they had run out of standard CDN-CM-4. This sample is also certified by Duncan Sanderson and Dr. Barry Smee. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of this standard as:

- Gold: 0.427 + 0.042 g/t
- Copper: 0.445 + 0.027 %
- Molybdenum: 0.027 + 0.002 %

Fifteen of these standards were submitted in 2010. ALS Chemex analyzed these standards in the same manner as standard CDN-CM-4, described above.

The results from sample standard CDN-CM-7 for 2010, for gold, copper and molybdenum analyses are plotted in Figure 11-6 through Figure 11-8 below:

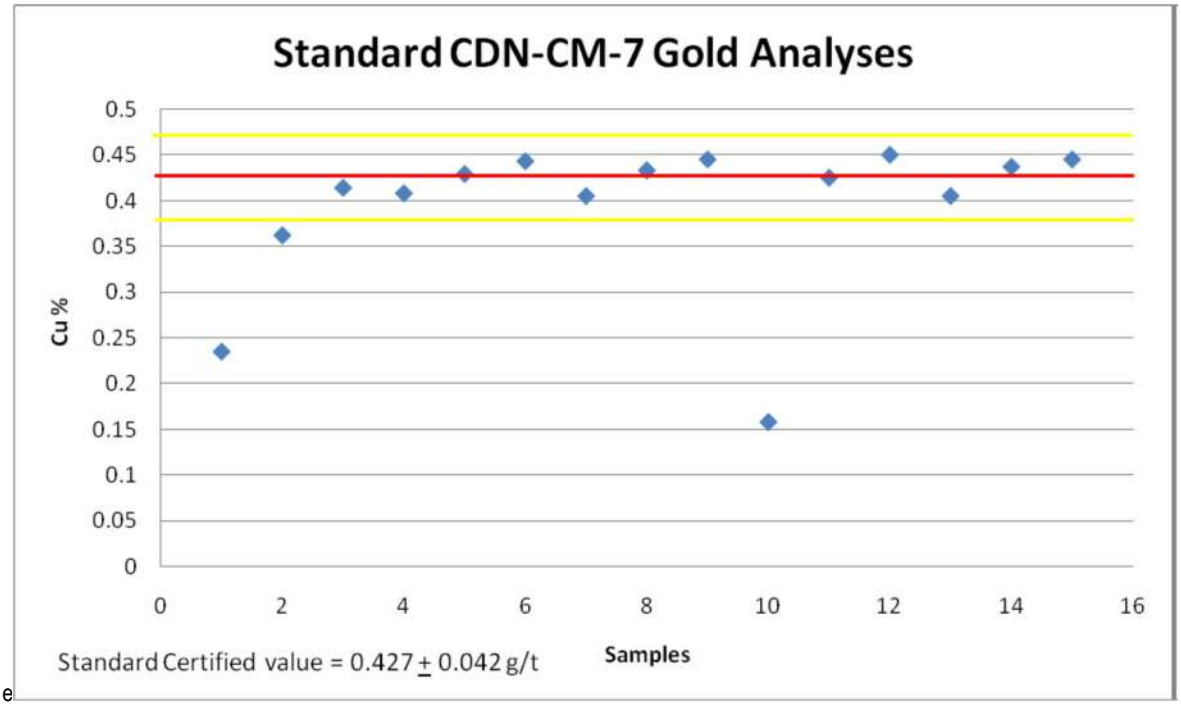


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Figure 11-6: Sample Standard CDN-CM-7 Gold Assay Results

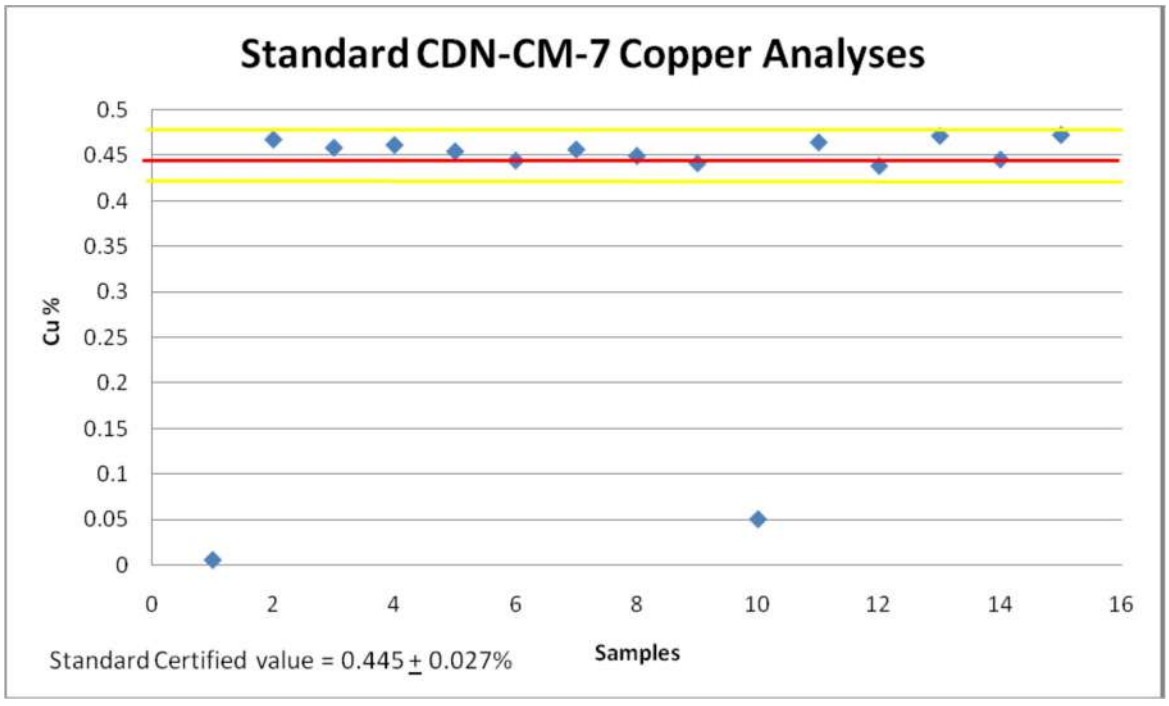


Figure 11-7: Sample Standard CDN-CM-7, Copper Assay Results

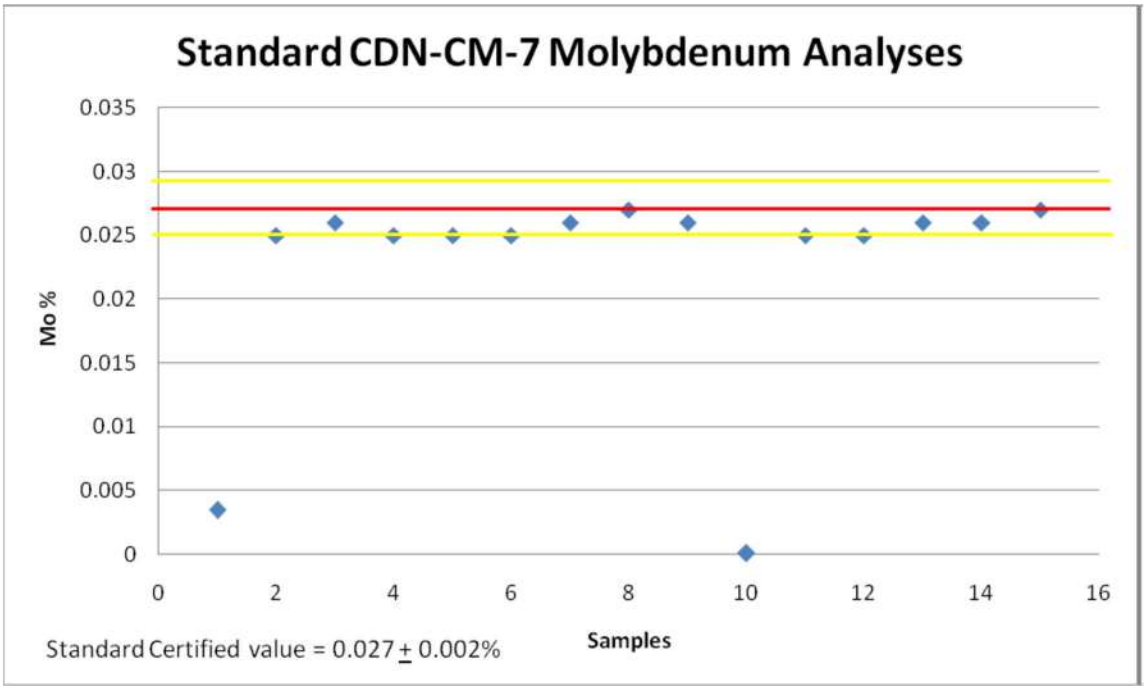


Figure 11-8: Sample Standard CDN-CM-7, Molybdenum Assay Results

The three plots show a good level of accuracy with the exception of samples 1 and 10 which are well below the expected values as certified by CDN. After checking the ALS Chemex internal standards and the duplicates from these batches there does not appear to be a systemic error in the batches. The error may have occurred when the sample standards were inserted in the field, or when the standards were originally placed in the geochemical run at the lab. These anomalous errors are not considered significant, as the great majority of standards were within the expected range.

11.5.1.2 Reference Material Standards, 2019

The SRM material used in 2019 were prepared by WCM Minerals in Burnaby, BC. Details of the standards are outlined in Table 11-2 below. Both standards were certified by Lloyd Twaites and Glen Armanini, who are both Registered Assayers in British Columbia.

Table 11-2: 2019 Standard reference Material from WCM Minerals

Standard	Copper (%)	Standard Deviation Cu	Molybdenum (%)	Standard Deviation Mo	Silver (g/t)	Standard Deviation Ag	Gold (g/t)	Standard Deviation Au
CU-185	0.400	0.0093	0.035	0.0019	15	0.6242	0.62	0.0217
CU-188	0.179	0.0068	0.018	0.0009	15	0.7883	0.4	0.0199

In 2019, 273 Standard samples (1 standard within every 20 samples) were submitted at regular intervals with the sample shipments. These comprised 154 CU-188 and 119 CU-185 standards. ALS Global analyzed the standards along with the drill core samples by gold, copper, and molybdenum assay, as well as multi-element ICP as described above.

The results from sample standards CU-185 and CU-188 for gold, silver, copper, and molybdenum analyses are plotted below in Figure 11-9 through Figure 11-16.

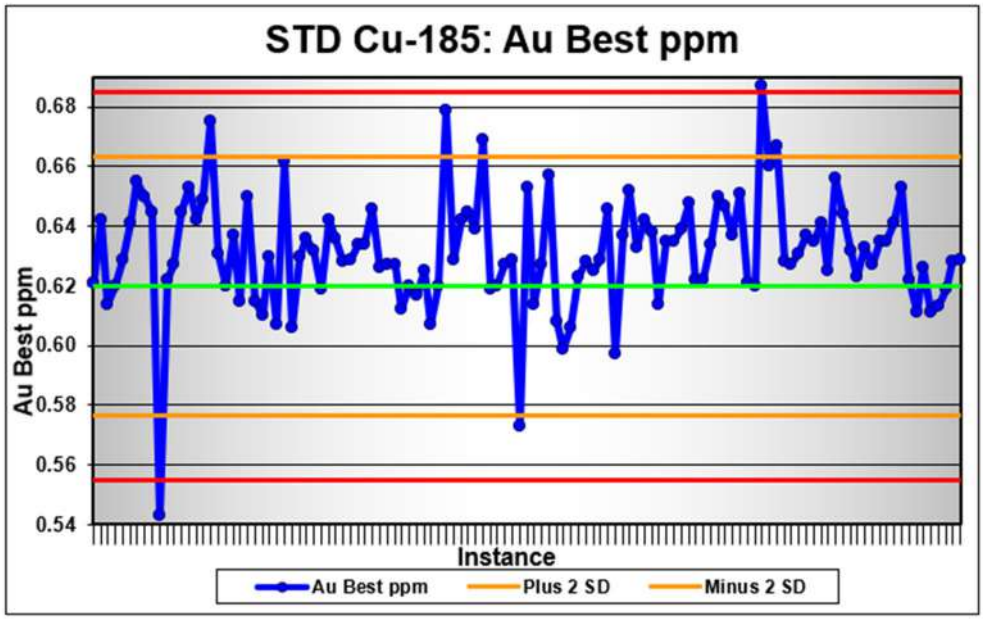


Figure 11-9: Gold Assay Results, Sample Standard CU-185

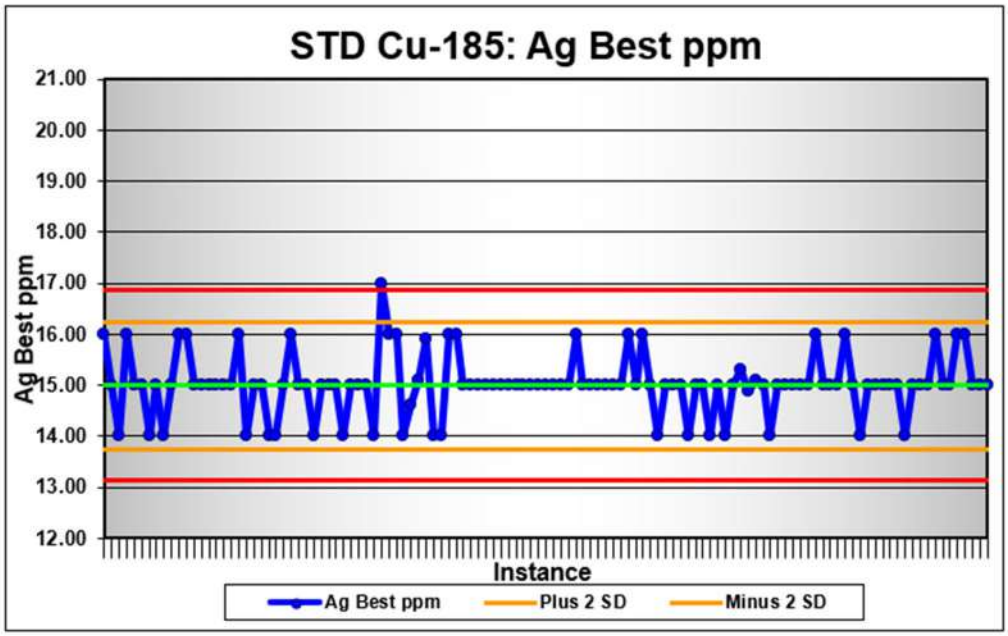


Figure 11-10: Silver Assay results, Sample Standard CU-185

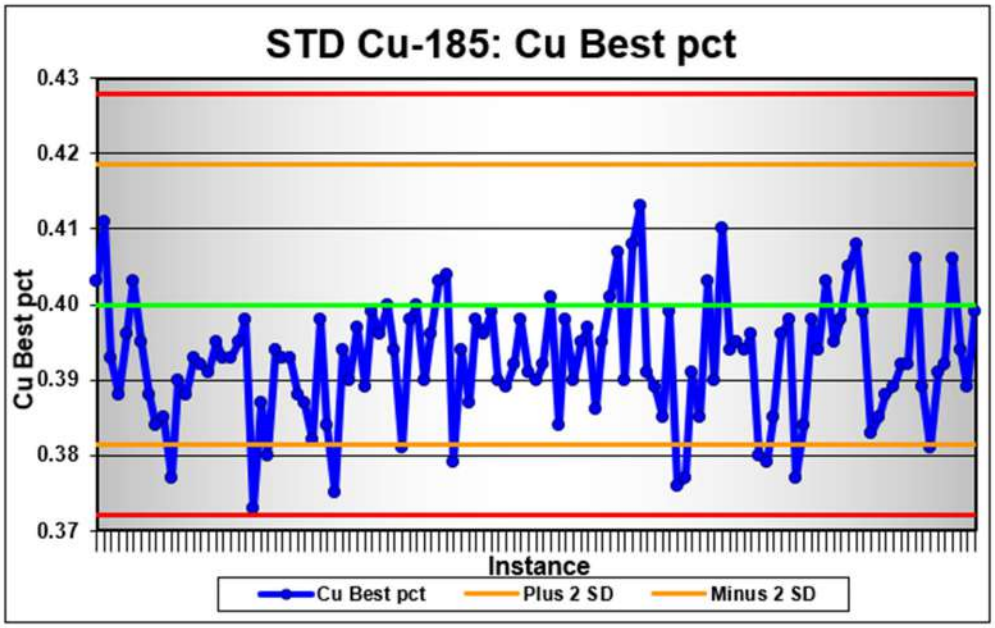


Figure 11-11: Copper Assay results, Sample Standard CU-185

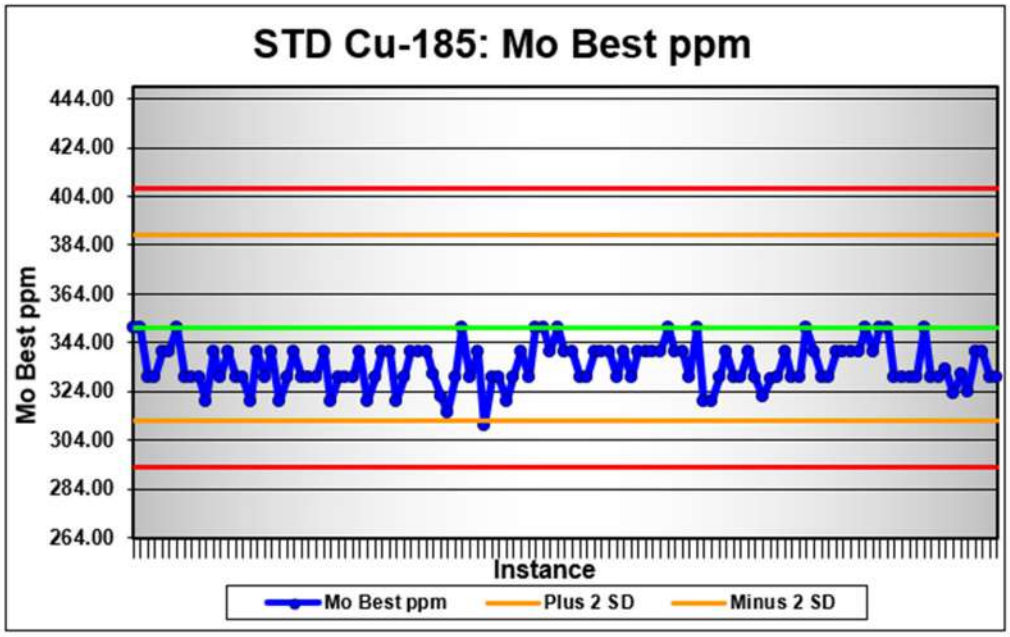


Figure 11-12: Molybdenum Assay results, Sample Standard CU-185

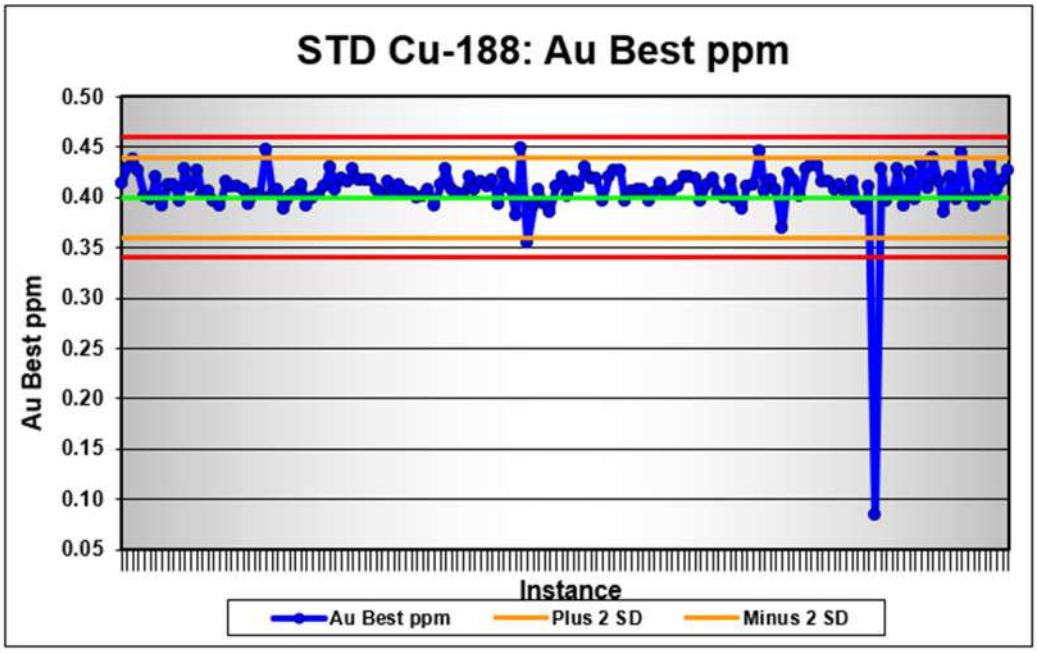


Figure 11-13: Gold Assay results, Sample Standard CU-188

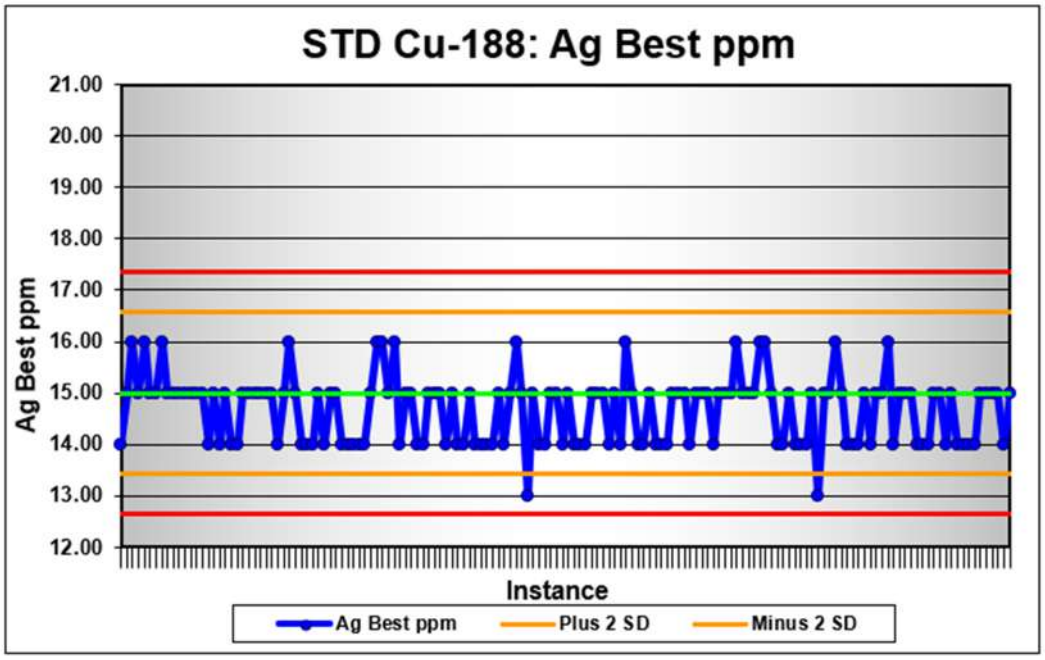


Figure 11-14: Silver Assay Results, Sample Standard CU-188

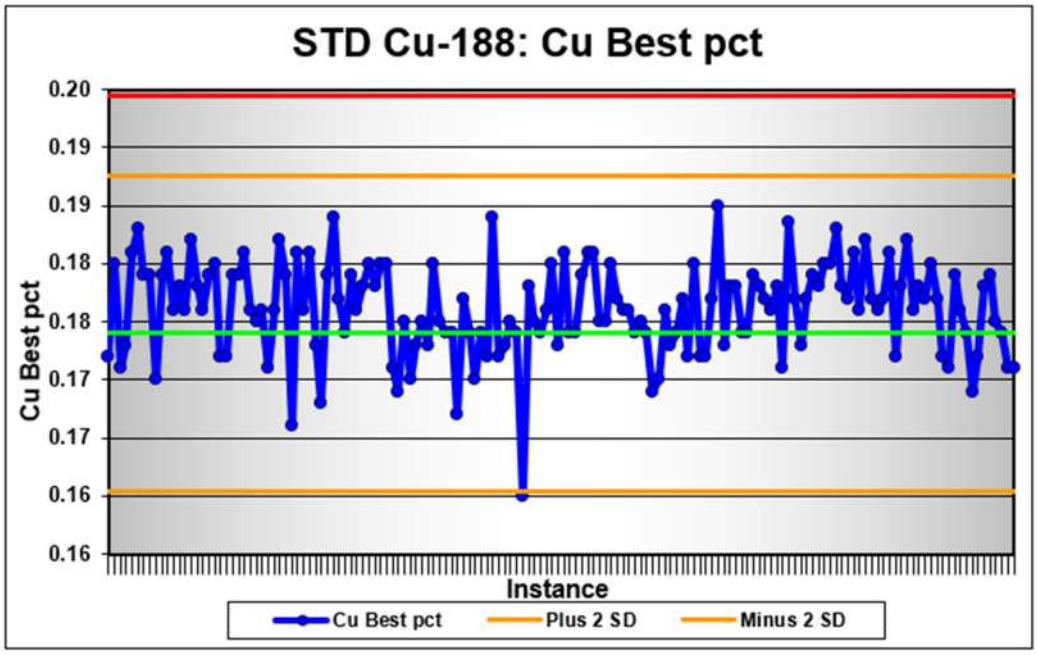


Figure 11-15: Copper Assay Results, Sample Standard CU-188

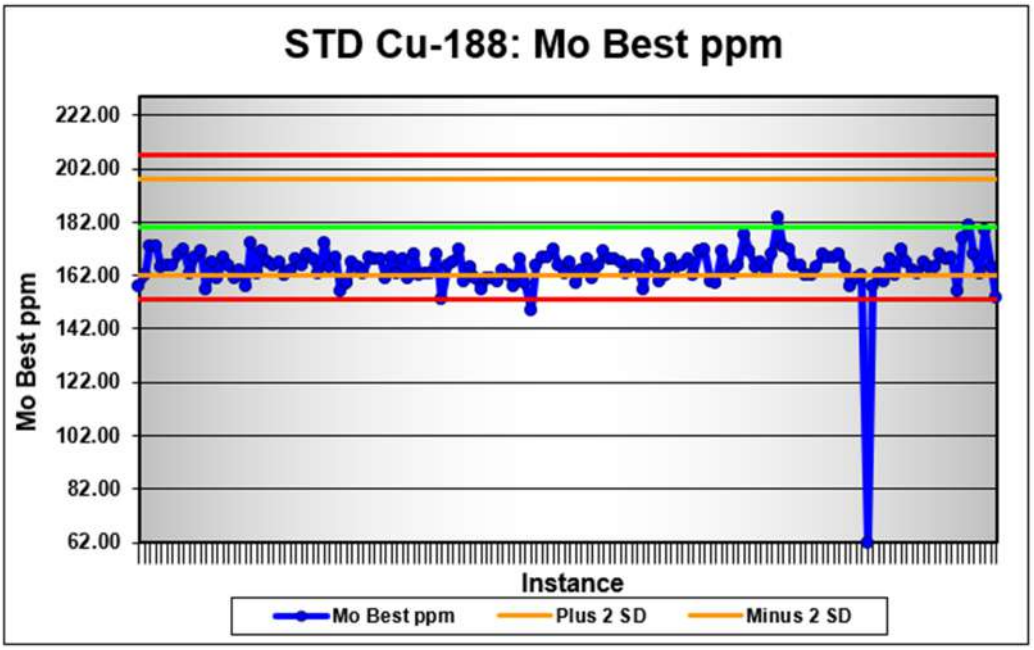


Figure 11-16: Molybdenum Assay results, Sample Standard CU-188

Standard CU-185 performed well for all elements of interest in 2019; all elements had higher than 90% passing rates within both two and three standard deviations of the mean expected values. In general, both copper and molybdenum values fell below the expected mean for CU-185, but still within an acceptable range. Silver showed good variation both above and below the mean value and gold values generally plotted slightly above the mean value. Table 11-3 summarizes the results for CU-185.

Table 11-3: Performance of Standard CU-185 during 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	7	94	2	98
Ag	1	99	1	99
Cu	12	90	0	100
Mo	1	99	0	100

Standard CU-188 also performed well for all elements of interest in 2019; all elements, except molybdenum (Mo) had higher than 90% passing rates within both two and three standard deviations of the mean expected values. In the case of the 32 standards that fell outside of the range of 2 standard deviations for Mo, the chart shows that, overall, this standard returned assay results below the expected mean value for Mo, as did those of CU-185. This indicates that both standards should be reassessed in a round robin process, and that the assay method ALS Global uses may tend toward a low bias for Mo. Even with the 32 Mo failures, 79.2% of the samples fell within 2 standard deviations and the range of values was acceptable. One sample, A0612554, failed outright for both Mo and Au. It is possible this sample became contaminated, as ALS Global had notified the project manager that this sample arrived with a torn plastic bag and had to be dried. Table 11-4 summarizes the results for CU-188.

Table 11-4: Performance of Standard CU-188 during 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	7	95	1	99
Ag	2	98.7	0	100
Cu	1	99	0	100
Mo	32	79.2	2	98.7

11.5.1.3 Reference Material Standards, 2020

The SRM “standards” used in 2020 were prepared by WCM Minerals in Burnaby, BC. Details of the standards are outlined in Table 11-5 below. Both standards were certified by Lloyd Twaites and Glen Armanini, who are both Registered Assayers in British Columbia.

Table 11-5: Reference material "Standards", utilized in 2021 (OREAS)

Standard	Copper (%)	Standard Deviation	Standard	Copper (%)	Standard Deviation	Standard	Copper (%)	Standard Deviation
CU-190	0.65	0.0188	0.032	0.0013	9	0.7580	0.68	0.0279
CU-188	0.179	0.0068	0.018	0.0009	15	0.7883	0.4	0.0199

In 2020, 250 sample standards (1 standard within every 20 samples) were submitted regularly with the sample shipments. Of these, 178 were of CU-188 and 72 were of CU-190. ALS Global Geochemistry (ALS) analyzed the

standard samples along with the drill core samples by gold, copper, and molybdenum assay, as well as multi-element ICP-AES.

The results from sample standards CU-188 and CU-190 for gold, silver, copper, and molybdenum analyses are plotted below in Figure 11-17 through Figure 11-24.

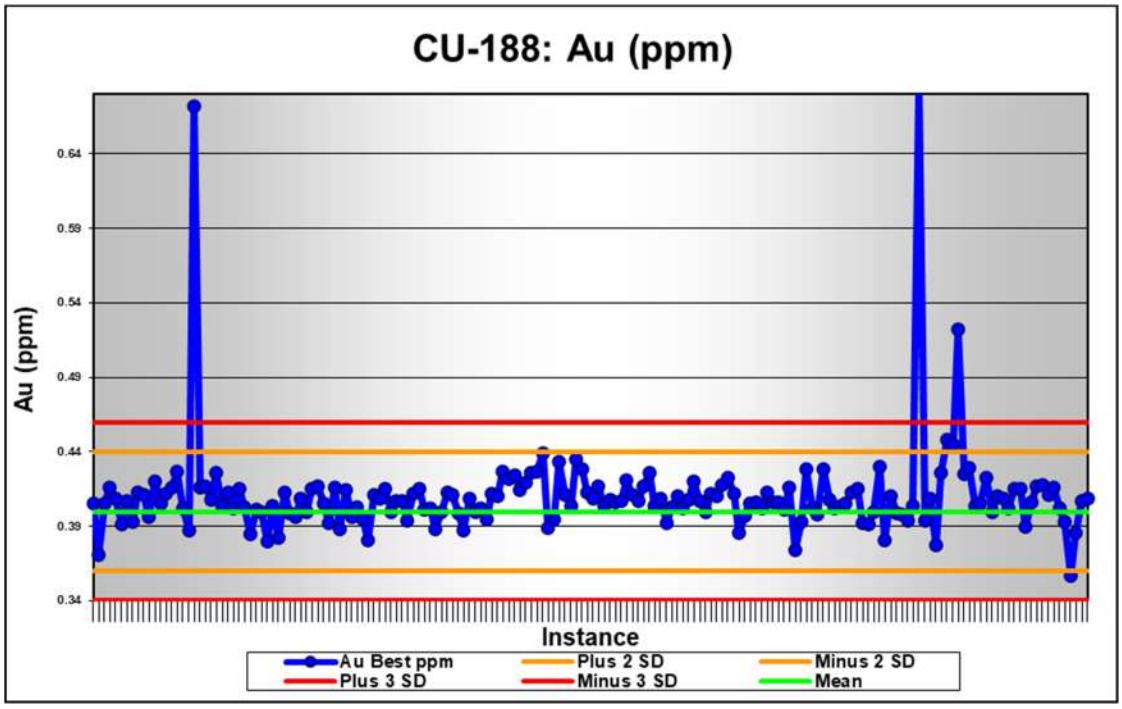


Figure 11-17: Gold assay results, Standard CU-188

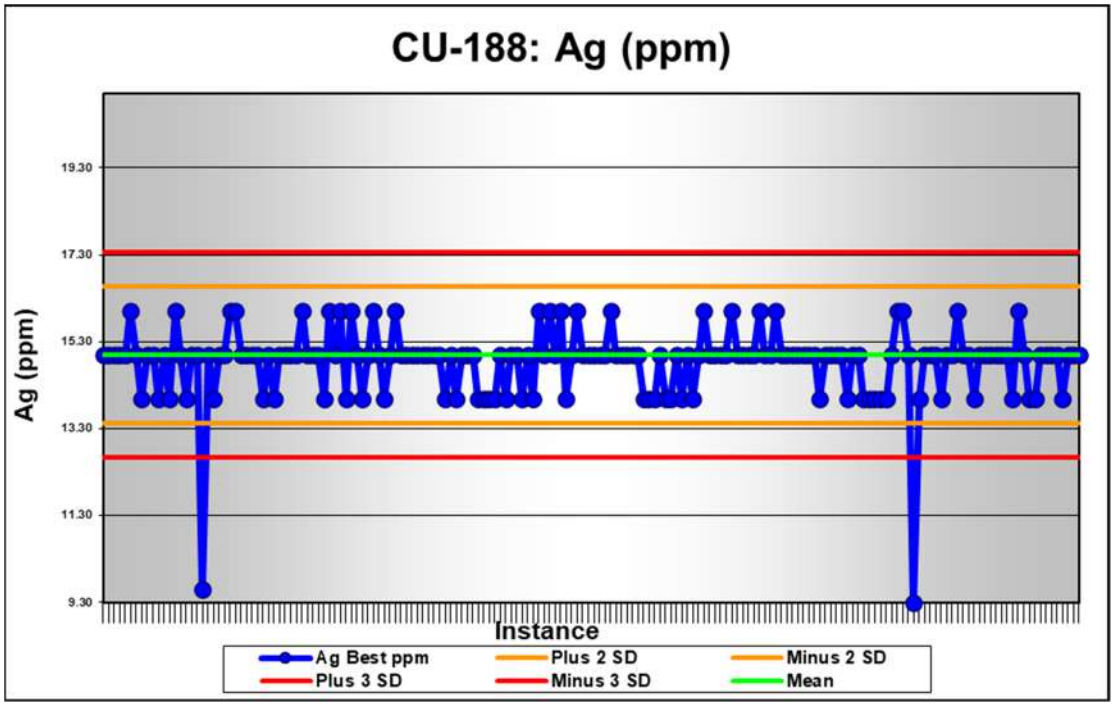


Figure 11-18: Silver assay results, Standard CU-188

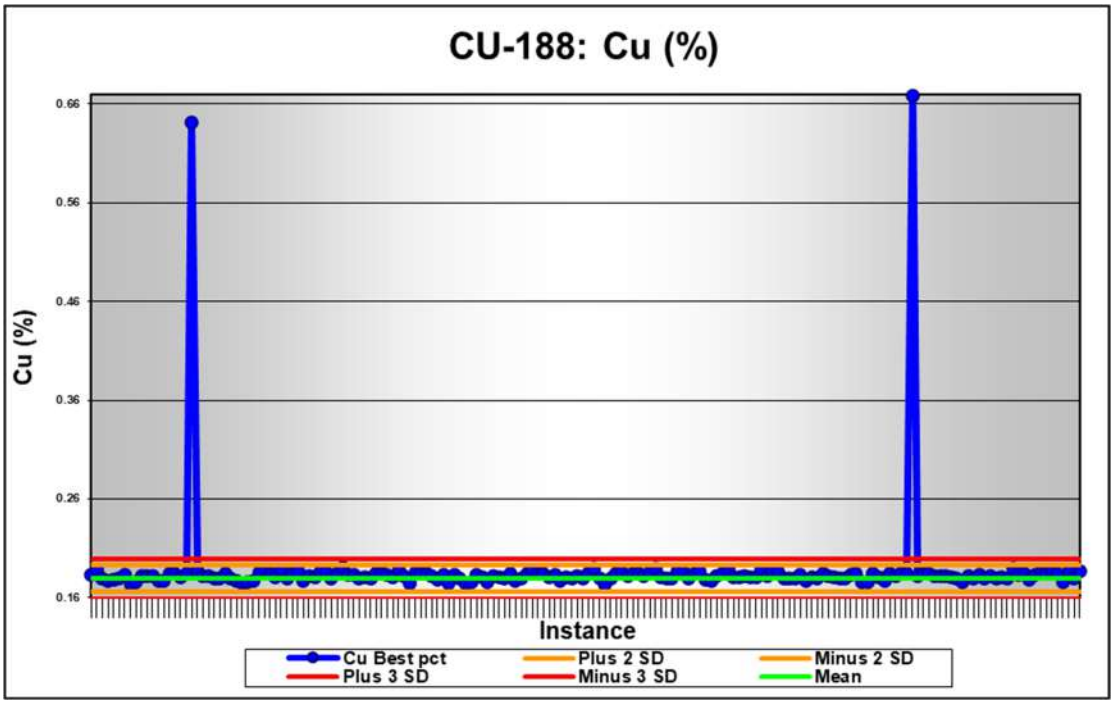


Figure 11-19: Copper assay results, Standard CU-188

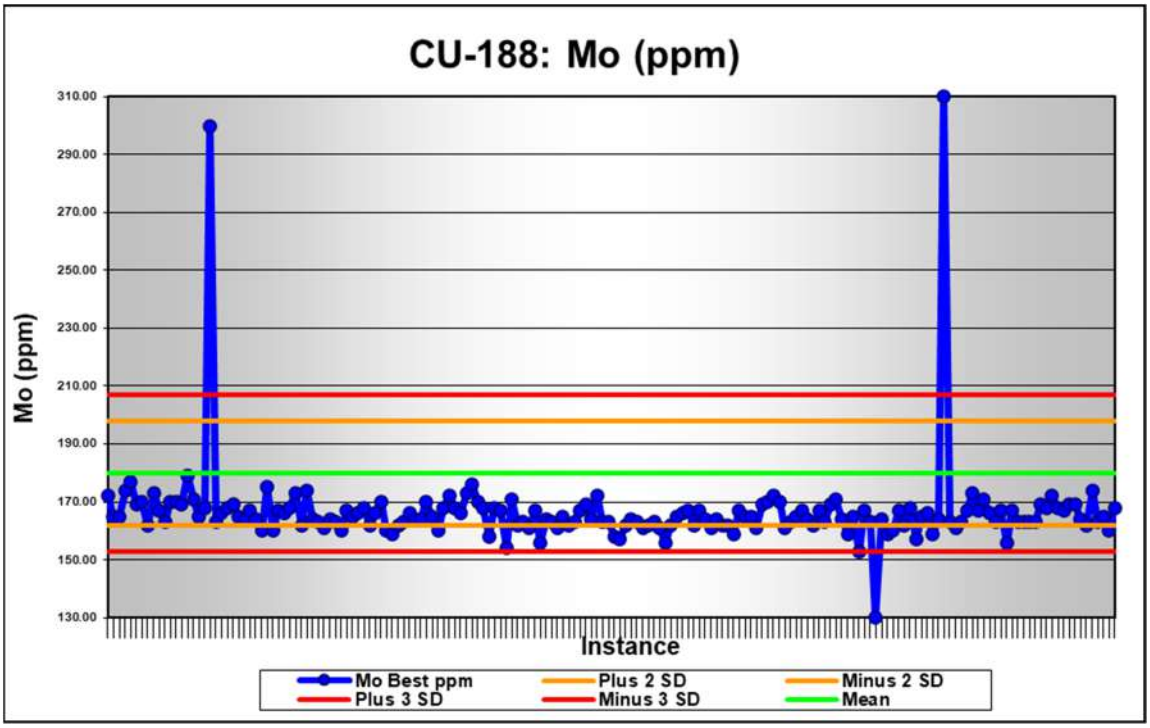


Figure 11-20: Molybdenum assay results, Standard CU-188

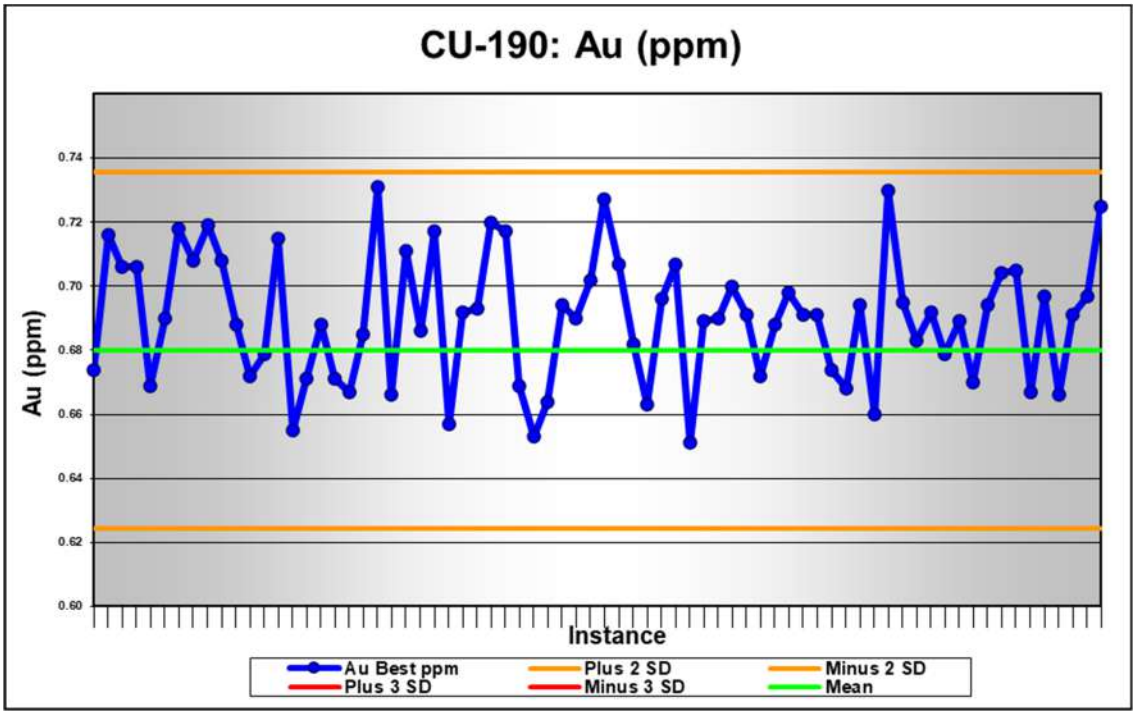


Figure 11-21: Gold assay results, Standard CU-190

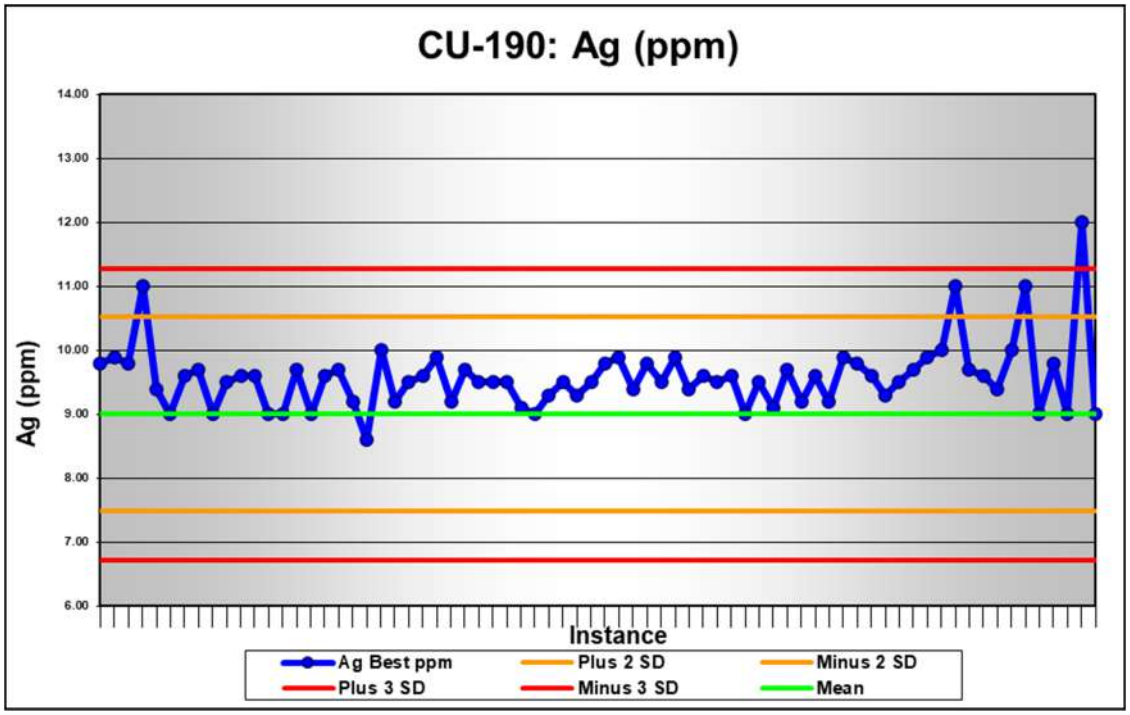


Figure 11-22: Silver assay results, Standard CU-190

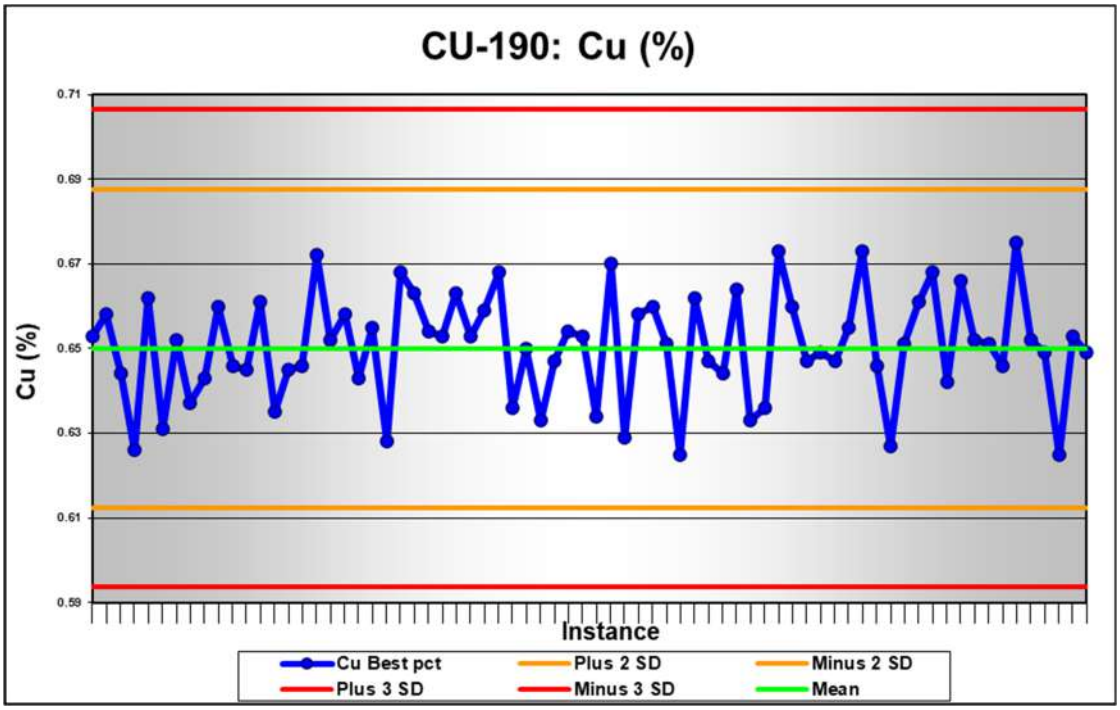


Figure 11-23: Copper assay results, Standard CU-190

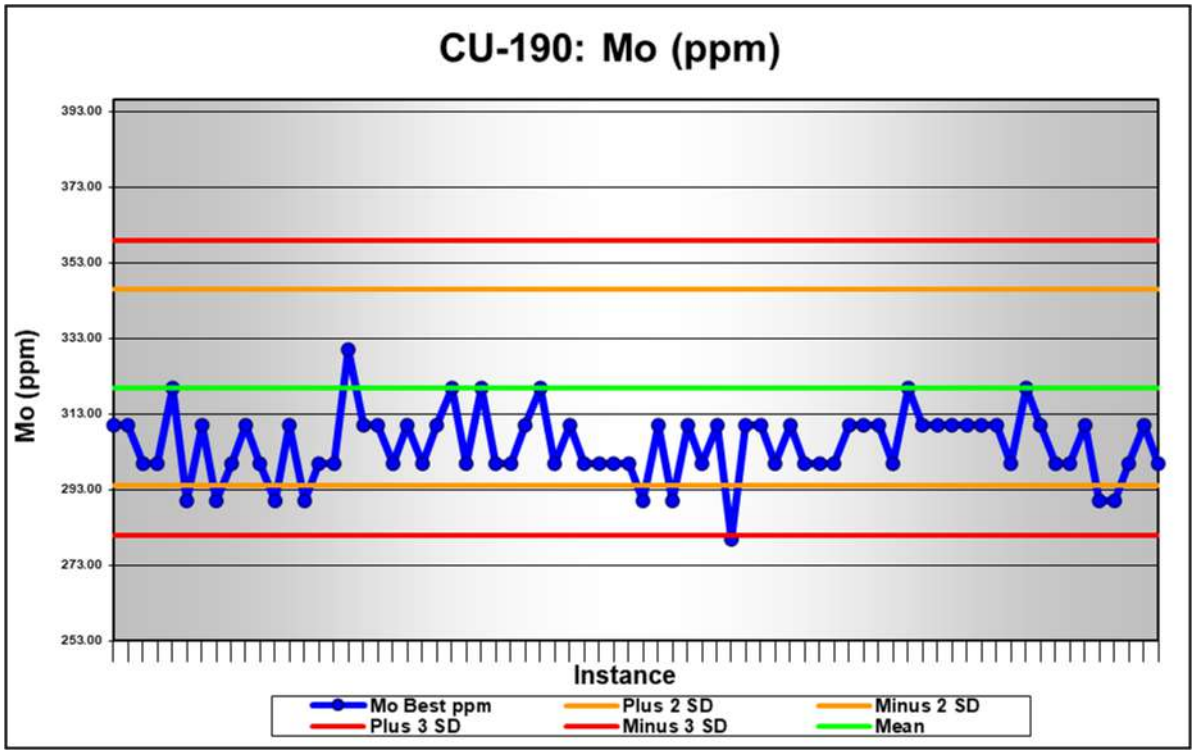


Figure 11-24: Molybdenum assay results, Standard CU-190

Standard reference material CU-188 performed well for gold, silver, and copper; all had higher than 96% passing rates within both two and three standard deviations of the mean expected values. CU-188 did not perform as well for molybdenum, but results were still acceptable, with 82% passing within two standard deviations and 98% within three standard deviations. Molybdenum values in general fell below the expected mean for CU-188, with an average closer to 166 ppm Mo, 14 ppm below the expected mean of 180 ppm. This also occurred in 2019 with CU-188 and indicates the standard itself should be reassessed by a Round Robin analysis. Two samples that failed for all the elements of interest (B660215 and A0610430) likely represent data entry errors as their values correspond to those of CU-190. Table 11-6 summarizes the results for CU-188.

Table 11-6: Performance of Standard CU-188 during 2020 drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	6	97	3	98
Ag	2	99	2	99
Cu	2	99	2	99
Mo	33	82	3	98

Standard reference material CU-190 also performed well for all elements of interest in 2020 (Table 11-7); all elements, except molybdenum (Mo), had higher than 93% passing rates within both two and three standard deviations of the mean expected values. As with CU-188, the overall average Mo value returned for CU-190 is 305 ppm, 15 ppm lower than the expected certified value of 320 ppm Mo. Over 81% of CU-190 sample values were within two standard

deviations, which is acceptable, but similar to results for CU-188. CU-190 likely requires an updated Round Robin analysis to determine the accuracy of the certified mean value for molybdenum.

Table 11-7: Performance of Standard CU-190 during 2020 drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	0	100	0	100
Ag	4	94	1	99
Cu	0	100	0	100
Mo	9	87.5	1	99

11.5.1.4 Reference Standards 2021

The CRMs "standards" used in 2021 were prepared by ORE Research & Exploration Pty Ltd. Details of the standards are outlined in Table 11-8 below. All standards were certified by Craig Hamlyn, Technical Manager of ORE Pty Ltd.

Table 11-8: Reference material "Standards", utilized in 2021 (OREAS)

Standard	Copper (%)	Standard Deviation Cu	Molybdenum (ppm)	Standard Deviation Mo	Silver (g/t)	Standard Deviation Ag	Gold (ppm)	Standard Deviation Au
OREAS 905	0.1533	0.0061	3.27	0.262	n/a	n/a	0.391	0.01
OREAS 506	0.444	0.01	87	3.6	1.88	0.075	0.364	0.0199
OREAS 502c	0.783	0.022	226	12	0.779	0.076	0.488	0.015
OREAS 151a	0.166	0.005	40	3	n/a	n/a	0.043	0.002

In 2021, 132 sample standards (1 standard within every 20 samples) were submitted regularly with the sample shipments. Of these, 61 were of OREAS 151a, 27 were OREAS 502c, 13 were OREAS 506 and 31 were OREAS 905. ALS Global Geochemistry (ALS) analyzed the standard samples along with the drill core samples by gold, copper, and molybdenum assay, as well as multi-element ICP-AES.

The results for the standards OREAS 151a are shown below in Figure 11-25 through Figure 11-27.

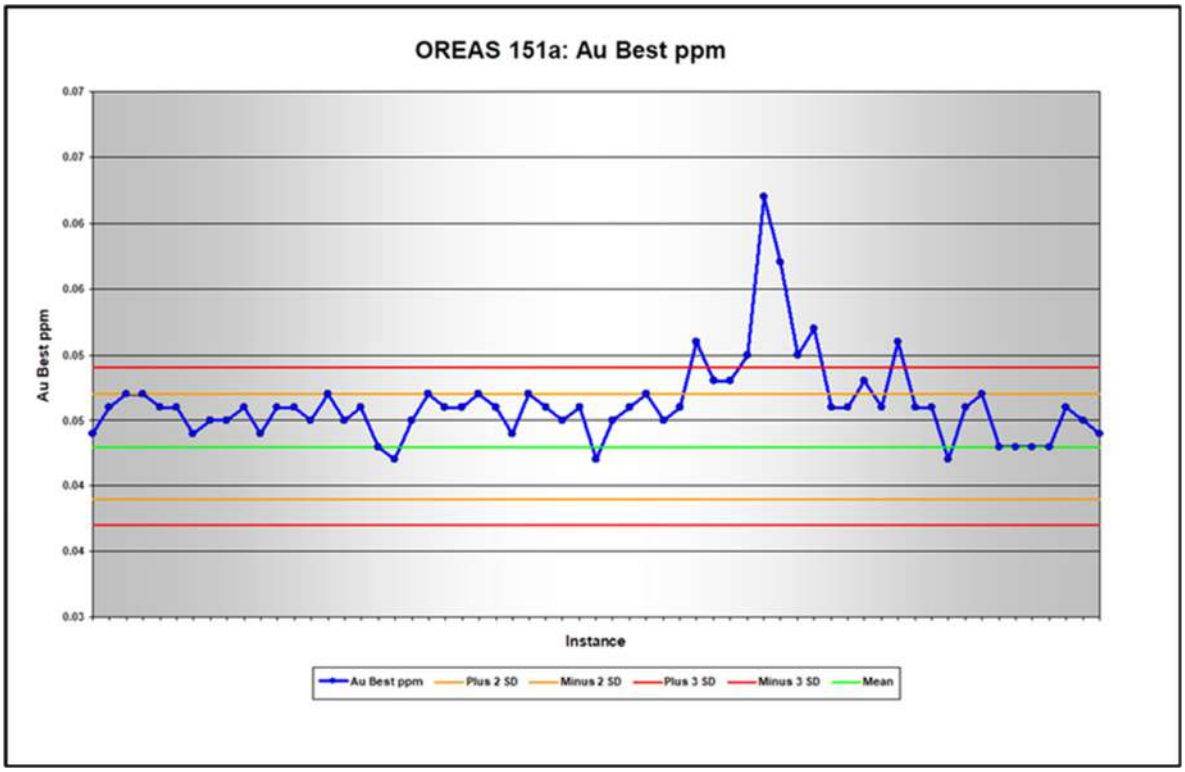


Figure 11-25: Gold assay results, OREAS 151a

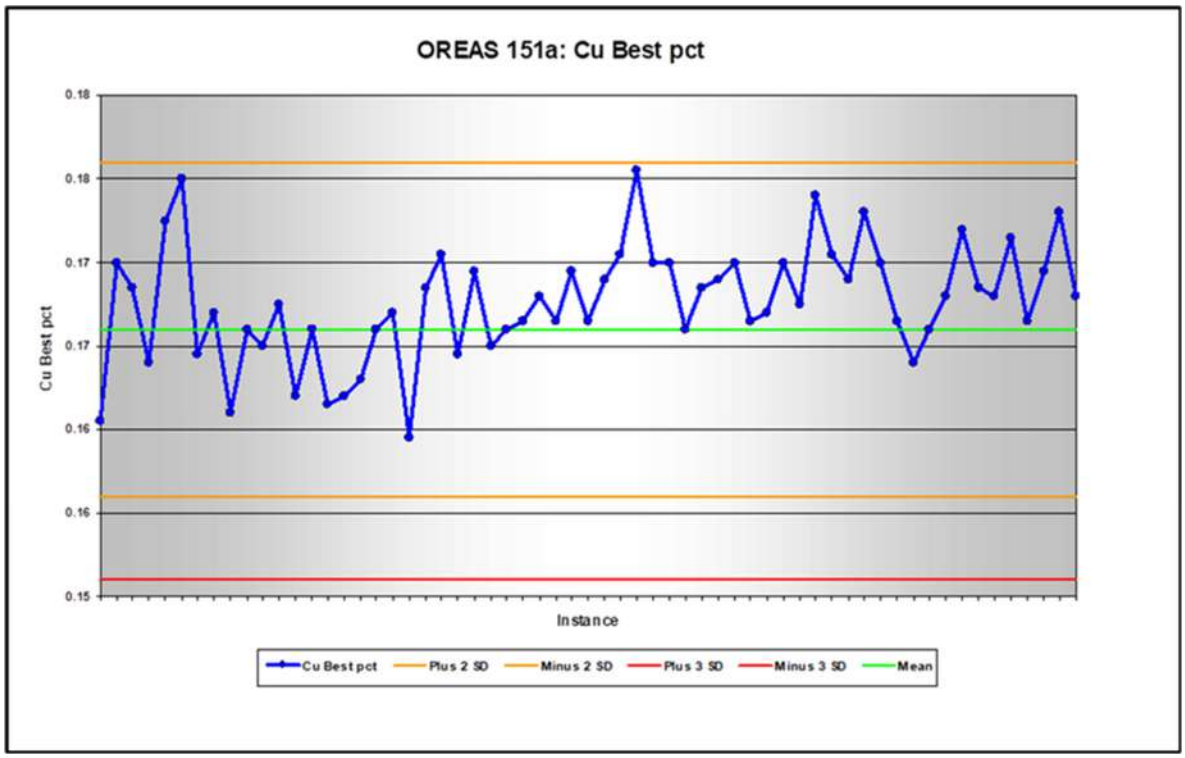


Figure 11-26: Copper assay results, OREAS 151a

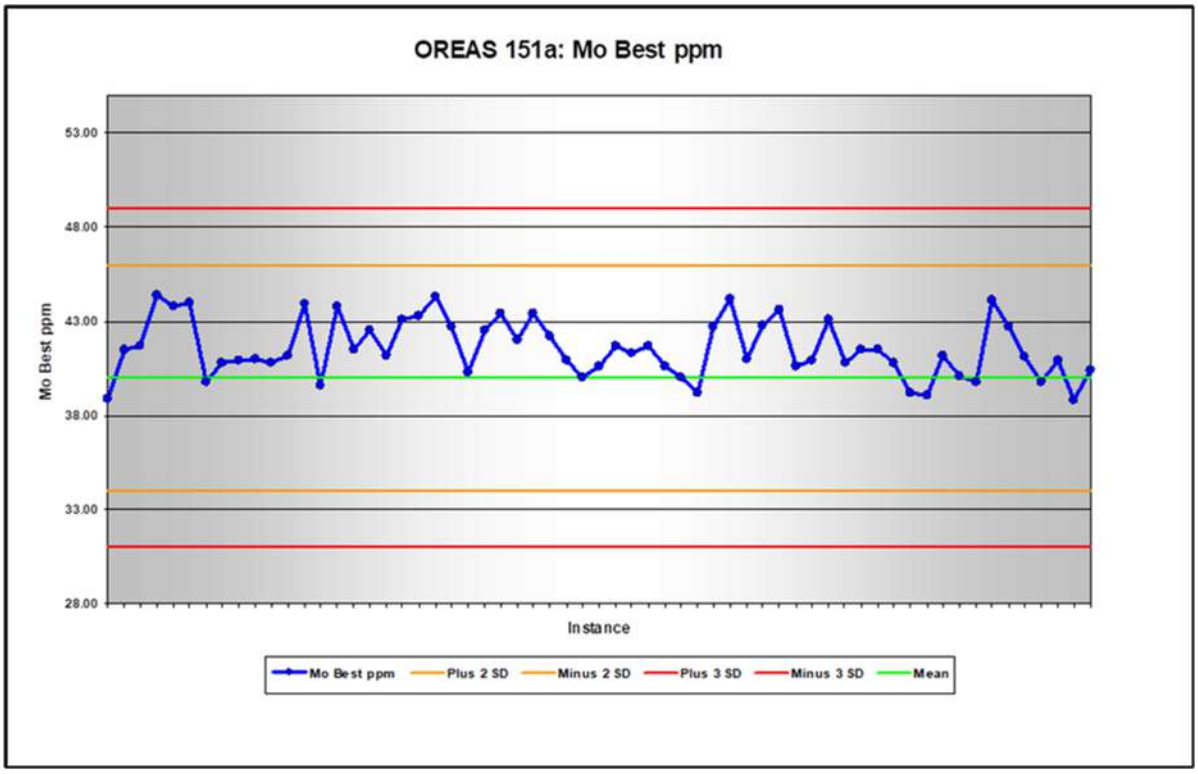


Figure 11-27: Molybdenum assay results, OREAS 151a

OREAS 151a performed well for all elements of interest in 2021; all elements had higher than 80% passing rates within two and three standard deviations of the mean expected values. Copper and molybdenum both had 100% passing rates. All three elements plotted above their mean expected values on average, but within acceptable limits for copper and molybdenum. Gold had several failures from certificate WH2122237 that were on the high end of the spectrum, thus a re-run was requested of these failures and surrounding core samples. While there was not enough material to re-assay the standards that failed, the surrounding core samples were re-assayed and the results were, on average, 0.001 ppm lower than the original assays. This is a relatively small difference but does indicate a high bias or potential contamination with gold fire assay at the time of the original analysis and for this reason. The core sample dataset has been corrected to reflect the re-assay values. At the time of this report, additional samples of OREAS 151a had yet to be supplied to the primary lab for re-analysis. Table 11-9 summarizes the results for OREAS 151a.

Table 11-9: Summary of Results for OREAS 151a

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	10	83.6	7	88.5
Cu	0	100	0	100
Mo	0	100	0	100

Figure 11-28 to Figure 11-31 illustrate the performance of SRM material OREAS 502c.

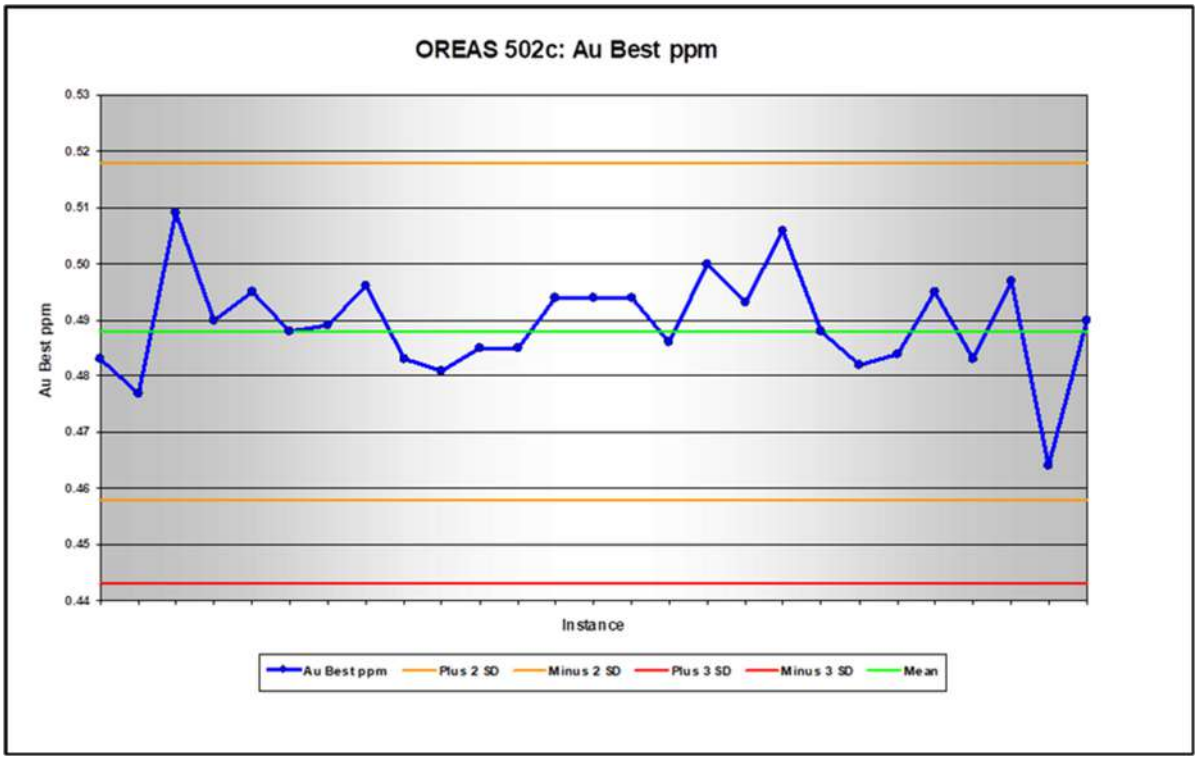


Figure 11-28: Gold assay results, OREAS 502c

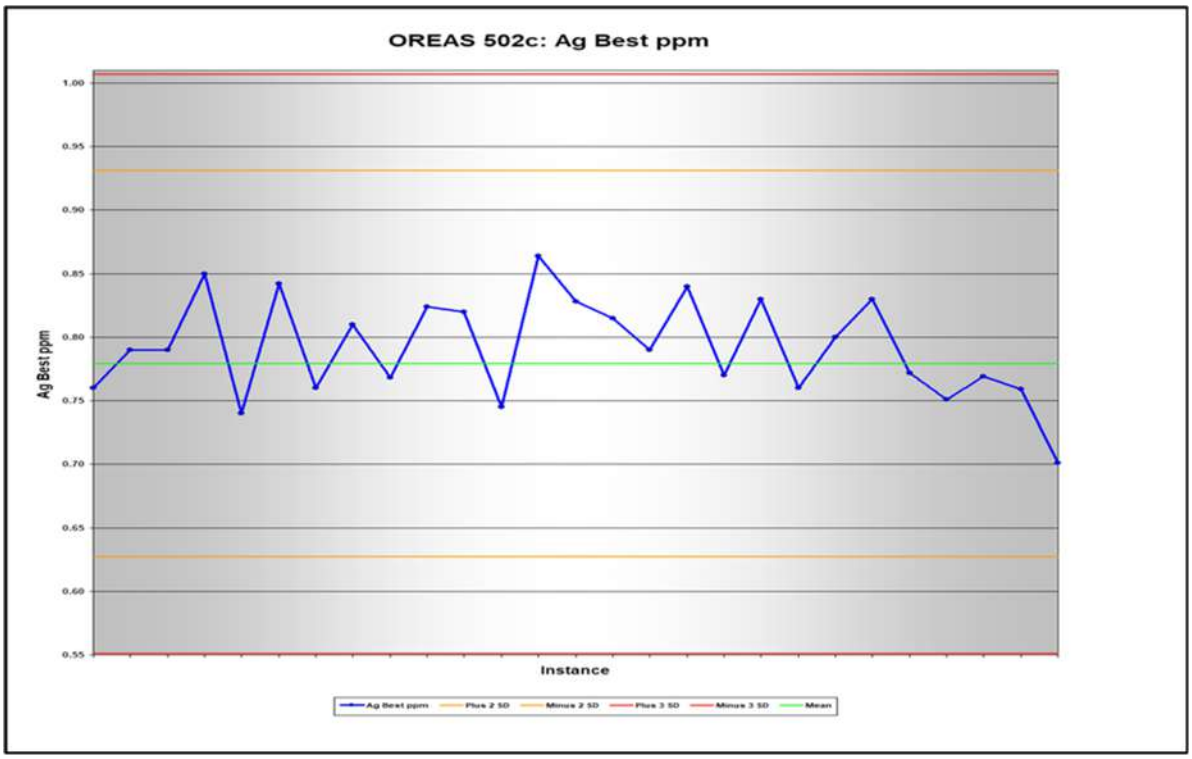


Figure 11-29: Silver assay results, OREAS 502c

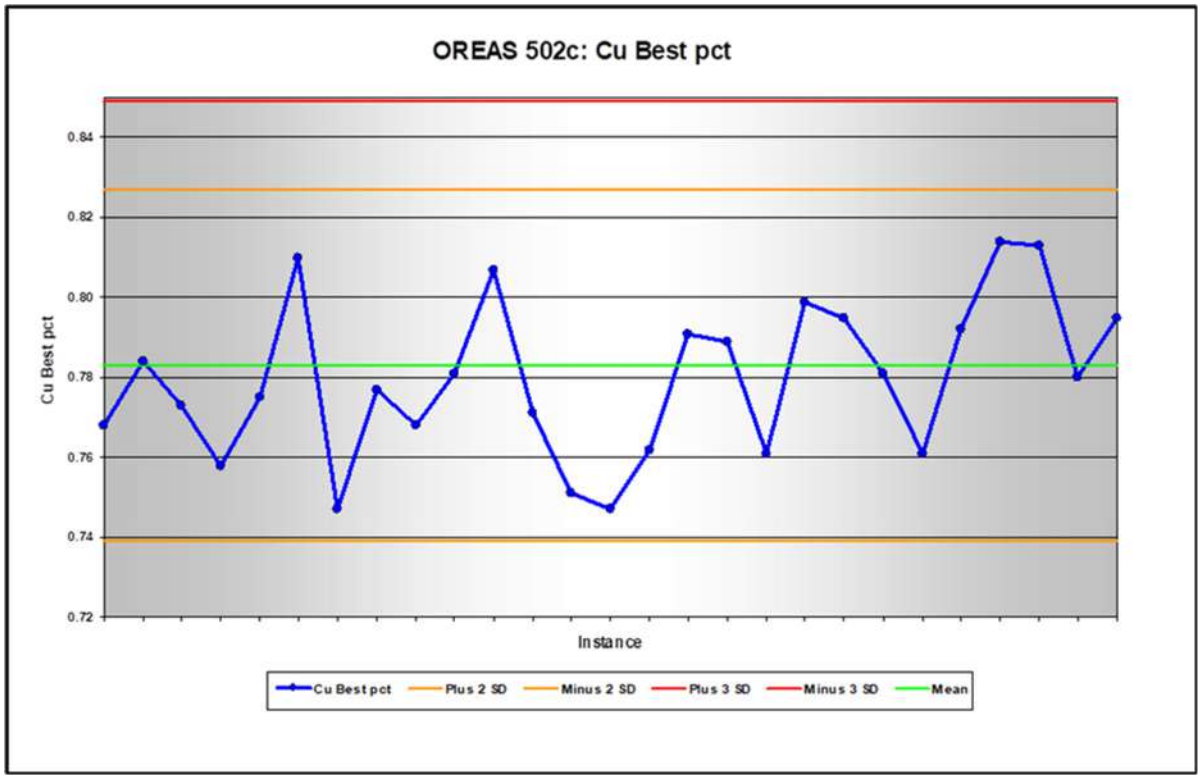


Figure 11-30: Copper assay results, OREAS 502c

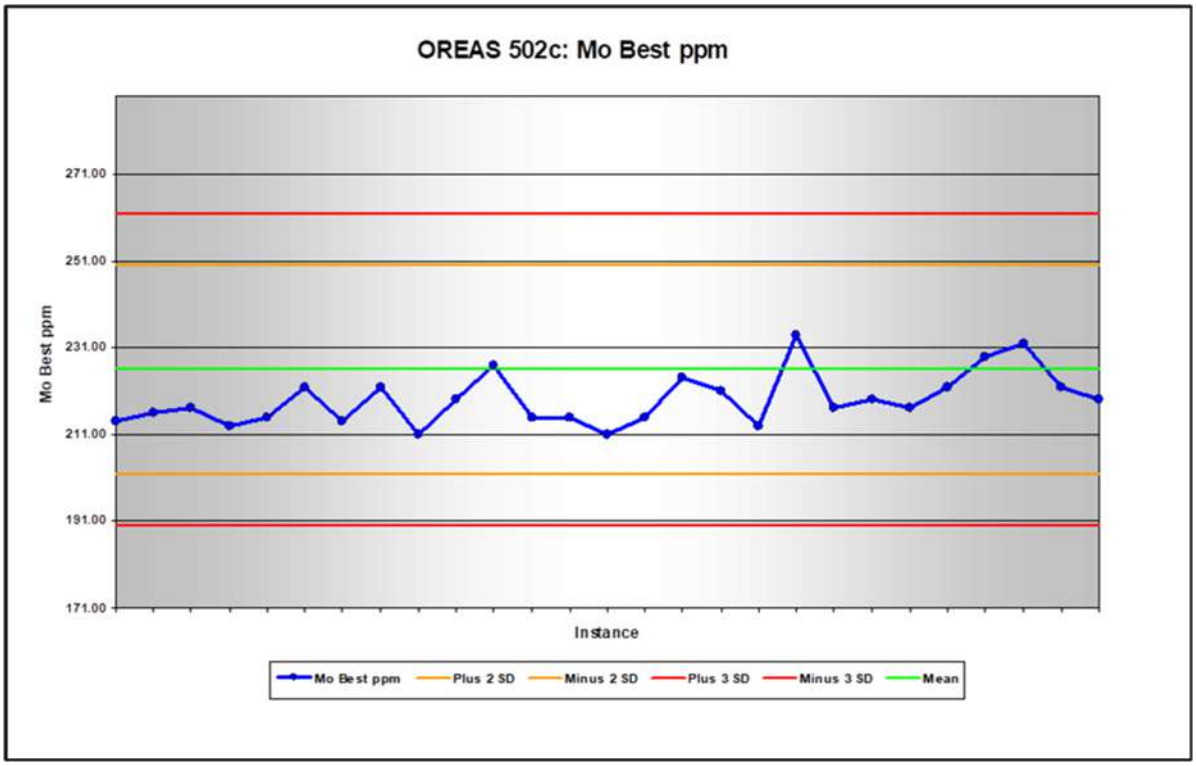


Figure 11-31: Molybdenum assay results, OREAS 502c

OREAS 502c performed well for all elements of interest in 2021 with no failures for any variables. On average, silver results plotted slightly above the mean for the standard and molybdenum plotted slightly below, but all within acceptable limits. Table 11-10 summarizes the results for OREAS 502c.

Table 11-10: Summary of results for OREAS 502c

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	0	100	0	100
Cu	0	100	0	100
Mo	0	100	0	100
Ag	0	100	0	100

Figure 11-32 through Figure 11-35 illustrate the results for SRM material OREAS 506.

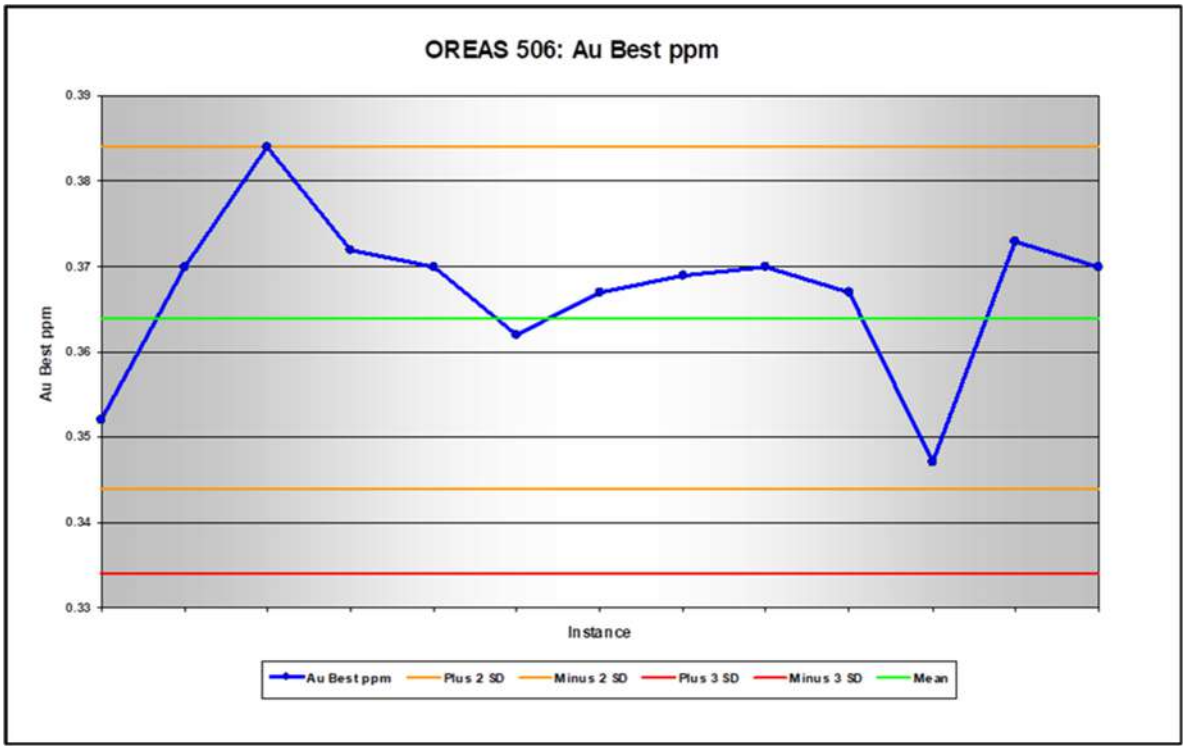


Figure 11-32: Gold assay results, OREAS 506

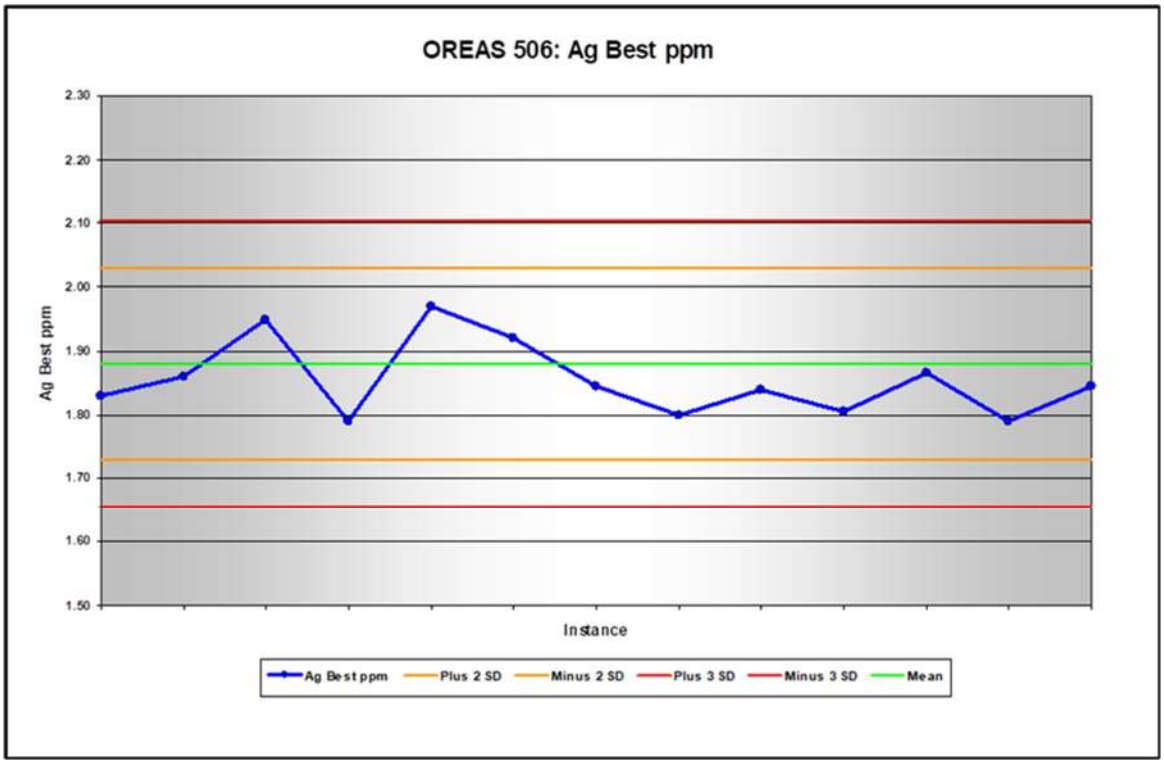


Figure 11-33: Silver assay results, OREAS 506

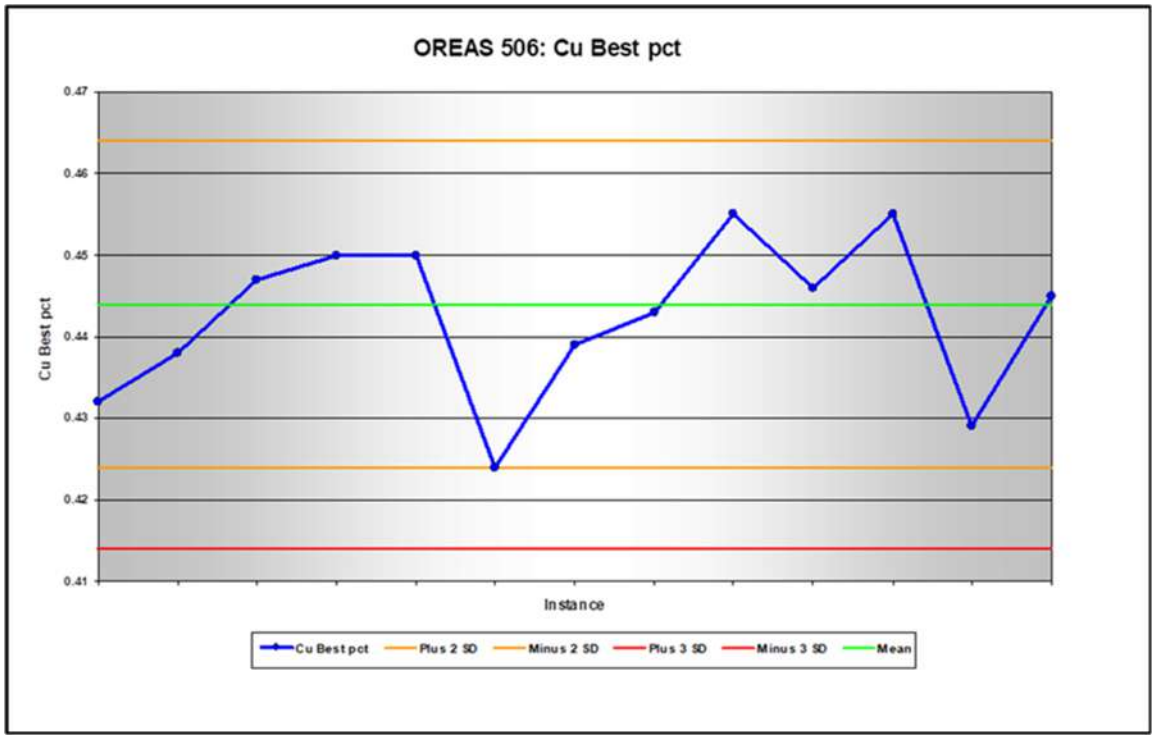


Figure 11-34: Copper assay results, OREAS 506

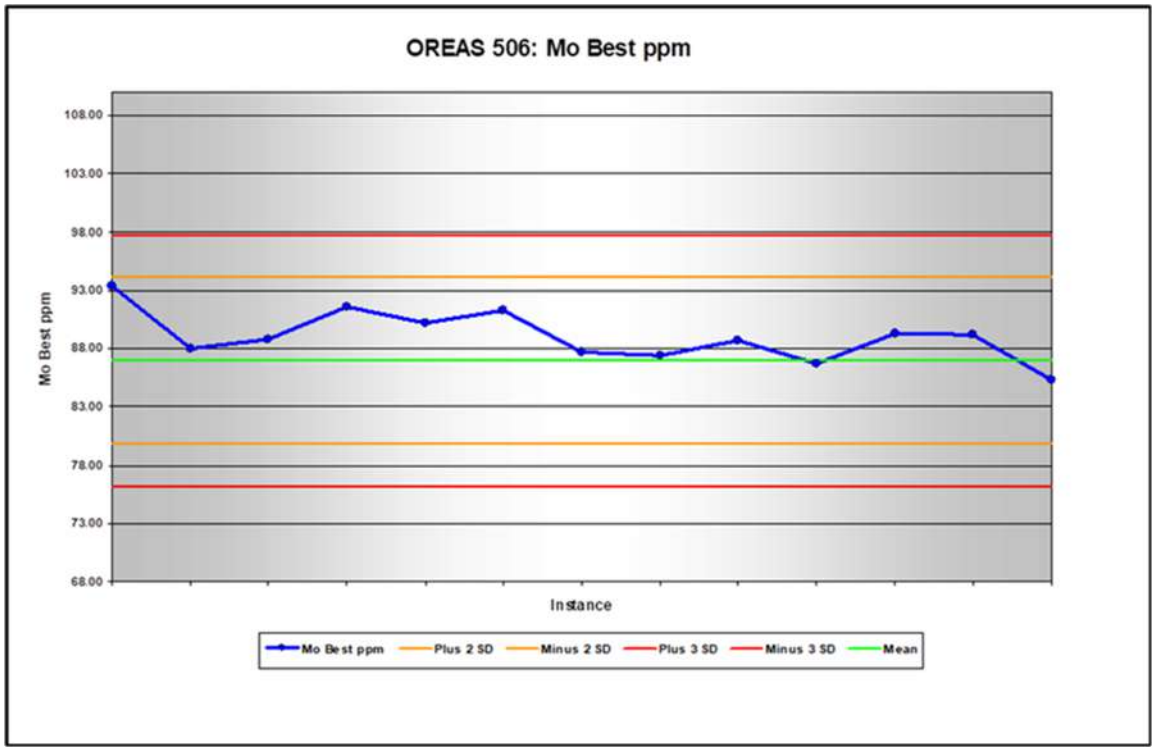


Figure 11-35: Molybdenum assay results, OREAS 506

OREAS 506 performed well for all elements of interest in 2021 with no failures for any variables. For gold, copper, and molybdenum there was one sample for each (all different samples) that plotted close to two standard deviations from the mean, but these were all still within acceptable limits. The population of samples (13) for OREAS 506 was relatively small compared to that of the other standards. A larger data set would improve the ability to assess this reference material. Table 11-11 summarizes the results for OREAS 506.

Table 11-11: Summary of results for OREAS 506

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	0	100	0	100
Cu	0	100	0	100
Mo	0	100	0	100
Ag	0	100	0	100

Figure 11-36 and Figure 11-37 illustrate the results for SRM material OREAS 905.

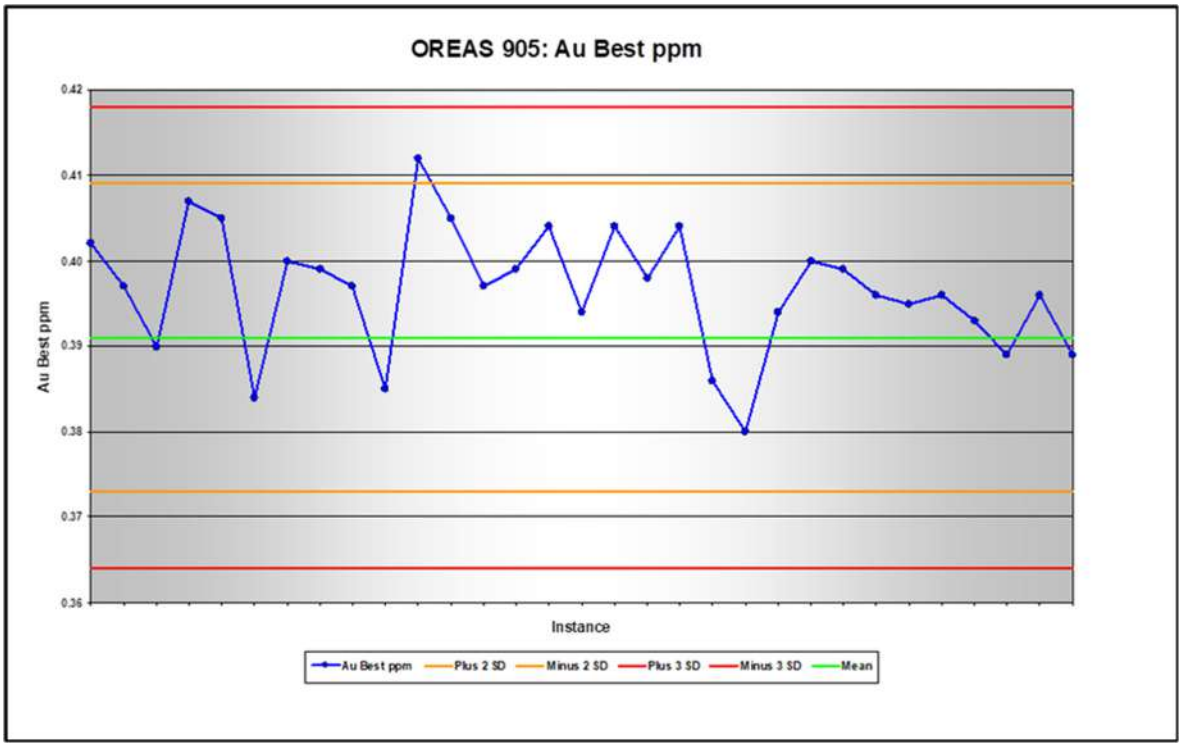


Figure 11-36: Gold assay results, OREAS 905

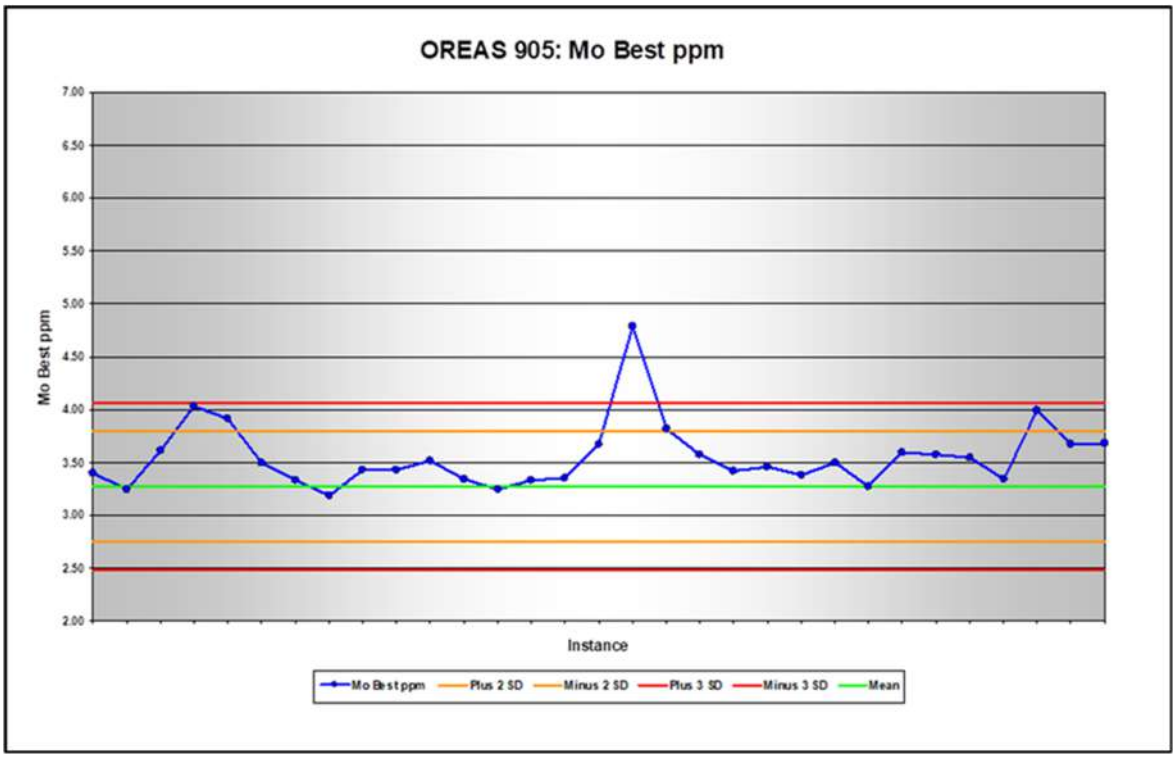


Figure 11-37: Copper assay results, OREAS 905

OREAS 905 performed well for all elements of interest in 2021 with all elements passing above 80%. Molybdenum had five failures outside of two standard deviations and one outside of three. The same samples did not fail for copper and were independent of the sample that failed for gold. Table 11-12 summarizes the results for OREAS 905.

Table 11-12: Summary of Results for OREAS 905

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	1	96.8	0	100
Cu	0	100	0	100
Mo	5	83.9	1	96.8

11.5.2 Blanks

11.5.2.1 Blanks, 2010-2012

Commencing in 2010, sample blanks were regularly inserted into the sample stream. Blanks are included as a check of the lower limit of the analytical range and to ensure that, at all stages in the process, the equipment and instruments are thoroughly cleaned prior to running subsequent samples. This is particularly important for precious metals. A total of 75 blanks were inserted during the 2010 program, nominally one every 50 samples.

The blank samples were also prepared by CDN Resource Laboratories Ltd (Reference material CDN-BL-6). They were certified for gold, platinum, and palladium. The recommended values for these elements were:

- Gold: <0.01 g/t
- Platinum: <0.01 g/t
- Palladium: <0.01 g/t

As the reported recommended gold values by CDN are below detection, they are not included in the plots. The gold values of the blanks analyzed ranged from below detection (<0.005 g/t) to a maximum of 0.046 g/t. The silver values ranged from <0.5 to 0.8 ppm.

11.5.2.2 Blanks, 2019

During the 2019 drill program, a landscape aggregate that was readily available in Whitehorse was used as blank material. It was sent to 4 different labs for a Round Robin analysis, with the following values calculated from the results:

- Gold: 0.002 ppm
- Silver: 0.2 ppm
- Copper: 0.00045 %
- Molybdenum: 0.39 ppm

Approximately 100 g of blank material were placed in each sample bag, and 1 blank sample was inserted randomly within every 20 core samples. A total of 277 blank samples were inserted into the sample stream in 2019.

The results from blank material for gold, silver, copper, and molybdenum analyses are plotted below in Figure 11-38 through Figure 11-41.

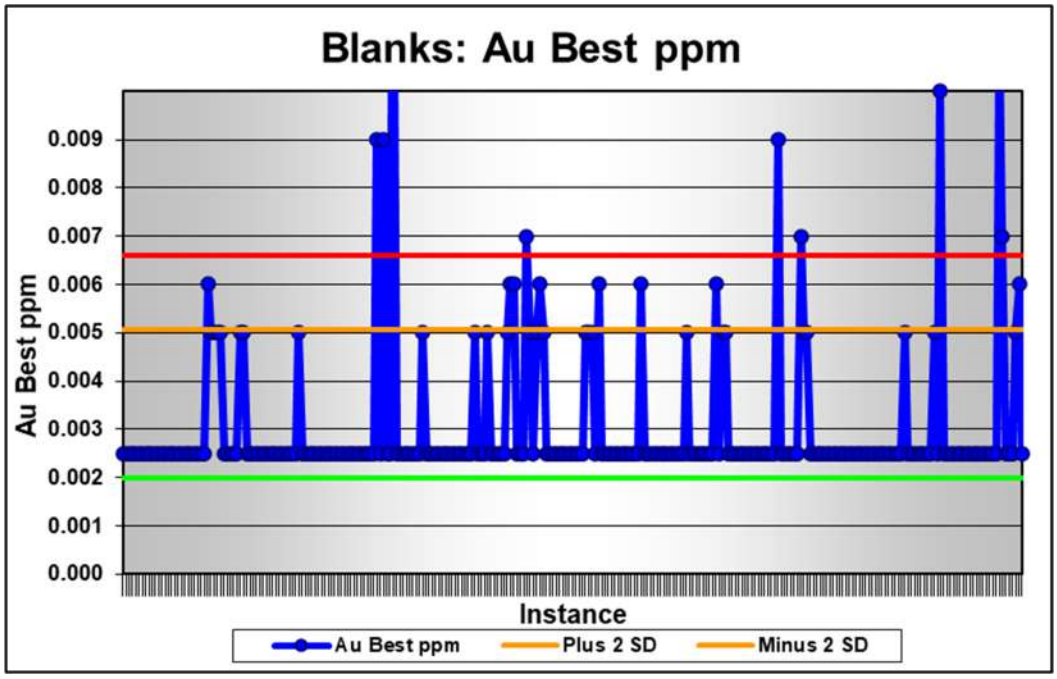


Figure 11-38: Gold Assay results, Blank Material

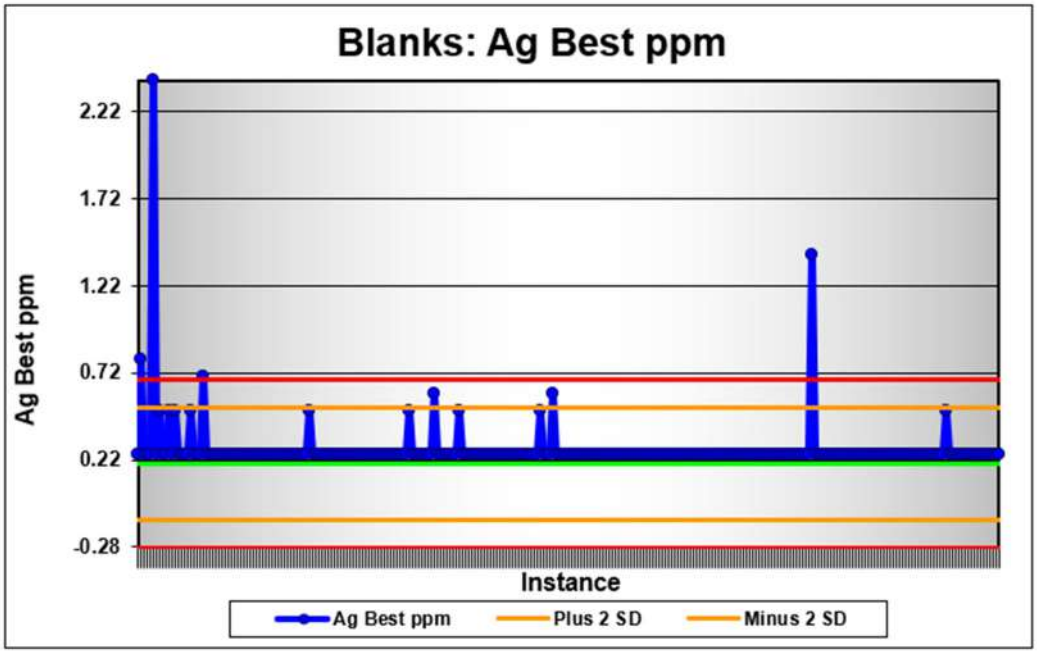


Figure 11-39: Silver Assay results, Blank Material

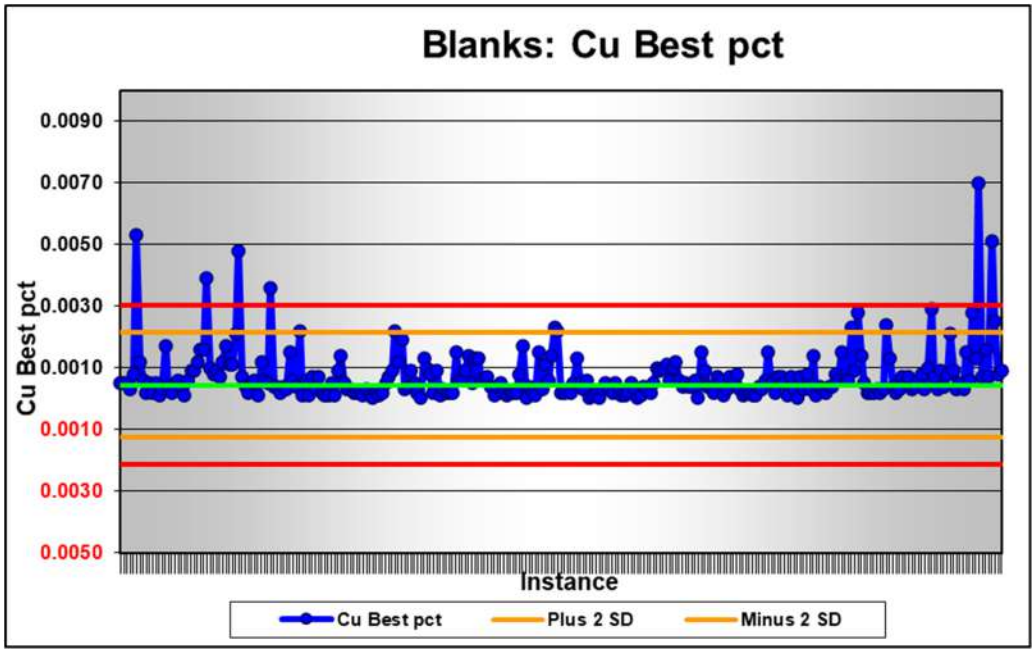


Figure 11-40: Copper Assay Results, Blank Material

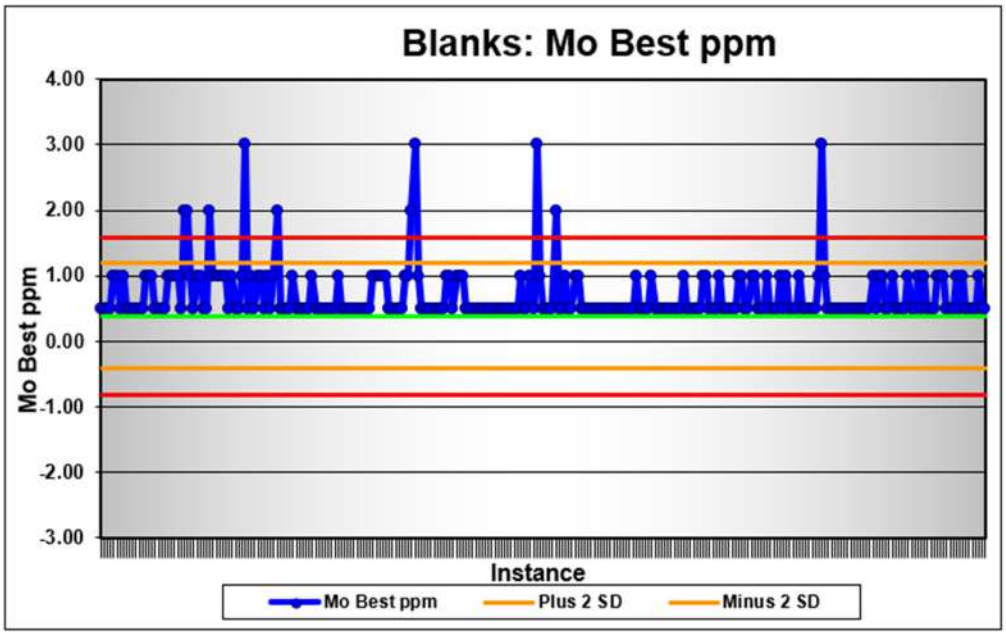


Figure 11-41: Molybdenum Assay Results, Blank Material

The blank material performed well for all elements of interest in 2019; all elements had higher than 90% passing rates within both two and three standard deviations of the mean expected values. On average gold values plotted well above the expected mean, but as Figure 11-38 shows, the detection limit for gold at ALS Global limits the lowest assay value to 0.0025 ppm, which is above the expected mean of 0.002 ppm for the blank material. The detection limits for silver and molybdenum are also higher than the expected mean of the blank material for those elements. Even with the detection limit cut-offs, the passing rates are still acceptable. Figure 11-38 and Figure 11-39, for gold and silver

respectively, do show a few off-chart potentially high-value failures. Upon investigation for gold the two samples with the greatest variation from the expected mean of 0.002 ppm Au varied by only 10-13%. The samples prior to these blanks returned 0.159 ppm Au and 0.283 ppm Au respectively, indicating the likelihood of some minor smear during the assaying. The failures for silver are somewhat less certain as there is no indication of high-grade material prior to the failed silver values. Table 11-13 below summarizes the overall performance of the Blank Material in 2019.

Table 11-13: Performance of Blank Material during 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	17	94	9	97
Ag	6	98	4	99
Cu	16	94	6	98
Mo	10	96	10	96

11.5.2.3 Blanks, 2020

During the 2020 drill program, a landscape aggregate (dolostone) that was readily available in Whitehorse was used as Blank Material. The material was of the same type as that submitted for round-robin analysis in 2019.

Each “blank” sample was comprised of approximately 200 g of material placed into a sample bag. One blank sample was inserted randomly within every 20 core samples. A total of 251 Blanks were inserted into the sample stream in 2020.

The results from blank material for gold, silver, copper, and molybdenum analyses are plotted below in Figure 11-42 through Figure 11-45.

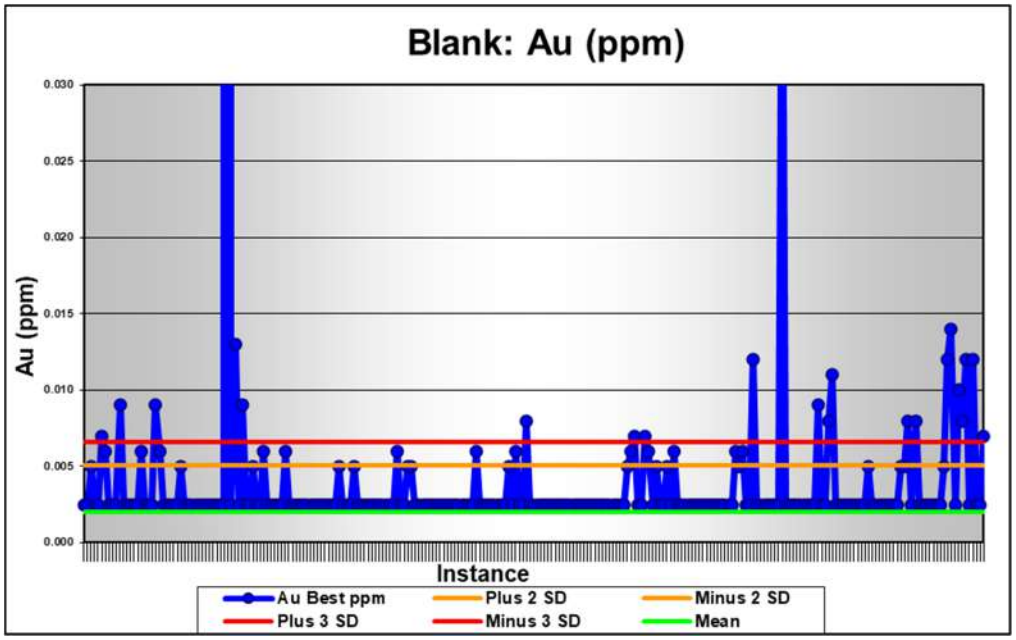


Figure 11-42: Gold Assay results, Blank Material

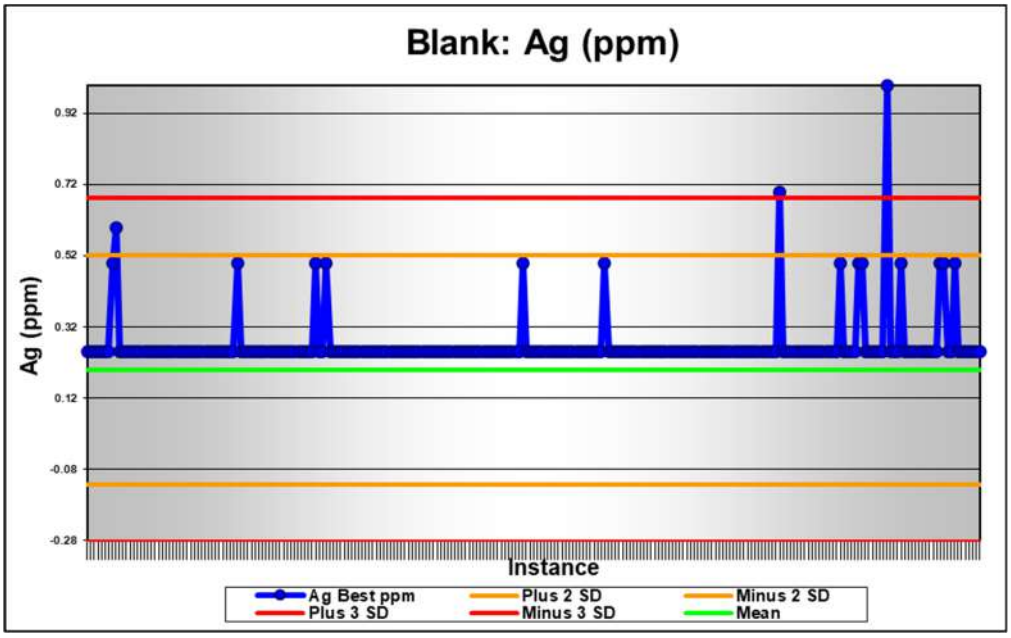


Figure 11-43: Silver Assay Results, Blank Material

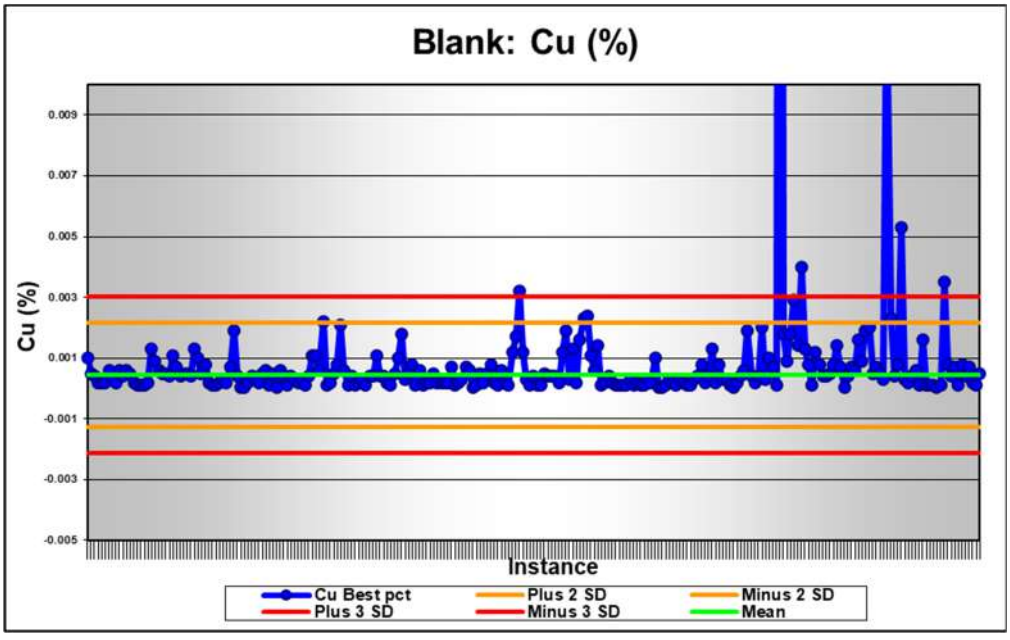


Figure 11-44: Copper Assay Results, Blank Material

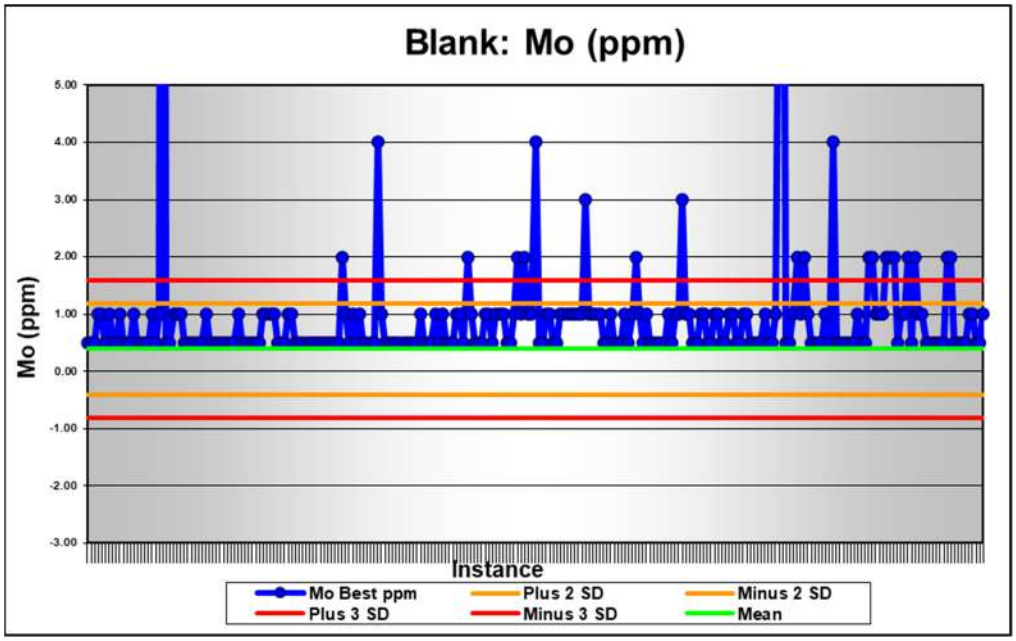


Figure 11-45: Molybdenum Assay Results, Blank Material

The blank material performed well for all elements of interest in 2020; all elements had higher than 85% passing rates within both two and three standard deviations of the mean expected values. On average, gold values plotted well above the expected mean, but, as Figure 11-42 shows, the detection limit for gold at ALS limits the lowest assay value to 0.0025 ppm, which is above the expected mean of 0.002 ppm for the blank material. The detection limits for silver and molybdenum are also higher than the expected means of the blank material for those elements. Even with the detection limit cut-offs, the passing rates are still acceptable. Figure 11-42 through Figure 11-45 do show a few off-chart, high value fails. Table 11-14 below summarizes the overall performance of the blank material in 2020.

Upon review of gold values, the sample with the greatest variation is sample B661435, at 2.35 ppm Au. This sample is only anomalous for gold as the other elements are all within the acceptable values for blank material. This sample was placed within a zone of anomalous gold values, but this zone is significantly less than 2 ppm Au. It's likely this is a lab error for gold. Another sample, B660416, returned 0.044 ppm Au, over twenty times the expected value; it was not placed in a zone of higher-grade material. The geochemical analysis for this sample shows fail values for all elements of interest; therefore, this is likely a data entry or sampling error. The remaining fail values for gold are all within 10 x the expected value of 0.002 ppm (e.g., B660460 is 0.012 ppm Au). They are all located either within zones of higher-grade material, or immediately following a "Standard" sample. The likely cause is laboratory equipment smear.

Of the three samples returning fail values for silver, one, as noted above for gold, is B660416, which likely represents a data entry error. The causes of the other two failures are somewhat less certain, as there are no indications of higher-grade material prior to these samples, and the other elements were within acceptable ranges.

The majority of the copper fail values were on the border of either two or three standard deviations, allowing for some leeway in accepting these as actual fail values (Table 11-14). These samples were all within zones of anomalous Cu grades. The remainder of the fails that exceeded three standard deviations account for only 2% of all the blanks within the 2020 season. One of these samples is B660416, which has been deemed a data entry error, and the other four are within higher grade zones and can be attributed to minor smear effects.

Table 11-14: Performance of Blank Material during 2020 Drill Program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	36	85.7	23	90.8
Ag	3	98.8	2	99.2
Cu	11	95.6	6	97.6
Mo	24	90.4	24	90.4

11.5.2.4 Blanks, 2021

During the 2021 drill program, a landscape aggregate (dolostone) that was readily available in Whitehorse was used as Blank Material. The material was of the same type as that submitted for round-robin analysis in 2019.

Each "blank" sample was comprised of approximately 200 g of material placed into a sample bag. One blank sample was inserted randomly within every 20 core samples. A total of 134 Blanks were inserted into the sample stream in 2021.

The results from blank material for gold, silver, copper, and molybdenum analyses are plotted below in Figure 11-46 through Figure 11-49.

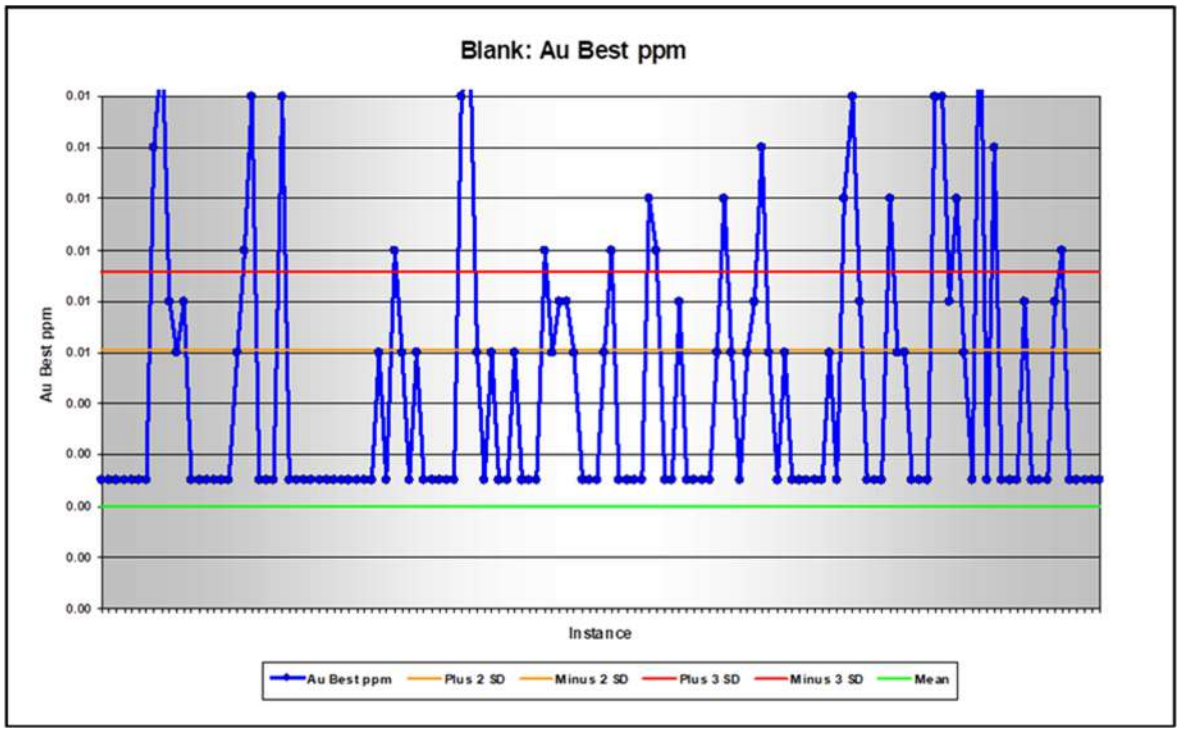


Figure 11-46: Gold assay results, Blank material

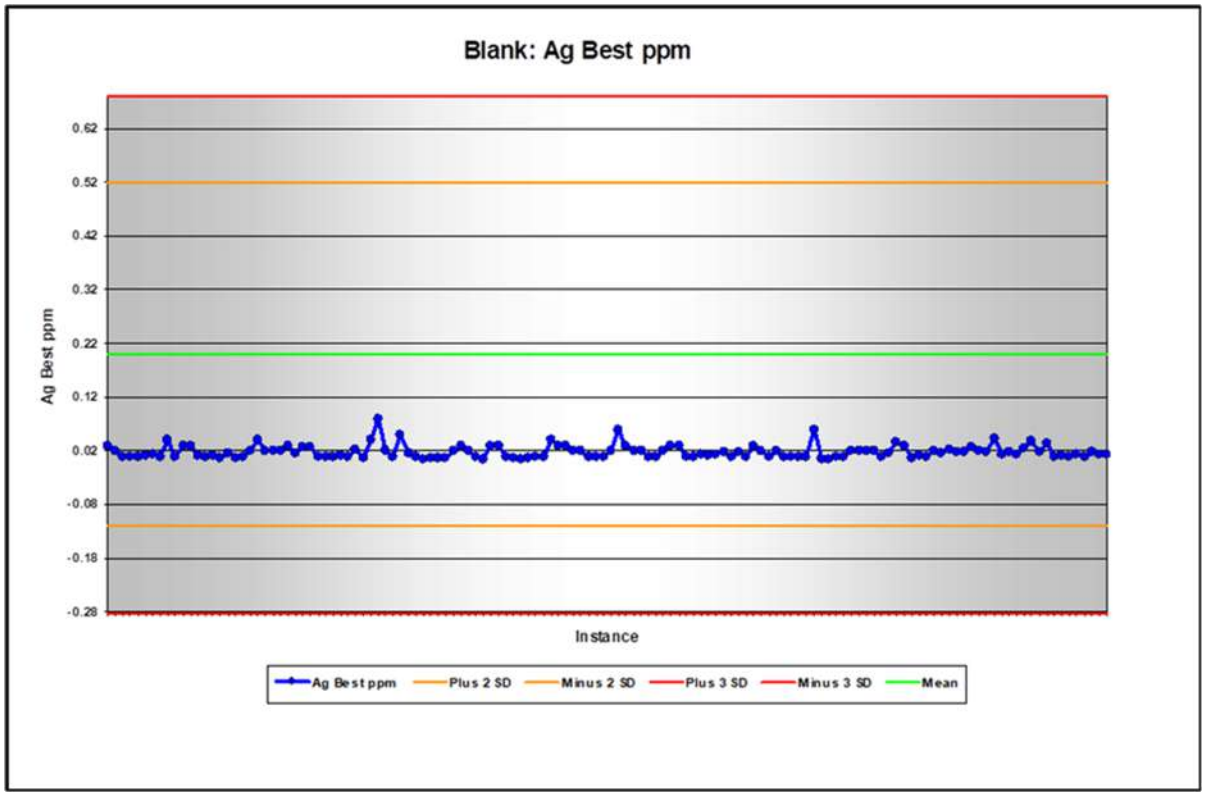


Figure 11-47: Silver assay results, Blank material

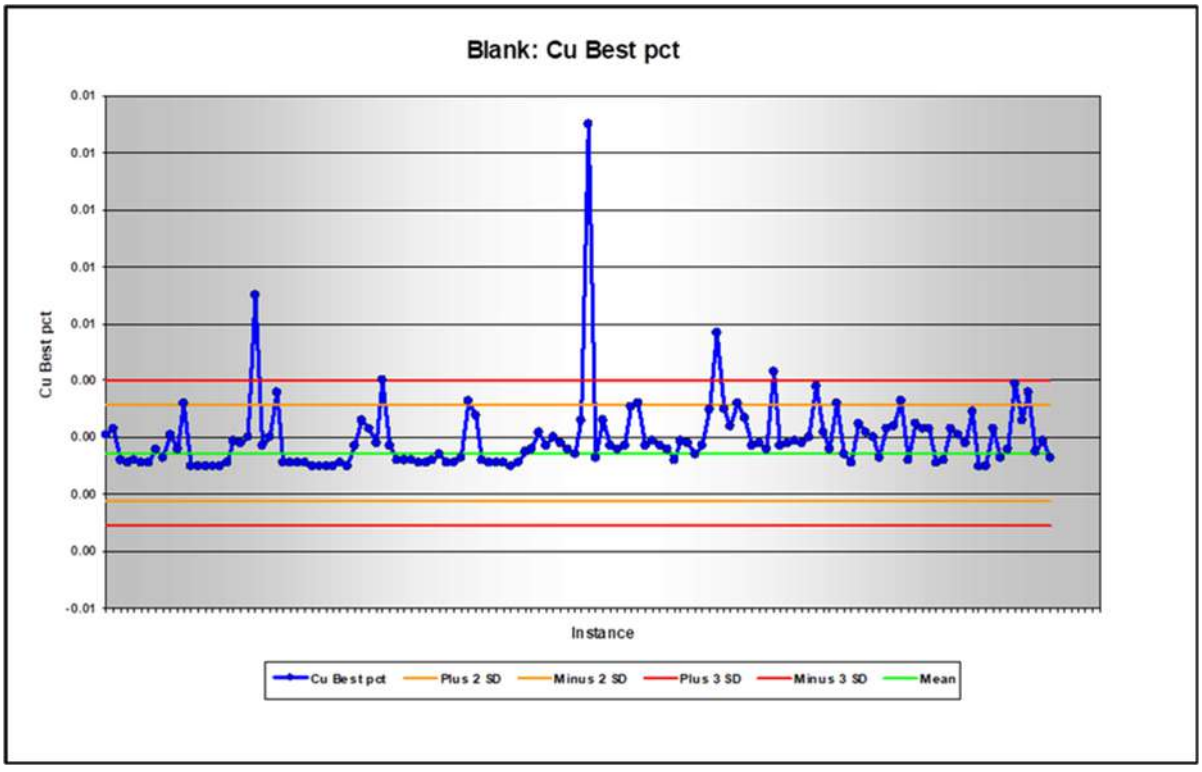


Figure 11-48: Copper assay results, Blank material

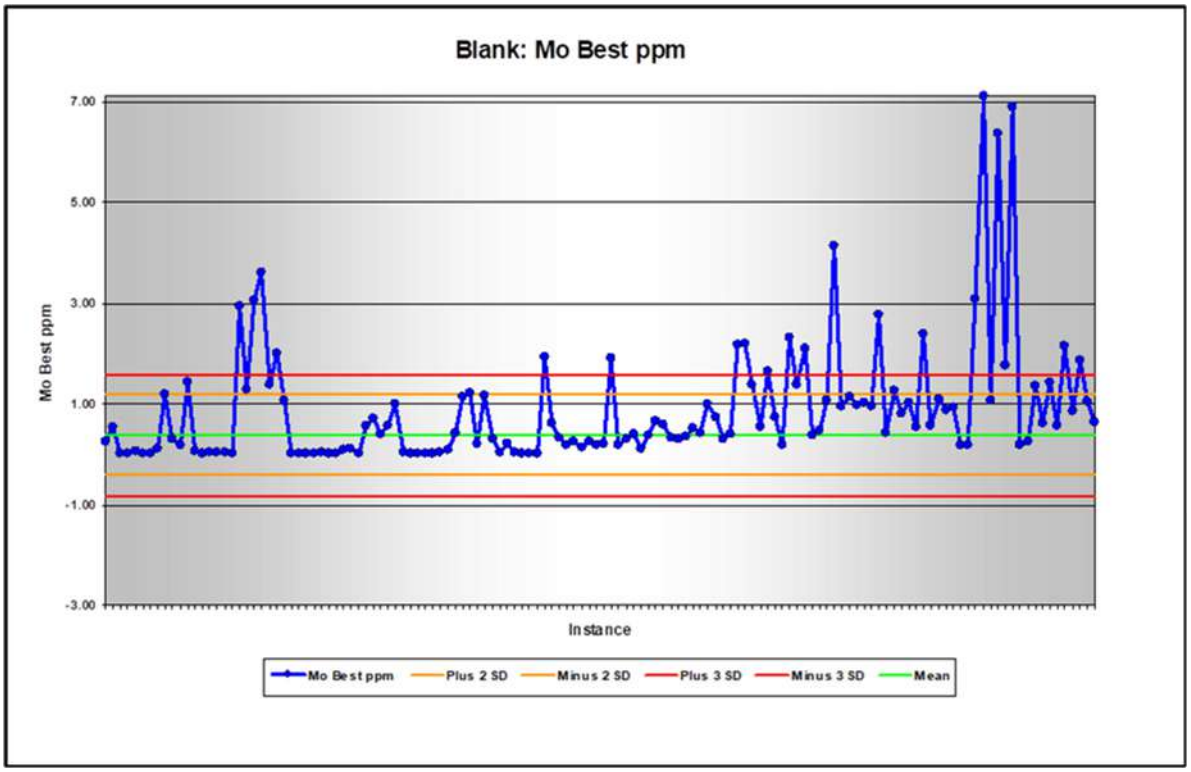


Figure 11-49: Molybdenum assay results, Blank material

Analysis of blank material in 2021 returned satisfactory results in 2021 but performed worse overall than in previous years. All elements had higher than 75% passing rates within three standard deviations of the mean expected values. On average, gold values plotted well above the expected mean, but, as Figure 11-46 shows, the detection limit for gold at ALS limits the lowest assay value to 0.0025 ppm, which is above the expected mean of 0.002 ppm for the blank material. Silver plotted well below the expected mean overall, but within acceptable limits. Molybdenum and copper both plotted above their expected means on average. These variances indicate the landscape aggregate should be sent for another round-robin analysis to determine whether the constituents have changed since 2019.

Several groups of failures from particular certificates have been requested to be re-run, but at the time of this report, those results have yet to be finalized. Table 11-15 lists the performance of blank sampling in 2021.

Table 11-15: Performance of Blank material during 2021 Drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	33	75.4	23	82.8
Ag	0	100	0	100
Cu	15	88.8	4	97
Mo	31	76.9	21	84.3

11.5.3 Field Duplicates, 2020

11.5.3.1 2008 through 2010

Field core duplicates are separate samples taken in the same manner and at the same core interval as the original sample. They are utilized to measure inherent variability in metal content from a single location and sample medium, and to provide an indication of sample reproducibility in the field. Field core duplicates were collected from the half-core that remained following the collection of the original sample. The duplicate was collected by sawing the half-core in half longitudinally, so that one quarter of the original core was collected. Duplicates were collected nominally for every 20th sample. Where duplicates were collected, only one quarter of the core remains stored in the core box on the property.

In 2008, 21 core duplicate pairs were collected; in 2009, 199 core duplicate pairs were collected; in 2010, 245 core duplicate pairs were collected. The original half-core samples were shipped to ALS Chemex and assayed for gold, copper, and molybdenum, as well as multi-element ICP analysis as described above. The duplicate quarter-core samples were shipped to Acme Labs for gold, copper, and molybdenum assay, as well as multi-element ICP analysis in a manner identical to that performed at ALS Chemex, as described above. The results for the duplicate analyses for gold, silver, copper, and molybdenum are demonstrated in comparison plots between the Acme and ALS Chemex values below (Figure 11-50 through Figure 11-53).

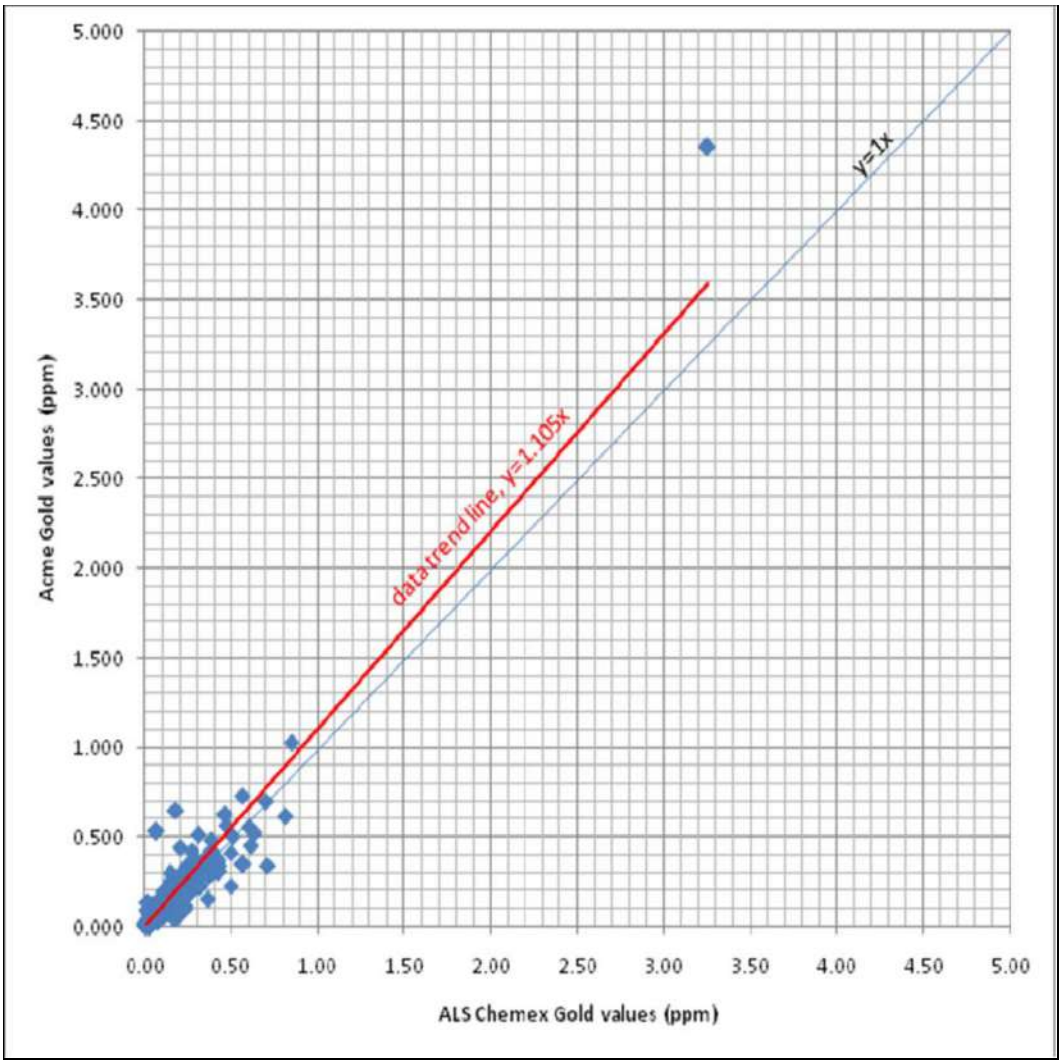


Figure 11-50: Plot of ALS Chemex Au assay vs Acme Labs Au assay for Field Duplicate Samples (2008, 2009 and 2010 samples)

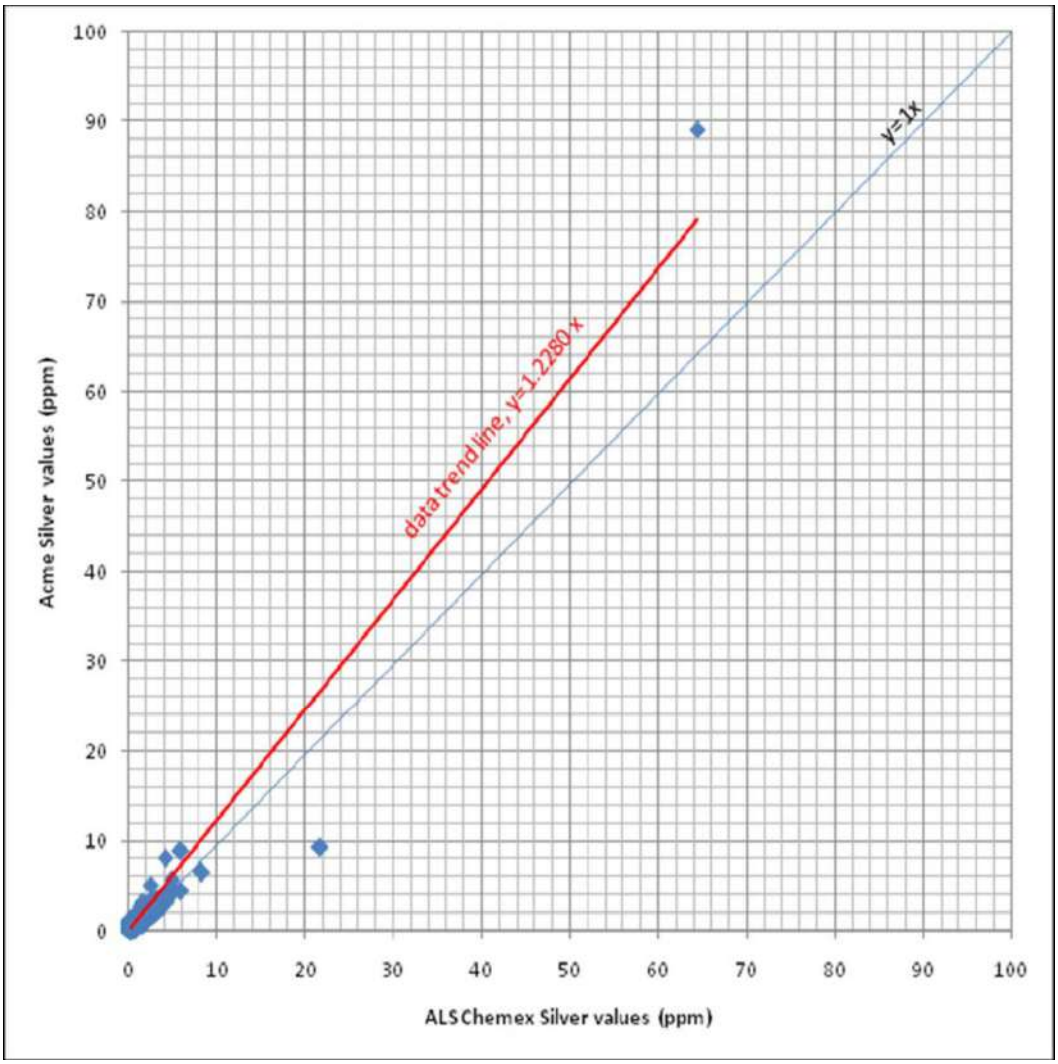


Figure 11-51: Plot of ALS Chemex Ag analyses vs. Acme Labs Ag Analysis for Field Duplicate Samples (2008, 2009, 2010 samples)

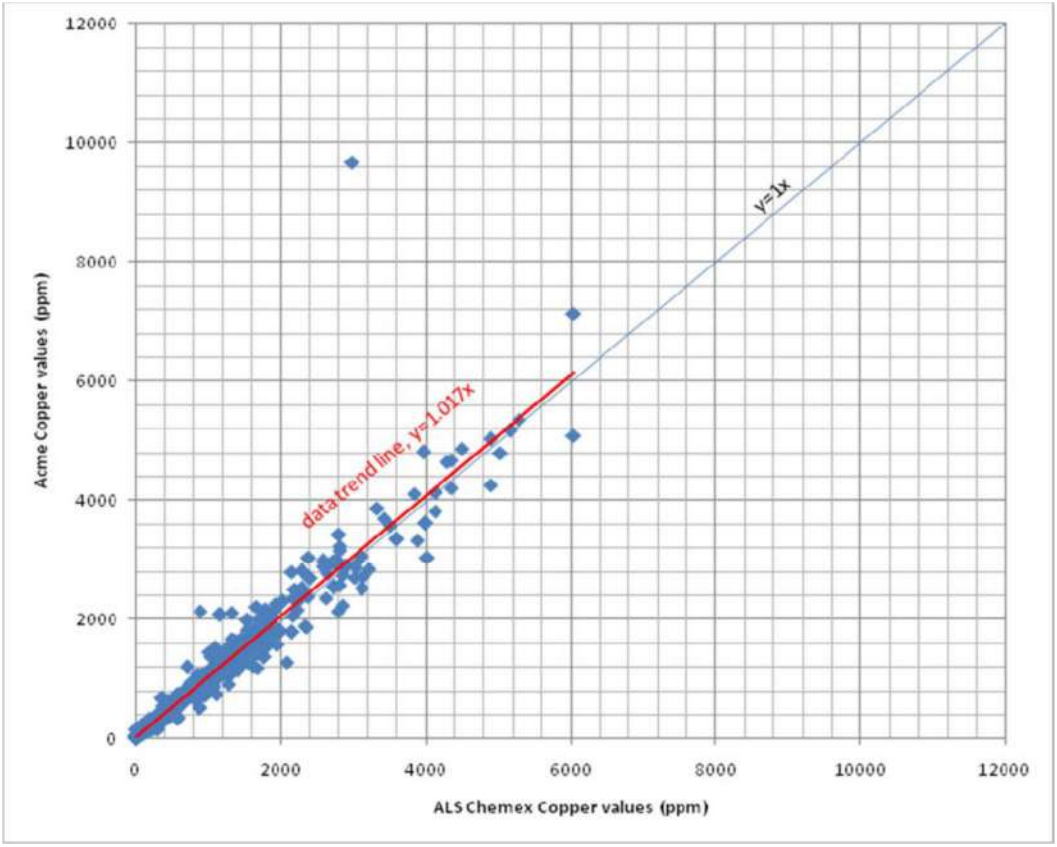


Figure 11-52: Plot of ALS Chemex Cu Assay vs. Acme Labs Cu Assay for Field Duplicate Samples (2008, 2009 and 2010 Data)

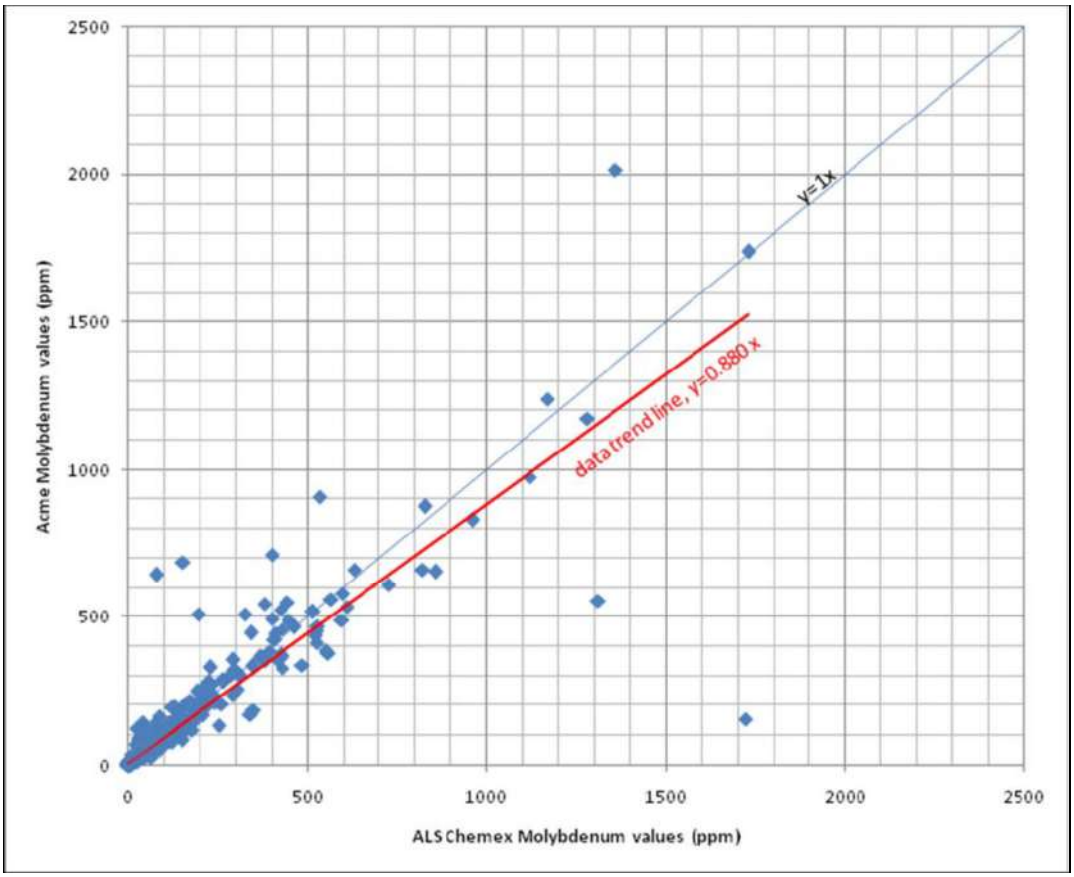


Figure 11-53: Plot of ALS Chemex Mo assay vs Acme Labs Mo assay for Field Duplicate Samples (2008, 2009 and 2010 Data)

The plots show generally good correlation between ALS Chemex and Acme Labs for all four elements of interest.

Often the "nugget effect" associated with gold and silver content will produce widely divergent values, which would plot as highly scattered data points. However, the gold and silver results from the duplicate samples show good correlation.

Ideally, a trend line of y=1x would show perfect reproducibility. This is rarely, if ever, the case due to the difference of mineral content between duplicate samples. The data trend line for gold returned y=1.105x. This demonstrates that Acme Lab results, as a whole, are 10.5% higher than ALS Chemex results. All samples cluster in close proximity to the trend line which indicates no strong "nugget effect" and good reproducibility.

The data trend line for silver is y=1.228x. This demonstrates that Acme Lab analytical results, as a whole, are 22.8% higher than ALS Chemex values. In general, the points cluster well around the trend line with the exception of one sample, also demonstrating good reproducibility.

The results for duplicate analyses for copper demonstrate excellent reproducibility. The data trend line returned y=1.017x. The copper data clusters tightly around trend line with the exception of one value. In general, the Acme results are very slightly higher (1.7%) than the ALS Chemex results.

The molybdenum plot demonstrates slightly more scattered results with 8 points plotting far off the trend line (y=0.880x). The trend line indicates that, in general, the Acme results for molybdenum are 12% lower than ALS Chemex results. Overall, the duplicate results show good correlation. Molybdenite mineralization was observed in quartz veins in the

drill core and it is possible that the 8 erratic values are reflecting a molybdenum "nugget effect", where there is a variability of molybdenite concentration between samples.

The results of analyses from the sample standards, blanks and duplicates provide for acceptable Quality Assurance and Quality Control (QA/QC) for the geochemical programs at Casino from 2008 through 2010. The results also indicate that there is no evidence of tampering during the sample collection process, shipping or at the laboratory. There is also no evidence of systemic errors in the sample preparation and analytical processes.

11.5.3.2 Field Duplicates 2019

In 2019, insertion of both field duplicates and pulp check duplicates were part of the overall sampling protocol at Casino.

Field Duplicates

Similar to standards and blanks, 1 field duplicate was inserted randomly within every 20 samples. The duplicate was quarter cored by the core cutter from the original sample and placed in a separate bag with its own sample tag number. This duplicate quarter-core sample would be set aside in a bin to be sent to ALS Global for analysis in a separate batch at a later date than its corresponding original sample. The purpose of this kind of duplicate sample is to test the reproducibility of the lab's analytical methods.

Figure 11-54 through Figure 11-57 show the comparison between the original core sample results and the duplicate core sample results for gold, silver, copper, and molybdenum.

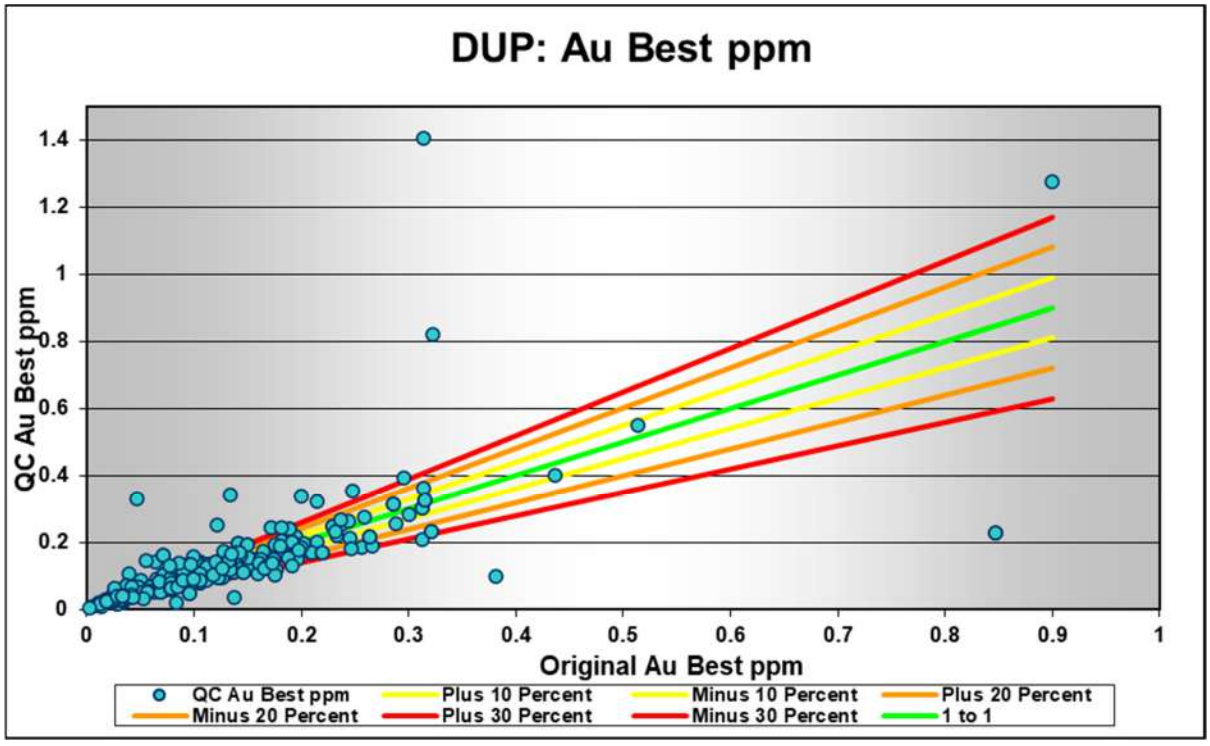


Figure 11-54: Comparison Plot between Original Gold Values and Duplicate Gold Values

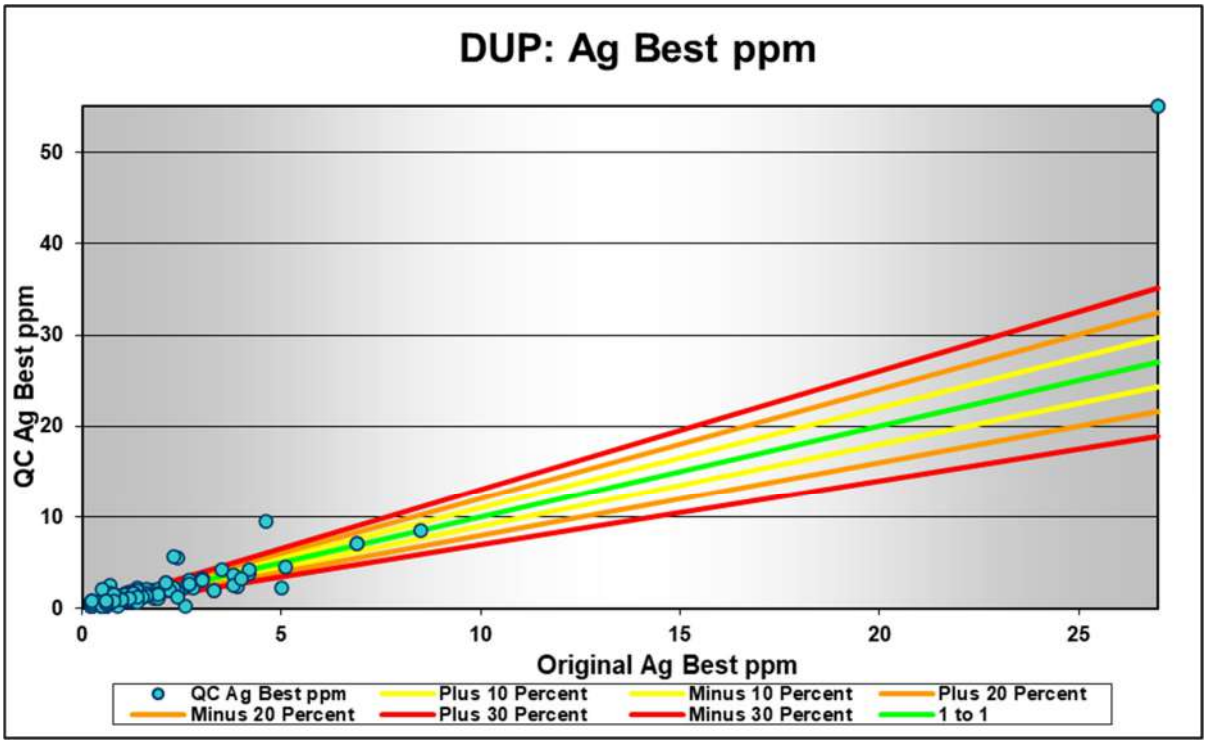


Figure 11-55: Comparison Plot between Original Silver Values and Duplicate Silver Values

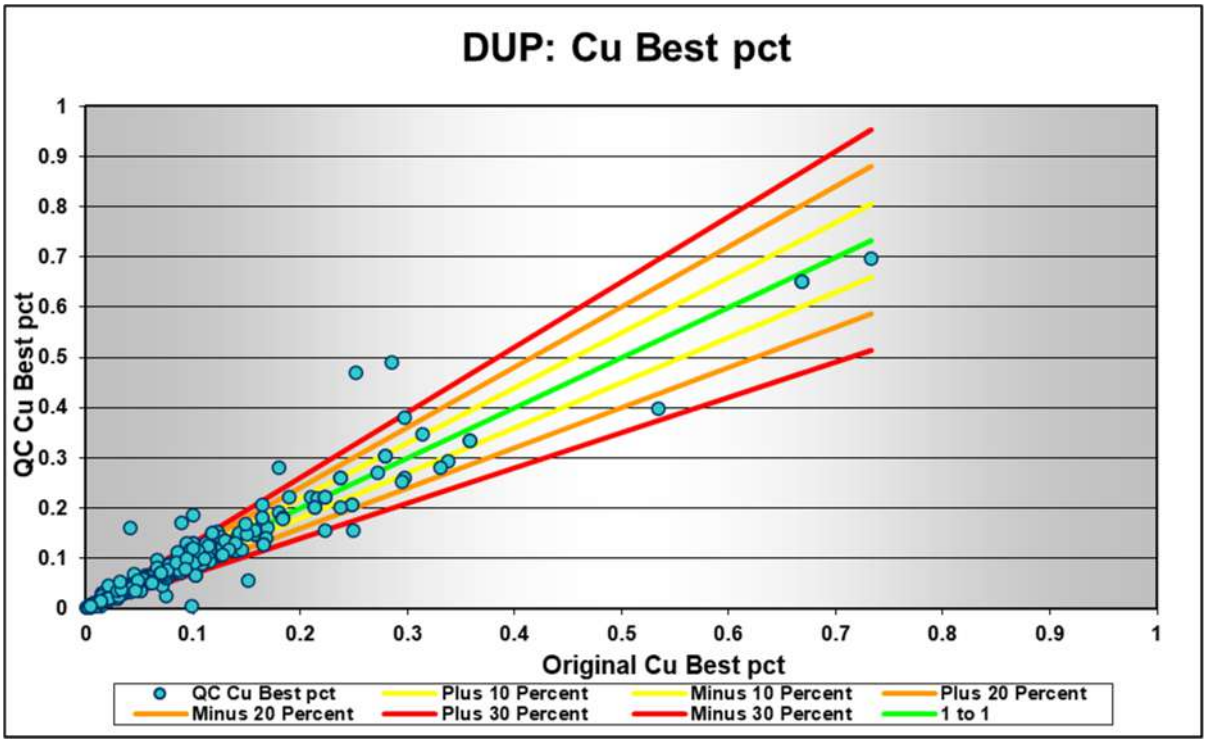


Figure 11-56: Comparison Plot between Original Copper Values and Duplicate Copper Values

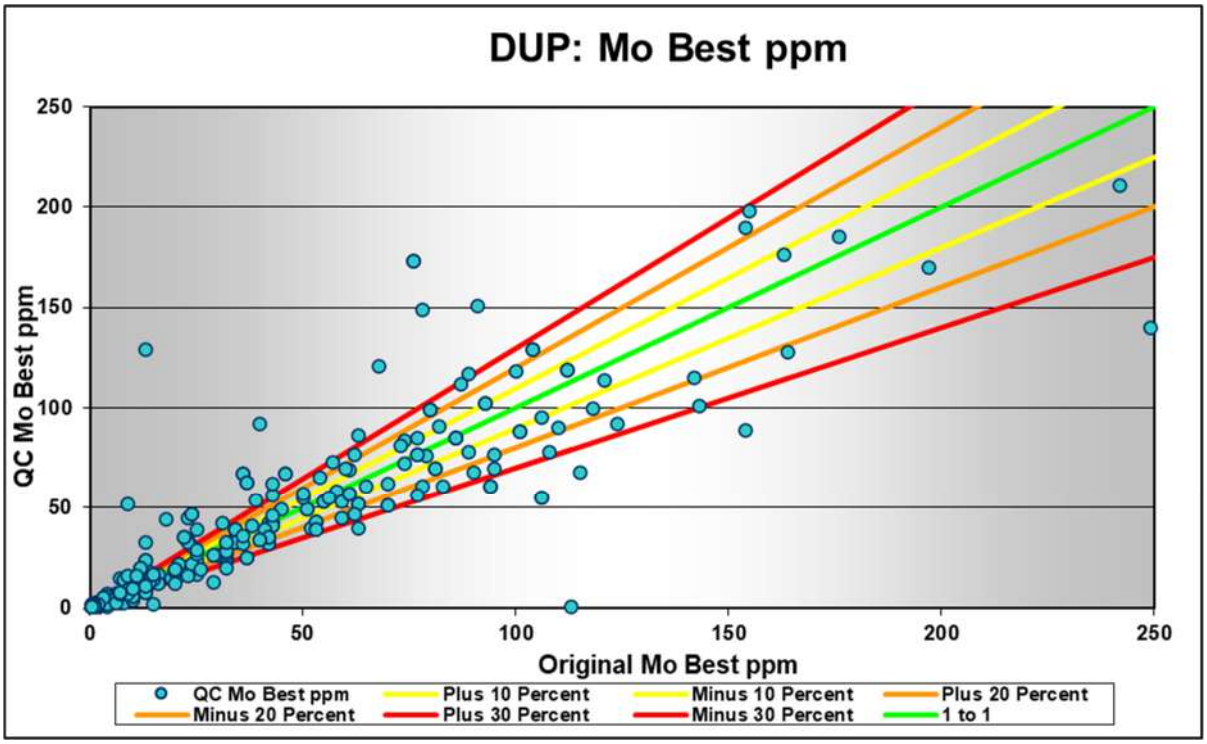


Figure 11-57: Comparison Plot between Original Copper Values and Duplicate Copper Values

Field duplicates for 2019 performed well, but not without issues (Table 11-16). The problem with field duplicates in this type of deposit is the difficulty to accurately cut a piece of core into two identical quarters.

Table 11-16: Summary of Duplicate (Core) Pair Performance during 2019 Drill Program Sampling

Element	Duplicate Pairs Within 10% Difference	% total pairs within 10%	Duplicate Pairs Within 20% Difference	% total pairs within 20%	Duplicate Pairs Within 30% Difference	% total Duplicates within 30%
Au	120	34.9	211	61.3	269	78.2
Ag	141	41	214	62.2	240	69.8
Cu	181	52.6	276	80.2	308	89.5
Mo	109	31.7	166	48.3	215	62.5

Check Duplicates

Check samples were selected at random from the entire sample population once the primary lab, ALS Global, had reported all the final assay results for the 2019 Casino Project. A list of 973 sample numbers (using a random selection in Excel) was sent to ALS Global in Whitehorse from the project manager/senior geologist, requesting ALS to pull the pulps for the samples listed and send them directly to SGS Canada Inc. in Burnaby, BC for processing. This represents a little over 20% of the entire 2019 sample population. Once received by SGS, these pulps were logged into their system, re-homogenized non-mechanically, then dry-screened randomly (1/100 samples were checked) to various mesh sizes to verify fineness. No major issues were found regarding fineness, and SGS proceeded with the full assay protocol.

The purpose of this kind of duplicate/check is to test the methodology of the primary lab to ensure there is no bias or systemic errors, and that other labs using similar methods can reproduce their results within a predetermined degree of variance.

Figure 11-58 through Figure 11-61 show the comparison between the original core sample results from ALS Global in Whitehorse and the check pulp sample results from SGS in Burnaby for gold, silver, copper, and molybdenum.

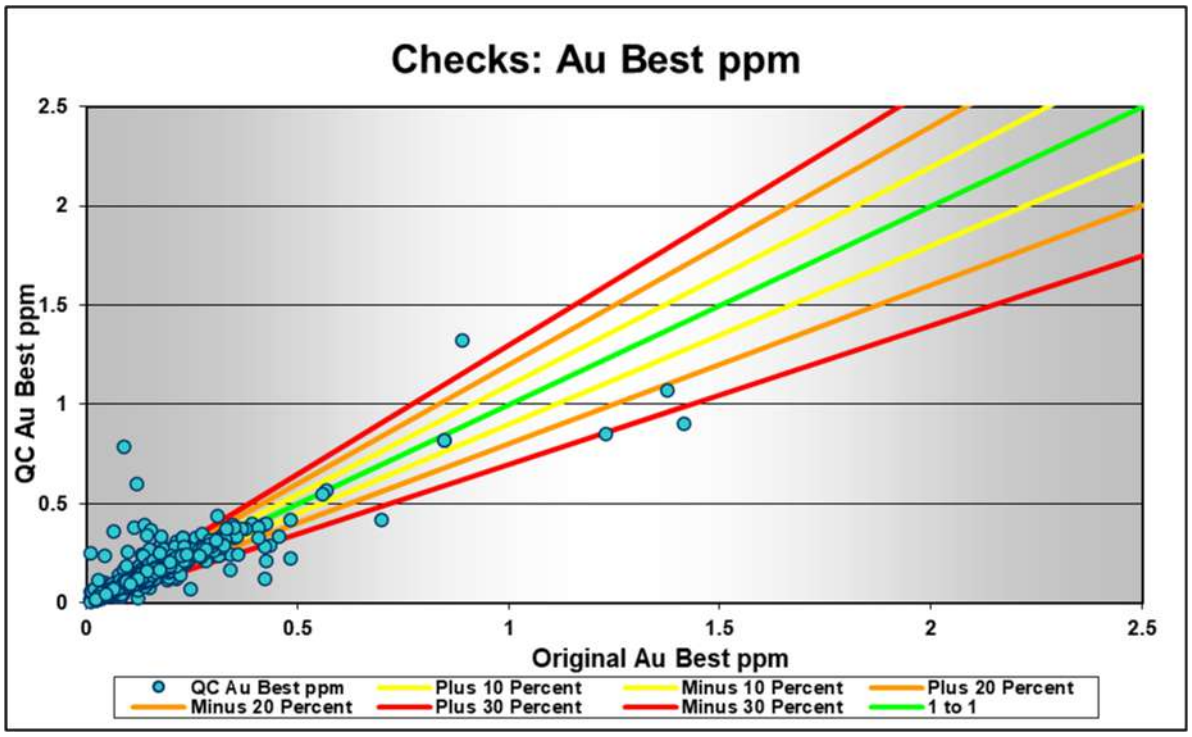


Figure 11-58: Comparison Plot between Gold Values from ALS Global and Gold Values from SGS

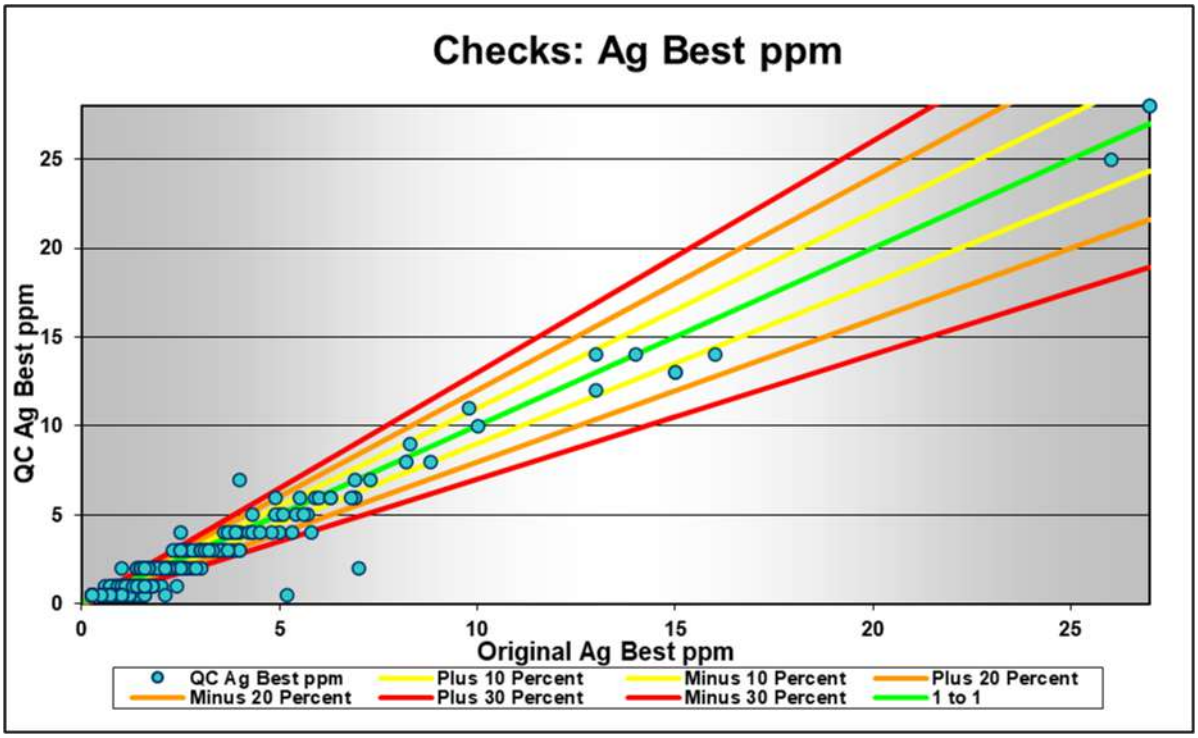


Figure 11-59: Comparison Plot between Silver Values from ALS Global and Silver Values from SGS

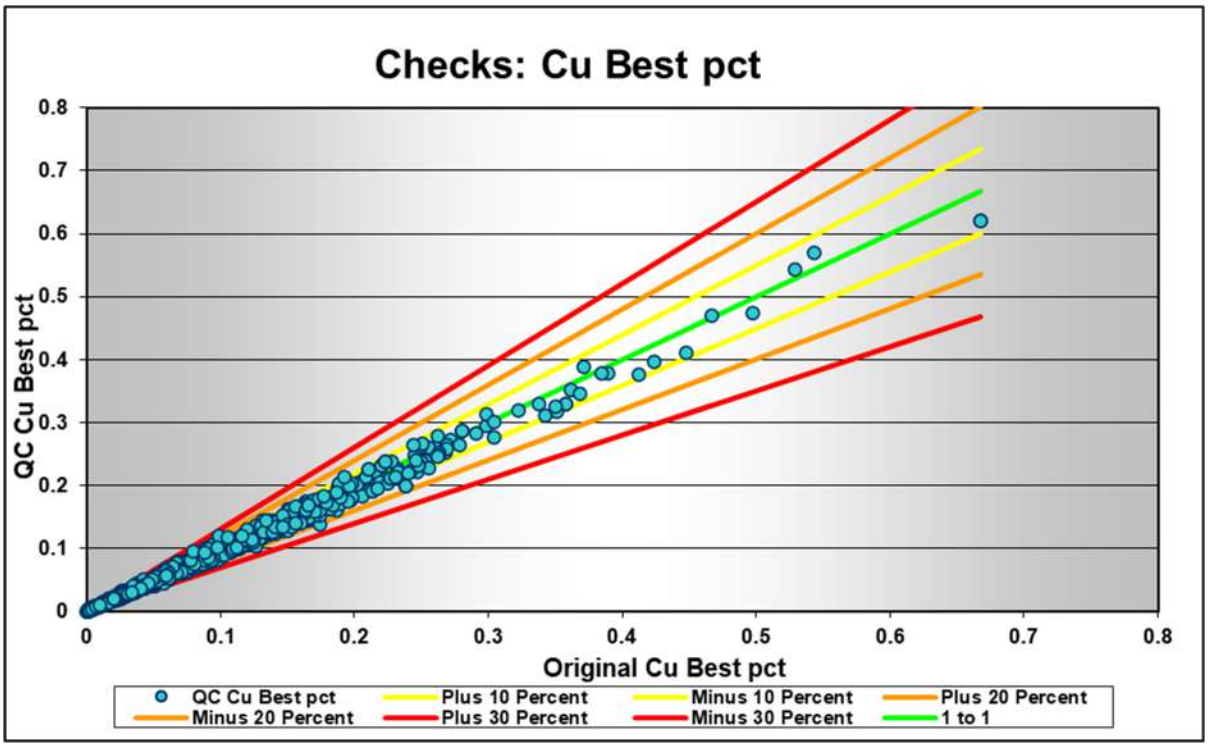


Figure 11-60: Comparison Plot between Copper Values from ALS Global and Copper Values from SGS

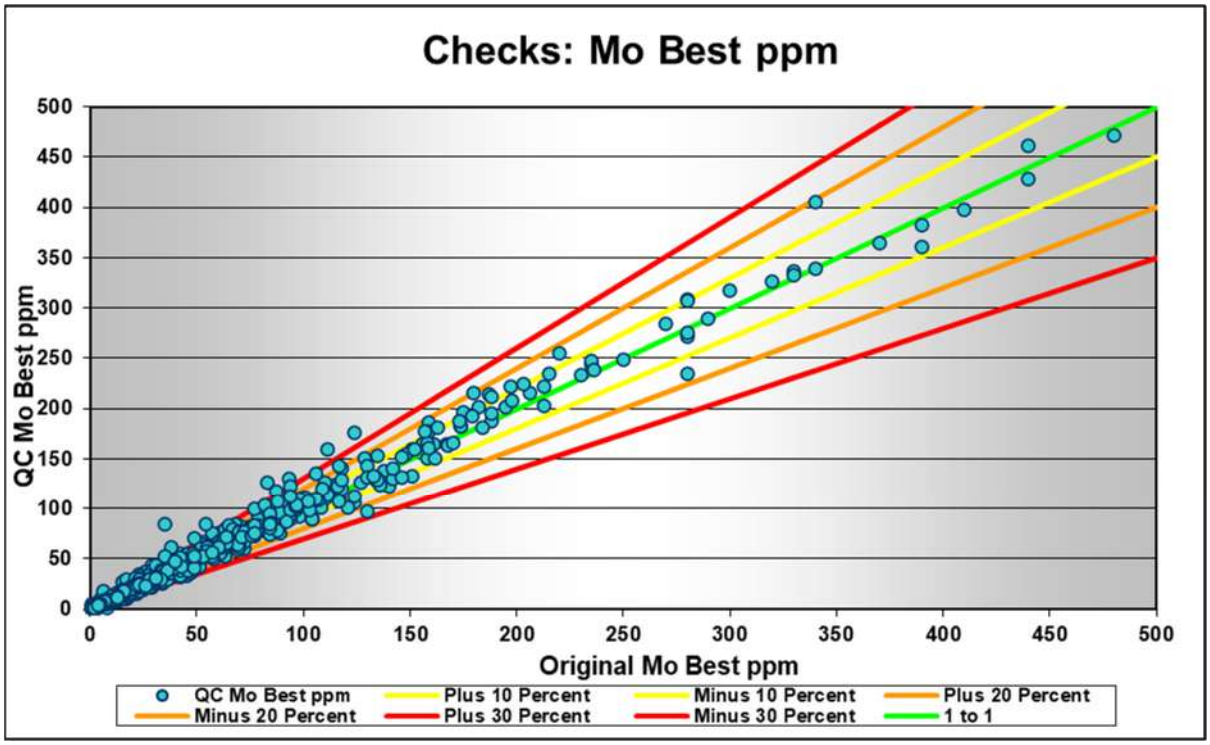


Figure 11-61: Comparison Plot between Molybdenum Values from ALS Global and Molybdenum Values from SGS

Check samples for 2019 performed well, better than most duplicates overall, but still not without issue (Table 11-17). In general, gold and silver showed lower correlation between pairs of pulps analyzed for all elements of interest than did copper and molybdenum. Silver showed the poorest correlation with less than 20% of samples having variation under 10%. The number of 2019 check samples (>20% of total) sent to the secondary lab represented a much larger population of samples than is industry practice, which is about 5%. The high percentage delivered in 2019 was partially due to utilization of a new laboratory for check samples, and to prove past success at these rates.

Table 11-17: Summary of Check (Pulps) Pair performance during 2019 Drill Program Sampling

Element	Check Pairs Within 10% Difference	% total pairs within 10%	Check Pairs Within 20% Difference	% total pairs within 20%	Check Pairs Within 30% Difference	% total Duplicates within 30%
Au	465	47.8	711	73.1	817	84
Ag	187	19.2	351	36	469	48.2
Cu	692	71.1	953	97.9	961	98.8
Mo	596	61.3	757	77.8	812	83.5

11.5.3.3 Field Duplicates, 2020

In 2020, both field duplicates and pulp check duplicates comprised part of the overall sampling protocol at Casino.

Field Duplicates

Similarly to standards and blanks, 1 field duplicate was inserted randomly within every 20 samples, equating to 252 duplicate samples in 2020. The duplicate sample was quarter cored by the core cutter and placed in a separate bag, along with its own sample tag, from the original sample. This duplicate quarter-core sample was set aside in a bin to be sent to ALS for analysis in a separate batch that was sent at a later date than its corresponding original sample. The purpose of this is to test for the uniformity of metal content in the core, and for the reproducibility of the lab's analytical methods.

Figure 11-62 through Figure 11-65 show the comparison between the original core sample results and the duplicate core sample results for gold, silver, copper, and molybdenum.

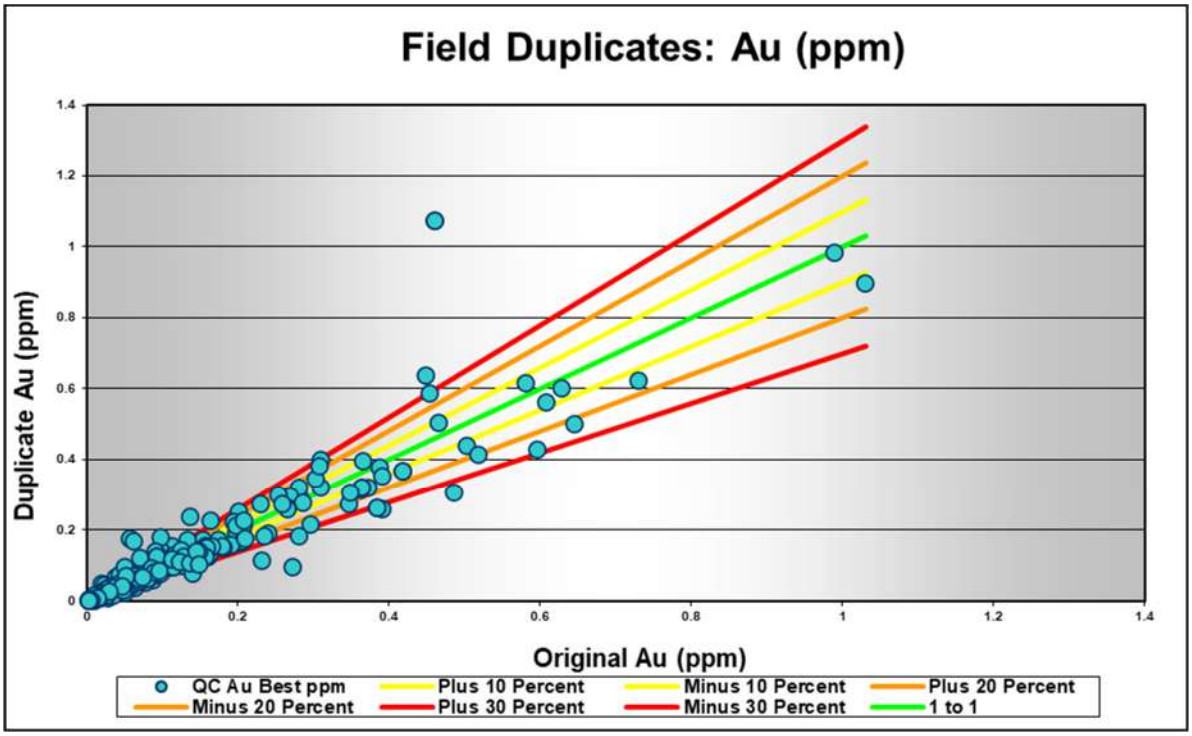


Figure 11-62: Comparison between Original and Duplicate Gold Values

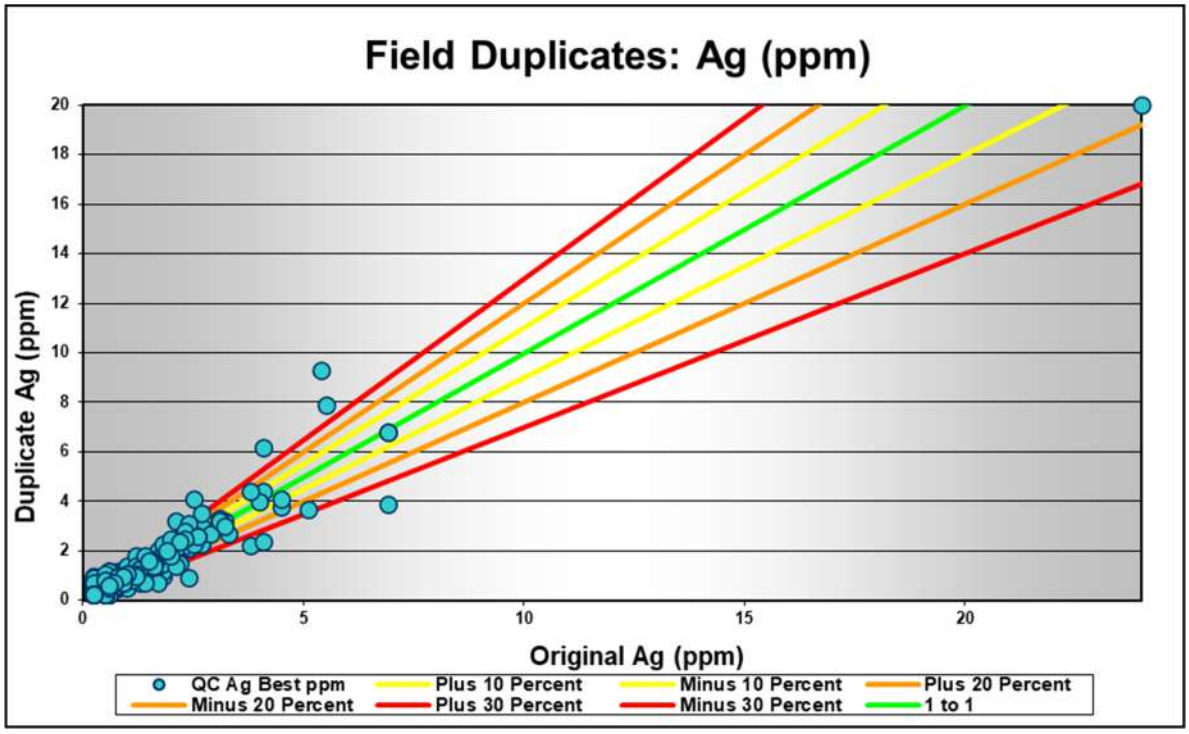


Figure 11-63: Comparison between Original and Duplicate Silver Values

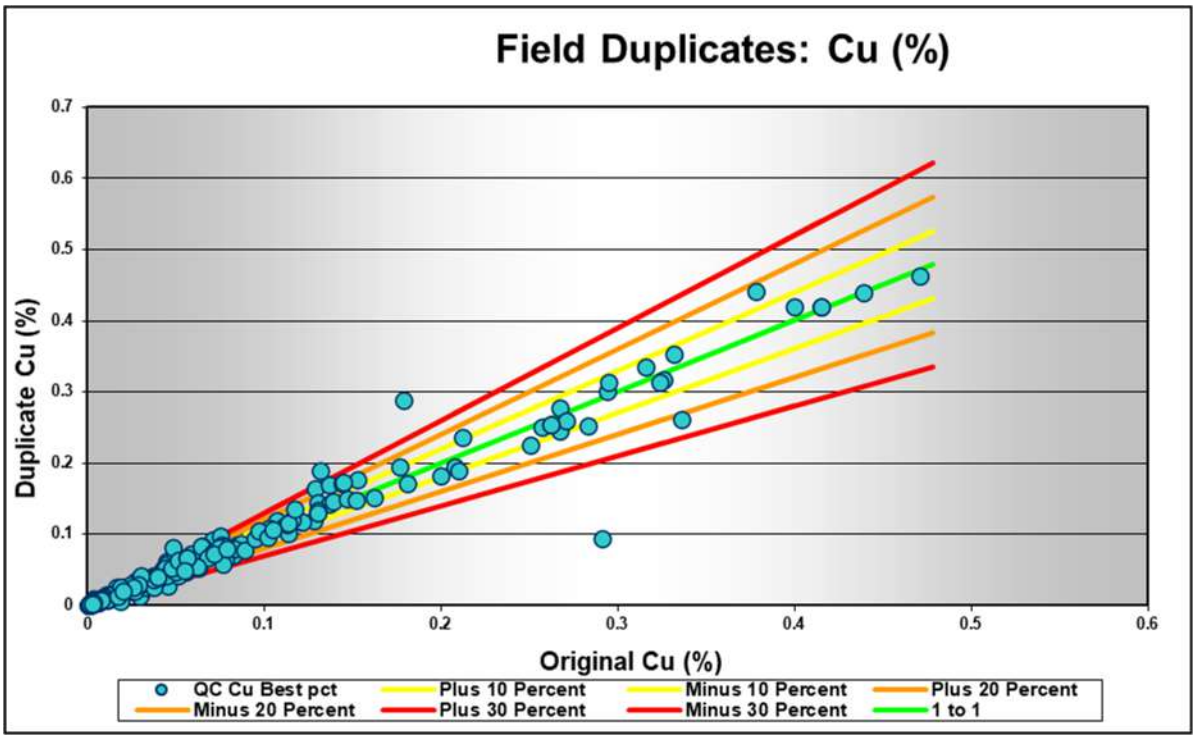


Figure 11-64: Comparison between Original and Duplicate Copper Values

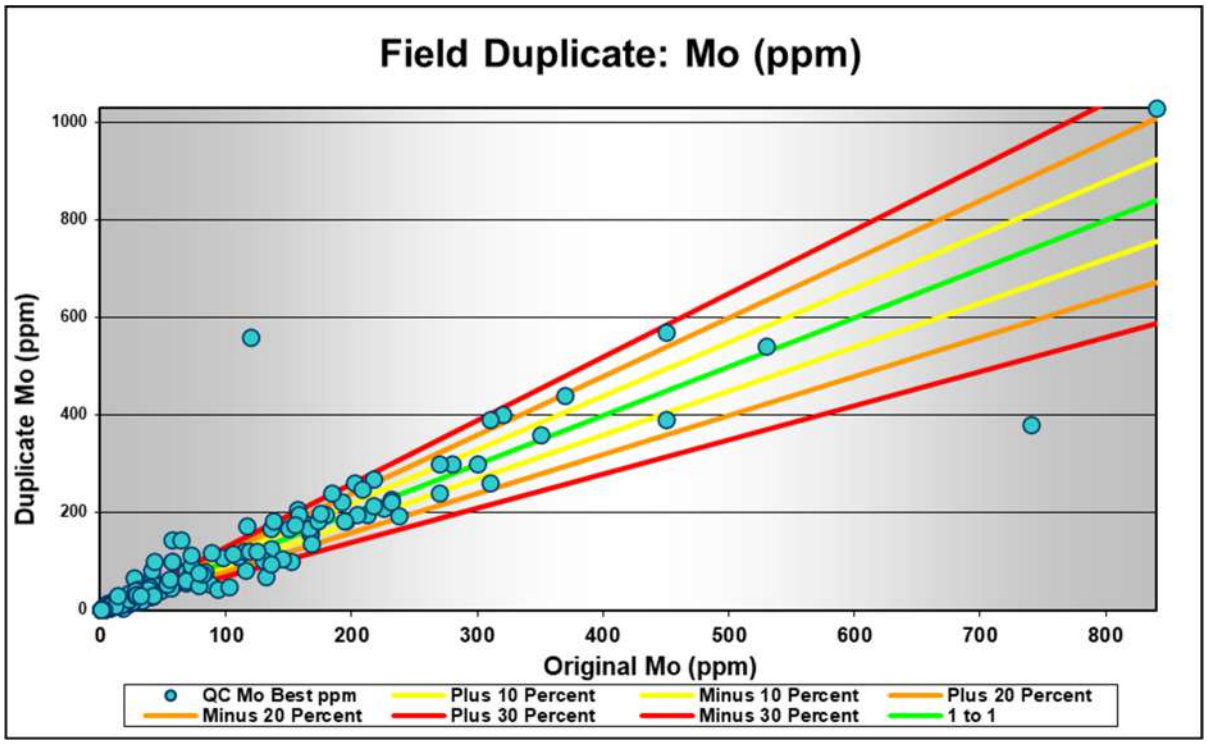


Figure 11-65: Comparison between Original and Duplicate Molybdenum Values

Field duplicates for 2020 performed well, but not excellently. The issue with field duplicates in this type of deposit is the difficulty to accurately cut the core into identical "quarters". In this way a pulp duplicate would test the lab's reproducibility more effectively. Table 11-18 summarizes the duplicate performance for each element of interest in 2020.

Table 11-18: Summary of Dupluicate (core) Pair Performance during 2020 Drill Program

Element	Duplicate Pairs Within 10% Difference	% total pairs within 10%	Duplicate Pairs Within 20% Difference	% total pairs within 20%	Duplicate Pairs Within 30% Difference	% total Duplicates within 30%
Au	88	34.9	164	65.1	191	75.8
Ag	121	48	163	64.7	240	69.8
Cu	181	52.6	276	80.2	308	89.5
Mo	109	31.7	166	48.3	215	62.5

Some degree of "coarse gold (nugget) effect" is likely here. Both Au and Ag commonly occur as fine nuggets, as is evident by visible gold occurring in DDH 19-21. This may account for the lower reproducibility rates for these elements. Chalcocite veins, which are fairly common, are prone to uneven distribution within core duplicates. The low rates for Mo may also be due to the sub-centimetre vein-associated nature of molybdenite mineralization, particularly prone to uneven distribution within core duplicates (Schulze, 2021).

Check Pulp Duplicates

Check samples were selected at random from the entire sample population once the primary lab, ALS Global, had reported all the final assay results for the 2020 Casino Project. A list of 200 sample numbers (using a random selection process in Microsoft Excel) was sent to ALS requesting them to pull the pulps for the samples listed and send them directly to the Bureau Veritas Minerals (BV) lab in Vancouver, BC for analysis. This represents 5% of the entire 2020 sample population. Once received by BV, these pulps were logged into their system, re-homogenized non-mechanically, then dry-screened and sent through various mesh sizes on a random basis to verify fineness. No major issues were found regarding fineness and BV proceeded with the full assay protocol.

The purposes of these kind of check samples are to test the methodologies of the primary lab to ensure there are no biases or systemic errors, and to verify that other labs using similar methods can reproduce the results within a certain degree of variance.

Figure 11-66 through Figure 11-73 show the comparison between the original core sample results from ALS and the check pulp sample results from BV for gold, silver, copper, and molybdenum.

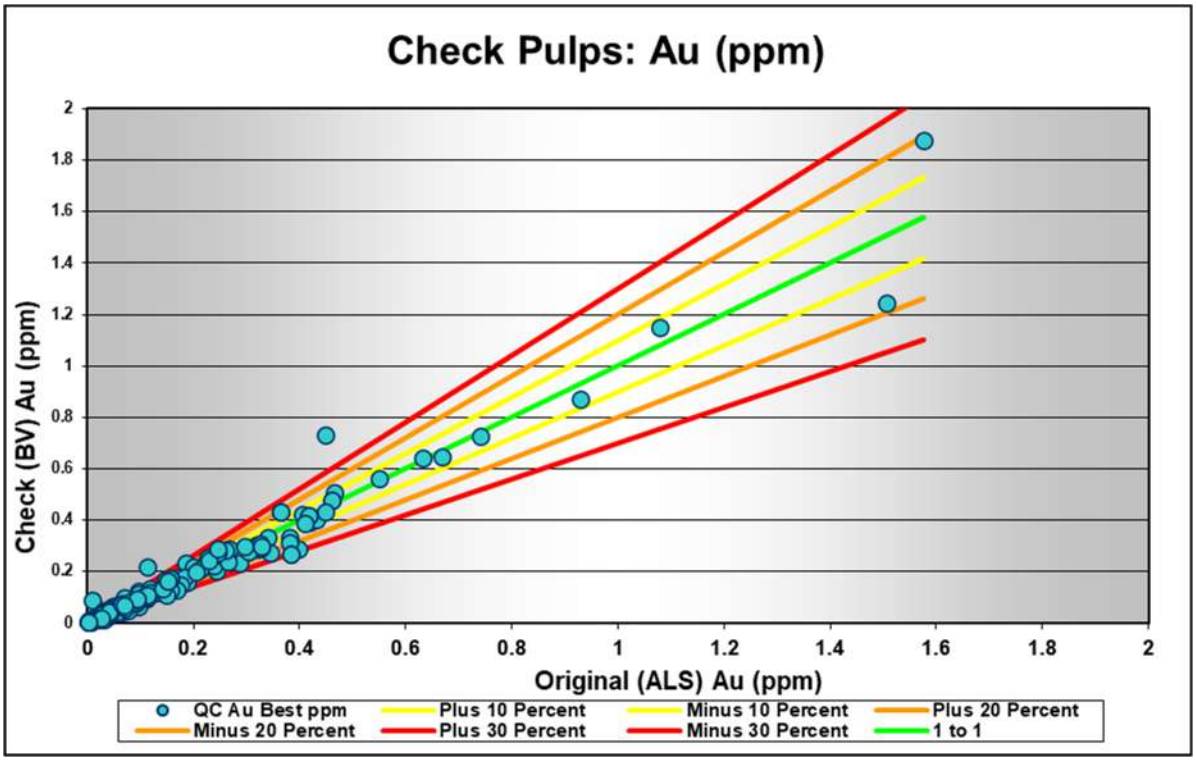


Figure 11-66: Comparison of Gold Values from ALS Global and Bureau Veritas

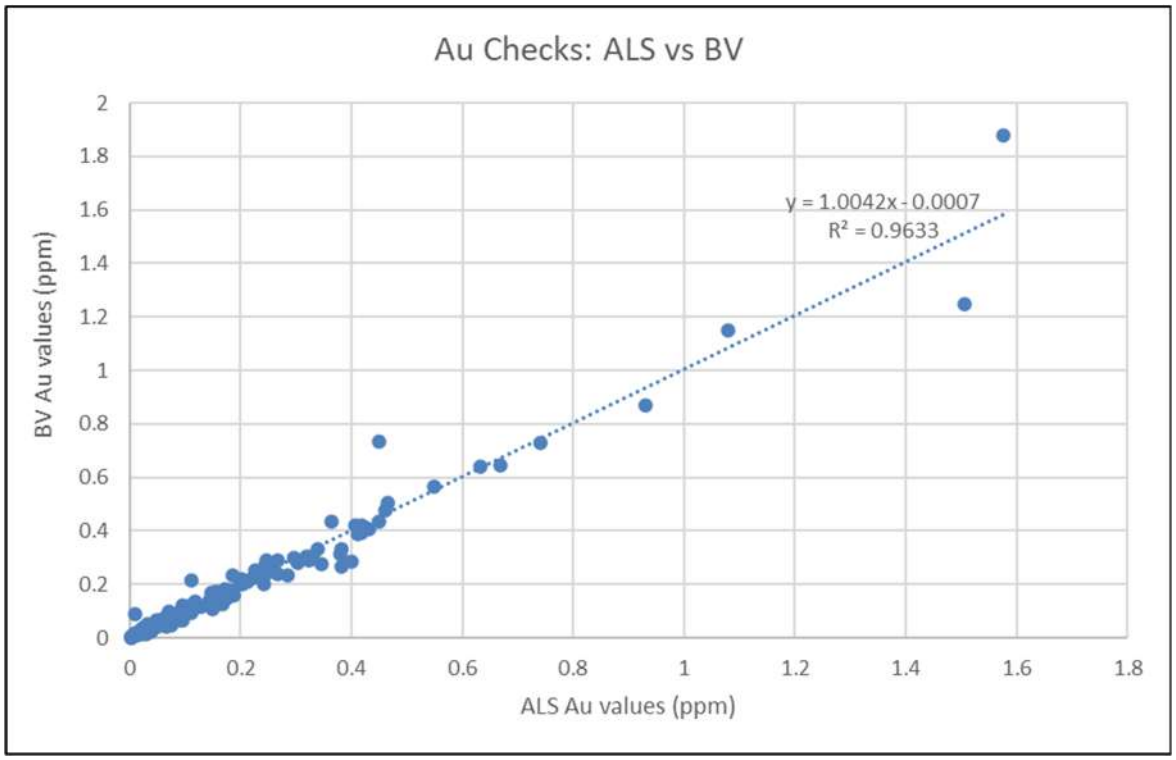


Figure 11-67: Correlation between Gold Values from ALS Global and Bureau Veritas

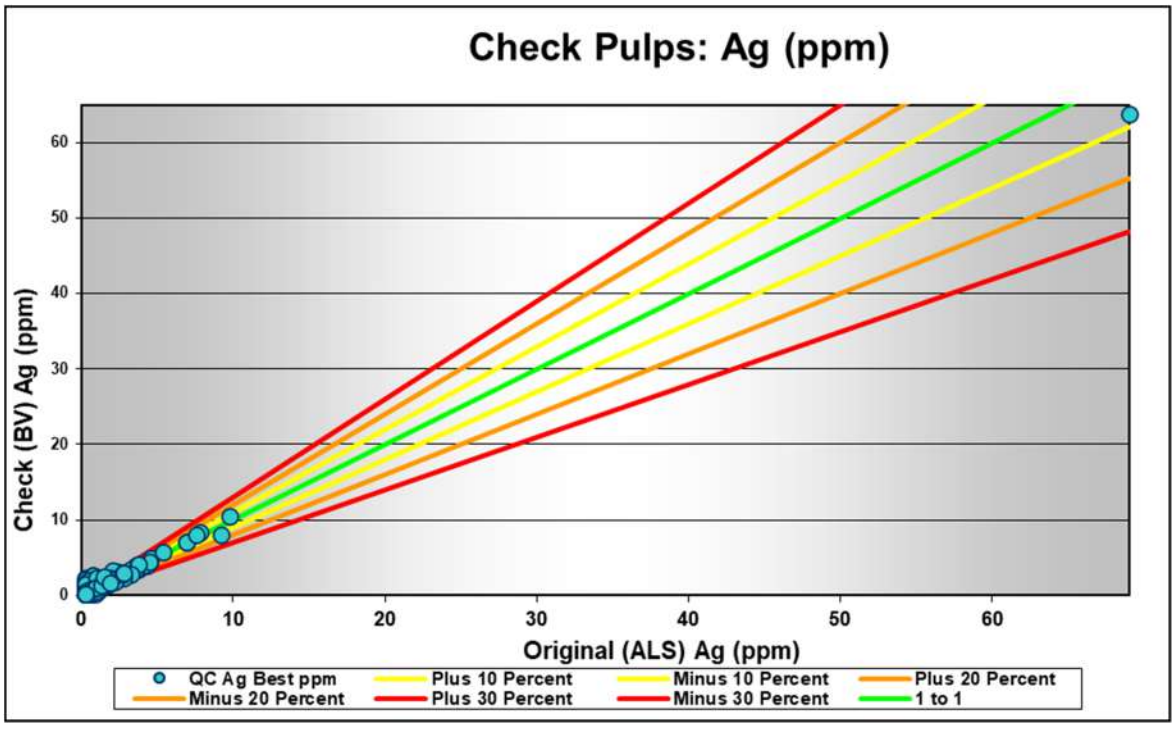


Figure 11-68: Comparison between Silver Values from ALS Global and Bureau Veritas

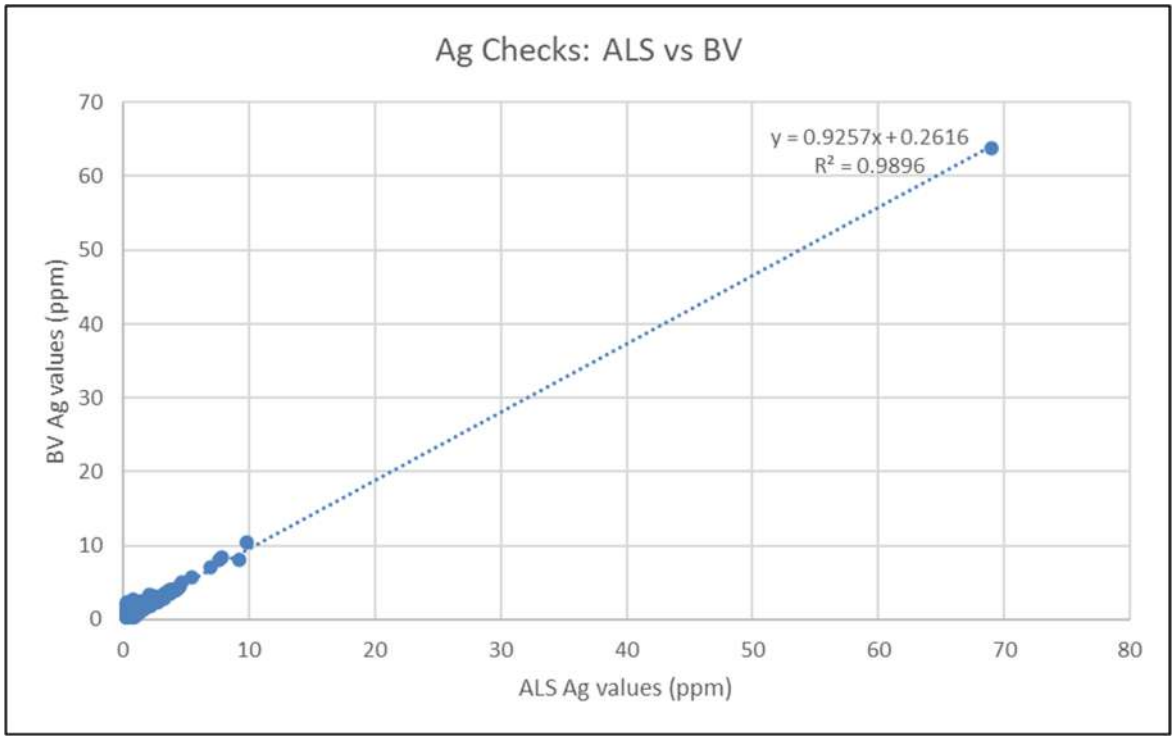


Figure 11-69: Correlation between Silver Values from ALS Global and Bureau Veritas

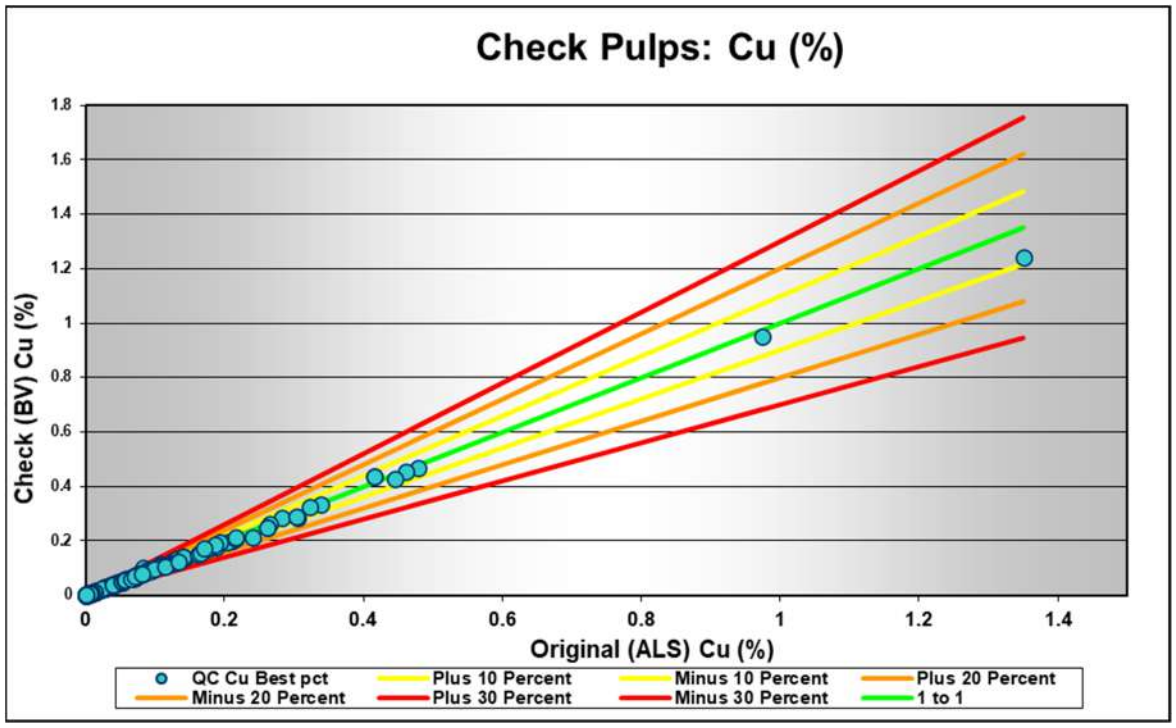


Figure 11-70: Comparison between Copper Values from ALS Global and Bureau Veritas

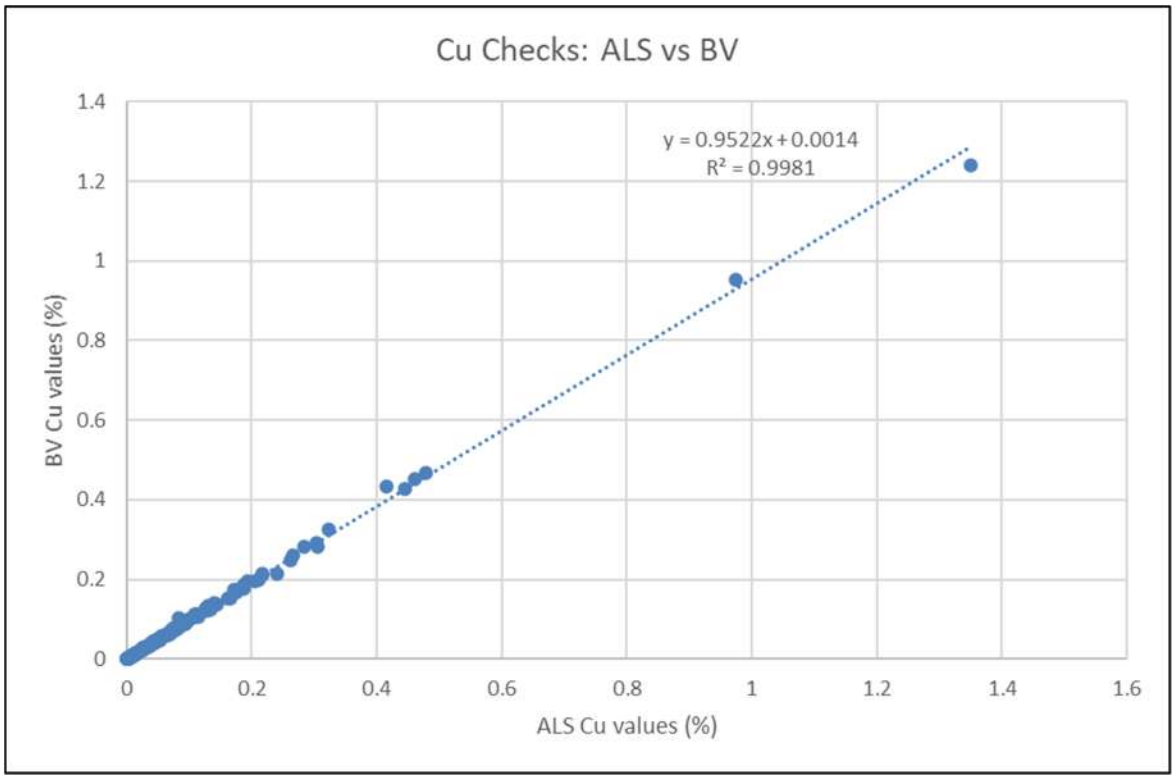


Figure 11-71: Correlation between Copper Values from ALS Global and Bureau Veritas

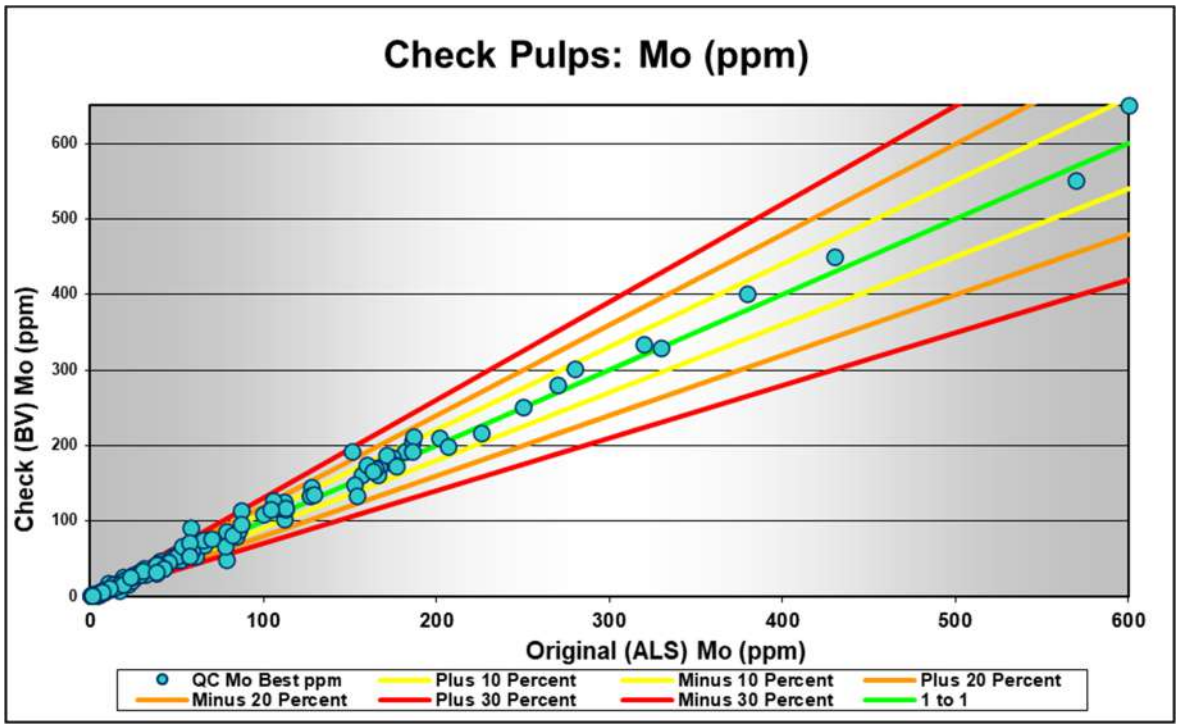


Figure 11-72: Comparison of Molybdenum Values from ALS Global and Bureau Veritas

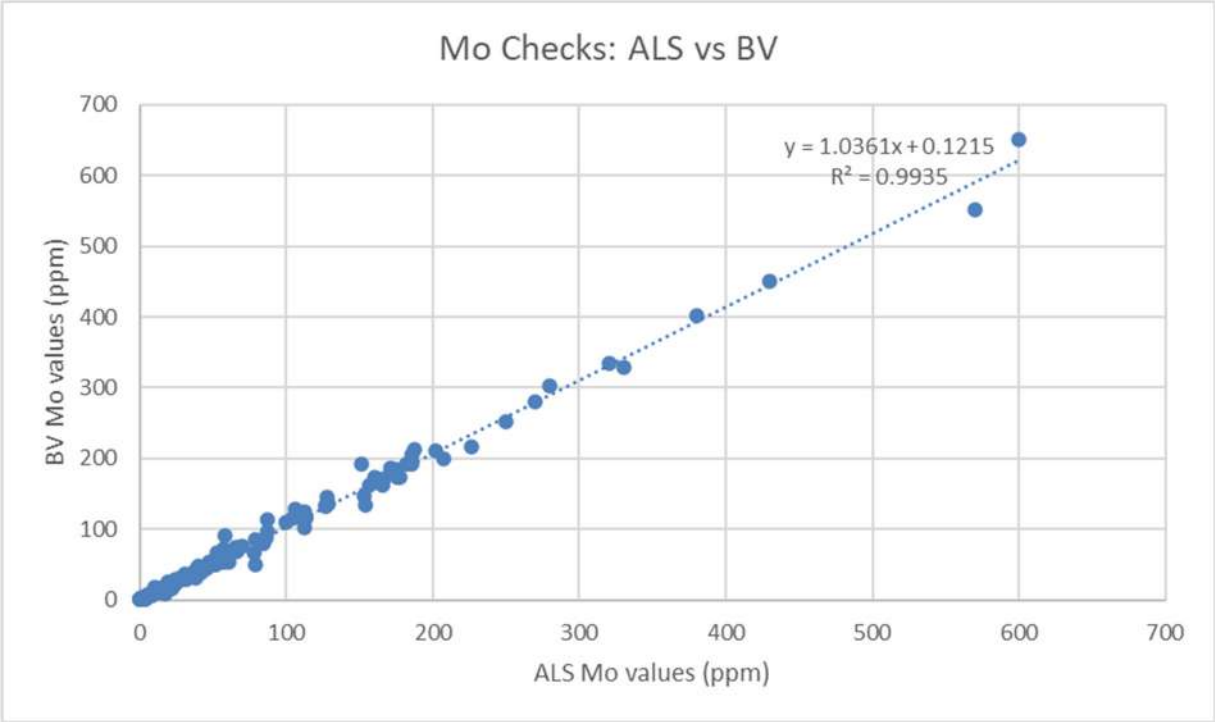


Figure 11-73: Correlation between Molybdenum Values from ALS Global and Bureau Veritas

Check Pulp Duplicates performed well in 2020 and better than Field Duplicates overall. Table 11-19 summarizes the percentages of check pairs that fall within 10%, 20% and 30% variation bands. Figure 11-67, Figure 11-69, Figure 11-71 and Figure 11-73 all show a good correlation between the pairs of check samples. Ideally, a trend line on these figures of y=1x would show perfect correlation and reproducibility, but this is rarely the case due to the differences in mineral content between duplicate samples.

Gold showed good correlation with a data trend line of y=1.0042x, demonstrating that gold results from BV are only 0.4% higher than ALS Global results. The data trend line for silver is y=0.9257x, demonstrating that BV results for silver are 7.4% lower than ALS results. For copper the data trend line is y=0.9522x, which indicates that BV results for copper are about 4.8% lower than ALS. Finally, for molybdenum, the data trend line is y=1.0361x, showing that BV results are 3.6% higher than ALS.

A "nugget effect" associated with gold and silver content can often produce widely divergent values, plotting as highly scattered data points. Aside from a small number of gold pairs, gold and silver results show good correlation, indicating pulp material was well-mixed by the labs during the preparation stage, and no strong "nugget effect" occurs within the samples.

Table 11-19: Summary of Check (Pulps) Pair Performance during 2020 Drill Program

Element	Check Pairs Within 10% Difference	% total pairs within 10%	Check Pairs Within 20% Difference	% total pairs within 20%	Check Pairs Within 30% Difference	% total Duplicates within 30%
Au	95	48.0	156	78.0	172	86.0
Ag	103	51.5	134	67.0	144	72.0
Cu	168	84	191	95.5	196	98.0
Mo	116	58.0	161	80.5	168	84

11.5.3.4 Field Duplicates, 2021

Field Duplicate analysis in 2021 returned results similar to those from 2019 and 2020. The issue with field duplicates at the Casino deposit is the difficulty to accurately cut the core into identical "quarters". Although a high level of diligence was employed, the potential for uneven cutting exists, particular in areas of fractured, broken, or rubbly core.

Table 11-20 summarizes the duplicate performance for each element of interest in 2021.

Table 11-20: Summary of Duplicate (core) Pair Performance during 2021 Drilling program

Element	Duplicate Pairs Within 10% Difference	% total pairs within 10%	Duplicate Pairs Within 20% Difference	% total pairs within 20%	Duplicate Pairs Within 30% Difference	% total Duplicates within 30%
Au	61	45.2	95	70.4	107	79.3
Ag	70	51.9	101	74.8	115	85.2
Cu	79	58.5	110	81.5	122	90.4
Mo	40	29.6	82	60.7	107	79.3

Some degree of coarse gold (nugget) effect is likely here. Both Au and Ag commonly occur as fine nuggets, the uneven distribution of which may account for their lower reproducibility rates. Molybdenum commonly occurs as flakes, leading to a similarly, potentially more extreme, uneven distribution, likely responsible for the low Mo reproducibility. Overall, duplicate performance was slightly better than in previous years, with higher percentages of pairs falling within the boundaries of 10%, 20% and 30% variance.

Figure 11-74 through Figure 11-77 show the variances between duplicate versus original assay values for Au, Cu, Ag, and Mo.

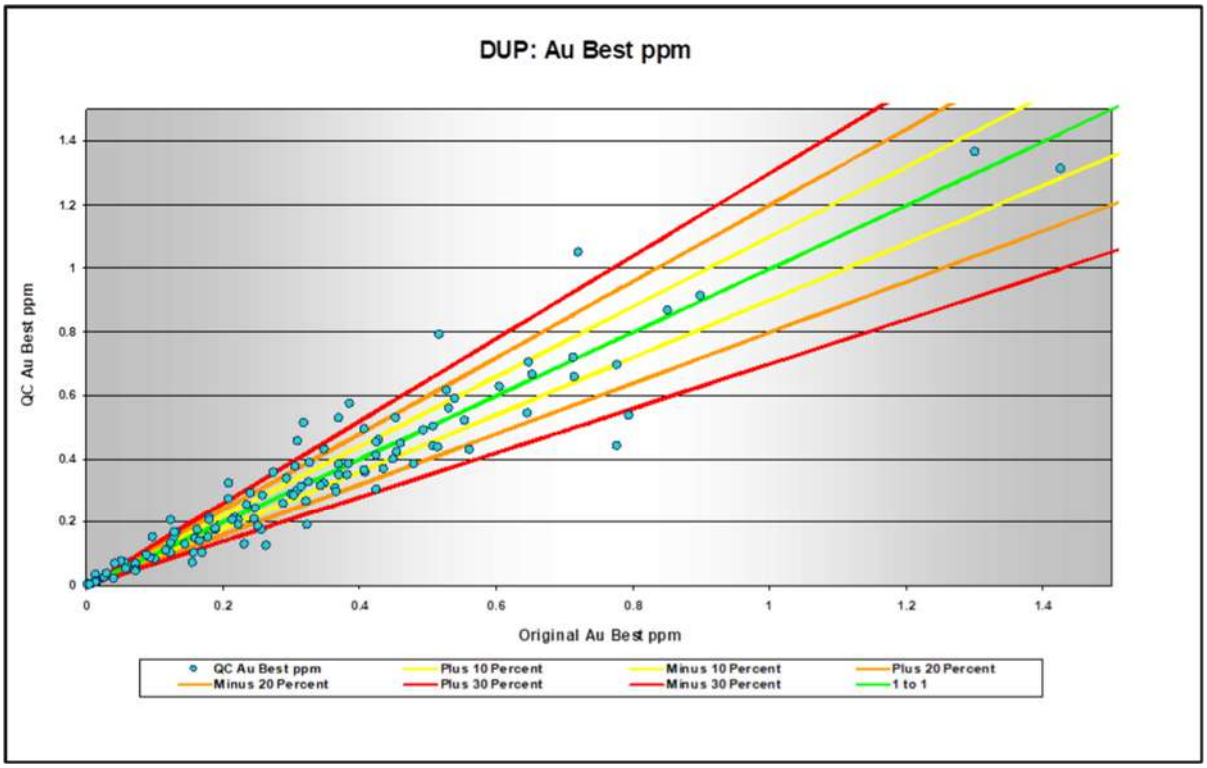


Figure 11-74: Comparison of Original versus Duplicate Gold Values

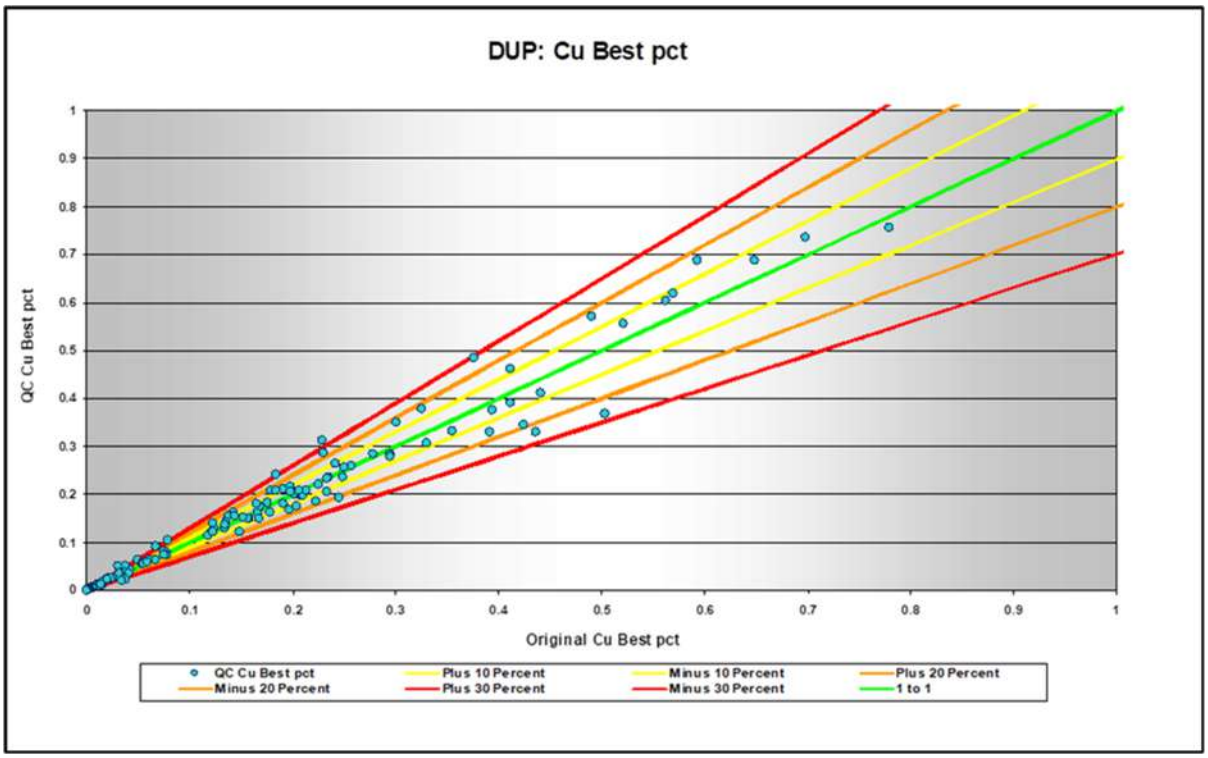


Figure 11-75: Comparison of Original versus Duplicate Copper values

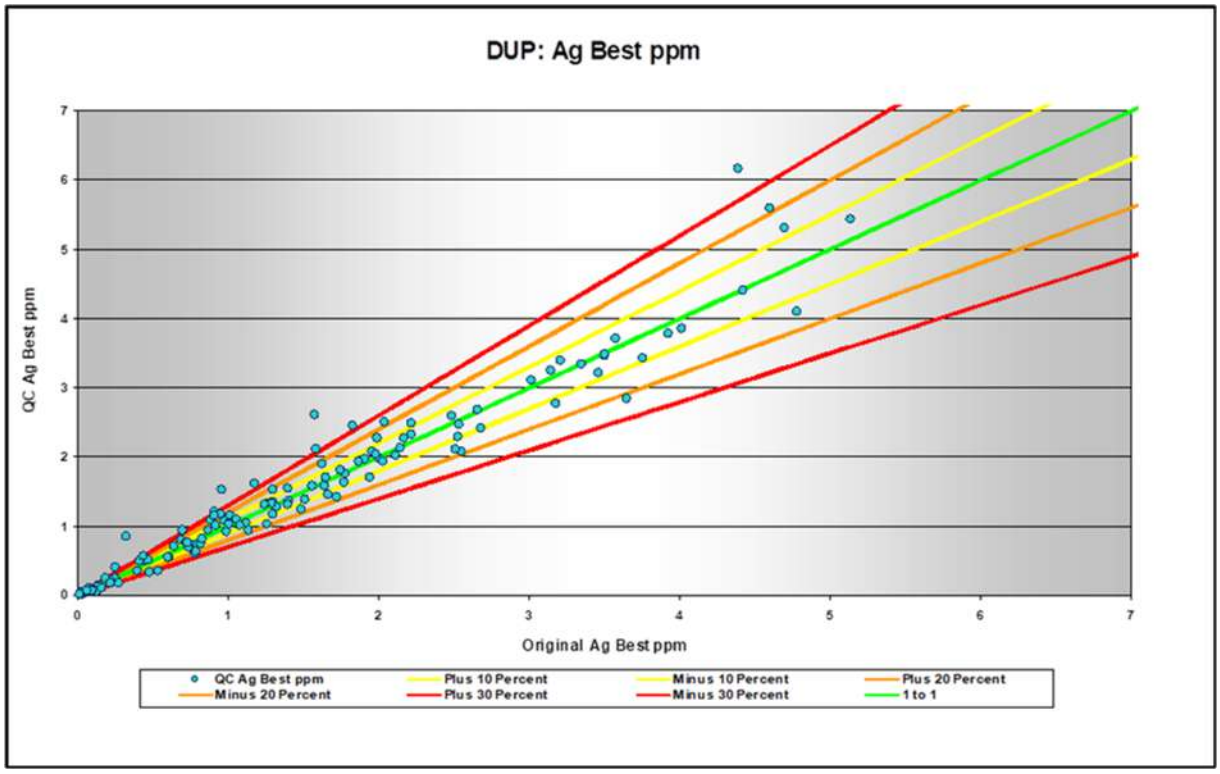


Figure 11-76: Comparison of Original versus Duplicate Silver values

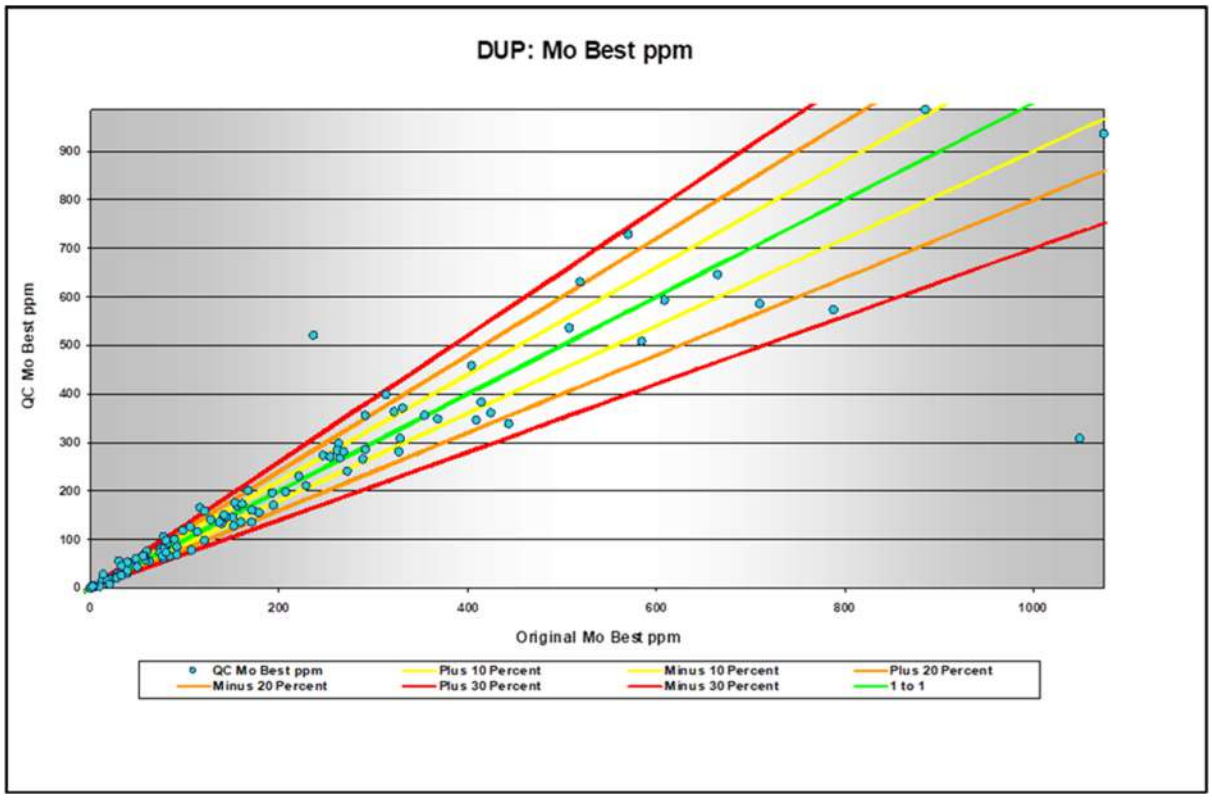


Figure 11-77: Comparison of Original versus Duplicate Molybdenum values

11.5.4 Opinion of Qualified Person

It is the author's opinion, as the Qualified Person responsible for this section of this report, that all parts of the data collection process commencing with drilling and advancing through geotechnical and geological core logging, core sampling, quality assurance during sampling and insertion of "Standard Reference Material" (SRMs), blank material and duplicate sampling, and security during shipping, meets NI 43-101 standards.

The analytical procedures employed are appropriate for the mineralogy of the Casino deposit that is the subject of this technical report. Additionally, the percentage and variety of SRM samples, quality of blank samples, and degree of duplicate sampling is adequate to provide an indication of the level of quality control employed by the labs involved. In 2021, duplicate sampling protocol varied depending on the drilling campaign (metallurgical, resource confirmation, geotechnical or exploration) conducted. All data are adequate for the purposes of resource estimation and for use in this technical report.

12 DATA VERIFICATION

This section contains a summary and review of the data entry and data verification related to the Casino Project. Several phases of exploration have taken place related to the project and the data entry and verification processes are discussed for each phase. The phases of exploration included diamond drilling programs dated 1992 through 1994 performed by PSG Exploration, followed by exploration by Western dated 2008 through 2010. Under Western, a transition to a new data system was implemented in 2013 and has continued since. The most recent exploration phase relevant to this report is dated 2021, although the resource estimate excludes the 2021 data.

Data entry and verification for the 1992 through 1994 programs were reviewed. Further, the author has reviewed a selection of original scanned drill logs and analytical certificates as compared with data in the current Casino Project database.

Data collection, entry and verification for the drilling programs conducted by Western from 2008 through 2010 has been reviewed. In addition, the author has reviewed original reports about the programs, and compared a selection of original drill logs and analytical results to the data within the current database.

A transition to an updated database system, GeoSpark Core, was implemented in 2013 by Western to streamline the data flow and provide automated data validation and checking. All data ranging from RC drill data from the 1960s to 2013 drill core logging details were merged and imported to the new database system. The author confirms a validation of the current database against a selection of original scanned drill logs and analytical certificates has been done, allowing for confidence in the data entry and data validation.

In 2019 the database system, GeoSpark Core, was used to combine core logging and assay data during the field season. Following the field season, a full audit/verification was done of all 2019 Collar, Survey and Assay, Alteration, Metallurgical and Lithology data. The 2019 data validation effort has been reviewed and, for this report, the author has also performed a review on the data entry and data validation related to the 2019 drilling program through comparison of original reports, original scans of drill logs, and analytical results to the data in the database.

In 2020 and 2021, a full (100%) audit was done for all logging data, and 20% of assay results were verified at random. Also, all 2020 and 2021 data were digitally logged, eliminating the need to incorporate scanned logging forms.

Ultimately this section of this report includes a comprehensive review of the data within the Casino Project database, thus confirming that the data has been generated using proper procedures, has been correctly entered digitally from the source files, and is suitable for use. Data validated to the database includes original scanned drill logs, and analytical certificates signed by an authorized individual.

12.1 DATA ENTRY

12.1.1 1992-1994

Original 1992 and 1993 field data were entered by Archer, Cathro and Assoc. and by Nowak and Assoc., both of Vancouver, B.C. Data was entered to a database on site and in the Vancouver office, by PSG personnel.

Assay, ICP, copper leach data, check assays and specific gravities were downloaded from the Chemex Labs computer-based data access system.

Pacific Sentinel Gold Corp. personnel entered the down-hole surveys and the collar surveys and were responsible for making corrections from the data verification process.

12.1.2 2008-2012

For the 2008 through 2012 exploration programs, field data processing and reporting were contracted to Casselman Geological Services Ltd. of Whitehorse, YT by Western.

Drill hole logging, sampling and geotechnical data were entered directly by the geologist or geotechnical logger working on the core into a Microsoft Excel spreadsheet. Upon completion of each hole, these files were submitted to the Project Manager for checking. Upon receipt of analytical data from the lab, the data was merged with the sample intervals by the Project Manager and the data was then verified.

All data were entered into Microsoft Excel spreadsheets organized into a standardized format. Once the data were checked it was posted on the Western FTP site. The data was then merged into Geosoft Target software for creation of drill plans, drill sections, and 3D modelling.

12.1.3 2013-2021

A transition to an updated database system, "GeoSpark Core", was implemented in 2013 by Western to streamline the data flow and provide automated data validation and checking. All data ranging from 1960s RC drill data to 2013 drill core logging details were merged and imported to the new database system.

Similar procedures to those used from 2008 through 2010 were used to collect the hydrogeological and water well drill data in 2013.

In 2019, all data were initially transcribed onto paper. Each part of the data logging was captured on a different piece of paper formatted for that specific data. Sample interval data was written directly onto the portion of the sample tag books that does not go into the sample bags during cutting.

The completed sample books were then checked by the Project Manager and stored in a secure cabinet in the geology office. The core logger was responsible for collecting all the data sheets in a file folder and scanning to digital files upon completion of the hole. These digital files were then uploaded to the Western remote server. Original paper copies were then filed in a secure cabinet in the geology office at the Casino Project site. The Project Manager would then ensure that each file folder for each hole had all the required data sheets, including Downhole Survey forms submitted by the drillers.

Down hole survey information was recorded digitally by the DeviShot downhole survey tool and then downloaded directly from the digital recorder by the Project Manager. This data was checked by the Project Manager and the digital files were uploaded to the Western server. Collar surveying was performed by surveyors from CAP Engineering and this data was provided to the Project Manager for addition to the main Casino Project database.

Upon completion of the field season, all 2019 data was entered into GeoSpark Core using the digital scans of the original core logging data. GeoSpark Core contains built-in checks to ensure a clean and usable dataset. Libraries of all data (e.g., lithology codes) link to each portion of the data entry so only codes that are checked and utilized by the project are accepted. Digital survey files that are downloaded from the downhole survey tool were imported directly into GeoSpark. Assay files from the lab were also directly imported without manipulation.

During the 2020 field season, all field data was entered directly into GeoSpark Core, using the same built-in checks and library codes as in 2019 to ensure a clean and usable dataset. The field geologists provided semi-weekly exports from GeoSpark to the Database Manager who then compiled all the core logging data in the central database using GeoSpark. Digital survey files were downloaded from the downhole survey tool and also provided to the Database

Manager for import into GeoSpark. Assay files received directly from the lab were imported by the Database Manager to GeoSpark.

During the 2021 field season, all field data was again entered directly into GeoSpark Core. A revised lithological dataset comprising a significantly more detailed stratigraphic column, subdivided into "Pre-mineral Wall Rock Units", "Early Mineral Units", "Intermediate Units" and "Late to Post-Mineral Units" was utilized. Otherwise, the 2021 program utilized essentially the same methodology. The field geologists provided semi-weekly exports from GeoSpark to the Database Manager who then compiled all the core logging data in the central database using GeoSpark. Digital survey files were downloaded from the downhole survey tool and also provided to the Database Manager for import into GeoSpark. Assay files received directly from the lab were imported by the Database Manager to GeoSpark.

The 2021 logging data and results were not incorporated into the updated resource estimate.

12.2 DATA VERIFICATION

For the purposes of this report, the author has visually verified five percent of original, scanned paper drill logs, and original, drill hole sample assay certificates, compared to the digital data used in the resource assessment. This verification amounted to review of 26 original drill logs and one excel file (containing re-logged primary lithologies and alterations), as well as 32 original, signed, assay certificates gold, silver, copper, and molybdenum analytical results, compared to the digital data within the project database.

The author has found no errors in the data transcription. This infers that the errors mentioned below have been addressed and provides further confidence in the data within the database.

12.2.1 1992-1994

The data verification process was performed under the supervision of a geologist familiar with the site logging procedures. In teams of two, one person read the original certificate, information sheet or logging form out loud while the other visually scanned the database printouts. Differences between the two were noted and corrected on the printout and the digital database. When required, a second pass was done on selected data.

The procedure for correcting errors was to highlight the value in question and to write the correct value beside it. Occasionally, the verification of field logs was followed up by a geologist familiar with logging and sampling techniques.

In addition, validations occurred throughout the exploration programs with ongoing monitoring and validation of field logs and analytical results, during the entire PSG Exploration endeavors.

12.2.2 2008-2013

The data verification process was performed under the supervision of the Project Manager. When errors were observed in geological, geotechnical, or sample intervals, the Project Manager and geologist or technician would go back to the core and/or original notes or sample tag booklets and sort out the error and make necessary corrections.

Data verification was performed on an ongoing basis. At times where data were first recorded on paper, original copies of the hand notes were kept for future reference.

In verification of field logs, when it was unclear which value was correct, a decision was made by a geologist familiar with logging and sampling techniques.

12.2.3 2019

The data verification process was performed under the supervision of the Project Manager/Senior Geologist on site. Digital scans of all the original core logs and related data were used to compare directly to the data that had been entered into GeoSpark Core software. The core log data was split up into sections (e.g., Assays were one section and Lithology was another) and assigned to two separate people to verify. Each person was also given a full Excel export of the master database, with which they would be comparing the original log scans. After each section was verified, the person who performed the work would submit a memo outlining the errors encountered and possible solutions to these errors to the Database Manager. The Database Manager would then work through the errors and make changes, when warranted. A complete review (100% of the records) was done for Assays, Alteration, Metallurgy, Lithology, Collars and Surveys. Assays were compared directly to the original assay certificates. Collar data was compared directly to the surveys performed by CAP Engineering and the logging forms and Downhole Survey data was compared directly to the exports from the DeviShot survey tool; all other data was compared directly to the core log scans. A partial (approximately 20%) review was then completed by the database manager for Geotechnical and Specific Gravity data by comparing the master database records to the original log scans.

Upon completion of 2019 data verification, the Project Manager reviewed the errors found and made changes where warranted. In cases where it was unclear what data were correct, the Project Manager would review related information (e.g., notes/comments on the logs and core photos) and make a final decision based on that related data and on extensive knowledge of the project itself. Overall, the data were in good shape, with occasional missing records or incorrect codes (e.g., POT instead of PRO for an alteration interval) entered during the first phase of data transcribing from original logs to GeoSpark. There were very few errors found during the partial review of the Geotechnical and Specific Gravity data; a complete review (100%) of this data was not conducted at the time due to the initial 20% pass finding so few errors and because a 20% verification is considered acceptable by the manager for this data.

12.2.4 2020

Verification of the 2020 data involved a three-step process. As in 2019, Step One in 2020 utilized the GeoSpark Core program in the field to collect core logging data. Assay results were imported directly from the laboratory into the program. As the data was collected and the logging of each hole was completed, geologists in the field would visually review the digital core log for accuracy and use the validation tools within GeoSpark Core to complete Step One of verification. The database manager compiled all core logging data received from the field, imported assay results received directly from the lab and visually verified the data during this process.

The second stage of verification was completed by two consulting geologists who did a complete audit of the 2020 Collar, Survey, Assay, Alteration, Metallurgical, Lithology, Structural and Geotechnical data. As in 2019, after each section was verified, the person who performed the work would submit a memo outlining the errors encountered, and possible solutions to these errors, to the Database Manager. The Database Manager would then work through the errors and make changes, when warranted. A 20% verification of assay data was completed by comparing the original assay certificate to the values imported into GeoSpark.

Upon completion of 2020 data verification, the Database Manager reviewed the errors found and made changes where warranted. In cases where it was unclear what data were correct, the Database Manager would review related information (e.g., notes/comments on the logs and core photos) and make a final decision based on that related data and on extensive knowledge of the project itself. Overall, the data were in good shape, with occasional missing records due to issues during import and typos in data entry (e.g., CAP litho code entered in the middle of 100 m of WRGD). Any missing data was located and imported, and typos were corrected.

The final stage of verification took place when the lithology, metallurgy and alteration surfaces were updated with the 2020 logging data, imported into a 3D model, and then compared with historic data. Finalized copper soluble assay

data were also used to confirm the metallurgical zones more precisely. There were no major errors in the logging data when compared against historic logging data in 2D and 3D views, but the soluble copper assays did provide clarity for the metallurgical zone boundaries and those new values were added to the database as a relog to preserve the original logged data while allowing for use of the updated information.

12.2.5 2021

In 2021 there was a two-stage verification process beginning with a check of 100% of all Collar, Downhole Survey, Alteration, Metallurgy, Lithology, Structure, Geotechnical and Specific Gravity data records. This initial stage was then followed by a check of 20% of all assay intervals.

The audit was performed using data exported from the Geospark Core database (Geospark). Geologists in the field were responsible for checking their core logs at the end of each hole, but the final verification was performed by staff of Western Copper and Gold and then reviewed by the Database Manager. The process for checking the database exports involved looking for discrepancies or gaps of information in each different section.

Verification commenced with a review of the collar data. Information recorded in GeoSpark was compared to the table of surveyed coordinates provided by Challenger Geomatics, and all Geospark records were found to be a match for coordinate data. Collar azimuth and dip data was then compared to downhole survey data. For holes with downhole surveys, this information was found to match. Finally, the depth of the hole was compared to logged data (e.g., lithology). One hole (DDH 21-21) was found to have an incorrect final depth listed on the collar data; this was corrected to match the related logging records.

The next section verified was the Downhole Survey database. For this, it was important to compare the survey information with that shown in the collars. If the surveys either did not commence near the top of the hole, didn't extend to the bottom of each drillhole, were missing a reading over a significant interval, or had significant variations in the azimuth readings, this information would be noted in the verification process. Several holes did not have downhole surveys provided by the drilling company and could not be located upon downloading directly from the survey tool. Drill holes with completed downhole surveys were found to be mostly correct and in line with collar azimuth, dip, and hole depth. In cases where the azimuth or dip did not match with the rest of the hole, the magnetic vector usually indicated a bad measurement, and this data was not used for plotting.

Drillhole samples were found to have no gaps, and all holes were sampled from the overburden-bedrock interface to the end of the hole.

There were few errors in the remaining drill log data and those that did occur were mainly missing intervals or gaps in data. These were located in the original exports sent by the core logging geologists and have been corrected.

For the final stage of assay verification, a random subset of 20% of all assay samples was chosen using the RAND function in Excel. This subset of 548 assayed samples was then compared directly with the original ALS Certificates.

12.3 VERIFICATION ERRORS

For the purposes of this report, the author has verified five percent of original drill logs and drill sample assays with the data used in the resource assessment. This involved visually comparing the analytical results for gold, silver, copper, and molybdenum within the original scans of signed assay certificates with assay data in the database, and visual comparison of primary lithology and alteration data and the corresponding intervals of the scanned paper logs to the data in the database, as well as review of re-logged data where applicable. There were no discrepancies found during this verification. This infers that the errors noted below related to earlier reviews have been addressed.

12.3.1 1992-1994

The geological logs had some errors introduced when the data entry personnel were unclear of the recording method the geologist was using. Additionally, changing definitions of many of the lithology types required re-logging of many of the holes in the 1992 and 1993 programs. The process of combining the information from the old and new logs introduced some errors into the database. Due to the number of discrepancies encountered in the Geology data of the 1992 and 1993 programs, a second verification of lithologies and alteration was performed after the errors detected in the first pass were corrected. Since the re-logging of the historic core in 2010, these errors associated with geological, mineralogical or alteration logging have been eliminated.

12.3.2 2008-2012

There were very few errors in the database. The most common error observed was in geological or sample intervals, where the "To" recording of a previous sample did not match the "From" recording of the subsequent sample. These were generally easy to sort out by the geologist or geotechnical logger.

Discrepancies with the assay, ICP and copper leach data involved values below the detection limit. Occasionally, "less than" symbols (<) were misplaced for the lower detection limit values. Anomalously high ICP values were occasionally rounded off differently in the assay certificates than in the assay data downloaded from the computer bulletin board.

The geotechnical logs were checked by the computer to find intervals with combinations of parameters that were suspect. These intervals were extracted from the database and the suspect values were checked against the originals and against other available information, such as core photos, to determine if they were in error. A large majority of the extracted parameters were correct and considered to be caused by normal variance of geotechnical characteristics.

Errors detected in the field data of the geological logs, geotechnical logs, synoptic logs, specific gravity logs and down-hole survey data were often a result of human error in recording the original or in transcription. Wherever possible, computer checks were done on the data; several types of errors were detected this way.

Errors found in the specific gravity data were due to the geotechnician assigning the wrong sample number to the interval from which the specific gravity was taken. These errors were detected by a computer check and confirmed by the data verification personnel.

12.3.3 2019

A complete data audit took place following the 2019 exploration program at the Casino Project. The data audit included a 100% audit of: Assay Data, Alteration Data, Metallurgical Data, Lithography Data, Survey Data and Collar Location.

The audit was performed using exported data from the Geospark Core database.

There were two main types of errors encountered during the verification process: missing records that had not been imported or entered and incorrect codes/typos. Overall, there were very few errors in the data entry, and all could be easily corrected by the project manager. Missing data was imported in the case of assay certificates or entered from original logs in the case of logging information.

As GeoSpark Core catches the inherent errors that crop up from manual entry into Excel or Access, the 2019 dataset was ready for import into other software for maps, cross sections and 3D modeling directly after the audit and verification process.

12.3.4 2020

In 2020, data verification identified several instances of missing data that was not imported into the main database. Data that was not imported but located in digital logs was subsequently imported. Any data missed during logging was entered or corrected based on core photographs, assay results, or results from nearby holes. Certain parameters, such as metallurgical zones and alteration, were omitted where drilling was outside the pit shell (metallurgical zones) or not present in visually detectable amounts (e.g., alteration). In rare cases, such as abandoned holes, portions of holes were found to be unlogged or to have certain aspects, such as specific gravity or geotechnical logging, omitted from the main database. Several overlaps of data, particularly lithological, metallurgical and alteration data, were also detected in Geospark during the logging process and rectified.

12.3.5 2021

For the purposes of this report, the author has verified five percent of original drill logs and drill sample assays to the data used in the resource assessment. For pulp check samples, 5% of the total represented 117 samples. This involved visually comparing the analytical results for gold, silver, copper, and molybdenum within the original scans of signed assay certificates to assay data in the database. It also involved visual comparison of scanned paper drill logs, focusing primary on lithology and alteration data and the corresponding intervals to the data in the database, as well as review of re-logged data where applicable. There were no discrepancies found during this verification. This infers that the errors noted below that are related to earlier reviews have been addressed.

The results in the certificates matched the values in the database in every sample that was checked. In some cases, where an overlimit or lower detection limit value is shown on the certificate, the database will replace these values with the value of the overlimit (e.g., >10,000 ppb Au on the certificate will be 10, 000 ppb in the database) or the negative value of the lower detection limit (e.g., <0.1 ppb Au on the certificate will be -0.1 ppb Au in the database). All of the overlimit or lower detection limit replacements were correct. One sample had no assay values but was shown to have "NSS" ("Not Sufficient Sample") on the certificate. One sample had no values for Germanium (Ge), and the missing value for Ge was re-imported.

12.4 OPINION OF QUALIFIED PERSON

It is the author's opinion, as the Qualified Person responsible for this section of this report, that the data for the Casino Project meets NI 43-101 standards and is adequate for the purposes of resource estimation and for use in this technical report.

All parts of the data collection process from drilling, sampling, and logging to shipping, assaying and verification have been reviewed by the author or personnel reporting to the author. It is the author's opinion that the Casino Project database has been maintained at high quality both in 2021 and previously.

In addition, the author can confirm that a five percent verification on scanned, original drill logs and signed, original assay certificates compared to the data in the Casino Project database has been done, and that no errors in the data transcription.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Casino Project will produce copper flotation concentrates with contained gold and silver values, and molybdenite flotation concentrates. Gold in the form of doré, and a high-grade copper sulphide product will also be produced from an oxide ore heap leach. All products will be shipped offsite for sale or further processing.

13.1 METALLURGICAL SAMPLES

In the test work commissioned by Pacific Sentinel Gold in the mid 90's, all samples used were assay rejects that were nominally minus 10 mesh in particle size. These assay rejects were combined to prepare composites that were sent to Lakefield Research for flotation and other testing under the direction of Melis Engineering, Ltd., to Brenda Process Technology for flotation testing, and to Kappes, Cassiday and Associates for copper and gold leaching.

The source of samples for all the 2008 work was split HQ core that was retrieved from site in September 2007. The core had been at site since it was drilled in 1993 and 1994 but was stored under cover.

Samples for the G&T Metallurgical Services (G&T) test program reported in early 2011 were split from fresh core from the 2010 drill program.

Samples for the comminution testing performed by Starkey and Associates (Starkey), FLSmidth, and comminution and flotation testing by G&T reported in early 2012 were retrieved in 2011 from the 1993 to 2010 drill programs and consisted of split core. Two shipments of half drill samples were received at G&T metallurgical. One shipment was used to construct five composites for metallurgical testing and one shipment was used to construct 11 composites for ore hardness testing. A portion of each of the 11 composites, crushed to minus 1.5 inches, was split out for SAG Design testing at FLSmidth.

A drill program to retrieve fresh hypogene core was completed in early 2012 and split core from this drilling campaign was used for the flotation tests reported by G&T in December 2012.

In June 2013, six bulk samples of different lithologies, identified as MET-02, MET-05, MET-11, MET-7, MET-19 and MET-20, were collected (William Dunn Enterprises) from just below the surface of the deposit using an excavator and were used for heap leaching studies performed by SGS E&S Engineering Solutions Inc. (SGS) in Tucson, Arizona reported in October 2014 and hydrodynamic characterization performed by HydroGeoSense in Tucson, Arizona reported in 2015. Three composites were made from the six samples, MET-02 and MET-05 samples were combined to generate Composite 1, MET-11 and MET-17 samples were combined to generate Composite 2 and MET-19 and MET-20 samples were combined to generate Composite 3. Approximately 89 kg of Composite 1 and 89 kg of Composite 2 were sent to HydroGeoSense.

In September 2020, Aurora Geosciences collected 10 met samples, each taken from near surface (0.5 m or greater below the overburden-bedrock interface) using an excavator for metallurgical testing. The samples were sent to SGS Canada Burnaby laboratory for metallurgical testing to investigate the optimal crush size for heap leaching reported in 2021. Three 300 kg composites representing fresh ore from the WR (MET-02/MET-05), PP (MET19/MET20), and IX (MET11/MET17) lithologies were prepared. The composites were each crushed to P100 of 19 mm. 50 kg of each composite was split out, screened and assayed. Approximately 60 kg of each composite sample was sent to HydroGeoSense in Tucson, Arizona for hydrodynamic characterization reported in 2022.

13.1.1 Data Verification

The QP was not involved in the collection of samples for testing, or the test work completed on the samples. The QP has reviewed the sample preparation, analysis, and security for collection of the metallurgical samples discussed in

the available test reports and considers it reliable. Data verification work for drill logs and drill sample assays is described in Section 9 of this report and is believed to be reliable. The QP is not aware of any limitations on or failure to conduct appropriate data verification for samples used for the metallurgical testing.

The QP believes that tests were performed using standard quality assurance / quality control (QA / QC) procedures. The QP reviewed all available test work to form an opinion regarding the mineral processing and metallurgical testing and is of the opinion that the data available is adequate for the purposes of the Feasibility Study.

13.2 COMMINUTION TESTING

In 2008 SGS Lakefield, under the direction of SGS MinnovEX, performed a comprehensive comminution study. Fifty (50) split drill core samples, representing the first six (6) years of production were sent to SGS and subjected to several tests.

A summary of the comminution results is presented in Table 13-1. As SGS reports, the samples tested were characterized as medium in hardness from the perspective of semi-autogenous milling and of medium in hardness with respect to ball milling.

Table 13-1: Summary of Comminution Results

Test Name	CEET CI	SPI (Min)	RWI (kWh/t)	BWI (kWh/t)	MBWI (kWh/t)	AI (g)
Average	**29.2**	**52.9**	**9.9**	**14.5**	**14.3**	**0.265**
Std. Dev.	13.9	20.8	5.6	2.6	1.6	0.046
Rel. Std. Dev.	47.5	39.3	56.5	18.1	11.3	17.0
Minimum	13.5	12.6	0.0	11.2	11.4	0.226
10th Percentile	15.3	31.4	4.4	12.1	12.5	0.232
25th Percentile	19.1	37.4	11.1	13.3	13.0	0.242
Median	**24.1**	**50.3**	**12.5**	**14.1**	**14.1**	**0.252**
75th Percentile	38.0	63.4	13.0	15.9	15.6	0.275
90th Percentile	52.3	82.5	13.0	17.3	16.3	0.309
Maximum	66.9	114.1	13.0	18.2	18.3	0.332

Additional comminution testing was performed in 2012 under the direction of both FLSmidth and Starkey and Associates at FLSmidth laboratories and G&T Metallurgical Services. This program tested 11 composites of ore representing a combination of different zones, lithologies and alterations. The 11 composites represent over 80% of the material that will be processed through the mill.

CMC geologists mapped zones, lithologies and alterations that were not tested to similar composites that were tested.

The 11 comminution composites were subjected to a series of tests at G&T's laboratory and FLSmidth's laboratory. The test results are summarised in the following Table 13-2 and Table 13-3.

Table 13-2: Summary of G&T SAG Mill Comminution (SMC) Test Results

Sample ID	DWi, kWh/m^3	DWi, %	Mia, kWh/t	Mih, kWh/t	Mic, kWh/t	A	B	SG	t_a
Composite 1	4.90	39	15.5	10.8	5.6	56.7	0.95	2.64	0.53
Composite 2	4.35	32	14.1	9.6	5.0	56.3	1.07	2.63	0.59
Composite 3	6.05	55	18.6	13.5	7.0	61.8	0.70	2.60	0.43
Composite 4	6.62	63	19.8	14.6	7.6	62.3	0.64	2.63	0.39
Composite 5	6.69	64	19.9	14.7	7.6	63.4	0.62	2.64	0.39
Composite 6	3.92	26	13.3	8.8	4.6	62.9	1.05	2.58	0.66

Sample ID	DWi, kWh/m³	DWi, %	Mia, kWh/t	Mih, kWh/t	Mic, kWh/t	A	B	SG	t_a
Composite 7	5.75	51	18.1	13	6.7	66.4	0.67	2.58	0.45
Composite 8	5.60	49	16.9	12.1	6.2	64.1	0.75	2.69	0.46
Composite 9	5.00	40	16.1	11.3	5.8	67.9	0.76	2.58	0.52
Composite 10	9.63	90	26.3	20.9	10.8	91.3	0.30	2.67	0.27
Composite 11	5.69	50	17.6	12.6	6.5	66.4	0.69	2.62	0.46
Average	5.84	51	17.8	12.9	6.7	65.4	0.75	2.62	0.47

Table 13-3: Summary of SAG Design Results and Crushed Bond Test Results

Sample ID	DML SAG Design Test Results			G&T Crushed Bond Test Results			
	Relative Density	Calc W_{SAG} to 1.7 mm (kWh/t)	SAG Dis. Bond BWi (kWh/t)	BWi (kWh/t)	RWi (kWh/t)	Ai (g)	CWi
Composite 1	2.66	8.19	16.18	13.5	12.9	0.162	9.41
Composite 2	2.60	6.78	17.26	14.1	12.3	0.176	10.00
Composite 3	2.66	9.39	15.70	14.1	14.5	0.198	13.62
Composite 4	2.72	12.41	18.36	15.5	15.5	0.199	13.84
Composite 5	2.64	9.56	18.26	15.3	14.6	0.156	11.20
Composite 6	2.67	5.05	16.37	13.7	10.4	0.118	10.22
Composite 7	2.69	7.45	16.12	13.4	12.4	0.155	14.57
Composite 8	2.82	7.71	17.82	15.2	14.1	0.170	12.27
Composite 9	2.57	6.48	14.35	12.9	11.4	0.158	11.03
Composite 10	2.71	11.68	18.93	16.6	14.9	0.161	13.23
Composite 11	2.67	8.50	17.23	15.1	13.5	0.170	10.33
Average	2.67	8.47	16.96	14.5	13.3	0.166	11.79

A circuit consisting of one 40 ft diameter (12.2 m) SAG mill and two 28 ft diameter (8.5 m) ball mills in closed circuit with two pebble crushers was selected, based on discussions with M3 and FLSmidth as a circuit that would likely meet the design tonnage. This circuit was modelled by FLSmidth using the parameters developed by SGS, G&T, and FLSmidth for the various composites. The results of this exercise are shown in Table 13-4.

Table 13-4: Predicted Production Rate

Project Sample Number	Client Sample Information	BWi G&T (kWh/t)	Production Rate (t/d)
1	Composite 1	13.5	133,805
2	Composite 2	14.1	128,064
3	Composite 3	14.1	128,064
4	Composite 4	15.5	116,582
5	Composite 5	15.3	118,018
6	Composite 6	13.7	131,818
7	Composite 7	13.4	134,798
8	Composite 8	15.2	118,790
9	Composite 9	12.9	139,987
10	Composite 10	16.6	108,854
11	Composite 11	15.1	119,674
Average		14.5	125,314

13.3 FLOTATION

13.3.1 2008 G&T Metallurgical Work

In 2008, Western and G&T Metallurgical reviewed the previous metallurgical work and developed an updated flotation program. To prevent oxidation, the program used split drill core rather than assay rejects as it had been done for the previous work.

The work focused on two composites at two different levels of oxide copper – an "oxide composite" and a "sulphide composite". The composites were prepared to be close to the average grade of ore received for the first 5 years. Assays for these composites are shown in Table 13-5.

Table 13-5: G&T Flotation Composite Assays

Composite	Cu (%)			Mo (%)		Fe	Au
	Total	WAS*	CNS**	Total	AS	(%)	(g/t)
Oxide Composite	0.275	0.132	0.042	0.019	0.006	3.225	0.345
Sulphide Composite	0.260	0.016	0.032	0.021	0.002	3.525	0.255

* Weak Acid Soluble
** Agent that affects the Central Nervous System

13.3.1.1 Oxide Composite

Copper recovery and grade from the oxide composite was very poor. Various combinations of sulphidizing the ore, changing grind size, using different reagents were attempted. Based on the poor performance of the oxide flotation, no further testing on the oxide composite was performed.

13.3.1.2 Sulphide Composite

Copper recovery from the sulphide composite was much better than that achieved for the oxide composite. Copper concentrate grades greater than 28% were routinely achieved.

Copper recoveries of 70-82% were obtained into concentrates grading from 26.8 to 32.2% copper in cleaner tests. Good recovery of copper was obtained with both a primary grind with K80's of 147 and 121 µm and regrinds with K80's less than 22 µm. A coarser grind with a K80 of 209 µm was examined in rougher tests and shown to be less favorable than the finer particle sizes selected for cleaner testing.

13.3.1.3 Locked Cycle Tests

Duplicate locked cycle tests at both primary grind K80's of 121 µm and 147 µm were performed as well as one locked cycle at a primary K80 of 209 µm. The results from these tests indicate that a grind with a K80 of 147 µm, 85.6% copper can be recovered into a 28.5% copper concentrate. Molybdenum recovery was variable and ranged from 26.5% to 69.4%. Gold recovery was more consistent and averaged 64.0%.

13.3.1.4 Variability Testing

A total of 63 individual split drill core intervals were tested for variability. These samples were chosen to primarily represent the first six years of production and covered a broad range of total copper, acid soluble copper, molybdenum, and gold values. Each of these samples was individually ground and floated in a cleaner test with regrind under the conditions determined from the locked cycle tests.

13.3.2 2009-2011 G&T Metallurgical Work

13.3.2.1 2009 Fresh Core Tests

The 2009 drilling campaign included two holes in the middle of the deposit – CAS-002 and CAS-003. A composite from CAS-002 had 92% copper recovery into a concentrate grading about 28% copper in cleaner tests. Similarly, a composite from CAS-003 had 87% of the copper in the feed recovered into a concentrate grading 26% copper. Molybdenum recoveries were high in both tests at approximately 90%.

13.3.2.2 2010 Supergene Sulphide Composite Tests

The material tested in the 2010 test program (reported at the beginning of 2011) was a composite of supergene material that was obtained from the drilling campaigns in 2009 and 2010. This material represented ore that will be fed to the mill in the later years of the operation. The feed grade averaged 0.30% copper and 0.037% molybdenum.

One of the main objectives of the 2010 test program was to evaluate coarser grinds than were tested in the 2008 test program. Results of this evaluation indicate that copper flotation response is virtually unaffected by primary grind size between 142 and 253 µm for this composite. Molybdenum flotation recovery to the bulk rougher concentrate was lower at grinds coarser than 179 µm. Molybdenum recovery was also reduced at elevated pH levels.

13.3.2.3 2010 Supergene Sulphide Composite Locked Cycle Tests

Locked cycle tests at primary grind K80's of 142 µm and 222 µm were performed. The results from these tests are presented in Table 13-6. The effect of regrind size on bulk concentrate copper grade and the effect of primary grind and regrind size on molybdenum recovery are indicated in the table below.

Table 13-6: 2010 Supergene Sulphide Composite Locked Cycle Test Results

Test	P. Grind K80 µm	Regrind K80 µm	Cycle	Assay				Distribution - percent			
				Cu (%)	Mo (%)	Fe (%)	Au (g/t)	Cu	Mo	Fe	Au
KM2721-33	222	19	IV	30.8	1.6	23.6	20.2	82.9	34.7	5.2	71.9
KM2721-33	222	19	V	28.2	1.4	26.5	19.9	81.6	34.2	6.2	69.7
KM2721-34	222	20	IV	26.1	1.6	25.7	17.8	88.6	48.8	7.0	68.4
KM2721-34	222	20	V	25.7	1.5	26.6	19.9	86.6	45.1	7.5	64.7
KM2721-35	142	19	IV	26.3	1.9	27.8	18.6	87.3	57.1	7.1	71.4
KM2721-35	142	19	V	25.1	1.7	27.6	16.1	86.4	54.3	7.6	66.1
KM2721-36	222	37	IV	17.8	1.4	31.1	13.1	81.7	55.7	9.6	67.0
KM2721-36	222	37	V	18.8	1.4	30.4	10.1	82.8	51.2	10.9	61.5
KM2721-37	222	31	IV	21.2	1.4	31.1	11.7	83.2	54.1	9.2	62.4
KM2721-37	222	31	V	20.8	1.7	31.3	11.7	83.9	59.7	9.9	65.7

13.3.2.4 Pyrite Flotation

Pyrite flotation was examined as a process to produce tailings samples that had low levels of residual sulphur, and thus could be deemed non-acid generating (NAG).

The locked cycle tests outlined in Table 13-6 included a pyrite rougher to reduce the sulphide concentration of the tailings. Pyrite flotation tailings from these tests obtained tailings averaging less than 0.08% sulphur.

13.3.3 2011-2012 G&T Metallurgical Work

Western retained International Metallurgical and Environmental to assist in the metallurgical testing and continued to perform the testing at G&T Metallurgical Services (name changed to ALS Metallurgy in late 2012).

13.3.3.1 Flowsheet Development

In previous testing campaigns, to achieve acceptable recoveries from the conventional copper flotation flowsheet's tested, 15-20% of the feed material needed to be reground. The focus of the flowsheet development was to reduce the material sent to the regrind mills.

The flowsheet development centered on a flowsheet where rougher concentrate was sent to the first cleaning stage prior to regrinding, the first cleaner concentrate went to regrinding and the second and third cleaner tails were returned to the first cleaner. By utilizing this flowsheet, the amount of feed material that needed to be reground dropped from 15-20% to 3 to 5%.

Locked cycle test results from the composites tested using this flowsheet are shown in Table 13-7. The results show similar recoveries to previous test work using a conventional copper flotation flowsheet.

Table 13-7: Flowsheet Development Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay				Distribution (%)			
				Cu (%)	Mo (%)	S (%)	Au (g/t)	Cu	Mo	S	Au
HYP1	38, 42	218	19	26.0	1.98	33.1	24.6	82.1	64.9	24.9	61.1
HYP2	39, 43	216	16	26.3	1.31	32.8	23.9	81.7	37.1	14.6	56.1
SUS1	44, 46	192	17.5	21.8	1.77	33.9	23.6	77.7	59.5	24.9	75.9
SUS2	47	190	14	24.1	0.85	38.1	28.3	62.8	32.8	20.4	64.4

13.3.3.2 Tests using Fresh Core

While supergene flotation tests were performed on fresh core obtained during the 2010 campaign, no flotation tests had been performed on fresh hypogene core except for a limited number of tests performed in 2009.

In 2012, a drilling campaign was executed to obtain fresh hypogene core from the first years of mining that represented the predominate mineralization that would be fed to the mill. In total, five holes were drilled (CAS-088 to CAS-093), and from these five holes, three composites were made representing lithologies: Patton porphyry (PP), Intrusion breccia (IX), and Dawson range batholith (WR).

Table 13-8: Hypogene Composites

	Cu (%)			Mo	Fe	Au
	Total	WAS	CNS	(%)	(%)	(g/t)
PP Composite	0.14	0.004	0.008	0.030	2.95	0.22
IX Composite	0.17	0.006	0.012	0.071	2.39	0.22
WR Composite	0.19	0.005	0.013	0.019	2.50	0.18

Locked cycle recoveries using these fresh composites were significantly better than previous testing on oxidized core and are shown in Table 13-9. Note that the primary grind size for these tests was also higher than the target of 200 µm, in some cases significantly, so it would be expected that actual plant recovery would be better than these tests indicate.

Table 13-9: Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay - percent or Au g/t				Distribution (%)			
				Cu (%)	Mo (%)	Ag (g/t)	Au (g/t)	Cu	Mo	Ag	Au
PP	23	234	31	18.6	7.5	126	15.6	89.9	77.9	46.5	57.3
IX	24	254	32	24.6	4.3	107	24.3	87.2	78.6	46.0	55.4
WR	25	211	31	17.5	1.50	82	13.5	91.9	89.4	53.8	67.2

13.3.3.3 Pilot Plant Testing and Copper/Molybdenum Separation

A pilot plant was performed on hypogene, and supergene composites taken from the drilling campaign to produce representative tailings for environmental testing, geotechnical testing, and thickener testing and to produce sufficient copper/molybdenum concentrate for copper moly separation tests. Unfortunately, there was not sufficient feed material to obtain operating information from the pilot plant.

Although suitable copper/molybdenum concentrate was produced to perform several copper/molybdenum separation tests, only one cleaner test was performed. The results from this test were sufficiently good to warrant no further testing. The results from this test are shown in Table 13-10.

Table 13-10: Copper/Molybdenum Separation Cleaner Test

Cumulative Product	Cum. Weight		Assay				Distribution (%)			
	%	grams	Cu (%)	Mo (%)	Fe (%)	S (%)	Cu	Mo	Fe	S
Final Conc.	3.1	31.2	0.39	57.4	0.8	37.9	0.1	94.1	0.1	2.6
Second Conc.	3.5	35.5	2.38	51.3	3.8	37.6	0.5	95.7	0.4	3.0
Rougher Conc.	6.1	62.5	9.04	29.7	15.3	38.0	3.5	97.4	2.9	5.3
Tails	93.9	953.8	16.5	0.05	33.9	44.3	96.5	2.6	97.1	94.7
Feed	100.0	1016.3	16.0	1.87	32.8	43.9	100	100	100	100

13.3.4 2022 ALS Metallurgy Program

Additional metallurgical work, focusing on flotation performance, is underway at ALS Metallurgy under the direction of International Metallurgical and Environmental and Rio Tinto personnel. Although the work is incomplete, the results are consistent with the 2011-2012 G&T Metallurgical program.

13.3.4.1 Interpretation of Flotation Test Results

The 2012 metallurgical program at ALS Metallurgy showed good copper recovery to copper concentrates that routinely achieve 28% or greater for various drill core samples from the deposit using the reagent scheme developed. The conclusions from this work are unambiguous and will be used as the basis of this study.

13.3.4.2 Supergene – Copper

It was difficult to achieve good copper concentrate grades from supergene oxide material that had copper oxide concentrations greater than 25-30% of the total copper. For this reason, during operation of the mill, supergene oxide ore will need to be blended in with the other ore types to achieve an oxide copper percentage less than 25%.

The supergene ore contains a certain percentage of oxide copper minerals (this is what defines it as being supergene material). Oxide copper minerals are poorly recovered by the flotation process; therefore, in the interpretation of the results, it is important to examine the recovery of sulphide copper to a copper concentrate. Sulphide copper can be

calculated by subtracting the concentration of oxide copper from the total copper. Supergene mineralization at Casino has been assayed for weak acid soluble copper (WAS), which is approximately equal to the amount of oxide copper in the sample assayed but may under or over represent the amount of oxide copper present depending on the specifics of the mineralization.

Sulphide copper recovery as a function of total copper grade and sulphide copper grade is shown in Table 13-11 for the supergene locked cycle tests by G&T Metallurgical. Recovery appears to be consistent.

Table 13-11: Supergene Locked Cycle Recoveries to Concentrate

	Feed Assays					Recovery to Concentrate			
	Cu (%)			Au	Mo	Total	Sulphide		
Test	Total	WAS	Sulphide	(g/t)	(%)	Cu	Cu	Au	Mo
KM2721									
33	0.3	0.03	0.27	0.25	0.036	82.3	91.4	70.7	34.5
34	0.3	0.03	0.27	0.25	0.036	87.6	97.3	66.4	46.9
35	0.3	0.03	0.27	0.25	0.036	86.8	96.4	68.8	55.7
36	0.3	0.03	0.27	0.25	0.036	82.3	91.4	64.4	53.3
37	0.3	0.03	0.27	0.25	0.036	83.5	92.8	64.1	57
KM3134									
44	0.3	0.056	0.244	0.37	0.022	79.9	98.2	75.5	64.6
46	0.3	0.056	0.244	0.47	0.022	75.6	93.0	76.1	54.6
47	0.3	0.094	0.206	0.47	0.028	62.8	91.5	64.4	32.8

Averaging the locked cycle tests results indicates that an average of 94% of the sulphide copper was recovered to a copper concentrate. This result also closely mirrors the variability results. Thus, the overall copper recovery for the supergene material will be:

$$\text{Cu Recovery} = 94 \times (Cu_{total} - Cu_{WAS})/(Cu_{total})$$

13.3.4.3 Supergene – Gold

Averaging the gold recovery from Table 13-11, an average gold recovery of 69% to copper concentrate is obtained:

Au Recovery = 69%

13.3.4.4 Supergene Molybdenum

In the majority of the tests, no attempt was made to optimise the molybdenum recovery. For this reason, the molybdenum recovery is quite variable.

Examining the locked cycle tests in Table 13-10, an average molybdenum recovery of 55% to copper concentrate was chosen, which represents the average molybdenum recovery when the two low outliers are removed.

Recovery of molybdenum from the copper-molybdenum concentrate to a molybdenum concentrate was not specifically tested for the supergene material, but it is expected to be similar to that obtained in hypogene tests that achieved approximately 95% molybdenum recovery to a molybdenum concentrate. Molybdenum recovery throughout the plant is equal to the recovery to the copper-molybdenum concentrate multiplied by recovery to a molybdenum concentrate and is shown below:

Mo Recovery = 52.3%

13.3.4.5 Supergene – Silver

Unfortunately, silver recovery was not determined in all test programs. The 2011 test program followed silver recovery. Averaging the silver recovery from these locked cycle tests indicates that a silver recovery of 60% should be achievable:

Ag Recovery = 60%

13.3.4.6 Hypogene

Hypogene recoveries are based on the December 2012 flotation work performed by ALS Metallurgy on "fresh" core that had been drilled earlier specifically for flotation test work.

Table 13-12 shows cleaner circuit recoveries for both copper and molybdenum for all three locked cycle tests with hypogene material. Copper concentrate grades have been corrected to reflect the removal of molybdenum and represent final concentrate grades in terms of copper.

Table 13-12: Cleaner Circuit Recoveries for Locked Cycle Test Results

Test and Cycle No.	Cu Con Grade %Cu	Cu Recovery %	Mo Recovery %	Au Recovery %
WR Composite				
Cycle 4	17.8	96.4	95.0	86.0
Cycle 5	17.9	96.9	95.6	88.0
IX Composite				
Cycle 4	22.8	96.9	81.7	83.3
Cycle 5	21.2	96.7	80.8	80.6
PP Composite				
Cycle 4	26.1	97.1	90.4	88.0
Cycle 5	26.5	97.1	91.1	84.9

Copper, molybdenum, and gold recovery, when a primary grind size of 200 to 220 µm is used, is summarised in Table 13-12 and is based on both locked cycle testing and open circuit rougher flotation tests. Molybdenum recovery was variable, and the higher-grade molybdenum sample (IX) had the lowest molybdenum recovery, indicating that reagent conditions could possibly improve this recovery. Within the cleaning circuit copper and gold recoveries were similar, irrespective of the final copper concentrate grade.

Table 13-13: Predicted Recoveries to Copper/Molybdenum Concentrate

Process Stream	Cu Recovery, %	Mo Recovery, %	Au Recovery, %
Rougher Circuit Recovery	95	92	78
Cleaner Circuit Recovery	97	90	85
Metal Recovery	92	82.8	66

13.3.4.7 Hypogene – Copper Molybdenum Separation

One test was performed to determine how well molybdenum could be separated from a copper/molybdenum concentrate. The test indicated that approximately 95% molybdenum recovery could be achieved. Thus, the overall recovery of molybdenum will be equal to:

Mo Recovery = 78.7%

13.3.4.8 Hypogene – Silver Recovery

Hypogene silver recovery was followed in the last set of tests on fresh core. Reviewing these recoveries, a silver recovery of 50% was chosen.

Ag Recovery = 50.0%

13.3.4.9 Concentrate Quality

Estimates of the chemistry of the copper concentrate are summarised in Table 13-14. The table is comprised of the best estimates of analysis of concentrates produced in test work. Concentrate chemistry estimation is based on detailed analysis of test products, conducted at various metallurgical test facilities.

Table 13-14: Copper Concentrate Chemistry

Element	Average Expected Value	High Range	Low Range
Copper - %	28	30	25
Gold – g/t	25	30	15
Silver – g/t	120	180	80
Molybdenum -%	0.05	0.1	0.02
Iron - %	26	30	24
Sulphur - %	36	40	28
Arsenic – g/t	200	500	100
Antimony – g/t	250	400	100
Mercury – g/t	1	2	0.1
Cadmium – g/t	40	80	20
Fluorine – g/t	100	200	50
Silica - %	2	5	1

Key analytical results for the Casino Project molybdenum concentrate are summarised in Table 13-15. Limited test work allows for only an average chemistry estimate to be made for the molybdenum concentrate at this time.

Table 13-15: Molybdenum Concentrate Chemistry

Element	Average Expected Value
Molybdenum -%	56.0
Copper - %	0.25
Gold – g/t	1
Silver – g/t	10
Iron - %	1
Sulphur - %	38
Arsenic – g/t	1500
Antimony – g/t	100
Mercury – g/t	<1
Cadmium – g/t	30
Silica - %	1.5
Rhenium – g/t	130

13.4 DEWATERING TESTS

Flotation tailing from the 2008 test program piloting were submitted to Outotec for dynamic high-rate thickening tests. Results were favorable and a thickener underflow of over 55 percent solids was achieved. Flocculant addition was 22 g/t. The solids loading rate of 1.05 t/m^{2}h was demonstrated. Rheology on the thickened material was low.

13.5 LEACHING TESTS

13.5.1 Kappes, Cassiday and Associates (KCA)

KCA performed two studies in 1995 on the leaching of the oxide cap (oxide gold zone) and supergene (oxide copper) material. In the first study they leached a selection of oxide cap material with cyanide. In the second study they examined pre-leaching both oxide cap and supergene material with acid followed by cyanidation of the residue.

Gold extraction was affected by the quantity of copper leached during cyanidation and ranged from 10-97.4%. Average gold extraction was 79.9%.

Lime consumption during cyanidation averaged 3.9 kg/t without the acid pre-leach, and 4.1 kg/t with the acid pre-leach. Cyanide consumption was significant, averaging 5.5 kg/t without the acid pre-leach. There was not a significant difference between the lime consumption for the oxide gold composites and oxide copper composites.

13.5.2 SGS E&S Engineering Solutions Inc.

SGS E&S Engineering Solutions Inc. (at the time METCON) ran two column tests in 2008 on a composite sample blended to create gold and copper concentrations similar to the average reserve concentrations.

The ore was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 6-metre columns, and irrigated at 12 L/h/m^2. One column was leached "open cycle" – a 0.5 g/L NaCN solution was fed to the top of the column and the pregnant solution was collected and assayed. The second column was conducted as a "locked cycle" and solution was recycled. In the locked cycle column when the copper concentration in solution exceeded 50 mg/L, the solution was treated through a Sulphidization, Acidification, Recycling and Thickening (SART) pilot plant discussed in the next section, and the gold was recovered on activated carbon.

The gold, silver, and copper extractions from the open and locked cycle tests compare favorably. Although the gold extraction was slightly higher for the open cycle test, both tests produced good gold recovery considering the coarse crush size.

Cyanide consumptions were similar based on titrations and the amount of cyanide added to the system for the locked cycle column at approximately 0.5 kg/t. Lime consumptions were comparable to the bottle roll test work at approximately 3 kg/t.

Table 13-16: Extractions and Reagent Consumptions from Open Cycle and Locked Cycle Cyanidation

	Assays (calculated head) (g/t)			Percent Extraction			Reagent Consumption (kg/t)		
	Au	Ag	Cu	Au	Ag	Cu	NaCN*	NaCN**	CaO
Open	0.47	1.92	693	69.52	25.14	17.4	0.39		2.83
Locked	0.42	1.61	654	65.79	27.31	18.2	0.48	0.54	3.06

*based on titrations
**based on additions

A second set of testing was performed in 2013, which investigated metal recovery as a function of lithology. Based on an examination of a mine plan developed in 2013, it was determined that the heap leach would be primarily composed of Granodiorite (WR), Intrusive Breccia (IX) and Patton Porphyry (PP) lithology types with argillic (ARG) alteration.

The breakdown of heap leach ore tested by lithology type is as below:

Lithology Type	% of Ore
WR - Dawson Granodiorite	64%
IX - Intrusive Breccia	28%
PP - Patton Porphyry	8%

The ore was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 3-metre columns, and irrigated at 9.78 L/h/m^2. Each column was run in duplicate. The columns were operated in "open cycle". Solution containing 0.75 g/L free NaCN and 300 mg/L Cu (added to approximate that steady state Cu concentration that would be used to leach the heap leach ore in practice) was added to the top of the column to irrigate.

Table 13-17: Extractions and Reagent Consumptions from Column Tests Investigating Lithology

Ore Type	Head Assays (g/t)			Percent Extraction		Reagent Consumption (kg/t)	
	Au	Ag	Cu	Au	Ag	NaCN	CaO
WR	0.27	0.85	72.3	82.56	27.97	0.26	4.34
(dup)	0.27	0.85	72.3	81.90	27.78	0.20	4.18
average	-	-	-	82.2	27.9	0.23	4.3
IX	0.54	2.70	46.2	64.55	22.71	0.68	3.11
(dup)	0.54	2.70	46.2	62.10	16.63	0.44	3.10
average	-	-	-	63.3	19.7	0.56	3.1
PP	0.63	2.76	73.0	75.09	26.29	0.47	3.51
(dup)	0.63	2.77	73.0	73.28	26.01	0.19	3.38
average	-	-	-	74.2	26.2	0.33	3.4
Weighted average	-	-	-	76.3	25.4	0.33	3.3

Gold extraction for WR and PP lithologies are higher than the gold recoveries in previous testing, and gold recovery for IX lithology is higher indicating that there is some variability in gold extraction based on lithology. Cyanide and lime consumption are more variable but are similar to what was obtained in previous work.

As described in the Metallurgical Sample section above, the samples used in the latest SGS work are from the first five (5) metres of the deposit. Samples used in previous SGS testing (2008 report) were from drill core that was spatially distributed in the heap leach resource. There is a significant difference in the gold recovery between the locked cycle test completed with SART in the early test and the latest tests as shown in Table 13-18 below.

Table 13-18: Comparison of Recovery

	Recovery		Consumption	
	Au	Ag	NaCN	Lime
Locked cycle w/SART 2008	66	26	0.50	3.3
Weighted average 2013	76.3	25	0.33	3.9

13.5.3 SGS Canada

SGS Canada Burnaby received ten variability samples in 2021 for testing. Each sample was screened at four size fractions with each size fraction being assayed. The weighted head grade of the samples ranged from 0.12 – 0.94 g/t Au, 0.95 to 10 g/t Ag, and 13.8 to 712 g/t Cu. A master composite was made from the screened size fractions by

combining equal weight amounts of each sample (reconstituted from each fraction according to the size distribution). The Master composite was estimated to have a head grade of 0.46 g/t Au, 2.92 g/t Ag, and 316 g/t Cu. The master composite was split into two composite samples. One composite was split into two test charges for column tests with one charge being crushed to 100% minus 50 mm (P80 37 mm) and one charge being crushed to 100% minus 19 mm (P80 12.5 mm). The second composite was not crushed and is considered as ROM for a third column test. Three bottle roll tests (BRT) were also completed on the master composite, one to evaluate lime consumption, and two standard coarse ore tests to evaluate affect on recovery from copper in solution.

Parameters for the BRT tests to evaluate affect of copper in solution included: 1) crush size of 100% minus 10 mesh: 2) 45% solids density: 3) pH of 11 maintained; 4) 1000 ppm NaCN concentration maintained; and 5) 96 hours leach time. One test had no copper addition (BRT-A), and the other test was spiked with 0.3 g/L Cu as copper sulphate (BRT-B). Both tests had similar results, gold extractions slightly higher than 80%, silver extraction 20%, and copper extraction about 6%. Gold leach kinetics were also similar with approximately 35-40% gold extraction achieved within the first 6 hours and approximately 70% gold extraction after 48 hours. Gold extraction appeared to be nearing completion after 96 hours but did not reach a definite plateau. Silver extraction reached a plateau at approximately 20% extraction withing the first 24 hours of leaching.

Parameters for the three column tests to evaluate effect of crush size on recovery included: 1) crush size (ROM, P80 37 mm, and P80 12.5 mm); 2) pH of 11 maintained, 3) 1000 ppm NaCN and 300 ppm Cu concentration maintained; 4) 10 L/h/m^2 irrigation rate target, 5) 75 days leach time. As expected, gold extraction was higher with a finer crush size; 47% for ROM ore, 69% for the minus 50 mm crushed ore, and 80% for the minus 19 mm crushed ore after 75 days. Silver extraction was estimated to follow the same trend; 13% for ROM ore, 20% for the 50 mm crushed ore, and 23% for the minus 19 mm crushed ore. An additional 4 - 7% gold and 1% silver was extracted during a ~14-day rinse. The results of the column tests suggest that an optimal crush size for heap leaching of the master composite would be a crush size between 19 mm and 50 mm.

Table 13-19 shows the head grade of the sample used for the column tests at various crush sizes compared to the LOM head grade expected for the project. The gold and silver grade in the sample tested is 1.5 to 2 times higher than expected for the heap leach.

Table 13-19: Comparison of Head Grade

	Au g/t	Ag g/t	Cu g/t
Master Composite	0.46	2.13	313
Life of Mine for Project	0.27	1.95	360

Previous tests described above with a similar grade to what is expected for the LOM grade for the project had a leach residue assay for gold of 0.04 to 0.05 g/t. Using this residue assay with the LOM head grade would result in a gold recovery of 80 to 84%.

13.6 SART COPPER RECOVERY

In this process, a cyanide solution containing copper is treated to remove copper—gold is not affected.

In the locked cycle test described previously, the pregnant leach solution from the column was treated using a SART pilot plant several times before removing the gold with carbon and recycling the treated fluid to the column. The SART results are summarised in Table 13-20.

Table 13-20: SART Results

Pregnant Solution				Barren Solution after SART & Carbon				Copper	Reagent Consumption		
Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Removal (%)	(g/L solution treated)		
									S^{2-}	H_2SO_4	CaO
0.25	81	0.21	0.30	0.39	6.8	0.04	0.02	91.3	0.024	0.64	0.37

13.7 HYDRODYNAMIC CHARACTERIZATION

13.7.1 2015 HydroGeoSense

HydroGeoSense performed hydrodynamic characterization of two Casino head samples in 2015 to define the physical and hydraulic response of the ore that was expected to have a crush size of P100 50 mm, P80 38 mm. The samples included Composites Met 02-05 (WR) and Met 11-17 (IX) which represent 1:1 blends of the individual samples Met 02 and Met 05; and Met 11 and Met 17, respectively. Table 13-21 shows the head grade of the samples compared to the expected LOM grade of the ore to be leached.

Table 13-21: Comparison of Head Grade

	Au g/t	Ag g/t	Cu g/t
Life of Mine for Project	0.27	1.95	360
Met 02-05	0.32	1.00	100
Met 11-17	0.50	3.00	530

The characterization program consisted of:

- Review of available particle size distribution (PSD) information,
- Agglomeration trials to define the optimal moisture content for each PSD,
- Review of metallurgical and mineralogical data,
- Stacking Tests to determine the potential hydrodynamic behavior of the samples to select the most promising agglomeration conditions, and
- Hydrodynamic Column Tests on three PSD to determine the potential operation conditions

The samples were tested to determine the specific gravity of the solids. The average estimated SG for the samples is shown in Table 13-22.

Table 13-22: Solids Specific Gravity

Met 02-05 (WR)	Met 11-17 (IX)
2.66	2.63

Samples were screened and assayed at eight (8) individual sizes. The grade by size data indicate that Au and Ag are not uniformly distributed along the PSD of these composites. Met 02-05 has most of its metal value (35%) in the fines, size fractions smaller than 1.7 mm and Met 11-17 has most of its metal value (>25%) in the coarse material, size fractions larger than 25.4 mm.

Mineral characterization of the two samples indicates that Au occurs primarily as sub-microscopic native gold. Potential cyanide consuming minerals include metal sulphides including copper, iron, arsenic, and zinc; iron oxide-hydroxides

such as Limonite and Jarosite which can also act as preg-robbing; and secondary aluminosilicates and swelling clays. Bulk mineralogy indicates the gangue is composed predominantly by quartz and aluminosilicate minerals. The quartz fraction is quite different from each of the composites, 55.5% for Met 02-05, 41.7% for Met 11-17 and 36.8% for Met 19-20. Secondary minerals include Kaolinite (Al2Si2O5(OH)) and swelling clays representing 5.5%, 6.0% and 7.7% for Met 02-05, Met 11-17, and Met 19-20, respectively.

Key findings from the hydrodynamic characterization on the samples tested are as follows:

- The results from the Hydrodynamic Column Tests (HCTs) confirm that composites Met 02-05 and Met 11-17 will effectively support percolation leaching during the operation.
    - The total porosity of the non-agglomerated Met 02-05 at a heap height of 71 m is 31% which compared to the minimal requirement of 30% may indicate that this condition may not support a heap height beyond 80 m on its own and may need to be blended or agglomerated to achieve porosity above this height.
    - At a 140-m lift height, composite Met 11-17 satisfies the HydroGeoSense requirement that the minimum saturated hydraulic conductivity be larger than 1,000 times the application rate (assumed for this study as 6 L/h/m^2 based on the results from the hydrodynamic characterization).
- Ore preparation practices (crushing, blending, and agglomeration) may have a significant impact on the metallurgical performance of the ore.

13.7.2 2022 HydroGeoSense

In 2022, HydroGeoSense performed hydrodynamic characterization of three (3) Casino near surface samples representing fresh ore from the main lithology types (WR, PP, and IX) to evaluate the physical and hydraulic response of the ore that is expected to have a top size of 19 mm and a P80 of approximately 16 mm. The characterization focused on the percolation capacity for a heap height of up to 110 m representing non-agglomerated ore.

Table 13-23 shows the head grade of the samples compared to the expected LOM grade of the ore to be leached.

Table 13-23: Comparison of Head Grade

	Lithology	Au g/t	Ag g/t	Cu g/t
Life of Mine for Project	-	0.26	1.95	340
Average Grade thru Year 5		0.36	2.76	427
	WR	0.44	1.9	196
	IX	0.47	4.3	393
	PP	0.36	2.0	514
Weighted average of samples		**0.44**	**2.58**	**277**

The characterization program consisted of:

- Sample drying to determine the as-received moisture content
- Determination of resulting Particle size distribution under dry and wet screening
- Test charge preparation.
- Determination of the specific gravity (SG) for the gravel, sand, and fines fractions.
- Determination of the moisture content to achieve agglomeration levels L1 to L3

- Physical and hydraulic response under the Stacking Test® (ST) procedure to define the ability of the samples to support a static heap leach process with the samples to represent a non-agglomerated condition
- Characterization via the Hydrodynamic Column Test® (HCT) to define the potential hydrodynamic response of the material under percolation leaching.

The samples were tested to determine the specific gravity of the solids. The average estimated SG for the samples is shown in Table 13-24.

Table 13-24: Solids Specific Gravity

WR	IX	PP
2.76	2.69	2.73

The bulk density of the samples were measured representing the first lift (9.1 m for this test) and the bottom lift (after compaction, 105 m) . The bulk density for each of the samples is shown in Table 13-25.

Table 13-25: Dry Bulk Density (t/m^3)

Lift	WR	IX	PP
First	1.767	1.702	1.674
Bottom	1.843	1.771	1.750

The saturation and drain down moistures of the samples tested were measured representing the first lift (9.1 m for this test) and the bottom lift (after compaction, 105 m). The saturation moisture for each of the samples is shown in Table 13-26 and the drain down moisture for each of the samples is shown in Table 13-27.

Table 13-26: Saturation Moisture (%)

Lift	WR	IX	PP
First	8.0	7.6	8.7
Bottom	8.4	7.4	8.3

Table 13-27: Drain Down Moisture (%)

Lift	WR	IX	PP
First	6.7	5.9	6.3
Bottom	6.2	5.7	6.1

Key findings from the hydrodynamic characterization on the samples tested are as follows:

- There is a low level of fines (<75 microns) in all three crushed samples tested. Tests indicate that agglomeration will likely improve the performance of the heap it is not required.
- All three samples have a sufficiently large percolation capacity to support an irrigation rate of 10 l/hr/m^2 without excess liquid saturation.
- The samples are competent ore which are minimally affected by compaction.
- The saturation (operation) moisture ranges from 7.4% to 9.0% versus the 12% used in design.
- The Drain down (residual) moisture content ranges from 5.9% to 6.7% versus 10% used in design.
- There is a good balance between the capillary and body forces which should result in an efficient leaching process.

The test samples were not leached with cyanide, so recovery is not estimated. Results of the HCT's performed by HydroGeoSense indicate that the hydrodynamic properties of the samples tested are adequate to support a multi-lift heap leaching process.

13.8 DETERMINATION OF RECOVERIES, REAGENT, AND OTHER CONSUMABLE CONSUMPTIONS

As described in the preceding sections, the recoveries, reagent, and other consumable consumptions shown in Table 13-28 and Table 13-29 will be used. Where values were unknown, typical values based on M3's experience are used.

Table 13-28: Flotation Operational Parameters

Parameter	Value	Units
Recovery		
Copper		
Supergene	Recovery = 94 x (Cu_{total} – Cu_{WAS})/(Cu_{total})	percent
Hypogene	92	percent
Gold		
Supergene	69	percent
Hypogene	66	percent
Molybdenum		
Supergene	52.3	percent
Hypogene	78.7	percent
Silver Recovery		
Supergene	60	percent
Hypogene	50	percent
Parameters		
Bond work index	14.5	kWh/t
Primary grind size (P80)	200	µm
Regrind size (P80)	25	µm
Reagents		
Lime (93% active)		
Supergene	2.7	kg/t Ore
Hypogene	1.1	kg/t Ore
Aerophine 3418A		
Supergene	8.4	g/t Ore
Hypogene	4.0	g/t Ore
Aerofloat 208		
Supergene	16.7	g/t Ore
Hypogene	8.0	g/t Ore
MIBC	10	g/t Ore
Fuel Oil	7.0	g/t Ore
PAX	40	g/t Ore
NaSH	0.053	kg/t Ore
Flocculant	25.4	g/t Ore
Liners		
SAG Mill – Liners	0.040	kg/t Ore
Ball Mill – Liners	0.048	kg/t Ore
Grinding Media		
SAG Mill – Balls	0.400	kg/t Ore
Ball Mill – Balls	0.400	kg/t Ore
Regrind – Balls	0.0410	kg/t Ore

Table 13-29: Heap Leach Operational Parameters

Parameter	Value	Units
Crush size (P100)	19	mm
Irrigation rate	12	L/h/m^2
Lift height	8	M
Leach solution application, primary	60	days
Recovery		
Gold recovery (initial)	77	percent
Gold recovery (final)	80	percent
Copper recovery	18	percent
Silver recovery	26	percent
Reagent Consumptions		
NaHS	0.025	kg/t Ore
Sulfuric acid	0.328	kg/t Ore
Hydrochloric acid	0.010	kg/t Ore
Lime (CaO) (93% active)	3.516	kg/t Ore
Sodium hydroxide	0.130	kg/t Ore
Sodium cyanide (NaCN)	0.500	kg/t Ore
Activated Carbon	0.011	kg/t Ore
Antiscalant	0.003	kg/t Ore
Flocculant	0.350	g/t Ore
Wear Consumptions		
Primary crusher liners	0.040	kg/t
Secondary crusher liners	0.085	kg/t
Tertiary crusher liners	0.085	kg/t

The amount of test work completed for the heap leach ore is quite small by comparison with a typical feasibility study. Most of the test work has been performed on samples representing the first years of production. Based on the latest tests completed to evaluate heap leach recovery by crush size a third stage of crushing with a crushed product size of minus 19 mm (P80 – 16 mm) has been included in this feasibility study. The gold recovery assumed for this feasibility study is 80% (77% from the leach cycle plus an additional 3% recovered during subsequent leaching/rinsing). However, additional metallurgical test work is recommended to measure the variability of the ore and its influence on the metallurgical response of the ore at the crush size selected.

14 MINERAL RESOURCE ESTIMATES

14.1 MINERAL RESOURCE

The Mineral Resource for the Casino Project includes Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resources amenable to heap leach recovery methods (leach material). Also, the Mineral Resource is reported inclusive of the Mineral Reserve presented in Section 15. Table 14-1 presents the Mineral Resource for mill material. Mill material includes the supergene oxide (SOX), supergene sulphide (SUS) and hypogene sulphide (HYP) mineral zones. Measured and Indicated Mineral Resources amount to 2.26 billion tonnes at 0.15% total copper, 0.18 g/t gold, 0.016% moly and 1.4 g/t silver and contained metal amounts to 7.45 billion pounds of copper, 12.9 million ounces gold, 791.2 million pounds of moly and 103.1 million ounces of silver. Inferred Mineral Resource is an additional 1.37 billion tonnes at 0.10% total copper, 0.14 g/t gold, 0.009% moly and 1.1 g/t silver and contained metal amounts to 3.03 billion pounds of copper, 6.1 million ounces of gold, 286.0 million pounds moly and 50.5 million ounces of silver for the Inferred Mineral Resource in mill material.

Table 14-2 presents the Mineral Resource for leach material. Leach material is oxide dominant leach cap (LC) mineralization. The emphasis of leaching is the recovery of gold in the leach cap. Copper grades in the leach cap are low, but it is expected some metal will be recovered. Measured and Indicated Mineral Resources amount to 231.7 million tonnes at 0.04% total copper, 0.25 g/t gold and 1.9 g/t silver and contained metal amounts to 196.9 million pounds of copper, 1.88 million ounces gold and 14.1 million ounces of silver. Inferred Mineral Resource is an additional 40.9 million tonnes at 0.05% total copper, 0.20 g/t gold and 1.4 g/t silver and contained metal amounts to 46.9 million pounds of copper, 270,000 ounces of gold and 1.9 million ounces of silver for the Inferred Mineral Resource in leach material.

Table 14-3 presents the Mineral Resource for combined mill and leach material for copper, gold, and silver. Measured and Indicated Mineral Resources amount to 2.49 billion tonnes at 0.14% total copper, 0.18 g/t gold and 1.5 g/t silver. Contained metal amounts to 7.64 billion pounds copper, 14.8 million ounces gold and 117.2 million ounces of silver for Measured and Indicated Mineral Resources. Inferred Mineral Resource is an additional 1.41 billion tonnes at 0.10% total copper, 0.14 g/t gold and 1.2 g/t silver. Contained metal amounts to 3.08 billion pounds of copper, 6.3 million ounces of gold and 52.3 million ounces of silver for the Inferred Mineral Resource. The Mineral Resource for molybdenum is as shown with mill material since it will not be recovered for leach material.

The Mineral Resources are based on a block model developed by IMC during December 2021. This updated model incorporated the 2020 Western drilling and updated geologic models.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a floating cone pit shell to demonstrate “reasonable prospects for eventual economic extraction” to meet the definition of Mineral Resources in NI 43-101.

Figure 14-1 shows the constraining pit shell that is based on Measured, Indicated, and Inferred Mineral Resource.

Table 14-1: Mineral Resource for Mill Material at C$6.11 NSR Cutoff

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq %	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Measured	144.9	40.09	0.30	0.38	0.024	2.1	0.64	953	1.8	75.2	9.6
Indicated	2,114.2	20.34	0.14	0.16	0.015	1.4	0.29	6,493	11.1	716.0	93.5
M+I	**2,259.0**	**21.60**	**0.15**	**0.18**	**0.016**	**1.4**	**0.31**	**7,446**	**12.9**	**791.2**	**103.1**
Inferred	1,371.5	15.41	0.10	0.14	0.009	1.1	0.21	3,029	6.1	286.0	50.5

Table 14-2: Mineral Resource for Leach material at C$6.61 NSR Cutoff

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	43.3	23.79	0.05	0.44	2.7	0.47	51.5	0.62	3.7
Indicated	188.4	11.47	0.04	0.21	1.7	0.23	145.4	1.27	10.4
M+I	**231.7**	**13.77**	**0.04**	**0.25**	**1.9**	**0.27**	**196.9**	**1.88**	**14.1**
Inferred	40.9	11.33	0.05	0.20	1.4	0.22	46.9	0.27	1.9

Table 14-3: Mineral Resource for Copper, Gold, and Silver (Mill and Leach)

Resource Category	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	188.2	36.34	0.24	0.40	2.2	1,005.0	2.4	13.3
Indicated	2,302.6	19.61	0.13	0.17	1.4	6,638.1	12.4	103.9
M+I	**2,490.7**	**20.88**	**0.14**	**0.18**	**1.5**	**7,643.1**	**14.8**	**117.2**
Inferred	1,412.5	15.30	0.10	0.14	1.2	3,075.5	6.3	52.3

Notes:

1. The Mineral Resources have an effective date of 29 April 2022, and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources for leach material are based on prices of US$3.50/lb copper, US$1650/oz gold and US$22/oz silver.
5. Mineral Resources for mill material are based on prices of US$3.50/lb copper, US$1650/oz gold, US$22/oz silver, and US$12.00/lb moly.
6. Mineral Resources are based on NSR Cutoff of C$6.61/t for leach material and C$6.11/t for mill material.
7. NSR value for leach material is as follows:
   NSR (C$/t) = $15.21 x copper (%) + $50.51 x gold (g/t) + $0.210 x silver (g/t), based on copper recovery of 18%, gold recovery of 80% and silver recovery of 26%.
8. NSR value for hypogene sulphide mill material is:
   NSR (C$/t) = $73.81 x copper (%) + $41.16 x gold (g/t) + $213.78 x moly (%) + $0.386 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 50% silver and 78.6% moly.
9. NSR value for supergene (SOX and SUS) mill material is:
   NSR (C$/t) = $80.06 x recoverable copper (%) + $43.03 x gold (g/t) + $142.11 x moly (%) + $0.464 x silver (g/t), based on recoveries of 69% gold, 60% silver and 52.3% moly. Recoverable copper = 0.94 x (total copper – soluble copper).
10. Table 14-6 accompanies this Mineral Resource and shows all relevant parameters.
11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. AuEq and CuEq values are based on prices of US$3.50/lb copper, US$1650/oz gold, US$22/oz silver, and US$12.00/lb moly, and account for all metal recoveries and smelting/refining charges.
13. The Mineral Resource is reported inclusive of the Mineral Reserve.

14.2 SENSITIVITY TO NSR CUTOFF

Table 14-4 shows resources at varying NSR Cut-offs for mill material. All tabulations are contained by the constraining pit shell used for the base case Mineral Resource at C$6.11 per tonne (highlighted). Increasing the NSR Cutoff by 30% to C$8/t has only a modest effect on the size of the Mineral Resource amenable to milling, decreasing Measured and Indicated Mineral Resource tonnes by 5% and the contained copper and gold by 1.2% and 2.3% respectively.

Table 14-5 shows resources at varying NSR Cut-offs for leach material. Again, all tabulations are contained by the constraining pit shell used for the base case Mineral Resource. The base case resource at an NSR Cutoff of C$6.61 per tonne is highlighted. Increasing the NSR Cutoff of leach material to C$8/t reduces the Measured and Indicated Mineral Resource tonnes by 14% and increases the gold grade by 8% so contained gold is reduced by only 7%.

Table 14-4: Mineral Resource – Mill Material by Various NSR Cut-offs (C$)

NSR Cog ($/t)	Resource Category	Tonnes Mt	NSR ($/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
6.11	Measured	144.9	40.09	0.30	0.38	0.024	2.1	0.64	953	1.8	75.2	9.6
	Indicated	2,114.2	20.34	0.14	0.16	0.015	1.4	0.29	6,493	11.1	716.0	93.5
	M+I	**2,259.0**	**21.60**	**0.15**	**0.18**	**0.016**	**1.4**	**0.31**	**7,446**	**12.9**	**791.2**	**103.1**
	Inferred	1,371.5	15.41	0.10	0.14	0.009	1.1	0.21	3,029	6.1	286.0	50.5
8	Measured	144.1	40.27	0.30	0.39	0.024	2.1	0.65	953	1.8	75.3	9.5
	Indicated	2,003.9	21.06	0.15	0.17	0.016	1.4	0.30	6,406	10.8	711.3	90.8
	M+I	**2,147.9**	**22.35**	**0.16**	**0.18**	**0.017**	**1.5**	**0.32**	**7,359**	**12.6**	**786.5**	**100.4**
	Inferred	1,202.8	16.58	0.11	0.15	0.011	1.2	0.23	2,864	5.6	278.4	46.4
12	Measured	141.5	40.81	0.30	0.39	0.024	2.1	0.66	949	1.8	74.9	9.5
	Indicated	1,686.3	23.14	0.16	0.18	0.018	1.5	0.33	5,985	9.8	680.3	81.9
	M+I	**1,827.8**	**24.51**	**0.17**	**0.20**	**0.019**	**1.6**	**0.35**	**6,934**	**11.5**	**755.2**	**91.3**
	Inferred	807.5	19.86	0.13	0.16	0.014	1.3	0.27	2,350	4.2	243.9	34.8
16	Measured	136.4	41.80	0.31	0.40	0.025	2.1	0.67	933	1.8	74.6	9.3
	Indicated	1,316.6	25.70	0.18	0.20	0.021	1.6	0.36	5,196	8.3	597.9	69.4
	M+I	**1,453.0**	**27.21**	**0.19**	**0.22**	**0.021**	**1.7**	**0.39**	**6,128**	**10.1**	**672.5**	**78.7**
	Inferred	434.1	25.04	0.17	0.20	0.018	1.5	0.34	1,637	2.8	167.5	21.1
24	Measured	120.8	44.59	0.33	0.43	0.027	2.2	0.72	873	1.7	71.9	8.7
	Indicated	628.6	32.25	0.22	0.25	0.026	1.9	0.45	3,104	5.0	365.8	38.4
	M+I	**749.4**	**34.24**	**0.24**	**0.28**	**0.026**	**2.0**	**0.49**	**3,977**	**6.6**	**437.7**	**47.1**
	Inferred	169.9	34.41	0.24	0.26	0.025	1.9	0.47	902	1.4	92.1	10.2
30	Measured	102.3	47.73	0.35	0.46	0.029	2.3	0.77	790	1.5	65.4	7.7
	Indicated	328.2	37.29	0.26	0.29	0.030	2.2	0.52	1,888	3.0	213.4	22.8
	M+I	**430.5**	**39.77**	**0.28**	**0.33**	**0.029**	**2.2**	**0.58**	**2,678**	**4.5**	**278.9**	**30.5**
	Inferred	108.7	38.69	0.27	0.29	0.027	2.1	0.53	652	1.0	65.0	7.2

Table 14-5: Mineral Resource – Leach Material by Various NSR Cut-offs (C$)

NSR Cog ($/t)	Resource Category	Tonnes Mt	NSR ($/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
6.61	Measured	43.3	23.79	0.05	0.44	2.7	0.47	51.5	0.62	3.7
	Indicated	188.4	11.47	0.04	0.21	1.7	0.23	145.4	1.27	10.4
	M+I	**231.7**	**13.77**	**0.04**	**0.25**	**1.9**	**0.27**	**196.9**	**1.88**	**14.1**
	Inferred	40.9	11.33	0.05	0.20	1.4	0.22	46.9	0.27	1.9
8	Measured	41.9	24.33	0.06	0.45	2.7	0.48	50.8	0.61	3.7
	Indicated	157.1	12.29	0.04	0.23	1.8	0.24	121.2	1.14	9.1
	M+I	**199.0**	**14.83**	**0.04**	**0.27**	**2.0**	**0.29**	**172.0**	**1.75**	**12.8**
	Inferred	29.1	12.96	0.06	0.23	1.4	0.26	36.6	0.22	1.3
10	Measured	40.0	25.04	0.06	0.47	2.8	0.50	49.4	0.60	3.6
	Indicated	104.4	13.96	0.03	0.26	2.0	0.28	78.2	0.87	6.7
	M+I	**144.4**	**17.03**	**0.04**	**0.32**	**2.2**	**0.34**	**127.7**	**1.47**	**10.3**
	Inferred	15.1	16.58	0.07	0.30	1.5	0.33	24.7	0.15	0.7
12	Measured	37.6	25.95	0.06	0.49	2.9	0.51	47.2	0.59	3.5
	Indicated	66.6	15.69	0.03	0.29	2.1	0.31	47.0	0.62	4.7
	M+I	**104.1**	**19.39**	**0.04**	**0.36**	**2.4**	**0.38**	**94.2**	**1.21**	**8.2**
	Inferred	10.6	19.04	0.08	0.35	1.6	0.38	17.6	0.12	0.5
14	Measured	34.7	27.03	0.06	0.51	3.0	0.54	43.6	0.56	3.3
	Indicated	42.8	17.21	0.03	0.32	2.4	0.34	31.1	0.44	3.2
	M+I	**77.5**	**21.61**	**0.04**	**0.40**	**2.6**	**0.43**	**74.7**	**1.01**	**6.5**
	Inferred	7.4	21.54	0.08	0.40	1.8	0.42	12.8	0.09	0.4

14.3 MINERAL RESOURCE PARAMETERS

14.3.1 Metal Prices

Table 14-6 shows the economic and recovery parameters for the Mineral Resource estimate. Metal prices for the Mineral Resource estimate are US$3.50 per pound copper, US$1,650 per ounce gold, US$22 per ounce silver and US$12 per pound moly. A conversion of US$0.80 = C$1.00 was used to convert the prices to C$. IMC believes these prices to be reasonable based on the following: 1) historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economists.

14.3.2 Cost and Recovery Estimates

Mining Cost

The base mining cost of C$1.934 per total tonne was estimated by IMC. The estimate is based on mining equipment requirements specific to Casino and typical prices for fuel, blasting agents, equipment parts, and labor, etc.

Processing of Mill Material

Mill material refers to the supergene oxide, supergene sulphide, and hypogene sulphide zones of the mineral deposit. The processing will be in a conventional sulphide flotation plant that will produce copper and molybdenum concentrates that will be sold to commercial copper smelters and molybdenum roasting plants. The base unit costs for processing and G&A are estimated at C$5.724 and C$0.385 per tonne, respectively. These are based on estimates developed by M3. The estimated plant recoveries for gold, moly, and silver in the supergene and hypogene zones are shown on Table 14-6. Copper recovery is estimated at 92.2% for hypogene sulphide material. The plant recovery for supergene material is estimated as follows:

Copper recovery = 94% (Cut% - Cuw%) / Cut%

Where,

Cut% = Total copper grade
Cuw% = Weak acid soluble copper grade

The copper, gold, and silver payable percentages shown on Table 14-6 are typical terms for copper concentrates, assuming a clean concentrate with a copper concentrate grade of 28% copper or greater. The off-site cost per pound of copper is estimated at C$0.486. Gold and silver refining is estimated at C$6.574/oz and C$1.347/oz, respectively.

Note that the off-site cost for molybdenum is assumed to be accounted in the 85% payable percentage for molybdenum in concentrate, i.e., this is assumed to be the net payable after treatment and transportation charges. This is applicable to a clean molybdenum concentrate with a moly grade of about 50% or greater.

The table also shows an NSR royalty of 2.75% and a concentrate loss of 0.5% are incorporated into the economic calculations.

Processing of Leach Material

Leach material refers to the leach capping of the mineral deposit. Processing is by crushing and heap leaching with cyanide. Gold and silver from the heap leach will report to a typical doré which will be sent to a refinery. The SART process will be used to extract copper from the cyanide solution and produce a copper concentrate that can be sold to conventional copper smelters. Heap leach processing is estimated at C$6.227 per tonne. The G&A cost of C$0.385 per tonne is also applied to leach material.

Heap leach recoveries are estimated at 18% for copper, 80% for gold, and 26% for silver. Typical terms for refining costs are shown and are C$2.122/oz gold and C$1.103/oz silver. The payable percentage is estimated at 98% for gold and silver.

It is also assumed that the SART process will produce a copper concentrate with a grade of about 60% copper. Smelting and refining terms are assumed the same as for the flotation concentrate. This results in a smelting, refining, and freight charge of about C$0.271 per pound.

14.3.3 NSR Calculations

Due to multiple mineral products and also the variable recovery for copper in the supergene zones, NSR values, in Canadian Dollars, were calculated for each model block to use to classify blocks into potential resource and waste. For the leach material:

NSR_au = ($2063 – $2.122) x 0.80 x 0.98 x 0.9725 x gold (g/t) / 31.103 = C$50.51 x gold (g/t)

NSR_cu = ($4.38 - $0.271) x 0.18 x 0.965 x 0.995 x 0.9725 x copper (%) x 22.046
= C$15.21 x copper (%)

NSR_ag = ($27.50 - $1.103) x 0.26 x 0.98 x 0.9725 x silver (g/t) / 31.103 = C$0.210 x silver (g/t)

NSR = NSR_au + NSR_cu + NSR_ag

The internal NSR cut-off for leach material is the processing + G&A cost of C$6.61 per tonne since all the recoveries and refining costs are accounted for in the NSR calculation. Internal cut-off grade applies to blocks that have to be removed from the pit, so the mining cost is a sunk cost. Internal cut-off is also generally the minimum cut-off that would

be evaluated for mine scheduling. The Mineral Resource tabulation for leach material on Table 14-2 is based on the internal cut-off. The breakeven NSR cut-off grade for leach material is C$8.55 per tonne (mining plus processing and G&A). The 0.9725 parameter in the equations accounts for the royalty.

For processing of hypogene sulphide material the NSR values are calculated as:

NSR_cu = ($4.38 – $0.486) x 0.922 x 0.965 x 0.995 x 0.9725 x copper (%) x 22.046
= C$73.81 x copper (%)

NSR_au = ($2063 – $6.574) x 0.66 x 0.975 x 0.995 x 0.9725 x gold (g/t) / 31.103
= C$41.16 x gold (g/t)

NSR_mo = $15.00 x 0.786 x 0.85 x 0.995 x 0.9725 x moly (%) x 22.046 = C$213.78 x moly (%)

NSR_ag = ($27.50 - $1.347) x 0.50 x 0.95 x 0.995 x 0.9725 x silver (g/t) / 31.103
= C$0.386 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

For processing of supergene material, the NSR values are calculated as:

NSR_cu = ($4.38 – $0.486) x 0.965 x 0.995 x 0.9725 x rec_cu (%) x 22.046
= C$80.06 x rec_cu (%)

NSR_au = ($2063 – $6.574) x 0.69 x 0.975 x 0.995 x 0.9725 x gold (g/t) / 31.103
= C$43.03 x gold (g/t)

NSR_mo = $15.00 x 0.523 x 0.85 x 0.995 x 0.9725 x moly (%) x 22.046 = C$142.11 x moly (%)

NSR_ag = ($27.50 - $1.347) x 0.60 x 0.95 x 0.995 x 0.9725 x silver (g/t) / 31.103
= C$0.464 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

where,

rec_cu = 0.94 x (Cut% – Cuw%)

The internal NSR cut-off for flotation is the processing plus G&A cost of C$6.11. Breakeven NSR cut-off is C$8.04. The 0.995 parameter in the equations accounts for concentrate loss and the 0.9725 parameter accounts for the royalty.

Table 14-6: Economic Parameters for Mineral Resource (C$)

Parameter	Units	Mill Material			Heap Leach
		SOX	SUS	HYP	
Commodity Prices and Exchange Rate:					
Copper Price Per Pound (US$)	(US$)	3.50	3.50	3.50	3.50
Gold Price Per Ounce (US$)	(US$)	1650	1650	1650	1650
Silver Price Per Ounce (US$)	(US$)	22.00	22.00	22.00	22.00
Molybdenum Price Per Pound (US$)	(US$)	12.00	12.00	12.00	12.00
Exchange Rate (CAD to $US)	(none)	0.80	0.80	0.80	0.80
Copper Price Per Pound (C$)	(C$)	4.38	4.38	4.38	4.38
Gold Price Per Ounce (C$)	(C$)	2063	2063	2063	2063
Silver Price Per Ounce (C$)	(C$)	27.50	27.50	27.50	27.50
Molybdenum Price Per Pound (C$)	(C$)	15.00	15.00	15.00	15.00
Mining Cost Per Total Tonne:					
Mining Cost Per Total Tonne	(C$)	1.934	1.934	1.934	1.934
Processing and G&A Per Tonne Processed					
Processing	(C$)	5.724	5.724	5.724	6.227
G&A	(C$)	0.385	0.385	0.385	0.385
Total Processing and G&A	(C$)	6.109	6.109	6.109	6.612
Average Plant Recoveries:					
Copper Recovery (Note 1)	(%)	59.5%	81.7%	92.2%	18.0%
Gold Recovery	(%)	69.0%	69.0%	66.0%	80.0%
Silver Recovery	(%)	60.0%	60.0%	50.0%	26.0%
Moly Recovery	(%)	52.3%	52.3%	78.6%	N.A.
Refinery Payables:					
Copper Payables	(%)	96.5%	96.5%	96.5%	96.5%
Gold Payables	(%)	97.5%	97.5%	97.5%	98.0%
Silver Payables	(%)	95.0%	95.0%	95.0%	98.0%
Molybdenum Payables	(%)	85.0%	85.0%	85.0%	N.A
Payable Concentrate (0.5% Conc Loss)	(%)	99.5%	99.5%	99.5%	Cu Only
Offsite Costs:					
Copper SRF Cost Per Pound	(C$)	0.486	0.486	0.486	0.271
Gold Refining Per Ounce	(C$)	6.574	6.574	6.574	2.122
Silver Refining Per Ounce	(C$)	1.347	1.347	1.347	1.103
Molybdenum Freight/Treatment Per Pound	(C$)	Note 2	Note 2	Note 2	N.A.
NSR Royalty:	%	2.75%	2.75%	2.75%	2.75%
NSR Factors:					
Copper Factor (Note 3)	(C$/t)	47.63	65.41	73.81	15.21
Gold Factor (Note 3)	(C$/t)	43.03	43.03	41.16	50.51
Silver Factor (Note 3)	(C$/t)	0.464	0.464	0.386	0.210
Moly Factor (Note 3)	(C$/t)	142.11	142.11	213.78	N.A.
Equivalency Factors		CuEq	CuEq	CuEq	AuEq
Copper Factor	(none)	1.00	1.00	1.00	0.301
Gold Factor	(none)	0.903	0.658	0.558	1.00
Silver Factor	(none)	0.0097	0.0071	0.0052	0.0042
Moly Factor	(none)	2.983	2.173	2.896	N.A
NSR Cutoff Grades:					
Breakeven Cutoff (C$/t)	(C$/t)	8.04	8.04	8.04	8.55
Internal Cutoff (C$/t)	(C$/t)	6.11	6.11	6.11	6.61

Note 1: Average recovery based on Recovery = 94% x (Cutotal – CuWAS)/(Cutotal) for SOX and SUS
Note 2: Moly offsite costs are accounted in payable percentage
Note 3: NSR factors are applied to model grades, copper factor for SOX and SUS is based on average recovery.

The copper and gold equivalent grades on the Mineral Resource tables account for all metal recoveries and smelting/refining charges. The equivalency factors shown on Table 14-6 are derived from the NSR factors as follows for hypogene sulphide mill material:

CuEq% = copper(%) + (41.16/73.81) x gold(g/t) + (213.78/73.81) x moly(%) + (0.386/73.81) x silver(g/t)
CuEq% = copper(%) + 0.558 x gold(g/t) + 2.896 x moly(%) + 0.0052 x silver (g/t)

The calculations are similar for the other material types. For the supergene oxide and supergene sulphide the copper NSR factor is based on the average copper recoveries.

14.3.4 Slope Angles

Slope angles recommendations were developed by Knight Piésold Ltd. (KP) and documented in the report "Open Pit Geotechnical Design", dated October 12, 2012.

Forty-five-degree inter-ramp angles were recommended for most of the slope sectors. The north sectors of the main pit and west pit were recommended to be designed at 42-degree inter-ramp angles. For the small amount of overburden on the north wall the recommended angle was 27 degrees. The slope angle recommendations also specified that there be no more than 200 m of vertical wall at the inter-ramp angle without an extra wide catch bench (16 m instead of 8 m).

IMC used an overall slope angle of 41 degrees in the floating cone runs to approximate overall slope angles with the KP inter-ramp angles.

14.4 ADDITIONAL INFORMATION

The Mineral Resources are classified in accordance with the May 2014 Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards – For Mineral Resources and Mineral Reserves" adopted by the CIM Council (as amended, the "CIM Definition Standards") in accordance with the requirements of NI 43-101. Mineral Reserve and Mineral Resource estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained.

There is no guarantee that any of the Mineral Resources will be converted to Mineral Reserve. The Inferred Mineral Resources included in this Technical Report meet the current definition of Inferred Mineral Resources. The quantity and grade of Inferred Mineral Resources are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as an Indicated Mineral Resource. It is, however, expected that the majority of Inferred Mineral Resource could be upgraded to Indicated Mineral Resource with continued exploration.

IMC does not believe that there are significant risks to the Mineral Resource estimates based on environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. The Project is in a jurisdiction friendly to mining. The most significant risks to the Mineral Resource are related to economic parameters such as prices lower than forecast, recoveries lower than forecast, or costs higher than the current estimates.

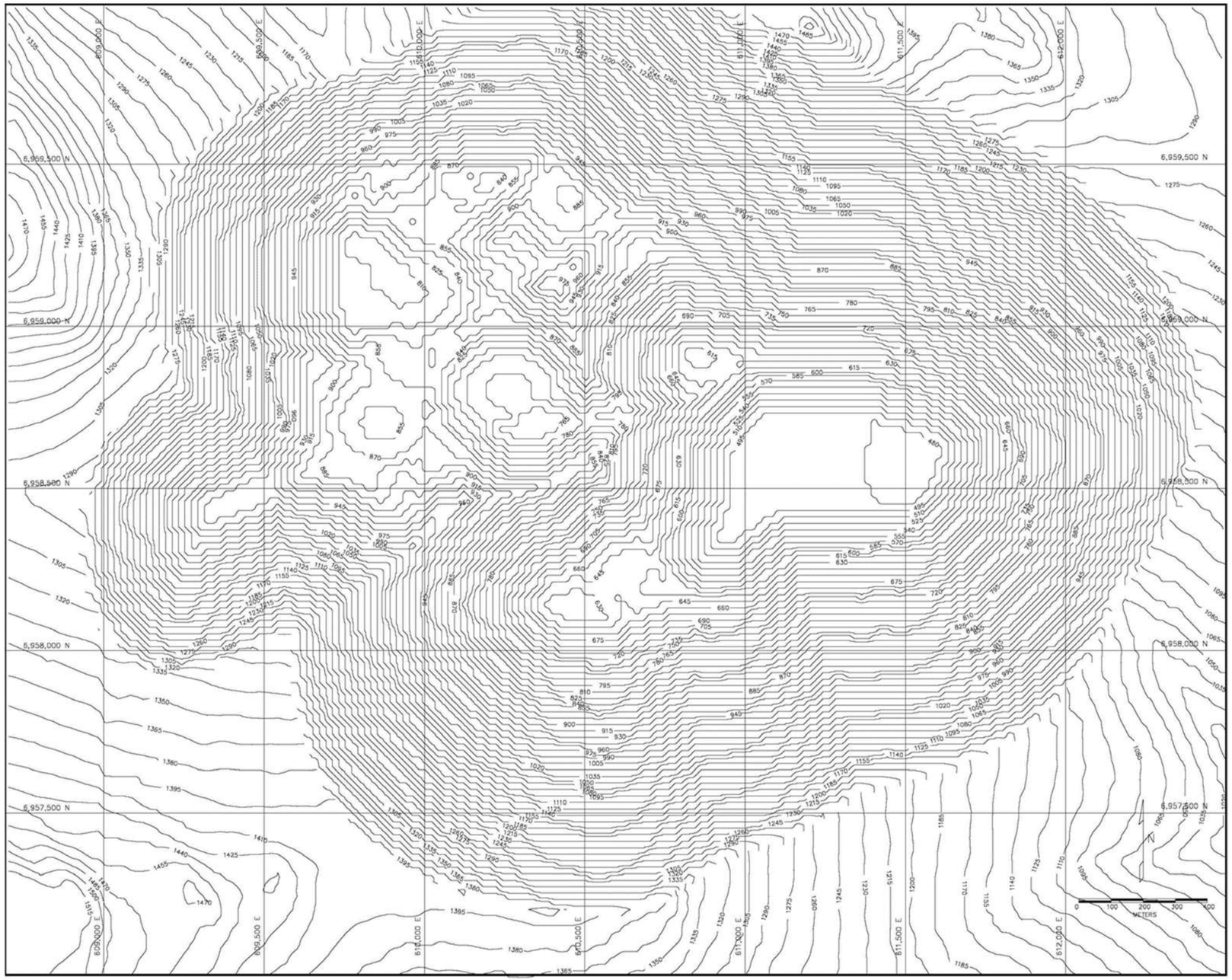


Figure 14-1: Floating Cone Shell for Mineral Resource (IMC, 2022)

14.5 DESCRIPTION OF THE BLOCK MODEL

14.5.1 General

A 3D block model was developed by IMC during December 2021. The block model is based on 20 m by 20 m by 15 m high blocks. The previous resource model of record was developed by IMC during June 2020. Prior to that, the 2010 model developed by G. Giroux was the model used for work from 2010 up to June 2020.

14.5.2 Drilling Data

The drillhole database provided to IMC included 464 holes that represented 128,129 metres of drilling. Table 14-7 summarizes the drilling by date and company.

Table 14-7: Casino Drilling by Date and Company

Years	Company	No. of Holes	Metres
1992-1994	Pacific Sentinel Gold Corp.	236	73,085
2008-2012	Western Copper and Gold	112	29,765
2019	Western Copper and Gold	69	13,458
2020	Western Copper and Gold	47	11,821
TOTAL		**464**	**128,129**

Figure 14-2 shows the hole locations and also the location of cross sections that will be presented for this report. The newest holes, i.e., the 2020 holes, are highlighted in red on the figure.

The analyses of interest for the study included total copper, weak acid soluble copper, gold, moly, and silver. Also available in the database is a complete suite of multi-element analyses. IMC's scope of work did not include a detailed review of the drilling data; that was performed by other QP's for this study.

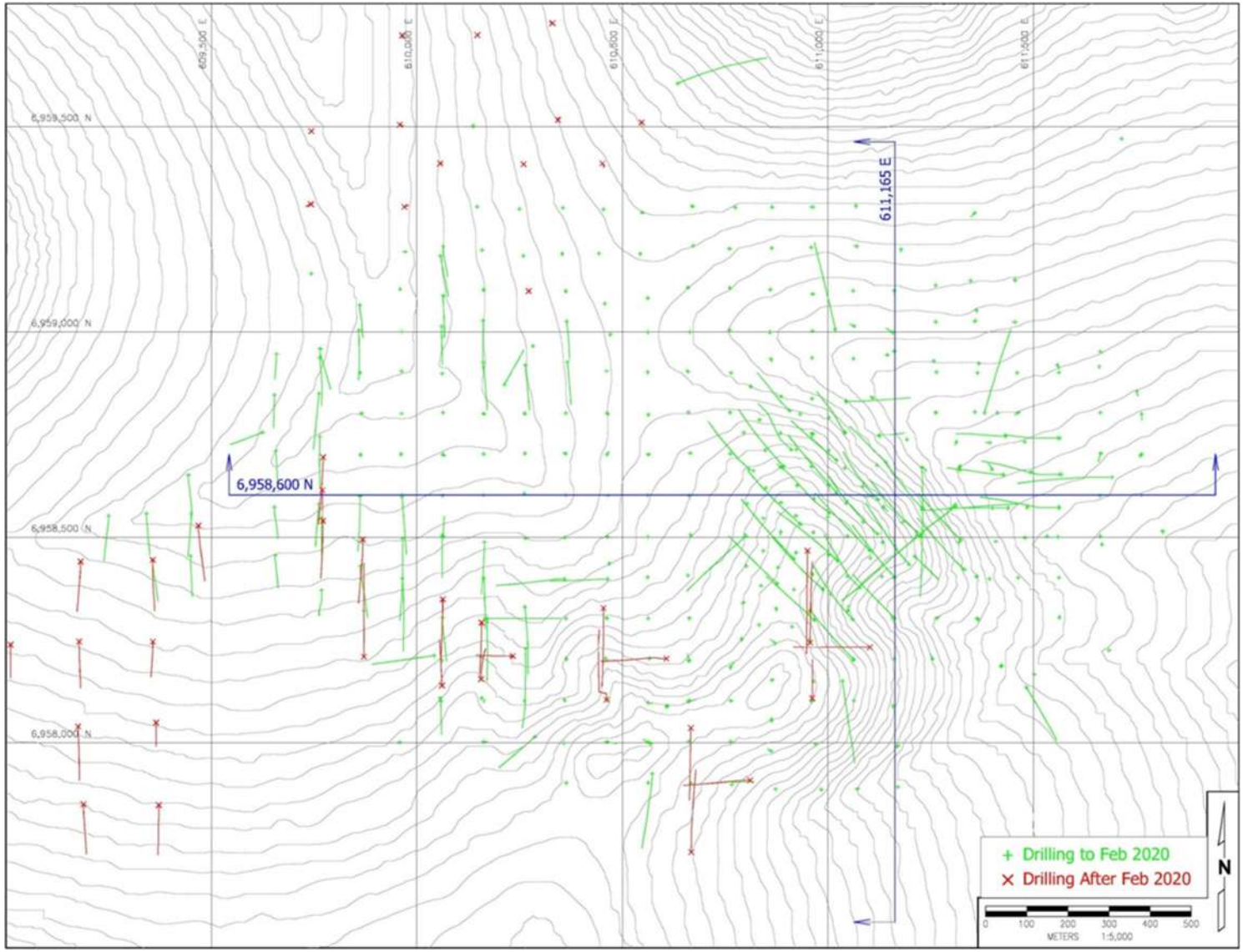


Figure 14-2: Hole Location Map (IMC, 2022)

14.5.3 Geologic Controls

Oxidation Zone Types

The most important geologic control, particularly for copper mineralization, is the oxidation zones. Table 14-8 shows the zone names, codes used for modeling, and a description. The overburden is a relatively thin, highly weathered zone, near the top of current topography. There are some mineralized intervals in the overburden. The leach cap (LC) is a highly oxidized domain where the copper mineralization has largely been dissolved in acids over time and transported to the underlying supergene zones. The gold, silver, and molybdenum mineralization was not subject to the dissolution, at least to any significant degree; in particular, there are significant gold values in the LC. The supergene domains have been divided into oxide dominant supergene oxide (SOX) and sulphide dominant supergene sulphide (SUS). Copper from the LC has been deposited in those zones, elevating the copper grade compared to the other domains. The hypogene sulphide (HYP) zone underlies the LC, SOX, and SUS zones. Mineralization is sulphidic in nature; percent of oxidation is very low, typically less than 10%.

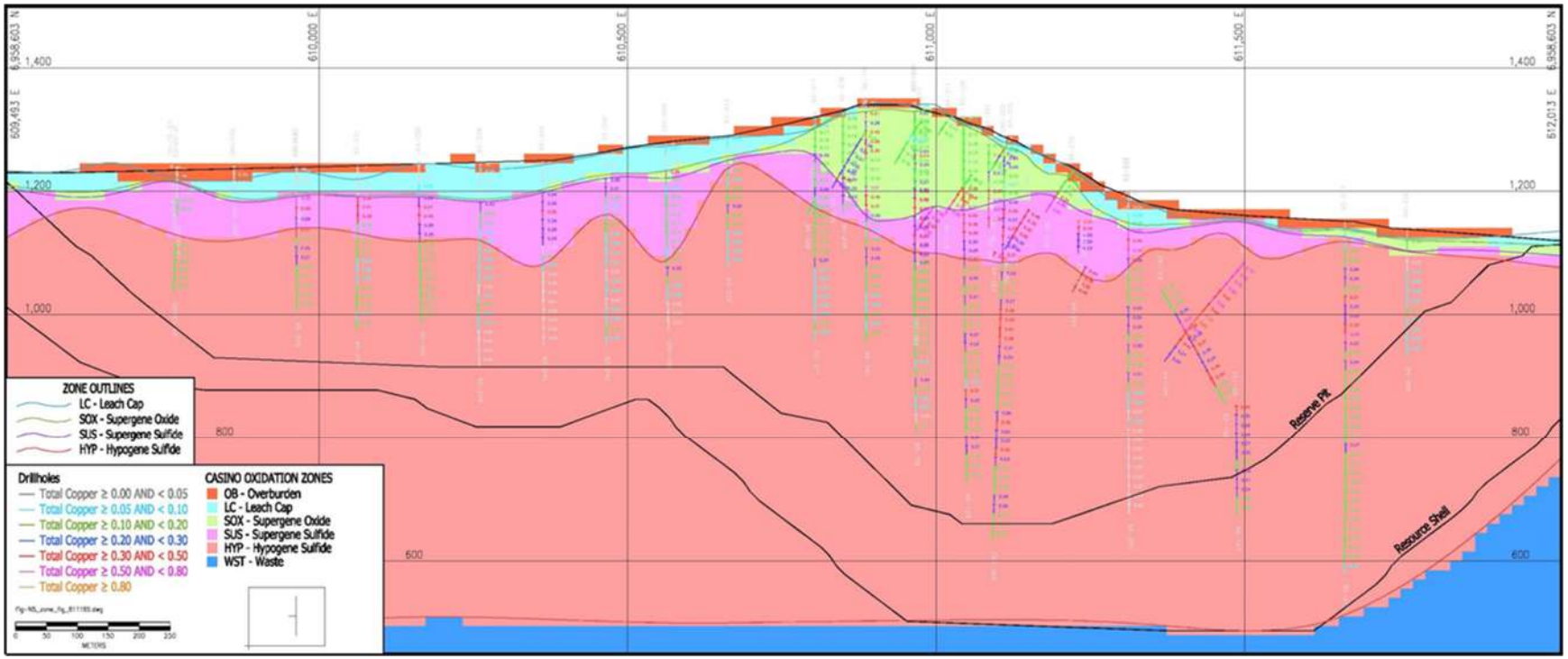
Western personnel provided IMC with solids to represent the LC, SOX, SUS, and HYP domains. IMC used these solids to assign oxidation zone types to model blocks. Code 6, waste, was used to denote blocks outside the provided solids. A surface was provided to denote the bottom of overburden. IMC assigned blocks above the leach cap as overburden.

Table 14-8: Oxidation Zone Types

Zone	Code	Description
OVB	1	Overburden
LC	2	Leach Cap
SOX	3	Supergene Oxide
SUS	4	Supergene Sulphide
HYP	5	Hypogene Sulphide
WST	6	Waste – Peripheral to Above Solids

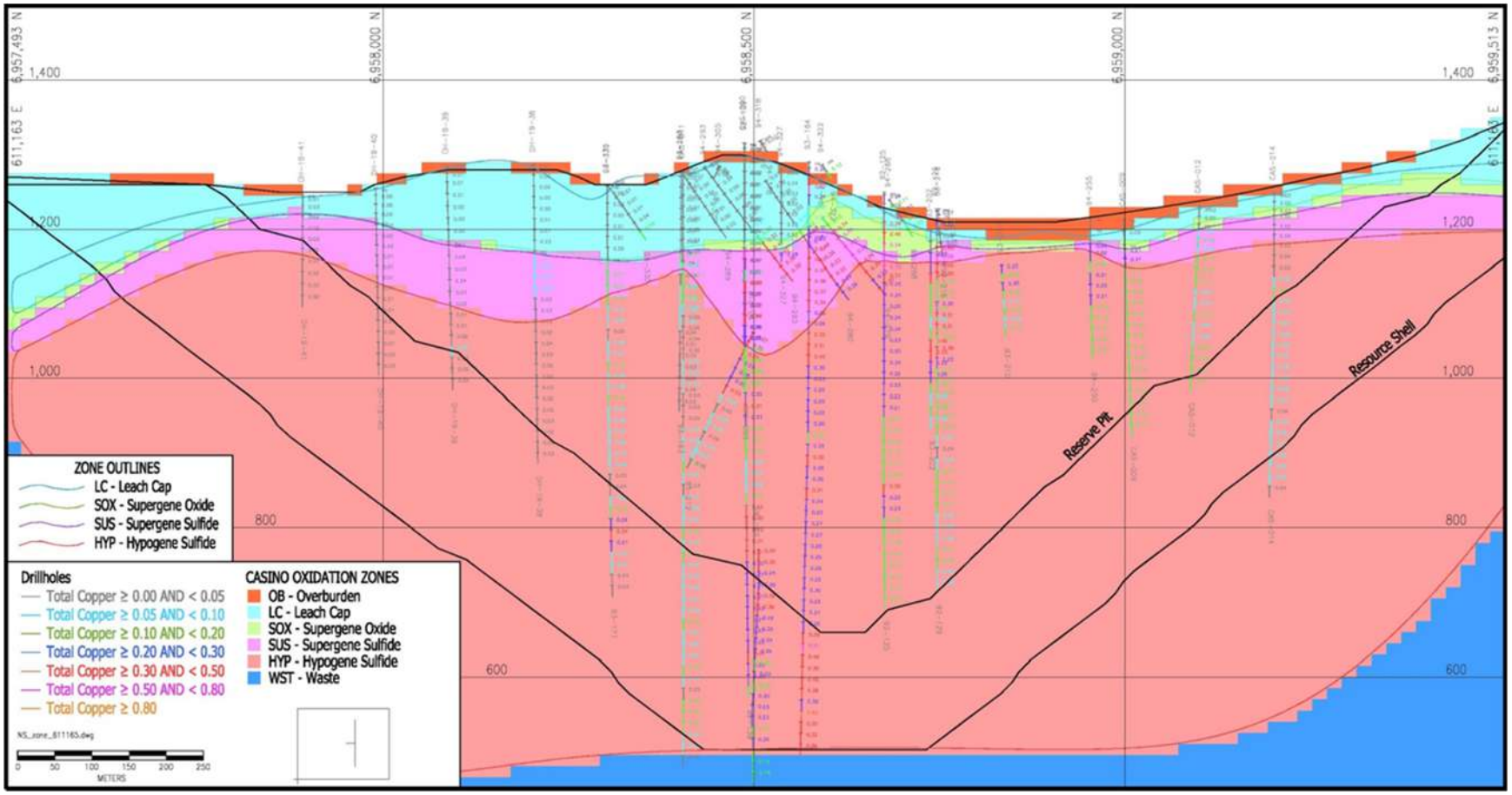
IMC also used the solids to back-assign the oxide domain codes to the assay database. It is noted that the assay database did include an oxide domain assignment from logging, but IMC used the back-assigned values for modeling so assay intervals would be consistent with the domains they are located.

Figure 14-3 and Figure 14-4 show the oxide zones on east-west and north-south cross sections, respectively. It can be seen that most of the Mineral Resource is in hypogene sulphide material.



(Source: IMC, 2020)

Figure 14-3: Oxidation Domains on East-West Section 6,958,600N



(Source: IMC, 2020)

Figure 14-4: Oxidation Domains on North-South Section 611,165E

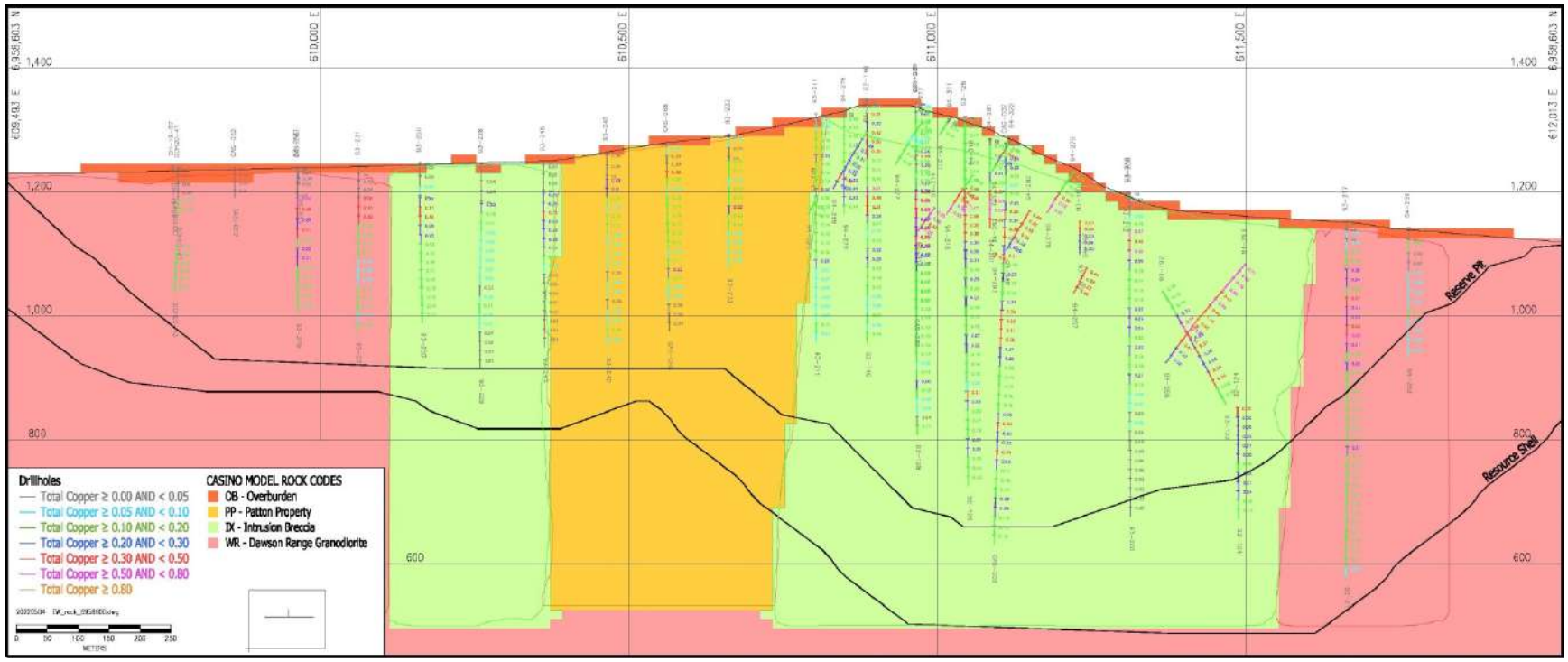
Rock Types

Rock type interpretations for four major rock types plus the overburden have been developed as 3D solids or a surface for the overburden. Table 14-9 shows the rock types. Figure 14-5 shows the rock types on east-west cross section 6,958,600N. It can be seen that the main host rock is the Dawson Range Granodiorite which has been intruded by the Intrusion Breccia and the Patton Porphyry. The third intrusion, the Post Mineral Explosive Breccia (MX) to the southwest of the pit, is post mineral in character.

IMC used the solids to assign rock codes to the model blocks. Rock codes were also assigned to the assay database by back-assignment from the solids. Note that there were rock type designations in the assay database, but the back-assigned values were used for the resource model so the assay assignments would be consistent with the block they were located.

Table 14-9: Model Rock Types

Rock	Code	Description
OVB	1	Overburden
PP	2	Patton Porphyry
IX	3	Intrusion Breccia
WR	4	Dawson Range Granodiorite
MX	5	Post Mineral Explosive Breccia



(Source: IMC, 2020)

Figure 14-5: Rock Types on East-West Section 6,958,600N

14.5.4 Cap Grades and Compositing

IMC reviewed the database to determine cap grades for the various minerals. The distribution of the length of sample intervals, when copper is assayed, is approximately as follows:

- About 24% are less than 3 m in length,
- About 69% are 3 m or 3.05 m (10 US ft), and
- About 7% are longer than 3.05 m.

IMC considers that a relatively consistent 3 m sample interval was used for the drilling and that cap grades may reasonably be applied to the assays.

IMC examined probability plots and sorted lists of the higher-grade assay intervals for copper, gold, moly, and silver by oxidation zones to determine cap grades. Table 14-10 shows the cap grades in the upper portion of the table and number of assays capped in the lower portion of the table. It can be seen that relatively small numbers of assays were capped for each metal in each population. The cap grades generally correspond to the upper 99.8 to 99.9 percentile of the populations.

The assay database was composited to nominal 7.5 m downhole composites, respecting the oxidation zones. It is noted this is one-half of the 15 m bench height used for the model. The smaller composite length allows capturing some of the narrowing zones and also tends to result in less grade smoothing during block grade estimation. Composited values included the total copper, weak acid soluble copper, gold, moly, and silver assays, the soluble copper to total copper ratio, and the rock type and oxidation zone codes.

The interpretation of nominal 7.5 m composites is described next. As noted, the composites do not cross oxidation zone boundaries. Composites within a zone are divided into equal length composites as close as possible to the target length. For example, a 28 m zone of supergene sulphide is composited into four 7 m composites. With this algorithm 93% of the composites are between 7 m and 8 m in length and 97.3% of the composites are between 6.5 m and 8.5 m in length; IMC does not consider the slight difference in the lengths of the composite's material for grade estimation purposes.

Table 14-10: Cap Grades and Number of Assays Capped

Metal	Units	OB	LC	SOX	SUS	HYP
Copper	(%)	none	0.70	1.60	2.00	1.70
Gold	(g/t)	none	2.00	2.10	3.20	3.75
Moly	(%)	none	0.20	0.17	0.70	0.26
Silver	(g/t)	none	35.0	25.0	25.0	95.0
Number of Assays Capped						
Metal	**Units**	**OB**	**LC**	**SOX**	**SUS**	**HYP**
Copper	(none)	0	6	5	5	10
Gold	(none)	0	13	8	9	12
Moly	(none)	0	9	8	5	10
Silver	(none)	2	15	9	7	13

14.5.5 Descriptive Statistics

Table 14-11 shows descriptive statistics for total copper, gold, moly, and silver for the assay intervals. The table shows values by the oxidation zones. The left side of the table shows uncapped values and the right side shows capped values. For copper it can be seen that values in the overburden and leach cap are very low, values in the SOX and SUS are somewhat elevated, and values in the hypogene tend to be lower than the supergene. Gold, moly, and silver do not have the corresponding depletion of values in the leach cap. Mean gold and moly grades are slightly elevated in the SOX compared to SUS. The table includes only non-zero values for each population, though many have placeholders for below detection limit values. The descriptive statistics calculations were limited to the data within the coordinate limits 6,957,500N to 6,960,000N and 609,000E to 612,000E, i.e., in the main deposit area. This was to exclude outlying exploration and geotechnical drilling.

Table 14-12 shows descriptive statistics for the 7.5 m composites. The table includes only non-zero values, but zero value assays are incorporated into the composites.

Figure 14-6 shows a probability plot of total copper grades for the composites for the various oxidation type domains. Figure 14-7 shows the probably plot for gold.

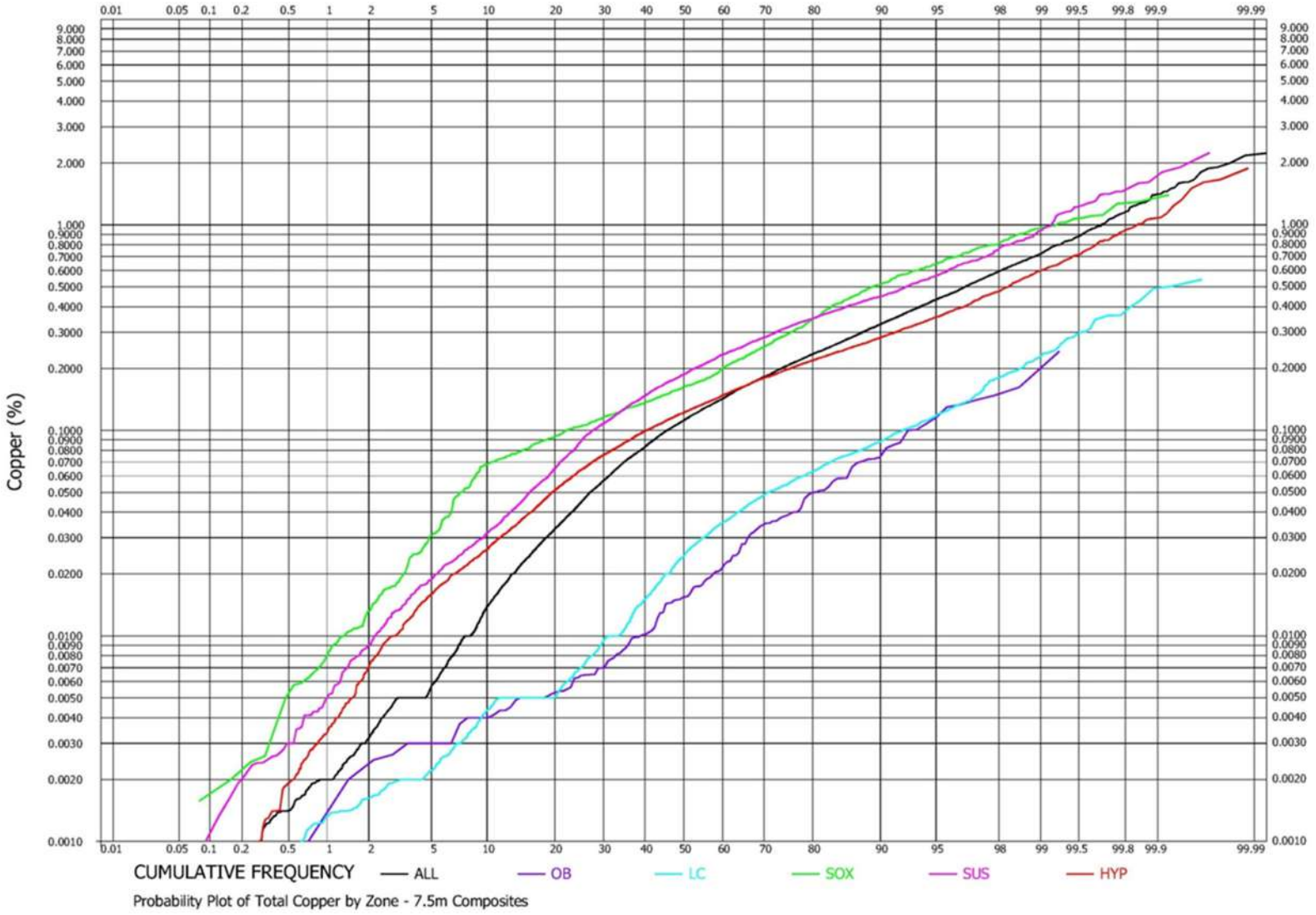
Table 14-11: Summary Statistics of Assays

Metal/Zone	Not Capped					Capped				
	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)	No. of Sample	Mean (%)	Std Dev (%)	Max (%)	Min (%)
Copper:										
All Samples	41,735	0.151	0.173	5.63	0.001	41,735	0.150	0.168	2.00	0.001
Overburden	420	0.039	0.061	0.50	0.001	420	0.039	0.061	0.50	0.001
Leach Cap	7,050	0.040	0.058	1.36	0.001	7,050	0.040	0.055	0.70	0.001
Supergene Oxide	3,368	0.230	0.231	2.90	0.001	3,368	0.229	0.227	1.60	0.001
Supergene Sulphide	8,820	0.226	0.227	5.63	0.001	8,820	0.226	0.218	2.00	0.001
Hypogene	22,077	0.146	0.138	2.99	0.001	22,077	0.146	0.134	1.70	0.001
Metal/Zone	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Sample**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Gold:										
All Samples	41,709	0.231	0.674	99.96	0.003	41,709	0.225	0.244	3.75	0.003
Overburden	421	0.153	0.189	1.68	0.003	421	0.153	0.189	1.68	0.003
Leach Cap	7,046	0.293	1.220	99.96	0.003	7,046	0.277	0.265	2.00	0.003
Supergene Oxide	3.366	0.366	0.323	2.64	0.003	3.366	0.365	0.320	2.10	0.003
Supergene Sulphide	8,821	0.241	0.336	18.79	0.003	8,821	0.238	0.256	3.20	0.003
Hypogene	22,055	0.188	0.561	55.10	0.003	22,055	0.183	0.204	3.75	0.003
Metal/Zone	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Sample**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Moly:										
All Samples	39,561	0.0177	0.0287	1.240	0.001	39,561	0.0176	0.0273	0.700	0.001
Overburden	376	0.0085	0.0127	0.098	0.001	376	0.0085	0.0127	0.098	0.001
Leach Cap	6,750	0.0169	0.0232	0.363	0.001	6,750	0.0167	0.0218	0.200	0.001
Supergene Oxide	3,290	0.0212	0.0240	0.398	0.001	3,290	0.0211	0.0227	0.170	0.001
Supergene Sulphide	8,239	0.0194	0.0437	1.240	0.001	8,239	0.0192	0.0413	0.700	0.001
Hypogene	20,906	0.0168	0.0232	0.681	0.001	20,906	0.0168	0.0223	0.260	0.001

Metal/Zone	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)	No. of Sample	Mean (%)	Std Dev (%)	Max (%)	Min (%)
Silver:										
All Samples	40,363	1.94	49.91	9999.9	0.10	40,363	1.66	3.01	95.0	0.10
Overburden	421	1.70	6.65	131.0	0.10	421	1.42	2.01	20.0	0.10
Leach Cap	6,890	2.11	4.97	200.0	0.10	6,890	2.01	2.49	35.0	0.10
Supergene Oxide	3,257	1.96	2.96	70.2	0.10	3,257	1.91	2.25	25.0	0.10
Supergene Sulphide	8,671	1.68	2.23	76.0	0.10	8,671	1.66	1.86	25.0	0.10
Hypogene	21,124	1.99	68.90	9999.9	0.10	21,124	1.51	3.60	95.0	0.10

Table 14-12:Summary Statistics of 7.5 m Composites

Metal/Zone	Not Capped					Capped				
	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)	No. of Sample	Mean (%)	Std Dev (%)	Max (%)	Min (%)
Copper:										
All Samples	15,943	0.152	0.160	4.12	0.000	15,943	0.151	0.155	2.00	0.000
Overburden	139	0.032	0.044	0.30	0.001	139	0.032	0.044	0.30	0.001
Leach Cap	2,723	0.039	0.049	0.73	0.001	2,723	0.039	0.048	0.49	0.001
Supergene Oxide	1,268	0.234	0.205	1.99	0.002	1,268	0.233	0.202	1.48	0.002
Supergene Sulphide	3,290	0.229	0.210	4.12	0.001	3,290	0.228	0.200	2.00	0.001
Hypogene	8,523	0.147	0.126	2.17	0.000	8,523	0.147	0.123	1.54	0.000
Metal/Zone	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Sample**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Gold:										
All Samples	15,943	0.231	0.372	24.93	0.000	15,943	0.226	0.210	3.13	0.000
Overburden	139	0.132	0.164	1.17	0.015	139	0.132	0.164	1.17	0.015
Leach Cap	2,723	0.287	0.529	24.93	0.003	2,723	0.277	0.235	1.80	0.003
Supergene Oxide	1,268	0.370	0.291	1.97	0.004	1,268	0.370	0.289	1.97	0.004
Supergene Sulphide	3,290	0.245	0.275	9.21	0.003	3,290	0.242	0.224	2.90	0.003
Hypogene	8,523	0.188	0.348	21.94	0.000	8,523	0.183	0.165	3.13	0.000
Metal/Zone	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Sample**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Moly:										
All Samples	15,454	0.0175	0.0254	0.679	0.0000	15,454	0.0174	0.0245	0.619	0.0000
Overburden	144	0.0061	0.0092	0.063	0.0000	144	0.0061	0.0092	0.063	0.0000
Leach Cap	2,643	0.0166	0.0210	0.320	0.0000	2,643	0.0165	0.0200	0.200	0.0000
Supergene Oxide	1,252	0.0210	0.0203	0.182	0.0000	1,252	0.0209	0.0198	0.134	0.0000
Supergene Sulphide	3,169	0.0193	0.0396	0.679	0.0000	3,169	0.0192	0.0381	0.619	0.0000
Hypogene	8,246	0.0168	0.0198	0.314	0.0000	8,246	0.0167	0.0193	0.220	0.0000
Metal/Zone	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Sample**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Silver:										
All Samples	15,690	1.79	14.52	1759.9	0.00	15,690	1.63	2.32	93.2	0.00
Overburden	139	1.75	3.77	38.1	0.10	139	1.40	1.45	9.7	0.10
Leach Cap	2,693	2.10	4.14	173.9	0.03	2,693	2.00	2.15	35.0	0.03
Supergene Oxide	1,243	1.96	2.41	41.0	0.00	1,243	1.90	1.89	20.0	0.00
Supergene Sulphide	3,269	1.66	1.53	20.0	0.08	3,269	1.65	1.43	17.8	0.08
Hypogene	8,346	1.72	19.72	1759.9	0.01	8,346	1.47	2.67	93.2	0.01



(Source: IMC, 2022)

Figure 14-6: Probability Plot of Total Copper Composites by Oxidation Type

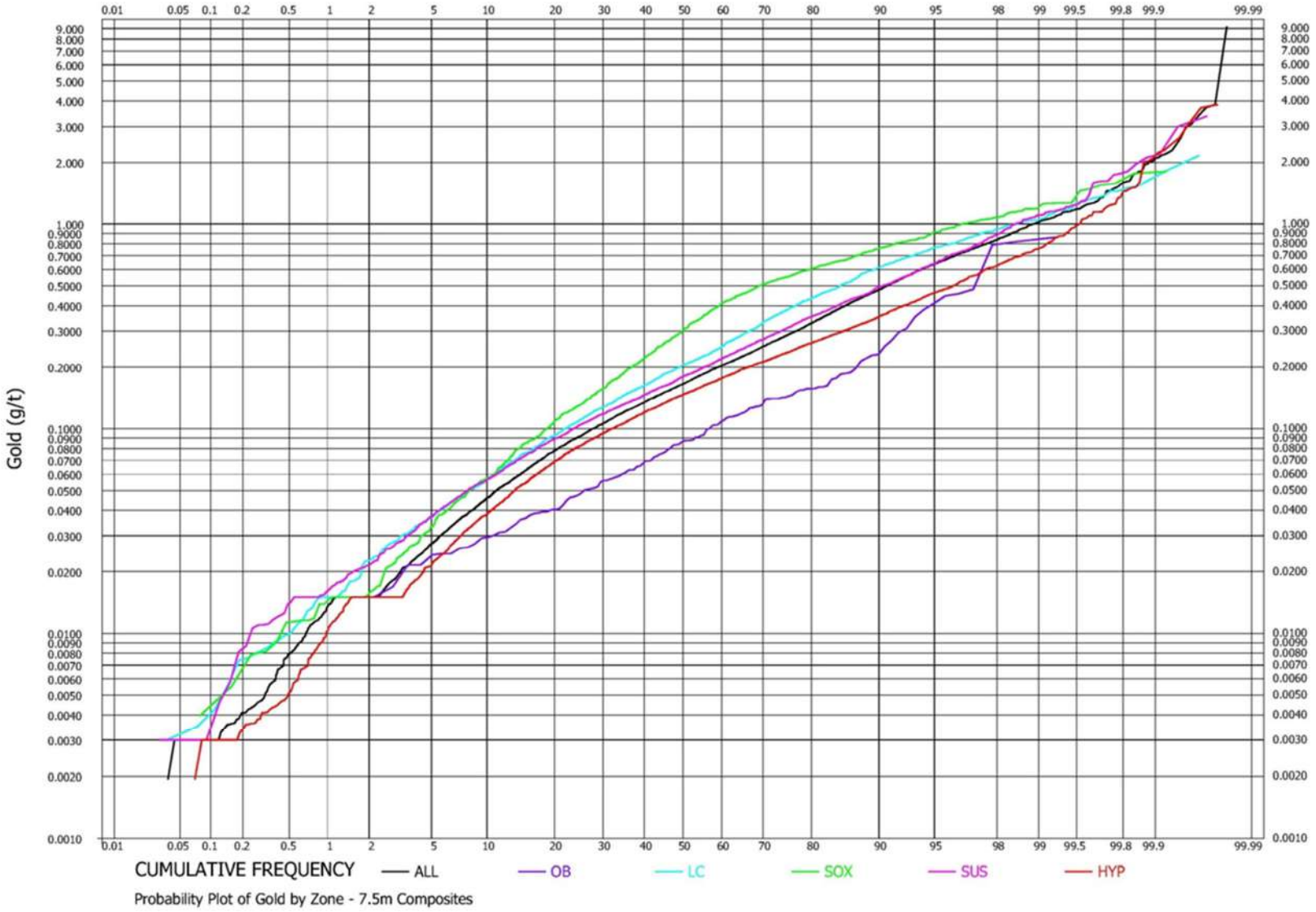


Figure 14-7: Probability Plot of Gold Composites by Oxidation Type (IMC, 2022)

14.5.6 Variogram Analysis

Copper

IMC conducted variogram analyses of total copper by oxidation type domains. The analysis was based on the 7.5 m composites. The leach cap, supergene oxide, and supergene sulphide domains are relatively flat lying, and the distribution of copper mineralization appears to not vary much by direction. Figure 14-8 and Figure 14-9 show variograms for supergene oxide and supergene sulphide, respectively. These variograms are calculated as the average of all horizontal directions which is consistent with the relatively flat lying mineralization in these domains. The ranges of the first variogram structures are 172 m for supergene oxide and 263 m for supergene sulphide. The variograms are of good clarity.

For the hypogene sulphide, IMC ran variograms in many directions. The various directional variograms tended to be similar, indicating a somewhat isotropic distribution of copper mineralization. Figure 14-10 shows the variogram for hypogene sulphide copper calculated as the average in all directions. The variogram has good clarity and the range of the first structure is 293 m. Geologic inference suggests that the range of influence should be slightly longer in the east-west direction than the north-south direction. This is indicated in the variograms. Figure 14-11 and Figure 14-12 show variograms for hypogene sulphide in the east-west and north-south directions, respectively.

The variograms were calculated with the pairwise relative variogram method. The variogram values shown on the graphs would be multiplied by the mean squared to convert them to % total copper units.

Gold, Moly, and Silver

Variograms were also calculated for gold, moly, and silver. For these metals there no evidence of significant grade changes across oxidation domain boundaries, so the calculations combined the data for all zones. As with copper, mineralization tends to be somewhat isotropic. Figure 14-13 shows the variogram for gold. The variogram has good clarity and the range of influence of the first structure is over 200 m. Though not shown, the molybdenum and silver variograms are also of good clarity and reasonably long ranges.

PAIRWISE RELATIVE VARIOGRAM OF: cap_cu
GLOBAL VARIOGRAM (AVG. OF ALL DIRECTIONS)
Azimuth: 0.0 Dip: 0.0 MARCH 9, 2020

Pairwise Relative Variogram
* variogram analysis of : cap_cu
data transformation : none
lag option : 1 class size 50.
file/variogram number : gamm-pairSOX-50-cap_ 3

azimuth	0.0	direction	North
dip angle	0.0	mean	0.2460
horizontal window	90.0	variance	0.0418
vertical window	90.0	no. of samples	995

spherical: c 0.1730E+00 range0.1725E+03
spherical: c 0.5920E-01 range0.7275E+03
nugget 0.1791E+00 sill 0.4113E+00

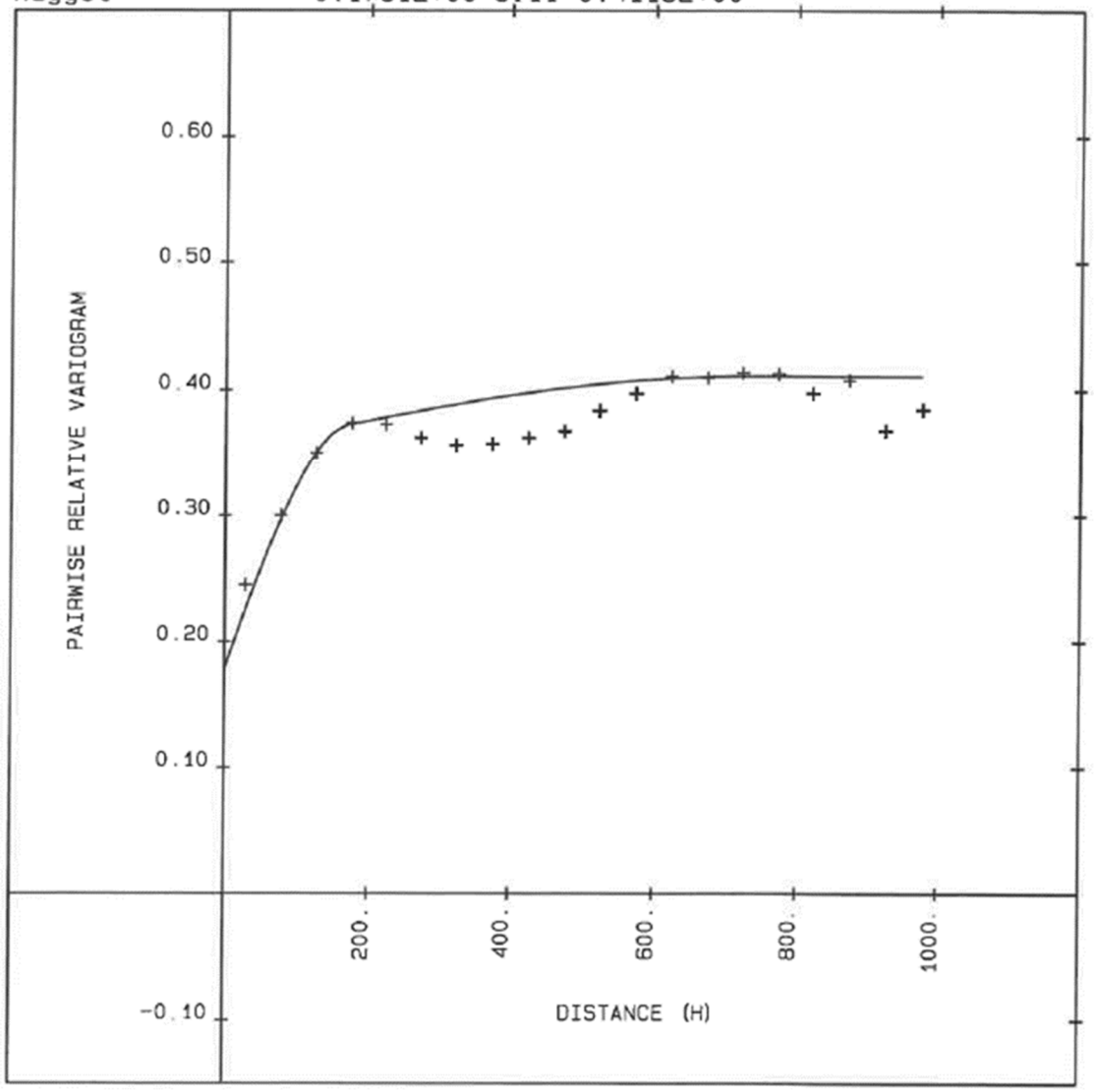


Figure 14-8: Total Copper Variogram – Supergene Oxide (IMC, 2022)

PAIRWISE RELATIVE VARIOGRAM OF: cap_cu
GLOBAL VARIOGRAM (AVG. OF ALL DIRECTIONS)
Azimuth: 0.0 Dip: 0.0 MARCH 9, 2020

Pairwise Relative Variogram
* variogram analysis of : cap_cu
data transformation : none
lag option : 1 class size 50.
file/variogram number : gamm-pair-SUS-50-cap 3

azimuth	0.0	direction	North
dip angle	0.0	mean	0.2520
horizontal window	90.0	variance	0.0411
vertical window	90.0	no. of samples	2885

spherical: c 0.1463E+00 range0.2631E+03
spherical: c 0.2252E+00 range0.8831E+03
nugget 0.1106E+00 sill 0.4821E+00

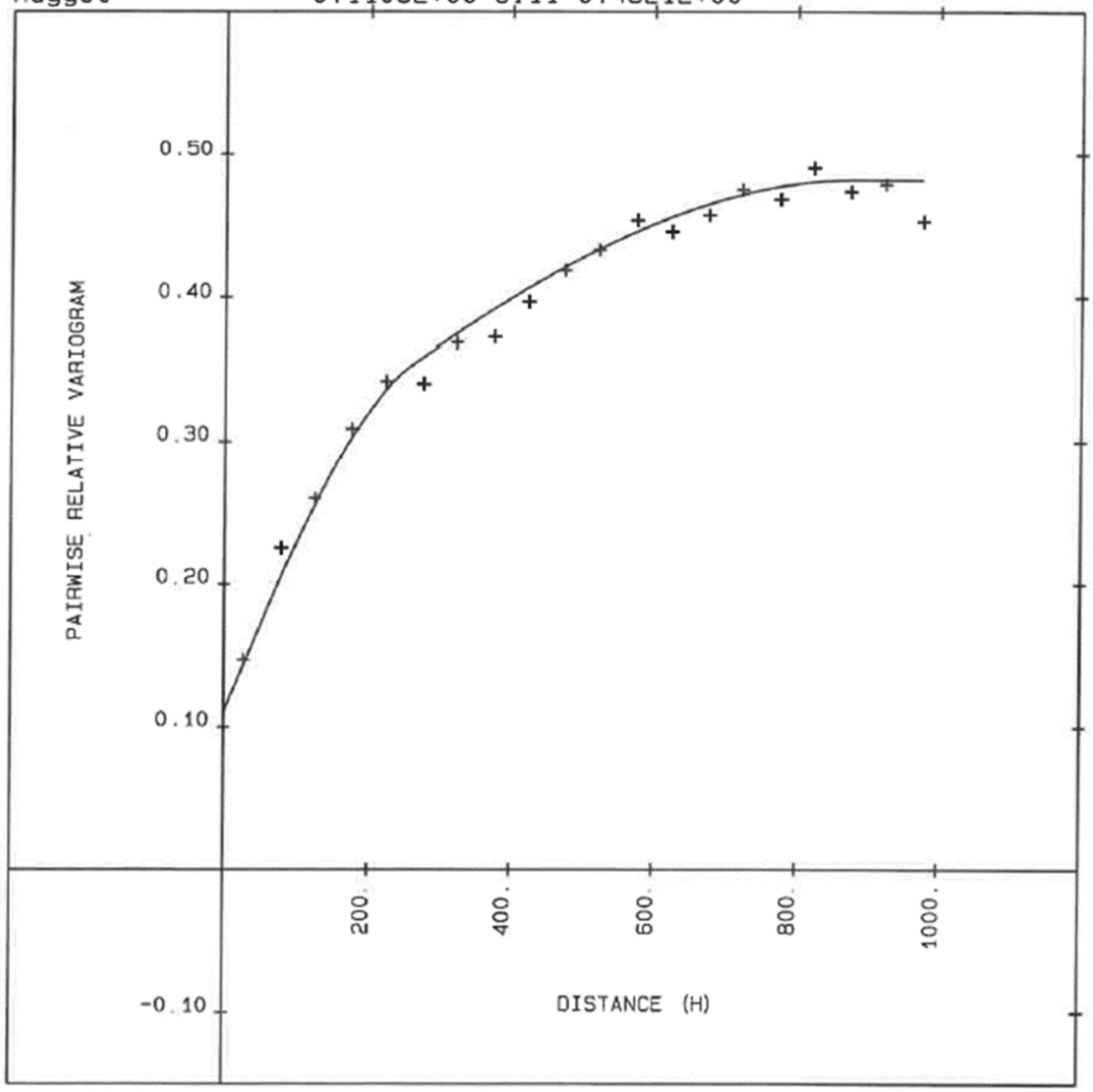


Figure 14-9: Total Copper Variogram – Supergene Sulphide (IMC, 2022)

Total Copper in Hypogene
GLOBAL VARIOGRAM (AVG. OF ALL DIRECTIONS)
Azimuth: 0.0 Dip: 90.0 APRIL 3, 2020

Pairwise Relative Variogram
* variogram analysis of : cap_cu
data transformation : none
lag option : 1 class size 50.
file/variogram number : gamm-pairHYP-50.avg 3

azimuth	0.0	direction	North
dip angle	90.0	mean	0.1580
horizontal window	90.0	variance	0.0155
vertical window	90.0	no. of samples	7560

spherical: c 0.2276E+00 range0.2933E+03
spherical: c 0.1335E+00 range0.6416E+03
nugget 0.8509E-01 sill 0.4462E+00

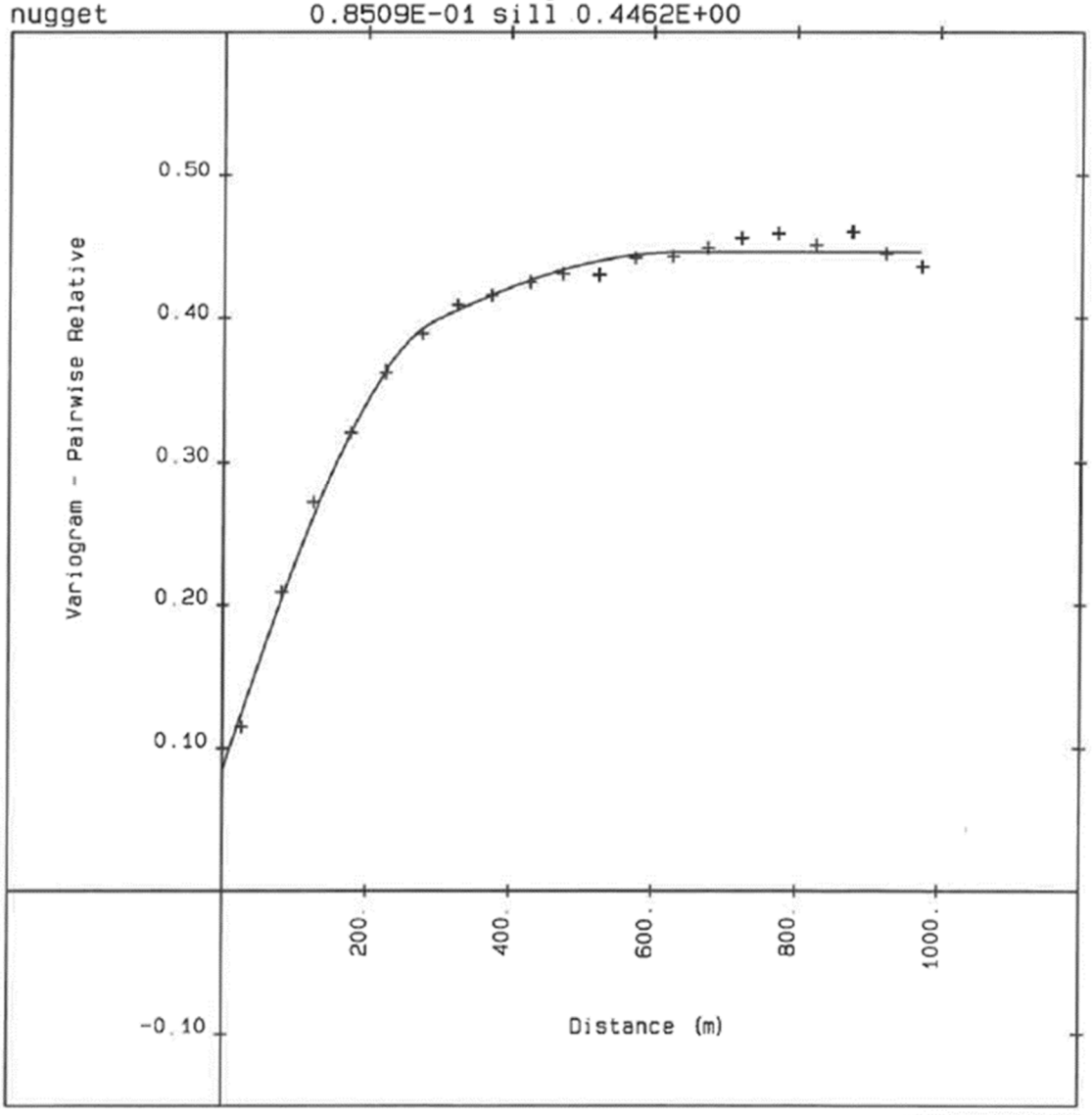


Figure 14-10: Total Copper Variogram – Hypogene Sulphide – Global (IMC, 2022)

Total Copper in Hypogene
East - NO DIP
Azimuth: 90.0 Dip: 0.0 APRIL 3, 2020

Pairwise Relative Variogram
* variogram analysis of : cap_cu
data transformation : none
lag option : 1 class size 50.
file/variogram number : gamm-pairHYP-50.avg 8

azimuth	90.0	direction	East
dip angle	0.0	mean	0.1580
horizontal window	15.0	variance	0.0155
vertical window	15.0	no. of samples	7560

spherical: c 0.2508E+00 range0.3188E+03
spherical: c 0.1509E+00 range0.7716E+03
nugget 0.7464E-01 sill 0.4764E+00

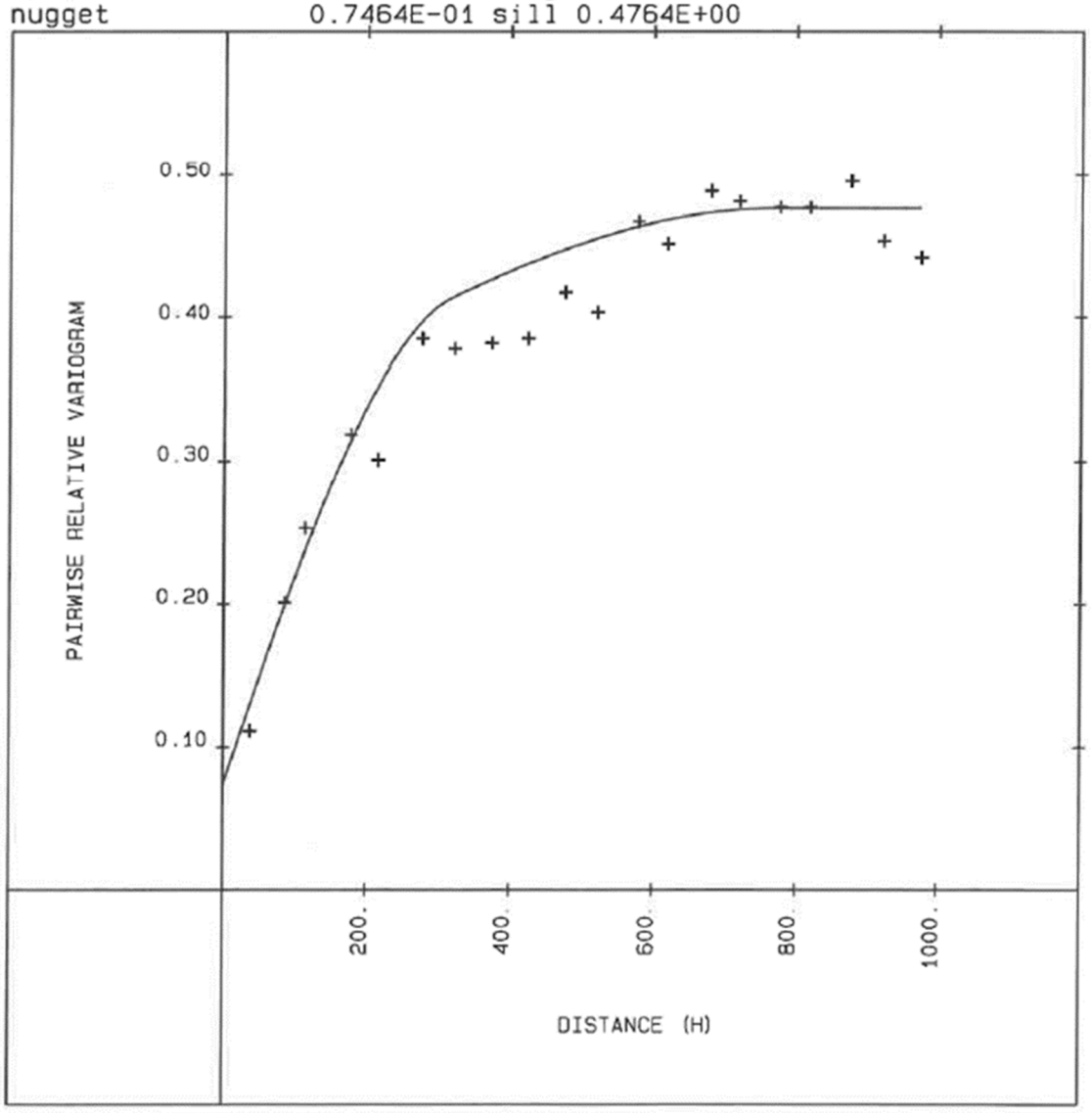


Figure 14-11: Total Copper Variogram – Hypogene Sulphide – East-West (IMC, 2022)

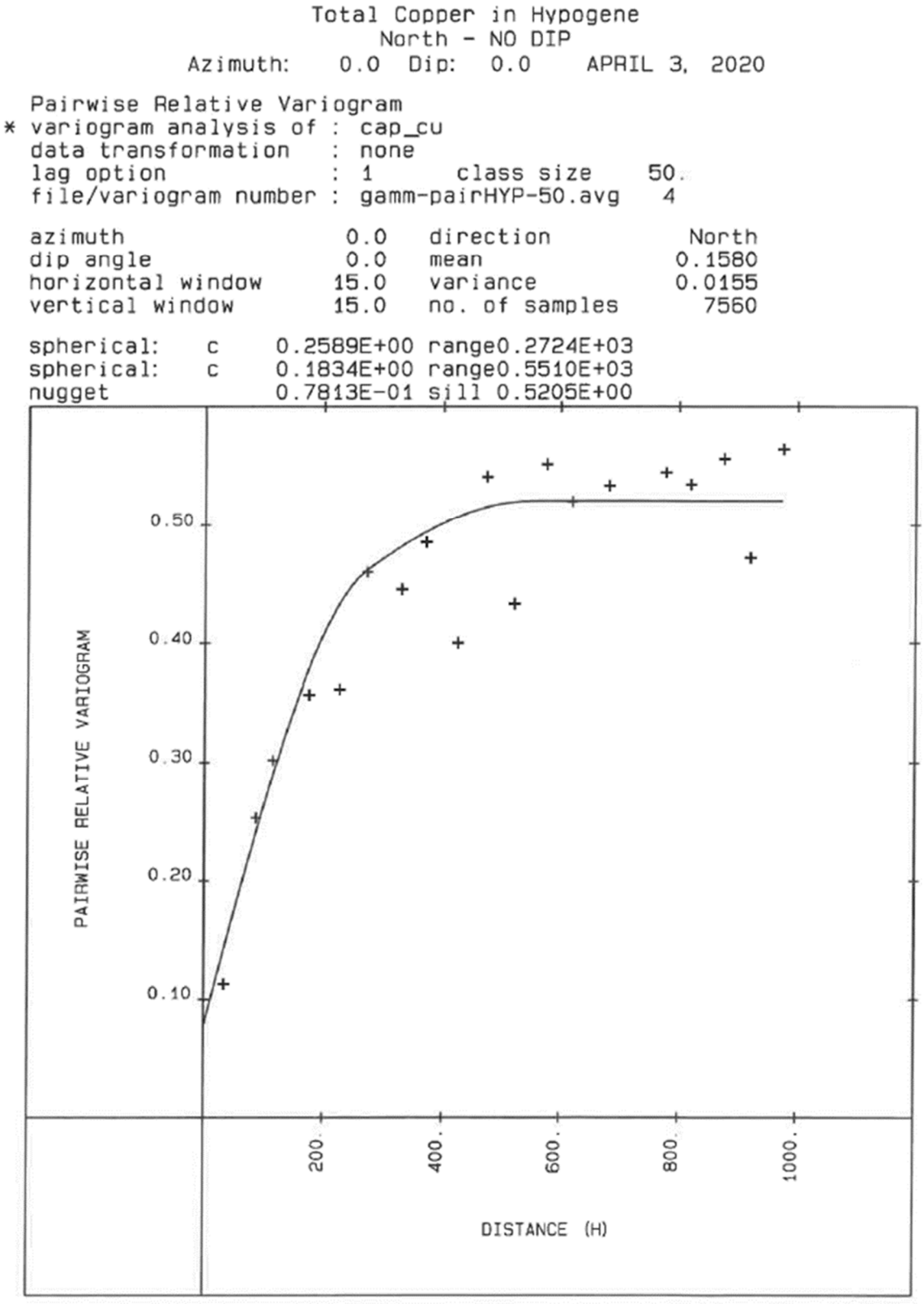


Figure 14-12: Total Copper Variogram – Hypogene Sulphide – North-South (IMC, 2022)

PAIRWISE RELATIVE VARIOGRAM OF: cap_au
GLOBAL VARIOGRAM (AVG. OF ALL DIRECTIONS)
Azimuth: 0.0 Dip: 0.0 MARCH 10, 2020

Pairwise Relative Variogram
* variogram analysis of : cap_au
data transformation : none
lag option : 1 class size 50.
file/variogram number : gamm-pairALL-cap_au. 3

azimuth	0.0	direction	North
dip angle	0.0	mean	0.2370
horizontal window	90.0	variance	0.0456
vertical window	90.0	no. of samples	13908

spherical: c 0.7369E-01 range0.2073E+03
spherical: c 0.2987E+00 range0.5348E+03
nugget 0.1217E+00 sill 0.4941E+00

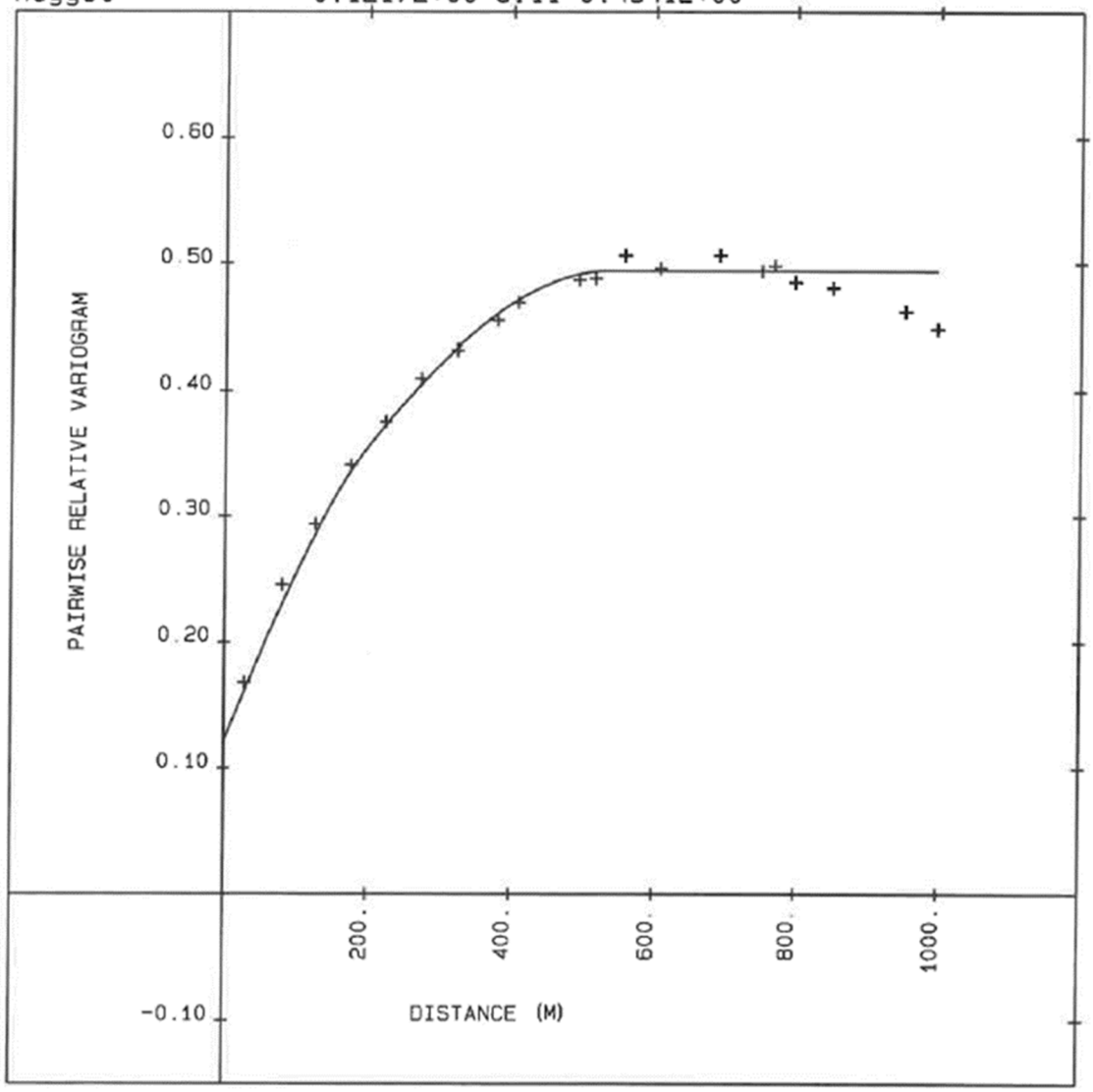


Figure 14-13: Gold Variogram (IMC, 2022)

14.5.7 Block Grade Estimation

Block grades for total copper, weak acid soluble copper, gold, moly, and silver were estimated with inverse distance with a power weight of 2 (ID2). The ID2 method was chosen because it generally results in less grade smoothing (smearing) than ordinary kriging (OK). For the 2020 model, estimates were also done by OK, inverse distance with a power weight of 3 (ID3), and nearest neighbour (NN) for comparison purposes. The ID2 method was chosen for the base case model. This comparison was not repeated for the current model.

Total and Soluble Copper

The leach cap, supergene oxide, supergene sulphide, and hypogene sulphide oxidation type boundaries were all considered hard boundaries for the estimation of total copper and weak acid soluble copper. The waste domain was not estimated; it is well outside the drilling data. Grades were not estimated for overburden blocks. The rock types (Patton Porphyry, Intrusion Breccia, Dawson Range Granodiorite, and Post Mineral Explosive Breccia) were also all considered as separate populations with hard boundaries. This is a change from the 2010 and 2020 models where the Post Mineral Explosive Breccia was treated as a separate population and the other rock types lumped together.

For leach cap, supergene oxide, and supergene sulphide the search radii for the estimations were 200 m (circular) in the horizontal direction and 30 m in the vertical direction. These search radii are well within the variogram ranges and are adequate to fill in the block grades. A maximum of 15 composites, a minimum of one composite, and a maximum of three composites per hole were used.

For hypogene sulphide the search radii were 220 m in the east-west direction, 180 m in the north-south direction, and 100 m in the vertical direction. A maximum of 24 composites, a minimum of two composites, and a maximum of six composites per hole were used.

Figure 14-14 and Figure 14-15 show copper grades on an east-west and north-south cross section, respectively.

Soluble copper block grade estimates were also conducted for the leach cap, supergene oxide, and supergene sulphide domains. Soluble copper estimates were not done for hypogene sulphide. Soluble copper assays were generally not done for hypogene material. A check was made for soluble copper assays that exceeded the total copper assay. There were only 4 intervals where this occurred, so it was not considered material. After estimation there were 784 blocks with the soluble copper estimate higher than total copper; these were capped at the total copper grade.

Gold, Molybdenum, and Silver

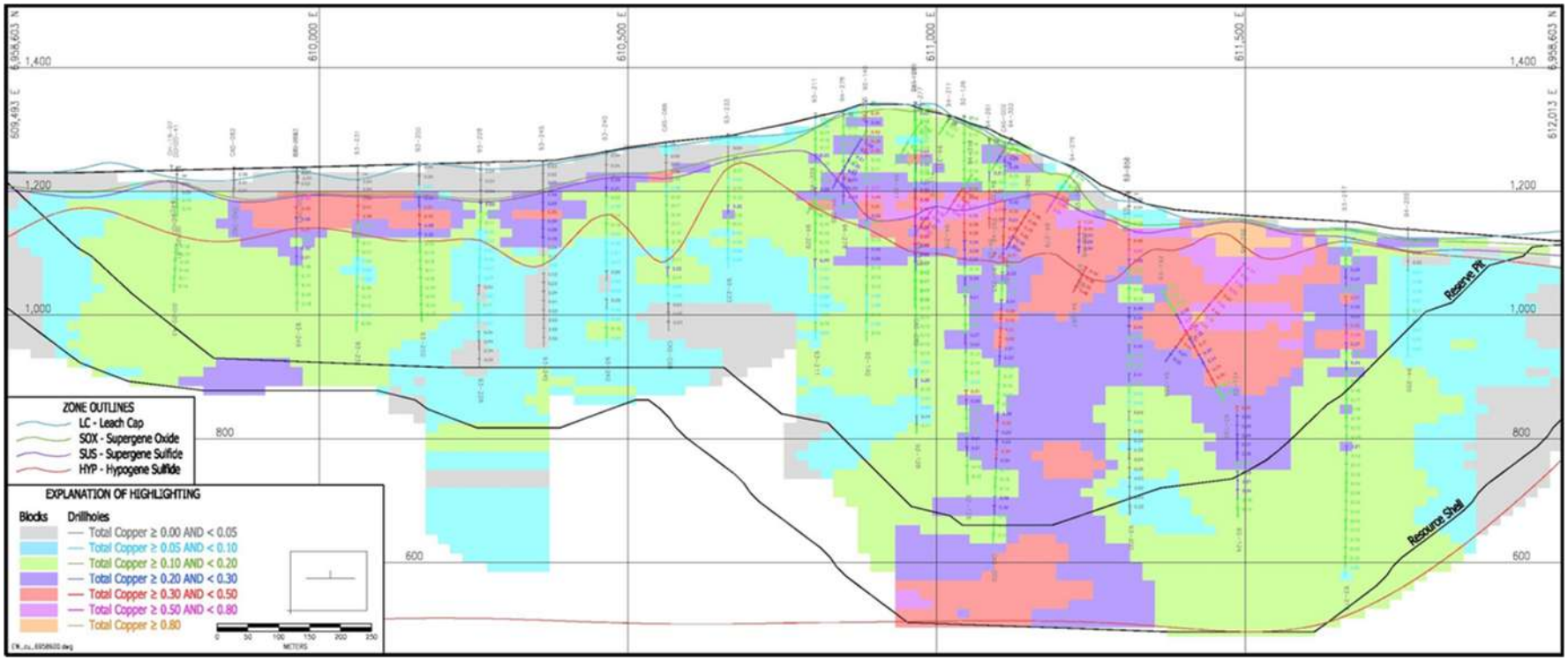
For gold, molybdenum, and silver the oxidation type boundaries were all considered hard boundaries for estimation. This is a change from the 2020 and 2010 models where they were not considered hard boundaries. Generally, remobilization of these metals between the various zones is minimal. All of the rock type boundaries were also considered as hard boundaries for the model update. This is also a change from the 2020 and 2010 models where only the Post Mineral Explosive Breccia was considered a separate domain from the other rock types.

Search radii parameters were the same as for copper. For leach cap, supergene oxide, and supergene sulphide the search radii were 200 m (circular) in the horizontal direction and 30 m in the vertical direction. A maximum of 15 composites, a minimum of one composite, and a maximum of three composites per hole were used. For hypogene sulphide the search radii were 220 m in the east-west direction, 180 m in the north-south direction, and 100 m in the vertical direction and maximum of 24 composites, a minimum of two composite, and a maximum of six composites per hole were used.

Figure 14-16 and Figure 14-17 show gold grades on an east-west and north-south section, respectively.

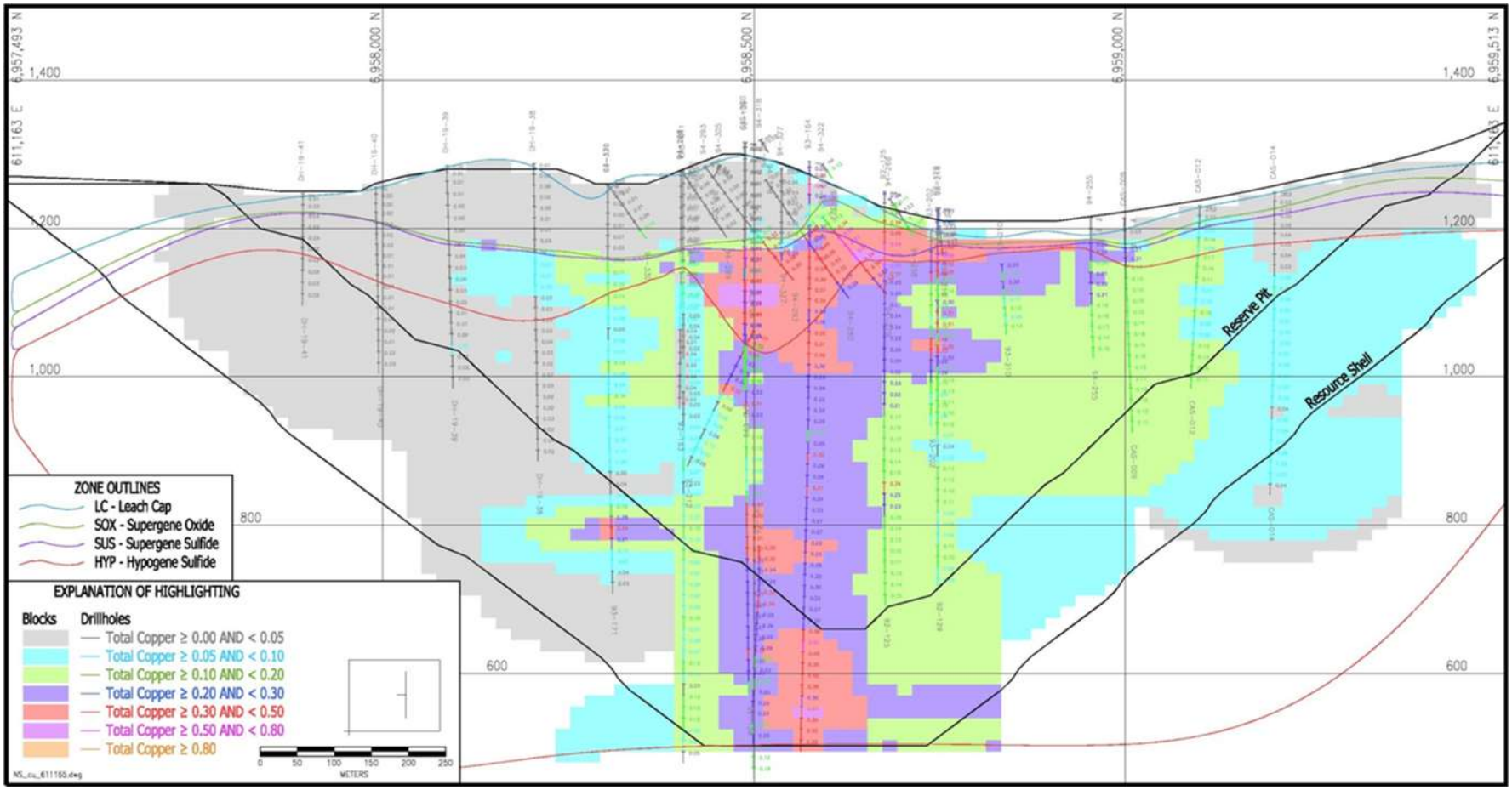
Arsenic, Antimony, and Bismuth

Block grade estimates were also conducted for arsenic, antimony, and bismuth. As with the other metals, all the oxide zones and rock type boundaries were considered hard boundaries. The methodology and search parameters were the same as for the other metals. Minor capping of the assays was conducted. Arsenic was capped at 3,000 ppm, antimony at 1,000 ppm, and bismuth at 400 ppm.



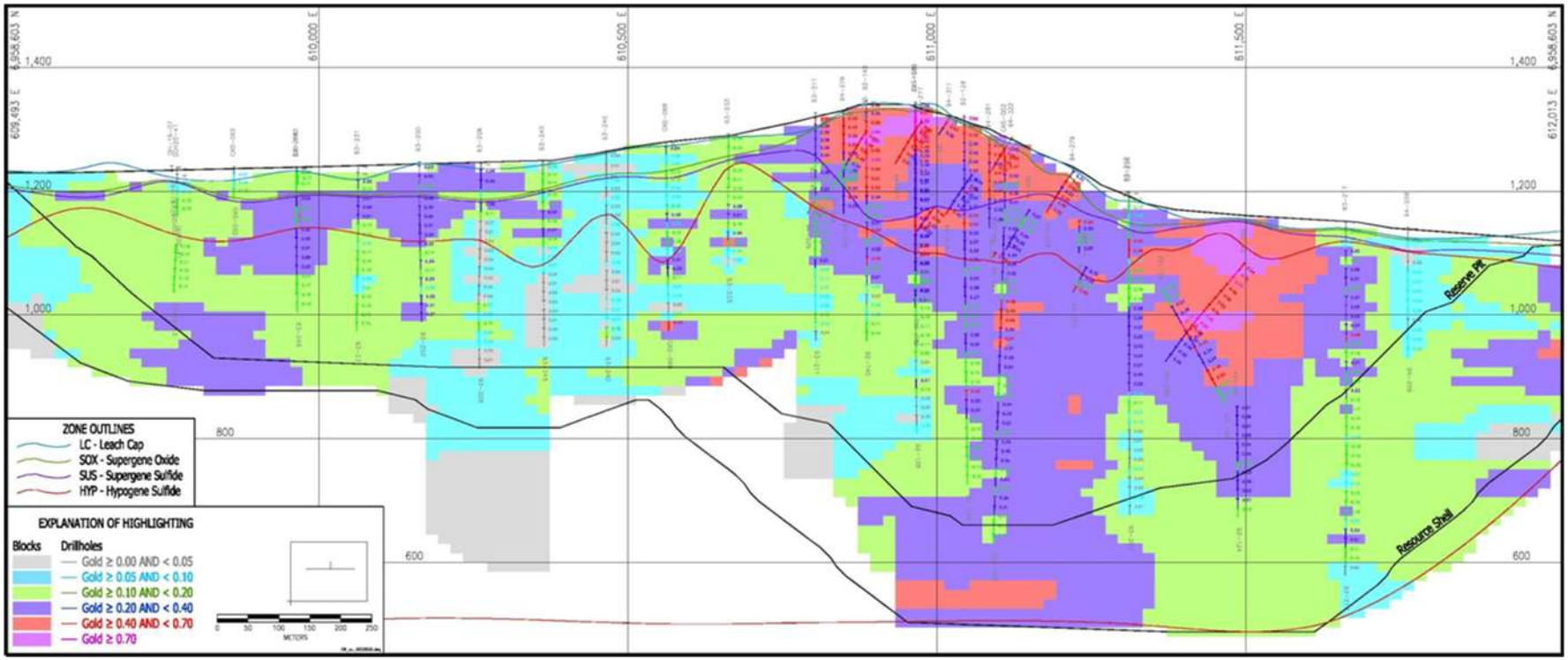
(Source: IMC, 2022)

Figure 14-14: Total Copper Grades on East-West Cross Section 6,958,600N



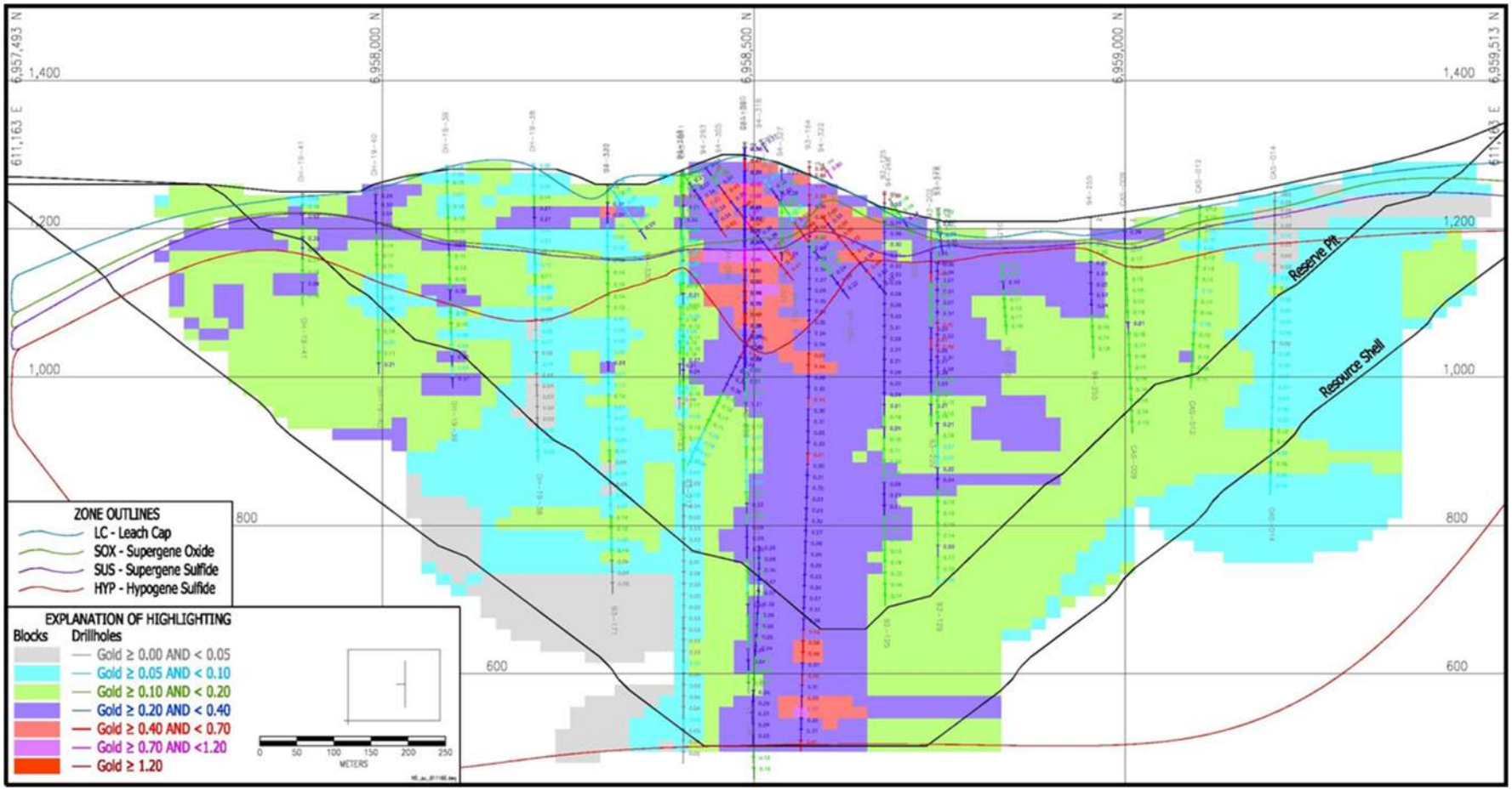
(Source: IMC, 2022)

Figure 14-15: Total Copper Grades on North-South Cross Section 611,165E



(Source: IMC, 2022)

Figure 14-16: Gold Grades on East-West Cross Section 6,958,600N



(Source: IMC, 2022)

Figure 14-17: Gold Grades on North-South Cross Section 611,165E

14.5.8 Bulk Density

Over 14,000 specific gravity measurements on core samples were included with the Casino assay database. IMC excluded six measurements that exceeded 4.0 and seven measurements less than 1.25, and also samples outside of the coordinate limits 6,957,500N to 6,960,000N and 609,000E to 612,000E and tabulated the remaining values by oxidation type as shown in Table 14-3.

Table 14-13: Statistics of Specific Gravity Measurement by Oxidation Zone

Oxidation Zone	Zone Code	No. of Samples	Mean S.G.	Std. Dev. S.G.
OVB	1	101	2.482	0.174
LC	2	2,442	2.517	0.129
SOX	3	999	2.578	0.153
SUS	4	3,062	2.626	0.129
HYP	5	7,337	2.652	0.112
TOTAL		13,941	2.616	0.133

It can be seen the mean values increase from OVB to LC to SOX to SUS to HYP to WST, i.e., the higher the level of oxidation the lower the specific gravity.

For the 2020 model update, IMC also examined the specific gravity measurements by rock type, but other than the overburden, the averages by rock type are very similar to each other, ranging from 2.612 to 2.637; it is more meaningful to group the data by oxidation type.

The average specific gravity values on the table were also assumed to represent bulk density measurements, in tonnes per cubic metre, without any adjustments, and assigned to the block model based on oxidation type.

There are sufficient measurements that IMC also investigated estimating values in a similar manner as the grade estimates. The means shown on the table were used as background values for blocks without sufficient close data to estimate them. The averages of blocks done by estimation tended to exceed the table values by a percent or so. Because of this IMC assigned values as the average zone values rather than estimation of the individual blocks.

14.5.9 Resource Classifications

For the purpose of classifying measured and indicated versus inferred mineral resources, an additional block estimate was done. This was based on the same search orientations and search radii as the grade estimates. The estimate was based on a maximum of three composites, a minimum of three, and a maximum of one composite per hole. This estimate provides the average distance to the nearest three holes to each block and was put into the block model. Note the grades from this estimate were not used.

Blocks with an average distance to the nearest three holes less than 100 m were assigned as indicated mineral resource. Blocks with an average distance to three holes greater than 100 m were assigned to inferred mineral resource. Generally (not specific to Casino) an average distance to the nearest three holes of 100 m corresponds to an average drill spacing of about 133 m. These estimates are approximate. It is noted that the nominal spacing for much of the Casino drilling is about 100 m.

Figure 14-18 shows the probability plots of the average distances to the nearest 3 holes for the supergene sulphide. Figure 14-19 shows the plot for the hypogene sulphide.

On Figure 14-2 it can be seen that there is an area on the eastern side of the deposit where there is a combination of vertical and angle holes that reduce the average sample spacing in the area to about 70 m or so. A solid was designed in this area to define measured mineral resources. Figure 14-20 and Figure 14-21 show the resource classification on and east-west and north-south cross section, respectively.

The analytical method of distinguishing between indicated and inferred mineral resources resulted in some small groupings of inferred blocks surrounded by indicated blocks. Some filtering was done to remove many, but not all, of these blocks. The filters identified inferred blocks that contacted two, three, or four indicated blocks and set them to indicated blocks. Several passes of filtering were done.

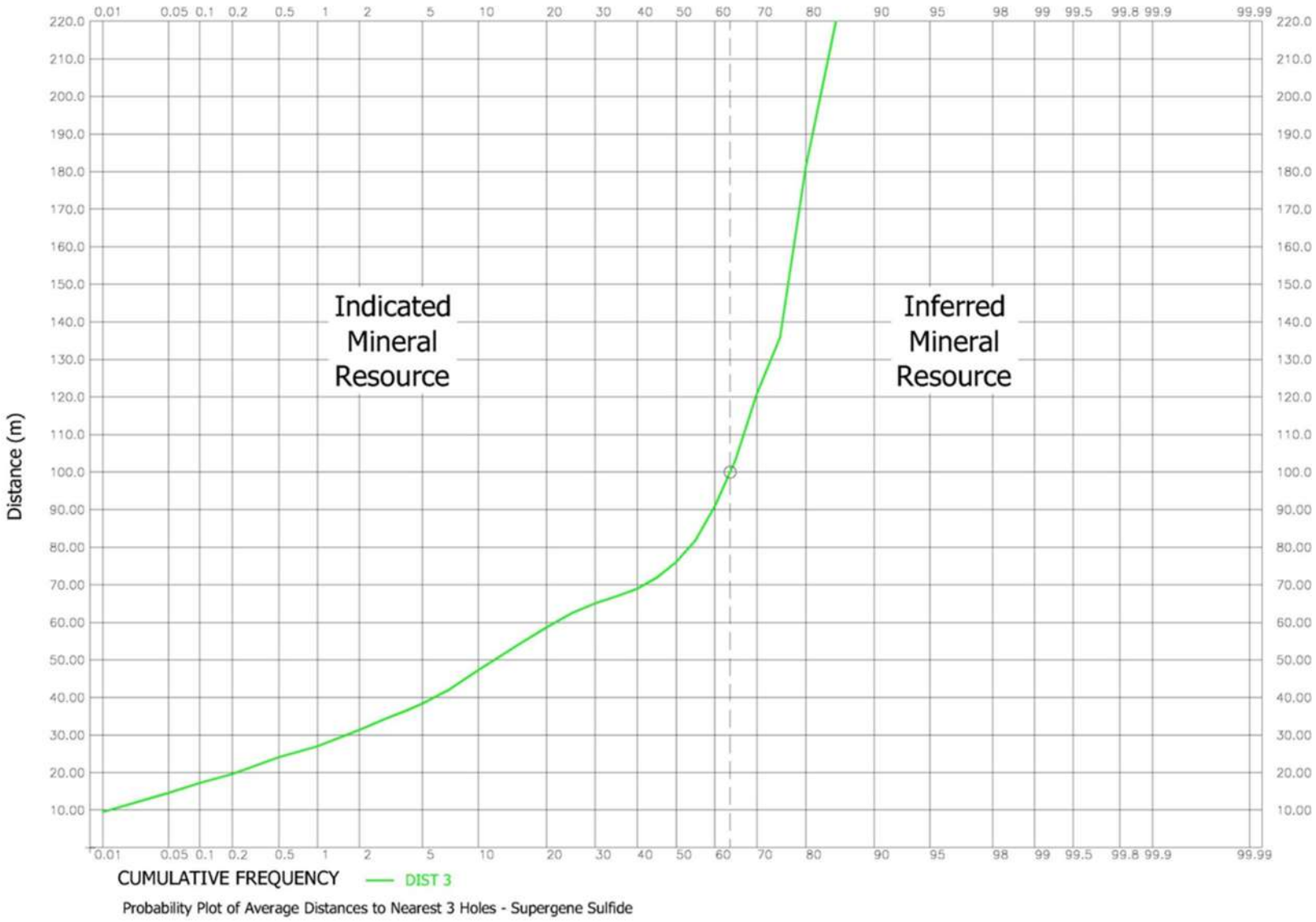


Figure 14-18: Probability Plot of Average Distance to Nearest 3 Holes – Supergene Sulphide (Source: IMC, 2022)

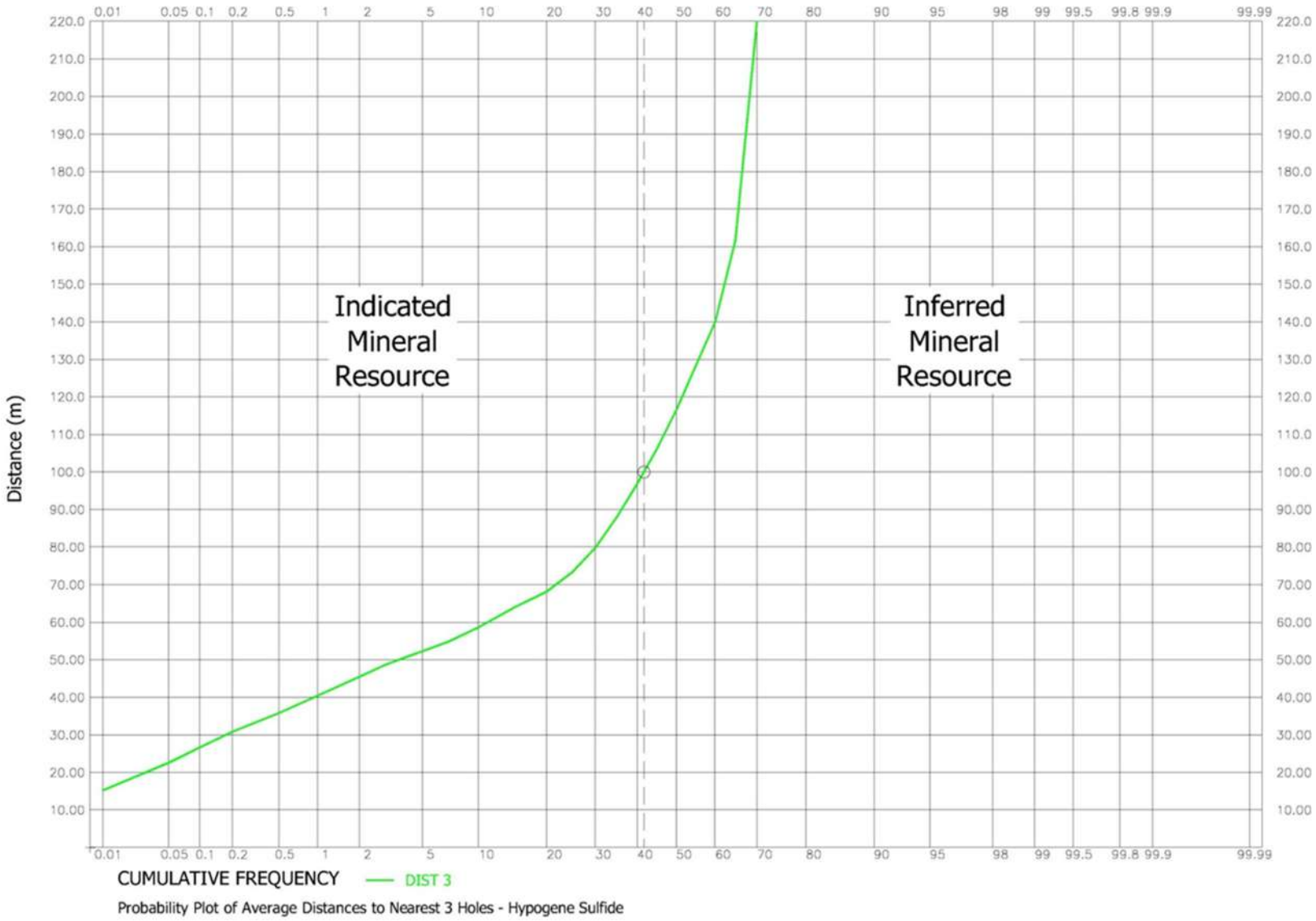


Figure 14-19: Probability Plot of Average Distance to Nearest 3 Holes – Hypogene Sulphide (Source: IMC, 2022)

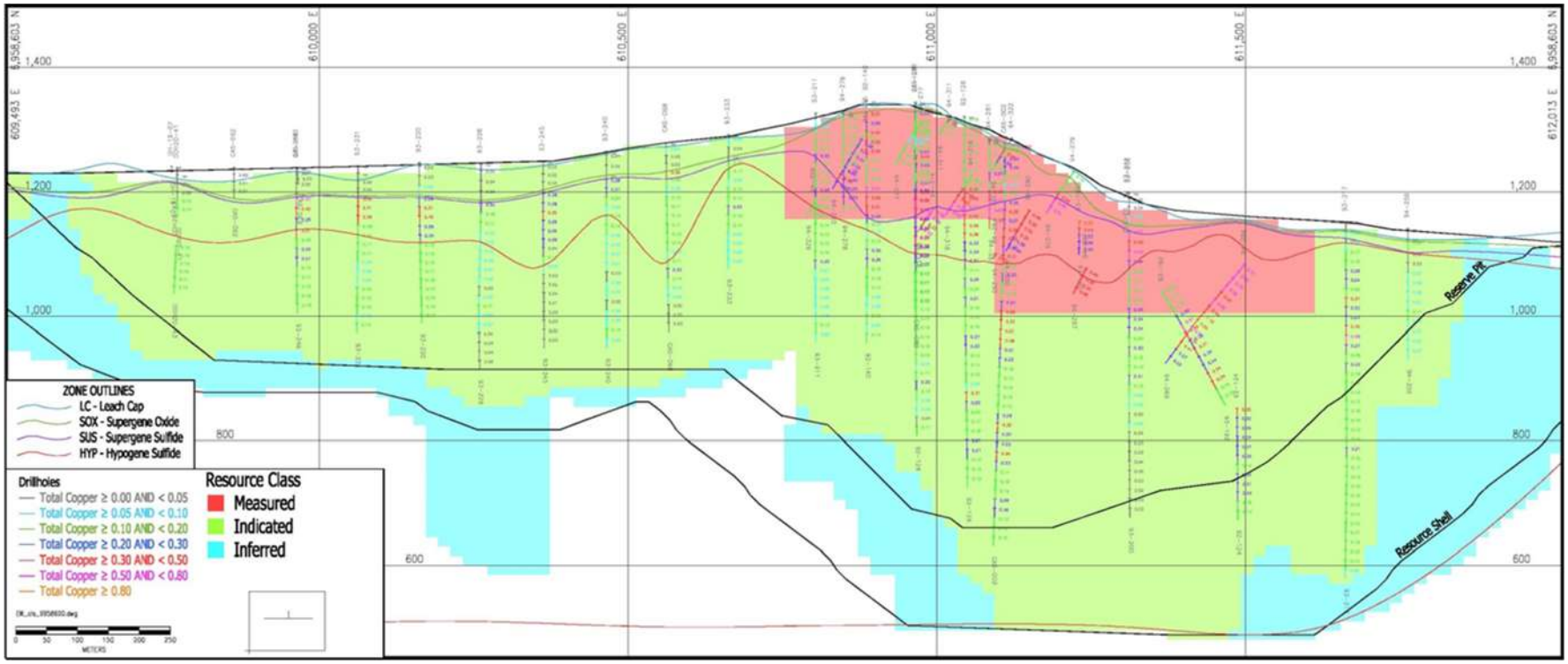


Figure 14-20: Resource Classification on East-West Cross Section 6,958,600N (Source: IMC, 2022)

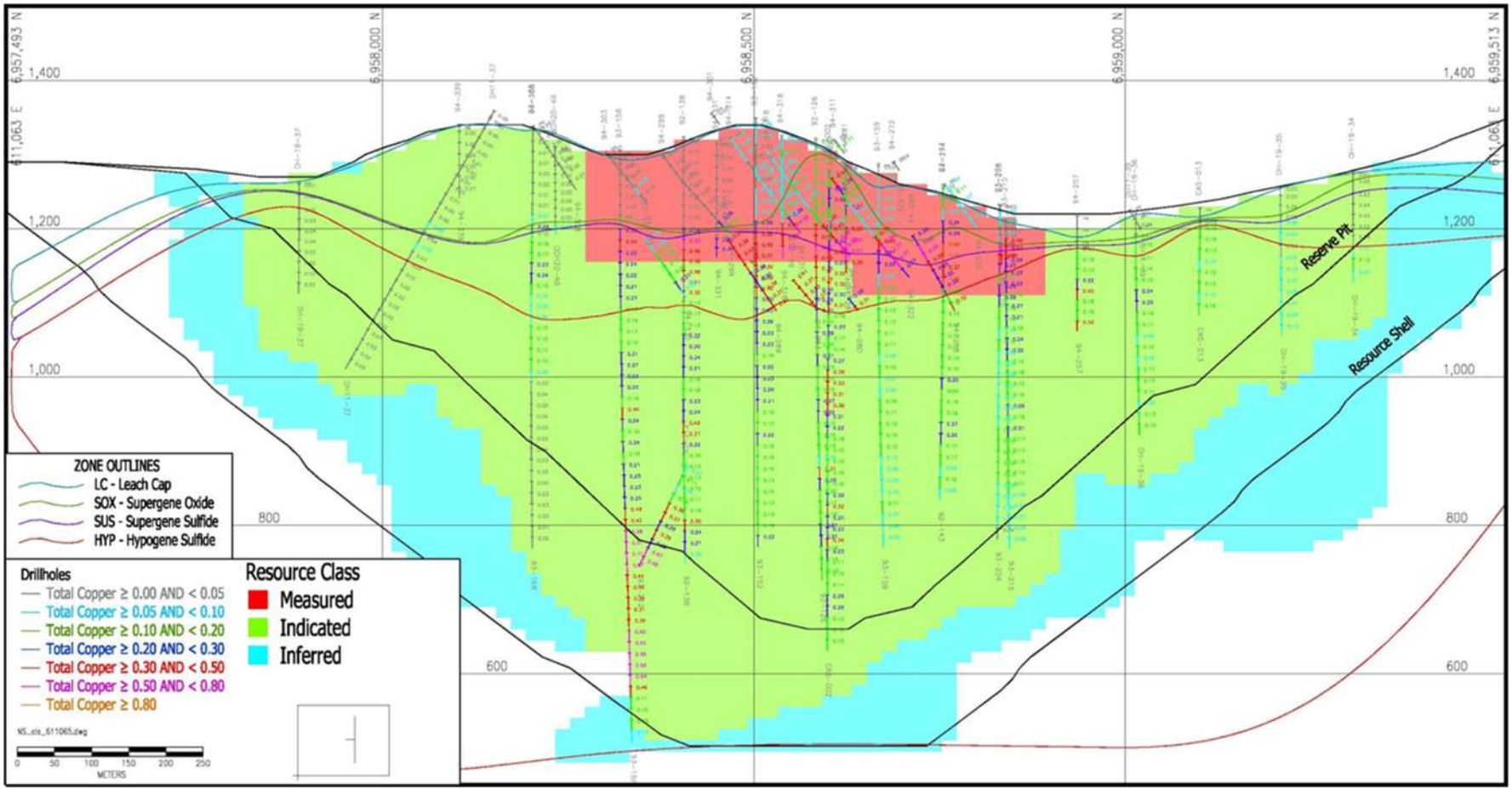


Figure 14-21: Resource Classification on North-South Cross Section 611,065E (Source: IMC, 2022)

14.5.10 Comparison of 2022 and 2020 Mineral Resource

This section compares the updated 2022 mineral resource with the 2020 mineral resource. Both are based on resource models and calculations developed by IMC. Note the following differences between the estimates:

- There was new drilling data and updated interpretations of the oxidation zones and rock types for the 2022 model.
- The 2022 model used more hard boundaries for oxidation zones and rock types for grade estimation than the 2020 model.
- Table 14-14 compares commodity prices and NSR cut-offs for the estimates. Commodity prices, but also costs and NSR cut-offs, are higher for the 2022 estimate.
- In spite of the above differences, the resource cones shells developed for the two estimates are very similar. The total volume of material incorporated into the two estimates is not materially different.

Table 14-15 compares mineral resources amenable to milling. These include the supergene oxide, supergene sulphide and hypogene sulphide materials. For the 2020 mineral resource, measured and indicated mineral resources amounted to 2.17 billion tonnes at 0.155% total copper, 0.182 g/t gold, 0.017% moly, and 1.43 g/t silver. This amounted to 7.43 billion pounds of contained copper, 12.7 million ounces of contained gold, 811.6 million pounds of contained molybdenum and 100.2 million ounces of contained silver. Inferred mineral resources was an additional 1.43 billion tonnes at 0.103% total copper, 0.139 g/t gold, 0.010% moly, and 1.16 g/t silver.

For the 2022 mineral resource, measured and indicated mineral resources amounted to 2.26 billion tonnes at 0.150% total copper, 0.177 g/t gold, 0.0159% moly, and 1.42 g/t silver. This amounts to 7.45 billion pounds of contained copper, 12.9 million ounces of contained gold, 791.2 million pounds of contained molybdenum and 103.1 million ounces of contained silver. Inferred mineral resources was an additional 1.37 billion tonnes at 0.100% total copper, 0.137 g/t gold, 0.010% moly, and 1.14 g/t silver.

For measured and indicated mineral resources the 2022 resource has 3.9% more resources tonnes at a 3.6% lower copper grade, a 2.6% lower gold grade, a 6.2% lower molybdenum grade, and a 1.0% lower silver grade. The overall result is 0.2% more contained copper, 1.3% more contained gold, 2.5% less contained moly, and 2.9% more contained silver. The reduced metal grades in the 2022 model are a consequence of the additional hard boundaries for the rock types. In previous models the higher-grade intrusive breccia composites were allowed to mix with Patton Porphyry and Dawson Creek Granodiorite. For measured mineral resources the 2022 model exhibits a transfer of resource from hypogene sulphide to supergene sulphide compared with the 2020 model. This is due to the updated geologic interpretation of the zones.

Table 14-16 compares mineral resources amenable to leaching. This is the leach cap material. For the 2020 mineral resource, measured and indicated mineral resource amounts to 217.4 million tonnes at 0.25 g/t gold, 0.035% total copper, and 1.90 g/t silver. This amounts to 1.76 million ounces of contained gold, 166.5 million pounds of contained copper and 13.3 million ounces of contained silver. Molybdenum will not be recovered in the leaching process. The 2020 mineral resource estimate was based on an NSR cut-off of C$5.46/t.

For the 2022 mineral resource estimate, measured and indicated mineral resource amounts to 231.7 million tonnes at 0.25 g/t gold, 0.039% total copper, and 1.89 g/t silver. This amounts to 1.88 million ounces of contained gold, 196.9 million pounds of contained copper and 14.1 million ounces of contained silver. This estimate is based on an NSR cut-off of C$6.61/t. The increased tonnage in the measured and indicated, as well as the inferred resource category, is due to the additional drilling, which expanded the interpretation of the leach cap zone. For measured and indicated resources, the copper grade is slightly higher for the 2022 estimate and the gold and silver grades are about

the same for the two estimates. For inferred mineral resources the gold grade is higher for the 2022 estimate, but the silver estimate is lower.

Table 14-14: Comparison of Economic Parameters for 2020 versus 2022 Mineral Resource

Year	Copper Price US$/lb	Gold Price US$/oz	Moly Price US$/lb	Silver Price US$/oz	Mill NSR Cutoff C$/t	Leach NSR Cutoff C$/t
2020	2.75	1500	11.00	18.00	5.70	5.46
2022	3.50	1650	12.00	22.00	6.11	6.61

Table 14-15: Comparison of 2022 and 2020 Mineral Resource – Mill Material

2022 Mineral Resource	Kt	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	Copper (mlbs)	Gold (koz)	Moly (%)	Silver (koz)
Measured Mineral Resource:									
Supergene Oxide – Mill	34,655	0.271	0.491	0.0251	2.24	207.0	547	19.2	2,496
Supergene Sulphide – Mill	57,084	0.354	0.373	0.0273	2.04	445.5	685	34.4	3,744
Hypogene Sulphide – Mill	53,114	0.257	0.327	0.0185	1.94	300.9	558	21.7	3,313
Measured Mineral Resource	144,853	0.299	0.384	0.0235	2.05	953.5	1,790	75.2	9,553
Indicated Mineral Resource:									
Supergene Oxide – Mill	73,692	0.194	0.155	0.0118	1.13	315.2	367	19.2	2,677
Supergene Sulphide – Mill	347,684	0.173	0.176	0.0135	1.41	1,326.1	1,967	103.5	15,762
Hypogene Sulphide – Mill	1,692,782	0.130	0.161	0.0159	1.38	4,851.5	8,762	593.4	75,107
Indicated Mineral Resource	2,114,158	0.139	0.163	0.0154	1.38	6,492.7	11,097	716.0	93,545
Meas/Ind Mineral Resource:									
Supergene Oxide – Mill	108,347	0.219	0.262	0.0161	1.49	522.2	914	38.3	5,173
Supergene Sulphide – Mill	404,768	0.199	0.204	0.0154	1.50	1,771.6	2,652	137.8	19,506
Hypogene Sulphide – Mill	1,745,896	0.134	0.166	0.0160	1.40	5,152.4	9,321	615.0	78,419
Meas/Ind Mineral Resource	2,259,011	0.150	0.177	0.0159	1.42	7,446.2	12,887	791.2	103,098
Inferred Mineral Resource:									
Supergene Oxide – Mill	18,628	0.156	0.108	0.0019	0.67	64.1	65	0.8	401
Supergene Sulphide – Mill	103,870	0.056	0.171	0.0019	1.04	128.2	571	4.4	3,473
Hypogene Sulphide – Mill	1,249,046	0.103	0.135	0.0102	1.16	2,836.3	5,421	280.9	46,584
Inferred Mineral Resource	1,371,544	0.100	0.137	0.0095	1.14	3,028.6	6,057	286.0	50,458
2020 Mineral Resource	**Kt**	**Copper (%)**	**Gold (g/t)**	**Moly (%)**	**Silver (g/t)**	**Copper (mlbs)**	**Gold (koz)**	**Moly (%)**	**Silver (koz)**
Measured Mineral Resource:									
Supergene Oxide – Mill	34,494	0.248	0.490	0.0253	2.31	188.6	543	19.2	2,562
Supergene Sulphide – Mill	52,176	0.383	0.398	0.0284	2.12	440.6	668	32.7	3,556
Hypogene Sulphide – Mill	58,614	0.276	0.343	0.0222	1.94	356.6	646	28.7	3,656
Measured Mineral Resource	145,284	0.308	0.398	0.0252	2.09	958.8	1,857	80.6	9,774
Indicated Mineral Resource:									
Supergene Oxide – Mill	65,636	0.178	0.154	0.0107	1.16	257.6	325	15.5	2,448
Supergene Sulphide – Mill	311,700	0.186	0.178	0.0141	1.42	1,278.1	1,784	96.9	14,231
Hypogene Sulphide – Mill	1,650,677	0.135	0.165	0.0170	1.39	4,912.8	8,757	618.6	73,769
Indicated Mineral Resource	2,028,013	0.144	0.167	0.0164	1.39	6,448.5	10,866	731.0	90,448
Meas/Ind Mineral Resource:									
Supergene Oxide – Mill	100,130	0.202	0.270	0.0157	1.56	446.2	868	34.7	5,010
Supergene Sulphide – Mill	363,876	0.214	0.210	0.0162	1.52	1,718.7	2,451	129.6	17,787
Hypogene Sulphide – Mill	1,709,291	0.140	0.171	0.0172	1.41	5,269.4	9,403	647.3	77,425
Meas/Ind Mineral Resource	2,173,297	0.155	0.182	0.0169	1.43	7,434.3	12,723	811.6	100,222

Inferred Mineral Resource:									
Supergene Oxide – Mill	8,997	0.096	0.118	0.0028	1.19	19.0	34	0.6	344
Supergene Sulphide – Mill	52,632	0.072	0.141	0.0021	1.25	83.5	239	2.4	2,115
Hypogene Sulphide – Mill	1,368,569	0.104	0.139	0.0106	1.16	3,137.8	6,116	319.8	51,041
Inferred Mineral Resource	1,430,198	0.103	0.139	0.0102	1.16	3,240.4	6,389	322.8	53,501
& Difference	**Kt**	**Copper (%)**	**Gold (g/t)**	**Moly (%)**	**Silver (g/t)**	**Copper (mlbs)**	**Gold (koz)**	**Moly (%)**	**Silver (koz)**
Measured Mineral Resource:									
Supergene Oxide – Mill	0.5%	9.3%	0.2%	-0.8%	-3.0%	9.8%	0.7%	-0.3%	-2.6%
Supergene Sulphide – Mill	9.4%	-7.6%	-6.3%	-3.9%	-3.8%	1.1%	2.5%	5.2%	5.3%
Hypogene Sulphide – Mill	-9.4%	-6.9%	-4.7%	-16.7%	0.0%	-15.6%	-13.6%	-24.5%	-9.4%
Measured Mineral Resource	-0.3%	-3.0%	-3.3%	-6.4%	-2.0%	-3.3%	-3.6%	-6.7%	-2.3%
Indicated Mineral Resource:									
Supergene Oxide – Mill	12.3%	9.0%	0.6%	10.3%	-2.6%	22.4%	13.0%	23.8%	9.4%
Supergene Sulphide – Mill	11.5%	-7.0%	-1.1%	-4.3%	-0.7%	3.7%	10.3%	6.8%	10.8%
Hypogene Sulphide – Mill	2.6%	-3.7%	-2.4%	-6.5%	-0.7%	-1.2%	0.1%	-4.1%	1.8%
Indicated Mineral Resource	4.2%	-3.4%	-2.0%	-6.0%	-0.8%	0.7%	2.1%	-2.1%	3.4%
Meas/Ind Mineral Resource:									
Supergene Oxide – Mill	8.2%	8.2%	-2.7%	2.1%	-4.6%	17.0%	5.3%	10.4%	3.3%
Supergene Sulphide – Mill	11.2%	-7.3%	-2.8%	-4.4%	-1.4%	3.1%	8.2%	6.4%	9.7%
Hypogene Sulphide – Mill	2.1%	-4.3%	-3.0%	-7.0%	-0.8%	-2.2%	-0.9%	-5.0%	1.3%
Meas/Ind Mineral Resource	3.9%	-3.6%	-2.6%	-6.2%	-1.0%	0.2%	1.3%	-2.5%	2.9%
Inferred Mineral Resource:									
Supergene Oxide – Mill	107.0%	62.5%	-8.5%	-32.1%	-43.7%	236.5%	89.5%	40.5%	16.6%
Supergene Sulphide – Mill	97.4%	-22.2%	21.3%	-9.5%	-16.8%	53.5%	139.3%	78.6%	64.2%
Hypogene Sulphide – Mill	-8.7%	-1.0%	-2.9%	-3.8%	0.0%	-9.6%	-11.4%	-12.2%	-8.7%
Inferred Mineral Resource	-4.1%	-2.5%	-1.1%	-7.6%	-1.7%	-6.5%	-5.2%	-11.4%	-5.7%

Table 14-16: Comparison of 2022 and 2020 Mineral Resource – Leach Material

2022 Mineral Resource	Tonnes Kt	Copper (%)	Gold (g/t)	Silver (g/t)	Copper (mlbs)	Gold (koz)	Silver (koz)
Measured	43,297	0.054	0.443	2.68	51.5	616.7	3,731
Indicated	188,395	0.035	0.209	1.71	145.4	1,265.9	10,358
M+I	**231,692**	**0.039**	**0.253**	**1.89**	**196.9**	**1,882.6**	**14,088**
Inferred	40,927	0.052	0.203	1.43	46.9	267.1	1,882
2020 Mineral Resource	**Kt**	**Copper (%)**	**Gold (g/t)**	**Silver (g/t)**	**Copper (mlbs)**	**Gold (koz)**	**Silver (koz)**
Measured	37,161	0.048	0.447	2.75	39.3	534.1	3,286
Indicated	180,241	0.032	0.212	1.73	127.2	1,228.5	10,025
M+I	**217,402**	**0.035**	**0.252**	**1.90**	**166.5**	**1,762.6**	**13,311**
Inferred	31,137	0.025	0.167	1.70	17.2	167.2	1,702
% Difference	**Kt**	**Copper (%)**	**Gold (g/t)**	**Silver (g/t)**	**Copper (mlbs)**	**Gold (koz)**	**Silver (koz)**
Measured	16.5%	12.5%	-0.9%	-2.5%	31.1%	15.5%	13.5%
Indicated	4.5%	9.4%	-1.4%	-1.2%	14.3%	3.0%	3.3%
M+I	**6.6%**	**11.0%**	**0.2%**	**-0.7%**	**18.3%**	**6.8%**	**5.8%**
Inferred	31.4%	108.0%	21.6%	-15.9%	173.4%	59.8%	10.6%

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE

Table 15-1 presents the Mineral Reserve estimate for the Casino Project. It can be seen that there are Mineral Reserves amenable to milling and Mineral Reserves amenable to heap leaching. The Proven and Probable Mineral Reserves amenable to milling amount to 1.22 billion tonnes at 0.19% total copper, 0.22 g/t gold, 0.021% moly and 1.7 g/t silver. The Proven and Probable Mineral Reserve amenable to heap leaching amounts to 209.6 million tonnes at 0.26 g/t gold, 0.036% copper and 1.9 g/t silver. The effective date of this Mineral Reserve estimate is June 13, 2022. The low-grade stockpile portion of the Mineral Reserve is economic, but lower grade, material that will be stockpiled and processed at the end of open-pit operations.

The Mineral Reserve estimate is based on an open-pit mine plan and mine production schedule developed by IMC. The Mineral Reserve estimate is based on commodity prices of US$ 3.25 per pound copper, US$1550 per ounce gold, US$ 12.00 per pound molybdenum and US$22.00 per ounce silver. Measured Mineral Resource in the mine production schedule was converted to Proven Mineral Reserve and Indicated Mineral Resource in the schedule was converted to Probable Mineral Reserve.

The Mineral Reserves are classified in accordance with the "CIM Definition Standards – For Mineral Resources and Mineral Reserves" adopted by the CIM Council (as amended, the "CIM Definition Standards") in accordance with the requirements of NI 43-101. Mineral Reserve estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained. The project is in a jurisdiction friendly to mining.

IMC does not believe that there are significant risks to the Mineral Reserve estimate based on metallurgical or infrastructure factors or environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. There has been a significant amount of metallurgical testing, however recoveries lower than forecast would result in loss of revenue for the project. Other risks to the Mineral Reserve estimate are related to economic parameters such as prices lower than forecast or costs higher than the current estimates. The impact of these is modeled in the sensitivity study with the economic analysis in Section 22.

All of the mineralization comprised in the Mineral Reserve estimate with respect to the Casino Project is contained on mineral titles controlled by Western Copper and Gold.

The Mineral Reserves include allowances for mining dilution and ore loss. IMC believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and ore loss effects.

Table 15-1: Mineral Reserve

Mineral Reserve (Milling):	Tonnes Mt	NSR (C$/t)	Tot Cu (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Proven Mineral Reserve	140.1	38.50	0.31	0.39	0.024	2.1	0.67	944	1.8	74.9	9.4
Mill Ore	124.2	41.20	0.32	0.43	0.027	2.2	0.72	885	1.7	72.6	8.8
Low Grade Stockpile	16.0	17.54	0.17	0.15	0.007	1.1	0.29	59	0.1	2.3	0.6
Probable Mineral Reserve	1,076.9	23.68	0.17	0.19	0.021	1.6	0.36	4,135	6.7	497.1	55.5
Mill Ore	825.1	26.15	0.19	0.21	0.024	1.7	0.40	3,484	5.6	430.9	45.9
Low Grade Stockpile	251.9	15.57	0.12	0.14	0.012	1.2	0.24	651	1.1	66.2	9.6
Proven/Probable Reserve	**1,217.1**	**25.38**	**0.19**	**0.22**	**0.021**	**1.7**	**0.40**	**5,079**	**8.5**	**571.9**	**64.9**
Mill Ore	949.2	28.12	0.21	0.24	0.024	1.8	0.44	4,369	7.3	503.5	54.7
Low Grade Stockpile	267.8	15.69	0.12	0.14	0.012	1.2	0.25	710	1.2	68.5	10.2
Mineral Reserve (Heap Leach):	Tonnes Mt	NSR (C$/t)	Gold (g/t)	Tot Cu (%)	Moly (%)	Silver (g/t)	AuEq (g/t)	Gold (Moz)	Copper (Mlbs)	Moly (Mlbs)	Silver (Moz)
Proven Mineral Reserve	42.9	22.52	0.45	0.055	n.a.	2.7	0.47	0.62	51.8	n.a.	3.7
Probable Mineral Reserve	166.8	11.14	0.22	0.031	n.a.	1.8	0.23	1.17	113.5	n.a.	9.4
Proven/Probable Leach Reserve	**209.6**	**13.47**	**0.26**	**0.036**	**n.a.**	**1.9**	**0.28**	**1.78**	**165.3**	**n.a.**	**13.1**

Notes:

1. The Mineral Reserve estimate has an effective date of 13 April 2022 and was prepared using the CIM Definition Standards (10 May 2014).
2. Columns may not sum exactly due to rounding.
3. Mineral Reserves are based on commodity prices of US$3.25/lb Cu, US$1550/oz Au, US$12.00/lb Mo, and US$22.00/oz Ag.
4. Mineral Reserves amenable to milling are based on NSR cut-offs that vary by time period to balance mine and plant production capacities (see Section 16). They range from a low of $6.11/t to a high of $25.00/t.
5. NSR value for supergene (SOX and SUS) mill material is NSR (C$/t) = $73.63 x recoverable copper (%) + $40.41 x gold (g/t) + $142.11 x moly (%) + 0.464 x silver (g/t), based on recoveries of 69% gold, 52.3% molybdenum and 60% silver. Recoverable copper = 0.94 x (total copper – soluble copper).
6. NSR value for hypogene (HYP) mill material is NSR (C$/t) = $67.88 x copper (%) + $38.66 x gold (g/t) + $213.78 x moly (%) + $0.386 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 78.6% molybdenum and 50% silver.
7. Mineral Reserves amenable to heap leaching are based on an NSR cut-off of $6.61/t.
8. NSR value for leach material is NSR (C$/t) = $14.05 x copper (%) + $47.44 x gold (g/t) + $0.210 x silver (g/t), based on recoveries of 18% copper, 80% gold and 26% silver.
9. AuEq and CuEq values are based on prices of US$ 3.25/lb Cu, US$ 1550/oz Au, US$ 12.00/lb Mo, and US$ 22.00/oz Ag, and account for all metal recoveries and smelting/refining charges.
10. The NSR calculations also account for smelter/refinery treatment charges and payables.
11. Table 15-2 accompanies this Mineral Reserve estimate and shows all relevant parameters.

15.2 ECONOMIC PARAMETERS

15.2.1 Commodity Prices

Table 15-2 shows the economic and recovery parameters used for mine design and production scheduling. In US dollar (US$) terms, commodity prices are US$3.25 per pound copper, US$1550 per ounce gold, US$22 per ounce silver, and US$12.00 per pound moly. A conversion of US$0.80 = C$1.00 was used to convert the prices to C$. IMC believes these prices to be reasonable based on 1) historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economists.

15.2.2 Cost and Recovery Estimates

15.2.2.1 Mining Cost

The base mining cost of C$1.934 per total tonne was estimated by IMC. The estimate is based on mining equipment requirements specific to Casino and typical prices for fuel, blasting agents, equipment parts, and labor, etc.

15.2.2.2 Processing of Mill Material

Mill material refers to the supergene oxide, supergene sulphide, and hypogene sulphide zones of the mineral deposit. The processing will be in a conventional sulphide flotation plant that will produce copper and molybdenum concentrates that will be sold to commercial copper smelters and molybdenum roasting plants. The base unit costs for processing and G&A are estimated at C$5.724 and C$0.385 per tonne, respectively. These are based on estimates developed by M3. The estimated plant recoveries for gold, moly, and silver in the supergene and hypogene zones are shown on Table 15-2. The copper recovery is estimated at 92.2% for hypogene sulphide material. The plant recovery for supergene oxide and supergene sulphide is estimated as follows:

Copper recovery = 94% (Cut% - Cuw%) / Cut%

Where,

Cut% = Total copper grade
Cuw% = Weak acid soluble copper grade

The copper, gold, and silver payable percentages shown on Table 15-2 are typical terms for copper concentrates, assuming a clean concentrate with a copper concentrate grade of 28% copper or greater. The off-site cost per pound of copper is estimated at C$0.486, a cost developed by M3. Gold and silver refining is estimated at C$6.574 and C$1.347 per ounce, respectively.

Note that the off-site cost for molybdenum is assumed to be accounted in the 85% payable percentage for molybdenum in concentrate, i.e., this is assumed to be the net payable after treatment and transportation charges. This is applicable to a clean molybdenum concentrate with a moly grade of about 50% or greater.

The table also shows an NSR royalty of 2.75% and a concentrate loss of 0.5% are incorporated into the economic calculations.

Table 15-2: Economic Parameters for Mine Design (C$)

Parameter	Units	Mill Material		Heap Leach
		SOX/SUS	HYP	
Commodity Prices and Exchange Rate:				
Copper Price Per Pound (US$)	(US$)	3.25	3.25	3.25
Gold Price Per Ounce (US$)	(US$)	1550.00	1550.00	1550.00
Silver Price Per Ounce (US$)	(US$)	22.00	22.00	22.00
Molybdenum Price Per Ounce (US$)	(US$)	12.00	12.00	12.00
Exchange Rate (CAD to $US)	(none)	0.80	0.80	0.80
Cooper Price Per Ounce (C$)	(C$)	4.06	4.06	4.06
Gold Price Per Ounce (C$)	(C$)	1937.50	1937.50	1937.50
Silver Price Per Ounce (C$)	(C$)	27.50	27.50	27.50
Molybdenum Price Per Ounce (C$)	(C$)	15.00	15.00	15.00
Mining Cost Per Total Tonne:				
Mining Cost Per Total Tonne	(C$)	1.934	1.934	1.934
Processing and G&A Per Tonne Processed				
Processing	(C$)	5.724	5.724	6.227
G&A	(C$)	0.385	0.385	0.385
Total Processing and G&A	(C$)	6.109	6.109	6.612
Average Plant Recoveries:				
Copper Recovery	(%)	Note 1	92.2%	18.0%
Gold Recover	(%)	69.0%	66.0%	80.0%
Silver Recovery	(%)	60.0%	50.0%	26.0%
Moly Recovery	(%)	52.3%	78.6%	N.A.
Refinery Payables:				
Copper Payable	(%)	96.5%	96.5%	96.5%
Gold Payable	(%)	97.5%	97.5%	98.0%
Silver Payable	(%)	95.0%	95.0%	98.0%
Molybdenum Payable	(%)	85.0%	85.0%	N.A.
Payable Concentrate (0.5% Conc Loss)	(%)	99.5%	99.5%	Cu Only
Offsite Costs:				
Copper SRF Cost Per Pound	(C$)	0.486	0.486	0.271
Gold Refining Per Ounce	(C$)	6.574	6.574	2.122
Silver Refining Per Ounce	(C$)	1.347	1.347	1.103
Molybdenum Freight/Treatment Per Pound	(C$)	Note 2	Note 2	N.A.
NSR Royalty	%	2.75%	2.75%	2.75%
NSR Factors:				
Copper Factor (Note 4)	(C$/t)	73.63	67.88	14.05
Gold Factor (Note 3)	(C$/t)	40.41	38.66	47.44
Silver Factor (Note 3)	(C$/t)	0.464	0.386	0.210
Moly Factor (Note 3)	(C$/t)	142.11	213.78	N.A.
NSR Cutoff Grades:				
Breakeven Cutoff (C$/t)	(C$/t)	8.04	8.04	8.55
Internal Cutoff (C$/t)	(C$/t)	6.11	6.11	6.61

Note 1: Recovery = 94% x (Cutotal – CuWAS)/(Cutotal)
Note 2: Moly offsite costs are accounted in payable percentage
Note 3: NSR factors are applied to model grades
Note 4: For copper, SOX/SUS factor is applied to recovered copper grade, for HYP and leach the factor is applied to total copper grade.

15.2.2.3 Processing of Leach Material

Leach material refers to the leach capping of the mineral deposit. Processing is by crushing and heap-leaching with cyanide. Gold and silver from the heap leach will report to a typical doré which will be sent to a refinery. The SART process will be used to extract copper from the cyanide solution and produce a copper concentrate that can be sold to conventional copper smelters. Heap leach processing is estimated at C$6.227 per tonne. The G&A cost of C$0.385 per tonne is also applied to leach material.

Heap leach recoveries are estimated at 18% for copper, 80% for gold, and 26% for silver. Typical terms for refining costs are shown and are C$2.122/oz gold and C$1.103/oz silver. The payable percentage is estimated at 98% for gold and silver.

It is also assumed that the SART process will produce a copper concentrate with a grade of about 60% copper. Smelting and refining terms are assumed the same as for the flotation concentrate. This results in a smelting, refining, and freight charge of about C$0.271 per pound.

15.2.3 NSR Calculation

Due to multiple mineral products and also the variable recovery for copper in the supergene zones, Net Smelter Return (NSR) values, in Canadian Dollars, were calculated for each model block to use to classify blocks into potential ore and waste. For the leach material:

NSR_au = ($1937.50 – $2.122) x 0.80 x 0.98 x 0.9725 x gold (g/t) / 31.103 = C$47.44 x gold (g/t)

NSR_cu = ($4.06 - $0.271) x 0.18 x 0.965 x 0.995 x 0.9725 x copper (%) x 22.046
= C$14.05 x copper (%)

NSR_ag = ($27.50 - $1.103) x 0.26 x 0.98 x 0.9725 x silver (g/t) / 31.103 = C$0.210 x silver (g/t)

NSR = NSR_au + NSR_cu + NSR_ag

The internal NSR cut-off for leach material is the processing + G&A cost of C$6.61 per tonne since all the recoveries and refining costs are accounted for in the NSR calculation. Internal cut-off grade applies to blocks that have to be removed from the pit so the mining cost is a sunk cost. Internal cut-off is also generally the minimum cut-off that would be evaluated for mine scheduling. The breakeven NSR cut-off grade for leach material is C$8.55 per tonne (mining plus processing and G&A). The 0.9725 parameter in the equations accounts for the royalty.

For processing of hypogene sulphide material, the NSR values are calculated as:

NSR_cu = ($4.06 – $0.486) x 0.922 x 0.965 x 0.995 x 0.9725 x copper (%) x 22.046
= C$67.88 x copper (%)

NSR_au = ($1937.50 – $6.574) x 0.66 x 0.975 x 0.995 x 0.9725 x gold (g/t) / 31.103
= C$38.66 x gold (g/t)

NSR_mo = $15.00 x 0.786 x 0.85 x 0.995 x 0.9725 x moly (%) x 22.046 = C$213.78 x moly (%)

NSR_ag = ($27.50 - $1.347) x 0.50 x 0.95 x 0.995 x 0.9725 x silver (g/t) / 31.103
= C$0.386 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

For processing of supergene material, the NSR values are calculated as:

NSR_cu = ($4.06 – $0.486) x 0.965 x 0.995 x 0.9725 x rec_cu (%) x 22.046
= C$73.63 x rec_cu (%)

NSR_au = ($1937.50 – $6.574) x 0.69 x 0.975 x 0.995 x 0.9725 x gold(g/t) / 31.103
= C$40.41 x gold (g/t)

NSR_mo = $15.00 x 0.523 x 0.85 x 0.995 x 0.9725 x moly (%) x 22.046 = C$142.11 x moly (%)

NSR_ag = ($27.50 - $1.347) x 0.60 x 0.95 x 0.995 x 0.9725 x silver (g/t) / 31.103
= C$0.464 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

where,

rec_cu = 0.94 x (Cut% – Cuw%)

The internal NSR cut-off for flotation is the processing plus G&A cost of C$6.11. Breakeven NSR cut-off is C$8.04.

16 MINING METHODS

16.1 OPERATING PARAMETERS AND CRITERIA

This FS is based on a conventional open pit mine plan. Mine operations will consist of drilling large diameter blast holes (31 cm), blasting with a bulk emulsion, and loading into large off-road trucks with cable shovels and a hydraulic shovel. Mineral reserves amenable to processing will be delivered to the primary crushers or various stockpiles. Waste rock will be placed inside the limits of the tailings management facility (TMF). There will be a fleet of track dozers, rubber-tired dozers, motor graders and water trucks to maintain the working areas of the pit, stockpiles, and haul roads.

The following general parameters guided the development of the mining plan:

- Mill material is limited to about 1.2 billion tonnes, CMC elected to limit the capacity of the TMF to be comparable to the concept and overall physical characteristics of the TMF design favored in the Best Available Tailings Technology Study (BATT study).
- Total mine waste to be co-disposed with tailings is limited to about 600 million tonnes,
- Mill capacity is a nominal 120,000 tonnes per day (t/d), but actual plant throughput for the schedule is based on hardness of the various material types, and usually exceeds 120,000 t/d.

The geotechnical parameters relevant to the mine plan are discussed in Section 16.2 and are adequate for this Technical Report.

16.2 SLOPE ANGLES

Slope angles recommendations were developed by Knight Piésold Ltd. (KP) and documented in the report “Open Pit Geotechnical Design” (Knight Piésold, 2012). Table 16-1 shows the recommended angles by design sector and Figure 16-1 shows the design sectors. Note the wall geology descriptions on Table 16-1 are described in the Figure 16-1 legend.

Forty-five-degree inter-ramp angles were recommended for most of the slope sectors. The north sectors of the main pit and west pit were recommended to be designed at 42-degree inter-ramp angles. For the small amount of overburden on the north wall, the recommended angle was 27 degrees. The slope angle recommendations also specified that there be no more than 200 m of vertical wall at the inter-ramp angle without an extra wide catch bench (16 m instead of 8 m). Figure 16-2 shows the final pit design for this study.

Table 16-1: Recommended Slope Angles (Knight-Piésold, 2012)

Design Sector	Max. Slope Height	Wall Geology	Bench Face Angle	Bench Height	Bench Width	Inter-ramp Angle	Max. Inter-ramp Slope Height	Max. Overall Slope Angle	Comments
	m		°	m	m	°	m	°	
M-North	630	Overburden	**40**	**5**	**4**	**27**	**200**	**39**	Reduction of inter-ramp angle to 42° will reduce the risk of multi-bench wedge failure in bedrock.
		DRB	**60**	**15**	**8**	**42**			
M-Northeast	630	Overburden	**40**	**5**	**4**	**27**	**100 (in Weathered Zone) 200 (in Fresh Bedrock)**	**40**	Weathered Zone down to 250 m deep, additional stepouts/ramps should be incorporated into the Weathered Zone slopes.
		DRB	**65**	**15**	**8**	**45**			
M-South	540	DRB, PMS	**65**	**15**	**8**	**45**	**200**	**42**	Potential planar and wedge failures are kinematically possible, but at the limit of the defect friction angle.
Central	210	PMS	**65**	**15**	**8**	**45**	**200**	**N/A**	Central lower pit walls with various orientations. The south facing maybe subject to potential wedge failure.
W-North	285	DRB	**60**	**15**	**8**	**42**	**200**	**39**	Reduction of the inter-ramp slope to 42° will reduce the risk of multi-bench planar and wedge failures.
W-South	480	DRB	**65**	**15**	**8**	**45**	**200**	**42**	Potential bench scale toppling failure is expected.
W-Southwest	345	PMS	**65**	**15**	**8**	**45**	**200**	**42**	
W-West	225	DRB	**65**	**15**	**8**	**45**	**200**	**42**	Potential bench scale planar failure is expected.

X:\p19808\report\KPSlopeAngles\[Table 7.1 - Recommended Pit Slope Angles.xlsm]Table 7.1

NOTES:

1. MAXIMUM SLOPE HEIGHT REPRESENTS THE HIGHEST WALL IN EACH DESIGN SECTOR.
2. RECOMMENDED SLOPE ANGLES DETERMINED BY THE KINEMATIC AND ROCK MASS STABILITY ANALYSES.
3. OVERBURDEN IS NEGLIGIBLE IN THE WESTERN AND SOUTHERN PIT WALLS.
4. THE OVERALL SLOPE ANGLES INCLUDED 1 TO 3 STEPOUTS OR HAUL RAMPS IN THE FINAL PIT WALLS.

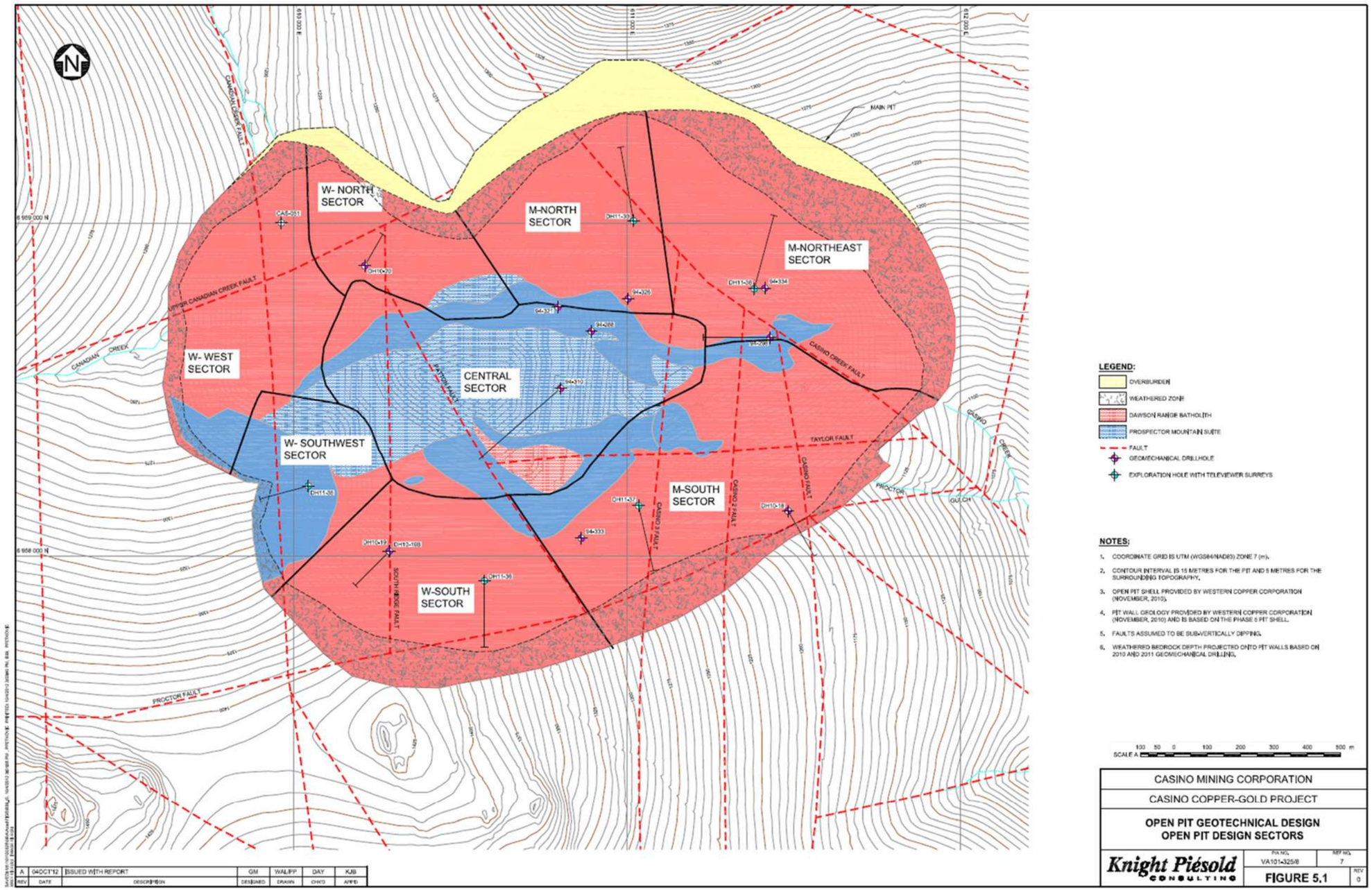


Figure 16-1: Open Pit Design Sectors (Knight-Piésold, 2012)

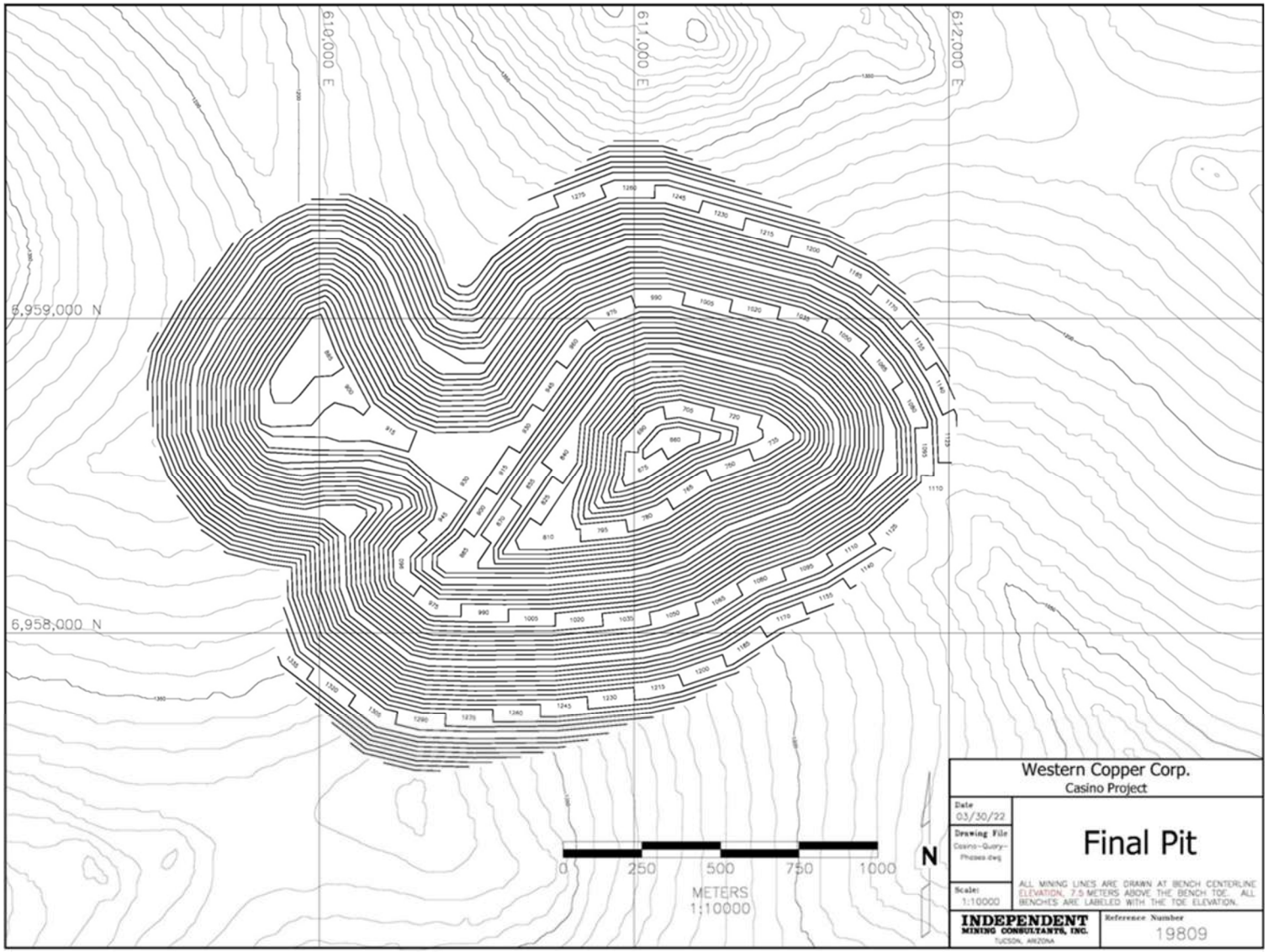


Figure 16-2: Final Pit Design (IMC, 2021)

16.3 MINING PHASES

The mine production schedule is based on five mining phases. The designs utilized 40 m wide roads at a maximum grade of 10%. The road width will accommodate trucks up to the 370-tonne class such as the Komatsu 980E. A suite of floating cone runs at various commodity prices were done to evaluate the final pit design (Phase 5). The final pit design is based on the floating cone shell at a copper price of US$1.75 per pound copper and US$835 per ounce gold. These are low prices compared to current market prices, CMC elected to limit the capacity of the TMF to be comparable to the concept and overall physical characteristics of the TMF design favored in the Best Available Tailings Technology Study (BATT study).

Figure 16-3 through Figure 16-7 show the mining phase designs. The mining phase 1 starter pit (Figure 16-3) is based on the cone shell run at US$1.75 copper and US$ 537 gold. Mining phase 2 (Figure 16-4) pushes phase 1 out in almost all directions and is based on the cone shell at US$1.375 copper and US$656 gold. Mining phase 3 (Figure 16-5) pushes the pit to the south and southwest to the US$1.50 copper and US$715 gold cone shell.

Mining phase 4 (Figure 16-6) is a small area in the northwest. The geometry approximates a portion of the US$1.375 copper / US$656 gold cone shell that was excluded from mining phase 2. Phase 5 (Figure 16-7) is a push to the north and northwest to the final pit limits.

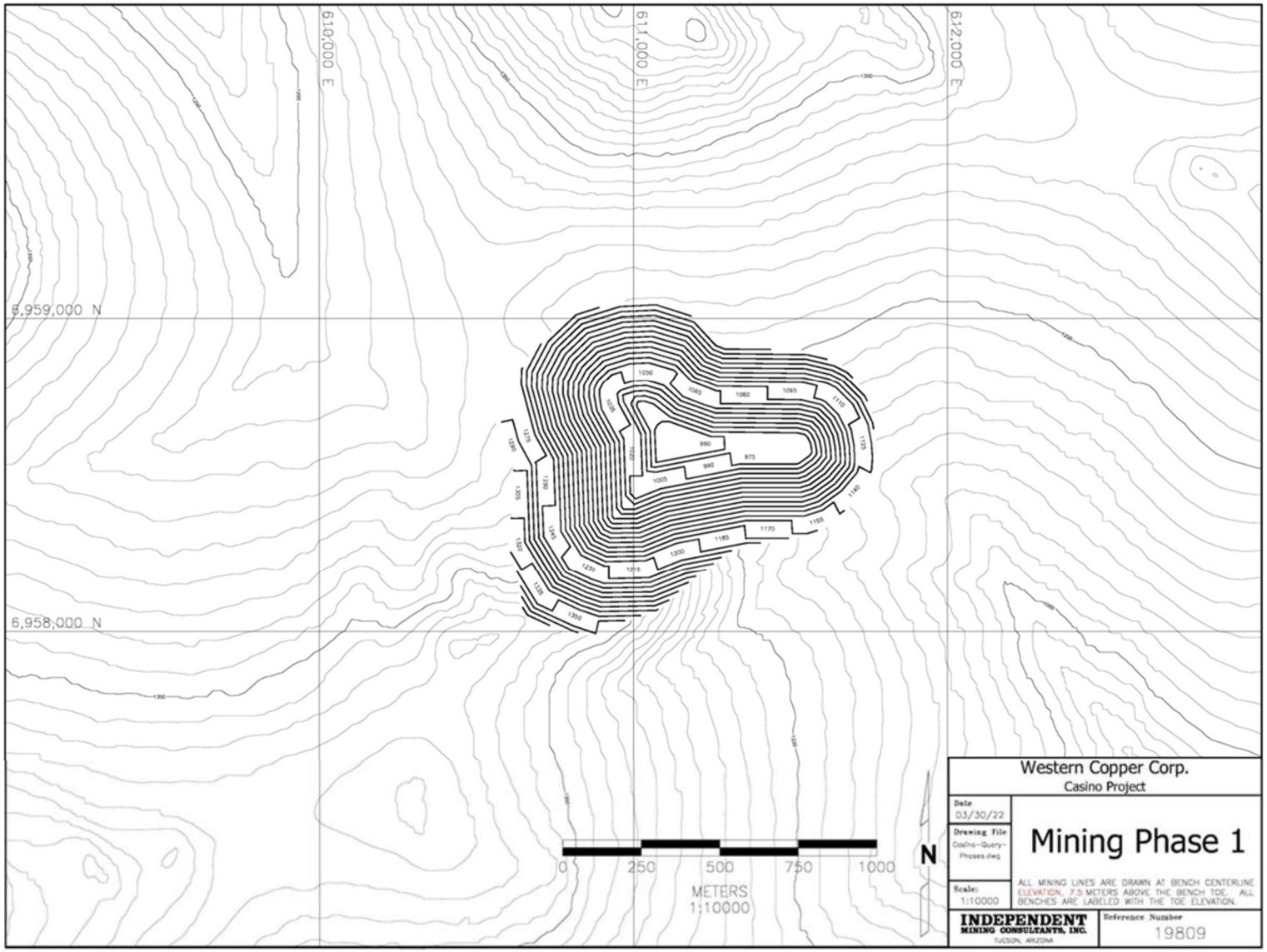


Figure 16-3: Mining Phase 1 (IMC, 2022)

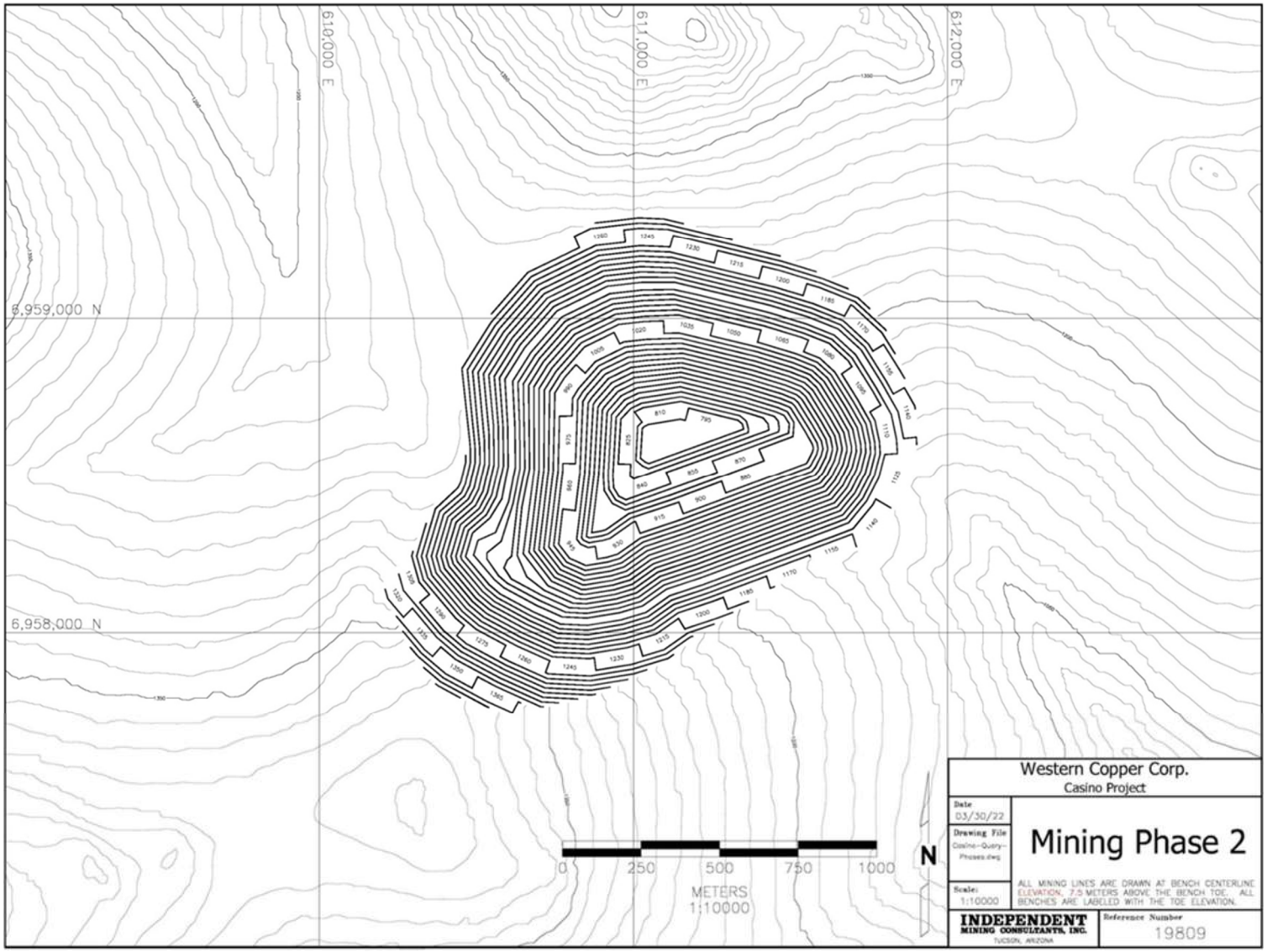


Figure 16-4: Mining Phase 2 (IMC, 2022)

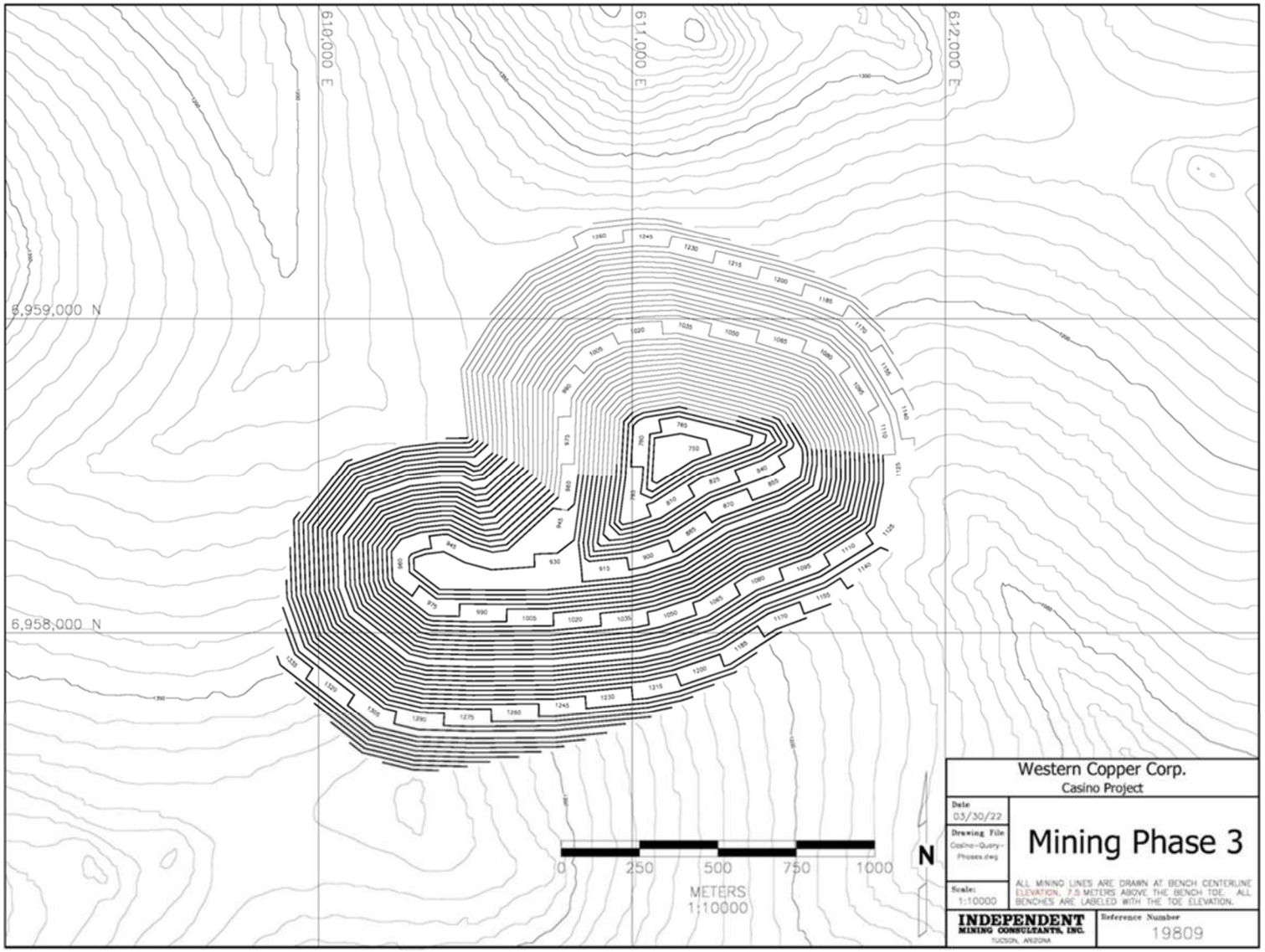


Figure 16-5: Mining Phase 3 (IMC, 2022)

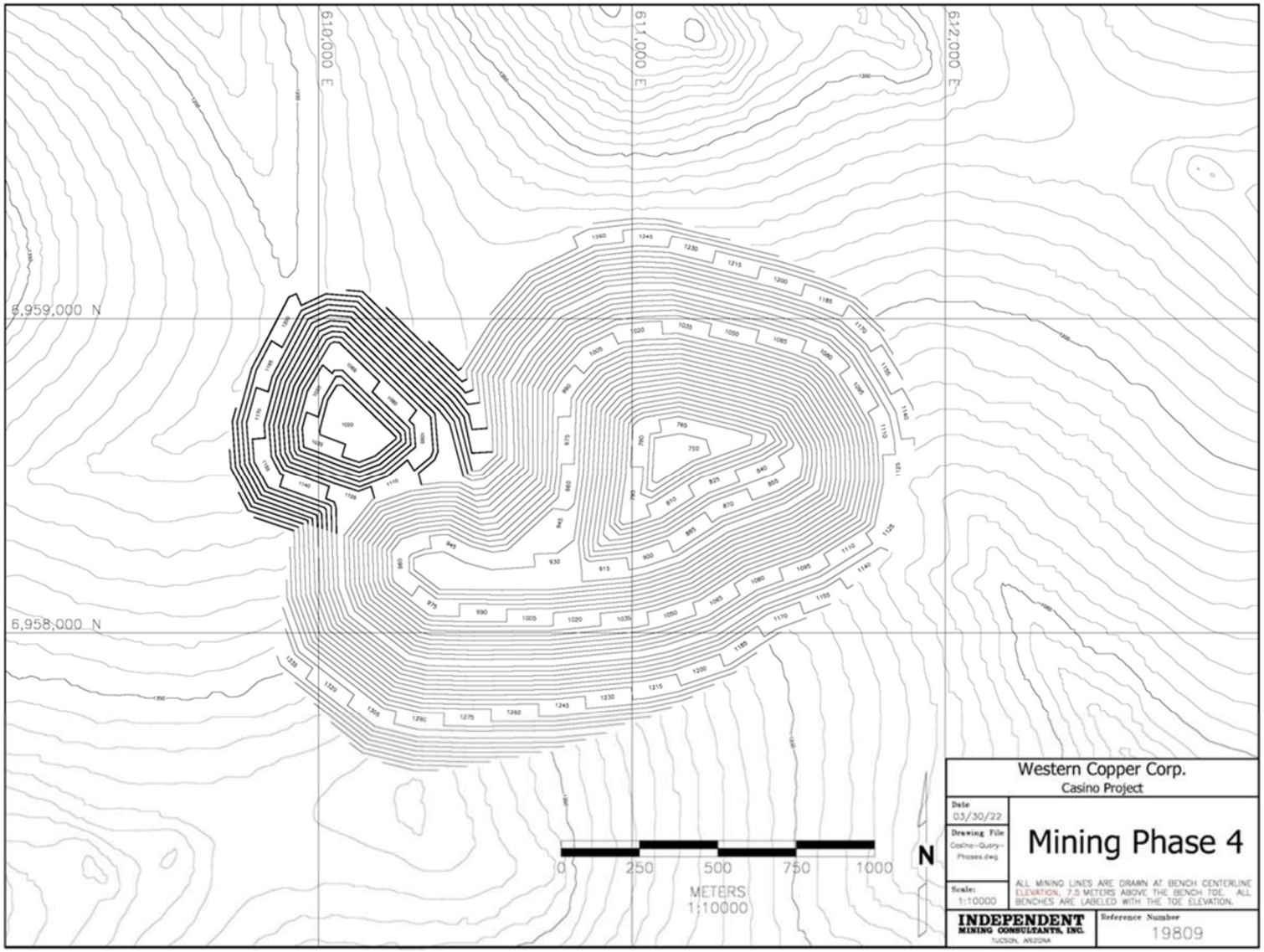


Figure 16-6: Mining Phase 4 (IMC, 2022)

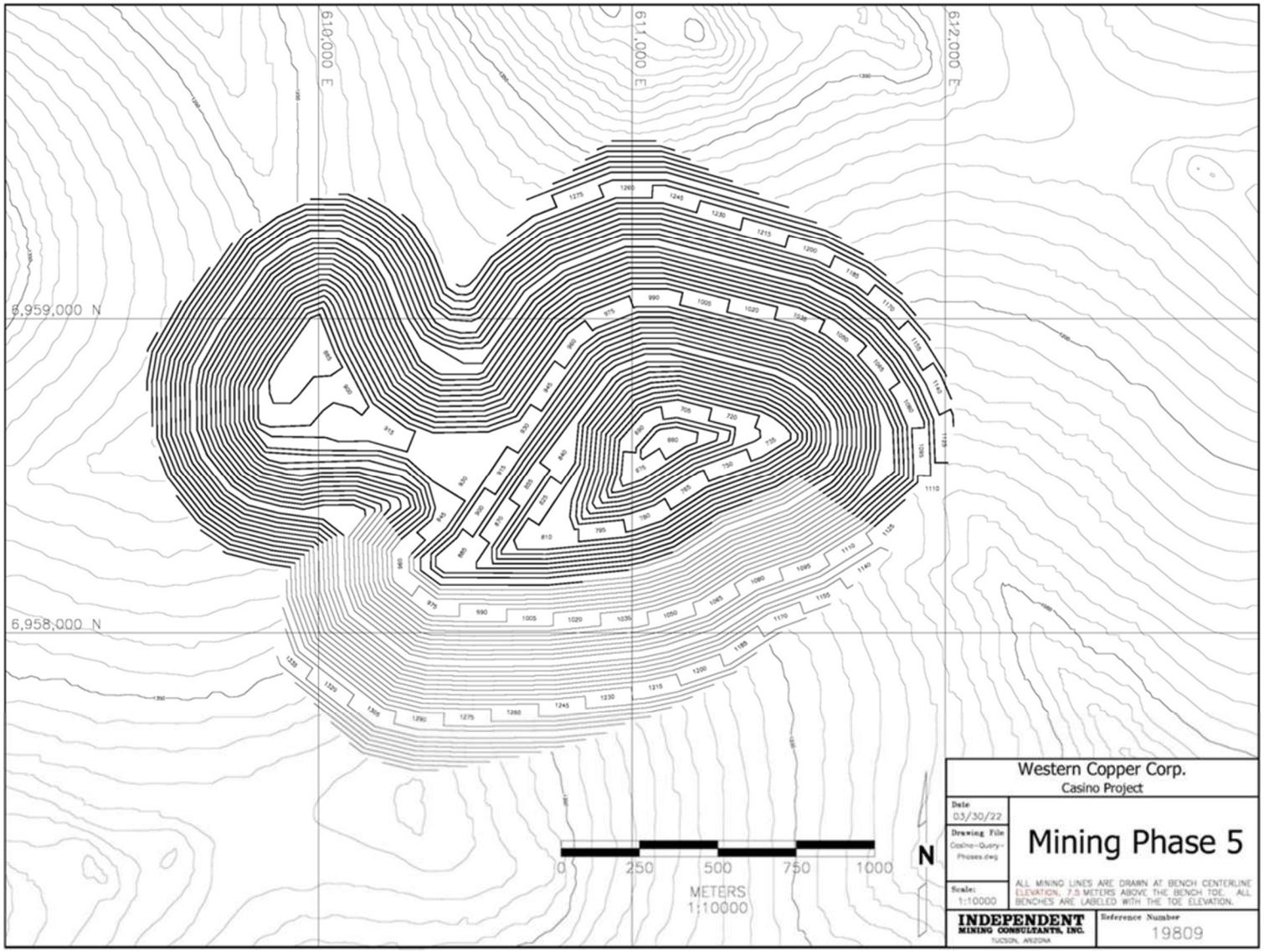


Figure 16-7: Mining Phase 5 (IMC, 2022)

16.4 MINE PRODUCTION SCHEDULE

The economic and recovery parameters used for mine design and scheduling are presented in Table 15-2 and Section 15.2 of this report.

The Bond Work Index and mill throughput rate have been assigned to model blocks based on rock type, oxidation zone, and alteration based on the recommendations in the FLSmidth 2012 report (FLSmidth, 2012). Generally, argillic altered rocks have the lowest work index, potassic altered rocks the highest work index, and rock with phyllic alteration tend to be in the middle. The production schedule has been developed based on plant hours, so throughput varies by year. It was reported to IMC that all necessary efficiency factors were incorporated in throughput rates; therefore, IMC has based the schedule on 8,760 plant hours per year.

The top section of Table 16-2 shows the proposed plant production schedule. Total mill ore is 1.22 billion tonnes at 0.189% copper, 0.217 g/t gold, 0.0213% moly, and 1.66 g/t silver. The average NSR value of this ore is $25.38 per tonne. Year 1 is shown by quarters and the rest of the schedule is full years. For Years 2 through 26, full production years, ore throughput varies from a low of 45.0 million tonnes in Year 19 to a high of 47.1 million tonnes in Year 2. The table also shows the average Bond Work Index (14.4) and throughput rate (0.19105 hours per ktonne or 5,234 tonnes per hour). The hours per ktonne units are unconventional, but a parameter that could be weight averaged by tonnes is required for the calculations. Copper recovery was also incorporated into the model on a block-by-block basis, based on total and soluble copper grades. The average recovered copper grade is 0.164%, indicating an average copper recovery of 86.4%.

The table also shows the various components of the mill ore. Direct feed ore is ore that is scheduled to be processed the same year it is mined. This amounts to 913.9 million tonnes at 0.206% total copper, 0.230 g/t gold, 0.0240% moly and 1.77 g/t silver. This is about 75% of total ore. The average NSR value of this ore is $27.77 per tonne. Note that Year 1 ore production is 34.6 million tonnes, about 73% of capacity in terms of plant hours and is made up of ore mined during preproduction and Year 1.

The SOX ore in mining phase 1 is stockpiled and processed during Years 4 through 13 at the rate of 3.60 million tonnes per year. This is done to maintain the ratio of weak soluble copper to total copper at relatively low levels by year. This ore amounts to 35.3 million tonnes at 0.275% total copper, 0.098% weak soluble copper, 0.500 g/t gold, 0.0254% moly, and 2.31 g/t silver. The NSR value for this ore is $37.14 per tonne.

The operating schedule also results in a significant amount of low-grade ore that is stockpiled and processed at the end of the mine life during Years 21 through 27. This amounts to 267.8 million tonnes at 0.120% total copper, 0.136 g/t gold, 0.0116 moly, and 1.19 g/t silver. The low grade represents material between an NSR cut-off of $12.50 per tonne and the operating cut-off for direct feed ore for the year. Additional low-grade resource is available if the low grade NSR cut-off is reduced. The $12.50/t cut-off gets to the maximum mill ore of 1.2 billion, the approximate limit for the current TMF design.

The reclaim schedules for both the SOX ore and low grade are on a last-in-first-out (LIFO) basis, consistent with stockpiles build up in lifts and reclaimed in reverse order.

Based on the schedule the commercial life of the project is 27 years after an approximate 3-year preproduction period.

Table 16-3 shows the mine production schedule. The upper section of the table shows the direct feed ore. 5.71 million tonnes of this is mined during preproduction and stockpiled to be part of the Year 1 plant feed.

As discussed in Section 15.2, an NSR value was calculated for each block to classify blocks into ore and waste. For the mine production schedule the NSR cut-off for direct feed ore varies by year to balance mine and plant capacities. It

ranges from a high of $25/t for preproduction and Year 1 Q1 feed and declines through the mine life down to the internal cut-off of $6.11/t for the last years of the mine life.

The second section of the table shows the SOX mill ore from phase 1 that is stockpiled and processed during Year 4 through 13. This is at an NSR cut-off grade of $20 per tonne to limit the total amount to about 36 million tonnes. Lower grade SOX goes to the low-grade stockpile. The third section of the table shows low grade ore produced by year. This is material with an NSR cut-off between $12.50 per tonne and the operating cut-off for the year. As previously discussed, this cut-off grade limited the material to what fits in the embankment.

The bottom of the table also shows the schedule of mineral reserve mined from the leach cap zone by year. This will be processed by crushing, and heap leaching. Leach ore is defined as leach cap material with an NSR above $6.61/t with leach economics and total copper less than 0.1%. This amounts to 209.6 million tonnes at 0.265 g/t gold, 1.95 g/t silver, and 0.036% total copper.

The bottom of the table also summarizes tonnages. It can be seen that life of mine total material from the pit is 2.04 billion tonnes. Preproduction is 75.0 million tonnes staged over three years, the same as the 2021 study. This is the approximate amount required to prepare mining phase 1 for commercial production. Year 1 total material is scheduled at about 95 million tonnes after which the peak material movement of 100 million tonnes per year is maintained through Year 14. Total waste is 611.3 million tonnes so the waste to ore ratio is about 0.44 if mill ore (including SOX), low grade, and leach ore are all counted as ore. Note that the 0.42 ratio is calculated based only on material mined during Years 1 through 22; it does not include preproduction or the processing of low-grade during Years 23 through 27.

The upper section of Table 16-4 shows a proposed stacking schedule for the leach resource. This is based on the ability to crush and stack 9,125 ktonnes per year (30,417 tpd for 300 days/year). Year -2 leach ore stacked on the pad is 3.90 million tonnes based on ramping up production between July and November of the year.

The second section of the table shows mine production of leach resource. The third and fourth sections show up to 9,125 ktonnes of mined ore as direct crusher feed and the excess going to stockpiles. Both are shown at average grades for the year. There are two stockpiles. Tonnages highlighted in green, about 33.3 million tonnes, are mined during preproduction and placed in a temporary stockpile within the final pit limits. This represents the highest-grade leach ore. Tonnages highlighted in blue go to the permanent stockpile east of the pit. The bottom of the table shows the stockpile reclaim on a last-in-first-out basis (LIFO). The temporary stockpile reclaim is completed early in Year 7, a couple of years before phase 4 stripping commences on that ground. This is a change from the 2021 study where some of this higher-grade material was stacked late in the mine life because it was buried by lower grade material. The temporary stockpile gets to a maximum capacity of 33.3 million tonnes and the permanent stockpile gets to a maximum capacity of 79.2 million tonnes.

Only measured and indicated mineral resource is included in the mine production schedule and converted to mineral reserves. The amount of potential mill resource in the pit that is inferred mineral resource is 34.1 million tonnes, an inconsequential amount.

The mine production schedule includes allowances for mining dilution and ore loss. IMC believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and mineralized material loss effects.

Table 16-2: Proposed Plant Production Schedule

	(Units)	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	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	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	TOTAL
Proposed Mill Schedule:																																
Ktonnes	(kt)	5,803	8,903	9,571	10,347	47,082	46,586	45,452	45,417	45,433	45,502	45,992	46,330	45,080	45,142	46,662	46,702	46,668	45,148	45,589	46,173	45,543	45,035	45,349	46,088	45,762	45,596	45,434	45,808	45,351	37,521	1,217,069
NSR Value	(C$/t)	37.89	40.00	40.17	44.29	41.87	38.01	30.88	27.64	28.22	29.61	29.82	29.20	29.80	30.82	28.85	27.99	23.93	21.45	20.92	20.44	22.38	23.76	21.52	21.75	18.08	14.66	14.73	16.42	16.11	16.89	25.38
Total Copper	(%)	0.270	0.317	0.337	0.380	0.334	0.271	0.269	0.228	0.211	0.205	0.203	0.188	0.218	0.263	0.192	0.193	0.182	0.203	0.158	0.148	0.152	0.162	0.132	0.143	0.118	0.108	0.113	0.114	0.136	0.160	0.189
Weak Soluble Copper	(%)	0.034	0.047	0.052	0.052	0.029	0.009	0.039	0.026	0.011	0.008	0.007	0.007	0.022	0.032	0.010	0.009	0.014	0.026	0.005	0.000	0.000	0.000	0.000	0.000	0.000	0.004	0.013	0.001	0.014	0.028	0.013
% Cuw/Cut	(%)	12.6%	15.0%	15.5%	13.6%	8.7%	3.2%	14.5%	11.2%	5.2%	4.1%	3.6%	3.8%	10.2%	12.3%	5.2%	4.4%	7.8%	12.7%	3.2%	0.1%	0.0%	0.0%	0.0%	0.0%	0.0%	3.6%	11.3%	1.0%	10.2%	17.6%	6.8%
Gold	(g/t)	0.454	0.438	0.398	0.403	0.373	0.346	0.285	0.257	0.263	0.261	0.253	0.241	0.256	0.243	0.246	0.228	0.172	0.166	0.187	0.178	0.186	0.182	0.155	0.152	0.138	0.122	0.139	0.144	0.144	0.151	0.217
Molybdenum	(%)	0.0144	0.0167	0.0222	0.0299	0.0312	0.0294	0.0173	0.0162	0.0182	0.0254	0.0290	0.0327	0.0306	0.0284	0.0306	0.0292	0.0247	0.0119	0.0134	0.0134	0.0187	0.0240	0.0287	0.0262	0.0200	0.0117	0.0107	0.0124	0.0086	0.0080	0.0213
Silver	(g/t)	2.70	2.84	2.44	2.07	2.00	2.00	1.85	1.73	1.74	2.21	1.68	1.74	1.99	1.97	1.80	1.59	1.79	1.56	1.53	1.66	2.27	1.56	1.17	1.39	1.27	1.15	1.11	1.25	1.27	1.12	1.66
Recovered Copper	(%)	0.221	0.252	0.268	0.308	0.285	0.242	0.215	0.188	0.185	0.181	0.180	0.168	0.183	0.217	0.169	0.171	0.155	0.166	0.142	0.136	0.140	0.149	0.122	0.132	0.108	0.096	0.093	0.104	0.113	0.124	0.164
Bond Work Index	(Kwh/t)	14.2	13.9	13.8	13.8	14.0	14.2	14.5	14.5	14.5	14.5	14.3	14.3	14.6	14.6	14.1	14.1	14.1	14.6	14.5	14.3	14.5	14.6	14.5	14.3	14.4	14.4	14.5	14.4	14.5	14.3	14.4
Hours Per Ktonne	(hr/kt)	0.18866	0.18535	0.18266	0.18362	0.18606	0.18802	0.19272	0.19287	0.19280	0.19252	0.19047	0.18908	0.19432	0.19406	0.18773	0.18758	0.18771	0.19403	0.19215	0.18972	0.19234	0.19451	0.19317	0.19007	0.19141	0.19212	0.19281	0.19124	0.19315	0.18998	0.19105
Mill Hours	(hours)	1,095	1,650	1,748	1,900	8,760	8,759	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,759	8,760	8,760	8,760	8,760	7,128	232,518
Copper Recovery	(%)	81.8%	79.7%	79.5%	81.0%	85.3%	89.5%	79.8%	82.4%	87.8%	88.4%	88.8%	89.2%	84.0%	82.5%	87.9%	88.5%	85.3%	81.6%	89.7%	91.9%	92.1%	92.0%	92.4%	92.0%	92.1%	89.2%	82.6%	91.4%	83.5%	77.1%	86.4%
Gold Recovery	(%)	68.4%	68.6%	68.6%	68.6%	67.7%	66.7%	68.1%	67.4%	66.4%	66.4%	66.5%	66.6%	67.9%	68.8%	67.0%	66.9%	66.5%	68.1%	66.9%	66.0%	66.0%	66.0%	66.0%	66.0%	66.0%	66.5%	67.7%	66.3%	67.2%	68.3%	67.1%
Moly Recovery	(%)	56.8%	54.8%	54.5%	55.8%	62.1%	72.7%	61.5%	64.9%	74.7%	76.1%	76.5%	76.2%	63.5%	54.3%	72.6%	73.0%	76.1%	60.4%	69.1%	78.4%	78.6%	78.6%	78.6%	78.6%	78.6%	74.9%	64.6%	78.2%	68.4%	57.5%	71.2%
Silver Recovery	(%)	57.9%	58.6%	59.0%	58.3%	55.1%	52.3%	57.0%	54.2%	51.1%	50.7%	51.1%	51.5%	56.6%	59.1%	52.8%	52.8%	51.5%	56.4%	53.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	51.3%	55.7%	50.6%	53.3%	57.4%	53.1%
Direct Feed Material:																																
Ktonnes	(kt)	5,803	8,903	9,571	10,347	47,082	46,586	41,864	41,817	41,833	41,902	42,392	42,735	41,485	41,547	43,067	43,730	46,668	45,148	45,589	46,173	45,543	45,035	45,349	34,548	9,176						913,893
NSR Value	(C$/t)	37.89	40.00	40.17	44.29	41.87	38.01	30.47	26.48	27.33	29.14	29.37	28.66	29.07	30.18	28.07	27.48	23.93	21.45	20.92	20.44	22.38	23.76	21.52	23.78	28.37						27.77
Total Copper	(%)	0.270	0.317	0.337	0.380	0.334	0.271	0.256	0.209	0.197	0.200	0.200	0.185	0.220	0.268	0.192	0.194	0.182	0.203	0.158	0.148	0.152	0.162	0.132	0.158	0.190						0.206
Weak Soluble Copper	(%)	0.034	0.047	0.052	0.052	0.029	0.009	0.032	0.015	0.001	0.000	0.000	0.001	0.018	0.029	0.005	0.004	0.014	0.026	0.005	0.000	0.000	0.000	0.000	0.000	0.000						0.011
Gold	(g/t)	0.454	0.438	0.398	0.403	0.373	0.346	0.284	0.247	0.250	0.242	0.233	0.218	0.224	0.211	0.214	0.204	0.172	0.166	0.187	0.178	0.186	0.182	0.155	0.165	0.203						0.230
Molybdenum	(%)	0.0144	0.0167	0.0222	0.0299	0.0312	0.0294	0.0172	0.0150	0.0171	0.0250	0.0290	0.0331	0.0313	0.0289	0.0312	0.0298	0.0247	0.0119	0.0134	0.0134	0.0187	0.0240	0.0287	0.0286	0.0335						0.0240
Silver	(g/t)	2.70	2.84	2.44	2.07	2.00	2.00	1.89	1.74	1.74	2.22	1.63	1.67	1.90	1.88	1.70	1.54	1.79	1.56	1.53	1.66	2.27	1.56	1.17	1.54	1.21						1.77
Recovered Copper	(%)	0.221	0.252	0.268	0.308	0.285	0.242	0.209	0.180	0.182	0.184	0.184	0.171	0.189	0.225	0.173	0.175	0.155	0.166	0.142	0.136	0.140	0.149	0.122	0.145	0.175						0.181
Bond Work Index	(Kwh/t)	14.2	13.9	13.8	13.8	14.0	14.2	14.6	14.6	14.6	14.6	14.4	14.3	14.7	14.7	14.1	14.1	14.1	14.6	14.5	14.3	14.5	14.6	14.5	14.3	14.2						14.4
Hours Per Ktonne	(hr/kt)	0.18866	0.18535	0.18266	0.18362	0.18606	0.18802	0.19354	0.19382	0.19374	0.19343	0.19117	0.18964	0.19533	0.19504	0.18815	0.18795	0.18771	0.19403	0.19215	0.18972	0.19234	0.19451	0.19317	0.18985	0.18860						0.19113
Mill Hours	(hours)	1,095	1,650	1,748	1,900	8,760	8,759	8,102	8,105	8,105	8,105	8,104	8,104	8,103	8,103	8,103	8,219	8,760	8,760	8,760	8,760	8,760	8,760	8,760	6,559	1,731						174,676
SOX Material (LIFO):																																
Ktonnes	(kt)							3,588	3,600	3,600	3,600	3,600	3,595	3,595	3,595	3,595	2,972															35,340
NSR Value	(C$/t)							35.63	41.21	38.45	35.03	35.11	35.60	38.21	38.21	38.21	35.48															37.14
Total Copper	(%)							0.417	0.456	0.367	0.257	0.236	0.216	0.199	0.199	0.199	0.192															0.275
Weak Soluble Copper	(%)							0.121	0.153	0.132	0.105	0.092	0.080	0.075	0.075	0.075	0.071															0.098
Gold	(g/t)							0.294	0.375	0.421	0.478	0.492	0.520	0.620	0.620	0.620	0.576															0.500
Molybdenum	(%)							0.0187	0.0305	0.0303	0.0301	0.0294	0.0279	0.0223	0.0223	0.0223	0.0191															0.0254
Silver	(g/t)							1.41	1.67	1.84	2.05	2.30	2.57	2.98	2.98	2.98	2.33															2.31
Recovered Copper	(%)							0.278	0.284	0.221	0.143	0.136	0.128	0.117	0.117	0.117	0.114															0.166
Bond Work Index	(Kwh/t)							13.8	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7															13.7
Hours Per Ktonne	(hr/kt)							0.18327	0.18179	0.18187	0.18198	0.18220	0.18243	0.18266	0.18266	0.18266	0.18207															0.18236
Mill Hours	(hours)							658	654	655	655	656	656	657	657	657	541															6,445
Low Grade Mill Stock (LIFO):																																
Ktonnes	(kt)																								11,540	36,586	45,596	45,434	45,808	45,351	37,521	267,836
NSR Value	(C$/t)																								15.69	15.50	14.66	14.73	16.42	16.11	16.89	15.69
Total Copper	(%)																								0.100	0.099	0.108	0.113	0.114	0.136	0.160	0.120
Weak Soluble Copper	(%)																								0.000	0.000	0.004	0.013	0.001	0.014	0.028	0.009
Gold	(g/t)																								0.115	0.122	0.122	0.139	0.144	0.144	0.151	0.136
Molybdenum	(%)																								0.0189	0.0166	0.0117	0.0107	0.0124	0.0086	0.0080	0.0116
Silver	(g/t)																								0.96	1.28	1.15	1.11	1.25	1.27	1.12	1.19
Recovered Copper	(%)																								0.093	0.091	0.096	0.093	0.104	0.113	0.124	0.103
Bond Work Index	(Kwh/t)																								14.3	14.4	14.4	14.5	14.4	14.5	14.3	14.4
Hours Per Ktonne	(hr/kt)																								0.19071	0.19212	0.19212	0.19281	0.19124	0.19315	0.18998	0.19190
Mill Hours	(hours)																								2,201	7,029	8,760	8,760	8,760	8,760	7,128	51,397

Table 16-3: Mine Production Schedule

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	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	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	TOTAL
Direct Feed Material:																																			
NSR Cutoff Grade	(C$)	25.00	25.00	25.00	25.00	6.11	6.11	24.00	23.00	19.75	18.00	19.25	20.25	20.00	19.25	18.25	18.25	17.50	17.00	15.00	15.00	14.25	16.00	17.00	18.00	18.00	6.11	6.11	6.11						
Ktonnes	(kt)		668	5,042	3,208	6,105	9,254	10,347	47,082	46,586	41,864	41,817	41,833	41,902	42,392	42,735	41,485	41,547	43,067	43,730	46,668	45,148	45,589	46,173	45,543	45,035	45,349	34,548	9,176						913,893
NSR Value	(C$/t)		32.16	34.40	40.22	42.96	40.44	44.29	41.87	38.01	30.47	26.48	27.33	29.14	29.37	28.66	29.07	30.18	28.07	27.48	23.93	21.45	20.92	20.44	22.38	23.76	21.52	23.78	28.37						27.77
Total Copper	(%)		0.236	0.238	0.295	0.353	0.341	0.380	0.334	0.271	0.256	0.209	0.197	0.200	0.200	0.185	0.220	0.268	0.192	0.194	0.182	0.203	0.158	0.148	0.152	0.162	0.132	0.158	0.190						0.206
Weak Soluble Copper	(%)		0.029	0.025	0.044	0.056	0.053	0.052	0.029	0.009	0.032	0.015	0.001	0.000	0.000	0.001	0.018	0.029	0.005	0.004	0.014	0.026	0.005	0.000	0.000	0.000	0.000	0.000	0.000						0.011
Gold	(g/t)		0.358	0.420	0.475	0.454	0.400	0.403	0.373	0.346	0.284	0.247	0.250	0.242	0.233	0.218	0.224	0.211	0.214	0.204	0.172	0.166	0.187	0.178	0.186	0.182	0.155	0.165	0.203						0.230
Molybdenum	(%)		0.0177	0.0127	0.0158	0.0182	0.0224	0.0299	0.0312	0.0294	0.0172	0.0150	0.0171	0.0250	0.0290	0.0331	0.0313	0.0289	0.0312	0.0298	0.0247	0.0119	0.0134	0.0134	0.0187	0.0240	0.0287	0.0286	0.0335						0.0240
Silver	(g/t)		1.93	2.20	3.07	3.17	2.45	2.07	2.00	2.00	1.89	1.74	1.74	2.22	1.63	1.67	1.90	1.88	1.70	1.54	1.79	1.56	1.53	1.66	2.27	1.56	1.17	1.54	1.21						1.77
Recovered Copper	(%)		0.194	0.199	0.236	0.279	0.271	0.308	0.285	0.242	0.209	0.180	0.182	0.184	0.184	0.171	0.189	0.225	0.173	0.175	0.155	0.166	0.142	0.136	0.140	0.149	0.122	0.145	0.175						0.181
Bond Work Index	(Kwh/t)		13.7	14.2	14.1	13.8	13.8	13.8	14.0	14.2	14.6	14.6	14.6	14.6	14.4	14.3	14.7	14.7	14.1	14.1	14.1	14.6	14.5	14.3	14.5	14.6	14.5	14.3	14.2						14.4
Hours Per Ktonne	(hr/kt)		0.18207	0.18925	0.18797	0.18408	0.18268	0.18362	0.18606	0.18802	0.19354	0.19382	0.19374	0.19343	0.19117	0.18964	0.19533	0.19504	0.18815	0.18795	0.18771	0.19403	0.19215	0.18972	0.19234	0.19451	0.19317	0.18985	0.18860						0.19113
Mill Hours	(hours)		122	954	603	1,124	1,690	1,900	8,760	8,759	8,102	8,105	8,105	8,105	8,104	8,104	8,103	8,103	8,103	8,219	8,760	8,760	8,760	8,760	8,760	8,760	8,760	6,559	1,731						174,676
SOX Stockpile:																																			
NSR Cutoff Grade	(C$)	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00																										
Ktonnes	(kt)	21	2,877	11,369	5,046	6,844	6,079	1,932	1,172																										35,340
NSR Value	(C$/t)	29.26	35.46	38.21	35.17	35.03	41.21	38.51	28.57																										37.14
Total Copper	(%)	0.189	0.192	0.199	0.219	0.257	0.456	0.478	0.300																										0.275
Weak Soluble Copper	(%)	0.087	0.071	0.075	0.081	0.105	0.153	0.146	0.066																										0.098
Gold	(g/t)	0.474	0.576	0.620	0.503	0.478	0.375	0.301	0.248																										0.500
Molybdenum	(%)	0.0141	0.0191	0.0223	0.0288	0.0301	0.0305	0.0200	0.0116																										0.0254
Silver	(g/t)	2.2000	2.3100	2.98	2.50	2.05	1.67	1.24	1.58																										2.31
Recovered Copper	(%)	0.096	0.114	0.117	0.130	0.143	0.284	0.311	0.220																										0.166
Bond Work Index	(Kwh/t)	13.7	13.7	13.7	13.7	13.7	13.7	13.7	14.0																										13.7
Hours Per Ktonne	(hr/kt)	0.18207	0.18205	0.18266	0.18239	0.18198	0.18179	0.18229	0.18549																										0.18236
Mill Hours	(hours)	4	524	2,077	920	1,245	1,105	352	217																										6,445
Low Grade:																																			
NSR Cutoff Grade	(C$)	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50									
Ktonnes	(kt)	0	9	876	1,063	665	446	1,963	11,868	6,610	15,642	34,451	25,055	17,441	8,774	8,721	12,377	8,668	10,492	3,830	3,142	6,021	20,806	28,134	23,554	17,228									267,836
NSR Value	(C$/t)	0.00	24.48	19.79	18.94	17.46	17.65	19.98	18.31	15.89	15.32	16.02	16.56	16.60	16.21	15.51	15.07	15.07	14.55	13.72	13.89	13.40	14.45	15.11	15.57	15.69									15.69
Total Copper	(%)	0.000	0.183	0.162	0.164	0.136	0.184	0.206	0.196	0.124	0.141	0.139	0.121	0.114	0.114	0.091	0.115	0.131	0.099	0.109	0.129	0.131	0.108	0.103	0.098	0.100									0.120
Weak Soluble Copper	(%)	0.000	0.029	0.034	0.048	0.053	0.065	0.035	0.032	0.011	0.028	0.016	0.003	0.000	0.000	0.003	0.017	0.019	0.002	0.006	0.030	0.024	0.003	0.000	0.000	0.000									0.009
Gold	(g/t)	0.000	0.272	0.236	0.220	0.208	0.165	0.158	0.141	0.168	0.136	0.143	0.147	0.138	0.136	0.176	0.153	0.124	0.135	0.114	0.125	0.110	0.125	0.121	0.124	0.115									0.136
Molybdenum	(%)	0.0000	0.0148	0.0054	0.0093	0.0184	0.0153	0.0086	0.0059	0.0059	0.0100	0.0080	0.0107	0.0143	0.0127	0.0099	0.0106	0.0121	0.0117	0.0088	0.0092	0.0080	0.0103	0.0140	0.0168	0.0189									0.0116
Silver	(g/t)	0.0000	1.5800	1.39	1.63	1.61	1.20	1.07	0.94	1.10	1.22	1.24	1.37	1.26	1.13	1.04	1.12	1.20	1.08	1.01	1.27	1.04	1.10	1.21	1.38	0.96									1.19
Recovered Copper	(%)	0.000	0.145	0.120	0.108	0.077	0.112	0.160	0.154	0.106	0.106	0.115	0.109	0.105	0.105	0.082	0.093	0.104	0.088	0.097	0.091	0.099	0.097	0.095	0.090	0.093									0.103
Bond Work Index	(Kwh/t)	0.000	13.700	13.8	13.9	13.7	13.7	14.3	14.3	14.2	14.4	14.5	14.5	14.4	14.1	14.4	14.9	14.9	14.1	14.3	14.1	14.6	14.4	14.4	14.5	14.3									14.4
Hours Per Ktonne	(hr/kt)	0.00000	0.18207	0.18271	0.18413	0.18226	0.18238	0.18993	0.19015	0.18889	0.19187	0.19337	0.19257	0.19168	0.18785	0.19149	0.19700	0.19792	0.18710	0.18889	0.18771	0.19341	0.19210	0.19189	0.19253	0.19071									0.19190
Mill Hours	(hours)	0.0000	1.6386	160	196	121	81	373	2,257	1,249	3,001	6,662	4,825	3,343	1,648	1,670	2,438	1,716	1,963	723	590	1,165	3,997	5,399	4,535	3,286									51,397
Leach Material:																																			
NSR Cutoff Grade	(C$)	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61													
Ktonnes	(kt)	6,058	14,872	25,359	7,388	5,676	3,034	5,351	30,021	25,707	13,056	450	559	5,038	10,838	11,324	5,900	21,983	206	6,255	7,084	3,478													209,637
NSR Value	(C$/t)	14.12	21.90	20.67	19.79	15.01	11.13	13.88	12.82	11.76	9.30	9.35	9.07	10.51	11.44	11.50	9.77	10.16	9.06	8.08	8.16	9.36													13.47
Gold	(g/t)	0.285	0.437	0.409	0.392	0.294	0.213	0.277	0.258	0.230	0.178	0.170	0.183	0.208	0.227	0.227	0.189	0.194	0.158	0.147	0.148	0.166													0.265
Silver	(g/t)	1.41	2.73	2.71	2.57	1.84	1.42	1.42	1.580	1.860	1.320	1.120	1.510	2.590	2.430	2.070	1.480	1.580	2.340	1.560	1.630	1.960													1.95
Total Copper	(%)	0.022	0.043	0.049	0.046	0.047	0.051	0.031	0.019	0.033	0.041	0.075	0.006	0.006	0.011	0.019	0.034	0.045	0.075	0.054	0.056	0.076													0.036
Weak Soluble Copper	(%)	0.005	0.010	0.015	0.019	0.020	0.016	0.008	0.005	0.010	0.012	0.019	0.002	0.002	0.005	0.007	0.006	0.008	0.029	0.017	0.016	0.014													0.010
Molybdenum	(%)	0.0064	0.0122	0.0171	0.0256	0.0269	0.0221	0.0042	0.0063	0.0105	0.0094	0.0039	0.0004	0.0111	0.0221	0.0215	0.0269	0.0253	0.0045	0.0077	0.0078	0.0053													0.0145
Tonnage Summary:																																			
Direct Feed Material:	(kt)	0	668	5,042	3,208	6,105	9,254	10,347	47,082	46,586	41,864	41,817	41,833	41,902	42,392	42,735	41,485	41,547	43,067	43,730	46,668	45,148	45,589	46,173	45,543	45,035	45,349	34,548	9,176						913,893
SOX Stockpile	(kt)	21	2,877	11,369	5,046	6,844	6,079	1,932	1,172	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0						35,340
Low Grade	(kt)	0	9	876	1,063	665	446	1,963	11,868	6,610	15,642	34,451	25,055	17,441	8,774	8,721	12,377	8,668	10,492	3,830	3,142	6,021	20,806	28,134	23,554	17,228	0	0	0						267,836
Leach Material:	(kt)	6,058	14,872	25,359	7,388	5,676	3,034	5,351	30,021	25,707	13,056	450	559	5,038	10,838	11,324	5,900	21,983	206	6,255	7,084	3,478	0	0	0	0	0	0	0						209,637
Total Material	(kt)	8,000	20,000	47,000	20,000	25,000	25,000	25,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	99,202	96,896	98,016	96,763	89,669	82,951	54,428	39,404	10,638					2,037,967
Waste Material	(kt)	1,921	1,574	4,354	3,295	5,710	6,187	5,407	9,857	21,097	29,438	23,282	32,553	35,619	37,996	37,220	40,238	27,802	46,235	46,185	42,308	42,249	31,621	22,456	20,572	20,688	9,079	4,856	1,462						611,261
Stockpile Rehandle (Mill)	(kt)	0	0	0	2,595	2,798	317	0	0	0	3,588	3,600	3,600	3,600	3,600	3,595	3,595	3,595	3,595	2,972	0	0	0	0	0	0	0	11,540	36,586	45,596	45,434	45,808	45,351	37,521	308,886
Waste Ratio (Year 1 - 22)	(none)				0.20	0.30	0.33	0.28	0.11	0.27	0.42	0.30	0.48	0.55	0.61	0.59	0.67	0.39	0.86	0.86	0.74	0.77	0.48	0.30	0.30	0.33	0.20	0.14	0.16						0.44

Table 16-4: Production Schedule for Leach Material

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	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	Year 21	Year 22	TOTAL
Leach Pad Stacking Schedule:																														
NSR Cutoff Grade	(C$)	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	
Ktonnes	(kt)	0	3,900	9,125	1,659	2,488	2,489	2,489	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	4,986	209,636
NSR Value	(C$/t)	0.00	21.90	20.67	19.79	17.28	14.96	16.61	16.26	15.67	14.28	21.11	16.82	10.51	11.44	11.50	10.37	10.16	10.14	8.73	8.68	9.61	11.76	11.76	12.67	12.82	12.96	14.41	19.79	13.47
Gold	(g/t)	0.000	0.437	0.409	0.392	0.340	0.292	0.330	0.324	0.308	0.279	0.420	0.338	0.207	0.227	0.227	0.202	0.194	0.193	0.162	0.160	0.180	0.230	0.230	0.254	0.258	0.261	0.283	0.392	0.265
Silver	(g/t)	0.00	2.73	2.71	2.57	2.19	1.94	1.94	2.08	2.23	1.93	2.65	1.93	2.03	2.43	2.07	1.73	1.58	1.60	1.57	1.67	1.63	1.86	1.86	1.62	1.58	1.56	1.71	2.57	1.95
Total Copper	(%)	0.000	0.043	0.049	0.046	0.048	0.050	0.038	0.032	0.040	0.045	0.045	0.029	0.020	0.011	0.019	0.028	0.045	0.046	0.051	0.051	0.053	0.033	0.033	0.021	0.019	0.021	0.043	0.046	0.036
Weak Soluble Copper	(%)	0.000	0.010	0.015	0.019	0.018	0.016	0.011	0.009	0.012	0.013	0.011	0.007	0.006	0.005	0.007	0.006	0.008	0.008	0.014	0.014	0.013	0.010	0.010	0.006	0.005	0.005	0.016	0.019	0.010
Molybdenum	(%)	0.0000	0.0122	0.0171	0.0256	0.0230	0.0201	0.0094	0.0110	0.0134	0.0128	0.0125	0.0083	0.0100	0.0221	0.0215	0.0251	0.0253	0.0248	0.0132	0.0110	0.0080	0.0105	0.0105	0.0069	0.0063	0.0060	0.0194	0.0256	0.0145
As Produced From Mine:																														
NSR Cutoff Grade	(C$)	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61	6.61								
Ktonnes	(kt)	6,058	14,872	25,359	7,388	5,676	3,034	5,351	30,021	25,707	13,056	450	559	5,038	10,838	11,324	5,900	21,983	206	6,255	7,084	3,478								209,637
NSR Value	(C$/t)	14.12	21.90	20.67	19.79	15.01	11.13	13.88	12.82	11.76	9.30	9.35	9.07	10.51	11.44	11.50	9.77	10.16	9.06	8.08	8.16	9.36								13.47
Gold	(g/t)	0.285	0.437	0.409	0.392	0.294	0.213	0.277	0.258	0.230	0.178	0.170	0.183	0.208	0.227	0.227	0.189	0.194	0.158	0.147	0.148	0.166								0.265
Silver	(g/t)	1.41	2.73	2.71	2.57	1.84	1.42	1.42	1.58	1.86	1.32	1.12	1.51	2.59	2.43	2.07	1.48	1.58	2.34	1.56	1.63	1.96								1.95
Total Copper	(%)	0.022	0.043	0.049	0.046	0.047	0.051	0.031	0.019	0.033	0.041	0.075	0.006	0.006	0.011	0.019	0.034	0.045	0.075	0.054	0.056	0.076								0.036
Weak Soluble Copper	(%)	0.005	0.010	0.015	0.019	0.020	0.016	0.008	0.005	0.010	0.012	0.019	0.002	0.002	0.005	0.007	0.006	0.008	0.029	0.017	0.016	0.014								0.010
Molybdenum	(%)	0.0064	0.0122	0.0171	0.0256	0.0269	0.0221	0.0042	0.0063	0.0105	0.0094	0.0039	0.0004	0.0111	0.0221	0.0215	0.0269	0.0253	0.0045	0.0077	0.0078	0.0053								0.0145
Direct to Crusher:		0.00%	26.22%	35.98%	22.46%	26.22%	49.08%	27.83%	17.07%	19.94%	39.25%	100.00%	100.00%	100.00%	84.19%	80.58%	100.00%	41.51%	100.00%	100.00%	100.00%	100.00%								
Ktonnes	(kt)	0	3,900	9,125	1,659	1,488	1,489	1,489	5,125	5,125	5,125	450	559	5,038	9,125	9,125	5,900	9,125	206	6,255	7,084	3,478								90,870
NSR Value	(C$/t)	14.12	21.90	20.67	19.79	15.01	11.13	13.88	12.82	11.76	9.30	9.35	9.07	10.51	11.44	11.50	9.77	10.16	9.06	8.08	8.16	9.36								12.16
Gold	(g/t)	0.285	0.437	0.409	0.392	0.294	0.213	0.277	0.258	0.230	0.178	0.170	0.183	0.208	0.227	0.227	0.189	0.194	0.158	0.147	0.148	0.166								0.237
Silver	(g/t)	1.41	2.73	2.71	2.57	1.84	1.42	1.42	1.58	1.86	1.32	1.12	1.51	2.59	2.43	2.07	1.48	1.58	2.34	1.56	1.63	1.96								1.96
Total Copper	(%)	0.022	0.043	0.049	0.046	0.047	0.051	0.031	0.019	0.033	0.041	0.075	0.006	0.006	0.011	0.019	0.034	0.045	0.075	0.054	0.056	0.076								0.036
Weak Soluble Copper	(%)	0.005	0.010	0.015	0.019	0.020	0.016	0.008	0.005	0.010	0.012	0.019	0.002	0.002	0.005	0.007	0.006	0.008	0.029	0.017	0.016	0.014								0.010
Molybdenum	(%)	0.0064	0.0122	0.0171	0.0256	0.0269	0.0221	0.0042	0.0063	0.0105	0.0094	0.0039	0.0004	0.0111	0.0221	0.0215	0.0269	0.0253	0.0045	0.0077	0.0078	0.0053								0.0157
To Leach Stockpile:		100.00%	73.78%	64.02%	77.54%	73.78%	50.92%	72.17%	82.93%	80.06%	60.75%	0.00%	0.00%	0.00%	15.81%	19.42%	0.00%	58.49%	0.00%	0.00%	0.00%	0.00%								
Ktonnes	(kt)	6,058	10,972	16,234	5,729	4,188	1,545	3,862	24,896	20,582	7,931	0	0	0	1,713	2,199	0	12,858	0	0	0	0								118,767
NSR Value	(C$/t)	14.12	21.90	20.67	19.79	15.01	11.13	13.88	12.82	11.76	9.30	9.35	9.07	10.51	11.44	11.50	9.77	10.16	9.06	8.08	8.16	9.36								14.47
Gold	(g/t)	0.285	0.437	0.409	0.392	0.294	0.213	0.277	0.258	0.230	0.178	0.170	0.183	0.208	0.227	0.227	0.189	0.194	0.158	0.147	0.148	0.166								0.286
Silver	(g/t)	1.41	2.73	2.71	2.57	1.84	1.42	1.42	1.58	1.86	1.32	1.12	1.51	2.59	2.43	2.07	1.48	1.58	2.34	1.56	1.63	1.96								1.93
Total Copper	(%)	0.022	0.043	0.049	0.046	0.047	0.051	0.031	0.019	0.033	0.041	0.075	0.006	0.006	0.011	0.019	0.034	0.045	0.075	0.054	0.056	0.076								0.035
Weak Soluble Copper	(%)	0.005	0.010	0.015	0.019	0.020	0.016	0.008	0.005	0.010	0.012	0.019	0.002	0.002	0.005	0.007	0.006	0.008	0.029	0.017	0.016	0.014								0.010
Molybdenum	(%)	0.0064	0.0122	0.0171	0.0256	0.0269	0.0221	0.0042	0.0063	0.0105	0.0094	0.0039	0.0004	0.0111	0.0221	0.0215	0.0269	0.0253	0.0045	0.0077	0.0078	0.0053								0.0136
Cumulative Stockpile:																														
Temporary	(kt)	6,058	17,030	33,264	33,264	32,264	31,264	30,264	26,264	22,264	18,264	9,589	1,023	0																
Permanent	(kt)	0	0	0	5,729	9,917	11,462	15,324	40,220	60,802	68,733	68,733	68,733	65,669	67,382	69,581	66,356	79,214	70,295	67,425	65,384	59,737	50,612	41,487	32,362	23,237	14,112	4,987		
Stockpile Reclaim:																														
Ktonnes	(kt)					1,000	1,000	1,000	4,000	4,000	4,000	8,675	8,566	4,087			3,225		8,919	2,870	2,041	5,647	9,125	9,125	9,125	9,125	9,125	9,125	4,986	118,766
NSR Value	(C$/t)					20.67	20.67	20.67	20.67	20.67	20.67	21.73	17.33	10.51			11.48		10.16	10.16	10.47	9.76	11.76	11.76	12.67	12.82	12.96	14.41	19.79	14.47
Gold	(g/t)					0.409	0.409	0.409	0.409	0.409	0.409	0.433	0.348	0.205			0.227		0.194	0.194	0.203	0.188	0.230	0.230	0.254	0.258	0.261	0.283	0.392	0.286
Silver	(g/t)					2.71	2.71	2.71	2.71	2.71	2.71	2.73	1.95	1.34			2.18		1.58	1.58	1.83	1.42	1.86	1.86	1.62	1.58	1.56	1.71	2.57	1.93
Total Copper	(%)					0.049	0.049	0.049	0.049	0.049	0.049	0.044	0.031	0.036			0.016		0.045	0.045	0.033	0.039	0.033	0.033	0.021	0.019	0.021	0.043	0.046	0.035
Weak Soluble Copper	(%)					0.015	0.015	0.015	0.015	0.015	0.015	0.011	0.007	0.010			0.006		0.008	0.008	0.008	0.012	0.010	0.010	0.006	0.005	0.005	0.016	0.019	0.010
Molybdenum	(%)					0.0171	0.0171	0.0171	0.0171	0.0171	0.0171	0.0129	0.0088	0.0086			0.0217		0.0253	0.0253	0.0220	0.0096	0.0105	0.0105	0.0069	0.0063	0.0060	0.0194	0.0256	0.0136

Temporary Stockpile	
Permanent Stockpile	
Combination	

16.5 WASTE MANAGEMENT

Total waste in the mine plan amounts to 611.3 million tonnes. The waste material by material type is as follows:

- 58.5 million tonnes of overburden.
- 144.6 million tonnes of leach cap material.
- 33.2 million tonnes of supergene oxide material.
- 125.1 million tonnes of supergene sulphide material.
- 249.8 million tonnes of hypogene material.

The overburden is placed in the overburden stockpile in Canadian Creek, north of the pit (Figure 16-8). The remaining waste is disposed in the tailing management facility in three facilities for mine waste: 1) the North Waste area which contains 248.4 million tonnes, 2) the Divider Dam which contains 134.4 million tonnes, and 3) the West Waste storage area which contains 164.6 million tonnes. About 5 million tonnes of mine waste will be used in the Starter Dam for the TMF embankment. The material will be placed by trucks and dozers; the rising water and tailings level in the TMF facility will cover the material before the end of the mine life.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 209.6 million tonnes of spent, non-reactive material, assuming all the potential leach material is processed.
- A low-grade stockpile southeast of the pit that has the capacity for 161.8 million tonnes, and a low-grade stockpile west of the pit that contains 106.1 million tonnes, both which will be processed at the end of the mine life.
- There will also be supergene oxide (SOX) stockpile south of the pit to store mining phase 1 SOX ore. It will be reclaimed during mining Years 4 through 13. The maximum size of this facility is estimated at 35.3 million tonnes. The SOX stockpile and the leach pad overlap by a small amount, but the SOX stockpile will be reclaimed before the leach pad gets to its final limits.
- There will be two stockpiles for leach ore. Leach ore mined during preproduction, 33.3 million tonnes, will be stockpiled in a temporary stockpile west of mining phase 1, but within the final pit limits (Figure 16-9). This material will be reclaimed and processed early in Year 7 a couple of years before waste stripping commences in that area. A larger facility for leach ore storage is located east of the pit. This is expected to reach a maximum size of 79.2 million tonnes during Year 11 and will be reclaimed by the end of Year 21.

Figure 16-9 also shows a temporary stockpile for run-of-mine (ROM) mill ore mined during the preproduction period, about 5.7 million tonnes. This material will be reclaimed and processed early in Year 1. This stockpile is also located within the limits of the final pit.

The stockpiles are all constructed in lifts from the bottom up. The low-grade stockpile, leach stockpile, and SOX stockpile are designed with 30 m lifts at angle of repose with a 20 m setback between lifts to make the overall slope angle about 2H:1V. This is assumed to be adequate since these are not permanent facilities.

Figure 16-8 also shows a quarry near the Divider Dam that is mined during preproduction and Year 1 to provide material for the Starter Dam. This material is not incorporated in the Casino mine production schedule; the cost is incorporated into infrastructure capital costs.

Figure 16-8: Maximum Extent of Waste Storage Areas and Stockpiles (IMC, 2022)

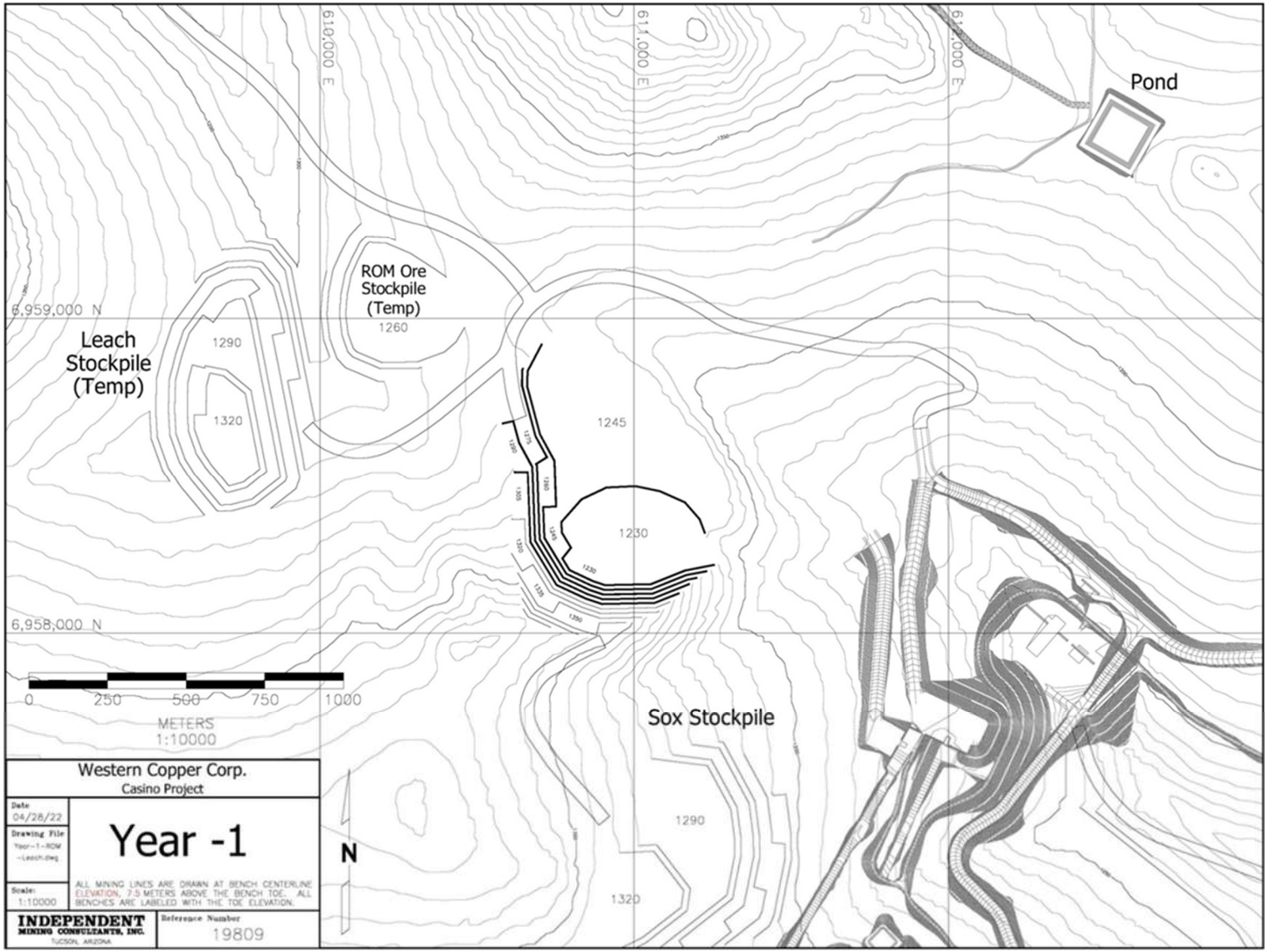


Figure 16-9: Year-1 Showing Temporary Leach and ROM Ore Stockpiles (IMC, 2022)

16.6 MINING EQUIPMENT

Mine equipment requirements were sized and estimated on a first principles basis, based on the mine production schedule, the mine work schedule, and estimated equipment shift productivity rates. The size and type of mining equipment is consistent with the size of the project, i.e., peak material movements of 100 Mt/y. The mine equipment estimate is based on owner operation and assumes a well-managed mining operation with a well-trained labor pool, and that all the equipment is new at the start of the operation.

Table 16-5 summarizes the major equipment requirements. The first column shows initial equipment for the first year of mine development (Year -3), the second column shows equipment required for the beginning of commercial production and the third column shows the peak fleet requirements. This represents the equipment required to perform the following duties:

- Develop access roads from the mine to the crushers, various stockpiles, and the waste storage areas.
- Mine and transport mill and leach ore to the crushers.
- Mine and transport material to various stockpiles as required.
- Reclaim stockpiled material and transport it to the crushers.
- Mine and transport waste to the overburden stockpile and the waste storage areas in the TMF.
- Maintain the haul roads and stockpiles and various truck dumping sites.

Table 16-5: Mining Equipment Requirements

Equipment Type	Capacity/ Power	Year -3	Year 1	Peak
P&H 320 XPC Drill	(314 mm)	0	3	4
Epiroc SmartROC D65	(178 mm)	1	2	2
P&H 4100XPC Cable Shovel	(67.6 cu m)	0	2	2
Komatsu PC8000-6 Hyd Shovel	(42 cu m)	0	1	1
Komatsu WA1200-6 Wheel Loader	(20 cu m)	1	1	2
Komatsu 980E Truck	(370 mt)	0	17	23
Komatsu HD1500 Truck	(144 mt)	6	4	8
Komatsu D475A Track Dozer	(664 kw)	0	3	3
Komatsu D375A Track Dozer	(455 kw)	2	3	3
Komatsu WD900 Wheel Dozer	(637 kw)	0	3	3
Komatsu GD825A Motor Grader	(209 kw)	1	3	3
Water Truck – 30,000 gal	(113,550 l)	1	3	3
Komatsu PC360LC-11 Excavator	(1.96 cu m)	1	2	2
Komatsu WA800-3 Wheel Loader	(11 cu m)	1	0	1
Komatsu HM 400 Art Truck	(40 mt)	2	0	2
Epiroc SmartROC T45	(114 mm)	1	0	1
TOTAL		17	47	63

17 RECOVERY METHODS

17.1 PROCESS DESCRIPTION

The Casino process plant will consist of two processing facilities, one for sulphide ore and one for oxide ore.

The sulphide ore processing facility will produce mineral concentrates of copper and molybdenum using conventional flotation technology. The copper concentrate will be dewatered and transported as a filtered cake by highway trucks. The molybdenum concentrate will be dewatered and packaged in super sacks for transport. Gold and silver contained in the sulphide ore will be recovered as a fraction of the copper concentrate.

17.2 SULPHIDE ORE PROCESS PLANT DESCRIPTION

17.2.1 Process Design Criteria and Major Equipment

Process design criteria were developed for the Sulphide facility based on a 120,000 t/d (43,800,000 t/y) plant design. The crushing circuit was designed to operate with an overall availability of 80%. The remainder of the sulphide facilities were designed to operate with an overall availability of 93%. The equipment was sized using these criteria. Table 17-1 is a summary of the main components of the sulphide process design criteria used for the study.

Table 17-2 is a summary of the major process equipment selected for the study.

Table 17-1: Sulphide Process Design Criteria

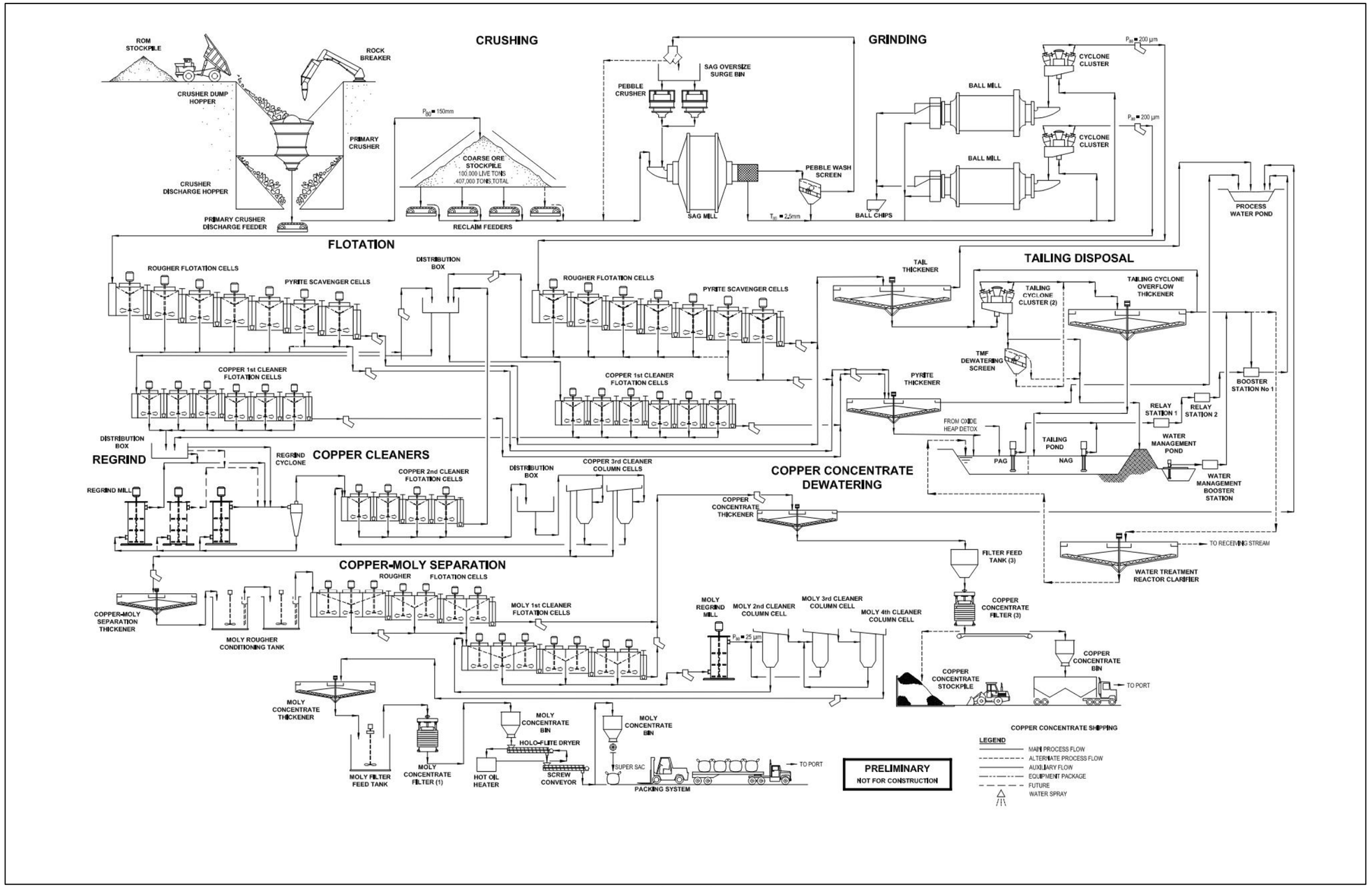
Description	Unit	Value
General		
Type of Deposit	-	Supergene Copper Oxide Zone Supergene Sulphide Zone Hypogene Zone
Plant Feed Grade (LOM)		
Copper (Cu)	%	0.189
Moly	%	0.023
Gold	g/t	0.217
Silver (Ag)	g/t	1.66
ROM Ore Characteristics		
Maximum mine-run ore size	mm	900
Ore Bulk Density, mine-run sulphide	t/m^3	1.68
Moisture Content	%	3
Crushing Work Index, SGS Method (CI)	-	11.8
Mill Feed Characteristics		
Specific Gravity	-	2.7
Moisture Content	%	3
Abrasion Index (Average)	-	0.265
Bond Rod Mill Work Index	kWh/t	9.9
Bond Ball Mill Work Index	kWh/t	14.5
Operating Schedule		
Shift/Day	-	2
Hours/Shift	H	12
Hours/Day	H	24
Days/Year	D	365
Plant Availability/Utilization		
Overall Plant Feed	t/y	43,800,000
Overall Plant Feed	t/d	120,000
Crusher Plant Availability	%	80
Grinding and Flotation Plant Availability	%	93
Crushing Rate	t/h	6,667
Grinding Rate	t/h	5,376
Flotation Rate	t/h	5,376
Design Factors		
Coarse ore storage thru grinding and final plant tailing	-	1.15
Bulk rougher flotation thru regrind and cleaner flotation	-	1.35
Moly circuit – rougher and 1st cleaner		1.35
Moly circuit –2nd, 3rd, 4th cleaners and conc handling	-	1.50
Head Grades (Design)		
Copper	%	0.29
Molybdenum	%	0.024
Gold	g/t	0.36
Silver	g/t	2.05
Copper Recovery Supergene Sulphide and Mixed Supergene Ores		
Copper recovery to copper concentrate	%	94
Gold recovery to copper concentrate	%	69
Silver recovery to copper concentrate	g/t	60
Molybdenum recovery to molybdenum concentrate	g/t	52

Description	Unit	Value
Copper Concentrate Grade		
Copper	%	28
Molybdenum	%	56
Copper Recovery Hypogene Ores		
Copper recovery to copper concentrate	%	92
Gold recovery to copper concentrate	%	66
Silver recovery to copper concentrate	g/t	50
Molybdenum recovery to molybdenum concentrate	g/t	79
Copper Concentrate Grade		
Copper	%	28
Molybdenum	%	56
Annual Concentrate Production (LOM)		
Copper	dry kt	264
Molybdenum	dry kt	12
Annual Concentrate Production (Design)		
Copper	dry kt	390
Molybdenum	dry kt	13

Table 17-2: Major Process Equipment

Item	Number	Description	Key Criteria (each)
Beneficiation			
Primary Crusher	1	1600 x 3000 TSU Top Service Gyratory Crusher	1,200 kW
SAG Mill	1	12.19 m x 8.84 (40 ft x 29 ft) (11.38 m ID x 7.99 m EGL)	29,000 kW
Pebble Crusher	2	Raptor XL 1300	970 kW
Ball Mill	2	8.5 m x 13.74 m (28 ft x 45 ft) (inside shell diameter x EGL)	22,000 kW
Cyclone Feed Pump	2	Horizontal Centrifugal, Hard Metal, 34x32-77	3000 kW
Flotation			
Copper Rougher	10	FLS nextSTEP, RT, 600 m^3	375 kW
Pyrite Scavenger	4	FLS nextSTEP, RT, 600 m^3	375 kW
Copper 1st Cleaner	12	FLS nextSTEP, RT, 100 m^3	132 kW
Copper 2nd Cleaner	4	FLS nextSTEP, RT, 100 m^3	132 kW
Copper 3rd Cleaner	2	Column Cell, 4.5 m diameter x 12.1 m high	75 kW
Copper-Moly Separation	7	Wemco 1+1, 8.5 m^3	30 kW
Moly 1st Cleaner	7	Wemco 1+1, 2.8 m^3	15 kW
Moly 2nd Cleaner	1	Column Cell, 1.75 m diameter x 8.6 m high	40 kW
Moly 3rd Cleaner	1	Column Cell, 1.5 m diameter x 7.5 m high	40 kW
Moly 4th Cleaner	1	Column Cell, 1.5 m diameter x 7.5 m high	40 kW
Cu. Regrind	2	Vertical Grinding, VTM 2250-WB	1,679 kW
Moly Regrind	1	SMD 18.5	19 kW
Dewatering and Filtration			
Cu-Moly Separation Thickener	1	High-rate thickener	45 m dia
Cu Concentrate Thickener	1	High-rate thickener	45 m dia
Mo Concentrate Thickener	1	High-rate thickener	6 m dia
Tailings Thickener	2	High-rate thickener	80 m dia
Tailing Cyclone Overflow Thickener	1	High-rate thickener	80 m dia
Pyrite Thickener	1	High-rate thickener	60 m dia
Copper Concentrate Filter	3	Tower Type - Pressure Filter, FLS Pneumapress M30-12	31.1 m^2 area
Moly Concentrate filter	1	Tower Type - Pressure Filter, FLS Pneumapress M19-2	3.3 m^2 area

Figure 17-1 is a simplified schematic of the overall process for the sulphide ore. This provides the basis for the process description that follows.



(Source: M3)

Figure 17-1: Simplified Sulphide Process Flowsheet

The following items summarise the process operations required to extract copper and molybdenum from the sulphide ore:

- Size reduction of the run-of-mine (ROM) material (900 mm max) to minus 200 mm.
- Stockpiling primary crushed ore and then reclaiming with feeders and a belt conveyor.
- Size reduction of the ore in a semi-autogenous (SAG) mill - ball mill grinding circuit with pebble crushing.
- Concentration and separation of the copper and molybdenum sulphide minerals by froth flotation to produce a bulk (copper/molybdenum) concentrate.
- Separation of the bulk concentrate into separate copper and molybdenum concentrates.
- Final copper concentrate will be thickened, filtered, and loaded in highway haul trucks for shipment.
- Final molybdenum concentrate will be thickened, filtered, dried, and packaged in bags for shipment.
- Concentration of the bulk flotation tailings in a pyrite flotation circuit. Pyrite flotation circuit tailings will have a low sulphide sulfur concentration.
- Subaqueous deposition of the pyrite flotation concentrate (PAG tailings) in the TMF.
- Flotation tailings will be thickened and transported by a gravity pipeline to a tailings impoundment area. The tailing will be cycloned with the underflow recovered as sand for tailing dam construction and overflow reporting to the tailing disposal impoundment site.
- Storing, preparing, and distributing reagents used in the sulphide ore process.
- Water from tailings and concentrate dewatering will be recycled for reuse in the process. Plant water stream types include process water, fresh water, potable water, and fire water

17.2.2 Crushing and Coarse Ore Stockpile

Run-of-Mine (ROM) sulphide ore will be trucked from the mine to the primary crusher and fed to the crusher via a dump pocket. The primary crusher will be a gyratory crusher, with an open side setting of 200 mm. The crushed ore will drop into a discharge bin equipped with an apron feeder. The apron feeder will discharge onto a belt conveyor that will discharge the primary crushed ore to a covered, conical ore stockpile.

Primary crushed ore will be stockpiled on the ground in a covered, conical ore stockpile. A reclaim tunnel will be installed beneath the stockpile. The stockpile will contain approximately 75,000 tonnes of “live” ore storage. Ore will be moved from the “dead” storage area to the “live” storage area by front-end loader or bulldozer.

Ore for the single grinding line will be withdrawn from the coarse ore reclaim stockpile by variable speed, apron feeders. The feeders will discharge to a conveyor belt which will provide new feed to the SAG mill in the primary grinding circuit.

17.2.3 Grinding and Classification

Ore will be ground to rougher flotation feed size in two stages: first, a SAG mill circuit with a single SAG Mill and, second, a ball mill circuit with two ball mills operating in parallel. The SAG mill will operate in closed circuit with a trommel, a pebble wash screen, and pebble crushers. The two ball mills will operate in closed circuit with hydrocyclones.

The SAG mill will be equipped with a 29 MW gearless wrap-around drive. SAG mill product (T80 = 2 to 2.5 mm) will discharge through a trommel screen. Trommel undersize will flow by gravity to the primary cyclone feed sump where

it will combine with the discharge of the ball mills. Trommel oversize will discharge to a pebble wash screen. Oversize from the pebble wash screen will be transported by belt conveyors to the pebble crushing circuit.

The pebble crushing circuit will consist of a surge bin, belt feeders and two shorthead, cone type crushers. The cone crushers will discharge onto the SAG feed conveyor. Pebbles (P80=12 mm) may bypass the pebble crushing circuit via a diverter gate, ahead of the pebble crusher surge bin, to the SAG feed conveyor.

Secondary grinding will be performed in two ball mills operated in parallel. Each ball mill will be equipped with a 22 MW gearless wrap-around drive. Each ball mill will operate in closed circuit with a single cluster of hydrocyclones. Discharge from both ball mills will be combined with the undersize from the SAG mill trommel and pebble wash screen in the primary cyclone feed sump and will be pumped to the hydrocyclone clusters via variable speed horizontal centrifugal slurry pumps. Hydrocyclone underflow will return by gravity to the ball mills. Hydrocyclone overflow (final grinding circuit product), with a target particle size distribution of 80 percent finer than 200 microns, will flow by gravity to the flotation circuit.

17.2.4 Flotation

17.2.4.1 Bulk (Copper/Molybdenum) Flotation

Hydrocyclone overflow will flow by gravity to the bulk (copper-moly) flotation circuit. The copper-moly flotation circuit will consist of two rows of mechanical rougher flotation cells, two rows of mechanical first cleaner flotation cells, two concentrate regrind mills operated in closed circuit with hydrocyclones, one row of mechanical second cleaner flotation cells, and two copper-moly third cleaner column flotation cells.

Rougher flotation concentrate will flow by gravity to a sump and will be pumped by variable speed, horizontal centrifugal pumps to the first cleaner flotation circuit. Tailing from the rougher flotation cells will flow by gravity to the pyrite scavenger flotation circuit.

Pyrite concentrate will join the first cleaner tailing at the pyrite thickener. Tailing from the pyrite flotation circuit (final tailing) will flow by gravity to the tailing thickeners.

First cleaner flotation concentrate will flow by gravity to the regrind cyclone feed sump. Tailing from the first cleaner flotation cells will be combined with the concentrate from the pyrite scavenger flotation section in the pyrite thickener.

Copper-moly concentrate regrinding will be performed in two vertical grinding mills operated in parallel. The vertical mills will operate in closed circuit with hydrocyclones. Vertical mill discharge will be combined with copper-moly first cleaner flotation concentrate in the regrind cyclone feed sump and will be pumped by variable speed, horizontal centrifugal slurry pump to a dedicated hydrocyclone cluster for each regrind mill. Hydrocyclone underflow will report back to the respective regrind mill. Hydrocyclone overflow (final regrind circuit product), with a target particle size distribution of 80 percent finer than 25 microns, will flow by gravity to the copper second cleaner flotation circuit.

Second cleaner concentrate will flow by gravity to the third cleaner feed sump and will be pumped by horizontal centrifugal pumps to the third cleaner columns for upgrading. Tailing from the second cleaner flotation cells will return to the first cleaner flotation circuit.

Third cleaner concentrate will flow by gravity to the copper-moly separation thickener. Tailing from the third cleaner flotation columns will return to the second cleaner flotation circuit.

The quantity and size of the flotation cells that will be installed in the bulk flotation circuit are shown in Table 17-3.

Table 17-3: Bulk Flotation Cells

Stage	Quantity of Cells	Size of Cells (m^3)
Copper Rougher	10	600
Pyrite Scavenger	4	600
Copper 1st Cleaner	12	100
Copper 2nd Cleaner	4	100
Copper 3rd Cleaner	2	4.5 m dia. column

Flotation reagents will be added at several points in the bulk flotation circuit as required.

17.2.4.2 Molybdenite Flotation

Concentrate from the final cleaner of the bulk flotation circuit will report to a copper-moly separation thickener. Thickened copper-moly concentrate will be pumped by variable speed, horizontal centrifugal slurry pumps to the molybdenite (moly) flotation circuit.

The moly flotation circuit will consist of two agitated rougher conditioning tanks, one row of separation (rougher) flotation cells, one row of first cleaner flotation cells, a concentrate regrind circuit, one second cleaner flotation column, one third cleaner flotation column, and one fourth cleaner flotation column.

Concentrate from the moly rougher cells will be pumped to the moly first cleaner flotation cells. Tailing from the moly rougher cells, which will be the final copper concentrate flotation product, will flow by gravity to the copper concentrate thickener.

Concentrate from the moly first cleaner cells will flow by gravity to the moly concentrate regrind circuit. Tailing from the moly first cleaner flotation cells will flow by gravity to the copper concentrate thickener.

To reduce consumption of NaHS and nitrogen, the rougher and first cleaner cells will be covered, and the flotation gas will be recycled.

Moly concentrate regrinding will be performed in a vertical mill, operated in open circuit. First cleaner moly concentrate will flow by gravity to a regrind sump and be pumped by a variable speed, horizontal centrifugal slurry pump to the mill. Reground moly first cleaner concentrate will be pumped to the moly second cleaner flotation column.

Concentrate from the moly second cleaner column will flow by gravity to the moly third cleaner flotation column. Tailing from the moly second cleaner flotation column will be pumped to the moly first cleaner flotation circuit.

Concentrate from the moly third cleaner column will flow by gravity to the moly fourth cleaner flotation column. Tailing from the moly third cleaner flotation column will be recycled to the moly second cleaner flotation column.

Concentrate from the moly fourth cleaner column, which will be the final moly concentrate flotation product, will flow by gravity to the moly concentrate dewatering circuit. Tailing from the moly fourth cleaner column will be recycled to the moly third cleaner flotation column.

The quantity and size of the flotation cells that will be installed in the moly flotation circuit are shown in Table 17-4.

Table 17-4: Moly Flotation Cells

Stage	Quantity of Cells	Size of Cells (m^3)
Moly Rougher	7	8.50
Moly 1st Cleaner	7	2.83
Moly 2nd Cleaner	1	2.0 m dia. column
Moly 3rd Cleaner	1	1.25 m dia. column
Moly 4th Cleaner	1	1.0 m dia. column

Flotation reagents will be added at several points in the moly flotation circuit as required.

17.2.5 Concentrate Dewatering and Storage

17.2.5.1 Copper Concentrate Dewatering

Moly rougher flotation tailing (copper concentrate) and moly first cleaner flotation tailing (copper concentrate) will flow by gravity to a copper concentrate thickener. Thickened copper concentrate will be filtered in three tower type copper concentrate pressure filters. Filter cake will discharge to a conveyor belt that will discharge to a covered copper concentrate stockpile.

Copper concentrate will be reclaimed by front-end loader onto highway haulage trucks. The loaded haul trucks will proceed to a wash station and be cleaned before exiting the concentrate load out area. This procedure will ensure against tracking of concentrate from the facility.

17.2.5.2 Molybdenite Concentrate Dewatering

Moly concentrate from the final moly cleaner flotation circuit will flow by gravity to an agitated filter feed tank. Moly concentrate slurry will be filtered in one tower type moly concentrate pressure filter. Filter cake will discharge to a Holo-Flite type dryer. The dryer will discharge to the moly concentrate storage bin.

Moly concentrate will be withdrawn from the dried moly concentrate storage bin by a packaging system and will be bagged in super-sacks for shipment by trucks to market.

17.2.6 Pyrite Concentrate Deposition

Pyrite scavenger concentrate and tailing from the copper first cleaner flotation circuit will collect in a pyrite thickener. Thickened pyrite concentrate will flow by gravity to the TMF for subaqueous deposition.

17.2.7 Tailings Dewatering

Tailing from each row of the pyrite scavenger flotation will flow by gravity to two tailing thickeners operated in parallel. Tailing thickener overflow will be pumped by vertical pumps to the process water pond for reuse in the mill. Overflow solution from the copper and moly concentrate thickeners will also be pumped to the tailing thickener overflow sump. Thickened tailing will flow by gravity to the tailing cyclone station either to produce sand via the cyclone plant or for direct deposition into the TMF. The cyclone tailings thickener receives cyclone overflow from the cyclone station. The thickener underflow flows by gravity to the tailings impoundment (may be pumped in later years). The thickener overflows to a concrete discharge sump (CTOS). Tailings discharge into the TMF will be via valved off-takes along a pipeline on the embankment crest. Discharge into the TMF will not be continuous from any one location in the pipeline. It will be rotated between off-takes as appropriate for tailings distribution within the TMF. This will ensure adequate beach development adjacent to the embankments and maximize the ability of the reclaim system to recover clean water.

Hydraulic placement of cyclone sand is not anticipated to be possible for a portion of the year due to freezing temperatures in the winter. The coarse cyclone underflow will be processed using dewatering screens during the winter months to produce sand that can be transported by conveyor to the toe of the dam for future use in construction of the Main Dam. The coarse cyclone underflow will flow by gravity to eight screens for dewatering. Screen undersize will be pumped using vertical pumps to the sand cyclone overflow thickener. Screen oversize will discharge on to the reversible dewatering screen collecting conveyor where it will be directed to the coarse sand conveyor and sent to the tailing dam crest, or it will be directed to the coarse sand stockpile feed conveyor which discharges to the coarse sand stacker ahead of a stockpile. The sand cyclone overflow flows by gravity to the sand cyclone overflow thickener. Thickener underflow flows by gravity to the TMF. Thickener overflow flows by gravity to the sand cyclone thickener overflow sump. A portion of the overflow is pumped using vertical pumps to the sand cyclone feed sump for cyclone feed dilution. The remaining overflow is pumped using vertical pumps to booster station No. 1 as part of the reclaim water supply back to the concentrator.

17.2.8 Reagents and Consumables

Reagent storage, mixing, and distribution will be provided for all reagents used in the sulphide processing circuits. Table 17-5 below is a summary of reagents used in the process plant.

Table 17-5: Process Consumables

Reagent & Consumables	Units	Consumption Rate
Copper Flotation Reagents		
Lime (hypogene)	kg/t	1.075
Lime (supergene sulphide)	kg/t	2.688
Fuel Oil	kg/t	0.007
3418A (hypogene)	kg/t	0.004
3418A (supergene sulphide)	kg/t	0.008
A208 (hypogene)	kg/t	0.008
A208 (supergene & sulphide)	kg/t	0.017
MIBC	kg/t	0.010
PAX	kg/t	0.040
Flocculant	kg/t	0.022
Moly Flotation Reagents		
NaHS	kg/t	0.053
Fuel Oil	kg/t	0.0004
Flocculant	kg/t	0.0034
Liners and Grinding Media		
Primary Crusher - Liners	kg/t	0.040
SAG Mill - Liners	kg/t	0.040
Ball Mill - Liners	kg/t	0.048
SAG Mill - Balls	kg/t	0.400
Ball Mill - Balls	kg/t	0.400
Regrind Mill – Balls	Kg/t	0.041

17.2.9 Water System

17.2.9.1 Fresh Water

The water source for Casino is the Yukon River, 17 km away. Fresh water will be pumped from the fresh-water intake Ranney well through a series of booster stations, pumps, and pipelines to the plant site area. Fresh water will collect

in a freshwater pond at the process site. The design capacity of the freshwater collection and transfer system will be approximately 3,200 m^3/h.

Fresh water stored in the freshwater pond will gravity feed a fresh/fire water tank. Fresh water will be pumped from the fresh/fire water tank to a camp and will gravity feed to distribution points in the plant. Fresh water will be distributed to:

- A chlorinator package and subsequent potable water tank, for use in offices, laboratory, dry, and rest rooms
- A mine water tank for mine road dust control
- A gland seal water tank to supply seal water for mechanical equipment
- The process water pond
- The fire water distribution system in the mill site area

17.2.9.2 Process Water

Drain water and seepage, designated as process water, is collected in the water management pond below the embankment. Water is pumped from the pond to the water management booster station by low head vertical pumps. High head vertical pumps installed in this transfer sump transfer the water up to the sand cyclone thickener overflow sump (CTOS). Low head vertical pumps in the CTOS pump water up to the concrete sand cyclone feed tank to dilute the tailings feeding the cyclone station. High head, vertical pumps in the CTOS pump the balance of the water, net of water required for cyclone dilution purposes, to Reclaim Booster Station #1. In addition, reclaim water is pumped to the ADR Reclaim/Fire water tank for distribution and to the barren solution tank as makeup water required for the oxide heap leach.

Vertical pumps installed on the NAG area barge pump reclaimed water to a concrete relay sump at about elevation 900 m. Vertical pumps installed on the two PAG barges pump reclaim water to the relay sump as well. Vertical pumps in Relay Station #1 sump transfer reclaimed water to Relay Station #2 where vertical pumps transfer water to Booster Station #1. Vertical pumps installed in Booster Station #1 transfer reclaimed water to the Process Water Pond. Process water will be distributed from the process water pond by gravity flow through a pipeline to mill process water usage points.

17.3 POWER

The Casino sulphide process facility has an estimated 146.0 megawatt (MW) total connected load and 132.0 MW total demand load. See Section 21 for power costs used in the operating cost estimate and financial model.

17.4 AIR SERVICE

Separate air supply systems will supply air to the following areas:

- Crushing
- Flotation – air will be provided by two compressors coupled with an air receiver.
- Concentrate Filtration
- Plant Air – high pressure air will be provided by two compressors coupled with an air receiver.
- Instrumentation – high pressure air will be provided from a dedicated compressor, receiver, and air dryer.

17.5 PROCESS CONTROL PHILOSOPHY – SULPHIDE PLANT

The sulphide plant control system will consist of a Siemens Simatic PCS 7 Plant Control System (PCS) with PC-based operator interface stations (OIS) located at the central control room. The PCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generations. The process

control system networks Operation Servers (OS), Process Historian/Information Servers (PH/IS), Web Server (WS), Operator Server (OS) Clients, Engineering Station (ES), and Web Clients to allow operators in the Control Rooms and in other designated control stations throughout the site, to monitor and control the industrial process. The plant central control room will be staffed by trained personnel 24 hours per day.

The process control will be enhanced with the installation of an automatic sampling system. The system will collect samples from various streams for on-line analysis and the daily metallurgical balance. Vendors' instrumentation packages will be integrated with the central control system. In addition, density and particle size metres will be installed in the cyclone feed sump to control the grind.

A closed-circuit television (CCTV) system will monitor various facilities and conveyors discharge points. The cameras will be monitored from the central control room.

17.6 QUALITY CONTROL

Samplers will be installed in locations that are required for metallurgical accounting and process control purposes. The final concentrate and intermediate streams will be monitored by an on-line x-ray diffraction analyzer, which may include pH control and reagent addition control systems. The assay data will be fed back to central control room and used to optimize process conditions. Routine samples of intermediate products and final products will be collected and analyzed in an assay laboratory where standard assays will be performed. The data obtained will be used for product quality control and routine process optimization. Feed and tailings samples will also be collected and subjected to routine assay.

The assay laboratory will consist of a full set of assay instruments for base metal analysis, including an atomic absorption spectrophotometer (AAS), and ICP, experimental balances, and other determination instruments such as pH and redox potential metres.

17.7 METALLURGICAL PERFORMANCE PROJECTION

According to the metallurgical performance projections developed from the metallurgical test results and the proposed mine plan, life of mine concentrate production is projected and shown in Table 17-6.

Table 17-6: Metallurgical Performance Estimate

Mill Feed			Copper Concentrate			Moly Concentrate		
Tonnage (kt)	Cu %	Mo %	Recovery Cu %	Grade Cu %	Production (kt)	Recovery Mo %	Grade Mo %	Production (kt)
1,217,069	0.189	0.021	86.5	28%	7,116	71.2	56	330

17.8 PROCESS DESCRIPTION

The oxide ore processing facility will produce gold and silver Doré bars via heap leach and carbon adsorption technology. Copper contained in the oxide ore will be recovered as a copper sulphide precipitate using SART technology.

17.9 OXIDE ORE PROCESS PLANT DESCRIPTION

17.9.1 Process Design Criteria and Major Equipment

Process design criteria were developed for the Project based on a 9,125,000 t/y plant design. Crushing, conveying, and stacking plant is designed to operate 300 days each year at 30,417 mtpd. Heap leach, ADR, and the SART plant

are designed to operate 365 days each year based on leaching 25,000 mtpd or ore on the leach pad each day. The crushing circuit was designed to operate with an overall availability of 75%. The remainder of the processing facilities were designed to operate with an overall availability of 98%. The equipment was sized using these criteria. Table 17-7 is a summary of the main components of the oxide process design criteria used for the study. Table 17-8 is a summary of the major process equipment selected for the study.

Table 17-7: Oxide Process Design Criteria

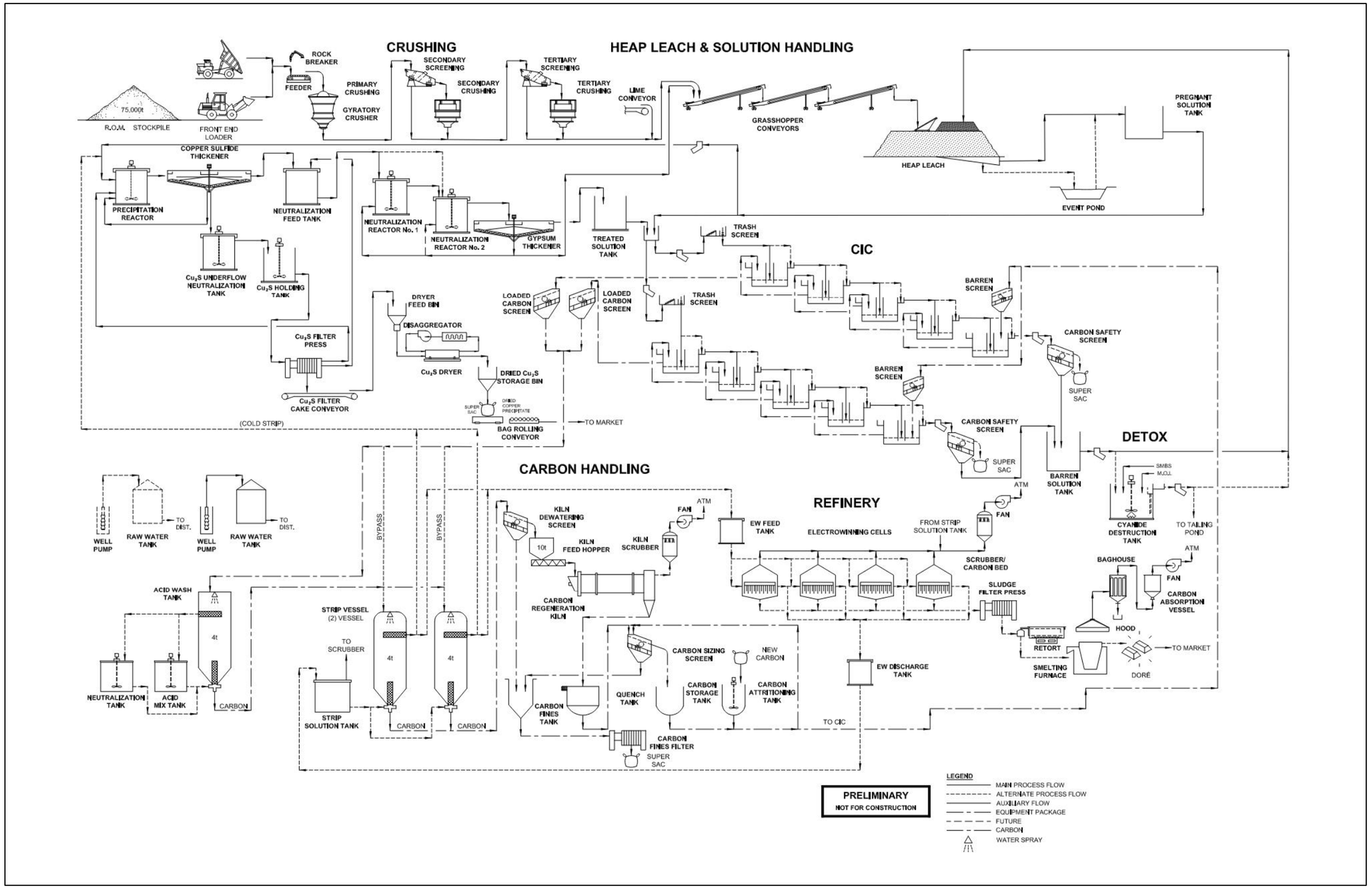
Description	Unit	Value
General		
Type of Deposit	-	Oxide Cap Material
Plant Feed Grade (LOM)		
Gold (Au)	g/t	0.44
Silver (Ag)	g/t	3.0
Copper (Cu)	%	0.05
ROM Ore Characteristics		
Maximum mine-run ore size	mm	900
Specific Gravity	-	2.65
Moisture Content	%	2
Ore Stacked Density	t/m^3	1.75
Crushing		
Primary Crushed Ore Product Size, P_{100}	mm	200
Final Crushed Ore Product Size, P_{100}	mm	50
Primary Crushed Ore Product Size, P_{100}	mm	25
Operating Schedule		
Shift/Day	-	2
Hours/Shift	h	12
Hours/Day	h	24
Days/Year- Crushing, Conveying, Stacking	d	300
Days/Year – Heap Leach/ADR/Refinery/SART	d	365
Plant Availability/Utilization		
Overall Plant Feed	t/y	9,125,000
Overall Crushing, Conveying, Stacking Plant Feed	t/d	30,417
Crushing Plant Availability	%	75
ADR/Refinery/SART Plant Availability	%	98
Crushing/Stacking Rate	t/h	1,690
Design Factors		
Continuous flows thru CIC and SART circuits	-	1.15
Batch sizing in the ADR plant	-	1.15
Heap Leach		
Pile		
Pad ore storage capacity	Tonne	210,000,000
Pile slope	-	2.5:1
Lift height	m	8.0
Ore pile setback	m	9
Precipitation	mm/a	500
Pan Evaporation	mm/a	308
Leach Solution		
Leach solution application method	-	Distribution network with emitters
Leach solution application, primary	d	60
Leach solution application, secondary	d	Through subsequent lifts
Application rate	$L/h/m^2$	12
Retained solution, leach pile moisture (wet)	%	8 to 10
Operating solution, leach pile, moisture (wet)	%	10 to 12

Description	Unit	Value
Barren solution flow rate (average)	m^3/h	1,312
Pregnant solution flow rate (average)	m^3/h	1,223

Table 17-8: Major Process Equipment

Item	Number	Description	Key Criteria (ea.)
Primary Crusher	1	1100 x 1800 TSU Top Service Gyratory Crusher	450 kW
Secondary Screen	1	3 m x 6.7 m Inclined; double deck; banana	19 kW
Secondary Crusher	1	Raptor XL 1300	970 kW
Tertiary Screen	1	3 m x 6.7 m Inclined; double deck; banana	19 kW
Tertiary Crusher	1	Raptor XL 1300	970 kW
Heap Leach Stacking System	1	Conveyors, Stacker	2,200 kW
Barren Solution Pump	2	Vertical Turbine; 703 m^3/hr @ 156 m TDH	448 kW
Pregnant Solution Pump	3	Submersible Vertical Pump; 754 m^3/hr, 125 m head	336 kW
Carbon Handling Plant	1 lot	ADR and Refinery	3 ton
SART plant	1 lot	Design basis: 400 mg/L Cu in the PLS	150 m^3/hr

Figure 17-2 is a simplified schematic of the overall process for the oxide ore. This provides the basis for the process description that follows.



(Source: M3)

Figure 17-2: Simplified Oxide Process Flowsheet

The following items summarise the process operations required to extract gold from the oxide gold ore.

- Size reduction of the run-of-mine (ROM) ore to minus 200 mm using a primary gyratory crusher.
- Size reduction of the primary crushed ore to minus 50 mm through screening and a secondary cone crusher.
- Size reduction of the secondary crushed ore to minus 25 mm through screening and a tertiary cone crusher.
- Stacking crushed ore by overland conveyors and a stacker onto a heap leach pad, and subsequently, leaching the ore with cyanide solution.
- Recovering gold and silver from the pregnant leach solution on activated carbon in carbon-in-column tanks (CIC).
- Recovering copper from the pregnant leach solution by the Sulphidization, Acidification, Recycling and Thickening (SART) process.
- Treating gold and silver loaded carbon recovered from the CIC circuit by acid washing, cold stripping with cyanide solution to remove copper, hot stripping with caustic solution to remove gold, and thermal reactivation of the carbon.
- Recovering gold from the pregnant carbon stripping solution as cathode sludge on stainless steel mesh cathodes in an electrowinning cell.
- Melting the cathode sludge with fluxes to produce a gold/silver Doré bar, the final product of the ore processing facility.
- Storing, preparing, and distributing reagents to be used in the process.

17.9.2 Crushing, Conveying, and Stacking

The Oxide ore will have a primary crusher and conveyor system separate from the Sulphide ore.

Run of mine (ROM) oxide ore will be trucked from the mine to the primary crusher apron feeder. Alternatively, ROM may be stockpiled in the ROM Stockpile if the primary crusher is down for maintenance. A front-end loader will reclaim ore from the stockpile and dump onto the primary crusher apron feeder. The apron feeder will provide the feed to the primary crusher. The primary crusher will be a gyratory crusher. The primary crusher will discharge onto a secondary screen feed belt conveyor.

Secondary screen feed conveyor will discharge onto the secondary screen. Screen undersize will discharge onto the fine ore transfer conveyor. Screen oversize will discharge onto secondary screen discharge belt conveyor, which discharges into secondary crusher feed bin.

Secondary crusher belt feeder will draw ore from the bin and provide feed to the secondary cone crusher. Cone crusher discharge will combine with undersize from the secondary screen on the tertiary screen feed belt conveyor.

Tertiary screen feed conveyor will discharge onto the tertiary screen. Screen undersize will discharge onto the fine ore transfer conveyor. Screen oversize will discharge onto the tertiary screen discharge belt conveyor, which discharges into tertiary crusher feed bin.

Tertiary crusher belt feeder will draw ore from the bin and provide feed to the tertiary cone crusher. Cone crusher discharge will combine with undersize from both the secondary and tertiary screens on the fine ore transfer conveyor. Lime will be added to this conveyor, which discharges onto the intermediate transfer conveyor.

The intermediate transfer conveyor discharges onto a series of overland transfer conveyors, with the last overland conveyor discharging onto the telescoping stacker feed conveyor. The telescoping stacker feed conveyor discharges onto the leach pile stacking conveyor, which places crushed ore onto the heap leach pile.

A hydraulically operated, pedestal-mounted, rock breaker will be installed over the primary crusher rock box. A belt magnet will be installed over the secondary screen feed conveyor to remove any tramp metal in the system. A metal detector will be installed above the secondary crusher belt feeder to alert the operator of metal still in the system and to protect the secondary cone crusher.

A belt scale will be installed on the secondary screen feed conveyor, the coarse ore conveyor and on the fine ore transfer conveyor to monitor primary crusher discharge. Dust control in the crushing area will be a dry type of dust collector system.

17.9.3 Heap Leaching

The heap leach pad will consist of liners and a low-permeability soil liner. A perforated pipe drainage system will enhance drainage of leach solutions away from the liner, reduce hydrostatic head, and facilitate pregnant solution recovery. A layer of crushed and screened rock will be placed on the liner. See section 18.8.3 for the liner system details.

Barren process solution will be applied to ore lots. Solution will be applied with drip emitters to minimize evaporation losses. When an ore lot has completed the primary leach cycle, solution application will be stopped, and another ore lift (or layer) will be placed on top of the previous lift. Leach solution application will resume. The process of layering and leaching the ore will repeat for a maximum of eight ore lifts or layers on the leach pad. When the last process leach cycle is completed on the last lift, the ore heaps will be rinsed with fresh water to recover the remaining gold and rinse the residue.

Pregnant solution discharging from the ore heaps will be collected in a network of pipe placed throughout the overliner material that will direct the solution to the in-heap collection area.

Pregnant solution will be pumped from the in-heap collection area using horizontal, centrifugal pregnant solution pumps. The pump discharge pipes will be combined in a single pipeline to the carbon-in-column (CIC) / SART circuit for recovery of gold and copper.

An events pond will be installed to handle any overflow that might occur during a large precipitation event. Water that accumulates in the events pond will be periodically pumped by a submersible pump to the barren solution system feeding leach solution to the ore heaps.

The pond system has been sized to contain normal operating solutions and stormwater. Ponds will be fenced to reduce the risk of danger to wildlife.

The design of the heap leach facility is such that the facility will be operated as a zero-discharge facility. A cyanide detox system will be installed as a contingency measure. If a surplus of solution exists, the excess solution will be routed to cyanide destruction prior to transfer to the tailing management facility, which is part of the sulphide concentrator.

17.9.4 Carbon ADR Plant/SART

Gold and silver will be recovered from heap leach pregnant solution by adsorption of their ions on activated carbon followed by desorption and electrowinning of a gold and silver solid product. Copper will be recovered from the pregnant

solution by the SART process where the copper will be precipitated by the addition of sulphide to produce a copper sulphide product.

The process steps required to recover gold and silver by the carbon adsorption method include:

- loading gold and silver on activated carbon in a CIC circuit,
- acid washing of the carbon to remove water scale and acid soluble copper,
- cold stripping of carbon (elution) to remove copper,
- stripping gold and silver from the carbon using a hot caustic solution,
- electrowinning gold and silver from the stripping solution in a precious metal sludge using an electrolytic cell,
- reactivating stripped carbon by thermal regeneration, and
- melting the precious metal sludge in a crucible furnace to produce Doré bars.

The process steps required to recover copper by the SART method include:

- bleeding a portion of the pregnant solution to the SART process,
- adding sodium hydrosulphide to the solution,
- decreasing the pH of the solution with acid, thereby precipitating copper,
- removing the copper precipitate from the solution by thickening, filtration, and drying,
- increasing the pH of the solution with lime, thereby precipitating gypsum,
- removing the gypsum from the solution by thickening, and
- shipping the filtered copper sulphide product to a smelter for refining.

17.9.4.1 Carbon-in-Column (CIC) Circuit

Gold will be recovered from pregnant leach solution in a two-train, five-stage carbon-in-column (CIC) circuit. Pregnant leach solution will pass through a stationary screen to remove trash prior to being introduced into the CIC tank line. Most of the pregnant solution will pass directly to the CIC circuit, and the remainder will be processed for copper recovery by the SART process and then be returned to the CIC circuit.

The CIC circuit will consist of two parallel trains of five CIC tanks operated in series. Each CIC tank will be a flat bottom tank with an internal distribution plate between the bottom and the carbon charge. It will be possible to bypass any tank by using a manually operated dart-valve.

Solution (barren solution) will exit the CIC lines, be combined, and pass through a safety, single deck, vibrating screen to capture carbon unintentionally washed from the carbon columns. Screen oversize (carbon) will be transferred by gravity to a carbon quench tank.

Carbon will be advanced through the circuit by a series of recessed impeller, horizontal centrifugal pumps. Loaded carbon advanced from the first CIC vessel will be pumped by the first carbon transfer pump from each train to a carbon distributor tank. Carbon will be advanced through from tank to tank by additional, dedicated carbon transfer pumps – one pump for each CIC vessel.

Regenerated and new carbon will be sized by screening on single deck vibrating screens. Screen undersize will flow by gravity to a carbon fines tank. Screen oversize will flow by gravity to the last carbon column on either of the CIC lines when carbon is advanced.

Barren solution will discharge from the CIC tank line and flow into a barren solution tank. Vertical turbine pumps will pump barren solution from the barren solution tank to the heap leach barren solution distribution system. The pump discharge lines will be combined to a single pipeline to the heap leach area.

Cyanide solution will be added to the barren solution tank and/or the first CIC columns.

17.9.4.2 SART Circuit

A portion of the heap leach pregnant solution, approximately 150 m^3/h, will be pumped to the SART process. Sodium hydrosulphide will be added to the solution through an in-line mixer. Downstream of this in-line mixer, sulfuric acid will be added to the solution prior to mixing in the precipitation reactor.

The discharge from the precipitation reactor will flow by gravity to a covered copper sulphide thickener. Most of the underflow from this thickener will be recirculated to the precipitation reactor, while the balance of the thickener underflow will be advanced to a copper sulphide neutralization tank. Overflow solution from the copper sulphide thickener will collect in a neutralization feed tank.

Sodium hydroxide will be added to the copper thickener underflow neutralization tank. Neutralized copper sulphide, thickened slurry will be fed to the copper sulphide filter press. The filter cake will discharge onto the copper sulphide filter cake belt conveyor, which discharges into a bin. The filtered cake will pass through a dryer. Dried copper sulphide will collect in a bin, be loaded into supersacks, and transported by flatbeds trucks to market.

Copper sulphide thickener overflow will collect in a neutralization feed tank. Neutralization feed pump will transfer the overflow solution to the first neutralization reactor. Lime and recirculated gypsum thickener underflow will be added to this reactor, which overflows into the second neutralization reactor.

Neutralized solution will flow by gravity to a covered, gypsum thickener. A portion of the thickener underflow will be recirculated to the first neutralization reactor. The balance of the underflow will be pumped to a gypsum holding tank.

Gypsum thickener overflow will collect in a treated solution tank, where an antiscalant will be added. This treated solution will be returned to the CIC circuit.

A scrubber will collect fumes from the precipitation reactor, the copper sulphide thickener, the copper sulphide neutralization tank, the neutralization feed tank, the neutralization reactors, the gypsum thickener, and the sodium hydro-sulphide storage tank. The pH of the scrubber will be adjusted by NaOH. Some scrubber discharge will be recirculated, but net scrubber discharge will report to the first neutralization reactor.

17.9.4.3 Carbon Acid Wash

Loaded carbon will be acid washed in a column. A dilute hydrochloric acid solution will be pumped into the bottom of the acid wash tank, flow up through the vessel, and overflow to the acid tank. This pump will either be operated to circulate solution through the carbon, or the carbon may be left to soak. After completion of the acid wash solution cycle, the batch of spent acid solution will be pumped to the acidifying reactor tank in the SART circuit.

A caustic (basic) solution will be pumped into the bottom of the acid wash tank at the end of its cycle, flow up through the vessel, and overflow to the acid tank. The caustic solution pump will either be operated to circulate solution through the carbon or to fill the acid wash tank and allow the carbon to soak. Upon completion of the acid wash cycle, the caustic solution will be used to neutralise the acid solution to pH 8 to 10. Neutralized solution will be pumped to the barren tank.

17.9.4.4 (Copper) Cold Stripping

The acid washed loaded carbon will be cold stripped to remove copper before hot stripping to remove gold.

Water will be added to the acid washed carbon and the carbon and water slurry will be pumped from the acid wash vessel to one of the two strip columns. Water transferred with the carbon (carbon transfer water) will be drained through an internal screen in the bottom of the cold strip tank and will drain to a carbon fines tank.

Cyanide solution will be pumped into the top of the cold strip tank, flow down through the vessel, and discharge through the internal screen. The carbon will then be soaked for the cold strip cycle. Upon completion of the cold strip cycle, the strip solution will be transferred to the SART circuit.

The drained carbon will be rinsed with barren solution to remove the copper strip solution. The rinse solution will also be transferred to the carbon fines tank.

17.9.4.5 Carbon Elution Circuit

Elution will be by pressure Zadra techniques. Barren strip solution, containing sodium hydroxide (caustic) will be heated by a solution heating system and circulated through the bottom of the elution vessel by a horizontal centrifugal pump. The solution will flow up through the column, exit the top as pregnant solution, and flow through a heat recovery heat exchanger. The cooled solution will flow to the electrowinning (EW) barren return tank.

After completion of the elution cycle, the strip solution will be drained through an internal screen in the bottom of the elution tank and will be transferred to the carbon fines tank.

17.9.4.6 Electrowinning

Gold and silver will be recovered from pregnant strip solution by the electrowinning process.

Pregnant strip solution will be pumped from the EW barren return tank to an electrowinning cell. Knitted stainless steel mesh cathodes will be used in the cells for deposition of the gold and silver. The stripped solution will discharge from the cell and flow back to the EW barren return tank.

When the electrowinning cycle has been completed, the solution will be pumped from the EW barren return tank to the barren strip solution tank. A high pressure, water wash system will remove the metal sludge from the cathodes and wash it into an electrowinning wash/ sludge tank. Sludge slurry will be pumped by a diaphragm pump to a plate and frame, pressure filter. Filter cake will be cleaned from the filters by hand and placed in filter cake boats for refining.

17.9.4.7 Smelting

Filter cake will be mixed with fluxing materials and charged to an electric, crucible furnace. The melted charge will be poured into conical molds. Doré (gold and silver) will sink to the bottom of the mold, and slag glass containing fused fluxes and impurities, will float to the top of the mold. Doré will be sampled for gold content using vacuum tube samples.

After cooling and solidifying, the molds will be dumped, and the slag will be knocked off the Doré buttons by hand. A vertical drill press will be available as a back-up sample method for Doré buttons.

Buttons will be cleaned under a water stream using a needle gun, weighed, and stamped with an identification number and weight. Doré buttons will be the final product of the operation and will be stored in a safe until shipment.

Slag will be crushed and screened to recover high-grade chips that will be returned to the melting furnace. Remaining slag will be returned to the heap leach.

Fumes from the melting furnace will be collected through ductwork and cleaned in a bag house dust collector system before discharging to atmosphere.

17.9.4.8 Carbon Regeneration

Barren, stripped carbon will be sized on a single deck, vibrating screen. Screen undersize will flow by gravity to the carbon fines tank. Screen oversize will flow by gravity to a kiln feed tank.

Carbon will be withdrawn from the kiln feed tank by a screw conveyor feeder. The feeder will discharge to a horizontal, propane gas fired, carbon regeneration kiln.

The carbon regeneration kiln will discharge into carbon quench tank. Quenching will be done in a conical bottom tank. Regenerated carbon will be transferred from this tank to an activated carbon storage tank. New carbon, after being soaked and attrition agitated in a separate tank, will be added as required. Carbon and water slurry will be pumped from this tank by recessed impeller, horizontal, centrifugal pump to carbon sizing screens for use in the CIC circuit.

17.9.5 Reagents and Consumables

Reagent storage, mixing, and distribution will be provided for all reagents used in the oxide processing circuits. Table 17-9 is a summary of reagents used in the process plant.

Table 17-9: Process Consumables

Reagent & Consumables	Units	Consumption Rate
Heap Leach/SART Reagents		
Sodium Cyanide (NaCN)	kg/t	0.500
Sodium Hydroxide (caustic, NaOH)	kg/t	0.130
Pebble Lime (CaO)	kg/t	3.516
Hydrochloric Acid (HCl)	kg/t	0.010
Sodium Hydrosulphide (NaHS)	kg/t	0.025
Sulfuric Acid H_2SO_4)	kg/t	0.328
Activated Carbon	kg/t	0.011
Antiscalant	kg/t	0.003
Flocculant	kg/t	0.00035
	kg/t	
Primary Crusher - Liners	kg/t	0.040
Secondary Crusher – Liners	kg/t	0.085
Tertiary Crusher – Liners	kg/t	0.085

17.9.6 Water System

Initial water requirements for the oxide ore heap leach and gold plant operations will be met by pumping water retained behind a temporary cofferdam located in the heap leach event pond. After start-up of the sulphide concentrator, process water will be provided from the TMF reclaim water system for use as makeup water at the heap leach and gold plant. Reclaim water from the sand cyclone thickener overflow sump will be pumped to the ADR Reclaim/Fire water tank for distribution and to the barren solution tank as makeup water required for the oxide heap leach.

17.10 POWER

The Casino oxide process facility has an estimated 12.4 megawatt (MW) total connected load and 9.8 MW total demand load. See Section 21 for power costs also used in the operating cost estimate and financial model.

17.11 AIR SUPPLY

An air distribution system will be included to supply required process air to the plant – primarily the crusher. Instrument air will be included for required instrumentation and controls. Blowers will supply air requirements for the Cyanide Destruction circuit.

17.12 PROCESS CONTROL PHILOSOPHY – OXIDE PLANT

The plant control system will consist of a Siemens Simatic PCS 7 Plant Control System (PCS) with PC-based operator interface stations (OIS) located at the ADR plant control room. The PCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generations. The process control system networks Operation Servers (OS), Process Historian/Information Servers (PH/IS), Web Server (WS), Operator Server (OS) Clients, Engineering Station (ES), and Web Clients to allow operators in the Control Rooms and in other designated control stations throughout the site, to monitor and control the industrial process. The ADR plant central control room will be staffed by trained personnel 24 hours per day.

Vendors' instrumentation packages will be integrated with the central control system.

A closed-circuit television (CCTV) system will monitor various facilities and discharge points. The cameras will be monitored from the central control room.

We would expect the following:

- Primary Crushing 100 Area– Control Shack
- Primary Crushing 110 Area– Control Shack
- Grinding 300 Area - Control Room/Server Room
- Concentrate 500 Area - Control Room (small)
- Sand Cyclone 640 Area - Control Room (small
- SART 450.ADR 580 Area - Control Room/Server Room

17.13 QUALITY CONTROL

Continuous samplers will be provided on selected streams to calculate the plant material balance and for control of the process. Routine samples of intermediate products and final products will be collected and analyzed in an assay laboratory where standard assays will be performed. The data obtained will be used for product quality control and routine process optimization. Feed, doré and solution samples will also be collected and subjected to routine assay.

17.14 METALLURGICAL PERFORMANCE PROJECTION

According to the metallurgical performance projections developed from the metallurgical test results average grades and recoveries for the heap leach are shown in Table 17-10.

Table 17-10: Metallurgical Performance Estimate

Heap Leach Feed	Grade			Recovery		
Tonnage (kt)	Cu %	Au g/t	Ag g/t	Cu %	Au %	Ag %
209,636	0.036	0.265	1.946	18	80	26

18 PROJECT INFRASTRUCTURE

The Project site is located about 300 km northwest of Whitehorse, about 200 km northwest of the Village of Carmacks, and about 18 km southwest of the Yukon River. The approximate elevation of the mine pit is 1,300 m. The plant site coordinates are 6,956,718.230 N, 611,800.867 E. The elevation of the plant site is approximately 1,190 m.

The mine site is remote from significant population centres, and accessed via helicopter year-round, or via a gravel airstrip designed to accommodate light aircraft during the summer months. A winter road has been established to site in intermittent years to bring large equipment to site, and the project is also accessible via barging up the Yukon River. The Yukon Highway network is paved, including the 180 km of roads from Whitehorse to Carmacks as well as south to Skagway, Alaska, the optimal port to utilize for concentrate exports and mine supply imports. Whitehorse has an international airport with daily flights to Vancouver, BC, and multiple flights a week to Calgary, Alberta.

The Yukon electrical grid is at a significant distance from the mine and at the current time, the utility does not have enough generating capacity to fulfill the requirements for the Casino mine. The cost-effective solution to provide power for the Project is to build and operate an on-site power plant.

18.1 ACCESS ROADS

18.1.1 Mine Site Access Road

Associated Engineering examined various route options for a year-round access road to Casino. The selected option which appears to have the least environmental impact and the greatest stakeholder support is a new road from the end of the existing Freegold Road approximately 70 km northwest of the Village of Carmacks. The proposed Casino/Carmacks access road would be a 132 km, two-lane, gravel resource road designed to accommodate B-Train Double (BTD) and Tridem trucks. The road design criteria satisfies the guidelines in the BC Ministry of Forests and Range Forest Road Engineering Guidebook (2nd Edition, 2002) for a 70 km/h design speed with some 50 km/h sections where road geometry is limited by the terrain.

In order to maximise the design speed and avoid unstable terrain, the route is located as much as possible in valley bottoms. The road surface elevation is designed to be 2.0 m above existing ground in these areas, with the fill material placed over undisturbed soils. This embankment height stabilises the road against washouts and protects against permafrost degradation under the road.

Where the road climbs out of the valley bottoms, the road construction method includes both cut and fill. Permafrost rich areas may require buttressing of cut slopes with a layer of angular rock fill on top of filter fabric. This will prevent permafrost degradation and act as a retaining structure to improve slope stability.

Most of the fill required for road construction will be developed from borrow pits located along the road alignment and then hauled to where it is needed. The section of road from the Selwyn River to the mine is located in soil that is mainly suitable for road embankment construction and can be utilized for fill. Further soil testing may reveal other locations with borrow suitable for road construction which will result in shorter haul distances and reduced road construction costs.

There are 16 major bridge crossings located along the main access road. These include crossings of Bow Creek, Big Creek, Hayes Creek, and Selwyn River along with several tributaries and side channels. Associated Engineering completed Hydro-technical analysis and topographic site surveys to provide the necessary information to then complete conceptual bridge designs for each crossing. There are also 71 major culvert crossings that have been identified for the main access road with estimated diameters ranging from 1,500 mm to 2,400 mm. Detailed hydro-technical analysis

has not been completed for the culvert crossings. Culverts in fish bearing streams will be embedded to provide a simulated natural stream bottom.

Smaller 500 mm and 600 mm culverts will be used to control road drainage with the culvert spacing determined by road gradient and natural depressions in the topography. The cost of upgrading this road will be shared between Industry (in this case the Casino Mine), the Yukon Government and the Canadian Government. The proportions will be 70% to Industry, and 30% to the Yukon and Canadian Governments. Figure 18-1 shows the proposed mine access road route.

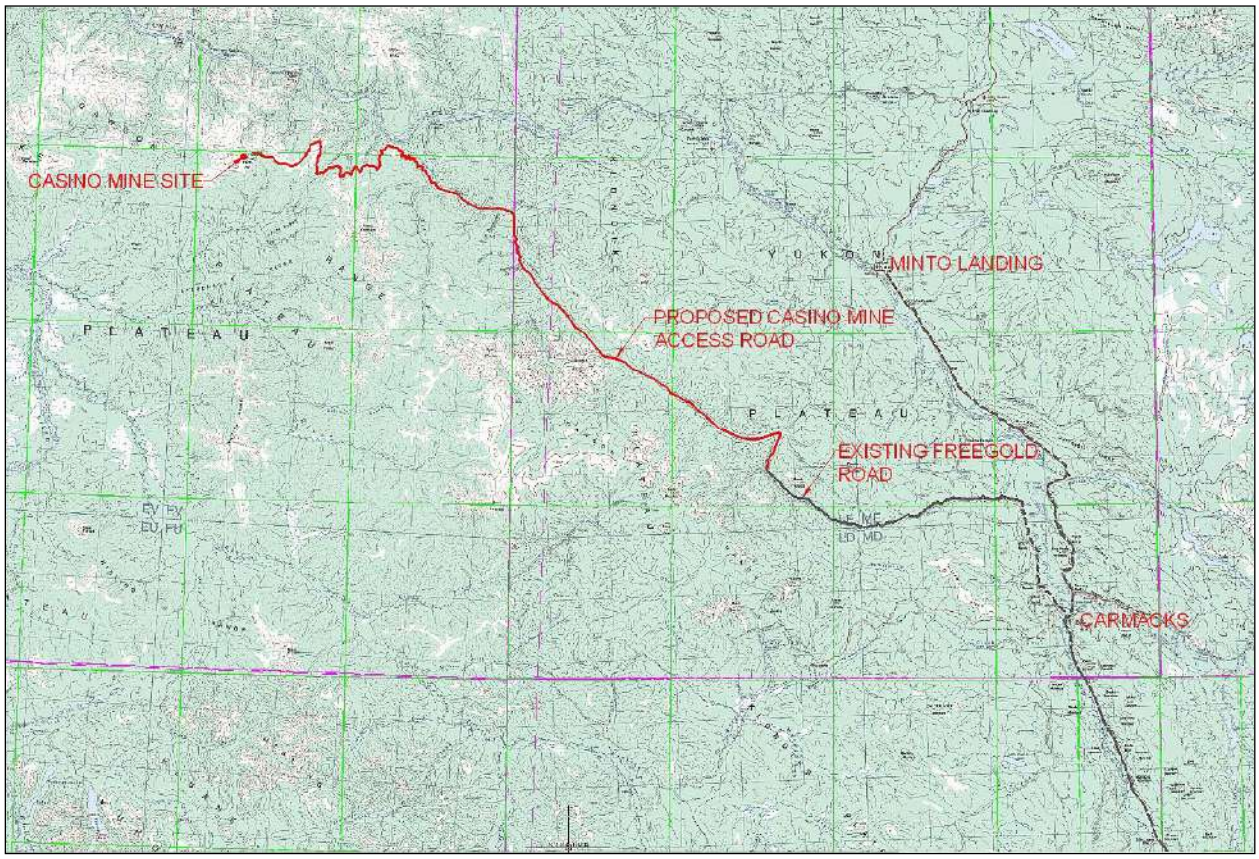


Figure 18-1: Proposed New Access Road

18.1.2 Service Roads

Service roads will connect the various mine and process facilities together with the ancillaries. The roads will be constructed with a minimum 4 m wide all-weather gravel surface. In general, the maximum grade for the road will be 10%.

An existing road leads northward from the mine facility to the Yukon River along Britannia Creek. The new freshwater pipeline will generally follow the road alignment. The roadway will be graded and improved as a service road to facilitate construction, access, inspection, and maintenance of the pipeline.

18.2 PORT FACILITIES

The Port of Skagway is located 560 km from the Casino site and has been selected as the port of export for the Project. The port has historically exported over 600,000 tonnes annually of lead and zinc concentrates and currently exports the copper concentrates from the Minto operation. An engineering evaluation has determined that the existing concentrate storage and handling facilities at Skagway can be economically upgraded to serve the Casino export requirements. The Port of Skagway and AIDEA have expressed interest in providing concentrate storage and load-out services to Casino in their to be expanded facilities, consistent with the conceptual design prepared by Western. The operating costs in this study reflect exporting of copper concentrate through the Port of Skagway facilities.

The upgrades to the Port facilities that are the responsibility of the Port of Skagway are the replacement of the Shiploader, extension of the existing conveyor that supports the Shiploader, and repair or replacement of the existing dock. Additionally, the Port of Skagway will provide a new Load-out Facility with the capability to store 30,000 to 40,000 tonnes of concentrate. This facility will be complete with dust collection, offloading and stacking capabilities of a bulk concentrate and access to that equipment. Power will be provided to CMC for operation of the conveying equipment, dust collection, lighting, the Pebble Lime silo, and the pneumatic equipment, and convenience.

Pebble Lime Storage (a new 3,000-ton silo) and the means to offload the Pebble Lime from barges and convey it to the silo pneumatically, are the responsibility of CMC.

18.3 SITE LAYOUT AND ANCILLARY FACILITIES

18.3.1 General

Project facilities will lie east of Patton Hill. The open pit mine is located between the headwaters of Casino Creek and Canadian Creek and will occupy an area of more than 300 ha. Figure 18-2 shows the overall project site layout.

A small valley about 1 km south of the pit, within the catchment of the Tailings Management Facility (TMF) area, will be filled with oxide mineralized material to form the heap leach pad. An earthen embankment at the eastern end of the pad will provide structural support for the heap leach. A spill and runoff control collection pond termed the “Events Pond” will be built directly downhill from the heap leach pad.

The TMF will be located southeast of the pit and plant site within the valley formed by headwaters of Casino Creek. It will store approximately 805 million tonnes (Mt) of tailings together with 615 Mt of potentially reactive waste rock. The TMF will include a water reclaim system to collect supernatant water and return it to the process.

There are two primary crushers: one for the oxide mineralized material and one for the sulphide mineralized material. The primary gyratory crusher pad where each primary crusher will be located will be constructed east of the pit. An overland conveyor belt will carry the sulphide mineralized material 0.86 km to the southwest from the crusher to the coarse mineralized material stockpile, which is located immediately northwest of the processing plant. The crushed oxide mineralized material will be carried to the heap leach by a separate overland conveyor system approximately 1.6 km long. The Oxide Crushing Facility consists of the Primary Crusher, a Secondary Crusher, and a Tertiary Crusher. The Secondary and the Tertiary Crushing Facilities have associated Screening Facilities.

Low-grade mineralized material will be placed in several temporary stockpiles adjacent to the plant site and crusher for processing in later years.

The concentrator consisting of the grinding mills, flotation circuits, reagent mixing, storage, and distribution systems, concentrate filter facility and thickeners will be located on an area of relatively flat terrain about 2 km south of the pit and crusher.

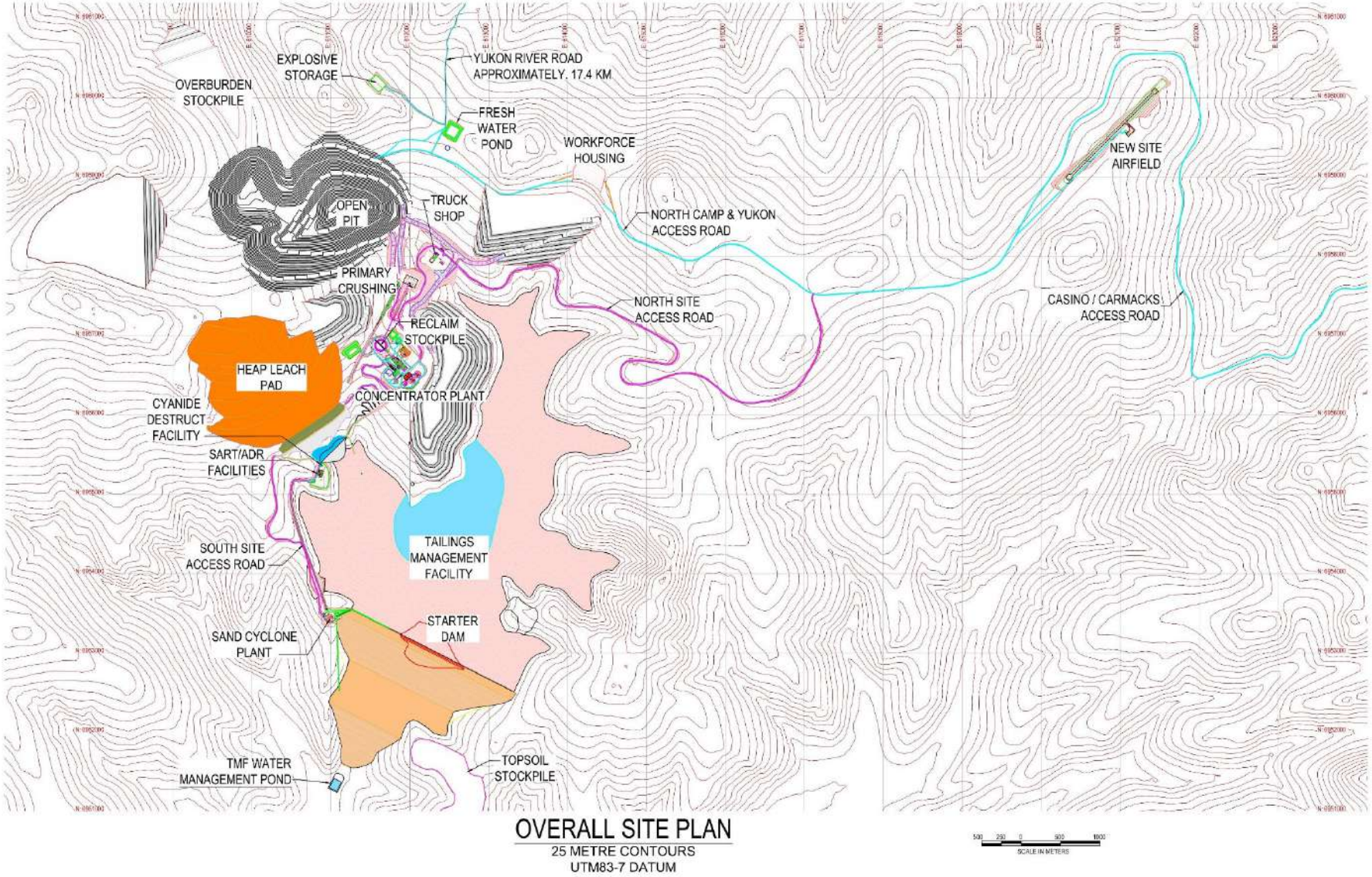


Figure 18-2: Overall Site Plan

18.3.2 Truck Shop

A truck shop and associated facilities will be constructed on a 4.5 ha pad adjacent to the open pit mines eastern exit. The other facilities located on the Truck Shop pad are the Truck Warehouse and the Truck wash. The Truck Shop also provides the alternate route for oversized vehicles that are too large to travel through the Haul Truck tunnel that provides access along the North Access Road the from the Carmacks/Casino Site Access road to the Casino Concentrator Plant.

18.3.3 Residence Camp

A nominal 1,400-person capacity construction and permanent camp will be installed. A 300-man, fabric structure, Pioneer Camp is to be constructed. The Pioneer Camp will be expanded by a 200-man dormitory expansion. The Permanent Main Camp will have approximately 774 “Jack and Jill” beds and approximately 108 Private suites for approximately 882 beds. Additionally, there will be construction camps for off-site construction provided by the contractor(s) for access road construction.

18.3.4 Operational Support Facilities

The majority of ancillary buildings are proposed to be located at the processing plant site. The Administration building will provide office space for both the construction effort and operations. The change house (mine dry) and laboratory buildings will be located near the mill and flotation buildings. A warehouse and laydown area will be provided for receiving and storage of parts and supplies, and for maintenance of plant mechanical and electrical equipment. A light vehicle maintenance building is proposed at the plant site apart from the truck shop. All of these buildings will be pre-engineered steel structures. See Figure 18-3.

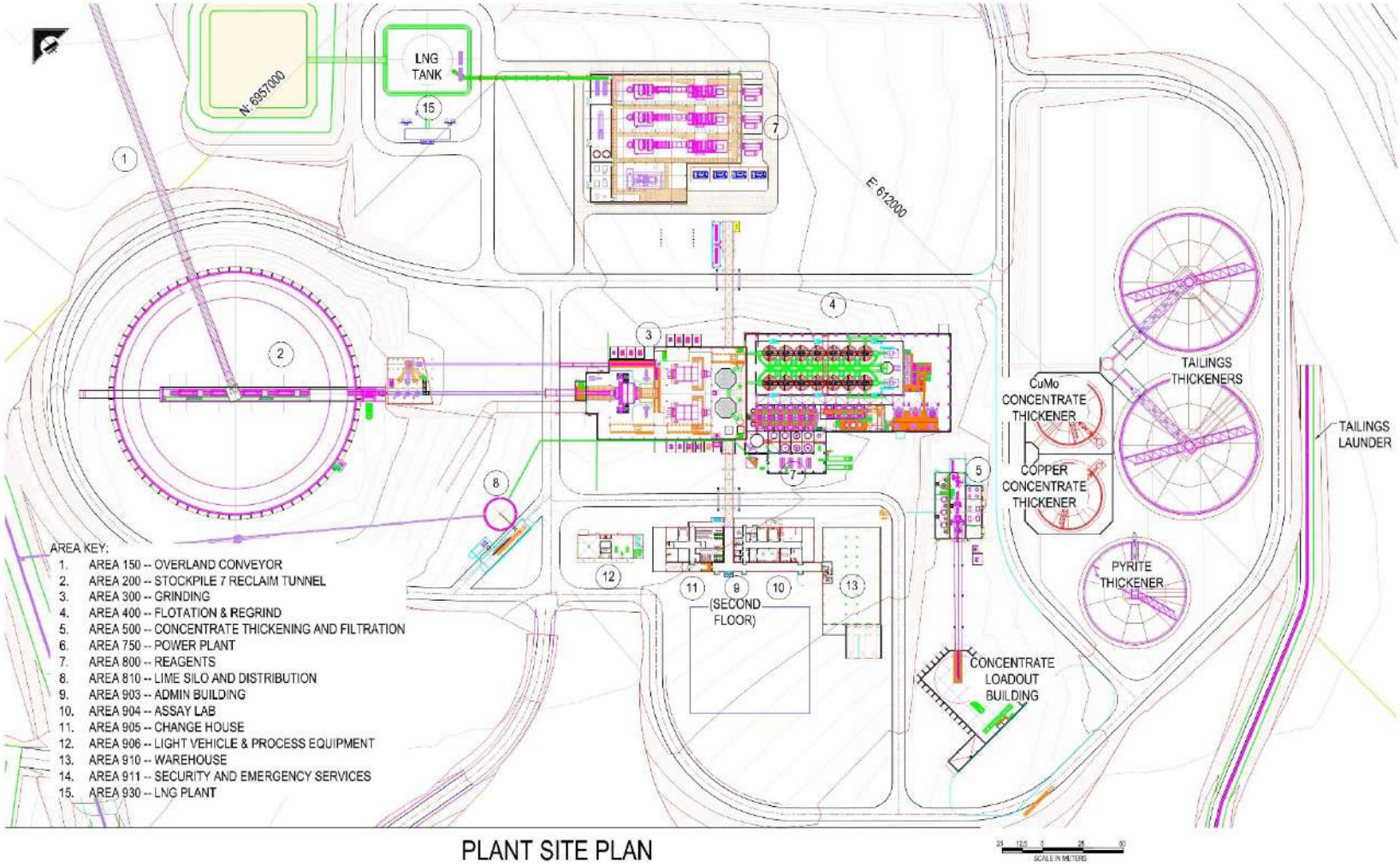


Figure 18-3: Plant Site Plan

18.3.5 Guard Shed/Scale House

A Guard Shed/Scale House at the facility entrance will provide 24/7 site security and truck scale service. The guard shed will be of modular construction.

18.3.6 Airstrip

The Casino Mine site is remote and the required workforce is expected to be approximately 1,400 with some operational personnel during construction and 600 to 700 during operations. Access to the site for this number of personnel will be best served by aircraft. The Project plan includes a new 1,600 m airstrip and pre-engineered air terminal building. The airstrip is planned to be located off the main access road into the site. It will be located approximately six kilometres east of the Permanent Camp Site.

Various aircraft have been considered to provide air service transport to the site. Aircraft considered suitable for this service includes the ATR 42 (capacity 42 passengers) and the Bombardier Dash 8- 200 series turbo-prop aircraft. Both aircraft are high-wing turboprop and well suited for the service conditions.

Flights to the site will originate from Whitehorse and other Yukon communities for the Yukon based workforce. Workers from other locations in Canada will connect with scheduled flights from Vancouver or other major centres.

The new airstrip site provides safe aircraft operations during all visible weather conditions. The airstrip design criteria have been developed to conform to the Transport Canada Aerodrome Standards and Recommended Practices (TP 312). The airstrip consists of a Code 3C non-instrument runway generally oriented northeast to southwest. It is 1,600 m long with 60 m overrun beyond the thresholds at each end for a total length of 1,720 m. The required runway width is 30 m, and the total graded width is 80 m. The airstrip embankment is raised a minimum of 1.5 m above the existing ground to provide a stable base and to prevent any ground ice from melting. There is a taxiway at the northeast end connecting to an apron for aircraft unloading and a parking area at the start of the airstrip access road. The new airfield is located on the right side of Figure 18-2.

18.4 PROCESS BUILDINGS

18.4.1 Crushing Plant

There will be two primary crushers: One crusher will do the initial crushing of the sulphide mineralized material, and the other will do the initial crushing of the oxide mineralized material. Each primary crusher will be housed within concrete structures. They will be placed only a few metres below existing grade in order to minimize blasting of bedrock. These structures will be surrounded by “U”-shaped Mechanically Stabilized Earth (Hilfiker or equivalent) retaining walls to form the truck maneuvering area around the dump hopper, roughly at the elevation of the exit from the pit.

18.4.2 Gold Recovery & SART Building

The oxide crushing facility will crush ore and transfer to the heap leach pad via a series of overland and grasshopper conveyors. Barren process solution will be applied to the oxide ore material lifts via irrigation lines. The irrigation lines will be buried to prevent freezing during winter conditions. Pregnant solution will be collected and pumped to the gold extraction plant for metal extraction and recycled for re-use in the leaching process. The ADR (Adsorption, Desorption and Recovery) and SART (Sulphidization, Acidification, Recycling, and Thickening) facilities will be located in a single structure.

18.4.3 Grinding

The Grinding Building (Mill) will house the 12.2 m (40 ft.) diameter SAG mill and two 8.5 m (28 ft.) diameter ball mills. The SAG mill and ball mill areas will each have a 90-tonne overhead bridge crane for maintenance.

18.4.4 Flotation/Reagent Storage & Mixing

The flotation circuits will be located within a structurally independent building adjacent and connected to the grinding building. Reagent storage and mixing facilities will be fully enclosed and placed adjacent to the flotation building. The 8,000-tonne lime silo will be located apart from the Mill/Flotation building.

18.5 WATER SUPPLY AND DISTRIBUTION

18.5.1 Fresh Water Supply

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above ground insulated 762 mm (30 in) diameter by 17.4 km long pipeline with four booster stations. The design flow rate of the system is 3,200 m^3/h.

The fire water requirement is 341 m^3/h for two hours. This demand is satisfied by designing the storage pond with a fire reserve capacity of 682 m^3 in the lower portion of the pond that will be unavailable for other uses.

18.5.2 Fire Water

The fire water requirement is 341 m^3/h for two hours. This demand is satisfied by installing a 10 m x 10 m storage tank that is kept full and ice free by passing the fresh water supply through. The entire Fresh Water Pond is an active backup for the fire system.

18.5.3 Potable Water

Potable water will be produced by filtering and chlorinating fresh water and will be stored and distributed separately.

18.5.4 Water Supply for the Leach, ADM and SART Facilities

Initial water requirements for the gold plant and oxide mineralized material heap leach operations will be met by pumping water retained behind a temporary cofferdam located in the TMF during construction of the starter dam. After start-up of the concentrator, process water will be provided from the TMF reclaim water system.

18.5.5 Process Water Supply

The NAG tailings is thickened to 55% solids in the Tailings Thickeners located at the concentrator. The underflow discharges through a 24” rubber lined steel pipeline down the steep slope. A pressure reducing choke station will be installed and the discharge will continue through a 48” concrete pipe launder to the Sand Cyclone Plant near the TSF. The cyclone underflow goes to the embankment and the overflow goes to the Sand Cyclone Thickener. The overflow of the Sand Cyclone Thickener reports to the Sand Cyclone Thickener Overflow Sump.

The Water Management Pond is located at the downstream toe of the TSF dam to collect seepage water. The pumps in the pond will deliver a design flow of 924 m^3/hr through a booster pump station to the Sand Cyclone Overflow Sump. There are two sets of pumps in the Sand Cyclone overflow sump. One set supplies Dilution water back to the Sand Cyclone Feed Sump. The second set of pumps delivers the balance of the overflow water to the reclaim system Booster Station No. 1. ADR plant will be fed from this stream.

The PAG Tailings is thickened to 45% solids in the Pyrite Thickener located at the concentrator. The underflow discharges through a 24" steel pipeline to a pressure reducing choke station and continues to the designated PAG area of the pond through HDPE piping.

There will be a total of three barges with three vertical turbine pumps per barge. Two barges will be in the PAG area, and one barge will be in the NAG area. The nominal flow for the barges is approximately 2500 m^3/hr. This would require four to six pumps operating out of the total of nine.

The flow from the barges will pass through two temporary relay pump stations to Booster Station No. 1.

Booster Station No. 1 delivers a nominal 3,458 m^3/hr to the Process Water Pond.

18.6 POWER GENERATION AND DISTRIBUTION

18.6.1 LNG Receiving, Storage and Distribution Facilities

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large 10,000 m^3 site-fabricated storage tank to provide fuel for the power plant. An LNG receiving station is provided to unload the LNG tankers and transfer the LNG into the storage facility. An LNG vaporization facility is provided to convert the LNG into gas at a suitable supply pressure to operate the power generation equipment.

18.6.2 Power Generation

Electrical power generation for the Project will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed in the vicinity of the main workforce housing complex to provide power to the camp, for construction activities, and to oxide crushing, conveying and heap leach facilities that go into operation before the main power plant is operational.

The Supplementary Power Plant will consist of three 2250 kW diesel internal combustion engines (ICE). Two of the generators will remain at the Workforce Housing complex and the third will be relocated to the Sand Cyclone (Area 640) facility to provide standby/emergency power to this area after the concentrator start-up.

A Main Power Plant will be constructed at the Casino main mill and concentrator complex to supply the electrical energy required for operations throughout the mine site. The primary electrical power generation will be provided by three Gas Turbine driven generators (two Single Fuel Gas Turbines, one Dual Fuel Gas Turbine) and a steam generator, operating in combined cycle mode (CCGT) with a total installed capacity of approximately 200 MW. The nominal running load to the mine and concentrator complex is about 130 MW. Three diesel internal combustion engine (ICE) driven generators will provide another 6.75 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines will be fuelled by natural gas (supplied as liquefied natural gas, or LNG). One of the three will have Dual Fuel capabilities - LNG and Diesel.

18.6.3 Power Distribution

The 34.5 kV distribution systems will radiate from a 34.5 kV switchgear line-up with feeders to the SAG mill, Ball Mill No. #1, Ball Mill No. #2, and feeders to the mill and flotation areas in cable tray using insulated copper conductors. Overhead line feeder circuits with aluminum conductor steel reinforced (ACSR) will be provided for the tailings reclaim water, fresh water from the Yukon River, crushing/conveying and SART/ADR, camp site and two feeders to the pit loop.

Electric power utilization voltages will be 4,160 volts for motors 300 horsepower (hp) and above, 575 volts for three-phase motors 250 hp and below. For lighting, small loads and building services 600/347 or 208/120 volts will be the utilization voltage.

18.7 TAILINGS MANAGEMENT FACILITY

18.7.1 Design Basis

The principal objectives for the design of the Tailings Management Facility (TMF) are safe and economic storage for tailings and waste rock, protection of the regional groundwater and surface waters both during operations and in the long term (post-closure), and to achieve effective reclamation at mine closure. The design of the TMF addresses the following requirements:

- Permanent, secure, and total confinement of all tailings and waste rock materials within an engineered facility
- Placement of potentially acid generating (PAG) tailings remote from the main embankment
- Maintaining surface drainage (including stormwater runoff) remote from the main embankment to the extent practical
- Control, collection, and removal of free draining liquids from the tailings during operations for recycling as process water to the maximum practical extent
- The use of hydrocyclones and vibratory dewatering screens to generate sand for embankment construction from the non-acid generating (NAG) tailings
- Monitoring the facility to confirm that the quantitative performance objectives (QPOs) are achieved, and the design intent is met
- Staged development of the facility over the life of the Project
- An end land use that meets the objectives of the affected First Nations, governments, and community stakeholders

The TMF impoundment comprises two confining embankments (Main, West Saddle), three waste storage areas (West Dump, North Dump, Divider Berm), two tailings cells (NAG, PAG), the cyclone sand plant, a cyclone overflow tailings thickener, tailings distribution pipelines, reclaim water systems, supernatant (surface water) pond, and water management systems (collection ditches, under-drains, water manage ponds and seepage recycle system).

- The Main Embankment will comprise a valley-fill impoundment developed in stages throughout the life of the mine using a combination of suitable quarried rock fill borrow materials and cyclone underflow sand. The initial Starter Embankment will store runoff as a fresh water source for mill start-up water during pre-production and accommodate tailings and PAG waste rock and overburden during the initial two years of operations (until the end of Year 2). Cyclone underflow sand will be used to construct ongoing raises of the Main Embankment, starting in Year 1, and continuing throughout the operating life of the Project.
- The West Saddle Embankment will be constructed in pre-production (starting in Year -3) to provide a pipeline corridor at a suitable grade for tailings delivery to the cyclone sand plant and the TMF. The embankment will be constructed using suitable earth and rockfill materials and an upstream liner will be installed prior to it being required for tailings and water storage in approximately Year 10.
- The West Waste Dump (WWD), North Waste Dump (NWD) will be developed within the TMF. The waste dumps will be developed concurrently with TMF filling to provide a dry, stable placement surface. The dumps will be maintained at an elevation above the supernatant pond with the eventual submergence by tailings by the end of operations.
- The Divider Berm will be constructed with waste rock and overburden to separate the NAG and PAG tailings within the TMF. It will provide access to the eastern extent of the project site and a dry, stable surface for the tailings distribution pipeline corridors.

- The PAG tailings line will discharge into the PAG Tailings Cell. Valved tees located along the line will enable deposition to be controlled to promote saturated conditions within the PAG Tailings Cell.
- NAG tailings may also be stored within the PAG Tailings Cell to promote even filling and minimize the hydraulic gradient between NAG and PAG tailings Cells.

The NAG Tailings Cell will primarily store the thickened cyclone sand plant overflow. Non-acid generating whole tailings from the mill during periods of cyclone plant maintenance, as well as screening losses from the tailings dewatering plant will also be discharged to the NAG Tailings Cell. The same piping system will be used to discharge the cyclone overflow stream and the undiluted Bulk NAG tailings to the TMF when the cyclone station is bypassed. Tailings discharge into the TMF will be from a valved spigots along the embankment crest. Valves will be spaced at approximately 100 metre intervals with rotation between off-takes as appropriate for good beach development within the NAG cell and to maintain a maximum tailings beach length between the supernatant pond and the main embankment.

18.7.2 Tailings Characteristics

Mining of the open pit will yield approximately 1.2 billion tonnes (Bt) of mineralized material. The mill will operate at a nominal throughput of approximately 120,000 tonnes per day (tpd), producing two primary tailings materials:

- 24% of the mill tailings are predicted to be Potentially-Acid Generating (PAG) that will be maintained in a sub-aqueous state within the TMF impoundment.
- 76% of the mill tailings are predicted to be NAG and will be used to produce sand for construction of the TMF Main Dam. The NAG tailings are relatively coarse with an average P80 of approximately 200 µm (the underflow has a P80 of 240 µm) which makes them amenable to efficient sand recovery with cyclones and dewatering screens.

It is estimated that approximately 50% of the NAG tailings can be recovered as cyclone underflow sand that is suitable for embankment construction when the cyclone plant is operating. The remaining 50% of NAG tailings (cyclone overflow) will be thickened prior to being discharged into the TMF impoundment. It is estimated that 85% of the underflow sand that reports to dewatering screens will be recovered; the remaining material lost through the screens will be thickened with the cyclone overflow and directed to the TMF NAG Tailings Cell. A schematic flow diagram showing the tailings mass balance split and distribution is shown on Figure 18-4.

Cyclone sand for embankment construction will initially be produced with a fines content (% passing a #200 sieve) of approximately 12% to 15% to achieve the required permeability and drainage characteristics. This criterion may be adjusted with ongoing experience and data collected during operations; other similar operations have been able to meet the design objectives with fines contents of 18% or greater (Barrera et al, 2011).

The current particle size distribution of the bulk NAG tailings (approximately 58% sand fraction) and information provided by cyclone suppliers indicates that single stage cycloning will be sufficient to achieve a suitable sand product with recovery of approximately 50% by weight from the bulk tailings stream. The cyclone feed will require water addition to dilute the solids content to approximately 36% and the sand (cyclone underflow) product will be produced at approximately 74% solids by weight. This will require dilution to a solids content of approximately 65% for distribution to the hydraulic cells.

Cyclone overflow tailings are described as an inorganic, non-plastic silt with sand (23% sand, 77% fines) with a liquid limit of 24%. Cyclone underflow tailings are described as a non-plastic sand with some silt (85% sand, 15% fines) with a specific gravity of 2.8. Additional laboratory testing included slurry consolidation permeability tests for the cyclone overflow tailings, and standard proctor and permeability tests for the cyclone underflow tailings.

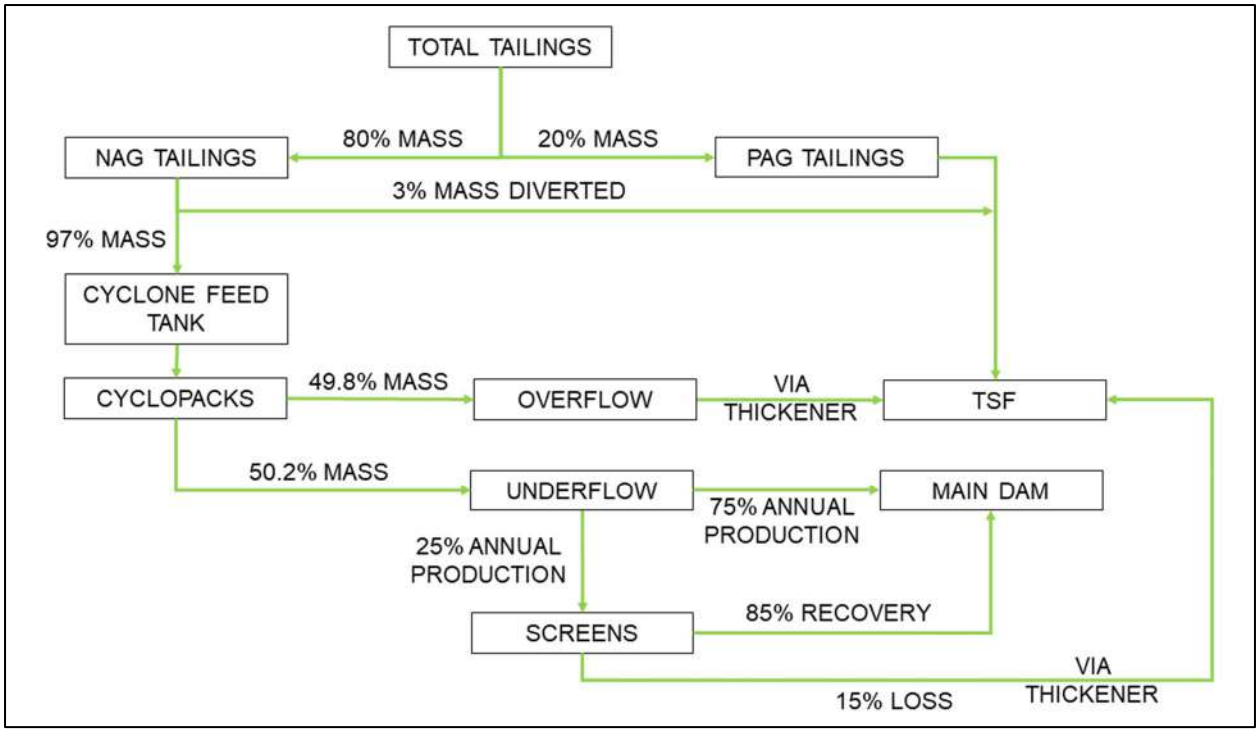


Figure 18-4: Tailings Conceptual Mass Balance

18.7.3 Storage Requirements

The storage requirements are based upon the mine plan developed by Independent Mining Consultants Inc. (IMC) dated April 5, 2022. The TMF is sized to provide sufficient capacity to store approximately 805 Mt of tailings and 615 Mt of PAG waste rock and overburden materials. The remaining 491 Mt of NAG tailings (cyclone underflow) will be used for embankment and buttress construction. Additional freeboard allowances are included in the design to manage the normal operating pond volume, seasonal inflows, and extreme storm events.

18.7.4 Hazard Classification

The Canadian Dam Association Dam Safety Guidelines (CDA, 2013; CDA, 2019) was used to determine the dam hazard classification during operations and suggested minimum target levels for some design criteria, such as the inflow design flood (IDF) and earthquake design ground motion (EDGM) for the TMF. A dam classification of VERY HIGH was assigned for the TMF embankments for operations. A classification of EXTREME was selected for closure.

The following design flood and design earthquake were adopted for the TMF for operations:

- IDF (operations) – two thirds between 1/1000 return period and probable maximum flood (PMF) for a 72-hour duration
- IDF (closure) - PMF
- EDGM (operations and closure) – the greater of the 1/10,000 return period or the maximum credible earthquake (MCE).

Seepage and stability analyses were completed along the critical sections of the TMF embankment and Waste Dumps (i.e., ultimate stage, maximum embankment height). Design basis criteria and model results are presented in the TMF Design Report (KP, 2022a).

18.7.5 Facility Design

Starter Embankment

The Starter Embankment will comprise an approximately 138 m high embankment (crest to downstream toe) constructed to an elevation of 845 m. The Starter Embankment will provide tailings, water, and waste rock containment until the end of Year 2 to allow enough cyclone underflow sand to be produced and placed to meet subsequent embankment raise requirements. The embankment shell zones for the Starter Embankment will require approximately 20.7 million cubic metres (Mm^3) of fill material for construction, sourced from suitable earth and rockfill material quarried from within the TMF basin; approximately 11.6 Mm^3 will be required during pre-production and approximately 9.1 Mm^3 will be placed in Year 1. Zone C1 is a selectively sized common fill material and Zone C2 is general rockfill.

The Starter Embankment will be constructed in downstream raises with 2.5H:1V upstream and 2H:1V downstream slopes. The upstream face of the embankment will be lined with a linear low-density polyethylene (LLDPE) geomembrane. A grout curtain will be installed along the upstream toe of the embankment, along with construction of a concrete plinth, and 10 m wide benches on the upstream face of the embankment to facilitate installation and tie-in of the LLDPE geomembrane. A liner bedding material and non-woven geotextile will be placed on the upstream face of the embankment prior to installation of the geomembrane. The grout curtain and geomembrane will limit seepage from the facility during initial operations when water storage may be required. A seepage collection system will collect and convey water away from the Starter Embankment. A conceptual section of the Starter Embankment is shown on Figure 18-5.

The Starter Embankment foundations will require clearing and stripping in preparation for fill placement. An average stripping depth of 0.3 metres has been assumed within the embankment footprint. Unsuitable material within the embankment foundation, including colluvial apron or other ice-rich overburden, will be excavated to competent foundation (i.e., sound bedrock), absent of frost susceptible soil. The average thickness of the colluvial apron and other ice-rich overburden is expected to be approximately 20 metres within the valley bottom and 2.5 metres along the valley sides based on the findings of site investigations. The excavation slopes within overburden will be laid back to maintain stability during construction. Localized overburden slope stabilization may be required in specific areas. Potential stabilization methods include grouted soil nails and groundwater depressurization and drainage.

The excavated material is assumed not to be suitable construction material and will be placed within the Divider Berm between the PAG and NAG Tailings Cells prior to establishment of a NAG tailings beach. Approximately 9.1 Mm^3 of unsuitable material will be excavated as part of the Starter Embankment foundation preparation.

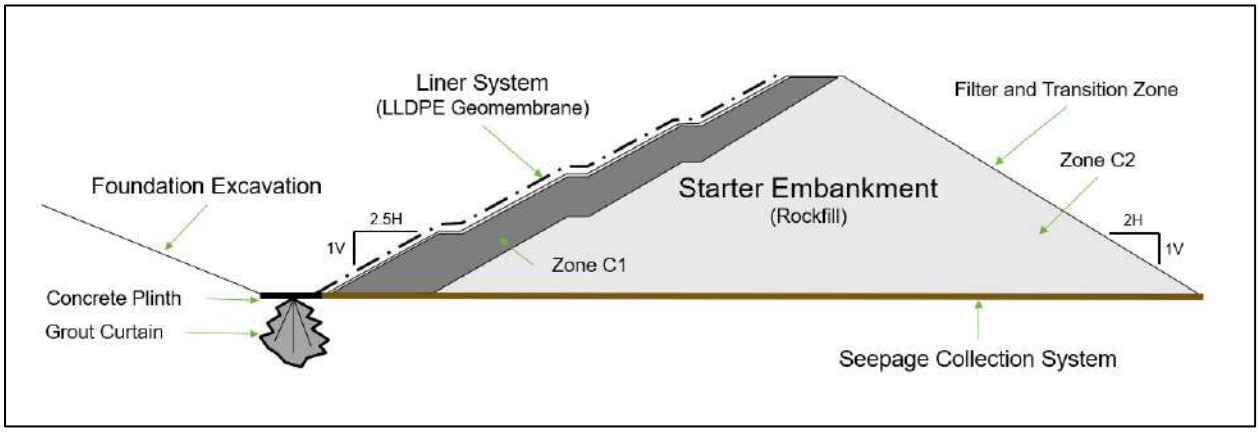


Figure 18-5: Tailings Management Facility Starter Embankment – Conceptual Section (KP, 2022a)

Main Embankment and Buttress

Cyclone overflow sand production beginning in Year 1 of operations will be used to extend the downstream shell of the TMF embankment, allowing the embankment crest to be raised above the Starter Embankment after approximately two years of operations. The sand will be placed directly in hydraulic cells or dewatered with vibratory screens for subsequent mechanical placement. Cyclone sand will be used to construct centerline raises of the embankment from Year 1 throughout operations to an ultimate crest elevation of 1000 m with a maximum embankment height of approximately 315 m (crest to downstream toe). The cyclone sand will be well-drained to progressively develop a zone of drained tailings solids within the facility along the embankment. A drainage zone underneath the maximum embankment section, with connection to the embankment abutments through regularly spaced toe drains, will facilitate drainage of the cyclone sand. The Main Embankment will be constructed with a minimum downstream slope of 3.5H:1V. Surplus sand produced through cycloning and screening operations will be used to construct a buttress for improved and robust embankment stability beyond the minimum requirements and to reduce the ultimate volume of tailings impounded within the TMF. A total of approximately 172 Mm^3 of cyclone sand will be used in construction of the Main Embankment, with an additional 56 Mm^3 of cyclone sand available for the buttress. A conceptual section of the Main Embankment is shown on Figure 18-6.

The TMF Main Embankment and Buttress foundations will be prepared in the same manner as the Starter Embankment foundation, including clearing, stripping, and excavation of unsuitable material. The excavated material will be placed within the Divider Berm. Approximately 20 Mm^3 of unsuitable material will be excavated as part of the Main Embankment and Buttress foundation preparation.

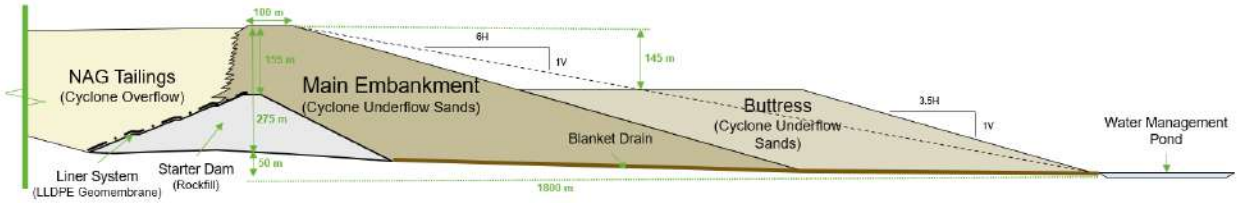


Figure 18-6: Tailings Management Facility Main Embankment – Conceptual Section (KP, 2022a)

The Main Embankment will be primarily constructed using hydraulic sand placement in cells to allow a high level of compaction. The hydraulic cell method involves the construction and maintenance of long, narrow cells (nominally assumed to be approximately 100 m by 20 m) along the embankment. Cyclone sand will be discharged hydraulically into each construction cell to depths of 0.5 m to 2.0 m and will be heavily track-packed by dozers and compacted with vibratory rollers if needed to achieve the target density. Cyclone underflow pipelines will be routed along pipeline benches above the sand cell operation or along confining flow control berms. Drainage recovery ditching will be routed to localized sumps for collection and transport to water collection ponds. Water expelled from the sand will drain via ditches away from the cells to the water management pond at the toe of the embankment and recovered to the TMF area. The in-situ density, moisture content, and fines content will be verified with quality control and quality assurance programs throughout construction and operations.

Construction of each of the embankment stages will correspond with cyclone sand availability and the storage requirements. Sand cell placement operations occur simultaneously in two or more cells to match sand cyclone plant operations. The cyclone plant and hydraulic sand placement activities are assumed to be operational for an average nine months of the year (with a 90% sand plant availability to account for maintenance and other downtime). Hydraulic placement of cyclone underflow sand is not anticipated to be practical for approximately three months of the year due to freezing winter temperatures. During this period, the coarse cyclone underflow will be processed using dewatering screens during the winter months to produce sand that can be transported by conveyor to the Main Embankment where

it will be stockpiled or used directly for construction. Screen sand will be placed, spread, and compacted to meet the density specifications.

A schematic of the embankment construction, including hydraulic cell sequencing, is shown on Figure 18-7.

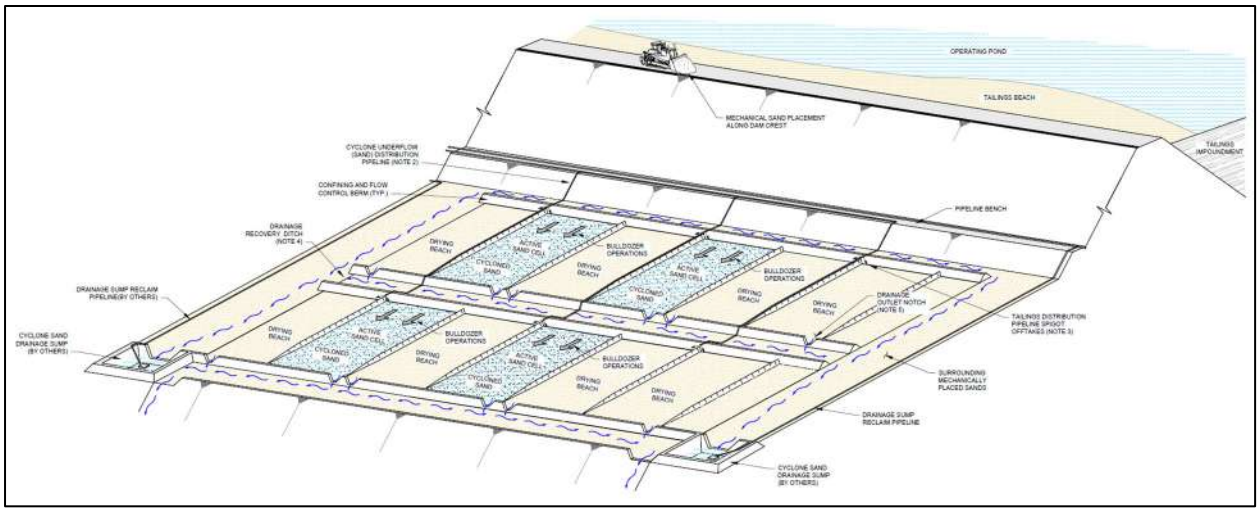


Figure 18-7: Main Embankment Construction – Conceptual Schematic (KP, 2022a)

West Saddle Embankment

The West Saddle Embankment will be developed in pre-production years using non-acid generating waste rock and overburden material available from initial open pit stripping operations or earth and rockfill from local borrow sources. The embankment will be raised to an elevation of 1,000 m with a maximum embankment height of approximately 40 m (crest to downstream toe). The upstream face of the embankment will be lined with a linear low-density polyethylene (LLDPE) geomembrane system similar to the Main Embankment.

The West Saddle Embankment foundation will be cleared and stripped in preparation for fill placement. Information collected during the site investigation programs indicate that the depth to bedrock throughout the area is approximately 2.5 metres and the shallow overburden likely comprises residual and colluvial soils. Areas of unsuitable material within the embankment foundation, including colluvial apron or other ice-rich overburden, will be excavated to competent foundation, absent of frost susceptible soil.

Water Management System

The Water Management System for the TMF has been designed to accommodate seepage through the TMF embankment, water recovered from the cyclone sand embankment construction, and surface runoff. The Water Management System consists of an under-drain system and a surface ditch system that discharge into a Water Management Pond located downgradient of the Main Embankment. The under-drain system comprises the following components:

- A 1 x 1 m² cross-sectional gravel seepage collection system with 300 mm diameter perforated collection pipes along the upstream toe and maximum cross-section alignment of the Starter Embankment to collect and convey drainage and seepage away from the Starter Embankment.

- A 150 x 6 m² cross-sectional gravel drainage blanket to drain the Main Embankment foundation to the downstream Water Management Pond. The blanket drain will be extended in two or more phases as the footprint of the embankment increases with staged embankment expansions.
- A series of 10 x 4 m² cross-sectional gravel drainage finger drains, to collect and convey embankment construction water and infiltrated surface water towards the blanket drain. The finger drains will be extended with the footprint of the Main Embankment throughout construction.

A filter and transition zone will surround both the blanket and finger drains to prevent cyclone sand and fines migration. The blanket and finger drains will be preferentially graded to promote gravity drainage. Excavation below the prepared foundation surface may be required to achieve the design grades.

The surface ditch system has been configured as a dual-ditch design. Two parallel ditches will be constructed: upgradient ditches for 'non-contact water' and downgradient ditches for 'contact water' generated by either precipitation or cyclone sand placement. The system will reduce the quantity of water that requires storage in the Water Management Pond and pumping back to the TMF. Scour protection of the ditching will be required to prevent erosion of the ditch channels. The ditch system will be developed in phases to accommodate staged embankment expansions, similar to the underdrain system.

Water management ponds will operate with minimal storage and are sized to contain the design storm event. An interim stage water management pond, sited immediately downstream of the Main Embankment footprint, will be actively managed for the first ten years of operations, during early stages of embankment expansion. A final water management pond, sited immediately downstream of the buttress footprint, will be actively managed from approximately Year 10 to the end of operations. A pump station will return the water from the water management pond back to the TMF. Supplemental booster pump stations will be required as the TMF embankment height increases.

Reclaim Water Systems

The Starter Embankment will impound water for mill commissioning and start-up during pre-production. Mill process water for ongoing operations and water for cyclone plant operations will be reclaimed from the TMF supernatant pond TMF water management ponds, and a thickener installed adjacent to the cyclone station that receives cyclone overflow and recovers water to sustain the cyclone operation thereby eliminating this volume from circulating through the TMF basin. The thickener will produce an underflow at approximately 50% by weight solids which is directed to the TMF. The Process water demand is supplemented by the Yukon River fresh water supply system. The fresh water supply system is designed to offset any predicted water deficit conditions during mine operations and is sized to supply all the process water requirements, if needed.

Water reclaimed from the TMF consists of supernatant from the settled tailings, runoff from precipitation, and snowmelt within the TMF catchment area. Pipelines will convey the reclaimed water to the settlement pond located upgradient of the mill and a water head tank upgradient of the cyclone plant.

The TMF is designed as a zero-surface water discharge facility during the operating life of the facility. A water treatment facility is installed adjacent to the thickener as a facility to treat and release tailings supernatant in the event a positive water balance arises, that is beyond design parameters, and requires a water release to restore the operational water balance.

18.7.6 Water Balance

A life-of-mine (LOM) water balance model (WBM) was completed to assess surplus and/or deficit conditions at the Casino site throughout the full life cycle of the Project. Water balance model objectives, design basis criteria and model results for the tailings management facility are presented in the TMF Design Report (KP, 2022a).

18.7.7 Instrumentation and Monitoring

Geotechnical instrumentation, comprising vibrating wire piezometers, thermistors, slope inclinometers, and movement (survey) monuments will be installed at selected planes along the Main Embankment and West Saddle Embankment during construction and over the life of the project. Additional remote sensing methods such as drone-based imagery and topographic data, satellite imagery, and InSAR may also be used for TMF monitoring. The instrumentation will be monitored during the construction, operation, and after closure of the TMF to assess embankment performance against the Quantitative Performance Objectives (QPOs) and to identify any conditions different to those assumed during design and analysis. Monitoring of civil and mechanical operational components will be a daily/shift requirement, to confirm that the systems are functioning correctly. Amendments to the on-going designs and/or remediation work can be implemented to respond to the changed conditions, should the need arise.

On-going water quality monitoring will be required to confirm the effectiveness of the Water Management System including the installation of groundwater wells at suitable locations downstream of the embankment footprint.

In addition to the routine inspections carried out by mine personnel on a shift/daily/weekly and monthly basis, the TMF will be regularly (e.g., annually) inspected by a qualified Geotechnical Engineer to ensure it is operating in a safe and efficient manner. Additionally, a formal dam safety review will be conducted every 5 years. The dam classification is reviewed during the dam safety review process and should also be routinely reviewed as part of the annual audit.

The Independent Engineering Review Panel (IERP) that was engaged prior to the Feasibility Study will be maintained throughout the mine life and into Closure, as necessary.

18.7.8 Closure and Reclamation

The primary objective of the closure and reclamation initiatives will be to eventually return the TMF site to a landform with pre-mining usage and capability. The TMF will be required to maintain long-term stability, protect the downstream environment, and safely manage surface water. Upon mine closure, surface facilities will be removed in stages and full reclamation of the TMF will be initiated. General aspects of the closure plan include:

- Selective discharge of NAG tailings or placement of other suitable materials around the facility during the final years of operations to establish a final tailings surface and water pond that will facilitate post closure surface water management and reclamation. The NAG tailings or other suitable material will be used to encapsulate and maintain the PAG tailings and waste rock in a saturated state.
- Dismantling and removal of the tailings and reclaim delivery systems, cyclone plant and all pipelines, structures and equipment not required beyond mine closure.
- Construction of semi-passive treatment system if required to treat water within TMF or Water Management Pond areas
- Removal of the seepage collection system at such time that suitable water quality for direct release is achieved.
- Construction of an overflow spillway and channel to allow surface water discharge downstream of the TMF.
- Removal and regrading of all access roads, ponds, ditches and borrow areas not required beyond mine closure.
- Long-term stabilization of all exposed erodible materials.
- During the closure phase of the project, suitable cover material and seeding will be placed on the beach surface after tailings deposition ends to promote revegetation and minimise dusting potential.

The water balance model developed for the TMF indicates that there will be a water surplus at closure. The spillway will be constructed at closure to safely route and discharge excess water accumulation within the TMF and to provide safe passage of stormwater volumes from the TMF. The closure spillway will be located close to the west abutment of

the West Saddle Embankment to provide long term protection against overtopping of the TMF during the post closure years. The spillway will be routed from the TMF to the south, ultimately terminating at an erosion protected plunge pool at the confluence of Brynelson Creek. The closure spillway will be comprised of a channel excavated in bedrock below the crest of the embankment. Sections of the spillway channel and plunge pool excavated within overburden will require armoring to prevent erosion.

The seepage collection ponds/sumps and recycle pumps will be retained until monitoring results indicate that seepage collected from the TMF at the Water Management Pond is of suitable quality for discharge to the downstream environment or a semi-passive treatment system has been commissioned, if required. The groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long-term monitoring devices.

Post-closure requirements will also include scheduled inspections of the TMF and an on-going evaluation of water quality, flow rates and instrumentation records to confirm the design assumptions for closure.

18.8 HEAP LEACH FACILITY

18.8.1 Design Basis

Heap leach facility operations will commence during pre-production stripping of the open pit. The heap leach pad will be stacked with ore and leached simultaneously from Year -2 through Year 22 of mine operations. The ore will be leached at a nominal rate of 9.125 Mt per year at a stacking rate of approximately 25,000 tpd. The proposed HLF is located on a southeast facing hillslope, approximately 1 km south of the open pit.

The leach pad will comprise a fully contained gold ore treatment facility. The leaching process will involve the irrigation of weak cyanide solution over successive lifts of heaped ore. The primary design objectives for the proposed heap leach pad are as follows:

- Provide a stable and cost-effective configuration for staged heap development
- Effectively collect and convey leachate solutions to the process plant or the events pond while ensuring maximum recovery
- Provide for secure containment of events solutions while monitoring and minimizing losses due to leakage
- Minimize surface runoff entering the leach pad area while providing for the collection of direct runoff from the heap area
- Sequential, staged development and leaching operations with particular emphasis on winter operations, and
- Effective decommissioning and reclamation of all heap leach facility components.

Gold ore to feed the HLF will be located in a temporary stockpile east of the open pit. The ore will be crushed and transported to the pad by conveyor. The ore will be placed on the pad for approximately 300 days each year. The stacked ore will be irrigated with cyanide solution year-round using a drip-type irrigation system.

The HLF consists of the following system components:

- A confining embankment, located at the toe of the proposed heap leach pad, to improve stability of the heap leach pad, collect leachate solution, and store/attenuate storm runoff prior to spilling to the Events Pond.
- A composite liner system consisting of ‘barrier’ and ‘drainage’ layers using a combination of synthetic liner (LLDPE) and compacted soil liner to maximize pregnant solution recovery and minimize leakage losses through the bottom and sides of the heap leach pad.
- A leachate solution collection system (LSCS) consisting of collection pipes and sumps designed to streamline solution collection and conveyance off the leach pad.

- A leak detection and recovery system (LDRS) designed to capture and convey any solution which leaks through the composite liner system via a combination of drainage layers, collection pipes, and drainage trenches. Leakage recovered by the LDRS will be conveyed into the LDRS sump at the toe of the confining embankment.
- A leach solution application system (LSAS) to irrigate the heap leach pad with chemical solution to leach the gold from the stacked leach ore.
- An events pond designed to provide storm storage that exceeds the in-heap storage capacity of the HLF. The pond is situated immediately down gradient of the HLF confining embankment, and the pond inflow is conveyed via the confining embankment spillway.
- A perimeter berm with diversion and runoff collection ditches intercept overland surface runoff around the HLF pad and to convey non-contact water flows to the TMF, and contact water flows to the in-heap events pond.

The HLF is shown in plan view on Figure 18-8, and in section view on Figure 18-9.

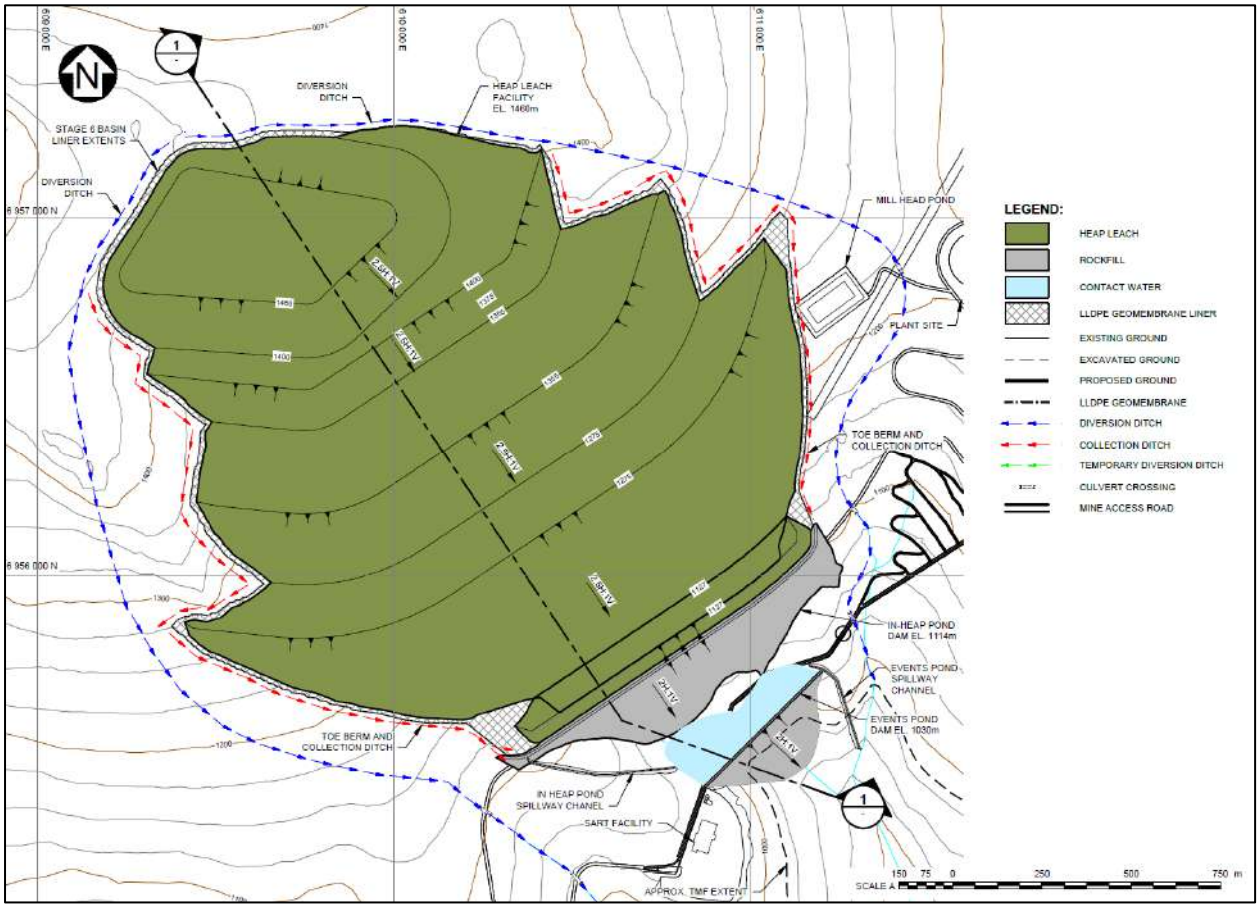


Figure 18-8: Heap Leach Facility General Arrangement (KP, 2022b - in progress)

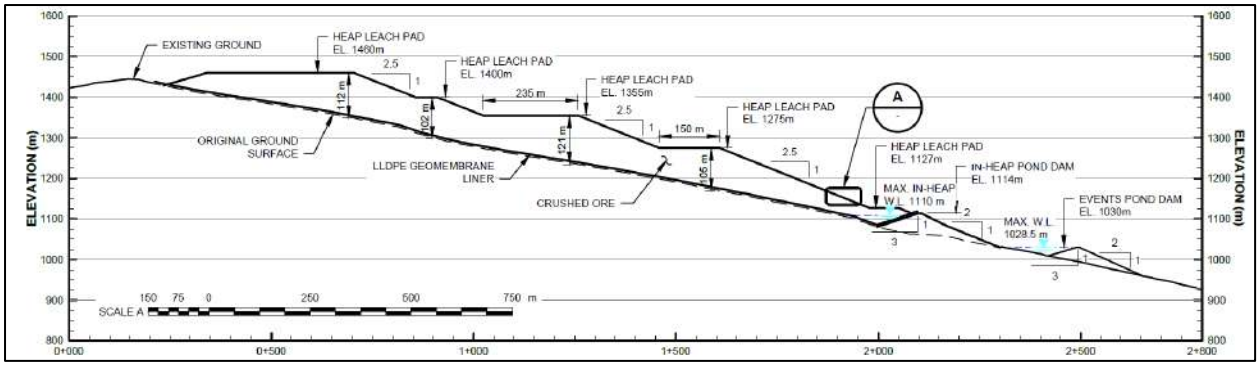


Figure 18-9: Heap Leach Facility Cross-Section (KP, 2022b - in progress)

18.8.2 Storage Requirements

The ore stacking schedule for the heap leach pad has been designed as 'development' stages, with each stage requiring advance expansion of the pad footprint. The HLF will be constructed in six 'pad development' stages, with the pad foundation preparation, liner installation and leachate collection piping developed as the footprint of the leach pad expands upslope to accommodate additional ore lifts. The duration for each stacking stages ranges from two to four years. Approximately 210 million tonnes (Mt) of mined ore will be processed at the Heap Leach Facility (HLF)

The initial HLF development (Stage 1) will also include the complete development of the confining embankment and the events pond prior to commencing ore stacking and leaching.

18.8.3 Facility Design

Heap Leach Pad

The heap leach pad is located on a uniformly sloping sidehill and has an approximate footprint area of 2.8 km^2. The heap leach pad is designed to be operated predominantly as a dry heap-leach facility with minimal leachate storage to occur behind the confining HLF embankment.

The heap leach pad will be developed in stages by loading in successive lifts upslope from the platform developed within the in-heap pond area behind the HLF confining embankment. This will provide initial stability and minimise initial capital costs. The pad will be developed in eight-metre lifts constructed at repose bench face angles of approximately 1.4H:1V. Bench widths approximately nine metres wide will be left at the toe of each lift to establish a final overall slope of 2.5H:1V. Intermittent wider benches will be constructed to limit the vertical height of the HLF along its profile to a maximum of 121 m (crest to downstream toe).

The heap leach pad and confining embankment foundation footprints will be cleared of vegetation, grubbed and the topsoil removed and stockpiled for use in reclamation. Frozen colluvial soils and weathered bedrock (residual soils) will be excavated down to a competent, stable bedrock foundation. A 3.5 m excavation depth has been estimated for foundation preparation to competent ground, based on site investigation date. Slopes in the heap leach pad area will be re-graded to a maximum 3H:1V slope with a dozer (as required) for liner installation. Any ice-rich materials will not be suitable for use as borrow materials for embankment construction, and therefore will be transported to local stockpile or the TMF impoundment.

Seepage and stability analyses were completed along the critical sections of the HLF facility (i.e., ultimate stage, maximum embankment height). Design basis criteria and model results are presented in the HLF Design Report (KP, 2022b - in progress).

Confining Embankment

The embankment will have a final crest elevation of 1,114 masl and will be constructed with an upstream slope of 3H:1V and downstream slope of 2H:1V. The embankment will be constructed from locally sourced rock and earth fill by placing the fill in lifts and compacting to a specified density. A 0.3 m thick bedding sand layer will be placed over the upstream slope of the embankment in preparation for installation of the LLDPE double liner system. The liner system anchored in a trench excavated along the crest of the embankment. A sloping leachate detection layer will also be constructed along the upstream face of the embankment. All liners will be tied into their corresponding liner along the foundation of the pad to provide a continuous seal zone.

Preparation of the embankment foundation will be undertaken in the same manner as the foundation preparation for the Heap Leach pad and will involve stripping the topsoil and excavating the underlying frozen colluvial and residual soils down to competent bedrock. The HLF confining embankment will require approximately 2.6 Mm^3 of embankment fill materials for construction.

The HLF will provide approximately 245,000 m^3 of in-heap storage (i.e., storage within the voids of the stacked mineralized material) behind the confining embankment for storage of leachate solution and contact water, with appropriate freeboard for a design storm event. In the event that the storage requirement is greater than the provided capacity, excess runoff will spill to the events pond downstream of the heap via the confining embankment spillway. The spillway channel will be excavated in bedrock and terminates in the Events Pond.

Liner Systems

Two liner systems have been developed for the Heap Leach pad, an engineered single liner design for the upper portion of the leach pad (above the in-heap leachate solution storage elevation) and a composite double liner design for the lower portion of the leach pad which will potentially have leachate solution storage.

The 'Upper', or single liner system consists of the following components:

- 1 metre thick overliner (2nd stage crushed and screened ore)
- 80 mil (2 mm) linear low-density polyethylene (LLDPE) geomembrane, and
- 0.3-metre-thick compacted low permeability soil liner.

The 'ponded', or double liner system is designed to be installed on the heap leach pad's lower slopes which may experience hydraulic loading from in-heap solution storage. The double liner system consists of the following components:

- 1 metre thick overliner (2nd stage crushed and screened ore)
- 80 mil (2 mm) linear low-density polyethylene (LLDPE) primary geomembrane
- 0.3-metre-thick compacted low permeability soil liner
- 0.6-metre-thick compacted Leak Detection sand layer
- Leak Detection and Recovery System (LDRS), and
- 80 mil (2 mm) linear low-density polyethylene (LLDPE) secondary geomembrane.

Development of the heap leach liner will be constructed in six stages, with liner expansions proposed every two to four years to meet ore stacking requirements. A protective layer approximately one metre thick of coarse crushed ore (from second stage crushing) will be placed over the entire liner system footprint to protect the liner's integrity from damage during ore placement.

The soil liner material may be sourced from local suitable overburden (residual and colluvial soils) and weathered bedrock. The liner system will require approximately 0.9 Mm^3 of earth fill and rockfill materials for construction. The total quantity of LLDPE liner required is approximately 2.9 km^2, approximately 285,000 m^2 of which is for the 'ponded' area.

Leachate Solution Collection System

Collection and recovery of the pregnant solution is undertaken by the Leachate Solution Collection System (LSCS) which works in conjunction with the heap leach liner, overliner, and Leak Detection and Recovery System (LDRS). Leachate will be collected in the Collection Header pipes, which then flow into the Main Header Collection pipes. The Main Header Collection pipes are positioned along the centre of the Heap Leach Pad and terminate at the upstream toe of the confining embankment at the three Leachate Collection Sumps, located at the toe of the confining embankment.

The sumps consist of two sections, the lower 'collection zone' and the upper zone. The lower zone consists of a three-metre-thick zone of clean screened gravel placed around a perforated steel vertical riser pipe. The upper zone consists of a three-metre-thick zone of compacted crushed ore placed around a non-perforated steel vertical riser pipe.

Leak Detection and Recovery System

The LDRS under the double lined area consists of a 0.9-metre-thick sand layer which is embedded with CPT collection pipes. The LDRS is sandwiched between the primary and secondary geomembrane layers in the ponded areas. The LDRS under the single lined area consists of a network of drainage 'trenches' which contain perforated CPT collection pipes surrounded by drainage sand. The trenches are aligned underneath the 'Collection Header' and 'Main Collection Header' pipes which are part of the Leachate Collection system embedded in the above overliner layer. These drainage trenches discharge into the LDRS layer underlying the double lined area in the lower heap leach portion. A single strip of geomembrane liner underlies the LDRS collection trenches for the non-ponded (i.e., single lined) area.

Leakage recovered by the LDRS will be conveyed into the LDRS sump at the toe of the confining embankment. A level-switch controlled submersible sump pump will transfer the recovered solution up the embankment slope via a pipe installed within the LDRS sand layer and connect into the main solution recovery line for processing. Monitoring of the leakage recovery will be undertaken through continuous monitoring of the pump hour records.

Events Pond

The Events Pond is designed to provide additional storage for flood events exceeding the design storm event of the HLF in-heap pond, as well as functioning as a seepage collection pond for leachate solution that may leak past the double composite liner system in the HLF in-heap pond area. The Events Pond will be operated as a dry pond during normal operations to have the maximum capacity available for storage of excess HLF surface runoff from storm events. Water collected in the Events Pond will be used to supplement the HLF water supply for irrigation. Water volumes exceeding the Events Pond storage capacity will be conveyed to the TMF via the Events Pond Spillway.

The Events Pond will be constructed to full size prior to commencing HLF operations. Construction of the Events Pond involves stripping of topsoil and excavation of 3.5 m of unsuitable overburden beneath the embankment and ponding footprint. The Events Pond embankment will be constructed of locally sourced earthfill and rockfill, with fine grained residual soils on the upstream face to provide a low permeability zone and an acceptable surface for installation of a HDPE double liner system. The embankment is designed with a 2H:1V downstream slope and a 3H:1V upstream slope. The embankment will be underlain with a 1 m thick drainage blanket layer to aid in maintaining unsaturated conditions in the embankment fill. The events pond embankment and ponded area will require approximately 1.1 Mm^3

of earthfill and rockfill materials for construction. The total quantity of liner required for the events pond is approximately 55,000 m^2.

Diversion and Collection Ditches

The ditches are designed to convey the 1 in 100-year 24-hour duration storm event. Lining and protection of the ditch channels from erosion and scouring is required for all permanent ditches due to the steep ditch grades associated with the natural topography and the anticipated high runoff flowrates.

18.8.4 Water Balance

The water balance indicates that the HLF will operate in a deficit condition, with makeup water sources required throughout the operation of the HLF.

18.8.5 Instrumentation and Monitoring

Geotechnical instrumentation will be installed to monitor the performance of the HLF during the construction stage and throughout the life of the facility. The purpose of the instrumentation will be to provide data to assess the stability of the heap leach pad and to evaluate the effectiveness and performance of the overliner and foundation drains.

The following instrumentation and monitoring are proposed:

- Piezometers will be installed to allow measurement of phreatic levels and pore water pressures within the HLF embankments and foundations drain.
- Surface water quality sampling at selected locations downstream of the HLF.
- Installation of monitoring wells around the facility to monitor groundwater quality during operations and at closure. These wells would be installed prior to development to obtain baseline information for comparative assessment.
- Installation of a LDRS to monitor and recover any leakage through the liner systems within the heap leach pad area and events pond.
- Slope movement monuments and survey control points installed and monitored to ensure the integrity and stability of the ore heap.
- The installation of flow monitoring devices in diversion ditches and creeks to confirm design flows.
- Review of thermistor data to confirm the thermal regime at the site.

Details of the instrumentation and monitoring plan will be developed in conjunction with the appropriate regulatory authorities. The instrumentation and monitoring plan will be summarized in an Operations, Maintenance and Surveillance (OMS) Manual for the HLF.

18.8.6 Closure and Reclamation

The reclamation plan for the HLF envisages the following actions:

- A rinse and drain-down period will occur to recover any final gold, and to remove or reduce residual cyanide and dissolved metals that could migrate out of the ore. As required, the water will be treated to ensure that the cyanide concentration meets the International Cyanide Management Code.
- Pipework and mechanical systems (including irrigation system, conveyors, stackers, etc.) will be removed for disposal.
- The Event Pond will be reclaimed and may be used to establish a semi-passive treatment system if required.

- The HLF confining embankment, events pond embankment, perimeter toe berm, and diversion and collection ditches will be levelled and regraded to a natural grade.
- All exposed, erodible surfaces will be covered and revegetated with native species.

The seepage collection ponds/sumps and recycle pumps will be retained until monitoring results indicate that any seepage from the HLF is of suitable quality for discharge to the open pit to aid flooding, to the TMF or direct release to downstream waters. The groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long term monitoring devices.

Post-closure requirements will also include inspections of the HLF and an on-going evaluation of water quality, flow rates and instrumentation records to confirm design assumptions for closure.

18.9 WASTEWATER DISPOSAL

Packaged sewage treatment plant systems will accept and treat all sanitary wastewater.

18.10 COMMUNICATIONS

No communications infrastructure exists in the area of the site. The Project will develop communications infrastructure to meet construction and operations requirements.

19 MARKET STUDIES AND CONTRACTS

No market studies were performed, and no sales contracts are in place. The commodities involved in this project are commonly traded on the open market.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section provides further details on environmental studies conducted to date, the environmental assessment process, territorial and federal regulatory approvals required to bring this property into production, and the status of First Nations' consultation and agreements. A brief overview of the closure concept and obligations is also provided.

20.1 ENVIRONMENTAL & SOCIAL STUDIES

Numerous environmental studies have been completed on the Casino property to support previous submissions to the Yukon Environmental and Socio-economic Assessment Board (YESAB) under the *Yukon Environmental and Socioeconomic Assessment Act* (YESAA). Environmental studies were conducted on terrain and terrain hazards, water quality and hydrology, geochemistry, hydrogeology, air quality, noise, fish and aquatic resources, rare plants and vegetation, and wildlife. Many of these studies were completed from 2012 through 2014, with bi-annual surface and groundwater monitoring and climate monitoring programs on-going, and monthly water monitoring occurring since May 2021. Socio-economic studies have been conducted focused on key socio-economic indicators. Figure 20-1 show the mine site and surrounding region. Figure 20-2 shows the mine site layout at the start of mine operations, and Figure 20-3 shows the mine site layout at the end of mine operations.

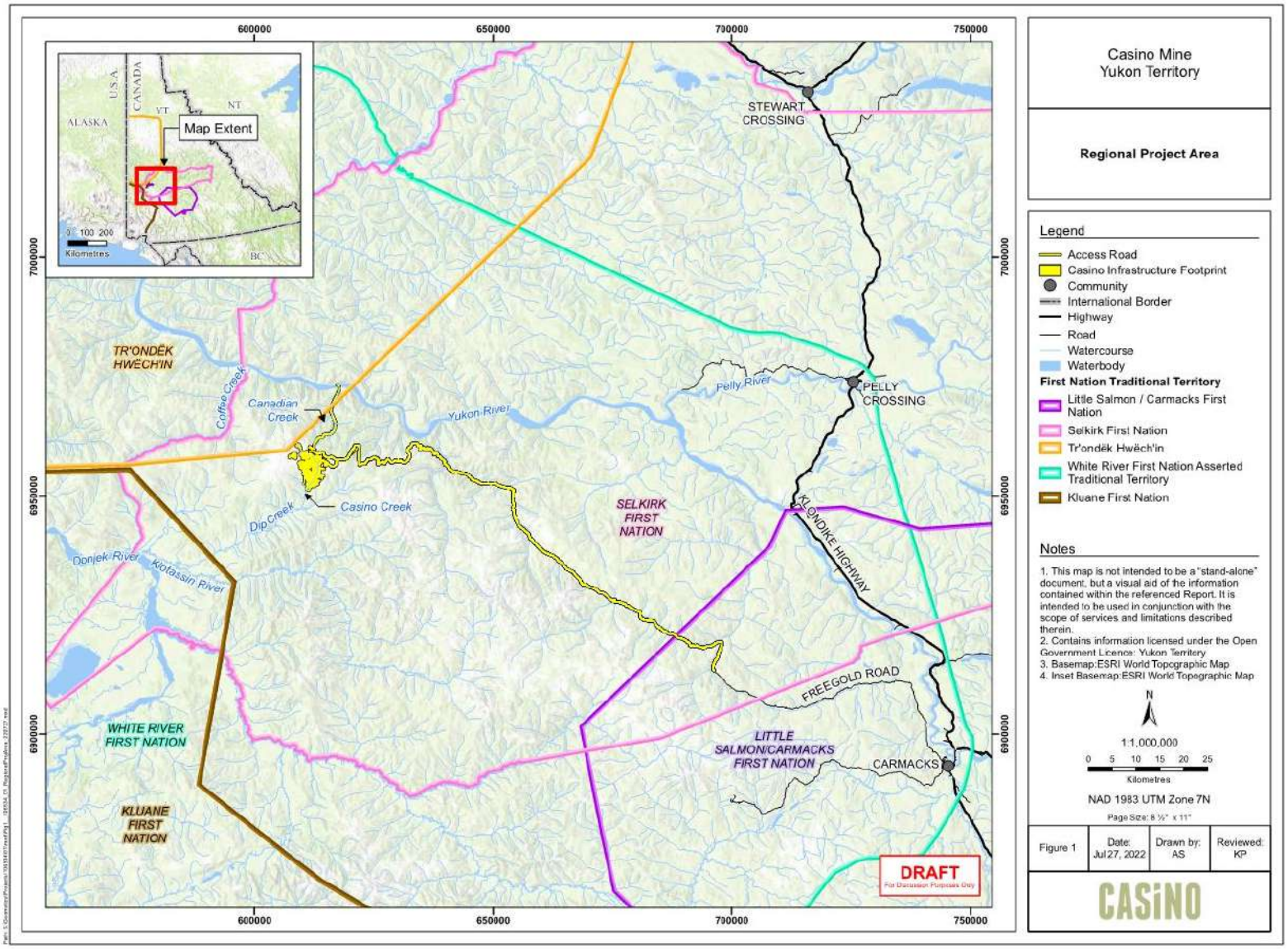


Figure 20-1: Regional Project Area

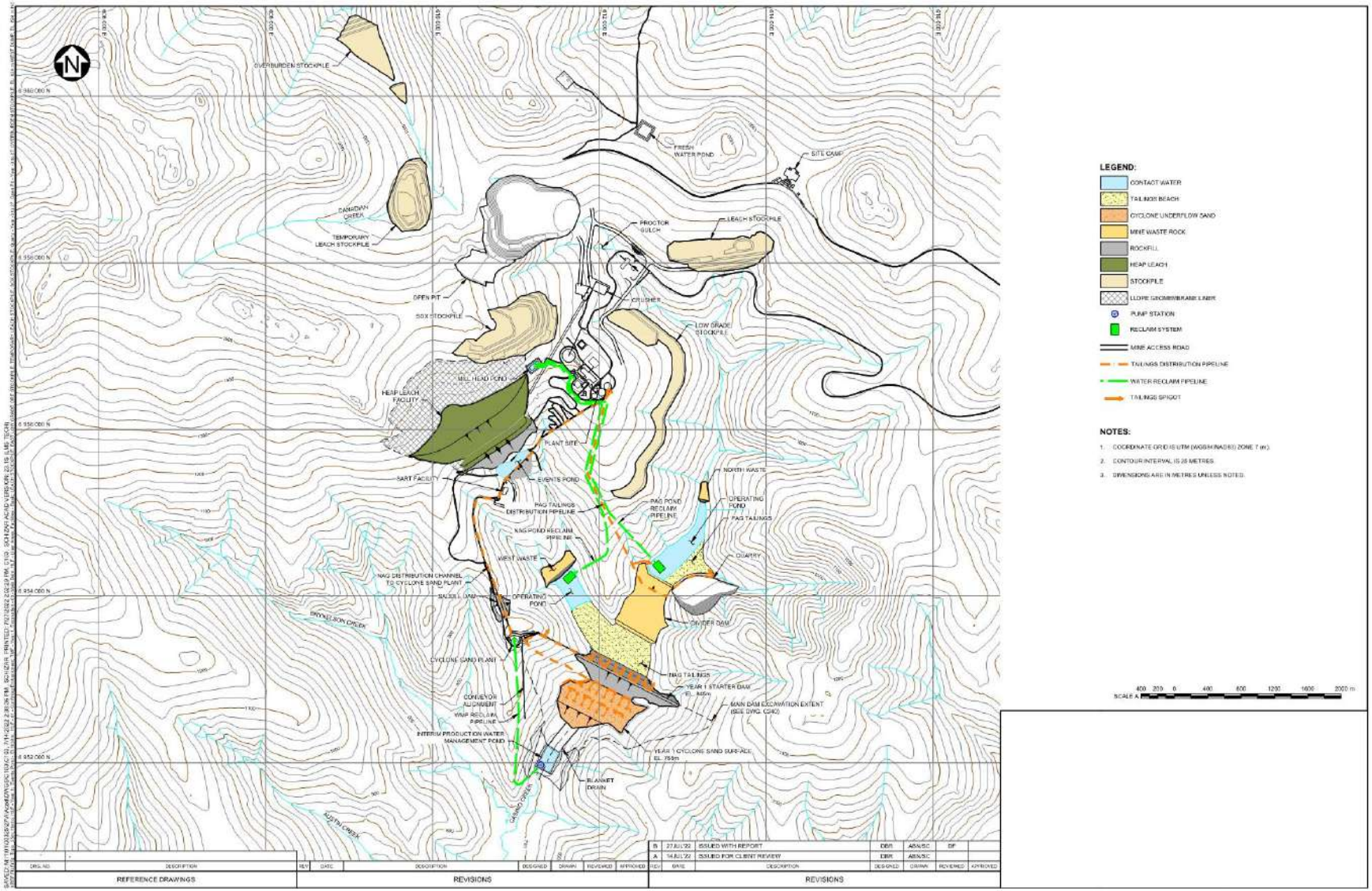


Figure 20-2: General Arrangement End of Year 1

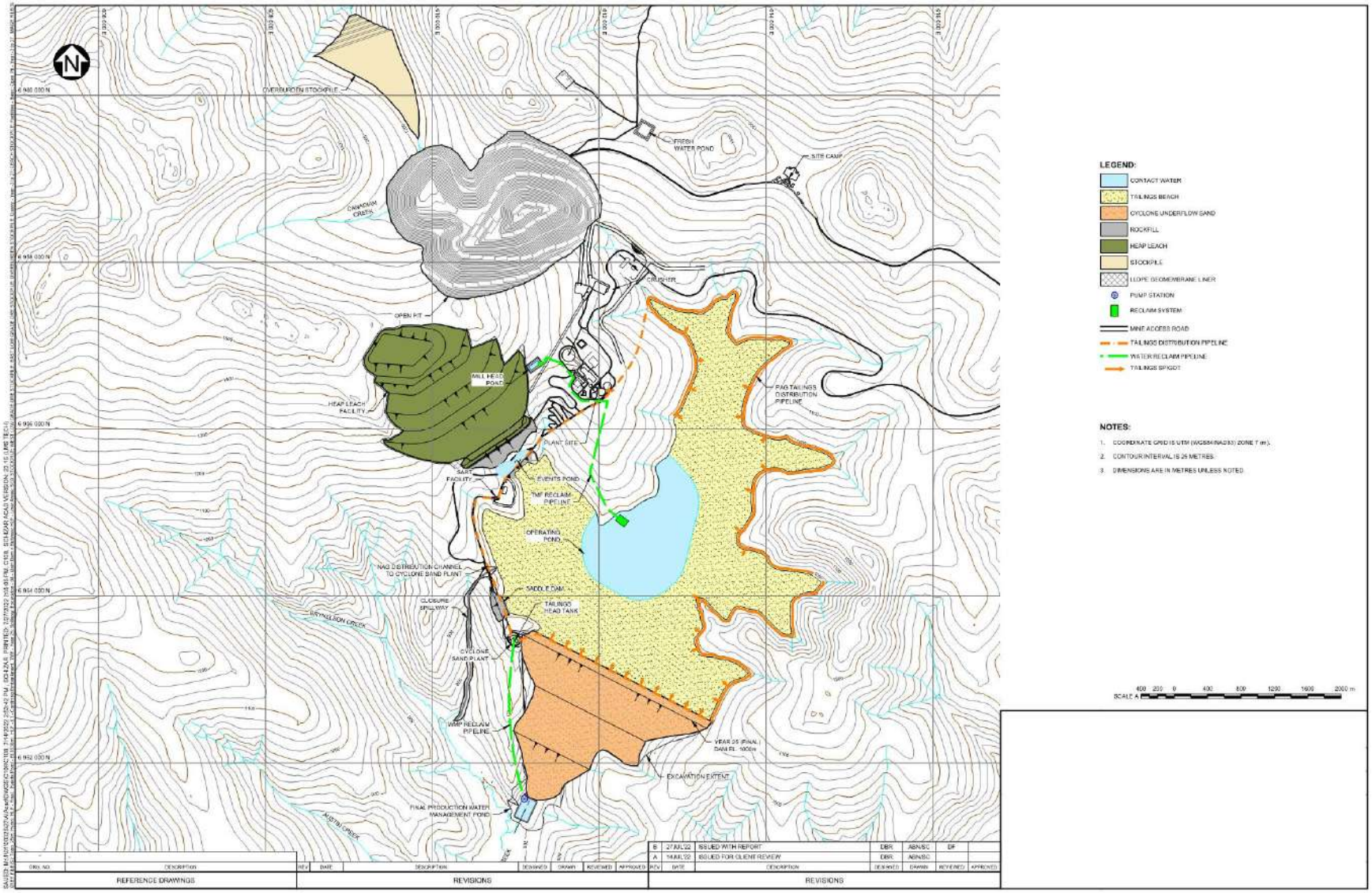


Figure 20-3: General Arrangement End of Year 27

20.1.1 Biophysical Setting

20.1.1.1 Vegetation

The Casino Project area lies within the Dawson Range of the Klondike Plateau Ecoregion, in the Boreal Cordillera Ecozone (Smith et al. 2004). The Project area is primarily located within the boreal bioclimate zone, with areas of subalpine and alpine bioclimate zones occurring mainly in the western portion of the Project area. The predominant ecosite types that occur in the Project study area include subalpine moist shrub, mixed wood forest, sparse coniferous forest, tall shrub, dry broadleaf forest and coniferous forest (2013b).

Within the vegetation study area, nine flora species were identified during the surveys in 2010 and 2012 as having conservation concern (EDI 2013b). Of these nine species, three were on the Yukon Conservation Data Centre (YCDC) Track list and the remainder were on the Watch list (EDI 2013b).

20.1.1.2 Wildlife

The Project area overlaps with the Klaza herd of woodland caribou (*Rangifer tarandus caribou*) year around, with parts of the herd's annual range overlapping with the proposed Freegold Road upgrade and extension. Moose are distributed throughout the wildlife study area with densities that are considered low for the Yukon (EDI 2014).

Grizzly bear (*Ursus arcto horribilis*) is a Federally-listed species of Special Concern, and is on the YCDC animal watch list (EDI 2014; YCDC 2019b). Grizzly bear and black bear (*Ursus americanus*), and bear dens have been observed in proximity to the proposed Project footprint, including the Freegold Road upgrade and extension (EDI 2014, 2020).

Furbearer species that have been detected within the study area include wolves (*Canis lupus*), wolverine (*Gulo gulo*), lynx (*Lynx canadensis*), American marten (*Marten americana*), and collard pika (*Ochotona collaris*), which is designated as a species of concern by COSEWIC (EDI 2014). Three species of bat were observed during bat surveys (EDI 2018). Little brown myotis (*Myotis lucifugus*) and northern myotis (*Myotis septentrionalis*) are two species identified that are on the YCDC track list (YCDC 2019a).

The Project area provides breeding habitat for a variety of bird species. Field surveys confirmed 82 species within the study area. Bird species selected as key indicator species include:

- Cliff nesting raptors, including peregrine falcon (*Falco peregrinus*), golden eagles (*Aquila chrysaotos*) and gyrfalcon (*Falco rusticolus*);
- Species at risk, including horned grebe (*Podiceps auratus*), barn swallow (*Hirundo rustica*), bank swallow (*Riparia riparia*), olive-sided flycatcher (*Contopus cooperi*), rusty blackbird (*Euphagus carolinus*) and short-eared owl (*Asio flammeus*); and
- Various species of passerine birds and waterfowl (CMC 2014a).

20.1.1.3 Fisheries & Aquatic Resources

The fisheries and aquatic resources study area includes Britannia Creek and Casino Creek watersheds. There is a 2 km reach of Yukon River at the northern edge of the study area. The southern portion of the Project area drains into Casino Creek, which flows south to Dip Creek and Yukon River via Klotassin River, Donjek River and White Rivers (PECGI 2013).

Fisheries sampling has occurred in Britannia Creek, Casino Creek, and Dip Creek watersheds, and have identified Arctic grayling (*Thymallus arcticus)* and slimy sculpin (*Cottus cognatus*), that make up most of the total catch in all three watersheds. Low numbers of Burbot (*Lota lota*), round whitefish (*Prosopium cylindraceum*), and chinook salmon (*Oncorhynchus tsawytscha*) were captured in Casino Creek, Dip Creek and Britannia Creek watersheds, respectively (PECGI 2011a).

20.1.1.4 Hydrology and Surface Water Quality

The study area for hydrology and surface water quality is Britannia Creek and Casino Creek watersheds, a 5-km reach of Dip Creek downstream from its confluence with Casino Creek, several Dip Creek tributaries within the airstrip and airstrip access road footprint, and several watersheds along the Freegold Road corridor (CMC 2014c).

Ten hydrometric monitoring stations have been operated since 2008 throughout the Project area to record baseline hydrometric conditions upstream and downstream of the Project infrastructure. In 2020 and 2021, hydrometric stations were re-developed with additional instrumentation and a monitoring program has been developed that aligns with the water quality program and an extensive monitoring program will be implemented in Spring 2022.

Water Quality sampling has occurred since 2008 in the Casino Creek, Dip Creek, Britannia Creek, Yukon River, Sunshine Creek, and Klosoassin River watersheds, and were analyzed for the full suite of physical parameters, anions, nutrients, and total and dissolved metals. Exceedances of the Canadian Council of Ministers of the Environment (CCME) guidelines for protection of freshwater aquatic life have been observed for pH, total cyanide (CN) and total metals, including: aluminum (Al), cadmium (Cd), chromium (Cr), copper (Cu), iron (Fe), lead (Pb), mercury (Hg), silver (Ag), uranium (U) and zinc (Zn) (AECOM 2009; PECGI 2011a, 2011b, 2013). The number of exceedances was highest for aluminum, cadmium, copper, and Iron and lowest for cyanide, mercury, and pH. Exceedances have been observed to be low in March in comparison to May, similar between July and September and least in October, indicating a seasonal trend most likely directly related to hydrological factors such as spring freshet and stream flow (AECOM 2009; PECGI 2011a, 2011b, 2013). Surface water quality monitoring is proposed to continue in 2022 to 2024.

20.1.1.5 Hydrogeology and Groundwater Quality

Hydrogeological site investigation in the summer of 2013 installed 11 monitoring wells at 6 locations. Data were collected at the site during six site visits in 2013 and 2014, including continuous groundwater level and ground temperature monitoring, groundwater quality sampling, and re-development of select historic monitoring wells (Knight Piésold Ltd. 2015). Monthly groundwater monitoring has continued since June 2021 continues to be tracked, and quality assurance and quality control of this data is currently underway.

The Project is situated within a region of discontinuous permafrost, which is inferred to be present at shallow depths on north-facing slopes and below organic soils in portions of the Casino Creek valley, and generally absent, or deeper, on south-facing slopes. A surface and inferred groundwater flow divide bisects the deposit area, directing surface water and groundwater either north towards Canadian Creek watershed or south and east towards Casino Creek watershed. Groundwater is inferred to discharge to surface in creek valleys and sustain winter flows in Casino Creek and Canadian Creek (Knight Piésold Ltd. 2013c). Groundwater elevations have been observed to be at the lowest in April and May immediately preceding the spring freshet and are typically highest in August following the snowmelt and summer rainfall.

Groundwater chemistry is variable throughout the Project area and has been characterized in terms of the proposed Open Pit area, the hillslope area, and the Casino Creek valley area (Knight Piésold Ltd. 2013c). Groundwater samples from the proposed Open Pit area have been characterized as primarily calcium-sulphate type and samples from the hillslope area were dominantly calcium-bicarbonate type. Samples from the Casino Creek valley were generally dominated by calcium, bicarbonate, and sulphate in the upper valley, calcium magnesium, bicarbonate, and sulphate

through the middle of the proposed Tailings Management Facility (TMF) area, and calcium and bicarbonate in the area of the proposed TMF embankment (CMC 2014c).

20.1.1.6 Sediment, Periphyton and Benthic Invertebrates

Aquatic surveys have included sampling for sediment quality, periphyton communities and benthic invertebrate communities in Britannia Creek, Casino Creek, and Dip Creek watersheds. Sediment samples were analyzed for physical parameters and total metals. Exceedances of the CCME Interim Sediment Quality Guidelines (ISQG) include total metals, including: arsenic (As), cadmium (Cd), copper (Cu), lead (Pb) and zinc (Zn) (AECOM 2009; PECGI 2011a). Arsenic exceedances throughout the study area are reflective of natural sources from weathered rocks and soils and are likely due to natural acid rock drainage (ARD) in the groundwater near the ore body (CMC 2014c).

Periphyton and benthic invertebrate samples within each watershed generally appear to be similar. However, the periphyton and benthic invertebrate communities in upper Casino Creek appear to be affected by the water flowing into Casino Creek from Proctor Gulch. Water from Proctor Gulch intermittently flows underground and interacts with the mineralized zone such that re-emerging water is highly acidic and has elevated metal concentrations. Further downstream Casino Creek, biological communities show higher levels of density, diversity, and richness (PECGI 2011a). Monitoring for sediment quality, periphyton communities and benthic invertebrate communities will continue in 2022 to 2024.

20.1.1.7 Air Quality

CMC is using the Government of Yukon ambient air quality standards to protect human and ecological health as a benchmark for the Project. Historical work and new information have highlighted air quality improvements for the Project. Information from the mining process, methods from feasibility study, and feasibility study will be used to construct an Air Quality Model, which will include emissions from stockpiles. The metrological dataset is gathered from a third party from wind fields over a 3-year period to determine dispersion of dust, metal concentrations in dust, and other contaminants. The Air Quality Model will produce metal concentrations in ambient air and metal deposition in the landscape.

20.1.1.8 Greenhouse Gas and Climate Change

CMC will prepare a detailed understanding of the Project's carbon emissions and strategies to reduce and/or offset emissions during the full project lifecycle. Key emission sources for the Project are the mining fleet and the power plant.

A study on the effects of climate change on the project over the life of the project is being developed using current climate change scenarios for the region.

20.1.2 Social and Community Setting

The Project is located on Crown land that is administered by the Government of Yukon. In 2013, a Socio-economic baseline report was produced, which focused on key socio-economic indicators related to population and demographics, community well-being, employment and income, employability, infrastructure and services, economic development and business sector, and cultural continuity (AMEC 2014).

20.1.2.1 Land Use

The Land Use study area is primarily located within the Selkirk First Nations (SFN) and Little Salmon/Carmacks First Nations (LSCFN) Traditional Territory. There are a variety of land use activities in the study area including: traditional and domestic use, parks and protected areas, hunting, trapping and guide outfitting, fishing, forest use (firewood

collection and gathering), recreation and tourism, mining and exploration activities, oil and gas, water resources and licences, other land tenures, transportation and access, and utilities (CMC 2014b).

Both the mine site and the Freegold Road Extension are in the SFN Traditional Territory. The SFN and LSCFN have historically lived in the Project area. In addition to hunting, trapping, and fishing activities SFN and LSCFN have traditionally and currently practiced a variety of traditional land uses, including the collection of plants for food, medicine and construction of baskets, containers, etc. (CMC 2014b). The Project is also within the asserted Traditional Territory of the White River First Nation.

Recreational hunting, guide outfitting and trapping take place within the study area, and there are 11 registered trapline concessions (CMC 2014b).

There are no parks or protected areas, campgrounds or picnic areas located within the study area. However, the general project area is known to be used for a variety of recreational land uses (CMC 2014b).

The Project is located within the Whitehorse Mining District and there are four different types of mining activities that occur within the Project area including: placer (gold), quartz (hard rock), coal, and quarries for borrow source material. Most mining activities are in proximity to the existing Freegold Road and the Freegold Road Extension (CMC 2014b).

20.1.2.2 Archaeology

Heritage resource impact assessments have been completed for portions of the study area. Remaining areas where heritage resources impact assessments are recommended are summarized in (Mooney, J. and Dale, J. 2014). The study identified:

- Seventeen (17) proposed borrow pits and approximately 7.45 km of right-of-way areas remain to be assessed;
- Thirty-three (33) archaeological sites to be managed along the proposed Freegold Road upgrade;
- Thirty-one (31) historical structures and resources to be managed along the Freegold Road;
- Seventeen (17) archaeological sites to be managed near the mine site, the proposed airstrip, the road to the airstrip, and the road to the Yukon River;
- Sixteen (16) historic structures and resources to be managed near the mine site, the proposed airstrip, the road to the airstrip , and the road to Yukon River; and
- Ten (10) previously noted First Nation use sites along the Freegold Road (Mooney, J. and Dale, J. 2014).

20.1.3 Environmental Disclosure

There are no known environmental issues that would materially impact CMC ability to extract mineral resources or mineral reserves.

20.2 WASTE MANAGEMENT AND WATER MANAGEMENT

This section presents the project mine waste management and water management approaches, including geochemical characterization of tailings, waste rock and ore.

20.2.1 Geochemical Characterization

20.2.1.1 Previous Testing

Extensive test work for static testing and kinetic testing programs have been completed for the project. Since the original application, over 1,490 individual static test samples have been completed and 50 kinetic tests were initiated. The kinetic tests are designed to replicate the waste storage conditions and has provided a strong understanding of leaching potential for unsaturated and saturated conditions.

20.2.1.2 Waste Rock and Ore Characterization

Most of the waste rock and ore is classified as potentially acid generating. The acid rock drainage and metal leaching (ARD/ML) properties varies between mineralization zones. Oxidized Leach Cap (CAP) is fully oxidized with trace quantities of sulphide resulting in sulphate acidity. Supergene oxide (SOX) and sulphide (SUS) zones are weathered, metal enriched rock, which generate sulphate and sulphide acidity. Some SOX and SUS zones contain carbonate minerals that will initially buffer drainage pH. Hypogene (HYP) zone is unaltered fresh rock that produces sulphide acidity.

Highly weathered rock types (CAP and SOX) containing sulphate acidity will be immediately acid generating. Saturated column results show that neutralization of these rock types minimizes metal leaching potential under saturated conditions. Fresh rock types (SUS and HYP) will initially produce neutral pH drainage and will only become acid generating over time. Sub-aqueous waste rock disposal will prevent HYP and SUS waste rock from becoming acid generating. Saturated column studies indicate that a combination of precipitation and surface sorption reactions will attenuate peak metal concentrations from waste rock in impoundment.

20.2.1.3 Tailings Characterization

There are two waste rock streams for the tailings. One is a cleaner tailings and pyrite concentrate that has low neutralization potential, which means these materials are to be disposed of directly into the pond or operating pond of the tailings management facilities (TMF). This potentially reactive material accounts for approximately 24% of the overall tailings mass. The second, non-acid generating tailings that are cycloned to produce sand, which will be used to build the embankment of the TMF.

20.2.2 Waste Rock and Tailings Management

The project will create waste rock from mine development and tailings as a by-product of mineral processing. The Tailings Management Facility (TMF) will provide storage for tailings and waste rock. The geochemical characteristics of the tailings produced in the milling mineralized material are predicted to be approximately 24% PAG and 76% non-acid generating (NAG) tailings. Geochemical waste characterization studies have indicated that most waste material is potentially acid generating (PAG) and/or metal leaching (ML). The TMF will store approximately 805 million tonnes (Mt) of tailings and 615 Mt of potentially reactive waste rock and overburden.

PAG tailings will be stored in the PAG Tailings Cell under saturated conditions to prevent acid generation. NAG tailings will be processed in the cyclone plant to produce sand for construction of the TMF Main Dam. The NAG tailings cyclone underflow will be used for embankment and buttress construction. The NAG tailings cyclone overflow will be thickened prior being discharged to the NAG Tailings Cell. The NAG Tailings Cell will primarily store thickened cyclone sand plant overflow and NAG whole tailings from the mill during cyclone plant maintenance periods, as well as screening losses from the tailings dewatering plant.

The waste rock and some overburden produced be deposited into three waste storage areas (West Waste Dump, North Waste Dump and Divider Berm) in the TMF. The West Waste Dump (WWD) will be constructed with NAG waste

rock and overburden upstream from the NAG tailings deposit. The North Waste Dump (NWD) will be constructed with PAG waste rock and overburden upstream from the PAG tailings deposit. The Divider Berm will separate the NAG and PAG tailings within the TMF and be constructed of waste rock and over burden material. Dumps will be constructed several metres above tailings level to provide a dry, stable placement surface, but will be submerged by tailings by the end of operation.

A low-grade ore stockpile proposed for the headwaters of Canadian Creek once it has been cut off by the Open Pit. Approximately 60 million tonnes of overburden from the pit will be stacked in Canadian Creek watershed instead of in the TMF to save it for reclamation.

20.2.3 Water Management

Water management objectives at the Project are to manage water in a manner that provides sufficient water to support ore processing, while minimizing the potential for storm flows to cause damage to mine structures and for mining operations to cause adverse effects to downstream water quality. The strategies applied to achieve these objectives are to keep non-contact water clean by diverting it around Project areas wherever possible, use the water within the project area to the maximum practical extent and manage sediment mobilization and erosion through Best Management Practices (BMP) before and during construction activities.

Groundwater and precipitation collected in the open pit mine will be dewatered throughout mine life, with dewatering flows from the pit sump pumped to the mill for use in the process plant. The open pit mine also intercepts Canadian Creek approximately mid-way through the mine life, which will be diverted into the open pit mine and water used as process water. Following the cessation of mining, the dewatering system will be decommissioned and the open pit mine will be allowed to fill with groundwater, precipitation, and runoff from the upstream catchments, including Canadian Creek. The open pit will also receive water pumped from the HLF and TMF pond early in the closure phase. Ultimately, once the open pit fills, it will discharge to the north end of the TMF. Refer to Figure 20-2 for mine site layout at the start of mine operations, and Figure 20-3 for mine site layout at the end of mine operations.

The TMF Starter Embankment will provide a fresh water source for mill start-up water during pre-production during the initial years of operations. Subsequently, during on-going construction of the Main Embankment, a phased dual-ditch system will route non-contact water around the TMF and contact water will be collected in the Water Management Pond. An interim stage water management pond, sited immediately downstream of the Main Embankment footprint, will be actively managed for the first ten years of operations, during early stages of embankment expansion. A final water management pond, sited immediately downstream of the buttress footprint, will be actively managed from Year 10 to the end of operations. A pump station will return the water from the water management pond back to the TMF. Supplemental booster pump stations will be required as the TMF embankment height increases. Water reclaimed from the TMF consists of supernatant from the settled tailings and runoff from precipitation. Pipelines will convey the reclaimed water to the settlement pond located upgradient of the mill and a water head tank upgradient of the cyclone plant for use in the ongoing process plant and cyclone plant operations. Upon mine closure, the tailings and reclaim delivery systems, cyclone plant and all pipelines, structures and equipment not required beyond mine closure will be dismantled and removed. Post-closure, an overflow spillway and channel will discharge excess water accumulating within the TMF to the south, ultimately terminating at an erosion protected plunge pool at the confluence of Brynelson Creek.

Initial water requirements for the gold plant and oxide mineralized material heap leach operations will be met by pumping water retained behind a temporary cofferdam located in the heap leach event pond. Diversion and runoff collection ditches will also be constructed at the HLF to intercept overland surface runoff around the HLF pad and convey non-contact flows to the TMF and contact water flows to the Events Pond. The HLF Events Pond is situated immediately down gradient of the HLF confining embankment, and the pond inflow is conveyed via the confining embankment spillway to the TMF. The Events Pond will be operated as a dry pond during normal operations but will

provide additional storage for flood events exceeding the design storm event of the HLF in-heap pond, as well as a seepage collection pond for leachate solution that may leak past the double composite liner system in the HLF in-heap pond area. Water collected in the Events Pond will be used to supplement the HLF water supply for irrigation. Volumes exceeding the Events Pond storage capacity will be conveyed to the TMF via the Events Pond Spillway. At closure, the Events Pond liner system will be removed and all water from the reclaimed HLF allowed to drain either to the TMF, or, if water quality meets post-closure objectives and seepage is suitable, will be directly released to downstream waters.

A freshwater system will offset any operational water deficit with water from the Yukon River. Fresh water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above ground insulated pipeline with booster stations. The freshwater pipeline will generally follow the road alignment of the existing road that leads northwards from the mine facility to the Yukon River along Britannia Creek. The fresh water will also be filtered and chlorinated to supply potable and fire water at the processing plant and residence camp, stored and distributed separately from the process freshwater supply system.

Seepage from site infrastructure will be controlled with a variety of seepage control and collection measures. Seepage from ore and topsoil stockpiles will be collected in collection ditches constructed around the downstream footprint edge of each stockpile. These collection ditches will collect and convey runoff from the stockpiles to ponds in topographic low points downstream of each stockpile which will be pumped to the TMF pond. Seepage at the HLF will be controlled by a multi-layered system comprising a confining embankment, composite liner system, leak detection and recovery systems, and perimeter berm with diversion and runoff collection ditches to intercept overland surface runoff around the HLF pad and to convey non-contact water flows to the TMF, and contact water flows to the in-heap events pond. For the first two years, seepage under the TMF will be limited by a grout curtain installed along the upstream toe of the Starter Embankment, together with a geomembrane liner installed on the upstream face of the Starter embankment. A seepage collection system will collect and convey water away from the Starter Embankment. During construction of the Main Embankment, an under-drain system and a surface ditch system will discharge into a Water Management Pond downgradient of the Main Embankment. In the unlikely event that the Water Management System is found to not effectively collect and recover seepage, it will be necessary to install additional seepage control provisions. Following the cessation of mining, the seepage collection systems, collection ditches, and pump back systems will be removed once suitable water quality for direct release is achieved.

On-going water quality monitoring will be required to assess the effectiveness of the Water Management System including the installation of groundwater wells at suitable locations downstream of site infrastructure. Flow monitoring devices will also be installed in diversion ditches and creeks to confirm design flows. The groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long-term monitoring devices. Post-closure requirements will also include scheduled inspections of remaining infrastructure (including the TMF) and will include an on-going evaluation of water quality, flow rates and instrumentation records to confirm the design assumptions for closure.

The Project is designed to be operated as a zero-discharge facility during operations. A water treatment facility will be included in the mine design as mitigation should the water balance be exceeded during operations. The design and capacity will be developed as part of the detailed mine design.

20.3 PERMITTING

Mining projects in the Yukon require several permits and licences issued either by the Yukon Government or by the various departments of the Government of Canada. The primary regulatory approvals are a Water Licence, issued under the Waters Act, and a Quartz Mining Licence, issued under the Quartz Mining Act. Federal authorizations are required under the Fisheries Act and Navigable Waters Act, amongst others. In advance of licence applications, mining

projects require a screening report issued by YESAB. The environmental assessment and permitting requirements are further detailed below.

20.3.1 Existing Assessments and Permits

Exploration activities at mining projects in Yukon are undertaken under a Mining Land Use approval, issued by Yukon Government, Department of Energy, Mines & Resources. Current exploration at the Casino property is approved under Class 4 Quartz Mining Land Use Approval LQ00510, and Class 3 Quartz Mining Land Use Approval LQ00320c. CMC recently underwent assessment through YESAB to combine these two approvals (YESAB project 2020-0083), and a decision document approving this assessment was issued in September 2020. Other existing permits include Waste Management Permit 81-079.

20.3.2 Licensing

Several key licenses/authorizations will be required for this project including a Quartz Mining Licence, a Type A Water Licence, a Fisheries Act Authorization, and a Schedule 2 amendment under the *Fisheries Act*. A Quartz Mining Licence will be required and must adhere to the regulations of the *Quartz Mining Act* particularly as per Section 135, issued and administered by the Yukon Government. Additionally, CMC will be required to obtain a Type A Water Licence under the *Waters Act* for mine operations with use of water and deposit of waste, as well as considerations of tailings creation and storage according to the project design. The Yukon Water Board would issue this licence and it would be administered by the Yukon Government. In addition, CMC will require a Fisheries Act Authorization and a Schedule 2 amendment under the Metal and Diamond Mining Effluent Regulations under the *Fisheries Act* because of the location of the tailings management facility in waters frequented by fish. Environment and Climate Change Canada will be responsible for approving the Compensation Plan for the loss of fish habitat and make the Schedule 2 amendment. Table 20-1 provides a summary of permit requirements for the Casino Mine.

Table 20-1: Summary of applicable Legislation, Regulations and Regulatory Approvals for the Casino Mine

Act	Regulation	Regulatory Approval	Purpose
YESAA			
Yukon Environmental and Socio-economic Assessment Act, S.C. 2003, c.7	Assessable Activities, Exceptions and Executive Committee Projects Regulations Decision Body Time Periods and Consultation Regulations	Panel Report and Recommendation (issued by the Panel of the Board) Decision Document (issued by Yukon Government)	Panel of the Board Review to assess the potential environmental and socio-economic impacts of the Project prior to permitting.
Water License			
Waters Act, SY 2003, c. 19	Waters Regulation, OIC 2003/58 (Schedule 7 Licensing Criteria for Quartz Mining Undertakings)	Water License Type A	Direct water use of water for milling at a rate of more than 100 tonnes of ore per day Deposition of waste from milling at a rate of more than 100 tonnes of ore per day Construction of permanent flood control structures Storage of water behind a dam with a maximum height higher than 8 m and more than 60,000 m^3 is stored
Quartz Mining License			
Quartz Mining Act, SY 2003, c. 14		Quartz Mining License	Development and operation of a quartz mine
Fisheries			
Fisheries Act, R.S.C. 1985, c. F-14		Section 35(2) Authorization	To carry on a proposed work, undertaking, or activity causing serious harm to fish that are part of a commercial, recreational, or Aboriginal fishery or to fish that support such a fishery.
	Metal and Diamond Mining Effluent Regulations, SOR/2002-222	Schedule 2 Amendment Authorization to deposit an effluent that contains a deleterious substance	Authorization for a tailings impoundment area in fish habitat Applies to mines that exceed an effluent flow rate of 50 m^3 per day, based on effluent deposited from all the final discharge points of the mine, and that deposit a deleterious substance in any water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water.

Act	Regulation	Regulatory Approval	Purpose
Land Tenure/Land Use			
Territorial Lands (Yukon) Act, SY 2003, c. 17	Land Use Regulation, OIC 2003/51	Land Use Permit	Use of territorial lands, as defined in the Act for the following: Construction of a solid waste disposal facility Storage and handling of petroleum products Establishment and use of fuel caches of more than 4,000 litres or any single container of more than 2,000 litres on Commissioner's Land Use of more than 50 kg of explosives on Commissioner's Land in any 30-day period Temporary use or occupation of Commissioner's Land Conduct of geotechnical studies Construction of new road access Construction of a new bridge crossing.
	Territorial Lands Regulation, OIC 2003/50	Application for land lease or purchase	Tenure for land lease or agreement of sale
Land Titles Act, SY 2002, c. 10	Land Titles Plans Regulation, OIC 2016/109 Land Titles General Regulation, OIC 2016/108	Registration of interest in land	Title to land
Lands Act, RSY 2002, c. 132	Quarry Regulations, OIC 1983/205	Quarry Permit, Quarry Lease	Removal of gravel/sand from a quarry on Yukon lands
Highways Act, RSY 2002, c. 108	Highways Regulation, OIC 2002/174	Permit under *Highways Act*, S. 7(2)	Construction of new road access
		Access Permit	Constructing road access on highway right-of-way
		Work in Right-of-Way Permit	Perform work within highway right-of-way
		Sign Permit	Erect sign within highway right-of-way
Highways Act, RSY 2002, c. 108		License of Occupation	Use of land within highway right-of-way
Forest Protection Act, RSY 2002, c. 94	Forest Protection Regulation (2003), OIC 2003/57	Burning Permit	Burning refuse (wood)
Forest Resources Act, SY 2008, c.15	Forest Resources Regulation, OIC 2010/171	Forest Resources Permit	Clearing of forest resources incidental to other activity (including mining, land use, road construction, FireSmart, work in the right-of-way, etc.)

Act	Regulation	Regulatory Approval	Purpose
Migratory Birds Convention Act, 1994	Migratory Birds Regulations, CRC, c. 1035	Conformance to Regulation	Inadvertent destruction of nests and eggs due to forest clearing is called "incidental take".
Construction/Building-Related			
Building Standards Act, RSY 2002, c. 19 amended by RSY 2013, c. 3		Building Permit	Construction of buildings outside of a municipality
		Plumbing Permit	Installation of plumbing outside of Whitehorse
Public Health and Safety Act, RSY 2002, C.176	Sewage Disposal Systems Regulation, OIC 1999/82	Permit to Install a Sewage Disposal System	Installation of an on-site sewage disposal system
	Regulations Respecting the Sanitation of Camps in the Yukon Territory, OIC 2009/187	During operation of the camp, provision of information to the Health Officer in respect of the Regulations, as required.	Maintenance of a camp in a sanitary condition
	Drinking Water Regulation, OIC 2007/139	Approval to construction a large public drinking water system Permit to operate a large public drinking water system	Construction, installation, modification, and operation of a drinking water system with 15 or more service connections to a piped distribution system; or Five or more delivery sites on a trucked distribution system, and includes the water source, any infrastructure (e.g., a well, pumphouse, water treatment plant, storage tank, reservoir, water delivery truck, or a piped or trucked distribution system).
	Eating and Drinking Places Regulation, CO 1961/001	Permit to Operate a Food Premise	Operation of a food premises
Electrical Protection Act, RSY 2002, c. 65	Electrical Protection Regulation, 1992, OIC 1992/017 Canadian Electrical Code, C22.1-15 (23rd edition)	Electric Permit	Electrical work
Gas Burning Devices Act, RSY 2002, c. 101	Gas Regulations, OIC 1998/213	Gas Installation Permit	Gas installation or modification
Environment Act, RSY 2002, c. 76	Air Emission Regulations, OIC 1998/207	Air Emissions Permit	Operation of fuel burning equipment greater than 5 Million British thermal units per hour (Mbtu/hr)

Act	Regulation	Regulatory Approval	Purpose
Boiler and Pressure Vessels Act, RSY 2002, c. 16	Design, Construction and Installation of Boilers and Pressure Vessels Regulations, OIC 1980/303	Registration	Installation of power boilers over 10 kW, heating boilers over 20 kW, pressure vessels and piping systems
Contaminants and Waste			
Environment Act, RSY 2002, c. 76	Solid Waste Regulations, OIC 2000/11	Solid Waste Management Permit	Solid waste disposal facility Operation of solid waste incinerator
	Special Waste Regulations, OIC 1995/47	Special Waste Permit Waste Manifest	Handling, disposal, generation, or storage of special (hazardous) waste Transportation of dangerous goods / waste
	Contaminated Sites Regulation, OIC 2002/171	Land Treatment Facility Permit	Operation of a land treatment facility for contaminated soils
		Relocation Permit	Relocation of contaminated material
	Air Emission Regulations, OIC 1998/207	Air Emissions Permit	Release of air pollutants (i.e., incinerator, diesel, generators) Operation of solid waste incinerator
	Ozone Depleting Substances and Other Halocarbon Regulation, OIC 2000/127	Ozone Depleting Substances and Other Halocarbons Permit	Servicing or installation of equipment containing ozone depleting substances (e.g., refrigeration equipment) Purchasing, handling, and services in/of ozone depleting substances equipment Use of ozone depleting substances and equipment
Environment Act, RSY 2002, c. 76	Storage Tank Regulations, OIC 1996/194	Application for Operation, Closure, Abandonment, or Renovations to Storage Tanks Above-Ground Storage Tank Permit	Storage and handling of petroleum products Use of storage tanks containing petroleum and allied petroleum products
Navigation Protection Act, R.S.C. 1985, c. N-22	Navigable Waters Works Regulations, CRC, c. 1232	Section 6(4) Approval to construct, place, alter, repair, rebuild, remove, or decommission a work in, on, over, under, through or across any navigable water that is listed in the Schedule to the Act	Water withdrawal from the Yukon River
Explosives			
Territorial Lands (Yukon) Act, SY 2003, c. 17	Land Use Regulation, OIC 2003/51	Land Use Permit	Use of more than 50 kg of explosives on Commissioner's land in any 30-day period

Act	Regulation	Regulatory Approval	Purpose
Explosives Act, R.S.C. 1985, c. E-17	Explosives Regulations, 2013 SOR/2013-211	Explosives Magazine Storage and Use Permit for Mining Purposes	Explosives storage
		ANFO Permit	Manufacture of ANFO
Occupational Health and Safety Act, RSY 2002, c. 159	Occupational Health and Safety Regulations, OIC 1986/164	Notification to Director OHS	Trenching excavations in excess of 6 m
		Underground and Surface Authorization to Conduct Blasting in Yukon	Blasting – underground or surface
		Temporary Blaster's Permit	Temporary blasting
Transportation			
Highways Act, RSY 2002, c. 108	Bulk Commodity Haul Regulations, OIC 1994/175	Bulk Commodity Agreement	Bulk commodity hauling
	Highways Regulation, OIC 2002/174	Over-dimensional or Over-weight Vehicle Permits	Oversize trucking
Transportation of Dangerous Goods Act, 1992, S.C. 1992, c.34	Transportation of Dangerous Goods Regulations, SOR/2001-286 Interprovincial Movement of Hazardous Waste Regulations, SOR/2002-301	There is no permit for transporting dangerous goods, but the driver must have the correct training, documentation and load security as required by Transport Canada. The Carrier must have a minimum of $2,000,000 liability coverage.	Transport of dangerous goods/waste

20.3.3 Environmental and Socio-Economic Assessment Process

Larger quartz mining projects (i.e., those that begin mining as opposed to just exploration activities) are typically categorized as assessable activities under the Assessable Activities, Exceptions and Executive Committee Projects Regulations (SOR/2005-379) and require an Executive Committee screening. Upon completion of the assessment, YESAB issues a screening report and a recommendation, which is sent to federal, territorial and/or First Nation governments who act as Decision Bodies. The recommendation will include one of four options. YESAB will recommend that the project:

- Be allowed to proceed;
- Be allowed to proceed with terms and conditions;
- Not proceed; or
- The Executive Committee can recommend that the project be required to undergo a review by a Panel of the Board.

The Decision Body for the assessment will be a regulating body or authority. Decision Bodies can be federal, territorial or First Nation governments and agencies that regulate and permit the proposed activity. The Decision Body will issue a Decision Document that accepts, varies, or rejects the recommendation. Once the Decision Document has been issued, an agency can issue authorizations or permits in accordance with their process.

A Project Proposal for the Casino Mine was submitted to the executive committee of the YESAB in January 2014 and underwent several rounds of adequacy review information requests from 2014 through 2016. On February 18, 2016, the Executive Committee determined that the Casino Mine Project requires a Panel Review, the highest level of environmental and socio-economic assessment under YESAA. A Panel Review is an assessment process by which a Panel of the Board (comprised of one YESAB board member nominated by the Council of Yukon First Nations, and two YESAB board members nominated by the territorial or federal governments) conducts technical analysis of an Environmental and Socio-economic Effects Statement submitted by CMC, followed by public hearings. The Panel of the Board then issues their recommendations (similar to the other levels of assessment under YESAA) to the relevant Decision Body(s), which can be federal, territorial and/or First Nation governments. The Decision Body(s) will then decide whether to accept, reject or vary the recommendation of YESAB and issue a Decision Document. The Decision body has 60 days to issue a decision document, or 45 days in which to refer the recommendations back to the Panel of the Board for reconsideration. Regulatory permitting would follow a positive decision document being issued.

Guidelines for the Environmental and Socio-economic Effects Statement were issued by YESAB on June 20, 2016. CMC is preparing the Environmental and Socio-economic Effects Statement in accordance with those guidelines.

20.4 FIRST NATIONS AND COMMUNITY ENGAGEMENT

In Yukon, assessment and regulatory processes provide opportunities for First Nation and community participation. Since 2008, CMC has been consulting, engaging, and sharing information with First Nations, local communities, Yukon government and federal agencies, non-government organizations (NGOs), and individuals. The 2016 Environmental and Socio-economic Effects (ESE) Statement Guidelines issued to CMC outline the information and consultation requirements related to First Nations and community engagement.

The Project has components that are located within the Traditional Territories of three self-governing Yukon First Nations: Selkirk First Nation, Little Salmon/Carmacks First Nation, and Tr'ondëk Hwëch'in. These First Nations each have a signed Final Agreement (also referred to as a Land Claim or Modern-Day Treaty) which outlines and defines their respective Settlement Land, rights, and financial compensation.

Kluane First Nation and White River First Nation have also been identified by the Yukon Environmental and Socio-economic Assessment Board (YESAB) as being potentially affected by the Project. Kluane First Nation has a signed Final Agreement. While Kluane First Nation Traditional Territory is downstream of the Project, no Project components overlap the First Nation's Traditional Territory. White River First Nation has not entered into a land claim or a self-government agreement, as a result White River First Nation is still governed under the Federal Indian Act. White River First Nation has an asserted Traditional Territory boundary that encompasses the entire Project footprint, and a large proportion of the Freegold Road.

CMC has signed agreements with Selkirk First Nation, Little Salmon/Carmacks First Nation, and Tr'ondëk Hwëch'in, which provided funding for participation in the Executive Committee review of the Project Proposal. Agreements were also reached with Selkirk First Nation, Tr'ondëk Hwëch'in, and White River First Nation for the funding of nation-specific Traditional Land Use studies. As a result, Selkirk First Nation prepared a comprehensive Traditional Land Use Study in 2017, Tr'ondëk Hwëch'in in 2018, and White River First Nation in 2018.

Subsequent updated agreements will be required to facilitate future participation in the preparation of the Environmental and Socio-economic Effects Statement, as well as the Panel Review process and/or in any other processes to be conducted under the proposed mining project. CMC is in regular communication with all five First Nations. CMC is planning to continue their ongoing, regular consultation and engagement with each First Nation's leadership and technical teams as the proposed Project continues to advance. This includes additional engagement about (but not limited to) past, present and future land use.

20.5 MINE CLOSURE AND RECLAMATION

A conceptual reclamation and closure plan (RCP) will be submitted with the Environmental and Socio-economic Effects Statement. A detailed RCP will be submitted as part of the application for a Quartz Mining License. The RCP will include a liability estimate for reclaiming and closing the mine. The RCP will demonstrate how CMC has considered and addressed the expectations and concerns throughout the mine planning process. Over the life of the mine, successive iterations of the plan may be expected every two years, each iteration providing more detail and greater certainty regarding the sequence of events to occur during reclamation and closure. The Yukon Government will require the company to post security for this project. Both the Yukon Waters Act and the Yukon Quartz Mining Act have provisions for security to be held by government.

20.5.1 Closure Plan, Cost Estimate and Financial Assurance

The conceptual mine closure occurs over four phases: operational closure, post-mining closure, active closure, and post-closure. Activities of each stage are outlined in Table 20-2.

Table 20-2: Proposed mine closure phases and activities for the Project

Closure Phase	Activities
Operational Closure	As and when each pit, TMF, waste rock storage facility (WRSF) and stages of the heap leach facility (HLF) are decommissioned they will be closed.
Post Mining Closure	Closure activities relating to terminating the mining operation, dismantling infrastructure, and reclamation.
Active Closure	Maintenance, monitoring, and closure of remaining facilities
Post-Closure	Monitoring only of mine closure and reclamation activities including the following: • Heap leach rinsing, re-slopping, and capping; • Managing hazardous waste; • Dismantling and disposing of all structures and equipment; • Landfilling all inert waste, including equipment drained of all oils and hazardous materials; • Transporting all hazardous waste from the Project site; • Disposing all liners and pipelines; • Collecting and treating all contaminated soils; • Re-contouring the site areas to provide positive drainage; and • Scarifying, placement of re-vegetation layer and seeding of disturbed surfaces.

Current proposed mine closure and reclamation means are described in Table 20-3, however, alternatives are being considered and plans are to be finalized.

Table 20-3: Proposed reclamation and closure of mine components

Mine Component	Current proposed reclamation and closure means
Open Pit	Open pit to be flooded and overflow directed to tailings management facility.
Ore Stockpiles	Any remaining Low-Grade Ore to be placed in open pit below flood elevation and remaining stockpile footprints will be graded, covered, and revegetated.
Process Facility and Infrastructure	All infrastructure to be removed or cut to surface. Materials will be appropriately disposed of, and disturbed areas will be covered with topsoil and revegetated.
Heap Leach Facility	Heap detoxified and rinsed; drained down water will be treated then pumped to open pit; heap resurfaced to stable slope, covered with low permeability cover, and revegetated.
Tailings Management Facility (TMF)	Development of a stable cover system. Semi-passive water treatment system(s) will be constructed if required to treat the TMF water including any water conveyed from other mine features such as the flooded open pit or HLF prior to discharging via a spillway, TMF closure spillway constructed for treated surface water discharge.
Airstrip and Site Access Roads	To be maintained for inspection and monitoring during closure.
Freegold Road	Extension decommissioned following completion of mining and active closure activities; public portion remains open.

Environmental monitoring will provide important indicators of the successful closure of the Project. A final post closure monitoring program will be compliant with the applicable guidelines and regulations. The post closure monitoring program will be implemented to ensure that the reclamation measures remain effective and continue to provide a high level of protection for the public and the environment.

The following assumptions were used to build up the closure cost estimate:

- Mobile equipment required for closure was assumed to be rental or third-party equipment;
- Labor was assumed to be provided by the owner using fully burdened labor rates;
- Unit cost estimates were based on the rental equipment fleet; and
- No salvage recover was included in the closure cost estimate.

Casino Mining Corporation will provide financial security for the anticipated closure cost for the Casino mine. The Yukon Government currently holds $672 in security for the Casino property. The Feasibility Study will include a provisional sum of $300 million for project closure. As part of the ongoing permitting process an estimate with supporting details will be provided in the Water License and Quartz Mine License applications. The estimate will present the ultimate cost at the end of mine life, and the amount at selected periods through the mine life. CMC will update the estimate and security provisions regularly throughout the mine life.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST

21.1.1 Initial Capital Cost

Table 21-1 summarizes the initial capital costs.

Table 21-1: Initial Capital Cost Summary

Cost Item	Total (C$M)
Process Plant and Infrastructure	
Project Directs including freight	2,116
Project Indirects	431
Contingency	369
Subtotal	**2,916**
Mine	
Mine Equipment	433
Mine Preproduction	228
Subtotal	**661**
Owner's Costs	41
Total	**3,617**

21.1.2 Basis of Process Plant and Infrastructure Capital Cost Estimate

In general, M3 based this capital cost estimate on its knowledge and experience of similar types of facilities and work in similar locations. To assist in the estimating, M3 used quantity estimates, and in some cases, costs supplied by specialist sub-consultants, such as the following:

- Associated Engineering (AE): Main access road design and costing from Carmacks to site.
- Knight Piésold (KP): Quantities, capital costs, and sustaining capital costs associated with the Heap Leach Facility, and the Tailing Management Facility.
- Independent Mining Consultants (IMC): Mine capital and operating costs.

"Initial Capital" is defined as all capital costs through to the end of construction or the end of Year 1 of the mine life. Capital costs predicted for later years are carried as sustaining capital in the financial model.

All costs 4th quarter 2021 Canadian dollars except as noted otherwise. Canadian to US exchange rate used is C$1.25 = US$1.00.

21.1.2.1 EPCM Execution

The capital costs are based on this project being executed by experienced EPCM contractor(s) in the hard rock mining industry with a recent record of bringing projects on budget or under budget. It is assumed that at least two sufficiently sized self-performing local contractors are in place for all trades, such as civil, concrete, steel, architectural, mechanical, electrical, instrumentation and controls, and process piping. Certain contractors will have multiple trade capabilities.

21.1.2.2 Exclusions and Qualifications

This capital cost estimate excludes the following items:

1. Future escalation.
2. The cost of all prior and future studies.
3. Start-up and initial operating expenses subsequent to Owner's acceptance of the plant ready to accept feed are excluded.
4. Future foreign currency exchange variation.
5. The cost to provide insurance coverage for the duration of the project.
6. Environmental and ecological considerations.

This capital cost estimate depends on the following qualifications:

1. Environmental permits and licenses required to operate the facilities are obtained in a timely manner.
2. Unfettered access to the project site is assured for the duration of the project development and operation.

21.1.2.3 Contingency

Contingency is a cost that statistically will occur based on historical data. The term is not used to cover changes in scope, errors, or lack of sufficient information to meet a desired accuracy range. Contingency is used to cover items of cost which fall within the scope of work but are not known or sufficiently detailed at the time that the estimate is developed. A 15% contingency was applied to the process plant and infrastructure direct and indirect costs.

21.1.2.4 Documents

Documents developed for this Feasibility Study include:

- Flowsheets
- Process design criteria
- Major equipment data sheets
- Civil earthworks and grading drawings
- Process and Instrumentation Diagrams (PIDs)
- Freshwater supply and yard piping drawings
- Electrical one-line diagrams
- Site power distribution drawings
- Control system architecture drawings

21.1.2.5 Construction Labor

Burdened construction labor rates used in the process plant and infrastructure capital cost estimate are shown in Table 21-2. The rates were provided in Q4 2021 by a BC industrial contractor experienced in the scope and scale of the Project. The work schedule considers a 70-hour work week with 10 hours per day at 7 days per week and a cycle of 21 working days and 10 days off.

Table 21-2: Burdened Labor Rates

Trade	Labor Rate (C$/ hour)
Civil Work	$122.67
Concrete	$134.12
Architectural	$135.30
Structural Steel	$134.12
Equipment Installation	$135.30
Piping	$135.30
Electrical	$129.72
Instrumentation	$129.72

21.1.3 Direct Costs

21.1.3.1 Civil

Civil earthwork quantities were taken off from a site wide Civil 3D cut and fill analysis and site civil earthwork grading drawings. It is assumed that the fist 5 metres of excavation is rippable rock and deeper excavations require blasting. Major cut earthworks for the process plant pad and airstrip are costed based on utilizing an early mobilized mining equipment fleet, priced by IMC.

Civil earthwork quantities and unit rates for the Heap Leach Facility and Tailing Management Facility were provided by Knight Piesold.

The 132km main access gravel road to site was designed and costed by Associated Engineering. It is assumed that 30% of the cost will be borne by the Yukon government under the Yukon Resource Gateway Program.

A quality assurance testing allowance of 2% on all civil costs is included.

A survey allowance of 1% on all civil costs is included.

21.1.3.2 Concrete

Concrete quantities were developed from 3D modelling and the general arrangement drawings.

In April 2022, unit rates including breakdown of materials, labor and construction equipment were provided by a BC industrial contractor experienced in the scope and scale of the Project.

A quality assurance testing allowance of 2% on all concrete costs is included.

A survey allowance of 1% on all concrete costs is included.

21.1.3.3 Structural Steel

Structural steel quantities were developed from the general arrangement drawings.

Fabricated steel was quoted. Offshore pricing was used for the fabrication of light, medium, and heavy steel.

In April 2022, steel erection rates were provided by a BC industrial contractor experienced in the scope and scale of the Project.

A quality assurance testing allowance of 2% on all steel costs is included.

A survey allowance of 1% on all steel costs is included.

21.1.3.4 Mechanical Steel

Takeoffs have been made for mechanical steel including platework, abrasion resistant liners, etc. based on the general arrangement drawings, equipment list and experience with similar installations.

21.1.3.5 Mechanical Equipment

M3 solicited prices for the major equipment depicted on the Flow Sheets and the Equipment Register from qualified suppliers. Budgetary quotes were received in Q4 2021 for the following major mechanical equipment:

- ADR Plant
- Agitators
- Apron Feeders
- Barge Pumps
- Belt Feeders
- Bridge Crane
- Coarse Ore Dome
- Column Cells
- Conveyors
- Crushers
- Cyclone Clusters
- Dewatering Screens
- Diesel Generators (Q1 2021)
- Dust Collector
- Filter Presses
- Flocculant System
- Flotation Cells
- Gearless Mill Drives
- Heap Leach Stackers
- Lime Storage and Slaking Package
- LNG Pressure Vessels
- LNG System (Q1 2022)
- Moly Concentrate Dryer
- Power Plant - equipment and bulk materials
- Pumps – Process and Freshwater Supply
- Regrind Mills
- Rock Breaker
- SAG & Ball Mills and Liner Handler
- Thickeners
- Vibrating Screens

Pricing sources for all mechanical equipment in this estimate were as follows:

- Approximately 95% (by value) budgetary quotes
- Approximately 5% (by value) from M3 historical data for similarly sized equipment

Installation labor hours for the SAG and ball mills were provided by FLSmidth. Installation labor hours for the gas turbine power plant was provided by Siemens. Remaining equipment installation labor hours are based on RS Means and M3 in-house data.

21.1.3.6 Architectural

Ancillary facilities costs are based on pre-engineered building quotes. Process plant roofing and siding quantities were developed from the general arrangement drawings and priced based on RS Means.

21.1.3.7 Piping

Process plant piping MTOs are based on the PIDs and general arrangement drawings. Pipe material and installation labor hours are priced based on RS Means.

Yard major overland piping MTOs were based on yard piping drawings with pipe material quoted. Installation labor hours based on RS Means and M3 in house data.

A small-bore pipe allowance of 15% of all process area piping costs is included.

A quality assurance testing allowance of 2% on all piping costs is included.

21.1.3.8 Electrical

Electrical MTOs are based on the electrical single line diagrams and site distribution drawings. All major equipment was quoted including E houses, gearless drives, medium voltage motors and large voltage motors. Materials and installation labor hours are priced based on RS Means.

An electrical miscellaneous allowance of 10% of labor and materials costs is included.

A commissioning assistance allowance of 2% on all electrical costs is included.

A start-up assistance allowance of 1% on all electrical costs is included.

A quality assurance testing allowance of 2% on all electrical costs is included.

21.1.3.9 Instrumentation

Instrumentation MTOs are based on the PIDs drawings. Instruments, material, and installation pricing are based on M3 historical data.

A commissioning assistance allowance of 2% on all instrumentation costs is included.

A start-up assistance allowance of 1% on all instrumentation costs is included.

A software and PLC programming allowance of 0.2% on project direct costs is included.

21.1.3.10 Capital and Commissioning Spares

Capital Spare parts are included at 2% of Plant Equipment Costs. Commissioning spares are also included at 0.5% of Plant Equipment Costs.

21.1.3.11 Freight

Freight is 9% of equipment and materials based on a Q2 2022 logistics evaluation.

21.1.4 Process Plant and Infrastructure Indirect Costs

Indirect costs include:

- Contractor mobilization

- Camp construction and operating costs
- Construction power
- Engineering, Procurement and Construction Management
- Vendor support
- Commissioning

21.1.5 Mine Capital Costs

The estimated mine capital cost includes the following items:

- Mine major equipment
- Mine support equipment
- Mine preproduction development expense

The estimated cost of the following mining facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Explosive storage facilities
- Office facilities

Table 21-3 summarizes the mine capital cost by category for initial and sustaining capital. The initial capital period is considered to be the four-year period from Years -3 through Year 1, as these are the years of significant capital build-up.

Table 21-3: Mining Capital – Mine Equipment and Mine Development (C$ x 1000)

Category		Initial Capital by Time Period				Initial Capital	Sustaining Capital	Total Capital
		Yr -3	Yr -2	Yr -1	Year 1			
Major Equipment		64,926	81,881	52,804	156,576	356,188	139,961	496,149
Support Equipment @	15.00%	9,739	12,282	7,921	23,486	53,428	20,994	74,422
Initial Spare Parts @	0.00%	0	0	0	0	0	0	0
Shop Tools @	0.00%	0	0	0	0	0	0	0
Equipment Subtotal		74,665	94,164	60,725	180,062	409,616	160,955	570,571
Equipment Contingency @	10.0%	7,466	9,416	6,072	0	22,955	0	22,955
Mine Development		34,184	70,206	123,636	0	228,026	0	228,026
TOTAL MINE CAPITAL		116,315	173,786	190,434	180,062	660,597	160,955	821,553
Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices.								

Mine preproduction development of C$228.0 million is based on the estimated mine operating costs during the preproduction period. The cost estimate is based on owner operating costs with large equipment plus a contingency to provide additional allowance for additional road construction or other site preparation and subcontracting portions of the mining, etc. Table 21-4 shows the components of the cost during mine development by year. Total preproduction development is estimated as 75.0 million tonnes, corresponding to a unit rate of C$3.04 per tonne.

Table 21-4: Mine Development Direct Costs Plus Contingency (C$ x 1000)

Item	Year -3	Year -2	Year -1	Total
Owner Operating Cost – Large Equipment	27,347	61,049	112,396	200,792
Mine Development Contingency	6,837	9,157	11,240	27,234
Total Mine Development Cost	34,184	70,206	123,636	228,026
% Contingency	25%	15%	10%	13.6%

21.1.6 Owner's Costs

Costs Included in Estimate

Owner's costs prior to the start of project engineering and construction are deemed sunk and not included in this estimate. Owner's costs incurred during the project development include but are not limited to:

- First fills and consumables
- Owner's On-Site team
- Consultants other than EPCM
- Shop tools & furnishings, special tools
- Office equipment, furniture, and hardware
- General supplies and safety equipment
- Temporary Offices

21.2 SUSTAINING CAPITAL COSTS

Project sustaining capital costs are capital costs incurred during operations and are included in LOM totals shown in Table 21-5 below. Mining capital includes equipment overhauls and replacements. The Tailings Management Facility capital includes earthworks to raise the embankment per annum. The Process Plant capital includes mobile equipment replacements every 7 years and heap leach facility expansions.

Table 21-5: LOM Sustaining Capital Costs

	Total (C$M)
Mine	161
Tailings Management Facility	326
Process Plant (includes Heap Leach)	264
Total	**751**

21.3 OPERATING COSTS

The operating costs for the Casino project are comprised of three components; mining, concentrator to process the sulphide mineral reserve and heap leach to process the oxide mineral reserve. The mining costs per tonne are shown in Table 21-6 below.

Table 21-6: Mining Cost

Area	LOM Cost (CAD$000's)	CAD$/mt (as noted)
Mining cost - Total	**$5,212,265**	
Mining cost per tonne (all material moved)		$2.30
Mining cost per tonne (concentrator + heap leach ore)		$3.65
Mining cost per tonne (concentrator ore only)		$4.28

Operating costs for the concentrator are shown in Table 21-7 below. Note that the $/tonne cost is calculated using the material processed through the concentrator over the life of the mine.

Table 21-7: Concentrator Cost

Area	LoM Cost (CAD$000s)	CAD$/mt mill ore
Concentrator (mill)	$7,811,035	$6.42
General & Administrative	$564,011	$0.46
Total	**$8,375,047**	**$6.88**

Heap leach operating costs are shown in Table 21-8 below. Note that the $/tonne cost is calculated using the material processed through the heap leach over the life of the mine.

Table 21-8: Heap Leach Cost

Area	LoM Cost (CAD$000s)	CAD$/mt heap leach ore
Heap Leach	$405,413	$1.93
ADR/SART	$1,006,778	$4.80
Total	**$1,412,192**	**$6.73**

Life of mine (LoM) operating cost is $11.16 per tonne of sulphide mineral reserve (concentrator), which includes the mining, and general and administrative costs for the mine. The LoM operating cost is $6.73 per tonne of processed oxide mineral reserve (heap leach).

Process operating costs were determined on a year-by-year basis. Table 21-9 below outlines costs for Year 2 only for illustration purposes. They are based on an annual mill mineral reserve tonnage of 47.0 million tonnes that will produce approximately 478,000 tonnes of copper concentrates and 16,000 tonnes of molybdenum concentrates. The heap leach plant costs are based on gold mineral reserve processed of 9.13 million tonnes and production of approximately 76,000 ounces of gold, 158,000 ounces of silver, and 1,000 tonnes of copper precipitates.

Table 21-9: Operating Cost – Mine Site Cost Summary

Concentrator Processing Units Base Rate (tonnes/year mineral reserve)	47,082,000	
Heap Leach Processing Units Base Rate (tonnes/year mineral reserve)	9,125,000	
Total Tonnes Mined	99,999,000	
	Year 2	
	Cost (C$)	**C$/tonne**
Mining Operations		
Drilling	$16,220,519	$0.34
Blasting	$32,604,850	$0.69
Loading	$25,422,385	$0.54
Hauling	$92,314,891	$1.96
Roads and Dumps	$24,893,798	$0.53
Mine Services	$13,469,925	$0.29
Mine Administration	$7,449,425	$0.16
Subtotal Mining	$212,375,793	$4.51
Processing Operations		
Concentrator		
Primary Crushing & Stockpile Feed	$16,094,186	$0.34
Grinding, Classification & Pebble Crushing	$165,382,234	$3.51
Flotation & Regrind	$75,016,079	$1.59
Concentrate Thickening/Filtration	$6,064,260	$0.13
Tailings Dewatering & Disposal	$37,237,452	$0.79
Fresh Water/Plant Water	$4,224,839	$0.09
Flotation Reagents	$2,021,930	$0.04
Ancillary Services	$4,658,431	$0.10
Subtotal Concentrator	$310,699,410	$6.60
Supporting Facilities		
Laboratory	$2,571,221	$0.05
General and Administrative	$19,753,485	$0.41
Subtotal Supporting Facilities	$22,324,707	$0.46
Total Mill Mineral Reserve Cost	$545,399,909	$11.58
Heap Leach		**C$/tonne Heap Leach**
Heap Leach - Gold Mineral Reserve	$17,850,228	$1.96
ADR/SART - Gold Mineral Reserve	$44,263,050	$4.85
Subtotal Heap Leach	$62,113,278	$6.81

21.3.1 Process Plant Operating & Maintenance Costs

The concentrator and heap leach operating costs are summarized by cost elements of labor, power, reagents, maintenance parts and supplies, and services. The concentrator operating costs are shown by cost element in Table 21-10. The heap leach operating costs are shown by cost element in Table 21-11.

Table 21-10: Concentrator Cost by Cost Element

Cost Element	LoM Cost (C$ 000s)	% of Total	C$/t mill ore
Labor	$633,729	8%	$0.52
Power	$2,497,720	32%	$2.05
Reagents	$1,308,326	17%	$1.07
Liners & Grinding Media	$2,349,021	30%	$1.93
Maintenance Parts & Services	$518,129	7%	$0.43
Supplies & Services	$504,110	6%	$0.41
Total	**$7,811,035**	**100%**	**$6.42**

Table 21-11: Heap Leach Cost by Cost Element

Cost Element	LoM Cost (C$ 000s)	% of Total	C$/t heap leach ore
Labor	$163,646	12%	$0.78
Power	$114,581	8%	$0.55
Reagents	$855,881	61%	$4.08
Liners	$143,086	10%	$0.68
Maintenance Parts & Services	$89,223	6%	$0.43
Supplies & Services	$45,775	3%	$0.22
Total	**$1,412,192**	**100%**	**$6.73**

21.3.1.1 Process Labor & Fringes

Process labor costs were derived from a staffing plan and based on prevailing annual labor rates referenced from an industry survey of Canadian wages and benefits. Labor rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes.

21.3.1.2 Power

Power costs were based on obtaining power from an LNG fueled power plant at a rate of C$0.098 per kWh. Power consumption is estimated using the equipment connected kW rating, discounted for operating time per day and anticipated operating load level. A summary of the power cost and consumptions are shown in Table 21-12.

Table 21-12: Power Cost Summary

Area	Average Annual kWh	Average Annual Cost (C$ 000s)
Concentrator		
Primary Crushing & Conveying	50,334,022	$4,935
Grinding, Classification & Pebble Crushing	655,864,901	$64,307
Flotation & Regrind	109,753,496	$10,761
Concentrate Thickening & Filtration	10,600,885	$1,039
Tailing Dewatering & Disposal	80,403,768	$7,884
Fresh Water/Plant Water	24,982,577	$2,450
Concentrator Reagents Area	5,358,502	$525
Ancillary Facilities	8,159,779	$800
Total	**945,457,930**	**$92,702**
Heap Leach		
Heap Leach Pre-crushing	32,672,825	$3,204
Leaching	5,119,786	$502
SART	1,859,783	$182
ADR Plant	12,156,409	$1,192
Cyanide Destruction Facility	1,682,055	$165
Heap Leach & Gold Recovery Water System	1,351,653	$133
Gold Leach Reagents Area	270,483	$27
Total	**55,112,994**	**$5,404**

21.3.1.3 Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied to Skagway, AK, or from local sources where available with allowance for freight to site. Grinding media and part consumption are based on industry practice for the crusher and grinding operations. Reagents and consumables for the process plants are shown below in Table 21-13.

Table 21-13: Reagents and Consumables

Reagent & Consumables	Average Annual Consumption (kg)	Average Annual Cost (C$ 000s)
Copper Flotation Reagents		
Lime (hypogene ore)	33,883,473	$10,614
Lime (supergene sulphide)	36,465,054	$11,422
Fuel Oil	315,536	$1,388
3418A (hypogene)	126,047	$2,231
3418A (supergene sulphide)	113,946	$1,139
A208 (hypogene)	252,093	$1,572
A208 (supergene & sulphide)	226,536	$850
MIBC	450,766	$1,846
PAX	1,803,065	$7,384
Flocculant	991,686	$5,335
Moly Flotation Reagents		
Sodium Hydrosulphide (NaHS)	2,389,061	$3,763
Flocculant	153,261	$825
Fuel Oil	19,834	$87

Reagent & Consumables	Average Annual Consumption (kg)	Average Annual Cost (C$ 000s)
Liners and Grinding Media		
Primary Crusher - Liners	1,805,858	$5,851
SAG Mill - Liners	1,805,858	$6,302
Ball Mill - Liners	2,163,678	$7,465
SAG Mill - Balls	18,030,652	$33,239
Ball Mill - Balls	18,030,652	$30,557
Regrind Mill - Balls	1,848,142	$3,587
Heap Leach/SART Reagents		
Sodium Hydrosulphide (NaHS)	220,991	$348
Sulfuric Acid (H2SO4)	2,865,025	$2,202
Hydrochloric Acid (HCl)	87,348	$64
Pebble Lime (CaO)	30,712,801	$9,620
Sodium Hydroxide (caustic, NaOH)	1,135,528	$1,960
Sodium Cyanide (NaCN)	4,367,417	$20,746
Activated Carbon	96,083	$577
Antiscalant	26,205	$127
Flocculant	3,057	$16
Liners		
Primary Crusher - Liners	349,934	$1,134
Secondary Crusher - Liners	745,081	$2,414
Tertiary Crusher - Liners	745,081	$2,414
Total Average Cost		**$177,081**

21.3.1.4 Maintenance Wear Parts and Consumables

An allowance was made to estimate the cost of equipment maintenance and the cost of facility maintenance. The allowance is 5.0% of the direct capital cost of equipment, which equates to $17.4 million annually for repair parts and $1.7 million annually for outside repairs for the concentrator. The allowance for the heap leach plant is $3.5 million annually for repair parts and $0.3 million annually for outside repairs.

21.3.1.5 Process Supplies & Services

Annual allowances were provided for operating supplies such as outside consultants, outside contractors, lubricants, safety items, and miscellaneous supplies. The allowances were estimated using historical information from other operations and projects.

21.3.2 General Administration

General and administration costs include labor and fringe benefits for the administrative personnel, human resources, and accounting. Also included are office supplies, communications, insurance, employee transportation and camp, and other expenses in the administrative area. Labor costs for G&A are based on a staff of 40. Labor rates are based on a daily rate and include benefits. All other G&A costs were developed as allowances based on historical information from other operations and other projects. Laboratory cost estimates are based on labor and fringe benefits, power, reagents, assay consumables, and supplies and services. All other laboratory costs were developed as allowances based on historical information from other operations and other projects. Note that laboratory services are likely to be contracted out. This estimate retains the costs under the assumption that contract and in-house laboratory services costs are equal.

Table 21-14: Operating Cost – Concentrator Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year mill ore)	47,082,000	
	Year 2	
	Annual Cost	**C$/t**
Primary Crushing & Stockpile Feed		
Labor and Fringes	$2,430,015	$0.05
Power	$5,112,212	$0.11
Liners	$6,111,277	$0.13
Maintenance	$2,120,848	$0.05
Supplies & Services	$319,835	$0.01
Subtotal Primary Crushing & Stockpile Feed	$16,094,186	$0.34
Grinding, Classification & Pebble Crushing		
Labor and Fringes	$2,330,748	$0.05
Power	$66,613,395	$1.41
Liners	$14,379,606	$0.31
Grinding Media	$70,380,403	$1.49
Maintenance	$11,117,085	$0.24
Supplies and Services	$560,996	$0.01
Subtotal Grinding, Classification & Pebble Crushing	$165,382,234	$3.51
Flotation & Regrind		
Labor and Fringes	$2,580,001	$0.05
Power	$11,147,194	$0.24
Reagents	$56,514,066	$1.20
Maintenance	$4,647,987	$0.10
Supplies and Services	$126,831	$0.00
Subtotal Flotation & Regrind	$75,016,079	$1.59
Concentrate Thickening/Filtration		
Labor and Fringes	$3,339,025	$0.07
Power	$1,076,687	$0.02
Maintenance	$1,521,717	$0.03
Supplies and Services	$126,831	$0.00
Subtotal Concentrate Thickening/Filtration	$6,064,260	$0.13
Tailings Dewatering & Disposal		
Labor and Fringes	$1,220,250	$0.03
Power	$8,166,267	$0.17
Maintenance	$5,216,244	$0.11
Supplies and Services	$22,634,691	$0.48
Subtotal Tailings Dewatering & Disposal	$37,237,452	$0.79
Fresh Water/Plant Water		
Labor and Fringes	$585,343	$0.01
Power	$2,537,374	$0.05
Maintenance	$1,008,377	$0.02
Supplies and Services	$93,745	$0.00
Subtotal Fresh Water/Plant Water	$4,224,839	$0.09

Flotation Reagents		
Labor and Fringes	$585,343	$0.01
Power	$544,240	$0.01
Maintenance	$809,631	$0.02
Supplies and Services	$82,716	$0.00
Subtotal Flotation Reagents	$2,021,930	$0.04
Ancillary Services		
Labor and Fringes	$1,291,023	$0.03
Power	$828,754	$0.02
Maintenance	$2,455,938	$0.05
Supplies and Services	$82,716	$0.00
Subtotal Ancillary Services	$4,658,431	$0.10
Total Process Plant	**$310,699,410**	**$6.60**

Table 21-15: Operating Cost – Heap Leach Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year ore)	**9,125,000**	
Processing Units Base Rate (dore ozs/year) - Gold Ore	**76,088**	
	Year 2	
	Annual Cost	**C$/t**
Heap Leach (includes Tertiary Crusher & conveyor)		
Labor and Fringes	$2,322,048	$ 0.25
Power	$3,705,546	$ 0.41
Liners	$6,228,217	$ 0.68
Maintenance	$4,181,630	$ 0.46
Supplies & Services	$1,412,787	$ 0.15
Subtotal Heap Leach	$17,850,228	$ 1.96
ADR/SART (Gold Ore)		
Labor and Fringes	$2,506,119	$ 0.27
Power	$1,698,254	$ 0.19
Reagents	$37,254,663	$ 4.08
Maintenance	$2,224,323	$ 0.24
Supplies and Services	$579,690	$ 0.06
Subtotal ADR/SART (Gold Ore)	$44,263,050	$ 4.85
Total Heap Leach	**$62,113,278**	**$ 6.81**

21.3.3 Mine Operating Costs

Table 21-16 summarizes the mine operating costs. Total cost, the cost per total tonne, and cost per mill tonne are shown by various time periods. During commercial production, the unit costs for mining are C$2.29 per total tonne and C$4.283 per mill tonne. The leach tonnes are not in the divisor for the cost per mill tonne calculations. Years 21 to 27 are low grade stockpile re-handle.

Table 21-16: Summary of Total and Unit Mining Costs

Category	Total Material (kt)	Mill Material (kt)	Total Cost (C$x1000)	Cost Per Total Tonne (C$/t)	Cost Per Mill Tonne (C$/t)
Mine Development (PP)	75,000	0	228,026	3.040	0.000
Commercial Production (Years 1 to 27)	2,271,856	1,217,069	5,212,256	2.294	4.283
All Time Periods	2,346,856	1,217,069	5,440,291	2.318	4.470
Commercial Production Years 1 - 5	507,897	219,161	1,121,820	2.209	5.119
Commercial Production Years 6 - 10	517,991	228,337	1,249,213	2.412	5.471
Commercial Production Years 11 - 15	506,262	230,322	1,227,214	2.424	5.328
Commercial Production Years 16 - 20	421,828	227,689	1,061,503	2.516	4.662
Commercial Production Yr 21 - 27 (LG)	317,878	311,560	552,516	1.738	1.773

The estimate is based on assumed prices for commodities such as fuel, explosives, parts, etc., that are subject to wide variations depending on market conditions. The current estimate is based on the following estimated prices for key commodities:

- Diesel fuel delivered to the site for C$1.375 per litre.
- Electrical power at C$0.098 per kWh.
- Bulk emulsion at C$1.123 per kg delivered to the site.
- Exchange rate of C$1.26 = US$1.00.

The following tables show additional details of the mine operating costs. Table 21-17 shows the mine operating cost by the various cost centres (drilling, blasting, loading, hauling, etc.). Table 21-18 breaks out the costs by parts and consumables and the various labor components. Table 21-19 shows details of the parts and consumables costs (fuel, power, explosives, etc.). These all reflect the commercial production period, Years 1 through 27.

Table 21-17: Mine Operating Cost by Cost Centre

Cost Centre	Total Cost (C$ x 1000)	Cost Per Total Tonne (C$/t)	Cost Per Mill Tonne (C$/t)	% of Total
Drilling	340,698	0.150	0.280	6.5%
Blasting	664,150	0.292	0.546	12.7%
Loading	572,101	0.252	0.470	11.0%
Hauling	2,587,914	1.139	2.126	49.7%
Roads and Dumps	535,766	0.236	0.440	10.3%
Mine Services	334,943	0.147	0.275	6.4%
Mine Administration	176,692	0.078	0.145	3.4%
Total Cost	5,212,265	2.294	4.283	100.0%
Total and Mill Ktonnes		2,271,856	1,217,069	

Table 21-18: Mine Operating Cost by Consumables versus Labor

Category	Total Cost (C$ x 1000)	Cost Per Total Tonne (C$/t)	Cost Per Mill Tonne (C$/t)	% of Total
Parts and Consumables	4,041,119	1.779	3.320	77.5%
Mine Administration Salaries	141,354	0.062	0.116	2.7%
Operating Labor	467,916	0.206	0.384	9.0%
Maintenance Labor	561,876	0.247	0.462	10.8%
Total Cost	5,212,265	2.294	4.283	100.0%
Total and Mill Ktonnes		2,271,856	1,217,069	

Table 21-19: Mine Parts and Consumables Costs

Category	Total Cost (C$ x 1000)	Cost Per Total Tonne (C$/t)	Cost Per Mill Tonne (C$/t)	% of Total
Fuel	1,617,650	0.712	1.329	40.0%
Power	53,201	0.023	0.044	1.3%
Tires	429,700	0.189	0.353	10.6%
Replacement Parts	529,823	0.233	0.435	13.1%
Lubricants and Filters	618,270	0.272	0.508	15.3%
Ground Engaging Wear Parts	107,582	0.047	0.088	2.7%
Blasting	649,555	0.286	0.534	16.1%
Mine Administration	35,338	0.016	0.029	0.9%
Other	0	0.000	0.000	0.0%
Total Cost	4,041,119	1.779	3.320	100.0%
Total and Mill Ktonnes		2,271,856	1,217,069	

22 ECONOMIC ANALYSIS

The economic analysis uses a discounted cash flow (DCF) methodology to determine the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production costs, and sales revenue. The sales revenue is based on the production of copper concentrate with gold and silver credits, molybdenum concentrate, and gold & silver doré. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

This economic analysis is based on measured and indicated mineral resources only. Inferred mineral resources are considered waste for this analysis.

All amounts are in Canadian dollars unless noted otherwise.

22.1 MINE PRODUCTION STATISTICS

Mine production is reported as direct feed mill mineralized material, SOX stockpile mineralized material, low grade mineralized material, leach mineralized material and waste material from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. The financial model assumes the stockpiling of low-grade mineralized material and its subsequent processing at the end of the mine life.

The life of mine mineralized material and waste quantities and mineralized material grade are presented in Table 22-1.

Table 22-1: Life of Mine Mineralized Material, Waste Quantities and Mineralized Material Grade

	Tonnes (000')	Copper %	Moly %	Gold g/t	Silver g/t
Direct Mill Feed Mineralized Material	913,893	0.206%	0.02%	0.230	1.773
SOX Stockpile Mineralized Material	35,340	0.413%	0.03%	0.569	2.582
Low Grade Mineralized Material	267,836	0.120%	0.01%	0.136	1.187
Leach Mineralized Material	209,637	0.036%		0.265	1.946
Waste	611,264				
Total Material Mined	2,037,970				

22.2 PLANT PRODUCTION STATISTICS

In the current plan, all the mill mineralized material and leach mineralized material is processed directly, with the low-grade mill mineralized material being stockpiled and processed at the end of the mine life. The leach mineralized material will begin to be processed in Year -2, which is two years before the mill mineralized material processing begins.

The gold mineralized material heap leach processing will produce three products, gold and silver doré and a copper precipitate. The estimated production over the life of the heap leach is 1,427,000 ounces of gold, 3,410,000 ounces of silver, and 29.8 million pounds of copper.

Production from the flotation plant will produce a copper-gold-silver concentrate and a molybdenum concentrate. The estimated copper concentrate production for the life of the flotation plant is 7.12 million tonnes containing 4.4 billion pounds of copper and 5.7 million ounces of gold and 34.5 million ounces of silver. The estimated molybdenum concentrate production for the life of the flotation plant is 330,000 tonnes containing 407.3 million pounds of molybdenum.

22.3 CAPITAL EXPENDITURE

22.3.1 Initial Capital

The base case financial indicators have been determined based on 100% equity financing of the initial capital. The analysis assumes LNG is used to fuel the power plant from initial start-up of the concentrator and the haulage units throughout the life of the mine. Any acquisition cost or expenditures prior to the full project production decision have been treated as "sunk" costs and have not been included in the analysis.

The total capital cost estimated for the project is $3.617 billion in Canadian dollars. The initial capital includes expenditures for Owner's cost, contingency, pre-production mining and the LNG power plant.

22.3.2 Sustaining Capital

Sustaining capital is scheduled over the operating life of the mine for tailings and heap leach expansion as well as equipment replacement. The total life of mine sustaining capital is estimated to be $751.3 million. This capital will be expended during a 25-year period.

22.3.3 Working Capital

An estimation of working capital has been included in the DCF analysis assuming 60 days for receipt of sales revenue (accounts receivable) and 30 days for payment to vendors (accounts payable). An allowance for initial replacement parts inventory for the heap leach plant (year -2) and concentrator (year -1) is also included. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

22.3.4 Salvage Value

An allowance of $22.8 million is included in the DCF analysis as a return of capital from the salvage and resale of equipment at the end of the mine life.

22.4 REVENUE

Annual revenue is determined by applying selected metal prices to the annual payable metal contained in the concentrates and doré estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The base case financial evaluation uses long term prices that were based on analyst consensus of US$3.60 per pound of copper, US$1,700 per oz of gold, US$22.00 per oz of silver and US$14.00 per pound of molybdenum with an exchange rate of US$0.80 per CAD$. Prices used are as shown in Table 22-2.

Table 22-2: Metal Prices Used in Economic Analysis

Metal	Long Term Price US $	Long Term Price CAD $
Copper (price/lb)	$3.60	CAD 4.50
Gold (price/oz)	$1,700.00	CAD 2,125.00
Silver (price/oz)	$22.00	CAD 27.50
Molybdenum (price/lb)	$14.00	CAD 17.50

The revenue is the gross value of payable metals sold before transportation and smelting charges.

22.5 TOTAL CASH OPERATING COST

The average sulphide cash operating costs are $13.24 per tonne of mill mineralized material processed. Included in these costs are the mining operations, concentrator, general administration, and smelter and transportation costs. Heap Leach cash operating cost is $6.79 per tonne of oxide mineralized material. Included in these costs are the heap leach operations, ADR/SART operations and transportation and refining.

Specifics of operating costs are discussed in Section 21.

22.6 TOTAL CASH PRODUCTION COST

Total Cash Production Cost is the Total Cash Operating Cost, plus royalties, carbon tax, reclamation & closure, property tax and salvage income. The sulphide average cash production cost including these items totals $15.84 per tonne of mill mineralized material processed. The Heap Leach costs remain the same at $6.79 per tonne of leach mineralized material as the sulphide process bears these additional costs alone.

22.6.1 Royalty

A net smelter return (NSR) royalty will be paid using a rate of 2.75% totalling $998.6 million over the life of the mine.

It is estimated that $2.26 billion will be paid in Yukon mining royalties. Yukon mining royalties are based on a sliding scale of 3-12% of revenues less operating expenses, depreciation, and pre-production expenses.

22.6.2 Taxes

22.6.2.1 Corporate Tax

Corporate income taxes are estimated to be $3.69 billion for the life of the mine based on a 27% combined federal and territorial corporate income tax rate. A deduction of depreciation for class 41A assets is being taken which results in no income tax being paid until initial capital is fully depreciated. These deductions against income are applied each year but cannot create a loss.

22.6.2.2 Property Tax

An allowance of $100,000 per year was included in the cash flow to account for property tax.

22.6.2.3 Carbon Tax

Carbon tax payments under the Output Based Pricing System (OBPS) are based on facility output by type of industrial process, with standards specified in the OBPS Regulation that establish the quantity (limit) of emissions a facility may emit before a tax is payable. A facility must pay carbon taxes on total eligible emissions above the limit, whereas a facility with total emissions below the limit will qualify for a credit. Credits can be held for use against future payment obligations or sold to another OBPS regulated facility.

The Greenhouse Gas (GHG) emission compensation cost was estimated for the Casino Mine on a year-by-year basis in accordance with the Canadian government's OBPS and engagement with Intergroup Consultants, Ltd, based in Winnipeg, Manitoba, Canada. The estimated carbon tax is included in the project financial model.

The schedule of annual compensation is based on:

1. The Canadian government's declared path toward a $170/tonne carbon price by 2030,

2. The decreasing Output Based Standards (emission limits) for electrical generation, and

3. Casino Mine operating at the production levels as indicated in the mine plan (for site hauling, and full electrical and heat production).

4. The estimated LNG fuel consumption for electrical power generation based on annual electrical demand projections and the thermal efficiency of the power plant.

5. The estimated annual quantity of diesel fuel for mining and other on-site uses was used to determine the GHG emissions from diesel consumption.

22.6.3 Reclamation & Closure

An allowance of $300.0 million is included in the analysis, which was estimated and provided by Western Copper and Gold. Expenditures for the closure activity begins in the last production year and spans 6 years.

22.7 TOTAL PRODUCTION COST

Total Production Cost is the Total Cash Cost plus depreciation. Depreciation is calculated by the 25% Declining Balance method starting with the first year of production. The last year of production is the catch-up year if the assets are not fully depreciated by that time. An additional deduction for the initial capital is being taken in the early years until the initial capital is fully depreciated.

22.8 PROJECT FINANCING

It is assumed the project will be 100% equity financed.

22.9 NET INCOME AFTER TAX

Net Income after tax amounts to $10.0 billion for the life of the mine.

22.10 NPV AND IRR

The base case economic analysis (Table 22-3) indicates that the project has an Internal Rate of Return (IRR) of 18.1% after taxes with a payback period of 3.3 years.

Table 22-3 shows the impact to the NPV (at various discount rates), IRR and payback economic indicators if percentage changes to metals prices, initial capital, operating costs, and mill recoveries where to occur as noted in the table. These sensitivity studies illustrate that the project's IRR sensitivity to variation in sales price has the most impact, while variation of operating cost, mill recovery, and initial capital cost are approximately equal. Table 22-4 further demonstrates the project's sensitivity to metals prices.

Table 22-3: Sensitivity Analysis (After tax, CAD$M)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	NPV @ 10%	IRR	Payback Years
Base Case (LTP)	$10,019	$4,059	$2,334	$1,568	18.1%	3.3
Base-Case Sensitivities						
Metals Price +10%	$12,582	$5,313	$3,211	$2,276	21.2%	2.8
Metals Price -10%	$7,456	$2,802	$1,454	$857	14.7%	4.0
Capex +10%	$9,786	$3,837	$2,122	$1,362	16.5%	3.6
Capex -10%	$10,253	$4,280	$2,547	$1,775	19.9%	3.0
Opex +10%	$9,052	$3,604	$2,024	$1,322	17.0%	3.5
Opex -10%	$10,986	$4,513	$2,645	$1,815	19.1%	3.1
Mill Recovery +5%	$11,125	$4,594	$2,707	$1,867	19.4%	3.1
Mill Recovery -5%	$8,914	$3,523	$1,962	$1,269	16.7%	3.6

Base Case Commodity Prices (CAD$)	
Copper	$4.50
Molybdenum	$17.50
Gold	$2,125.00
Silver	$27.50

Table 22-4: Metal Price Sensitivity

Exchange Rate - .80$US			Copper Price ($US)						
			$3.00	$3.50	$3.60	$4.00	$4.50	$5.00	$5.50
Gold Price ($US)	$1,300	NPV 8% (CAD $M)	$866	$1,437	$1,551	$2,006	$2,571	$3,136	$3,701
		IRR%	12.1%	14.5%	14.9%	16.7%	18.7%	20.7%	22.5%
		Payback (years)	5.0	4.1	4.0	3.6	3.2	2.9	2.7
	$1,500	NPV 8% (CAD $M)	$1,261	$1,831	$1,944	$2,396	$2,961	$3,526	$4,091
		IRR%	13.8%	16.1%	16.5%	18.2%	20.2%	22.1%	23.9%
		Payback (years)	4.3	3.7	3.6	3.3	3.0	2.8	2.6
	$1,700	NPV 8% (CAD $M)	$1,655	$2,221	$2,334	$2,786	$3,351	$3,917	$4,481
		IRR%	15.5%	17.7%	18.1%	19.7%	21.6%	23.5%	25.2%
		Payback (years)	3.8	3.4	3.3	3.0	2.8	2.6	2.4
	$1,850	NPV 8% (CAD $M)	$1,949	$2,514	$2,627	$3,079	$3,644	$4,209	$4,772
		IRR%	16.7%	18.8%	19.2%	20.8%	22.7%	24.5%	26.2%
		Payback (years)	3.6	3.2	3.1	2.9	2.7	2.5	2.3
	$2,050	NPV 8% (CAD $M)	$2,339	$2,904	$3,017	$3,469	$4,034	$4,597	$5,159
		IRR%	18.3%	20.3%	20.7%	22.2%	24.0%	25.8%	27.5%
		Payback (years)	3.2	2.9	2.9	2.7	2.5	2.4	2.2
	$2,200	NPV 8% (CAD $M)	$2,632	$3,197	$3,310	$3,762	$4,326	$4,888	$5,450
		IRR%	19.4%	21.4%	21.8%	23.3%	25.1%	26.8%	28.4%
		Payback (years)	3.0	2.8	2.8	2.6	2.4	2.3	2.2

Exchange Rate - .80$US		Molydenum Price ($US)						
		$10.00	$12.00	$14.00	$16.00	$18.00	$20.00	$22.00
NPV 8% (CAD $M)		$1,997	$2,166	$2,334	$2,503	$2,672	$2,840	$3,009
IRR%		16.9%	17.5%	18.1%	18.6%	19.2%	19.7%	20.2%
Payback (years)		3.5	3.4	3.3	3.2	3.1	3.0	3.0

Mineralized material mined during the first four years is substantially higher in copper, gold, silver, and molybdenum than the life-of-mine average. This has a favorable impact to cash flow during those years allowing payback in 3.3 years under the base case. Table 22-5 illustrates the difference during these early years.

Table 22-5: Project Cash Flow

	Years 1-4	Life of Mine
Average Annual Pre-Tax Cashflow (C$ 000s)	$1,032,581	$661,703
Average Annual After-Tax Cashflow (C$ 000s)	$950,781	$517,271
Average NSR (sulphide mineralized material C$)	$43.15	$29.08
Average Annual Mill Feed Grade		
Copper (%)	0.300%	0.189%
Gold (g/t)	0.352	0.217
Silver (g/t)	2.054	1.659
Molybdenum (%)	0.025%	0.021%
Average Concentrate Production		
Copper (dry kt)	390	264
Molybdenum (dry kt)	13	12
Average Annual Metal Production		
Copper & Molybdenum Concentrate		
Copper (Mlbs)	241	163
Gold (koz)	333	211
Silver (koz)	1,596	1,277
Molybdenum (Mlbs)	15.5	15.1
Gold/Silver Doré		
Gold (koz)	73	57
Silver (koz)	159	136
Copper Precipitate		
Copper (Mlbs)	1.5	1.2

The Financial Model is shown on the following page in Table 22-6.

Table 22-6: Financial Model

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Mining Operations																																					
Direct Feed Mill Mineralized Material																																					
Beginning Inventory (kt)	913,893		913,893	913,893	913,225	908,183	879,269	832,187	785,601	743,737	701,920	660,087	618,185	575,793	533,058	491,573	450,026	406,959	363,229	316,561	271,413	225,824	179,651	134,108	89,073	43,724	9,176	-	-	-	-	-	-	-	-	-	-
Mined (kt)	913,893		668	5,042	28,914	47,082	46,586	41,864	41,817	41,833	41,902	42,392	42,735	41,485	41,547	43,067	43,730	46,668	45,148	45,589	46,173	45,543	45,035	45,349	34,548	9,176	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		913,893	913,225	908,183	879,269	832,187	785,601	743,737	701,920	660,087	618,185	575,793	533,058	491,573	450,026	406,959	363,229	316,561	271,413	225,824	179,651	134,108	89,073	43,724	9,176	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.206%		0.000%	0.236%	0.238%	0.352%	0.334%	0.271%	0.256%	0.209%	0.197%	0.200%	0.200%	0.185%	0.220%	0.268%	0.192%	0.194%	0.182%	0.203%	0.158%	0.148%	0.152%	0.162%	0.132%	0.158%	0.190%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.024%		0.0000%	0.0177%	0.0127%	0.0235%	0.0312%	0.0294%	0.0172%	0.0150%	0.0171%	0.0250%	0.0290%	0.0331%	0.0313%	0.0289%	0.0312%	0.0298%	0.0247%	0.0119%	0.0134%	0.0134%	0.0187%	0.0240%	0.0287%	0.0286%	0.0335%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.230		-	0.358	0.420	0.421	0.373	0.346	0.284	0.247	0.250	0.242	0.233	0.218	0.224	0.211	0.214	0.204	0.172	0.166	0.187	0.178	0.186	0.182	0.155	0.165	0.203	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	1.773		-	1.930	2.199	2.535	1.998	1.997	1.887	1.740	1.737	2.220	1.630	1.667	1.901	1.881	1.703	1.538	1.791	1.563	1.534	1.662	2.270	1.560	1.170	1.540	1.210	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	4,155,875		-	3,476	26,441	224,614	346,376	278,287	236,374	192,539	181,887	184,756	186,849	174,760	200,964	245,773	182,023	186,633	187,308	202,499	159,053	151,058	152,616	160,842	131,970	120,341	38,436	-	-	-	-	-	-	-	-	-	-
Contained Molybdenum (klbs)	483,780		-	261	1,416	14,957	32,414	30,207	15,853	13,850	15,775	23,094	27,084	31,155	28,640	26,449	29,668	28,768	25,411	11,833	13,421	13,669	18,776	23,828	28,693	21,783	6,777	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	6,751		-	8	68	391	565	518	382	332	336	326	318	299	299	282	297	287	258	241	274	265	272	264	226	183	60	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	52,083		-	41	356	2,356	3,024	2,990	2,540	2,339	2,336	2,991	2,222	2,290	2,536	2,513	2,358	2,162	2,688	2,269	2,248	2,467	3,324	2,259	1,706	1,711	357	-	-	-	-	-	-	-	-	-	-
SOX Stockpile Mineralized Material																																					
Beginning Inventory (kt)	35,340		35,340	35,319	32,442	21,073	1,172	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	35,340		21	2,877	11,369	19,901	1,172	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		35,319	32,442	21,073	1,172	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.413%		0.189%	0.192%	0.199%	0.575%	0.300%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.031%		0.0141%	0.0191%	0.0223%	0.0385%	0.0116%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.569		0.474	0.576	0.620	0.559	0.248	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	2.582		2.200	2.310	2.980	2.453	1.580	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	322,042		88	12,178	49,878	252,147	7,751	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Molybdenum (klbs)	23,980		7	1,211	5,589	16,873	300	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	647		0	53	227	357	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	2,934		1	214	1,089	1,570	60	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade Mineralized Material																																					
Beginning Inventory(kt)	267,836		267,836	267,836	267,827	266,951	262,814	250,946	244,336	228,694	194,243	169,188	151,747	142,973	134,252	121,875	113,207	102,715	98,885	95,743	89,722	68,916	40,782	17,228	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	267,836		9	876	4,137	11,868	6,610	15,642	34,451	25,055	17,441	8,774	8,721	12,377	8,668	10,492	3,830	3,142	6,021	20,806	28,134	22,554	17,228	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		267,836	267,827	266,951	262,814	250,946	244,336	228,694	194,243	169,188	151,747	142,973	134,252	121,875	113,207	102,715	98,885	95,743	89,722	68,916	40,782	17,228	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.120%		0.000%	0.183%	0.162%	0.181%	0.196%	0.124%	0.141%	0.139%	0.121%	0.114%	0.114%	0.091%	0.115%	0.131%	0.099%	0.109%	0.129%	0.131%	0.108%	0.103%	0.098%	0.100%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.012%		0.0000%	0.0148%	0.0054%	0.0111%	0.0059%	0.0059%	0.0100%	0.0080%	0.0107%	0.0143%	0.0127%	0.0099%	0.0106%	0.0121%	0.0117%	0.0088%	0.0092%	0.0080%	0.0103%	0.0140%	0.0168%	0.0189%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.136		-	0.272	0.236	0.183	0.141	0.168	0.136	0.143	0.147	0.138	0.136	0.176	0.153	0.124	0.135	0.114	0.125	0.110	0.125	0.121	0.124	0.115	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	1.187		-	1.580	1.395	1.314	0.936	1.096	1.225	1.240	1.368	1.261	1.126	1.044	1.117	1.196	1.083	1.011	1.268	1.038	1.101	1.213	1.380	0.960	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	710,611		-	36	3,129	16,536	51,282	18,139	48,646	105,925	66,565	43,739	22,126	17,496	31,444	24,987	22,785	9,225	8,915	17,336	49,486	63,942	50,889	37,981	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Molybdenum (klbs)	68,426		-	3	103	1,011	1,534	861	3,450	6,057	5,931	5,517	2,463	1,896	2,883	2,303	2,712	743	638	1,056	4,703	8,660	8,724	7,178	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	1,171		-	0	7	24	54	36	68	158	119	78	38	49	61	35	46	14	13	21	83	109	94	64	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	10,219		-	0	39	175	357	233	616	1,373	1,102	707	318	293	445	333	365	124	128	201	736	1,097	1,045	532	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Mill Mineralized Material																																					
Beginning Inventory(kt)	1,217,069	-	1,217,069	1,217,048	1,213,494	1,196,207	1,143,255	1,083,133	1,029,937	972,431	896,163	829,275	769,932	718,766	667,310	613,448	563,233	509,674	462,114	412,304	361,135	294,740	220,433	151,336	89,073	43,724	9,176	-	-	-	-	-	-	-	-	-	-
Mined (kt)	1,217,069	-	21	3,554	17,287	52,952	60,122	53,196	57,506	76,268	66,888	59,343	51,166	51,456	53,862	50,215	53,559	47,560	49,810	51,169	66,395	74,307	69,097	62,263	45,349	34,548	9,176	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		1,217,048	1,213,494	1,196,207	1,143,255	1,083,133	1,029,937	972,431	896,163	829,275	769,932	718,766	667,310	613,448	563,233	509,674	462,114	412,304	361,135	294,740	220,433	151,336	89,073	43,724	9,176	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.193%	0.000%	0.189%	0.200%	0.208%	0.423%	0.306%	0.253%	0.225%	0.178%	0.168%	0.175%	0.185%	0.169%	0.196%	0.245%	0.173%	0.187%	0.179%	0.195%	0.142%	0.131%	0.134%	0.145%	0.132%	0.158%	0.190%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.021%	0.000%	0.014%	0.019%	0.028%	0.026%	0.026%	0.026%	0.015%	0.012%	0.015%	0.022%	0.026%	0.029%	0.027%	0.026%	0.027%	0.028%	0.024%	0.011%	0.012%	0.014%	0.018%	0.023%	0.029%	0.029%	0.034%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Gold Grade (g/t)	0.219	-	0.474	0.534	0.542	0.454	0.325	0.324	0.244	0.200	0.211	0.212	0.216	0.211	0.208	0.196	0.199	0.197	0.169	0.159	0.167	0.157	0.165	0.163	0.155	0.165	0.203	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	1.667	-	2.200	2.237	2.672	2.409	1.780	1.885	1.707	1.514	1.599	1.938	1.544	1.561	1.721	1.763	1.582	1.495	1.758	1.501	1.398	1.492	1.967	1.394	1.170	1.540	1.210	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	5,188,527	-	88	15,690	79,448	493,297	405,409	296,426	285,020	298,464	248,452	228,495	208,976	192,256	232,409	270,760	204,808	195,858	196,223	219,835	208,539	215,000	203,505	198,823	131,970	120,341	38,436	-	-	-	-	-	-	-	-	-	-
Contained Molybdenum (klbs)	576,186	-	7	1,475	7,109	32,840	34,247	31,068	19,303	19,907	21,706	28,612	29,547	33,051	31,523	28,751	32,380	29,511	26,049	12,889	18,124	22,329	27,500	31,007	28,693	21,783	6,777	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	8,568	-	0	61	301	773	628	554	451	491	455	404	356	349	360	316	342	301	271	262	357	374	366	327	226	183	60	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	65,236	-	1	256	1,485	4,101	3,441	3,223	3,156	3,712	3,438	3,698	2,539	2,583	2,980	2,846	2,723	2,286	2,816	2,470	2,985	3,564	4,369	2,790	1,706	1,711	357	-	-	-	-	-	-	-	-	-	-
Leach Mineralized Material																																					
Beginning Inventory(kt)	209,637		209,637	203,579	188,707	163,348	141,899	111,878	86,171	73,115	72,665	72,106	67,068	56,230	44,906	39,006	17,023	16,817	10,562	3,478	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	209,637		6,058	14,872	25,359	21,449	30,021	25,707	13,056	450	559	5,038	10,838	11,324	5,900	21,983	206	6,255	7,084	3,478	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		203,579	188,707	163,348	141,899	111,878	86,171	73,115	72,665	72,106	67,068	56,230	44,906	39,006	17,023	16,817	10,562	3,478	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.036%		0.0220%	0.0430%	0.0490%	0.0432%	0.0190%	0.0330%	0.0410%	0.0750%	0.0060%	0.0060%	0.0110%	0.0190%	0.0340%	0.0450%	0.0750%	0.0540%	0.0560%	0.0760%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.265		0.285	0.437	0.409	0.312	0.258	0.230	0.178	0.170	0.183	0.208	0.227	0.227	0.189	0.194	0.158	0.147	0.148	0.166	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	1.946		1.410	2.730	2.710	1.927	1.580	1.860	1.320	1.120	1.510	2.590	2.430	2.070	1.480	1.580	2.340	1.560	1.630	1.960	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	165,400		2,938	14,098	27,394	20,442	12,575	18,702	11,801	744	74	666	2,628	4,743	4,422	21,809	341	7,447	8,746	5,827	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	1,784		56	209	333	215	249	190	75	2	3	34	79	83	36	137	1	30	34	19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	13,115		275	1,305	2,209	1,329	1,525	1,537	554	16	27	420	847	754	281	1,117	15	314	371	219	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste																																					
Beginning Inventory(kt)	611,264		611,264	609,343	607,769	603,415	582,816	572,960	551,864	522,425	499,143	466,589	430,970	392,975	355,755	315,516	287,713	241,479	195,293	152,985	110,735	79,113	56,657	36,085	15,397	6,318	1,462	-	-	-	-	-	-	-	-	-	-
Mined (kt)	611,264		1,921	1,574	4,354	20,599	9,856	21,096	29,439	23,282	32,554	35,619	37,995	37,220	40,239	27,803	46,234	46,186	42,308	42,250	31,622	22,456	20,572	20,688	9,079	4,856	1,462	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-		609,343	607,769	603,415	582,816	572,960	551,864	522,425	499,143	466,589	430,970	392,975	355,755	315,516	287,713	241,479	195,293	152,985	110,735	79,113	56,657	36,085	15,397	6,318	1,462	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined (kt)	2,037,970		8,000	20,000	47,000	95,000	99,999	99,999	100,001	100,000	100,001	100,000	99,999	100,000	100,001	100,001	99,999	100,001	99,202	96,897	98,017	96,763	89,669	82,951	54,428	39,404	10,638	-	-	-	-	-	-	-	-	-	-
Waste to Mineralized Material Ratio	0.43		0.32	0.09	0.10	0.28	0.11	0.27	0.42	0.30	0.48	0.55	0.61	0.59	0.67	0.39	0.86	0.86	0.74	0.77	0.48	0.30	0.30	0.33	0.20	0.14	0.16	-	-	-	-	-	-	-	-	-	-
Mine Re-handle (kt)	226,014		-	-	5,710	-	-	3,588	3,600	3,600	3,600	3,600	3,595	3,595	3,595	3,595	2,972	-	-	-	-	-	-	-	-	11,540	36,586	45,596	45,434	45,808	-	-	-	-	-	-	-	-
Process Plant Operations - Concentrator																																					
Supergene Oxide Mineralized Material																																					
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	80,264	-	-	-	-	-	89	1,855	10,237	5,751	3,646	3,600	3,605	4,226	5,243	7,702	3,661	3,248	4,161	4,982	-	-	-	-	-	-	-	624	3,083	630	4,385	9,535	-	-	-	-	-	-
Mined Mineralized Material - Processed (kt)	80,264	-	-	-	-	-	89	1,855	10,237	5,751	3,646	3,600	3,605	4,226	5,243	7,702	3,661	3,248	4,161	4,982	-	-	-	-	-	-	-	624	3,083	630	4,385	9,535	-	-	-	-	-	-
Ending Mineralized Material Inventory	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.240%		0.000%	0.000%	0.000%	0.000%	0.227%	0.188%	0.308%	0.378%	0.363%	0.257%	0.236%	0.204%	0.191%	0.185%	0.198%	0.187%	0.297%	0.283%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.184%	0.149%	0.151%	0.203%	0.172%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.020%		0.0000%	0.0000%	0.0000%	0.0000%	0.0118%	0.0210%	0.0178%	0.0304%	0.0299%	0.0301%	0.0295%	0.0306%	0.0294%	0.0218%	0.0220%	0.0179%	0.0101%	0.0074%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0067%	0.0180%	0.0019%	0.0084%	0.0121%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.315		-	-	-	-	0.409	0.338	0.232	0.307	0.422	0.478	0.491	0.475	0.464	0.394	0.614	0.546	0.170	0.163	-	-	-	-	-	-	-	0.099	0.124	0.176	0.123	0.143	-	-	-	-	-	-
Silver Grade (g/t)	1.698		-	-	-	-	1.920	1.770	1.390	1.303	1.819	2.050	2.294	2.471	2.556	2.185	2.938	2.547	1.460	1.210	-	-	-	-	-	-	-	0.692	1.149	0.662	0.642	1.032	-	-	-	-	-	-
Contained Copper (klbs)	424,581		-	-	-	-	445	7,688	69,623	47,904	29,166	20,397	18,769	18,996	22,021	31,345	15,977	13,412	27,245	31,083	-	-	-	-	-	-	-	2,533	10,159	2,100	19,599	36,117	-	-	-	-	-	-
Contained Molybdenum (klbs)	35,342		-	-	-	-	23	859	4,028	3,853	2,407	2,389	2,341	2,848	3,399	3,705	1,772	1,282	927	813	-	-	-	-	-	-	-	92	1,220	26	812	2,547	-	-	-	-	-	-
Contained Gold (kozs)	814		-	-	-	-	1	20	76	57	50	55	57	64	78	98	72	57	23	26	-	-	-	-	-	-	-	2	12	4	17	44	-	-	-	-	-	-
Contained Silver (kozs)	4,383		-	-	-	-	5	106	458	241	213	237	266	336	431	541	346	266	195	194	-	-	-	-	-	-	-	14	114	13	91	316	-	-	-	-	-	-
Recovery Copper (%)	61.90%		0.00%	0.00%	0.00%	0.00%	53.74%	63.30%	64.81%	62.79%	60.21%	55.64%	57.50%	60.65%	62.57%	66.56%	58.89%	60.70%	51.85%	68.90%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	57.99%	62.13%	68.15%	59.16%	62.95%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Molybdenum (%)	52.30%		0.00%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Copper Concentrate (kt)	426		-	-	-	-	0	8	73	49	28	18	17	19	22	34	15	13	23	35	-	-	-	-	-	-	-	2	10	2	19	37	-	-	-	-	-	-
Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Copper (klbs)	262,809		-	-	-	-	239	4,867	45,122	30,080	17,561	11,349	10,793	11,520	13,778	20,863	9,408	8,141	14,127	21,418	-	-	-	-	-	-	-	1,469	6,312	1,432	11,594	22,736	-	-	-	-	-	-
Recovered Gold (kozs)	561		-	-	-	-	1	14	53	39	34	38	39	45	54	67	50	39	16	18	-	-	-	-	-	-	-	1	8	2	12	30	-	-	-	-	-	-
Recovered Silver (kozs)	2,630		-	-	-	-	3	63	275	145	128	142	160	201	258	325	208	160	117	116	-	-	-	-	-	-	-	8	68	8	54	190	-	-	-	-	-	-
Molybdenum Concentrate (kt)	15		-	-	-	-	0	0	2	2	1	1	1	1	1	2	1	1	0	0	-	-	-	-	-	-	-	0	1	0	0	1	-	-	-	-	-	-
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Molybdenum (klbs)	18,484		-	-	-	-	12	449	2,106	2,015	1,259	1,249	1,225	1,490	1,778	1,938	927	671	485	425	-	-	-	-	-	-	-	48	638	14	425	1,332	-	-	-	-	-	-
Supergene Sulphide Mineralized Material																																					
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	285,991	-	-	-	-	29,926	25,191	8,946	24,103	14,745	796	-	43	969	22,160	33,538	7,746	8,352	4,109	28,388	11,566	174	-	-	-	-	84	7,132	21,829	2,421	14,292	19,481	-	-	-	-	-	-
Mined Mineralized Material - Processed (kt)	285,991	-	-	-	-	29,926	25,191	8,946	24,103	14,745	796	-	43	969	22,160	33,538	7,746	8,352	4,109	28,388	11,566	174	-	-	-	-	84	7,132	21,829	2,421	14,292	19,481	-	-	-	-	-	-
Ending Mineralized Material Inventory	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.245%		0.000%	0.000%	0.000%	0.347%	0.397%	0.304%	0.256%	0.225%	0.219%	0.000%	0.137%	0.155%	0.247%	0.289%	0.250%	0.219%	0.197%	0.210%	0.209%	0.253%	0.000%

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Molybdenum Concentrate (kt)	53		-	-	-	6	9	3	3	2	0	-	0	0	6	9	2	2	1	3	2	0	-	-	-	0	1	1	2	0	1	1	-	-	-	-	-
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Molybdenum (klbs)	65,686		-	-	-	7,693	10,631	3,094	3,780	2,414	168	-	1	72	7,359	11,717	2,831	2,687	763	3,862	2,547	42	-	-	-	-	-	19	827	2,339	92	1,316	1,433	-	-	-	-	-
Hypogene Mineralized Material																																					
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	850,814	-	-	-	-	4,698	21,802	35,785	11,112	24,921	40,991	41,902	42,344	41,135	17,677	3,902	35,255	35,102	38,398	11,778	34,023	45,999	45,543	45,035	45,349	46,088	45,678	37,839	20,522	42,757	26,674	8,504	-	-	-	-	-
Mined Mineralized Material - Processed (kt)	850,814	-	-	-	-	4,698	21,802	35,785	11,112	24,921	40,991	41,902	42,344	41,135	17,677	3,902	35,255	35,102	38,398	11,778	34,023	45,999	45,543	45,035	45,349	46,088	45,678	37,839	20,522	42,757	26,674	8,504	-	-	-	-	-
Ending Mineralized Material Inventory	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.166%		0.000%	0.000%	0.000%	0.245%	0.261%	0.267%	0.260%	0.196%	0.197%	0.200%	0.200%	0.187%	0.190%	0.192%	0.179%	0.188%	0.168%	0.154%	0.141%	0.148%	0.152%	0.162%	0.132%	0.143%	0.118%	0.103%	0.097%	0.116%	0.120%	0.132%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.022%		0.0000%	0.0000%	0.0000%	0.0186%	0.0251%	0.0297%	0.0248%	0.0142%	0.0171%	0.0250%	0.0290%	0.0335%	0.0332%	0.0245%	0.0312%	0.0305%	0.0272%	0.0140%	0.0114%	0.0134%	0.0187%	0.0240%	0.0287%	0.0262%	0.0200%	0.0121%	0.0111%	0.0131%	0.0090%	0.0070%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.201		-	-	-	0.426	0.361	0.350	0.361	0.257	0.249	0.242	0.233	0.217	0.233	0.233	0.220	0.210	0.175	0.186	0.178	0.178	0.186	0.182	0.155	0.152	0.138	0.123	0.135	0.139	0.146	0.153	-	-	-	-	-
Silver Grade (g/t)	1.640		-	-	-	2.750	2.100	2.000	2.250	1.830	1.720	2.220	1.630	1.670	1.740	2.130	1.710	1.520	1.850	2.130	1.430	1.660	2.270	1.560	1.170	1.395	1.268	1.198	1.065	1.267	1.437	1.280	-	-	-	-	-
Contained Copper (klbs)	3,109,005		-	-	-	25,358	125,450	210,643	63,694	107,685	178,028	184,756	186,705	169,585	74,045	16,517	139,126	145,487	142,217	39,988	105,761	150,087	152,616	160,842	131,970	145,782	118,351	86,204	43,895	108,881	70,586	24,747	-	-	-	-	-
Contained Molybdenum (klbs)	411,061		-	-	-	1,923	12,064	23,431	6,075	7,802	15,453	23,094	27,072	30,380	12,938	2,108	24,250	23,603	23,026	3,635	8,551	13,589	18,776	23,828	28,693	26,592	20,096	10,131	5,006	12,343	5,283	1,317	-	-	-	-	-
Contained Gold (kozs)	5,492		-	-	-	64	253	403	129	206	328	326	317	287	132	29	249	237	216	70	195	263	272	264	226	226	203	149	89	191	125	42	-	-	-	-	-
Contained Silver (kozs)	44,865		-	-	-	415	1,472	2,301	804	1,466	2,267	2,991	2,219	2,209	989	267	1,938	1,715	2,284	807	1,564	2,455	3,324	2,259	1,706	2,067	1,862	1,458	703	1,741	1,233	350	-	-	-	-	-
Recovery Copper (%)	92.20%		0.00%	0.00%	0.00%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Molybdenum (%)	78.60%		0.00%	0.00%	0.00%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Gold (%)	66.00%		0.00%	0.00%	0.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Silver (%)	50.00%		0.00%	0.00%	0.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Copper Concentrate (kt)	4,644		-	-	-	38	187	315	95	161	266	276	279	253	111	25	208	217	212	60	158	224	228	240	197	218	177	129	66	163	105	37	-	-	-	-	-
Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Copper (klbs)	2,866,502		-	-	-	23,380	115,665	194,212	58,726	99,286	164,142	170,345	172,142	156,357	68,270	15,228	128,274	134,139	131,124	36,869	97,512	138,380	140,712	148,296	121,676	134,411	109,120	79,480	40,471	100,388	65,081	22,817	-	-	-	-	-
Recovered Gold (kozs)	3,625		-	-	-	42	167	266	85	136	217	215	209	189	87	19	165	156	143	46	129	174	180	174	149	149	134	98	59	126	83	28	-	-	-	-	-
Recovered Silver (kozs)	22,432		-	-	-	208	736	1,151	402	733	1,133	1,495	1,110	1,104	494	134	969	858	1,142	403	782	1,227	1,662	1,129	853	1,033	931	729	351	871	616	175	-	-	-	-	-
Molybdenum Concentrate (kt)	262		-	-	-	1	8	15	4	5	10	15	17	19	8	1	15	15	15	15	2	5	9	12	15	18	17	13	6	3	8	3	1	-	-	-	-	-
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Molybdenum (klbs)	323,094		-	-	-	1,511	9,483	18,417	4,775	6,132	12,146	18,152	21,279	23,879	10,170	1,657	19,060	18,552	18,098	2,857	6,721	10,681	14,758	18,729	22,553	20,901	15,795	7,963	3,935	9,702	4,153	1,035	-	-	-	-	-
Total Mill Mineralized Material																																					
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	1,217,069		-	-	-	34,624	47,082	46,586	45,452	45,417	45,433	45,502	45,992	46,330	45,080	45,142	46,662	46,702	46,668	45,148	45,589	46,173	45,543	45,035	45,349	46,088	45,762	45,596	45,434	45,808	45,351	37,521	-	-	-	-	-
Mined Mineralized Material - Processed (kt)	1,217,069		-	-	-	34,624	47,082	46,586	45,452	45,417	45,433	45,502	45,992	46,330	45,080	45,142	46,662	46,702	46,668	45,148	45,589	46,173	45,543	45,035	45,349	46,088	45,762	45,596	45,434	45,808	45,351	37,521	-	-	-	-	-
Ending Mineralized Material Inventory	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.189%	0.000%	0.000%	0.000%	0.000%	0.333%	0.334%	0.271%	0.269%	0.228%	0.211%	0.205%	0.203%	0.188%	0.218%	0.263%	0.192%	0.193%	0.182%	0.203%	0.158%	0.148%	0.152%	0.162%	0.132%	0.143%	0.118%	0.108%	0.113%	0.114%	0.136%	0.160%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.021%	0.000%	0.000%	0.000%	0.000%	0.022%	0.031%	0.029%	0.017%	0.016%	0.018%	0.025%	0.029%	0.033%	0.031%	0.028%	0.031%	0.029%	0.025%	0.012%	0.013%	0.013%	0.019%	0.024%	0.029%	0.026%	0.020%	0.012%	0.011%	0.012%	0.009%	0.008%	0.000%	0.000%	0.000%	0.000%	0.000%
Gold Grade (g/t)	0.217		-	-	-	0.419	0.373	0.346	0.285	0.257	0.263	0.261	0.253	0.241	0.256	0.243	0.246	0.228	0.172	0.166	0.187	0.178	0.186	0.182	0.155	0.152	0.138	0.122	0.139	0.144	0.144	0.151	-	-	-	-	-
Silver Grade (g/t)	1.659		-	-	-	2.474	1.998	1.997	1.850	1.734	1.745	2.207	1.682	1.737	1.987	1.969	1.801	1.588	1.791	1.563	1.534	1.662	2.270	1.560	1.170	1.395	1.267	1.146	1.114	1.254	1.265	1.125	-	-	-	-	-
Contained Copper (klbs)	5,080,992		-	-	-	254,531	346,376	278,287	269,350	228,730	211,037	205,153	205,604	191,892	216,736	261,545	197,795	199,223	187,308	202,499	159,053	151,058	152,616	160,842	131,970	145,782	118,556	108,254	113,014	115,436	135,618	132,728	-	-	-	-	-
Contained Molybdenum (klbs)	571,998		-	-	-	16,633	32,414	30,207	17,330	16,270	16,211	25,483	29,416	33,365	30,407	28,216	31,435	30,022	25,411	11,833	13,421	13,669	18,776	23,828	28,693	26,592	20,131	11,804	10,700	12,545	8,611	6,604	-	-	-	-	-
Contained Gold (kozs)	8,490		-	-	-	467	565	518	416	376	385	381	375	359	371	353	369	342	258	241	274	265	272	264	226	226	203	178	203	212	209	182	-	-	-	-	-
Contained Silver (kozs)	64,927		-	-	-	2,754	3,024	2,990	2,703	2,532	2,549	3,228	2,487	2,587	2,880	2,857	2,702	2,384	2,688	2,269	2,248	2,467	3,324	2,259	1,706	2,067	1,864	1,680	1,628	1,847	1,845	1,357	-	-	-	-	-
Recovery Copper (%)	86.46%	0.00%	0.00%	0.00%	0.00%	80.34%	85.25%	89.54%	79.79%	82.57%	87.61%	88.57%	89.03%	88.90%	83.84%	82.50%	87.91%	88.61%	85.28%	81.63%	89.68%	92.16%	92.20%	92.20%	92.20%	92.20%	92.17%	89.14%	82.68%	91.35%	83.35%	77.08%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Molybdenum (%)	71.20%	0.00%	0.00%	0.00%	0.00%	55.34%	62.09%	72.70%	61.52%	64.91%	74.65%	76.13%	76.50%	76.25%	63.49%	54.26%	72.59%	72.98%	76.13%	60.38%	69.06%	78.45%	78.60%	78.60%	78.60%	78.60%	78.55%	74.87%	64.61%	78.18%	68.44%	57.55%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Gold (%)	67.06%	0.00%	0.00%	0.00%	0.00%	58.59%	67.66%	66.67%	68.07%	67.36%	66.44%	66.46%	66.46%	66.60%	67.93%	68.75%	66.97%	65.92%	66.49%	68.12%	66.87%	66.02%	66.00%	66.00%	66.00%	66.00%	66.00%	66.49%	67.68%	66.30%	67.21%	68.31%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovery Silver (%)	53.09%	0.00%	0.00%	0.00%	0.00%	58.49%	55.13%	52.31%	57.03%	54.21%	51.11%	50.74%	51.08%	51.46%	56.57%	59.06%	52.83%	52.81%	51.50%	56.45%	53.04%	50.05%	50.00%	50.00%	50.00%	50.00%	50.01%	51.32%	55.68%	50.57%	53.32%	57.42%	0.00%	0.00%	0.00%	0.00%	0.00%
Copper Concentrate (kt)	7,116		-	-	-	331	478	404	348	306	300	294	297	276	294	350	282	286	259	268	231	226	228	240	197	218	177	156	151	171	183	166	-	-	-	-	-
Copper Concentrate Grade (%)	28.00%	0.00%	0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Copper (klbs)	4,392,828		-	-	-	204,486	295,287	249,174	214,906	188,854	184,897	181,694	183,041	170,591	181,711	215,762	173,885	176,528	159,745	165,306	142,644	139,221	140,712	148,296	121,676	134,411	109,275	96,493	93,436	105,456	113,034	102,306	-	-	-	-	-
Recovered Gold (kozs)	5,693		-	-	-	320	382	346	283	253	256	253	249	239	252	243	247	229	172	164	183	175	180	174	149	149	134	119	137	141	141	124	-	-	-	-	-
Recovered Silver (kozs)	34,470		-	-	-	1,611	1,667	1,564	1,541	1,373	1,303	1,638	1,271	1,331	1,629	1,688	1,428	1,259	1,384	1,281	1,193	1,235	1,662	1,129	853	1,033	932	862	906	934	984	779	-	-	-	-	-
Molybdenum Concentrate (kt)	330		-	-	-	7	16	18	9	9	11	16	18	21	16	12	18	18	16	6	8	9	12	15	18	17	13	7	6	8	5	3	-	-	-	-	-
Molybdenum Concentrate Grade (%)	56.00%	0.00%	0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Molybdenum (klbs)	407,264		-	-	-	9,205	20,125	21,960	10,661	10,561	13,573	19,402	22,505	25,440	19,306	15,311	22,818	21,909	19,345	7,145	9,268	10,723	14,758	18,729	22,553	20,901	15,814	8,838	6,913	9,807	5,893	3,800	-	-	-	-	-
Heap Leach																																					
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Heap Leach (kt)	209,636		-	3,900	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	4,986	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material - Processed (kt)	209,636		-	3,900	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	4,986	-	-	-	-	-	-	-	-	-	-	-
Ending Mineralized Material Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.036%		0.000%	0.043%	0.049%	0.046%	0.032%	0.040%	0.045%	0.045%	0.029%	0.020%	0.011%	0.019%	0.028%	0.045%	0.046%	0.051%	0.051%	0.053%	0.033%	0.033%	0.021%	0.019%	0.021%	0.043%	0.046%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Gold Grade (g/t)	0.265		-	0.437	0.409	0.334	0.324	0.308	0.279	0.420	0.338	0.207	0.227	0.227	0.202	0.194	0.193	0.162	0.160	0.180	0.230	0.230	0.254	0.258	0.261	0.283	0.392	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	1.946		-	2.730	2.710	2.122	2.075	2.233	1.929	2.648	1.927	2.031	2.430	2.070	1.729	1.580	1.597	1.566	1.675	1.627	1.860	1.860	1.619	1.580	1.559	1.706	2.570	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	165,400		-	3,697	9,857	9,157	6,468	8,050	8,954	9,131	5,863	3,932	2,213	3,822	5,592	9,053	9,189	10,294	10,231	10,744	6,639	6,639	4,213	3,822	4,143	8,641	5,056	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	1,784		-	55	120	98	95	90	82	123	99	61	67	67	59	57	57	47	47	53	67	67	75	76	76	83	63	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs)	13,115		-	342	795	622	609	655	566	777	565	596	713	607	507	464	469	460	491	477	546	546	475	464	457	501	412	-	-	-	-	-	-	-	-	-	-	-
Recovery Copper (%)	18%		18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	0%	0%	0%	0%	0%
Recovery Gold (%)	80%		80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	0%	0%	0%	0%	0%
Recovery Silver (%)	26%		26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	0%	0%	0%	0%	0%
Copper Precipitate (kt)	23		-	1	1	1	1	1	1	1	1	1	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Copper Precipitate Grade	60%		60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	0%	0%	0%	0%	0%
Recovered Copper (klbs)	29,772		-	665	1,774	1,648	1,164	1,449	1,612	1,644	1,055	708	398	688	1,007	1,629	1,654	1,853	1,842	1,934	1,195	1,195	758	688	746	1,555	910	-	-	-	-	-	-	-	-	-	-	-
Recovered Gold Dore - Heap Leach (kozs)	1,427		-	44	96	78	76	72	66	99	79	48	53	53	48	46	45	38	38	42	54	54	60	61	61	66	50	-	-	-	-	-	-	-	-	-	-	-
Total Recovered Gold Dore (kozs)	1,427		-	44	96	78	76	72	66	99	79	48	53	53	48	46	45	38	38	42	54	54	60	61	61	66	50	-	-	-	-	-	-	-	-	-	-	-
Recovered Silver Dore (kozs)	3,410		-	89	207	162	158	170	147	202	147	155	185	158	132	121	122	119	128	124	142	142	123	121	119	130	107	-	-	-	-	-	-	-	-	-	-	-
Payable Metals																																					
Copper Concentrate																																					
Payable Copper (klbs)	4,239,079	-	-	-	-	197,329	284,952	240,453	207,384	182,244	178,425	175,335	176,635	164,620	175,351	208,210	167,799	170,350	154,154	159,520	137,652	134,348	135,787	143,106	117,418	129,707	105,450	93,115	90,166	101,765	109,078	98,725	-	-	-	-	-
Payable Gold (kozs)	5,551	-	-	-	-	312	373	337	276	247	249	247	243	233	246	237	241	223	167	160	179	170	175	170	145	145	131	116	134	137	137	121	-	-	-	-	-
Payable Silver (kozs)	32,746	-	-	-	-	1,530	1,584	1,486	1,464	1,304	1,238	1,556	1,207	1,265	1,548	1,603	1,356	1,196	1,315	1,217	1,133	1,173	1,579	1,073	810	982	886	819	861	887	934	740	-	-	-	-	-
Molybdenum Concentrates																																					
Payable Molybdenum (klbs)	346,174	-	-	-	-	7,824	17,107	18,666	9,062	8,977	11,537	16,491	19,129	21,624	16,410	13,015	19,395	18,623	16,444	6,073	7,878	9,114	12,544	15,920	19,170	17,766	13,442	7,512	5,876	8,336	5,009	3,230	-	-	-	-	-
Gold/Silver Dore																																					
Payable Metal Gold (kozs)	1,399		-	43	94	77	75	71	64	97	78	48	52	52	47	45	44	37	37	41	53	53	58	59	60	65	49	-	-	-	-	-	-	-	-	-	-	-
Payable Metal Silver (kozs)	3,342		-	87	203	159	155	167	144	198	144	152	182	155	129	118	119	117	125	122	139	139	121	118	117	128	105	-	-	-	-	-	-	-	-	-	-	-
Copper Precipitate																																					
Payable Metal (klbs)	28,730		-	642	1,712	1,591	1,123	1,398	1,555	1,586	1,018	683	384	664	971	1,572	1,596	1,788	1,777	1,866	1,153	1,153	732	664	720	1,501	878	-	-	-	-	-	-	-	-	-	-	-
Income Statement (C$000)																																					
Copper (C$/lb)	$ 4.50		$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ 4.50	$ -	$ -	$ -	$ -	$ -
Molybdenum (C$/lb)	$ 17.50		$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ 17.50	$ -	$ -	$ -	$ -	$ -
Gold (C$/oz)	$ 2,125.00		$ 2,125.00	$ 2,125.00	$ 2,125.00

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Gold Refining Charges	$ 1,855		$ -	$ 57	$ 125	$ 102	$ 99	$ 94	$ 85	$ 128	$ 103	$ 63	$ 69	$ 69	$ 62	$ 59	$ 59	$ 49	$ 49	$ 55	$ 70	$ 70	$ 78	$ 79	$ 79	$ 86	$ 65	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Silver Refining Charges	$ 1,023		$ -	$ 27	$ 62	$ 49	$ 47	$ 51	$ 44	$ 61	$ 44	$ 46	$ 56	$ 47	$ 40	$ 36	$ 37	$ 36	$ 38	$ 37	$ 43	$ 43	$ 37	$ 36	$ 36	$ 39	$ 32	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Transportation	$ 1,505		$ -	$ 41	$ 94	$ 75	$ 73	$ 75	$ 66	$ 93	$ 70	$ 63	$ 74	$ 66	$ 56	$ 52	$ 52	$ 49	$ 51	$ 52	$ 61	$ 61	$ 57	$ 56	$ 56	$ 61	$ 49	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Copper Precipitate																																					
Treatment Charges	$ 1,688		$ -	$ 38	$ 101	$ 93	$ 66	$ 82	$ 91	$ 93	$ 60	$ 40	$ 23	$ 39	$ 57	$ 92	$ 94	$ 105	$ 104	$ 110	$ 68	$ 68	$ 43	$ 39	$ 42	$ 88	$ 52	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Refining Charges	$ 2,233		$ -	$ 50	$ 133	$ 124	$ 87	$ 109	$ 121	$ 123	$ 79	$ 53	$ 30	$ 52	$ 75	$ 122	$ 124	$ 139	$ 138	$ 145	$ 90	$ 90	$ 57	$ 52	$ 56	$ 117	$ 68	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Transportation	$ 3,864		$ -	$ 86	$ 230	$ 214	$ 151	$ 188	$ 209	$ 213	$ 137	$ 92	$ 52	$ 89	$ 131	$ 211	$ 215	$ 240	$ 239	$ 251	$ 155	$ 155	$ 98	$ 89	$ 97	$ 202	$ 118	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Cash Operating Cost	$ 17,539,221		8,084	35,907	73,301	649,042	774,548	755,154	747,079	733,909	733,965	732,963	734,523	731,034	734,424	762,194	735,248	744,376	732,564	709,211	690,794	685,747	678,582	675,641	590,260	568,410	478,320	409,785	415,131	417,414	428,109	373,504	-	-	-	-	-
Royalty	$ 998,570		$ -	$ 2,486	$ 5,419	$ 46,961	$ 62,524	$ 56,930	$ 46,289	$ 43,968	$ 43,725	$ 43,436	$ 43,251	$ 42,714	$ 42,054	$ 43,328	$ 42,203	$ 40,791	$ 35,519	$ 31,343	$ 31,443	$ 31,229	$ 33,507	$ 35,023	$ 32,332	$ 33,281	$ 27,178	$ 19,763	$ 19,767	$ 22,021	$ 21,431	$ 18,653	$ -	$ -	$ -	$ -	$ -	$ -
Property Tax	$ 3,003	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 101	$ 102	$ -	$ -	$ -	$ -	$ -	$ -
Carbon Tax	$ 1,866,838	$ 1,801	$ 5,871	$ 12,955	$ 57,637	$ 72,421	$ 74,308	$ 73,500	$ 74,258	$ 75,983	$ 76,302	$ 76,730	$ 77,303	$ 74,243	$ 74,957	$ 76,860	$ 78,343	$ 78,435	$ 71,546	$ 71,388	$ 72,453	$ 71,590	$ 71,017	$ 63,540	$ 63,848	$ 57,689	$ 54,350	$ 53,835	$ 55,201	$ 55,085	$ 43,389	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Salvage Value	$ (22,807)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (22,807)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation & Closure	$ 300,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 25,000	$ 50,000	$ 60,000	$ 60,000	$ 60,000	$ 60,000	$ 45,000
Total Cash Production Cost	$ 20,684,826	$ 9,985	$ 44,364	$ 91,775	$ 753,740	$ 909,592	$ 886,491	$ 866,968	$ 852,235	$ 853,773	$ 852,801	$ 854,604	$ 851,151	$ 850,821	$ 880,579	$ 854,411	$ 863,609	$ 846,618	$ 812,200	$ 793,727	$ 789,529	$ 783,779	$ 781,782	$ 686,232	$ 665,639	$ 563,287	$ 483,998	$ 488,833	$ 471,928	$ 504,726	$ 460,649	$ 50,000	$ 60,000	$ 60,000	$ 60,000	$ 60,000	$ 45,000
Operating Income	$ 20,338,420	$ -	$ (9,985)	$ 52,212	$ 121,405	$ 1,151,260	$ 1,675,700	$ 1,440,691	$ 999,506	$ 903,288	$ 888,170	$ 900,702	$ 941,596	$ 916,809	$ 896,869	$ 936,480	$ 893,600	$ 826,016	$ 616,288	$ 484,649	$ 495,637	$ 490,267	$ 593,444	$ 662,978	$ 642,714	$ 713,644	$ 560,483	$ 334,691	$ 327,687	$ 447,885	$ 391,034	$ 317,699	$ (50,000)	$ (60,000)	$ (60,000)	$ (60,000)	$ (45,000)
Yukon Mining Royalty	$ 2,256,636	$ -	$ 5,860	$ 15,366	$ 82,588	$ 147,812	$ 118,192	$ 63,273	$ 51,048	$ 48,734	$ 72,237	$ 121,489	$ 119,377	$ 116,630	$ 121,175	$ 115,874	$ 107,710	$ 81,919	$ 64,870	$ 66,012	$ 66,147	$ 79,467	$ 88,527	$ 85,394	$ 93,933	$ 74,277	$ 46,277	$ 45,441	$ 60,259	$ 53,512	$ 43,236	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income before Depreciation	$ 18,081,784	$ -	$ (9,985)	$ 46,352	$ 106,039	$ 1,068,672	$ 1,527,888	$ 1,322,499	$ 936,234	$ 852,240	$ 839,436	$ 828,464	$ 820,107	$ 797,432	$ 780,239	$ 815,305	$ 777,726	$ 718,306	$ 534,370	$ 419,779	$ 429,626	$ 424,120	$ 513,977	$ 574,451	$ 557,321	$ 619,711	$ 486,206	$ 288,414	$ 282,246	$ 387,625	$ 337,522	$ 274,463	$ (50,000)	$ (60,000)	$ (60,000)	$ (60,000)	$ (45,000)
Capital Cost Depreciation	$ 1,851,365	$ -	$ 46,352	$ 106,039	$ 866,269	$ 602,884	$ 229,822	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Additional Deduction (Class 41A Assets)	$ 1,766,101	$ -	$ -	$ 187,272	$ 889,364	$ 689,465	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Depreciation	$ 751,278		$ -	$ 15,132	$ 35,640	$ 42,654	$ 42,961	$ 39,831	$ 37,977	$ 37,736	$ 47,717	$ 33,317	$ 30,412	$ 32,729	$ 36,372	$ 39,119	$ 37,554	$ 28,645	$ 28,368	$ 29,680	$ 26,263	$ 20,724	$ 17,192	$ 15,330	$ 13,167	$ 14,823	$ 9,395	$ 9,281	$ 8,452	$ 20,807	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Depreciation	$ 4,368,744	$ -	$ 46,352	$ 106,039	$ 1,068,672	$ 1,527,888	$ 961,941	$ 42,961	$ 39,831	$ 37,977	$ 37,736	$ 47,717	$ 33,317	$ 30,412	$ 32,729	$ 36,372	$ 39,119	$ 37,554	$ 28,645	$ 28,368	$ 29,680	$ 26,263	$ 20,724	$ 17,192	$ 15,330	$ 13,167	$ 14,823	$ 9,395	$ 9,281	$ 8,452	$ 20,807	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income After Depreciation	$ 13,713,040	(9,985)	-	-	-	360,558	893,272	812,409	801,459	790,728	772,390	764,115	749,827	782,576	741,354	679,188	496,816	391,134	401,258	394,440	487,714	553,727	540,128	604,381	473,039	273,591	272,851	378,344	329,070	253,656	(50,000)	(60,000)	(60,000)	(60,000)	(45,000)
Tax Loss Carry Forward Applied	$ (41,985)	-	-	-	-	(41,985)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Income After Tax Loss Carry Forward	$ 13,671,055	(9,985)	-	-	-	318,573	893,272	812,409	801,459	790,728	772,390	764,115	749,827	782,576	741,354	679,188	496,816	391,134	401,258	394,440	487,714	553,727	540,128	604,381	473,039	273,591	272,851	378,344	329,070	253,656	(50,000)	(60,000)	(60,000)	(60,000)	(45,000)
Taxable Income	$ 13,671,055	$ (9,985)	$ -	$ -	$ -	$ 318,573	$ 893,272	$ 812,409	$ 801,459	$ 790,728	$ 772,390	$ 764,115	$ 749,827	$ 782,576	$ 741,354	$ 679,188	$ 496,816	$ 391,134	$ 401,258	$ 394,440	$ 487,714	$ 553,727	$ 540,128	$ 604,381	$ 473,039	$ 273,591	$ 272,851	$ 378,344	$ 329,070	$ 253,656	$ (50,000)	$ (60,000)	$ (60,000)	$ (60,000)	$ (45,000)
Taxes at 27%	$ 3,693,881	-	-	-	-	86,015	241,184	219,350	216,394	213,497	208,545	206,311	202,453	211,296	200,166	183,381	134,140	105,606	108,340	106,499	131,683	149,506	145,835	163,183	127,720	73,869	73,670	102,153	88,849	68,487	(13,500)	(16,200)	(16,200)	(16,200)	(12,150)
Net Income After Taxes	$ 10,019,159	(9,985)	-	-	-	274,543	652,089	593,058	585,065	577,232	563,845	557,804	547,374	571,280	541,189	495,807	362,676	285,528	292,918	287,941	356,031	404,221	394,294	441,198	345,318	199,721	199,181	276,191	240,221	185,169	(36,500)	(43,800)	(43,800)	(43,800)	(32,850)
Cash Flow																																				
Net Income before Depreciation plus Tax Loss Carry Forward	$ 18,081,784	$ (9,985)	$ 46,352	$ 106,039	$ 1,068,672	$ 1,527,888	$ 1,322,499	$ 936,234	$ 852,240	$ 839,436	$ 828,464	$ 820,107	$ 797,432	$ 780,239	$ 815,305	$ 777,726	$ 718,306	$ 534,370	$ 419,779	$ 429,626	$ 424,120	$ 513,977	$ 574,451	$ 557,321	$ 619,711	$ 486,206	$ 288,414	$ 282,246	$ 387,625	$ 337,522	$ 274,463	$ (50,000)	$ (60,000)	$ (60,000)	$ (60,000)	$ (45,000)
Working Capital																																				
Account Receivable	$ -	$ -	$ (15,876)	$ (19,168)	$ (278,107)	$ (111,829)	$ 42,429	$ 75,733	$ 18,239	$ 2,232	$ (1,900)	$ (7,019)	$ 4,642	$ 3,332	$ (11,403)	$ 9,598	$ 37,269	$ 27,297	$ 1,230	$ 1,573	$ (16,015)	$ (11,102)	$ 19,038	$ (8,275)	$ 42,002	$ 50,150	$ 357	$ (16,980)	$ 151,202	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Accounts Payable	$ -	$ 664	$ 2,287	$ 3,073	$ 47,321	$ 10,316	$ (1,594)	$ (664)	$ (1,082)	$ 5	$ (82)	$ 128	$ (287)	$ 279	$ 2,282	$ (2,215)	$ 750	$ (971)	$ (1,919)	$ (1,514)	$ (415)	$ (589)	$ (242)	$ (7,018)	$ (1,796)	$ (7,405)	$ (5,633)	$ 439	$ 188	$ (34,308)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Inventory - Parts, Supplies	$ -	$ (500)	$ -	$ (5,800)	$ -	$ -	$ -	$ 500	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5,800	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Working Capital	$ -	$ 164	$ (13,589)	$ (21,894)	$ (230,786)	$ (101,513)	$ 40,835	$ 75,569	$ 17,156	$ 2,237	$ (1,982)	$ (6,891)	$ 4,355	$ 3,611	$ (9,121)	$ 9,136	$ 10,348	$ 36,298	$ 25,378	$ (283)	$ 1,158	$ (16,604)	$ (11,344)	$ 12,020	$ (10,070)	$ 40,398	$ 44,517	$ 796	$ (16,792)	$ 116,894	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Expenditures																																				
Initial Capital																																				
Mine	$ 660,597	$ -	$ 116,315	$ 173,786	$ 190,434	$ 180,062	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 2,915,976	$ 98,843	$ 857,902	$ 1,176,654	$ 737,708	$ 44,868	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners Cost	$ 40,892	$ 1,636	$ 11,041	$ 17,175	$ 8,996	$ 2,045	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital																																				
Mine	$ 160,955		$ -	$ -	$ -	$ -	$ 20,456	$ -	$ 20,415	$ 11,883	$ 10,228	$ -	$ -	$ 13,022	$ 9,019	$ 10,152	$ -	$ 34,370	$ -	$ -	$ 20,374	$ 3,527	$ 3,527	$ 3,982	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Tailings Management Facility	$ 326,277		$ -	$ -	$ -	$ 116,054	$ 48,286	$ 28,656	$ 28,656	$ 1,810	$ 787	$ 787	$ 787	$ 1,067	$ 1,307	$ 41,108	$ 39,347	$ 1,302	$ 754	$ 1,302	$ 754	$ 754	$ 1,088	$ 754	$ 1,088	$ 754	$ 754	$ 754	$ 754	$ 6,815	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant (includes Heap Leach)	$ 264,045		$ -	$ -	$ -	$ 5,000	$ 24,991	$ 5,000	$ 5,000	$ 5,000	$ 24,937	$ 25,403	$ 5,000	$ 5,000	$ 31,888	$ 5,000	$ 5,000	$ 5,000	$ 25,403	$ 5,000	$ 15,216	$ 5,000	$ 5,000	$ 5,000	$ 5,804	$ 25,403	$ 5,000	$ 5,000	$ 5,000	$ 5,000	$ 5,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures	$ 4,368,744	$ 100,479	$ 985,259	$ 1,367,615	$ 937,138	$ 348,029	$ 73,277	$ 54,112	$ 33,656	$ 27,226	$ 37,606	$ 36,418	$ 5,787	$ 19,089	$ 42,215	$ 56,261	$ 44,347	$ 40,671	$ 26,157	$ 6,302	$ 36,345	$ 9,281	$ 9,614	$ 9,736	$ 6,891	$ 26,157	$ 5,754	$ 5,754	$ 5,754	$ 11,815	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow before Taxes	$ 13,713,040	$ (100,479)	$ (995,079)	$ (1,334,852)	$ (852,993)	$ 489,857	$ 1,353,098	$ 1,309,221	$ 978,147	$ 842,170	$ 804,067	$ 790,064	$ 807,430	$ 782,699	$ 741,635	$ 749,924	$ 742,514	$ 687,983	$ 544,511	$ 438,855	$ 392,998	$ 415,998	$ 487,758	$ 553,372	$ 562,449	$ 583,484	$ 520,849	$ 327,177	$ 277,288	$ 359,018	$ 454,416	$ 274,463	$ (50,000)	$ (60,000)	$ (60,000)	$ (60,000)	$ (45,000)
Cumulative Cash Flow before Taxes		$ (100,479)	$ (1,095,558)	$ (2,430,411)	$ (3,283,404)	$ (2,793,547)	$ (1,440,449)	$ (131,228)	$ 846,919	$ 1,689,089	$ 2,493,156	$ 3,283,220	$ 4,090,650	$ 4,873,349	$ 5,614,984	$ 6,364,908	$ 7,107,422	$ 7,795,405	$ 8,339,915	$ 8,778,770	$ 9,171,768	$ 9,587,765	$ 10,075,523	$ 10,628,895	$ 11,191,345	$ 11,774,829	$ 12,295,678	$ 12,622,855	$ 12,900,143	$ 13,259,161	$ 13,713,577	$ 13,988,040	$ 13,938,040	$ 13,878,040	$ 13,818,040	$ 13,758,040	$ 13,713,040
						1.0	1.0	1.0	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxes																																				
Income Taxes	$ 3,693,881	$ -	$ -	$ -	$ -	$ 86,015	$ 241,184	$ 219,350	$ 216,394	$ 213,497	$ 208,545	$ 206,311	$ 202,453	$ 211,296	$ 200,166	$ 183,381	$ 134,140	$ 105,606	$ 108,340	$ 106,499	$ 131,683	$ 149,506	$ 145,835	$ 163,183	$ 127,720	$ 73,869	$ 73,670	$ 102,153	$ 88,849	$ 68,487	$ (13,500)	$ (16,200)	$ (16,200)	$ (16,200)	$ (12,150)
Cash Flow after Taxes	$ 10,019,159	$ (100,479)	$ (995,079)	$ (1,334,852)	$ (852,993)	$ 489,857	$ 1,353,098	$ 1,223,207	$ 736,963	$ 622,820	$ 587,673	$ 576,568	$ 598,884	$ 576,387	$ 539,182	$ 538,628	$ 542,349	$ 504,602	$ 410,370	$ 333,249	$ 284,658	$ 309,499	$ 356,075	$ 403,866	$ 416,615	$ 420,301	$ 393,129	$ 253,307	$ 203,618	$ 256,865	$ 365,567	$ 205,976	$ (36,500)	$ (43,800)	$ (43,800)	$ (43,800)	$ (32,850)
Cumulative Cash Flow after Taxes		$ (100,479)	$ (1,095,558)	$ (2,430,411)	$ (3,283,404)	$ (2,793,547)	$ (1,440,449)	$ (217,243)	$ 519,721	$ 1,142,541	$ 1,730,214	$ 2,306,781	$ 2,905,666	$ 3,482,053	$ 4,021,235	$ 4,559,863	$ 5,102,212	$ 5,606,814	$ 6,017,184	$ 6,350,433	$ 6,635,091	$ 6,944,590	$ 7,300,665	$ 7,704,531	$ 8,121,146	$ 8,541,447	$ 8,934,576	$ 9,187,883	$ 9,391,501	$ 9,648,366	$ 10,013,933	$ 10,219,909	$ 10,183,409	$ 10,139,609	$ 10,095,809	$ 10,052,009	$ 10,019,159
						1.0	1.0	1.0	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes

NPV @ 0%	0%	$ 13,713,040
NPV @ 5%	5%	$ 5,767,739
NPV @ 8%	8%	$ 3,473,249
NPV @ 10%	10%	$ 2,454,440
IRR		21.2%
Payback		3.1

Economic Indicators after Taxes

NPV @ 0%	0%	$ 10,019,159
NPV @ 5%	5%	$ 4,058,702
NPV @ 8%	8%	$ 2,334,396
NPV @ 10%	10%	$ 1,568,307
IRR		18.1%
Payback	Years	3.3

23 ADJACENT PROPERTIES

Several Yukon quartz mineral claim blocks and placer claim blocks registered to other owners are held adjacent to, or in the general vicinity of, CMC's claim block. The entire extent of Canadian Creek, including its upper extent within the Casino and Canadian Creek blocks are covered by placer claims held by W. Pilkington and B. Tinnelly, operating as Batavia Mining. Active operations from 2019 onward covering the Canadian Creek "bend", are located within the projected pit shell, and are worked by their owners on a seasonal basis with small heavy equipment. Also, the lower 5 km of Casino Creek are covered by a placer lease held by Sphinx Exploration Inc. The entire extent of Meloy Creek, a tributary of Casino Creek the source of which is near the Bomber Vein, is covered by a separate placer lease held by M. Prins.

The northwestern boundary of the Casino property adjoins the Coffee Creek project of Newmont Mining. The property hosts a structurally controlled gold deposit in metamorphic rocks of the Yukon Tanana terrane and granitoids of mid Cretaceous age. The mineralization is associated with quartz- carbonate and illite alteration and is best described as an orogenic deposit.

The Coffee Creek project is at a pre-feasibility stage of development. Operations will involve gold extraction from four deposits: the Latte, Double-Double, Supremo and Kona pits. As of early March of 2022, the Government of Yukon was satisfied with the environmental assessment of the project, which still requires granting of a Yukon Quartz Mining Licence and a Yukon Water Licence.

The northeastern boundary of the Casino property abuts the "Betty and Hayes" property held by White Gold Corp. This property abuts the northern boundary of the narrow eastern extension of the Casino property. The property has undergone Rotary Air Blast (RAB) drilling for similar orogenic-style gold mineralization to that within the Coffee Creek property. Two significant zones, the Betty Ford, and Betty White targets, have been drilled, with the highest result of 1.08 g/t Au across 50.29 m returned from the Betty Ford target (Website, White Gold Corp).

The west boundary of the Canadian Creek block is in contact with the TEA claim block held by White Gold Corp. The TEA block also abuts the southern boundary of Newmont's Coffee Creek block. Part of the south boundary of the Canadian Creek block is in contact with the SOT block held by J. Milton. Part of the southern boundary of the eastern extension of the Casino property is in contact with the IDAHO claim block held by Atac Resources Ltd.

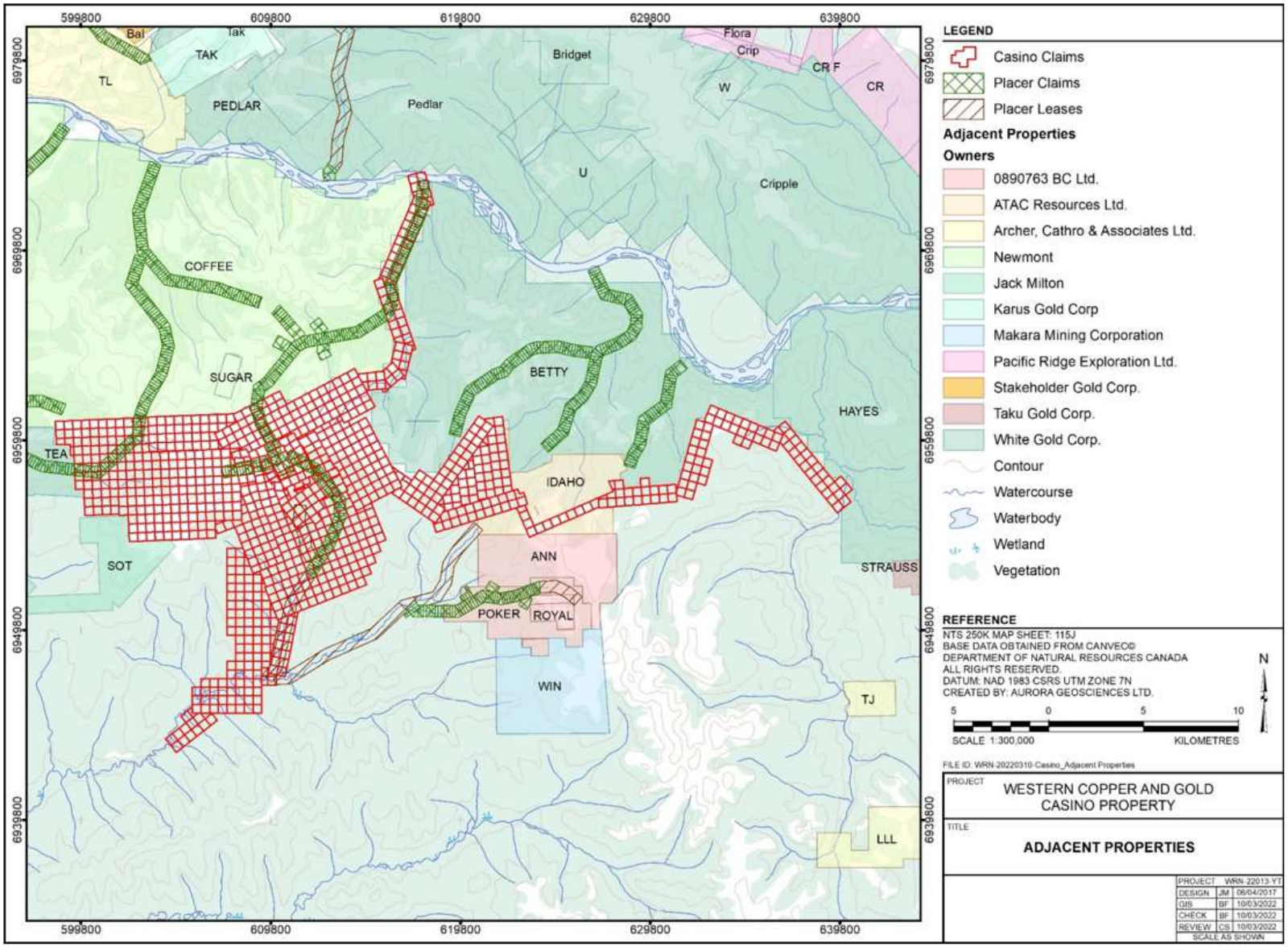


Figure 23-1: Adjacent properties, Casino Property Area (as of March 8, 2022)

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

The Project Execution Plan describes, at a high level, how the project will be carried out. This plan contains an overall description of what the main work focuses are, project organization, the estimated schedule, and where important aspects of the project will be carried out. The plan is to start operations with the Heap Leach Facility, to be followed by the concentrator. Key milestones include the following:

- Full Notice to Proceed and construction early works – first quarter 2026
- Heap Leach operation start-up – second quarter 2028
- Concentrator start-up – fourth quarter 2029
- Commercial Production – first quarter 2030

24.1.1 Focus

The proposed project execution plan incorporates an integrated strategy for engineering, procurement, and construction management (EPCM). The primary objective of the execution methodology is to deliver the project at the lowest capital cost, on schedule, and consistent with the project standards for quality, safety, and environmental compliance.

The majority of mechanical and electrical equipment required for the project will be procured within North America. Concrete, building construction materials and timber products will be sourced primarily in the Yukon. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be largely sourced within Canada, and to the extent practical, within the region. Some commodities, such as structural steel, may be sourced out of country.

24.1.2 Engineering

The project will enter the basic engineering phase in 2023 followed by detailed engineering in 2024. It is particularly important to identify priorities for long lead procurement and priorities for early construction that support the overall construction schedule. Engineering must be completed to the point that key procurement and construction activities have been decided contractually prior to the project's Notice to Proceed. Some funding may need to be committed to achieve this status.

24.1.3 Procurement

Equipment and bulk material suppliers will be selected via a competitive bidding process. Similarly, construction contractors will be selected through a pre-qualification process followed by a competitive bidding process. The project will employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contracts are awarded.

Procurement of long lead equipment and materials will be scheduled with their relevant engineering tasks. This will ensure that the applicable vendor information is incorporated into the design drawings and that the equipment will be delivered to site at the appropriate time, as well as support the overall project schedule. Particular emphasis will be placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

Construction documents need to be completed for bidding, bids offered and received, and contractors accepted and prepared to begin work before the "Full Notice to Proceed." Contractors will be selected, and a hold will be put on their

contracts awaiting the release of funds and the notice to proceed. January 2026 is the anticipated "Full Notice to Proceed."

24.1.4 Project Services

The EPCM contractor will be responsible for management and control of the various project activities and will ensure that the team has appropriate resources to accomplish WCGC's objectives. The EPCM integrated project services system from construction documents through procurement, cost control, project accounts, warehousing, and start-up.

24.1.5 Construction Management

The construction program is scheduled to start after the full notice to proceed, identified at this time as January 2026. The completion of one lane of the all-weather road by third quarter 2026 will be of the highest priority. It will be necessary to get the large earth-moving equipment onto the site to begin major work. This work will include the clearing and grubbing of the airfield and approaches, the access road to the airfield, the mine site including the main plant building pad, the heap leach pad, the north and south access roads within the mine site, and the Tailings Management Facility. The full access 2 lane all-weather road will be completed by third quarter 2027.

Erection of the construction camp will be divided into two phases. The first phase, scheduled to start in June 2026, will be the relocation of the existing camp from the new mill plant site and the construction of the new pioneer camp, which will include three Worker's dorms, one Supervisor's dorm, and a kitchen/diner/recreation unit for approximately 500 personnel. The first phase should be completed by September 2026. The second phase will begin the following construction season with further site preparation and construction of the foundations. The second phase will expand the construction camp by approximately 880 personnel for a total of approximately 1380 personnel. It will include seven additional Worker's dorms, one additional Supervisor's dorm, and two new Executive dorms. It will also include additional kitchen/dining facilities, and recreation facilities. The entire construction camp will be completed in September 2027.

Though the concrete plant is not expected to be able to be transported to the site until 2027, it is anticipated that sand and aggregate, as well as cement if appropriate storage can be assured, can be staged in 2026 by utilizing the limited access road from Carmacks as well as barges during the four months that barge traffic is available (June, July, August, and September). Additionally, it is anticipated that a low-volume batch or portable concrete mixer will be utilized in 2026 to facilitate the construction schedule.

Processing construction will begin with the Oxide Primary Crushing Building and the ADR/SART Facility in in the second quarter of 2027. Processing construction will finish with full commercial production of the concentrator in the first quarter of 2030. The objective is to have sufficient structures enclosed by the third quarter of 2027 so that mechanical and electrical work can continue during the colder months.

24.1.6 Contracting

Contracting is an integral function in the project's overall execution. Contracting for the Casino Project will be done in full accord with the provisions of the WCGC EPCM contract.

A combination of vertical, horizontal, and design construction contracts will be employed as best suits the work to be performed, and as best suits the degree of engineering and scope definition available at the time of award. A site-installed concrete batch plant will supply concrete to all construction contractors. All construction contractors will utilize the Owner-furnished construction camp. Camp operations will be provided by the EPCM through 2029 and will be taken over by WCGC-contracted service providers in 2030 when production starts. Early earthwork contractors will be expected to provide their own camps, as they will be on site prior to the erection of the site camp.

The estimate assumes that the mine fleet will perform bulk excavation of the concentrator pad and airstrip.

24.1.7 Labor

The labour market in northwestern Canada at this time is scarce. Many contractors in the Yukon are open shop. Construction labour will likely need to be imported from major population centres by air, including possibly Alaska.

24.1.8 Construction Completion and Turn-Over Procedure

The Construction Completion Procedure is part of the Construction Quality Plan. Contractors are to enter into contractual agreements with WCGC to perform certain portions of the work, which includes quality control of their work. Facilities will be verified and accepted in a stepwise documented process of mechanical completion and pre-operational testing. The main steps are as follows:

- Mechanical completion of components,
- Pre-commissioning of instrumentation,
- Pre-operational testing of overall systems,
- Start-up by Owner, and
- Full commercial production.

24.1.9 Quality Plan

A project-specific Quality Plan will be developed and implemented. The Quality Plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of a project. The plan, which consists of many different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction. The Quality Plan ensures compliance with various technical and accounting activities that will take place.

24.1.10 Health and Safety Plan

The Health and Safety Plan (HASP) will be established for the construction of the Casino Project and any other authorized work at the project site. The HASP covers both contractor personnel and operational personnel working at the project, and any on any other authorized work for the project.

The HASP specifies regulatory compliance requirements, training, certifications, and medical requirements necessary for Contractors to complete the project. Along with the Operations Procedures, the HASP is to be followed by all Contractor personnel working at the site.

24.1.11 Camp Transition

The camp is being built as a construction camp. However, it is expected that this camp will also become the permanent operations personnel camp. The camp usage will transition from construction to operations during the latter stages of construction prior to start-up.

24.1.12 Project Schedule

At the present time, the overview schedule is shown in Table 24-1.

Table 24-1: Overview Schedule

	2026				2027				2028				2029				2030			
	Year -4				Year -3				Year -2				Year -1				Year 1			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Full Notice to Proceed																				
One lane access road construction																				
Existing camp relocation																				
Mass excavations with mining fleet																				
Mining																				→
Two lane access road construction																				
Second phase construction camp																				
Heap leach circuit construction																				
Heap leach gold production																				→
Concentrator construction																				
Concentrate production																				→

25 INTERPRETATION AND CONCLUSIONS

25.1 MINING

This study has identified 1.22 billion tonnes of Mineral Reserve at 0.19% copper, 0.22 g/t gold, 0.021% moly and 1.7 g/t silver that is amenable to conventional milling and sulphide flotation. There is also 210 million tonnes of Mineral Reserve at 0.26 g/t gold, 1.9 g/t silver, and 0.036% copper that is amenable to conventional heap leaching. The project is based on conventional open pit mining and typical, well understood, processing methods.

The current Mineral Reserve is constrained by the current engineering design of the tailings management facility (TMF). There is potential to expand the TMF capacity by constructing an additional embankment south (downstream) of the current design. Significant work is required for the potential TMF expansion, but it would result in increased Mineral Reserve.

A mine plan was developed to supply mill ore to a conventional copper sulphide flotation plant with the capacity to process material at a nominal rate of 120,000 t/d or 43.8 Mt/y. Actual annual throughput will vary depending on the hardness of the ore encountered during the period. At this rate, the current Mineral Reserve indicates the commercial life of the project is 27 years following a three-year preproduction period.

25.2 PROCESS

Comprehensive metallurgical test work programs have been completed on Casino ores over the years. Flotation testing from 2008 to 2012 has confirmed that the Casino ores respond well to proven and widely used sulphide mineral processing techniques. Metallurgical results obtained in 2021 on oxide samples indicated that gold recovery from the heap leach could be maximized by crushing the ore going to the heap leach to minus 19 mm. Hydrodynamic characterization testing completed in 2022 on the oxide ore indicates that with three stage crushing percolation through the heap should not be an issue.

Both sulphide copper-molybdenum mill material and oxide gold leach material will be processed. Copper-molybdenum mill material will be transported from the mine to the concentrator facility. The sulphide facility design is based on processing 120,000 mtpd of material. Oxide gold leach material will be transported from the mine to a crushing facility ahead of a heap leaching and gold recovery facility. A review of heap leaching operations in cold climates indicates that year-round leaching at the Casino project site is feasible. Design provisions are considered in the design to mitigate cold weather affects. The oxide crushing, conveying, stacking facilities design is based on crushing and stacking oxide material 300 days per year at a rate of 30,400 mtpd. The carbon plant design is based on leaching 25,000 mtpd of oxide material.

Copper-molybdenum mill material will be processed by crushing, grinding using a single SABC circuit, and flotation to produce copper and molybdenum sulphide mineral concentrates. Copper concentrate will be loaded into highway haul trucks and transported to the Port of Skagway for ocean shipment to market. Molybdenum concentrate will be bagged and loaded onto highway haul trucks for shipment to market.

Oxide gold mill material will be processed using a three-stage crushing circuit and conveyed to a heap leach pad where it will be leached with a cyanide leach solution. Gold in the enriched (or pregnant) leach solution will be recovered using carbon absorption technology to produce gold doré bars. A portion of the enriched leach solution will also be treated to recover copper and cyanide and produce a copper sulphide precipitate. The copper sulphide precipitate will be bagged and loaded onto highway haul trucks for shipment to market.

The typical risks associated with open pit mining related to dilution, geotechnical and hydrogeological conditions, equipment availability and productivity, and personnel productivity are generally like those expected at other operations.

The ability to achieve the estimated OPEX costs are important elements of project success. Investigation of cost reduction measures will mitigate over runs.

Crushing, leaching, and ADR plant performance have all been designed using engineering analyses that are based on small but representative samples that may be different than actuality. Ongoing consideration of potential variances in engineering assumptions during operations will mitigate risks.

Lower than expected metallurgical recovery from either the sulphide ore or oxide ore is a potential risk. Additional sampling, test work, and operational experience will mitigate risks.

25.3 TAILINGS MANAGEMENT FACILITY

The highlighted risks identified during the feasibility study include:

- Geotechnical Foundation Conditions – There is a risk that the extent or depth of permafrost conditions or other unsuitable materials in the foundations may be greater than currently anticipated, requiring more removal of material prior to construction of the TMF embankments. This risk can be mitigated through additional geotechnical investigations.

- Geotechnical and Hydrogeological Performance of the Cyclone Sand – There is a risk that cyclone sand within the main embankment does not meet the geotechnical design criteria for drainage and strength at the specified range of fines content. Industry experience and published literature support that the current assumptions in this study are reasonable; however, the site-specific sampling and testing is limited. This risk will be mitigated through advanced laboratory testing of additional representative tailings samples to determine the material parameters under high stresses and assess the influence of variability in the particle size distribution.

- Main Embankment Raises – There is a risk that the main embankment cannot be raised quickly enough using the assumed construction methods, especially in the early years of operation when the required rate of rise is high. This risk will be mitigated by more detailed staging of the raises and contingency plans for raising the dam using screened sand stockpiled in the winter months.

The key potential opportunities include:

- Potential to use mine waste rock and material excavated from the TMF embankment foundation in the starter dam construction to reduce the reliance on a dedicated rock quarry for initial construction; this may reduce initial capital costs.

- Potential to optimize the main embankment geometry if further geotechnical investigation, site characterization, and cyclone sand testing identify more favorable conditions than currently assumed.

- Potential to relax the maximum allowable fines content in the cyclone sand underflow to increase operation flexibility and increase sand production.

- Potential to optimize and reduce the volume of the initial foundation excavation for the Starter Dam through additional site investigation and improved site characterization.

25.4 EXPLORATION INTERPRETATIONS

From 2010 through 2021 a total of 173 diamond drill holes for 39,372.91 m were completed at the Casino and Canadian Creek properties. The 2019 and 2020 programs were the most extensive, designed to upgrade the resource categories of portions of the deposit, and, in 2020, to test additional interpreted mineralized zones, particularly the Gold, North Porphyry and Canadian (now called the Casino West) zones. In 2021, the program focused to resource confirmation, metallurgical testing, and geotechnical holes in the eastern portion of the deposit area, including low-lying areas directly east of Patton Hill. The 2021 program also included six exploration holes east and south of the Casino deposit.

The "Gold Zone" was delineated along the southern and western margins of the deposit, covering an area of widely spaced narrow high-grade intercepts returned from 1992 – 1994 drilling by Pacific Sentinel, and from drilling by Western from 2008 – 2012 and 2019. One such occurrence, returning 55.1 g/t gold across 2.97 m including several specks of visible gold, was returned from DDH 19-21 within the theoretical Gold Zone. However, drilling in 2020 did not intersect any significant high-grade zones, and the Gold Zone was determined not to be a discrete feature within the deposit.

Drilling in 2020 at the potential North Porphyry zone essentially disproved the presence of a second porphyry centre. Assay values for Cu, Mo, Au, and Ag progressively diminish northward from the deposit core, and mineralization becomes increasingly vein-dominated and intermittent to the north. Drilling at the Canadian zone returned low to background values in its southern extent, with metal values increasing somewhat northwards, towards Canadian Creek. However, grades remained comparatively low, and the 2020 drilling did not result in significant expansion of the Casino deposit.

Detailed review of results to 2020 revealed the presence of a "Deposit Core" area of increased Cu-Au grades occurring within intrusive breccias in the east-central property area. The Deposit Core extends roughly NNW, centered on holes DDH20-05 and DDH20-08, from the crest of Patton Hill. This area was interpreted in early 2021 as a potential target for initial ore extraction.

The 2021 drilling included several holes east of, and topographically considerably lower than, the Deposit Core area on Patton Hill. Hole DDH21-09, a geotechnical hole, returned a high-grade interval at shallow depths. The pathfinder signature shows weakly to moderately elevated As, Bi, Sb and Zn values, including a narrow interval of strongly elevated Au-Ag-As-Bi-Sb values. This assemblage is indicative of fault-hosted hydrothermal mineralization. This hole, as well as DDH21-07, is located along the interpreted trace of the Casino Fault, which may have acted as a conduit for late-phase hydrothermal mineralization. The fault may have caused displacement of stratigraphy along its northeast side.

Soil sampling followed by XRF analysis in the field identified three anomalies (Anomalies A through C) east and south of the main deposit. All three underwent exploration diamond drilling and returned low to background Cu-Mo-Au-Ag values. Increased resolution during follow-up soil sampling may result in delineation of more specific targets. Of the other three exploration holes, DDH21-14, the closest hole east of the deposit, returned a 54.35-metre intercept of anomalous Au-Ag mineralization, with weakly elevated Cu and background Mo values. Highly anomalous As and Sb, and moderately anomalous Bi values, indicate the mineralization is likely fault-controlled and of late-stage hydrothermal origin.

Analysis of lab assay results correlate fairly well with those from XRF analysis in the field, indicating that XRF analysis may be used as a guide to immediate follow-up exploration. The soil survey was incomplete when Anomalies A through C were drilled, with outlying areas still to be surveyed. Lab analyses revealed three further anomalous areas: Anomaly D, in the extreme northeast corner; Anomaly E, along the northwest boundary, and Anomaly F, southeast of the main

deposit. Of these, only Anomaly F has a geochemical signature indicative of porphyry-style mineralization, comprising anomalous Au-Ag-Cu-Mo values but lacking anomalous As values. This target has not been "ground-truthed" and warrants further detailed exploration.

25.5 EXPLORATION CONCLUSIONS

The following conclusions can be made from the results of 2019 through 2021 diamond drilling, and the 2021 soil sampling program:

- The 2019, 2020 and 2021 programs comprised 39,372.91 m of diamond drilling completed on the Casino and Canadian Creek properties. These drilling programs effectively delineated the extent of the Casino deposit. The 2021 drilling results are not incorporated into this feasibility study.

- The 2020 program focused on evaluating the potential presence of the Gold, North Porphyry and Canadian (Casino West) zones. Results indicate the Canadian zone does not have significant mineral potential, and that there are no actual discrete Gold and North Porphyry zones.

- Results of drilling inclusive of 2020 indicate the presence of a "Deposit Core" of higher-grade material in the east-central deposit area, both within the leached cap and underlying sulphide mineralized zones. This NNW-trending zone may represent a target for initial mineral extraction.

- The 2021 drilling included several holes east of, and topographically considerably lower than, Patton Hill. Of these, hole DDH21-09 returned a high-grade CuEq interval that coincides with the surface trace of the Casino fault, indicating the fault trace may represent a target for higher-grade mineralization within the deposit.

- Individual samples of the 2021 soil sampling program were analyzed by a portable XRF unit, results of which were determined to be sufficiently accurate to identify targets for follow-up exploration while still in the field. Three anomalies were identified and underwent drilling, although no significant results were returned.

- Lab analytical results for the soil sampling roughly support those from XRF analysis for base and pathfinder elements. Lab results revealed three further anomalies, D through F, although only Anomaly F has a geochemical signature indicative of porphyry-style Cu-Mo-Ag-Au mineralization.

- Exploration hole DDH21-14 returned a 54.35 m intercept of anomalous Au-Ag values, likely representing fault-controlled hydrothermally derived mineralization. The geochemical signature is indicative of outlying "Bonanza-style" veining, although mineralization is of lower grade than typical Bonanza-style zones.

26 RECOMMENDATIONS

Based on the results of this study, it is recommended that the project advance into the execution planning phase and an application for environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act be prepared to continue the permitting process.

26.1 METALLURGY AND MINERAL PROCESSING

For the next stage of the project, detailed mill specifications will be required and final design aspects, such as SAG discharge grate aperture size and type/number of shell lifters should be considered. Also sizing details regarding the trommel screen and SAG discharge screen will be finalized. Additionally, simulations relating to the pulp lifter depth will be required.

Makeup water requirements to the heap leach and carbon column sizing are recommended to be evaluated with consideration given to evaporation, precipitation, ROM moisture, and saturation moisture. It is recommended to perform additional column testing with finer crushed ore to validate metal recovery. It is recommended to perform analysis of pregnant solution from leaching finer crushed ore to measure the metal concentrations to validate sizing of carbon handling plant. Cost included with design consideration in section 26.2

It is recommended to perform SART testing on leach solution to optimize size of SART plant equipment and confirm the consumption of reagents for the next phase of engineering. It is also recommended to define the characteristics of the copper precipitate produced and market requirements.

The estimated cost for the above tests is $100,000.

26.2 TAILINGS MANAGEMENT FACILITY

The following is a list of recommendations for additional site investigations, test work and design studies for the TMF:

- Additional drilling and sampling of the overburden materials is recommended in the area of the TMF to better define the composition and expected variability in material depths, particularly within the footprint of the TMF starter dam.
- Preparation of a comprehensive site characterization report.
- Updated laboratory testing to confirm the physical characteristics and assumed performance of the tailings cyclones. The current assumptions are based on historical test work as new tailings were not available prior to initiation of this study.
- Additional laboratory testing to confirm the physical characteristics of the cyclone sand (material classification, strength, and permeability tests) is recommended. This should include strength and permeability tests at very high confining stresses, representative of the height of the Main Embankment, and testing to examine the influence of cyclone sand fines content on available percentage of sand recovery and the impact on permeability.
- Investigate the feasibility of using waste rock to construct the TMF starter dam.
- Laboratory testing to determine the physical characteristics of the PAG tailings (material classification, slurry settling, consolidation and permeability tests).
- Development of a detailed tailings and waste rock deposition strategy to optimize material handling, and tailings discharge line and reclaim barge locations throughout operations.
- Assessment of the sequencing of sand cell construction in relation to embankment raising requirements to optimise material demand and placement strategies.
- Carry out stress-deformation modelling for the main embankment once a suitable level of characterization has been completed on the in-situ foundation and construction materials.

- Update the hydrometeorology inputs for the TMF design as further studies are completed.
- Update the seismic hazard analysis.
- Basic engineering design of all civil earthworks and mechanical systems, including sumps, intakes, outlets, pumps, pipe systems etc.

The estimated cost for the above tailings management facility recommendations is $3.5 M.

26.3 HEAP LEACH FACILITY

The following is a list of geotechnical recommendations for additional site investigations, test work, and design studies for the HLF:

- Geotechnical Investigations and Test Work:
    - Additional test pits/drill holes to confirm assumptions and characterization of foundation and permafrost conditions.
    - Additional test pits/drill holes to assess suitability, availability, and quantity of borrow materials for earthworks construction.
    - Geotechnical laboratory testing of potential borrow materials for pad foundation (low permeability soil layer) and confining embankment or events pond dam construction (including particle size distribution, Atterberg limits, specific gravity, moisture-density relationship, permeability, and shear strength tests).
    - Laboratory direct shear testing of liner interfaces, to determine the interface shear strength relationships for heap stability assessment. It is recommended that the shear strength tests are carried out for each of the liner interfaces within the composite liner systems once the potential material sources have been confirmed.
    - Mineralized Material testing (including particle size distribution, specific gravity, permeability under load, load-percolation, and direct shear/triaxial shear strength tests).
- Design Studies and Analyses:
    - Leach testing to determine optimal mineralized material densities, leaching rates and resulting in-heap moisture contents.
    - Detailed water balance analyses, based on results of hydrology and leach testing, to estimate solution storage area water volumes, peak storm flows for ditch design, and to confirm sizing of pump and pipework systems.
    - Heap stability assessment based on results of laboratory shear strength and liner interface strength testing.
    - Seepage analyses to predict seepage flow patterns and solution losses for the design of the LDRS.
    - Basic engineering design of all civil and mechanical works, including sumps, intakes, outlets, pumps, pipe systems etc.
    - Advanced studies on the staging and sequencing of mineralized material placement and development of the HLF.

The estimated cost for the above heap leach facility recommendations is $800,000.

26.4 ADDITIONAL FACILITIES

The following is a list of recommendations for additional site investigations, test work, and design studies required to carry the additional facilities to final design and construction:

- Additional drilling and sampling of the overburden materials is recommended in the area of the Processing facility to better define the composition and expected variability in material depths.
- Additional drilling and sampling of the overburden materials is recommended in the area of the proposed new airstrip to better define the composition and expected variability in material depths.

26.5 EXPLORATION

The remaining exploration holes proposed for 2021 are recommended to undergo drill testing, to the depths proposed. Additional holes targeting the surface strike projection of the gold-silver enriched fault-controlled interval in DDH21-14 are also recommended.

Based on results from DDH21-09, further drilling along the trace of the Casino Fault, particularly to the southeast, is also recommended, to a minimum depth of 300 m. Holes are recommended to be oriented at azimuths of 060°, and dips of -60°. At least one exploration hole is recommended to be collared about 100 m east of DDH21-09, to test for porphyry-style mineralization potentially offset by the Casino fault.

Detailed B-horizon soil sampling at a 100-metre line spacing and 50-metre station spacing are recommended for soil anomalies D, E and F. At Anomaly D, lines are recommended to be oriented at 135° – 315°; at Anomaly E, they should be oriented north-south; and at Anomaly F the orientation should be east-west. If funding permits, detailed B-horizon soil sampling at the same line and station spacing is also recommended for anomalies A, B and C. Survey lines at Anomaly A and B are recommended to be oriented east-west, whereas lines at Anomaly C are recommended to be oriented north-south.

The estimated cost for the above exploration recommendations is CAD$854,000.

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APPENDIX A

Feasibility Study Contributors and Professional Qualifications

Certificate of Qualified Person ("QP")

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Roth, P.E., P. Eng. do hereby certify that:

1. I am currently employed as a project manager and civil engineer at M3 Engineering & Technology Corp. located at 2051 West Sunset Rd, Suite 101, Tucson, AZ 85704.
2. This certificate applies to the technical report entitled, "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I graduated with a Bachelor of Science degree in Civil Engineering from The University of Manitoba in 1990.
4. I am a registered professional engineer in good standing in the following jurisdictions:
    - Yukon, Canada (No. 1998)
    - British Columbia, Canada (No. 38037)
    - Alberta, Canada (No. 62310)
    - Ontario, Canada (No. 100156213)
    - New Mexico, USA (No. 17342)
    - Arizona, USA (No. 37319)
    - Alaska, USA (No. 102317)
    - Minnesota, USA (No. 54138)
    - Nevada, USA (No. 029423)
5. I have worked continuously as a design engineer, engineering, and project manager since 1990, a period of 30 years. I have worked in the minerals industry as a project manager for M3 Engineering & Technology Corporation since 2003, with extensive experience in hard rock mine process plant and infrastructure design and construction, environmental permitting review, as well as development of capital cost estimates, operating cost estimates, financial analyses, preliminary economic assessments, pre-feasibility, and feasibility studies.
6. 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.
7. I am responsible for Sections 1,1.1-1.4, 1.14.1, 1.15, 1.16, 1.17, 1.19, 2, 3, 4, 5, 18, 18.1-18.4, 18.9, 18.10, 19, 21 (except 21.1.5, 21.3.1 and 21.3.3), 22, 24, 26, 26.4 and corresponding section 27 of the Technical Report.
8. I have prior involvement with the property that is the subject of the Technical Report. I have developed various capital and operating cost tradeoff studies for Western from 2014 through 2020. I was a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Mineral Resource Statement, Yukon, Canada" dated effective July 3, 2020 and the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021. I visited the property that is the subject of the Technical Report on August 6, 2021 for one day.
9. As of the date of this certificate, 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.
10. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 8 day of August, 2022.

"Signed"
Signature of Qualified Person

Daniel Roth
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Michael G. Hester, do hereby certify that:

1. I am currently employed as Vice President and Principal Mining Engineer by Independent Mining Consultants, Inc. ("IMC") of 3560 E. Gas Road, Tucson, Arizona, 84714, USA.
2. This certificate applies to the technical report entitled, "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I graduated with a Bachelor of Science degree in Mining Engineering from the University of Arizona in 1979 and a Master of Science degree in Mining Engineering from the University of Arizona in 1982.
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101").
5. I have worked in the minerals industry as an engineer continuously since 1979, a period of 43 years. I am a founding partner, Vice President, and Principal Mining Engineer for IMC, a position I have held since 1983. I have been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I have also been a member of the Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration since March 2012. During my career I have had extensive experience developing mineral resource models, developing open pit mine plans, estimating equipment requirements for open pit mining operations, developing mine capital and operating cost estimates, performing economic analysis of mining operations and managing various preliminary economic assessments, pre-feasibility, and feasibility studies. I have extensive experience with large open pit base metal mines and with large copper-gold porphyry geologic systems similar to the property that is the subject of the Technical Report.
6. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am responsible for Section 1.9, 14, and corresponding parts of section 27 of the Technical Report.
8. I have prior involvement with the property that is the subject of the Technical Report. I was a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Mineral Resource Statement, Yukon, Canada" dated effective July 3, 2020 and the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021. I was also a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" dated effective January 23, 2013 and the technical reports of Western Copper Corporation entitled "Casino Project, NI 43-101 Technical Report, Pre-Feasibility Study Update, Yukon Territory, Canada" dated April 29, 2011 and "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada" dated August 5, 2008. I also worked on studies of the property for Pacific Sentinel Corporation in or around September 1995. I most recently inspected the property that is the subject of the Technical Report on September 7, 2021, for a period of one day.
9. As of the date of this certificate, 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.
10. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 08 day of August 2022.

"Signed"
Signature of Qualified Person

Michael G. Hester, FAusIMM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

a) I am currently employed as the President and a Senior Mining Engineer by:

Independent Mining Consultants, Inc.
3560 E. Gas Road
Tucson, Arizona, USA 85714

b) This certificate applies to the technical report entitled "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of 13 June 2022, (the "Technical Report").

c) I graduated with the following degrees from the Colorado School of Mines:

- Bachelors of Science, Mineral Engineering – Physics – 1974
- Masters of Science, Mining Engineering – 1976

d) I am a member of the following professional associations as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"):

- I am a Registered Professional Mining Engineer in the State of Arizona, USA – Registration # 12772
- I am a Registered Professional Engineer in the State of Colorado, USA – Registration # 16191
- I am a Professional Engineer, Yukon Territory, Canada.
- I am a Professional Engineer, Ontario Province, Canada.
- I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 46 years. My work experience includes mine planning, equipment selection, mine cost estimation and mine feasibility studies for base and precious metals projects worldwide for over 46 years. I have extensive experience with large open pit base metal mines and with large copper-gold porphyry geologic systems similar to the property that is the subject of the Technical Report.

e) I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

f) I visited the property that is the subject of the Technical Report on September 7, 2021 for one day.

g) I am responsible for sections 1.10, 1.11, 15, 16, 21.1.5, 21.3.3, 25.1 and the corresponding components of section 27 of the Technical Report.

h) I am independent of Western and its subsidiaries as defined by Section 1.5 of National Instrument 43-101.

i) I have prior involvement with the property that is the subject of the Technical Report. I was a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021.

CERTIFICATE OF QUALIFIED PERSON

j) I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

k) 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.

Dated: August 08, 2022

Signed and Sealed

John M. Marek
Professional Engineer
Yukon Territory

CERTIFICATE OF QUALIFIED PERSON

I, Laurie M. Tahija, MMSA-Q.P. do hereby certify that:

1. I am currently employed as Senior Vice President by M3 Engineering & Technology Corporation, 2051 W. Sunset Road, Ste. 101, Tucson, Arizona 85704, USA.
2. This certificate applies to the technical report entitled, "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I am a graduate of Montana College of Mineral Science and Technology, in Butte, Montana and received a Bachelor of Science degree in Mineral Processing Engineering in 1981. I have practiced mineral processing for 40 years. I have over twenty (20) years of plant operations and project management experience at a variety of mines including both precious metals and base metals. I have worked both in the United States (Nevada, Idaho, California) and overseas (Papua New Guinea, China, Chile, Mexico) at existing operations and at new operations during construction and startup. My operating experience in base metal processing includes copper heap leaching with SX/EW and zinc recovery using ion exchange, SX/EW, and casting. My operating experience in precious metals processing includes heap leaching, agitation leaching, gravity, flotation, Merrill-Crowe, and ADR (CIC & CIL). I have been responsible for process design for new plants and the retrofitting of existing operations. I have been involved in projects from construction to startup and continuing into operation. I have worked on scoping, pre-feasibility and feasibility studies for mining projects in the United States and Latin America, as well as worked on the design and construction phases of some of these projects.
4. I am recognized as a Qualified Professional (QP) member (#01399QP) with special expertise in Metallurgy/Processing by the Mining and Metallurgical Society of America (MMSA).
5. I have not visited the property that is the subject of the Technical Report.
6. I am responsible for Sections 1.12, 1.13, 13, 17, 21.3.1, 25.2, 26.1 and corresponding section 27 of the Technical Report.
7. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
8. I have prior involvement with the property that is the subject of the Technical Report. I was a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Mineral Resource Statement, Yukon, Canada" dated effective July 3, 2020, and the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021.
9. 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.
10. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
11. As of the date of this certificate, 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.

Dated this 08 day of August 2022.

"Signed"
Signature of Qualified Person

Laurie M. Tahija
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Carl Schulze, P. Geo. with a business address at 34A Laberge Rd, Whitehorse, Yukon Y1A 5Y9, hereby certify that:

1. I am a Project Manager employed by: Aurora Geosciences Ltd., 34A Laberge Rd, Whitehorse, Yukon Y1A 5Y9.
2. This certificate applies to the technical report titled: "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I am a graduate of Lakehead University, Bachelor of Science Degree in Geology, 1984. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (EGBC), Lic. No. 25393. I have worked as a geologist for a total of 38 years since my graduation from Lakehead University. I have worked extensively in Yukon, British Columbia, northern Ontario, and Alaska, as well as the Northwest Territories, Saskatchewan, and Manitoba. Specifically, I have worked on copper-molybdenum-silver-gold porphyry-style deposits and prospects in Yukon and British Columbia. I served as President of the Yukon Chamber of Mines, where I was also a Director from 2003 to 2015. I have acted in various capacities with numerous private and publicly-traded mining and exploration companies, and also served as the Resident Geologist for the Government of Nunavut from 2000 to 2002.
4. 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.
5. I performed a site visit of the property that is subject of the Technical Report for 18 days from September 9, 2020 through September 26, 2020. I was also on site for 13 days from May 21 to June 3, and for 22 days from June 10 to July 1, 2021.
6. I am responsible for Sections 1.4-1.8, 1.18, 1.19, 6, 7, 8, 9, 10, 11, 12, 23, 25.4, 25.5, 26.5 and 27 of the Technical Report.
7. I have prior involvement with the property that is subject of the Technical Report. I was a qualified person for the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Mineral Resource Statement, Yukon, Canada" dated effective July 3, 2020 and the technical report of Western entitled "Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021.
8. I am independent of Western and its subsidiaries applying the test in section 1.5 of NI 43-101.
9. I have read NI 43-101and Form 43-101F1, and the sections of the Technical Report that I am responsible for has been prepared in compliance with that instrument and form;
10. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading, and;

Dated at Whitehorse, Yukon this 08 day of August 2022.

"Signed" Carl Schulze

Signature of Qualified Person

Carl Schulze

Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Friedman, P. Eng. do hereby certify that:

1. This certificate applies to the technical report entitled, “Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada” prepared for Western Copper and Gold Corporation (the “Issuer”) with an effective date of June 13, 2022 (the “Technical Report”)
2. I am employed as a Specialist Civil Engineer of Knight Piésold Ltd. with an office at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of McGill University, Montreal, Canada, B.Eng. (Civil), 2003. I have practiced my profession continuously since 2004. My principal experience is in the areas of water and waste management for mining projects, including open pit copper-gold projects, and hydrotechnical engineering.
4. I am a registered professional engineer in good standing in the following jurisdictions:
    - Yukon, Canada (No. 3404)
    - British Columbia, Canada (No. 32571)
    - New Brunswick, Canada (No. L5001)
    - Arizona, USA (No. 53722)
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 have not visited the property that is the subject of the Technical Report.
7. I am responsible for Sections 1.14.6, 1.14.7, 18.7, 18.8, 25.3, 26.2, 26.3, and corresponding section 27 of the Technical Report and accept professional responsibility for these sections of the Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have prior involvement with the property that is the subject of the Technical Report. I have developed various engineering studies related to tailings and water management for Western Copper and Gold Corporation (“Western”) from 2016 through 2021. I was a qualified person for the technical report of Western entitled “Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada” dated effective June 22, 2021.
10. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report that I am responsible for have been prepared in accordance with that instrument and form.

Knight Piésold Ltd. | Suite 1400 - 750 West Pender Street | Vancouver, British Columbia | Canada, V6C 2T8
T +1 604 685 0543 | E vancouver@knightpiesold.com | www.knightpiesold.com

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 13, 2022
Signing Date: August 8, 2022

Signed

Daniel Friedman, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Patrick W. Dugan, P.E. do hereby certify that:

1. I am currently employed as Piping Quality Control Manager at M3 Engineering & Technology Corp. located at 2051 West Sunset Rd, Suite 101, Tucson, AZ 85704.
2. This certificate applies to the technical report entitled, "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I graduated with a Bachelor of Science degree in Mechanical Engineering from The University of Arizona in 1972. I completed 24 graduate units in Physical Metallurgy from the University of Arizona 1972 -1974. Master's program not completed.
4. I am a registered professional engineer in good standing in the following jurisdictions:
    - Arizona, USA (No. 15308)
5. I have worked continuously as a design engineer, construction engineer, engineering manager and project manager since 1974, a period of 48 years. I have worked in the minerals industry since 1979, a period of 43 years. I have been involved in design, startup and construction of uranium, precious metal, and base metal projects. I have worked on projects in the US (Arizona, New Mexico, Utah, Nevada, Colorado, Idaho, Wyoming, and South Carolina) and Overseas (Mexico, Chile, Argentina, Costa Rica, Philippines, Papua New Guinea, Saudi Arabia, and Serbia). I have designed tailings disposal pipelines and pump station systems for plants from 10,000 TPD to 150,000 TPD. My experience in power includes construction management for a 350 MW coal fired power plant and management of piping design and construction assistance for several simple cycle and combined cycle natural gas fire power stations including a 350 MW combined cycle plant in California. I have also been involved in numerous retrofit projects for waster heat boiler power stations at several smelters in the US (Arizona and New Mexico.). I have been involved in numerous scoping, pre-feasibility and feasibility studies for mineral projects in the US, Mexico, South America and the middle east over the last 40 years.
6. I have been a member of Society for Mining, Metallurgy and Exploration (SME) since 1983.
7. 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.
8. I am responsible for Sections 1.14.2-1.14.5, 18.5, 18.6 and corresponding section 27 of the Technical Report.
9. I have prior involvement with the property that is the subject of the Technical Report. I have assisted in development of various water supply and tailing disposal designs and was involved in the initial specifications and vendor selection for the combined cycle power plant to be constructed at the site for Western from 2012 through 2020. I have not visited the property that is the subject of the Technical Report.
10. As of the date of this certificate, 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.
11. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 08 day of August 2022.

"Signed"
Signature of Qualified Person

Patrick W Dugan P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Scott Weston, P.Geo., do hereby certify that:

1. I am employed as Vice President, Business Development and Strategy with Hemmera Envirochem Inc, a wholly owned subsidiary of Ausenco Canada ("Ausenco"), with an office address of 4515 Central Boulevard, Burnaby, BC, Canada.
2. This certificate applies to the technical report entitled, "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" prepared for Western Copper and Gold Corporation ("Western") with an effective date of June 13, 2022 (the "Technical Report").
3. I graduated from University of British Columbia, Vancouver, BC, Canada, 1995 with a Bachelor of Science, Physical Geography, and Royal Roads University, Victoria, BC, Canada, 2003 with a Master of Science, Environment and Management.
4. I am a Professional Geoscientist of Engineers and Geoscientists British Columbia; 124888.
5. I have practiced my profession for 25 years as an environmental consultant in Canada and the Americas. I have a long history of working in Yukon. This provides me a strong understanding of Yukon biophysical and social situations, analysis and interpretation of baseline data, and Yukon regulatory processes.
6. 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.
7. I am responsible for Section 20 and corresponding section 27 of the Technical Report.
8. I have no prior involvement with the property that is the subject of the Technical Report. I have not visited the property that is the subject of the Technical Report.
9. As of the date of this certificate, 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.
10. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 08 day of August 2022.

"Signed"
Scott Weston, P. Geo.

APPENDIX B

List of Claims

Casino Property

List of Casino Placer Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	GRANT NUMBER	TENURE TYPE	CLAIM NAME	CLAIM NUMBER	RECORDED DATE	EXPIRY DATE
1	P 508065	Placer	CAS PL	4	2011-08-11	2022-02-11
2	P 508066	Placer	CAS PL	5	2011-08-11	2022-02-11
3	P 508067	Placer	CAS PL	6	2011-08-11	2022-02-11
4	P 508068	Placer	CAS PL	7	2011-08-11	2022-02-11
5	P 508069	Placer	CAS PL	8	2011-08-11	2022-02-11
6	P 508070	Placer	CAS PL	9	2011-08-11	2022-02-11
7	P 508071	Placer	CAS PL	10	2011-08-11	2022-02-11
8	P 508072	Placer	CAS PL	11	2011-08-11	2022-02-11
9	P 508073	Placer	CAS PL	12	2011-08-11	2022-02-11
10	P 508074	Placer	CAS PL	13	2011-08-11	2022-02-11
11	P 508075	Placer	CAS PL	14	2011-08-11	2022-02-11
12	P 508076	Placer	CAS PL	15	2011-08-11	2022-02-11
13	P 508077	Placer	CAS PL	16	2011-08-11	2022-02-11
14	P 508078	Placer	CAS PL	17	2011-08-11	2022-02-11
15	P 508079	Placer	CAS PL	18	2011-08-11	2022-02-11
16	P 508080	Placer	CAS PL	19	2011-08-11	2022-02-11
17	P 508081	Placer	CAS PL	20	2011-08-11	2022-02-11
18	P 508082	Placer	CAS PL	21	2011-08-11	2022-02-11
19	P 508083	Placer	CAS PL	22	2011-08-11	2022-02-11
20	P 508084	Placer	CAS PL	23	2011-08-11	2022-02-11
21	P 508085	Placer	CAS PL	24	2011-08-11	2022-02-11
22	P 508086	Placer	CAS PL	25	2011-08-11	2022-02-11
23	P 508087	Placer	CAS PL	26	2011-08-11	2022-02-11
24	P 508088	Placer	CAS PL	27	2011-08-11	2022-02-11
25	P 508089	Placer	CAS PL	28	2011-08-11	2022-02-11
26	P 508090	Placer	CAS PL	29	2011-08-11	2022-02-11
27	P 508091	Placer	CAS PL	30	2011-08-11	2022-02-11
28	P 508092	Placer	CAS PL	31	2011-08-11	2022-02-11
29	P 508093	Placer	CAS PL	32	2011-08-11	2022-02-11
30	P 508094	Placer	CAS PL	33	2011-08-11	2022-02-11
31	P 508095	Placer	CAS PL	34	2011-08-11	2022-02-11
32	P 508096	Placer	CAS PL	35	2011-08-11	2022-02-11
33	P 508097	Placer	CAS PL	36	2011-08-11	2022-02-11
34	P 508098	Placer	CAS PL	37	2011-08-11	2022-02-11
35	P 508099	Placer	CAS PL	38	2011-08-11	2022-02-11
36	P 508100	Placer	CAS PL	39	2011-08-11	2022-02-11
37	P 509301	Placer	CAS PL	40	2011-08-11	2022-02-11
38	P 509302	Placer	CAS PL	41	2011-08-11	2022-02-11
39	P 509303	Placer	CAS PL	42	2011-08-11	2022-02-11
40	P 509304	Placer	CAS PL	43	2011-08-11	2022-02-11
41	P 509305	Placer	CAS PL	44	2011-08-11	2022-02-11
42	P 509306	Placer	CAS PL	45	2011-08-11	2022-02-11
43	P 509307	Placer	CAS PL	46	2011-08-11	2022-02-11
44	P 509308	Placer	CAS PL	47	2011-08-11	2022-02-11
45	P 509309	Placer	CAS PL	48	2011-08-11	2022-02-11
46	P 509310	Placer	CAS PL	49	2011-08-11	2022-02-11
47	P 509311	Placer	CAS PL	50	2011-08-11	2022-02-11
48	P 509312	Placer	CAS PL	51	2011-08-11	2022-02-11
49	P 509313	Placer	CAS PL	52	2011-08-11	2022-02-11
50	P 509314	Placer	CAS PL	53	2011-08-11	2022-02-11
51	P 509315	Placer	CAS PL	54	2011-08-11	2022-02-11
52	P 509316	Placer	CAS PL	55	2011-08-11	2022-02-11
53	P 509317	Placer	CAS PL	56	2011-08-11	2022-02-11
54	P 509318	Placer	CAS PL	57	2011-08-11	2022-02-11
55	P 509319	Placer	CAS PL	58	2011-08-11	2022-02-11

Casino Property

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1	95740	CAT	63	1965-12-05	2025-03-25	115J10	
2	95741	CAT	64	1965-12-05	2025-03-25	115J10	
3	95742	CAT	65	1965-12-05	2025-03-25	115J10	
4	95743	CAT	66	1965-12-05	2025-03-25	115J10	
5	95745	CAT	68	1965-12-05	2025-03-25	115J10	
6	95747	CAT	70	1965-12-05	2025-03-25	115J10	
7	Y 35195	MOUSE	4	1969-06-04	2025-03-25	115J10	
8	Y 35197	MOUSE	6	1969-06-04	2025-03-25	115J10	
9	Y 35484	MOUSE	90	1969-06-22	2025-03-25	115J10	
10	YD04376	FLY	2	2011-02-02	2025-03-25	115J10	
11	YD04377	FLY	3	2011-02-02	2025-03-25	115J10	
12	YD04378	FLY	4	2011-02-02	2025-03-25	115J10	
13	YD04379	FLY	5	2011-02-02	2025-03-25	115J10	
14	YD04380	FLY	6	2011-02-02	2025-03-25	115J10	
15	YD04381	FLY	7	2011-02-02	2025-03-25	115J10	
16	YD04382	FLY	8	2011-02-02	2025-03-25	115J10	
17	YD04383	FLY	9	2011-02-02	2025-03-25	115J10	
18	YD04384	FLY	10	2011-02-02	2025-03-25	115J10	
19	YD04385	FLY	11	2011-02-02	2025-03-25	115J10	
20	YD04386	FLY	12	2011-02-02	2025-03-25	115J10	
21	YD04387	FLY	13	2011-02-02	2025-03-25	115J10	
22	YD04388	FLY	14	2011-02-02	2025-03-25	115J10	
23	YD04399	FLY	15	2011-02-02	2025-03-25	115J10	
24	YD04400	FLY	16	2011-02-02	2025-03-25	115J10	
25	YD04401	FLY	17	2011-02-02	2025-03-25	115J10	
26	YD04402	FLY	18	2011-02-02	2025-03-25	115J10	
27	YD04375	FLY	1	2011-02-02	2025-03-26	115J10	
28	YC82855	BL	1	2008-07-31	2025-08-01	105 E12	
29	YC82856	BL	2	2008-07-31	2025-08-01	105 E12	
30	YC82857	BL	3	2008-07-31	2025-08-01	105 E12	
31	YC82858	BL	4	2008-07-31	2025-08-01	105 E12	
32	YC82859	BL	5	2008-07-31	2025-08-01	105 E12	
33	YC82860	BL	6	2008-07-31	2025-08-01	105 E12	
34	YC82861	BL	7	2008-07-31	2025-08-01	105 E12	
35	YC82862	BL	8	2008-07-31	2025-08-01	105 E12	
36	YE94141	CAS19	1	2019-08-28	2025-09-03	115J10	
37	YE94142	CAS19	2	2019-08-28	2025-09-03	115J10	
38	YE94143	CAS19	3	2019-08-28	2025-09-03	115J10	
39	YE94144	CAS19	4	2019-08-28	2025-09-03	115J10	
40	YE94145	CAS19	5	2019-08-28	2025-09-03	115J10	
41	YE94146	CAS19	6	2019-08-28	2025-09-03	115J10	
42	YE94147	CAS19	7	2019-08-28	2025-09-03	115J10	
43	YE94148	CAS19	8	2019-08-28	2025-09-03	115J10	
44	YE94149	CAS19	9	2019-08-28	2025-09-03	115J10	
45	YE94150	CAS19	10	2019-08-28	2025-09-03	115J10	
46	YE94151	CAS19	11	2019-08-28	2025-09-03	115J10	
47	YE94152	CAS19	12	2019-08-28	2025-09-03	115J10	
48	YE94153	CAS19	13	2019-08-28	2025-09-03	115J10	
49	YE94154	CAS19	14	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
50	YE94155	CAS19	15	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
51	YE94156	CAS19	16	2019-08-28	2025-09-03	115J10	
52	YE94157	CAS19	17	2019-08-28	2025-09-03	115J10	
53	YE94158	CAS19	18	2019-08-28	2025-09-03	115J10	
54	YE94159	CAS19	19	2019-08-28	2025-09-03	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
55	YE94160	CAS19	20	2019-08-28	2025-09-03	115J10	
56	YE94161	CAS19	21	2019-08-28	2025-09-03	115J10	
57	YE94162	CAS19	22	2019-08-28	2025-09-03	115J10	
58	YE94163	CAS19	23	2019-08-28	2025-09-03	115J10	
59	YE94164	CAS19	24	2019-08-28	2025-09-03	115J10	
60	YE94165	CAS19	25	2019-08-28	2025-09-03	115J10	
61	YE94166	CAS19	26	2019-08-28	2025-09-03	115J10	
62	YE94167	CAS19	27	2019-08-28	2025-09-03	115J10	
63	YE94168	CAS19	28	2019-08-28	2025-09-03	115J10	
64	YE94169	CAS19	29	2019-08-28	2025-09-03	115J10	
65	YE94170	CAS19	30	2019-08-28	2025-09-03	115J10	
66	YE94171	CAS19	31	2019-08-28	2025-09-03	115J10	
67	YE94172	CAS19	32	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
68	YE94173	CAS19	33	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
69	YE94174	CAS19	34	2019-08-28	2025-09-03	115J10	
70	YE94175	CAS19	35	2019-08-28	2025-09-03	115J10	
71	YE94176	CAS19	36	2019-08-28	2025-09-03	115J10	
72	YE94177	CAS19	37	2019-08-27	2025-09-03	115J10	
73	YE94178	CAS19	38	2019-08-27	2025-09-03	115J10	
74	YE94179	CAS19	39	2019-08-27	2025-09-03	115J10	
75	YE94180	CAS19	40	2019-08-27	2025-09-03	115J10	
76	YE94181	CAS19	41	2019-08-27	2025-09-03	115J10	
77	YE94182	CAS19	42	2019-08-27	2025-09-03	115J10	
78	YE94183	CAS19	43	2019-08-27	2025-09-03	115J10	
79	YE94184	CAS19	44	2019-08-27	2025-09-03	115J10	
80	YE94185	CAS19	45	2019-08-27	2025-09-03	115J10	
81	YE94186	CAS19	46	2019-08-27	2025-09-03	115J10	
82	YE94187	CAS19	47	2019-08-27	2025-09-03	115J10	
83	YE94188	CAS19	48	2019-08-27	2025-09-03	115J10	
84	YE94189	CAS19	49	2019-08-27	2025-09-03	115J10	Full Quartz fraction (25+ acres)
85	YE94190	CAS19	50	2019-08-27	2025-09-03	115J10	
86	YE94191	CAS19	51	2019-08-27	2025-09-03	115J10	
87	YE94192	CAS19	52	2019-08-27	2025-09-03	115J10	
88	YE94193	CAS19	53	2019-08-27	2025-09-03	115J10	Full Quartz fraction (25+ acres)
89	YC99925	KANA	46	2010-06-05	2026-06-08	115J15	Full Quartz fraction (25+ acres)
90	YB37540	AZTEC	1	1992-09-12	2026-09-21	115J10	
91	YB37541	AZTEC	2	1992-09-12	2026-09-21	115J10	
92	YB37542	AZTEC	3	1992-09-12	2026-09-21	115J10	
93	YB37543	AZTEC	4	1992-09-12	2026-09-21	115J10	
94	YB37544	AZTEC	5	1992-09-12	2026-09-21	115J10	
95	YB37545	AZTEC	6	1992-09-12	2026-09-21	115J10	
96	YB37546	AZTEC	7	1992-09-12	2026-09-21	115J10	
97	YB37547	AZTEC	8	1992-09-12	2026-09-21	115J10	
98	YB37548	AZTEC	9	1992-09-12	2026-09-21	115J10	
99	YB37549	AZTEC	10	1992-09-12	2026-09-21	115J10	
100	YB37622	MAYA	31	1992-09-12	2026-09-21	115J10	
101	YB37623	MAYA	32	1992-09-12	2026-09-21	115J10	
102	YB37624	MAYA	33	1992-09-12	2026-09-21	115J10	
103	YB37625	MAYA	34	1992-09-12	2026-09-21	115J10	
104	YB37626	MAYA	35	1992-09-12	2026-09-21	115J10	
105	YB37627	MAYA	36	1992-09-12	2026-09-21	115J10	
106	YB37628	MAYA	37	1992-09-12	2026-09-21	115J10	
107	YB37629	MAYA	38	1992-09-12	2026-09-21	115J10	
108	YB37630	MAYA	39	1992-09-12	2026-09-21	115J10	
109	YB37631	MAYA	40	1992-09-12	2026-09-21	115J10	
110	YC99915	KANA	37	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
111	YC99916	KANA	38	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
112	YC99917	KANA	39	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
113	YC99918	KANA	40	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
114	YC99919	KANA	41	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
115	YC99920	KANA	42	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
116	YC99921	KANA	43	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
117	YC99922	KANA	44	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
118	YC99923	KANA	45	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
119	YB37830	ICE	30	1993-01-22	2027-01-27	115J11	
120	YB37831	ICE	31	1993-01-22	2027-01-27	115J11	
121	YB37832	ICE	32	1993-01-22	2027-01-27	115J11	
122	YB37833	ICE	33	1993-01-22	2027-01-27	115J11	
123	YB37841	ICE	41	1993-01-22	2027-01-27	115J10	
124	YB37842	ICE	42	1993-01-22	2027-01-27	115J10	
125	YB37843	ICE	43	1993-01-22	2027-01-27	115J11	
126	YB37844	ICE	44	1993-01-22	2027-01-27	115J11	
127	YB37845	ICE	45	1993-01-22	2027-01-27	115J11	
128	YB37846	ICE	46	1993-01-22	2027-01-27	115J11	
129	YB37847	ICE	47	1993-01-22	2027-01-27	115J11	
130	YD17559	AXS	1	2009-10-05	2027-03-25	115J15	
131	YD17560	AXS	2	2009-10-05	2027-03-25	115J15	
132	YD17561	AXS	3	2009-10-05	2027-03-25	115J15	
133	YD17562	AXS	4	2009-10-05	2027-03-25	115J15	
134	YD17563	AXS	5	2009-10-05	2027-03-25	115J15	
135	YD17564	AXS	6	2009-10-05	2027-03-25	115J15	
136	YD17565	AXS	7	2009-10-05	2027-03-25	115J10	
137	YD17566	AXS	8	2009-10-05	2027-03-25	115J10	
138	YD17567	AXS	9	2009-10-05	2027-03-25	115J10	
139	YD17568	AXS	10	2009-10-05	2027-03-25	115J10	
140	YD17569	AXS	11	2009-10-06	2027-03-25	115J10	
141	YD17570	AXS	12	2009-10-06	2027-03-25	115J10	
142	YD17571	AXS	13	2009-10-06	2027-03-25	115J10	
143	YD17572	AXS	14	2009-10-06	2027-03-25	115J10	
144	YD17573	AXS	15	2009-10-06	2027-03-25	115J10	
145	YD17574	AXS	16	2009-10-06	2027-03-25	115J10	
146	YD17575	AXS	17	2009-10-06	2027-03-25	115J10	
147	YD17576	AXS	18	2009-10-06	2027-03-25	115J10	
148	YD17577	AXS	19	2009-10-05	2027-03-25	115J10	
149	YD17578	AXS	20	2009-10-05	2027-03-25	115J10	
150	YD17579	AXS	21	2009-10-05	2027-03-25	115J10	
151	YD17580	AXS	22	2009-10-05	2027-03-25	115J10	
152	YD17581	AXS	23	2009-10-05	2027-03-25	115J10	
153	YD17582	AXS	24	2009-10-05	2027-03-25	115J10	
154	YD17583	AXS	25	2009-10-05	2027-03-25	115J10	
155	YD17584	AXS	26	2009-10-05	2027-03-25	115J10	
156	YD17585	AXS	27	2009-10-05	2027-03-25	115J10	
157	YD17586	AXS	28	2009-10-05	2027-03-25	115J10	
158	YD17587	AXS	29	2009-10-05	2027-03-25	115J10	
159	YD17588	AXS	30	2009-10-05	2027-03-25	115J10	
160	YD17589	AXS	31	2009-10-05	2027-03-25	115J10	
161	YD17590	AXS	32	2009-10-05	2027-03-25	115J10	
162	YD17591	AXS	33	2009-10-05	2027-03-25	115J10	
163	YD17592	AXS	34	2009-10-05	2027-03-25	115J10	
164	YD17593	AXS	35	2009-10-05	2027-03-25	115J10	
165	YD17594	AXS	36	2009-10-05	2027-03-25	115J10	
166	YD17595	AXS	37	2009-10-05	2027-03-25	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
167	YD17596	AXS	38	2009-10-05	2027-03-25	115J10	
168	YD17597	AXS	39	2009-10-05	2027-03-25	115J10	
169	YD17598	AXS	40	2009-10-05	2027-03-25	115J10	
170	YD17599	AXS	41	2009-10-05	2027-03-25	115J10	
171	YD17600	AXS	42	2009-10-05	2027-03-25	115J10	
172	YD17601	AXS	43	2009-10-05	2027-03-25	115J10	
173	YD17602	AXS	44	2009-10-05	2027-03-25	115J10	
174	YD17603	AXS	45	2009-10-05	2027-03-25	115J10	
175	YD17604	AXS	46	2009-10-05	2027-03-25	115J10	
176	YD17605	AXS	47	2009-10-05	2027-03-25	115J10	
177	YD17606	AXS	48	2009-10-05	2027-03-25	115J10	
178	YD17607	AXS	49	2009-10-06	2027-03-25	115J10	
179	YD17608	AXS	50	2009-10-06	2027-03-25	115J10	
180	YD17609	AXS	51	2009-10-06	2027-03-25	115J10	
181	YD17610	AXS	52	2009-10-06	2027-03-25	115J10	
182	YD17611	AXS	53	2009-10-06	2027-03-25	115J10	
183	YD17612	AXS	54	2009-10-06	2027-03-25	115J10	
184	YD17613	AXS	55	2009-10-05	2027-03-25	115J10	
185	YD17614	AXS	56	2009-10-05	2027-03-25	115J09	
186	YD17615	AXS	57	2009-10-05	2027-03-25	115J10	
187	YD17616	AXS	58	2009-10-05	2027-03-25	115J10	
188	YD17617	AXS	59	2009-10-05	2027-03-25	115J10	
189	YD17618	AXS	60	2009-10-05	2027-03-25	115J10	
190	YD17619	AXS	61	2009-10-05	2027-03-25	115J09	
191	YD17620	AXS	62	2009-10-05	2027-03-25	115J09	
192	YD17621	AXS	63	2009-10-05	2027-03-25	115J09	
193	YD17622	AXS	64	2009-10-05	2027-03-25	115J09	
194	YD17623	AXS	65	2009-10-05	2027-03-25	115J09	
195	YD17624	AXS	66	2009-10-05	2027-03-25	115J09	
196	YD17625	AXS	67	2009-10-05	2027-03-25	115J09	
197	YD17626	AXS	68	2009-10-05	2027-03-25	115J09	
198	YD17627	AXS	69	2009-10-06	2027-03-25	115J09	
199	YD17628	AXS	70	2009-10-06	2027-03-25	115J09	
200	YD17629	AXS	71	2009-10-06	2027-03-25	115J09	
201	YD17630	AXS	72	2009-10-06	2027-03-25	115J09	
202	YD17631	AXS	73	2009-10-06	2027-03-25	115J09	
203	YD17632	AXS	74	2009-10-06	2027-03-25	115J09	
204	YD17633	AXS	75	2009-10-05	2027-03-25	115J09	
205	YD17634	AXS	76	2009-10-05	2027-03-25	115J09	
206	YD17635	AXS	77	2009-10-05	2027-03-25	115J09	
207	YD17636	AXS	78	2009-10-05	2027-03-25	115J09	
208	YD17637	AXS	79	2009-10-05	2027-03-25	115J09	
209	YD17638	AXS	80	2009-10-05	2027-03-25	115J09	
210	YD17639	AXS	81	2009-10-05	2027-03-25	115J09	
211	YD17640	AXS	82	2009-10-05	2027-03-25	115J09	
212	YD17641	AXS	83	2009-10-05	2027-03-25	115J09	
213	YD17642	AXS	84	2009-10-05	2027-03-25	115J09	
214	YD17643	AXS	85	2009-10-05	2027-03-25	115J09	
215	YD17644	AXS	86	2009-10-05	2027-03-25	115J09	
216	YD17645	AXS	87	2009-10-06	2027-03-25	115J09	
217	YD17646	AXS	88	2009-10-06	2027-03-25	115J09	
218	YD17647	AXS	89	2009-10-06	2027-03-25	115J09	
219	YD17648	AXS	90	2009-10-06	2027-03-25	115J09	
220	YD17649	AXS	91	2009-10-06	2027-03-25	115J09	
221	YD17650	AXS	92	2009-10-06	2027-03-25	115J09	
222	YD17651	AXS	103	2009-10-07	2027-03-25	115J09	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
223	YD17652	AXS	102	2009-10-07	2027-03-25	115J16	
224	YD17653	AXS	101	2009-10-07	2027-03-25	115J16	
225	YD17654	AXS	100	2009-10-06	2027-03-25	115J16	
226	YD17655	AXS	99	2009-10-06	2027-03-25	115J16	
227	YD17656	AXS	98	2009-10-06	2027-03-25	115J16	
228	YD17657	AXS	97	2009-10-06	2027-03-25	115J16	
229	YD17658	AXS	96	2009-10-06	2027-03-25	115J16	
230	YD17659	AXS	95	2009-10-06	2027-03-25	115J16	
231	YD17660	AXS	94	2009-10-06	2027-03-25	115J16	
232	YD17661	AXS	93	2009-10-06	2027-03-25	115J16	
233	YD17662	AXS	104	2009-10-07	2027-03-25	115J09	
234	YD17663	AXS	105	2009-10-07	2027-03-25	115J09	
235	YD17664	AXS	106	2009-10-07	2027-03-25	115J09	
236	YD17665	AXS	107	2009-10-07	2027-03-25	115J09	
237	YD17666	AXS	108	2009-10-07	2027-03-25	115J09	
238	YD17667	AXS	109	2009-10-07	2027-03-25	115J09	
239	YD17668	AXS	110	2009-10-07	2027-03-25	115J09	
240	YD17669	AXS	111	2009-10-07	2027-03-25	115J09	
241	YD17670	AXS	112	2009-10-07	2027-03-25	115J09	
242	YD17694	AXS	136	2009-10-06	2027-03-25	115J09	
243	YD17671	AXS	113	2009-10-06	2027-03-25	115J09	
244	YD17672	AXS	114	2009-10-06	2027-03-25	115J09	
245	YD17673	AXS	115	2009-10-06	2027-03-25	115J09	
246	YD17674	AXS	116	2009-10-06	2027-03-25	115J09	
247	YD17675	AXS	117	2009-10-06	2027-03-25	115J09	
248	YD17676	AXS	118	2009-10-06	2027-03-25	115J09	
249	YD17677	AXS	119	2009-10-06	2027-03-25	115J09	
250	YD17678	AXS	120	2009-10-06	2027-03-25	115J09	
251	YD17679	AXS	121	2009-10-06	2027-03-25	115J09	
252	YD17680	AXS	122	2009-10-06	2027-03-25	115J09	
253	YD17681	AXS	123	2009-10-06	2027-03-25	115J09	
254	YD17682	AXS	124	2009-10-06	2027-03-25	115J09	
255	YD17683	AXS	125	2009-10-06	2027-03-25	115J09	
256	YD17684	AXS	126	2009-10-06	2027-03-25	115J09	
257	YD17685	AXS	127	2009-10-06	2027-03-25	115J09	
258	YD17686	AXS	128	2009-10-06	2027-03-25	115J09	
259	YD17687	AXS	129	2009-10-06	2027-03-25	115J09	
260	YD17688	AXS	130	2009-10-06	2027-03-25	115J09	
261	YD17689	AXS	131	2009-10-06	2027-03-25	115J09	
262	YD17690	AXS	132	2009-10-06	2027-03-25	115J09	
263	YD17691	AXS	133	2009-10-06	2027-03-25	115J09	
264	YD17692	AXS	134	2009-10-06	2027-03-25	115J09	
265	YD17693	AXS	135	2009-10-06	2027-03-25	115J09	
266	YD08825	BERG	3	2010-06-06	2027-06-08	115J15	
267	YD08824	BERG	4	2010-06-06	2027-06-08	115J15	
268	YD08823	BERG	5	2010-06-06	2027-06-08	115J15	
269	YD08822	BERG	6	2010-06-06	2027-06-08	115J15	
270	YD08821	BERG	7	2010-06-06	2027-06-08	115J14	
271	YD08820	BERG	8	2010-06-06	2027-06-08	115J14	
272	YD08819	BERG	9	2010-06-06	2027-06-08	115J14	
273	YD08818	BERG	10	2010-06-06	2027-06-08	115J14	
274	YD08817	BERG	11	2010-06-06	2027-06-08	115J14	
275	YD08816	BERG	12	2010-06-06	2027-06-08	115J14	
276	YD08815	BERG	13	2010-06-06	2027-06-08	115J14	
277	YD08814	BERG	14	2010-06-06	2027-06-08	115J14	
278	YD08813	BERG	15	2010-06-06	2027-06-08	115J14	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
279	YD08812	BERG	16	2010-06-06	2027-06-08	115J14	
280	YD08811	BERG	17	2010-06-06	2027-06-08	115J14	
281	YD08810	BERG	18	2010-06-06	2027-06-08	115J14	
282	YD08809	BERG	19	2010-06-07	2027-06-08	115J14	
283	YD08808	BERG	20	2010-06-07	2027-06-08	115J14	
284	YD08807	BERG	21	2010-06-07	2027-06-08	115J14	
285	YD08806	BERG	22	2010-06-07	2027-06-08	115J14	
286	YD08827	BERG	27	2010-06-06	2027-06-08	115J15	
287	YD08828	BERG	28	2010-06-06	2027-06-08	115J15	
288	YD08829	BERG	29	2010-06-06	2027-06-08	115J15	
289	YD08830	BERG	30	2010-06-06	2027-06-08	115J15	
290	YD08831	BERG	31	2010-06-06	2027-06-08	115J14	
291	YD08832	BERG	32	2010-06-06	2027-06-08	115J14	
292	YD08833	BERG	33	2010-06-06	2027-06-08	115J14	
293	YD08834	BERG	34	2010-06-06	2027-06-08	115J14	
294	YD08835	BERG	35	2010-06-06	2027-06-08	115J14	
295	YD08836	BERG	36	2010-06-06	2027-06-08	115J14	
296	YD08837	BERG	37	2010-06-06	2027-06-08	115J14	
297	YD08838	BERG	38	2010-06-06	2027-06-08	115J14	
298	YD08839	BERG	39	2010-06-06	2027-06-08	115J14	
299	YD08840	BERG	40	2010-06-06	2027-06-08	115J14	
300	YD08841	BERG	41	2010-06-06	2027-06-08	115J14	
301	YD08842	BERG	42	2010-06-06	2027-06-08	115J14	
302	YD08847	BERG	47	2010-06-05	2027-06-08	115J11	
303	YD08848	BERG	48	2010-06-05	2027-06-08	115J11	
304	YD08849	BERG	49	2010-06-05	2027-06-08	115J11	
305	YD08850	BERG	50	2010-06-05	2027-06-08	115J11	
306	YD08854	BERG	54	2010-06-05	2027-06-08	115J11	
307	YD08855	BERG	55	2010-06-05	2027-06-08	115J11	
308	YD08856	BERG	56	2010-06-05	2027-06-08	115J11	
309	YD08853	BERG	53	2010-06-05	2027-06-08	115J11	Partial Quartz fraction (<25 acres)
310	YD08802	BERG	59	2010-06-07	2027-06-08	115J11	Full Quartz fraction (25+ acres)
311	YC99926	KANA	47	2010-06-05	2027-06-08	115J15	Partial Quartz fraction (<25 acres)
312	YC99924	KANA	58	2010-06-04	2027-06-08	115J10	Partial Quartz fraction (<25 acres)
313	YC99927	KANA	48	2010-06-08	2027-06-08	115J15	
314	YC99928	KANA	49	2010-06-08	2027-06-08	115J15	
315	YC99929	KANA	50	2010-06-08	2027-06-08	115J15	
316	YC99930	KANA	51	2010-06-08	2027-06-08	115J15	
317	YC99931	KANA	52	2010-06-08	2027-06-08	115J15	
318	YC99932	KANA	53	2010-06-08	2027-06-08	115J15	
319	YC99933	KANA	54	2010-06-08	2027-06-08	115J15	
320	YC99934	KANA	55	2010-06-08	2027-06-08	115J15	
321	YC99935	KANA	56	2010-06-05	2027-06-08	115J15	
322	YC99936	KANA	57	2010-06-05	2027-06-08	115J15	
323	YC99879	KANA	1	2009-06-20	2027-06-22	115J15	
324	YC99880	KANA	2	2009-06-20	2027-06-22	115J15	
325	YC99881	KANA	3	2009-06-20	2027-06-22	115J15	
326	YC99882	KANA	4	2009-06-20	2027-06-22	115J15	
327	YC99883	KANA	5	2009-06-20	2027-06-22	115J15	
328	YC99884	KANA	6	2009-06-20	2027-06-22	115J15	
329	YC99885	KANA	7	2009-06-20	2027-06-22	115J15	
330	YC99886	KANA	8	2009-06-20	2027-06-22	115J15	
331	YC99887	KANA	9	2009-06-20	2027-06-22	115J15	
332	YC99888	KANA	10	2009-06-20	2027-06-22	115J15	
333	YC99889	KANA	11	2009-06-20	2027-06-22	115J15	
334	YC99890	KANA	12	2009-06-20	2027-06-22	115J15	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
335	YC99891	KANA	13	2009-06-20	2027-06-22	115J15	
336	YC99892	KANA	14	2009-06-20	2027-06-22	115J15	
337	YC99893	KANA	15	2009-06-20	2027-06-22	115J15	
338	YC99894	KANA	16	2009-06-20	2027-06-22	115J15	
339	YC99895	KANA	17	2009-06-20	2027-06-22	115J15	
340	YC99896	KANA	18	2009-06-20	2027-06-22	115J15	
341	YC99897	KANA	19	2009-06-20	2027-06-22	115J15	
342	YC99898	KANA	20	2009-06-20	2027-06-22	115J15	
343	YC99899	KANA	21	2009-06-20	2027-06-22	115J15	
344	YC99900	KANA	22	2009-06-20	2027-06-22	115J15	
345	YC99901	KANA	23	2009-06-20	2027-06-22	115J15	
346	YC99902	KANA	24	2009-06-20	2027-06-22	115J15	
347	YC99903	KANA	25	2009-06-20	2027-06-22	115J15	
348	YC99904	KANA	26	2009-06-20	2027-06-22	115J15	
349	YC99905	KANA	27	2009-06-20	2027-06-22	115J15	
350	YC99906	KANA	28	2009-06-20	2027-06-22	115J15	
351	YC99907	KANA	29	2009-06-20	2027-06-22	115J15	
352	YC99908	KANA	30	2009-06-20	2027-06-22	115J15	
353	YC99909	KANA	31	2009-06-20	2027-06-22	115J15	
354	YC99910	KANA	32	2009-06-20	2027-06-22	115J15	
355	YC99911	KANA	33	2009-06-20	2027-06-22	115J15	
356	YC99912	KANA	34	2009-06-20	2027-06-22	115J15	
357	YC99913	KANA	35	2009-06-20	2027-06-22	115J15	
358	YD08861	BERG F	61	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
359	YD08862	BERG F	62	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
360	YD08863	BERG F	63	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
361	YD08864	BERG F	64	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
362	YD08865	BERG F	65	2010-08-09	2027-08-13	115J11	Full Quartz fraction (25+ acres)
363	YD08866	BERG F	66	2010-08-09	2027-08-13	115J11	Full Quartz fraction (25+ acres)
364	YD08867	BERG F	67	2010-08-09	2027-08-13	115J14	Partial Quartz fraction (<25 acres)
365	YB37482	KOFFEE	1	1992-09-12	2027-09-21	115J10	
366	YB37483	KOFFEE	2	1992-09-12	2027-09-21	115J10	
367	YB37484	KOFFEE	3	1992-09-12	2027-09-21	115J10	
368	YB37485	KOFFEE	4	1992-09-12	2027-09-21	115J10	
369	YB37486	KOFFEE	5	1992-09-12	2027-09-21	115J10	
370	YB37487	KOFFEE	6	1992-09-12	2027-09-21	115J10	
371	YB37488	KOFFEE	7	1992-09-12	2027-09-21	115J10	
372	YB37489	KOFFEE	8	1992-09-12	2027-09-21	115J10	
373	YB37490	KOFFEE	9	1992-09-12	2027-09-21	115J10	
374	YB37491	KOFFEE	10	1992-09-12	2027-09-21	115J10	
375	YB37492	KOFFEE	11	1992-09-12	2027-09-21	115J10	
376	YB37493	KOFFEE	12	1992-09-12	2027-09-21	115J10	
377	YB37494	KOFFEE	13	1992-09-12	2027-09-21	115J10	
378	YB37495	KOFFEE	14	1992-09-12	2027-09-21	115J10	
379	YB37496	KOFFEE	15	1992-09-12	2027-09-21	115J10	
380	YB37497	KOFFEE	16	1992-09-12	2027-09-21	115J10	
381	YB37498	KOFFEE	17	1992-09-12	2027-09-21	115J10	
382	YB37499	KOFFEE	18	1992-09-12	2027-09-21	115J10	
383	YB37500	KOFFEE	19	1992-09-12	2027-09-21	115J10	
384	YB37501	KOFFEE	20	1992-09-12	2027-09-21	115J10	
385	YB37502	KOFFEE	21	1992-09-12	2027-09-21	115J10	
386	YB37503	KOFFEE	22	1992-09-12	2027-09-21	115J10	
387	YB37504	KOFFEE	23	1992-09-12	2027-09-21	115J10	
388	YB37505	KOFFEE	24	1992-09-12	2027-09-21	115J10	
389	YB37506	KOFFEE	25	1992-09-12	2027-09-21	115J10	
390	YB37507	KOFFEE	26	1992-09-12	2027-09-21	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
391	YB37508	KOFFEE	27	1992-09-12	2027-09-21	115J10	
392	YB37509	KOFFEE	28	1992-09-12	2027-09-21	115J10	
393	YB37510	KOFFEE	29	1992-09-13	2027-09-21	115J10	
394	YB37511	KOFFEE	30	1992-09-13	2027-09-21	115J10	
395	YB37512	KOFFEE	31	1992-09-13	2027-09-21	115J10	
396	YB37513	KOFFEE	32	1992-09-13	2027-09-21	115J10	
397	YB37514	KOFFEE	33	1992-09-13	2027-09-21	115J10	
398	YB37515	KOFFEE	34	1992-09-13	2027-09-21	115J10	
399	YB37516	KOFFEE	35	1992-09-13	2027-09-21	115J10	
400	YB37517	KOFFEE	36	1992-09-13	2027-09-21	115J10	
401	YB37518	KOFFEE	37	1992-09-13	2027-09-21	115J10	
402	YB37519	KOFFEE	38	1992-09-13	2027-09-21	115J10	
403	YB37520	KOFFEE	39	1992-09-13	2027-09-21	115J10	
404	YB37521	KOFFEE	40	1992-09-13	2027-09-21	115J10	
405	YB37522	KOFFEE	41	1992-09-13	2027-09-21	115J10	
406	YB37523	KOFFEE	42	1992-09-13	2027-09-21	115J10	
407	YB37524	KOFFEE	43	1992-09-13	2027-09-21	115J10	
408	YB37525	KOFFEE	44	1992-09-13	2027-09-21	115J10	
409	YB37526	KOFFEE	45	1992-09-13	2027-09-21	115J10	
410	YB37527	KOFFEE	46	1992-09-13	2027-09-21	115J10	
411	YB37528	KOFFEE	47	1992-09-13	2027-09-21	115J10	
412	YB37529	KOFFEE	48	1992-09-13	2027-09-21	115J10	
413	YB37530	KOFFEE	49	1992-09-13	2027-09-21	115J10	
414	YB37531	KOFFEE	50	1992-09-13	2027-09-21	115J10	
415	YB37532	KOFFEE	51	1992-09-13	2027-09-21	115J10	
416	YB37533	KOFFEE	52	1992-09-13	2027-09-21	115J10	
417	YB37534	KOFFEE	53	1992-09-13	2027-09-21	115J10	
418	YB37535	KOFFEE	54	1992-09-13	2027-09-21	115J10	
419	YB37536	KOFFEE	55	1992-09-13	2027-09-21	115J10	
420	YB37537	KOFFEE	56	1992-09-13	2027-09-21	115J10	
421	YB37538	KOFFEE	57	1992-09-13	2027-09-21	115J10	
422	YB37539	KOFFEE	58	1992-09-13	2027-09-21	115J10	
423	YC99914	KANA	36	2009-09-01	2027-09-29	115J15	Partial Quartz fraction (<25 acres)
424	YB37801	ICE	1	1993-01-21	2028-01-27	115J10	
425	YB37802	ICE	2	1993-01-21	2028-01-27	115J11	
426	YB37803	ICE	3	1993-01-21	2028-01-27	115J11	
427	YB37804	ICE	4	1993-01-21	2028-01-27	115J11	
428	YB37805	ICE	5	1993-01-21	2028-01-27	115J11	
429	YB37809	ICE	9	1993-01-22	2028-01-27	115J10	
430	YB37810	ICE	10	1993-01-22	2028-01-27	115J10	
431	YB37811	ICE	11	1993-01-22	2028-01-27	115J11	
432	YB37812	ICE	12	1993-01-22	2028-01-27	115J11	
433	YB37813	ICE	13	1993-01-22	2028-01-27	115J11	
434	YB37814	ICE	14	1993-01-22	2028-01-27	115J11	
435	YB37815	ICE	15	1993-01-22	2028-01-27	115J11	
436	YB37816	ICE	16	1993-01-22	2028-01-27	115J11	
437	YB37817	ICE	17	1993-01-22	2028-01-27	115J11	
438	YB37818	ICE	18	1993-01-22	2028-01-27	115J11	
439	YB37825	ICE	25	1993-01-22	2028-01-27	115J10	
440	YB37826	ICE	26	1993-01-22	2028-01-27	115J10	
441	YB37827	ICE	27	1993-01-22	2028-01-27	115J11	
442	YB37828	ICE	28	1993-01-22	2028-01-27	115J11	
443	YB37829	ICE	29	1993-01-22	2028-01-27	115J11	
444	YA86735	ANA	1	1985-04-25	2028-02-17	115J10	
445	YA86736	ANA	2	1985-04-25	2028-02-17	115J10	
446	YA86737	ANA	3	1985-04-25	2028-02-17	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
447	YA86738	ANA	4	1985-04-25	2028-02-17	115J10	
448	YA86739	ANA	5	1985-04-25	2028-02-17	115J10	
449	YA86740	ANA	6	1985-04-25	2028-02-17	115J10	
450	YA86741	ANA	7	1985-04-25	2028-02-17	115J10	
451	YA86742	ANA	8	1985-04-25	2028-02-17	115J10	
452	YA86743	ANA	9	1985-04-25	2028-02-17	115J10	
453	YA86744	ANA	10	1985-04-25	2028-02-17	115J10	
454	YA86749	ANA	15	1985-04-25	2028-02-17	115J10	
455	YA86750	ANA	16	1985-04-25	2028-02-17	115J10	
456	YA86751	ANA	17	1985-04-25	2028-02-17	115J10	
457	YA86752	ANA	18	1985-04-25	2028-02-17	115J10	
458	YA86753	ANA	19	1985-04-25	2028-02-17	115J10	
459	YA86754	ANA	20	1985-04-25	2028-02-17	115J10	
460	YA86755	ANA	21	1985-04-25	2028-02-17	115J10	
461	YA86756	ANA	22	1985-04-25	2028-02-17	115J10	
462	YA86757	ANA	23	1985-04-25	2028-02-17	115J10	
463	YA86758	ANA	24	1985-04-25	2028-02-17	115J10	
464	YA86759	ANA	25	1985-04-25	2028-02-17	115J10	
465	YA86760	ANA	26	1985-04-25	2028-02-17	115J10	
466	YA86763	ANA	29	1985-04-25	2028-02-17	115J10	
467	YA86764	ANA	30	1985-04-25	2028-02-17	115J10	
468	YA86765	ANA	31	1985-04-25	2028-02-17	115J10	
469	YA86766	ANA	32	1985-04-25	2028-02-17	115J10	
470	YA86767	ANA	33	1985-04-25	2028-02-17	115J10	
471	YA86768	ANA	34	1985-04-25	2028-02-17	115J10	
472	YA86769	ANA	35	1985-04-25	2028-02-17	115J10	
473	YA86770	ANA	36	1985-04-25	2028-02-17	115J10	
474	YA86771	ANA	37	1985-04-25	2028-02-17	115J10	
475	YA86772	ANA	38	1985-04-25	2028-02-17	115J10	
476	YA86773	ANA	39	1985-04-25	2028-02-17	115J10	
477	YA86774	ANA	40	1985-04-25	2028-02-17	115J10	
478	YA86777	ANA	43	1985-04-25	2028-02-17	115J10	
479	YA86778	ANA	44	1985-04-25	2028-02-17	115J10	
480	YA86779	ANA	45	1985-04-25	2028-02-17	115J10	
481	YA86780	ANA	46	1985-04-25	2028-02-17	115J10	
482	YA86781	ANA	47	1985-04-25	2028-02-17	115J10	
483	YA86782	ANA	48	1985-04-25	2028-02-17	115J10	
484	YA86783	ANA	49	1985-04-25	2028-02-17	115J10	
485	YA86784	ANA	50	1985-04-25	2028-02-17	115J10	
486	YA86785	ANA	51	1985-04-25	2028-02-17	115J10	
487	YA86786	ANA	52	1985-04-25	2028-02-17	115J10	
488	YA86787	ANA	53	1985-04-25	2028-02-17	115J10	
489	YA86788	ANA	54	1985-04-25	2028-02-17	115J10	
490	YE32245	PAL	1	2016-05-17	2028-03-25	115J10	
491	YE32246	PAL	2	2016-05-17	2028-03-25	115J10	
492	YE32247	PAL	3	2016-05-17	2028-03-25	115J10	
493	YE32248	PAL	4	2016-05-17	2028-03-25	115J10	
494	YE32249	PAL	5	2016-05-17	2028-03-25	115J10	
495	YE32138	PAL	6	2016-05-16	2028-03-25	115J10	
496	YE32139	PAL	7	2016-05-16	2028-03-25	115J10	
497	YE32140	PAL	8	2016-05-16	2028-03-25	115J10	
498	YE32141	PAL	9	2016-05-16	2028-03-25	115J10	
499	YE32142	PAL	10	2016-05-16	2028-03-25	115J10	
500	YE32143	PAL	11	2016-05-16	2028-03-25	115J10	
501	YE32144	PAL	12	2016-05-16	2028-03-25	115J10	
502	YE32145	PAL	13	2016-05-16	2028-03-25	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
503	YE32146	PAL	14	2016-05-17	2028-03-25	115J10	
504	YE32147	PAL	15	2016-05-17	2028-03-25	115J10	
505	YE32148	PAL	16	2016-05-17	2028-03-25	115J10	
506	YE32149	PAL	17	2016-05-17	2028-03-25	115J10	
507	YE32150	PAL	18	2016-05-17	2028-03-25	115J10	
508	YE32151	PAL	19	2016-05-17	2028-03-25	115J10	
509	YE32152	PAL	20	2016-05-17	2028-03-25	115J10	
510	YE32153	PAL	21	2016-05-17	2028-03-25	115J10	
511	YE32154	PAL	22	2016-05-17	2028-03-25	115J10	
512	YE32155	PAL	23	2016-05-17	2028-03-25	115J10	
513	YE32156	PAL	24	2016-05-17	2028-03-25	115J10	
514	YE32157	PAL	25	2016-05-17	2028-03-25	115J10	
515	YE32196	PAL	26	2016-05-17	2028-03-25	115J10	
516	YE32197	PAL	27	2016-05-17	2028-03-25	115J10	
517	YE32178	PAL	28	2016-05-17	2028-03-25	115J10	
518	YE32179	PAL	29	2016-05-17	2028-03-25	115J10	
519	YE32180	PAL	30	2016-05-16	2028-03-25	115J10	
520	YE32181	PAL	31	2016-05-16	2028-03-25	115J10	
521	YE32182	PAL	32	2016-05-16	2028-03-25	115J10	
522	YE32183	PAL	33	2016-05-16	2028-03-25	115J10	
523	YE32184	PAL	34	2016-05-16	2028-03-25	115J10	
524	YE32185	PAL	35	2016-05-16	2028-03-25	115J10	
525	YE32186	PAL	36	2016-05-16	2028-03-25	115J10	
526	YE32187	PAL	37	2016-05-16	2028-03-25	115J10	
527	YE32101	PAL	38	2016-05-16	2028-03-25	115J10	
528	YE32102	PAL	39	2016-05-16	2028-03-25	115J10	
529	YE32103	PAL	40	2016-05-16	2028-03-25	115J10	
530	YE32104	PAL	41	2016-05-16	2028-03-25	115J10	
531	YE32106	PAL	42	2016-05-16	2028-03-25	115J10	
532	YE32105	PAL	43	2016-05-16	2028-03-25	115J10	
533	YE32108	PAL	44	2016-05-16	2028-03-25	115J10	
534	YE32107	PAL	45	2016-05-16	2028-03-25	115J10	
535	YE32110	PAL	46	2016-05-16	2028-03-25	115J10	
536	YE32109	PAL	47	2016-05-16	2028-03-25	115J10	
537	YE32112	PAL	48	2016-05-17	2028-03-25	115J10	
538	YE32111	PAL	49	2016-05-17	2028-03-25	115J10	
539	YE32114	PAL	50	2016-05-17	2028-03-25	115J10	
540	YE32113	PAL	51	2016-05-17	2028-03-25	115J10	
541	YE32115	PAL	52	2016-05-17	2028-03-25	115J10	
542	YE32116	PAL	53	2016-05-17	2028-03-25	115J10	
543	YE32118	PAL	54	2016-05-17	2028-03-25	115J10	
544	YE32117	PAL	55	2016-05-17	2028-03-25	115J10	
545	YE32232	PAL	56	2016-05-17	2028-03-25	115J10	
546	YE32231	PAL	57	2016-05-17	2028-03-25	115J10	
547	YE32233	PAL	58	2016-05-17	2028-03-25	115J10	
548	YE32234	PAL	60	2016-05-17	2028-03-25	115J10	
549	YE32235	PAL	62	2016-05-17	2028-03-25	115J10	
550	YE32236	PAL	64	2016-05-17	2028-03-25	115J10	
551	YE32237	PAL	65	2016-05-17	2028-03-25	115J10	
552	YE32135	PAL	66	2016-05-17	2028-03-25	115J10	
553	YE32136	PAL	67	2016-05-17	2028-03-25	115J10	
554	YE32133	PAL	68	2016-05-17	2028-03-25	115J10	
555	YE32134	PAL	69	2016-05-17	2028-03-25	115J10	
556	YE32119	PAL	70	2016-05-16	2028-03-25	115J10	
557	YE32120	PAL	71	2016-05-16	2028-03-25	115J10	
558	YE32121	PAL	72	2016-05-16	2028-03-25	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
559	YE32122	PAL	73	2016-05-16	2028-03-25	115J10	
560	YE32123	PAL	74	2016-05-16	2028-03-25	115J10	
561	YE32124	PAL	75	2016-05-16	2028-03-25	115J10	
562	YE32125	PAL	108	2016-05-17	2028-03-25	115J10	
563	YE32126	PAL	109	2016-05-17	2028-03-25	115J10	
564	YE32127	PAL	110	2016-05-17	2028-03-25	115J10	
565	YE32128	PAL	111	2016-05-17	2028-03-25	115J10	
566	YE32129	PAL	112	2016-05-17	2028-03-25	115J10	
567	YE32130	PAL	113	2016-05-17	2028-03-25	115J10	
568	YE32131	PAL	114	2016-05-17	2028-03-25	115J10	
569	YE32132	PAL	115	2016-05-17	2028-03-25	115J10	
570	YE32188	PAL	143	2016-05-17	2028-03-25	115J10	
571	YE32200	PAL	150	2016-05-17	2028-03-25	115J10	
572	YE32189	PAL	151	2016-05-17	2028-03-25	115J10	
573	YE32190	PAL	152	2016-05-17	2028-03-25	115J10	
574	YE32191	PAL	153	2016-05-17	2028-03-25	115J10	
575	YE32192	PAL	154	2016-05-17	2028-03-25	115J10	
576	YE32193	PAL	155	2016-05-17	2028-03-25	115J10	
577	YD60030	AXS	137	2010-05-11	2028-03-25	115J09	
578	YD60031	AXS	138	2010-05-11	2028-03-25	115J09	
579	YD60032	AXS	139	2010-05-11	2028-03-25	115J09	
580	YD60033	AXS	140	2010-05-11	2028-03-25	115J09	
581	YD60034	AXS	141	2010-05-11	2028-03-25	115J09	
582	YD60035	AXS	142	2010-05-11	2028-03-25	115J09	
583	YD60036	AXS	143	2010-05-11	2028-03-25	115J10	
584	YD60037	AXS	144	2010-05-11	2028-03-25	115J10	
585	YD60038	AXS	145	2010-05-11	2028-03-25	115J10	
586	YD60039	AXS	146	2010-05-11	2028-03-25	115J10	
587	YD60040	AXS	147	2010-05-11	2028-03-25	115J10	
588	YD60041	AXS	148	2010-05-11	2028-03-25	115J10	
589	YD60042	AXS	149	2010-05-11	2028-03-25	115J10	
590	YD60043	AXS	150	2010-05-11	2028-03-25	115J10	
591	YD60044	AXS	151	2010-05-11	2028-03-25	115J10	
592	YD60045	AXS	152	2010-05-11	2028-03-25	115J10	
593	YD60046	AXS	154	2010-05-11	2028-03-25	115J10	
594	YD60047	AXS	153	2010-05-11	2028-03-25	115J10	
595	YD60048	AXS	155	2010-05-11	2028-03-25	115J10	
596	YD60049	AXS	156	2010-05-11	2028-03-25	115J10	
597	YD60050	AXS	157	2010-05-11	2028-03-25	115J10	
598	YD60051	AXS	158	2010-05-11	2028-03-25	115J10	
599	YD60052	AXS	159	2010-05-11	2028-03-25	115J10	
600	YD60053	AXS	160	2010-05-11	2028-03-25	115J10	
601	YD60054	AXS	161	2010-05-11	2028-03-25	115J10	
602	YD60055	AXS	162	2010-05-11	2028-03-25	115J10	
603	YD60056	AXS	163	2010-05-11	2028-03-25	115J10	
604	YD60057	AXS	164	2010-05-11	2028-03-25	115J10	
605	YD60058	AXS	165	2010-05-11	2028-03-25	115J10	
606	YD60059	AXS	166	2010-05-11	2028-03-25	115J10	
607	YD60060	AXS	167	2010-05-11	2028-03-25	115J10	
608	YD60061	AXS	168	2010-05-11	2028-03-25	115J10	
609	YD60062	AXS	169	2010-05-11	2028-03-25	115J10	
610	YD60063	AXS	170	2010-05-11	2028-03-25	115J10	
611	YD60064	AXS	171	2010-05-11	2028-03-25	115J10	
612	YD60065	AXS	172	2010-05-11	2028-03-25	115J10	
613	YD60066	AXS	173	2010-05-11	2028-03-25	115J09	
614	YD60067	AXS	174	2010-05-11	2028-03-25	115J09	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
615	YD60068	AXS	175	2010-05-11	2028-03-25	115J09	
616	YD60069	AXS	176	2010-05-11	2028-03-25	115J09	
617	YD60070	AXS	177	2010-05-11	2028-03-25	115J09	
618	YD60071	AXS	178	2010-05-11	2028-03-25	115J09	
619	YD60072	AXS	179	2010-05-11	2028-03-25	115J09	
620	YD60073	AXS	180	2010-05-11	2028-03-25	115J09	
621	YD60074	AXS	181	2010-05-11	2028-03-25	115J09	
622	YD60075	AXS	182	2010-05-11	2028-03-25	115J09	
623	YD60076	AXS	183	2010-05-11	2028-03-25	115J09	
624	YD60077	AXS	184	2010-05-11	2028-03-25	115J09	
625	YD60078	AXS	185	2010-05-11	2028-03-25	115J09	
626	YD60079	AXS	186	2010-05-11	2028-03-25	115J09	
627	YD61120	AXS	187	2010-05-11	2028-03-25	115J10	
628	YD61121	AXS	188	2010-05-11	2028-03-25	115J10	
629	YD61122	AXS	189	2010-05-11	2028-03-25	115J10	
630	YD61123	AXS	190	2010-05-11	2028-03-25	115J10	
631	YD61124	AXS	191	2010-05-11	2028-03-25	115J10	
632	YD61125	AXS	192	2010-05-11	2028-03-25	115J10	
633	YD61126	AXS	193	2010-05-11	2028-03-25	115J10	
634	YD61127	AXS	194	2010-05-11	2028-03-25	115J10	
635	YD61128	AXS	196	2010-05-11	2028-03-25	115J10	
636	YD61129	AXS	195	2010-05-11	2028-03-25	115J10	
637	YD61130	AXS	197	2010-05-11	2028-03-25	115J10	
638	YD61131	AXS	198	2010-05-11	2028-03-25	115J10	
639	YD61132	AXS	199	2010-05-11	2028-03-25	115J10	
640	95744	CAT	67	1965-12-05	2031-03-25	115J10	
641	95746	CAT	69	1965-12-05	2031-03-25	115J10	
642	Y 35194	MOUSE	3	1969-06-04	2031-03-25	115J10	
643	Y 35196	MOUSE	5	1969-06-04	2031-03-25	115J10	
644	Y 35198	MOUSE	7	1969-06-04	2031-03-25	115J10	
645	Y 35199	MOUSE	8	1969-06-04	2031-03-25	115J10	
646	Y 35200	MOUSE	9	1969-06-04	2031-03-25	115J10	
647	Y 35201	MOUSE	10	1969-06-04	2031-03-25	115J10	
648	Y 35202	MOUSE	11	1969-06-04	2031-03-25	115J10	
649	Y 35203	MOUSE	12	1969-06-04	2031-03-25	115J10	
650	Y 35204	MOUSE	13	1969-06-04	2031-03-25	115J10	
651	Y 35205	MOUSE	14	1969-06-04	2031-03-25	115J10	
652	Y 35206	MOUSE	15	1969-06-04	2031-03-25	115J10	
653	Y 35207	MOUSE	16	1969-06-04	2031-03-25	115J10	
654	Y 35483	MOUSE	89	1969-06-22	2031-03-25	115J10	
655	Y 35491	MOUSE	97	1969-06-22	2031-03-25	115J10	
656	Y 35492	MOUSE	98	1969-06-22	2031-03-25	115J10	
657	Y 35517	MOUSE	123	1969-06-22	2031-03-25	115J10	
658	Y 35518	MOUSE	124	1969-06-22	2031-03-25	115J10	
659	Y 35519	MOUSE	125	1969-06-22	2031-03-25	115J10	
660	Y 35520	MOUSE	126	1969-06-22	2031-03-25	115J10	
661	Y 35521	MOUSE	127	1969-06-22	2031-03-25	115J10	
662	Y 35522	MOUSE	128	1969-06-22	2031-03-25	115J10	
663	YB36618	CAS	31	1991-11-28	2031-03-25	115J10	
664	YB36619	CAS	32	1991-11-28	2031-03-25	115J10	
665	YB36620	CAS	33	1991-11-28	2031-03-25	115J15	
666	YB36621	CAS	34	1991-11-28	2031-03-25	115J15	
667	YB36622	CAS	35	1991-11-28	2031-03-25	115J15	
668	YB36623	CAS	36	1991-11-28	2031-03-25	115J15	
669	YB37242	E	23	1992-09-01	2031-03-25	115J15	
670	YB37243	E	24	1992-09-01	2031-03-25	115J15	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
671	YB37244	E	25	1992-09-01	2031-03-25	115J15	
672	YB37246	E	27	1992-09-01	2031-03-25	115J10	
673	YB37247	E	28	1992-09-01	2031-03-25	115J10	
674	YB37248	E	29	1992-09-01	2031-03-25	115J15	
675	YB37249	E	30	1992-09-01	2031-03-25	115J15	
676	YB37250	E	31	1992-09-01	2031-03-25	115J15	
677	YB37251	E	32	1992-09-01	2031-03-25	115J15	
678	YB37278	F	27	1992-08-30	2031-03-25	115J10	
679	YB37279	F	28	1992-08-30	2031-03-25	115J10	
680	YB37640	I	1	1992-09-09	2031-03-25	115J10	
681	YB37641	I	2	1992-09-09	2031-03-25	115J10	
682	YB37642	I	3	1992-09-09	2031-03-25	115J10	
683	YB37643	I	4	1992-09-09	2031-03-25	115J10	
684	YB37658	I	19	1992-09-09	2031-03-25	115J10	
685	YB37659	I	20	1992-09-09	2031-03-25	115J10	
686	YC81316	BRIT	1	2008-06-10	2034-03-05	115J15	
687	YC81317	BRIT	2	2008-06-10	2034-03-05	115J15	
688	YC81318	BRIT	3	2008-06-10	2034-03-05	115J15	
689	YC81319	BRIT	4	2008-06-10	2034-03-05	115J15	
690	YC81320	BRIT	5	2008-06-10	2034-03-05	115J15	
691	YC81321	BRIT	6	2008-06-10	2034-03-05	115J15	
692	YC81322	BRIT	7	2008-06-10	2034-03-05	115J15	
693	YC81323	BRIT	8	2008-06-10	2034-03-05	115J15	
694	YC81324	BRIT	9	2008-06-10	2034-03-05	115J15	
695	YC81325	BRIT	10	2008-06-10	2034-03-05	115J15	
696	YC81326	BRIT	11	2008-06-10	2034-03-05	115J15	
697	YC81327	BRIT	12	2008-06-10	2034-03-05	115J15	
698	YC81328	BRIT	13	2008-06-10	2034-03-05	115J15	
699	YC81329	BRIT	14	2008-06-10	2034-03-05	115J15	
700	YC81330	BRIT	15	2008-06-10	2034-03-05	115J15	
701	YC81331	BRIT	16	2008-06-10	2034-03-05	115J15	
702	YC81332	BRIT	17	2008-06-10	2034-03-05	115J15	
703	YC81333	BRIT	18	2008-06-10	2034-03-05	115J15	
704	YC81334	BRIT	19	2008-06-10	2034-03-05	115J15	
705	YC81335	BRIT	20	2008-06-10	2034-03-05	115J15	
706	YC81336	BRIT	21	2008-06-10	2034-03-05	115J15	
707	YC81337	BRIT	22	2008-06-10	2034-03-05	115J15	
708	YC81338	BRIT	23	2008-06-10	2034-03-05	115J15	
709	YC81339	BRIT	24	2008-06-10	2034-03-05	115J15	
710	YC81340	BRIT	25	2008-06-10	2034-03-05	115J15	
711	YC81341	BRIT	26	2008-06-10	2034-03-05	115J15	
712	YC81342	BRIT	27	2008-06-10	2034-03-05	115J15	
713	YC81343	BRIT	28	2008-06-10	2034-03-05	115J15	
714	YC81344	BRIT	29	2008-06-10	2034-03-05	115J15	
715	YC81345	BRIT	30	2008-06-10	2034-03-05	115J15	
716	YC81346	BRIT	31	2008-06-10	2034-03-05	115J15	
717	YC81347	BRIT	32	2008-06-10	2034-03-05	115J15	
718	YC81348	BRIT	33	2008-06-10	2034-03-05	115J15	
719	YC81349	BRIT	34	2008-06-10	2034-03-05	115J15	
720	YC81350	BRIT	35	2008-06-10	2034-03-05	115J15	
721	YC81351	BRIT	36	2008-06-10	2034-03-05	115J15	
722	YC81352	BRIT	37	2008-06-10	2034-03-05	115J15	
723	YC81353	BRIT	38	2008-06-10	2034-03-05	115J15	
724	YC81354	BRIT	39	2008-06-10	2034-03-05	115J15	
725	YC81355	BRIT	40	2008-06-10	2034-03-05	115J15	
726	YC81356	BRIT	41	2008-06-10	2034-03-05	115J15	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
727	YC81357	BRIT	42	2008-06-10	2034-03-05	115J15	
728	YC81358	BRIT	43	2008-06-10	2034-03-05	115J15	
729	YC81359	BRIT	44	2008-06-10	2034-03-05	115J15	
730	YC81360	BRIT	45	2008-06-10	2034-03-05	115J15	
731	YC81361	BRIT	46	2008-06-10	2034-03-05	115J15	
732	YC81362	BRIT	47	2008-06-10	2034-03-05	115J15	
733	YC81363	BRIT	48	2008-06-10	2034-03-05	115J15	
734	YC81364	BRIT	49	2008-06-10	2034-03-05	115J15	
735	YC81365	BRIT	50	2008-06-10	2034-03-05	115J15	
736	YC81366	BRIT	51	2008-06-10	2034-03-05	115J15	
737	YC81367	BRIT	52	2008-06-10	2034-03-05	115J15	
738	YC81368	BRIT	53	2008-06-10	2034-03-05	115J15	
739	YC81369	BRIT	54	2008-06-10	2034-03-05	115J15	
740	YC81370	BRIT	55	2008-06-10	2034-03-05	115J15	
741	YC81371	BRIT	56	2008-06-10	2034-03-05	115J15	
742	YC81372	BRIT	57	2008-06-10	2034-03-05	115J15	
743	YC81373	BRIT	58	2008-06-10	2034-03-05	115J15	
744	YC81374	BRIT	59	2008-06-10	2034-03-05	115J15	
745	YC81375	BRIT	60	2008-06-10	2034-03-05	115J15	
746	YC81376	BRIT	61	2008-06-10	2034-03-05	115J15	
747	YC81377	BRIT	62	2008-06-10	2034-03-05	115J15	
748	YC81378	BRIT	63	2008-06-10	2034-03-05	115J15	
749	YC81379	CC	1	2008-06-12	2034-03-05	115J10	
750	YC81380	CC	2	2008-06-12	2034-03-05	115J10	
751	YC81381	CC	3	2008-06-12	2034-03-05	115J10	
752	YC81382	CC	4	2008-06-12	2034-03-05	115J10	
753	YC81383	CC	5	2008-06-12	2034-03-05	115J10	
754	YC81384	CC	6	2008-06-12	2034-03-05	115J10	
755	YC81385	CC	7	2008-06-11	2034-03-05	115J10	
756	YC81386	CC	8	2008-06-11	2034-03-05	115J10	
757	YC81387	CC	9	2008-06-11	2034-03-05	115J10	
758	YC81388	CC	10	2008-06-11	2034-03-05	115J10	
759	YC81389	CC	11	2008-06-11	2034-03-05	115J10	
760	YC81390	CC	12	2008-06-11	2034-03-05	115J10	
761	YC81391	CC	13	2008-06-11	2034-03-05	115J10	
762	YC81392	CC	14	2008-06-11	2034-03-05	115J10	
763	YC81393	CC	15	2008-06-11	2034-03-05	115J10	
764	YC81394	CC	16	2008-06-11	2034-03-05	115J10	
765	YC81395	CC	17	2008-06-11	2034-03-05	115J10	
766	YC81396	CC	18	2008-06-11	2034-03-05	115J10	
767	YC81397	CC	19	2008-06-11	2034-03-05	115J10	
768	YC81398	CC	20	2008-06-11	2034-03-05	115J10	
769	YC81399	CC	21	2008-06-12	2034-03-05	115J10	
770	YC81400	CC	22	2008-06-12	2034-03-05	115J10	
771	YC81401	CC	23	2008-06-12	2034-03-05	115J10	
772	YC81402	CC	24	2008-06-12	2034-03-05	115J10	
773	YC81403	CC	25	2008-06-12	2034-03-05	115J10	
774	YC81404	CC	26	2008-06-12	2034-03-05	115J10	
775	YC81405	CC	27	2008-06-12	2034-03-05	115J10	
776	YC81406	CC	28	2008-06-12	2034-03-05	115J10	
777	YC81407	CC	29	2008-06-12	2034-03-05	115J10	
778	YC81408	CC	30	2008-06-11	2034-03-05	115J10	
779	YC81409	CC	31	2008-06-11	2034-03-05	115J10	
780	YC81410	CC	32	2008-06-11	2034-03-05	115J10	
781	YC81411	CC	33	2008-06-11	2034-03-05	115J10	
782	YC81412	CC	34	2008-06-11	2034-03-05	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
783	YC81413	CC	35	2008-06-11	2034-03-05	115J10	
784	YC81414	CC	36	2008-06-11	2034-03-05	115J10	
785	YC81415	CC	37	2008-06-11	2034-03-05	115J10	
786	YC81416	CC	38	2008-06-11	2034-03-05	115J10	
787	YC81417	CC	39	2008-06-11	2034-03-05	115J10	
788	YC81418	CC	40	2008-06-12	2034-03-05	115J10	
789	YC81419	CC	41	2008-06-12	2034-03-05	115J10	
790	YC81420	CC	42	2008-06-12	2034-03-05	115J10	
791	YC81421	CC	43	2008-06-12	2034-03-05	115J10	
792	YC81422	CC	44	2008-06-12	2034-03-05	115J10	
793	YC81423	CC	45	2008-06-12	2034-03-05	115J10	
794	YC81424	CC	46	2008-06-12	2034-03-05	115J10	
795	YC81425	CC	47	2008-06-12	2034-03-05	115J10	
796	YC81426	CC	48	2008-06-12	2034-03-05	115J10	
797	YC81427	CC	49	2008-06-12	2034-03-05	115J10	
798	YC81428	CC	50	2008-06-12	2034-03-05	115J10	
799	YC81429	CC	51	2008-06-12	2034-03-05	115J10	
800	YC81430	CC	52	2008-06-12	2034-03-05	115J10	
801	YC81431	CC	53	2008-06-12	2034-03-05	115J10	
802	YC81432	CC	54	2008-06-12	2034-03-05	115J10	
803	YC81433	CC	55	2008-06-12	2034-03-05	115J10	
804	YC81434	CC	56	2008-06-12	2034-03-05	115J10	
805	YC81435	CC	57	2008-06-11	2034-03-05	115J10	
806	YC81436	CC	58	2008-06-11	2034-03-05	115J10	
807	YC81437	CC	59	2008-06-11	2034-03-05	115J10	
808	YC81438	CC	60	2008-06-11	2034-03-05	115J10	
809	YC81439	CC	61	2008-06-11	2034-03-05	115J10	
810	YC81440	CC	62	2008-06-11	2034-03-05	115J10	
811	YC81441	CC	63	2008-06-12	2034-03-05	115J10	
812	YC81442	CC	64	2008-06-12	2034-03-05	115J10	
813	YC81443	CC	65	2008-06-12	2034-03-05	115J10	
814	YC81444	CC	66	2008-06-12	2034-03-05	115J10	
815	YC81445	CC	67	2008-06-12	2034-03-05	115J10	
816	YC81446	CC	68	2008-06-12	2034-03-05	115J10	
817	YC81447	CC	69	2008-06-12	2034-03-05	115J10	
818	YC81448	CC	70	2008-06-12	2034-03-05	115J10	
819	YC81449	CC	71	2008-06-12	2034-03-05	115J10	
820	YC81450	CC	72	2008-06-12	2034-03-05	115J10	
821	YC81451	CC	73	2008-06-12	2034-03-05	115J10	
822	YC81452	CC	74	2008-06-12	2034-03-05	115J10	
823	YC81453	CC	75	2008-06-12	2034-03-05	115J10	
824	YC81454	CC	76	2008-06-12	2034-03-05	115J10	
825	YC81455	CC	77	2008-06-12	2034-03-05	115J10	
826	YC81456	CC	78	2008-06-12	2034-03-05	115J10	
827	YC81457	CC	79	2008-06-12	2034-03-05	115J10	
828	YC81458	CC	80	2008-06-11	2034-03-05	115J10	
829	YC81459	CC	81	2008-06-11	2034-03-05	115J10	
830	YC81460	CC	82	2008-06-11	2034-03-05	115J10	
831	YC81461	CC	83	2008-06-13	2034-03-05	115J10	
832	YC81462	CC	84	2008-06-13	2034-03-05	115J10	
833	YC81463	CC	85	2008-06-13	2034-03-05	115J10	
834	YC81464	CC	86	2008-06-13	2034-03-05	115J10	
835	YC81465	CC	87	2008-06-13	2034-03-05	115J10	
836	YC81466	CC	88	2008-06-13	2034-03-05	115J10	
837	YC81467	CC	89	2008-06-13	2034-03-05	115J10	
838	YC81468	CC	90	2008-06-13	2034-03-05	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
839	YC81469	CC	91	2008-06-13	2034-03-05	115J10	
840	YC81470	CC	92	2008-06-13	2034-03-05	115J10	
841	YC81471	CC	93	2008-06-13	2034-03-05	115J10	
842	YC81472	CC	94	2008-06-13	2034-03-05	115J10	
843	Y 10693	JOE	89	1966-09-24	2036-03-05	115J10	
844	Y 10694	JOE	90	1966-09-24	2036-03-05	115J10	
845	Y 10695	JOE	91	1966-09-24	2036-03-05	115J10	
846	Y 10696	JOE	92	1966-09-24	2036-03-05	115J10	
847	Y 10697	JOE	93	1966-09-24	2036-03-05	115J10	
848	Y 10698	JOE	94	1966-09-24	2036-03-05	115J10	
849	Y 10699	JOE	95	1966-09-24	2036-03-05	115J10	
850	Y 10700	JOE	96	1966-09-24	2036-03-05	115J10	
851	Y 10702	JOE	98	1966-09-24	2036-03-05	115J10	
852	Y 10703	JOE	99	1966-09-24	2036-03-05	115J10	
853	Y 10705	JOE	101	1966-09-24	2036-03-05	115J10	
854	Y 10706	JOE	102	1966-09-24	2036-03-05	115J10	
855	Y 10707	JOE	103	1966-09-24	2036-03-05	115J15	
856	Y 10708	JOE	104	1966-09-24	2036-03-05	115J15	
857	Y 35192	MOUSE	1	1969-06-04	2036-03-05	115J10	
858	Y 35193	MOUSE	2	1969-06-04	2036-03-05	115J10	
859	Y 51850	JOE	91	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
860	Y 51851	JOE	92	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
861	Y 51852	JOE	93	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
862	Y 51853	JOE	94	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
863	Y 51854	JOE	95	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
864	Y 51855	JOE	96	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
865	YC64893	VIK	1	2007-05-27	2036-03-05	115J10	
866	YC64894	VIK	2	2007-05-27	2036-03-05	115J10	
867	YC64895	VIK	3	2007-05-27	2036-03-05	115J10	
868	YC64896	VIK	4	2007-05-27	2036-03-05	115J10	
869	YC64897	VIK	5	2007-05-27	2036-03-05	115J10	
870	YC64898	VIK	6	2007-05-27	2036-03-05	115J10	
871	YC64899	VIK	7	2007-05-27	2036-03-05	115J10	
872	YC64900	VIK	8	2007-05-27	2036-03-05	115J10	
873	YC64901	VIK	9	2007-05-27	2036-03-05	115J10	
874	YC64902	VIK	10	2007-05-27	2036-03-05	115J10	
875	YC64903	VIK	11	2007-05-27	2036-03-05	115J10	
876	YC64904	VIK	12	2007-05-27	2036-03-05	115J10	
877	YC64905	VIK	13	2007-05-27	2036-03-05	115J10	
878	YC64906	VIK	14	2007-05-27	2036-03-05	115J10	
879	YC64907	VIK	15	2007-05-27	2036-03-05	115J10	
880	YC64908	VIK	16	2007-05-27	2036-03-05	115J10	
881	YC64909	VIK	17	2007-05-27	2036-03-05	115J10	
882	YC64910	VIK	18	2007-05-28	2036-03-05	115J10	
883	YC64911	VIK	19	2007-05-28	2036-03-05	115J10	
884	YC64912	VIK	20	2007-05-28	2036-03-05	115J10	
885	YC64913	VIK	21	2007-05-28	2036-03-05	115J10	
886	YC64914	VIK	22	2007-05-25	2036-03-05	115J10	
887	YC64915	VIK	23	2007-05-25	2036-03-05	115J10	
888	YC64916	VIK	24	2007-05-25	2036-03-05	115J10	
889	YC64917	VIK	25	2007-05-25	2036-03-05	115J10	
890	YC64918	VIK	26	2007-05-25	2036-03-05	115J10	
891	YC64919	VIK	27	2007-05-25	2036-03-05	115J10	
892	YC64920	VIK	28	2007-05-25	2036-03-05	115J10	
893	YC64921	VIK	29	2007-05-25	2036-03-05	115J10	
894	YC64922	VIK	30	2007-05-25	2036-03-05	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
895	YC64923	VIK	31	2007-05-25	2036-03-05	115J10	
896	YC64924	VIK	32	2007-05-25	2036-03-05	115J10	
897	YC64925	VIK	33	2007-05-25	2036-03-05	115J10	
898	YC64926	VIK	34	2007-05-25	2036-03-05	115J10	
899	YC64927	VIK	35	2007-05-25	2036-03-05	115J10	
900	YC64928	VIK	36	2007-05-25	2036-03-05	115J10	
901	YC64929	VIK	37	2007-05-25	2036-03-05	115J10	
902	YC64930	VIK	38	2007-05-25	2036-03-05	115J10	
903	YC64931	VIK	39	2007-05-25	2036-03-05	115J10	
904	YC64932	VIK	40	2007-05-25	2036-03-05	115J10	
905	YC64933	VIK	41	2007-05-31	2036-03-05	115J10	
906	YC64934	VIK	42	2007-05-31	2036-03-05	115J10	
907	YC64935	VIK	43	2007-06-05	2036-03-05	115J10	
908	YC64936	VIK	44	2007-06-05	2036-03-05	115J10	
909	YC64937	VIK	45	2007-06-05	2036-03-05	115J10	
910	YC64938	VIK	46	2007-06-05	2036-03-05	115J10	
911	YC64939	VIK	47	2007-06-05	2036-03-05	115J10	
912	YC64940	VIK	48	2007-06-05	2036-03-05	115J10	
913	YC64941	VIK	49	2007-05-25	2036-03-05	115J10	
914	YC64942	VIK	50	2007-05-25	2036-03-05	115J10	
915	YC64943	VIK	51	2007-05-25	2036-03-05	115J10	
916	YC64944	VIK	52	2007-05-25	2036-03-05	115J10	
917	YC64945	VIK	53	2007-05-25	2036-03-05	115J10	
918	YC64946	VIK	54	2007-05-25	2036-03-05	115J10	
919	YC64947	VIK	55	2007-05-25	2036-03-05	115J10	
920	YC64948	VIK	56	2007-05-24	2036-03-05	115J10	
921	YC64949	VIK	57	2007-05-24	2036-03-05	115J10	
922	YC64950	VIK	58	2007-05-24	2036-03-05	115J10	
923	YC64951	VIK	59	2007-05-24	2036-03-05	115J10	
924	YC64952	VIK	60	2007-05-24	2036-03-05	115J10	
925	YC64953	VIK	61	2007-05-24	2036-03-05	115J10	
926	YC64954	VIK	62	2007-05-24	2036-03-05	115J10	
927	YC64955	VIK	63	2007-05-24	2036-03-05	115J10	
928	YC64956	VIK	64	2007-05-24	2036-03-05	115J10	
929	YC64958	VIK	66	2007-05-24	2036-03-05	115J10	
930	YC64959	VIK	67	2007-05-24	2036-03-05	115J10	
931	YC64960	VIK	68	2007-05-24	2036-03-05	115J10	
932	YC64961	VIK	69	2007-05-24	2036-03-05	115J10	
933	YC64962	VIK	70	2007-05-24	2036-03-05	115J10	
934	YC64963	VIK	71	2007-05-24	2036-03-05	115J10	
935	YC64964	VIK	72	2007-05-24	2036-03-05	115J10	
936	YC64965	VIK	73	2007-05-24	2036-03-05	115J10	
937	YC64966	VIK	74	2007-05-24	2036-03-05	115J10	
938	YC64967	VIK	75	2007-05-24	2036-03-05	115J10	
939	YC64968	VIK	76	2007-05-24	2036-03-05	115J10	
940	YC64969	VIK	77	2007-05-24	2036-03-05	115J10	
941	YC64970	VIK	78	2007-05-24	2036-03-05	115J10	
942	YC64971	VIK	79	2007-05-24	2036-03-05	115J10	
943	YC64972	VIK	80	2007-05-27	2036-03-05	115J10	
944	YC64973	VIK	81	2007-05-25	2036-03-05	115J10	
945	YC64974	VIK	82	2007-05-26	2036-03-05	115J10	
946	YC64975	VIK	83	2007-05-26	2036-03-05	115J10	
947	YC64976	VIK	84	2007-05-26	2036-03-05	115J10	
948	YC64977	VIK	85	2007-05-26	2036-03-05	115J10	
949	YC64978	VIK	86	2007-05-26	2036-03-05	115J10	
950	YC64979	VIK	87	2007-05-26	2036-03-05	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
951	YC64980	VIK	88	2007-05-26	2036-03-05	115J10	
952	YC64981	VIK	89	2007-05-26	2036-03-05	115J10	
953	YC64982	VIK	90	2007-05-26	2036-03-05	115J10	
954	YC64983	VIK	91	2007-05-26	2036-03-05	115J10	
955	YC64984	VIK	92	2007-05-26	2036-03-05	115J10	
956	YC64985	VIK	93	2007-05-26	2036-03-05	115J10	
957	YC64986	VIK	94	2007-05-26	2036-03-05	115J10	
958	YC64987	VIK	95	2007-05-26	2036-03-05	115J10	
959	YC64988	VIK	96	2007-05-28	2036-03-05	115J10	
960	YC64989	VIK	97	2007-05-28	2036-03-05	115J10	
961	YC64990	VIK	98	2007-05-28	2036-03-05	115J10	
962	YC64991	VIK	99	2007-05-28	2036-03-05	115J10	
963	YC64992	VIK	100	2007-05-28	2036-03-05	115J10	
964	YC64993	VIK	101	2007-05-28	2036-03-05	115J10	
965	YC64994	VIK	102	2007-05-28	2036-03-05	115J10	
966	YC64995	VIK	103	2007-05-28	2036-03-05	115J10	
967	YC64996	VIK	104	2007-05-26	2036-03-05	115J10	
968	YC64997	VIK	105	2007-05-26	2036-03-05	115J10	
969	YC64998	VIK	106	2007-05-26	2036-03-05	115J10	
970	YC64999	VIK	107	2007-05-26	2036-03-05	115J10	
971	YC65000	VIK	108	2007-05-26	2036-03-05	115J10	
972	YC65001	VIK	109	2007-05-26	2036-03-05	115J10	
973	YC65002	VIK	110	2007-05-26	2036-03-05	115J10	
974	YC65003	VIK	111	2007-05-26	2036-03-05	115J10	
975	YC65004	VIK	112	2007-05-26	2036-03-05	115J10	
976	YC65005	VIK	113	2007-05-26	2036-03-05	115J10	
977	YC65006	VIK	114	2007-05-26	2036-03-05	115J10	
978	YC65007	VIK	115	2007-05-26	2036-03-05	115J10	
979	YC65008	VIK	116	2007-05-26	2036-03-05	115J10	
980	YC65009	VIK	117	2007-05-26	2036-03-05	115J10	
981	YC65010	VIK	118	2007-05-26	2036-03-05	115J15	
982	YC65011	VIK	119	2007-05-26	2036-03-05	115J15	
983	YC65012	VIK	120	2007-05-26	2036-03-05	115J15	
984	YC65013	VIK	121	2007-05-26	2036-03-05	115J15	
985	YC65014	VIK	122	2007-05-26	2036-03-05	115J15	
986	YC65015	VIK	123	2007-05-26	2036-03-05	115J15	
987	YC65016	VIK	124	2007-05-26	2036-03-05	115J15	
988	YC65017	VIK	125	2007-05-26	2036-03-05	115J15	
989	YC65018	VIK	126	2007-05-29	2036-03-05	115J15	
990	YC65019	VIK	127	2007-05-29	2036-03-05	115J15	
991	YC65020	VIK	128	2007-05-29	2036-03-05	115J15	
992	YC65021	VIK	129	2007-05-29	2036-03-05	115J15	
993	YC65022	VIK	130	2007-05-29	2036-03-05	115J15	
994	YC65023	VIK	131	2007-05-29	2036-03-05	115J15	
995	YC65024	VIK	132	2007-05-29	2036-03-05	115J15	
996	YC65025	VIK	133	2007-05-29	2036-03-05	115J15	
997	YC65026	VIK	134	2007-05-29	2036-03-05	115J15	
998	YC65027	VIK	135	2007-05-29	2036-03-05	115J15	
999	YC65028	VIK	136	2007-05-29	2036-03-05	115J15	
1000	YC65029	VIK	137	2007-05-29	2036-03-05	115J15	
1001	YC65030	VIK	138	2007-05-29	2036-03-05	115J15	
1002	YC65031	VIK	139	2007-05-29	2036-03-05	115J15	
1003	YC65032	VIK	140	2007-05-29	2036-03-05	115J15	
1004	YC65033	VIK	141	2007-05-29	2036-03-05	115J15	
1005	YC65034	VIK	142	2007-05-29	2036-03-05	115J15	
1006	YC65035	VIK	143	2007-05-29	2036-03-05	115J15	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1007	YC65036	VIK	144	2007-05-29	2036-03-05	115J15	
1008	YC65037	VIK	145	2007-05-29	2036-03-05	115J15	
1009	YC65038	VIK	146	2007-05-29	2036-03-05	115J15	
1010	YC65039	VIK	147	2007-05-29	2036-03-05	115J15	
1011	YC65040	VIK	148	2007-05-29	2036-03-05	115J15	
1012	YC65041	VIK	149	2007-05-29	2036-03-05	115J15	
1013	YC65042	VIK	150	2007-05-29	2036-03-05	115J15	
1014	YC65043	VIK	151	2007-05-29	2036-03-05	115J15	
1015	YC65044	VIK	152	2007-05-29	2036-03-05	115J15	
1016	YC65045	VIK	153	2007-05-29	2036-03-05	115J15	
1017	YC65046	VIK	154	2007-05-29	2036-03-05	115J15	
1018	YC65047	VIK	155	2007-05-29	2036-03-05	115J15	
1019	YC65048	VIK	156	2007-05-29	2036-03-05	115J15	
1020	YC65049	VIK	157	2007-05-29	2036-03-05	115J15	
1021	YC65050	VIK	158	2007-05-29	2036-03-05	115J15	
1022	YC65051	VIK	159	2007-05-29	2036-03-05	115J15	
1023	YC65052	VIK	160	2007-05-29	2036-03-05	115J15	
1024	YC65053	VIK	161	2007-05-29	2036-03-05	115J15	
1025	YC65054	VIK	162	2007-05-29	2036-03-05	115J15	
1026	YC65055	VIK	163	2007-05-29	2036-03-05	115J15	
1027	YC65056	VIK	164	2007-05-29	2036-03-05	115J15	
1028	YC65057	VIK	165	2007-05-29	2036-03-05	115J15	
1029	YC65058	VIK	166	2007-05-29	2036-03-05	115J15	
1030	YC65059	VIK	167	2007-05-29	2036-03-05	115J15	
1031	YC65060	VIK	168	2007-05-29	2036-03-05	115J15	
1032	YC65061	VIK	169	2007-05-29	2036-03-05	115J15	
1033	YC65062	VIK	170	2007-05-30	2036-03-05	115J15	
1034	YC65063	VIK	171	2007-05-30	2036-03-05	115J15	
1035	YC65064	VIK	172	2007-05-30	2036-03-05	115J15	
1036	YC65065	VIK	173	2007-05-30	2036-03-05	115J15	
1037	YC65066	VIK	174	2007-05-30	2036-03-05	115J15	
1038	YC65067	VIK	175	2007-05-30	2036-03-05	115J15	
1039	YC65068	VIK	176	2007-05-30	2036-03-05	115J15	
1040	YC65069	VIK	177	2007-05-30	2036-03-05	115J15	
1041	YC65070	VIK	178	2007-05-30	2036-03-05	115J15	
1042	YC65071	VIK	179	2007-05-30	2036-03-05	115J15	
1043	YC65072	VIK	180	2007-05-30	2036-03-05	115J15	
1044	YC65073	VIK	181	2007-05-30	2036-03-05	115J15	
1045	YC65074	VIK	182	2007-05-30	2036-03-05	115J15	
1046	YC65075	VIK	183	2007-05-30	2036-03-05	115J15	
1047	YC65076	VIK	184	2007-05-28	2036-03-05	115J15	
1048	YC65077	VIK	185	2007-05-28	2036-03-05	115J15	
1049	YC65078	VIK	186	2007-05-28	2036-03-05	115J15	
1050	YC65079	VIK	187	2007-05-29	2036-03-05	115J15	
1051	YC65080	VIK	188	2007-05-29	2036-03-05	115J15	
1052	YC64957	VIK	65	2007-05-26	2036-03-05	115J10	
1053	4252	HELICOPTER		1943-09-04	2036-03-25	115J10	
1054	56979	#1 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1055	56980	#3 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1056	56981	#5 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1057	56983	#1 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1058	56984	#3 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1059	56985	#5 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1060	56987	#2 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1061	56988	#6 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1062	56990	#2 AIRPORT GROU		1947-08-07	2036-03-25	115J10	

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1063	56991	#4 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1064	56992	#6 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1065	56993	#8 AIRPORT GROU		1947-07-07	2036-03-25	115J10	
1066	92201	CAT	1	1965-06-29	2036-03-25	115J10	
1067	92202	CAT	2	1965-06-29	2036-03-25	115J10	
1068	92203	CAT	3	1965-06-29	2036-03-25	115J10	
1069	92204	CAT	4	1965-06-29	2036-03-25	115J10	
1070	92205	CAT	5	1965-06-29	2036-03-25	115J10	
1071	92206	CAT	6	1965-06-29	2036-03-25	115J10	
1072	92207	CAT	7	1965-06-29	2036-03-25	115J10	
1073	92208	CAT	8	1965-06-29	2036-03-25	115J10	
1074	92209	CAT	9	1965-06-29	2036-03-25	115J10	
1075	92210	CAT	10	1965-06-29	2036-03-25	115J10	
1076	92211	CAT	11	1965-06-29	2036-03-25	115J10	
1077	92212	CAT	12	1965-06-29	2036-03-25	115J10	
1078	92213	CAT	13	1965-06-29	2036-03-25	115J10	
1079	92214	CAT	14	1965-06-29	2036-03-25	115J10	
1080	92215	CAT	15	1965-06-30	2036-03-25	115J10	
1081	92216	CAT	16	1965-06-30	2036-03-25	115J10	
1082	92217	CAT	17	1965-06-30	2036-03-25	115J10	
1083	92218	CAT	18	1965-06-30	2036-03-25	115J10	
1084	92219	CAT	19	1965-06-30	2036-03-25	115J10	
1085	92220	CAT	20	1965-06-30	2036-03-25	115J10	
1086	92221	CAT	21	1965-06-30	2036-03-25	115J10	
1087	92222	CAT	22	1965-06-30	2036-03-25	115J10	
1088	92764	CAT	23	1965-09-10	2036-03-25	115J10	
1089	92765	CAT	24	1965-09-10	2036-03-25	115J10	
1090	92766	CAT	25	1965-09-10	2036-03-25	115J10	
1091	92776	CAT	35	1965-09-11	2036-03-25	115J10	
1092	92777	CAT	36	1965-09-11	2036-03-25	115J10	
1093	92778	CAT	37	1965-09-11	2036-03-25	115J10	
1094	92779	CAT	38	1965-09-11	2036-03-25	115J10	
1095	92780	CAT	39	1965-09-12	2036-03-25	115J10	
1096	92781	CAT	40	1965-09-12	2036-03-25	115J10	
1097	92782	CAT	41	1965-09-12	2036-03-25	115J10	
1098	92783	CAT	42	1965-09-12	2036-03-25	115J10	
1099	95724	CAT	47	1965-12-02	2036-03-25	115J10	
1100	95725	CAT	48	1965-12-02	2036-03-25	115J10	
1101	95726	CAT	49	1965-12-02	2036-03-25	115J10	
1102	95727	CAT	50	1965-12-02	2036-03-25	115J10	
1103	95728	CAT	51	1965-12-02	2036-03-25	115J10	
1104	95729	CAT	52	1965-12-02	2036-03-25	115J10	
1105	95730	CAT	53	1965-12-02	2036-03-25	115J10	
1106	95731	CAT	54	1965-12-02	2036-03-25	115J10	
1107	95732	CAT	55	1965-12-02	2036-03-25	115J15	
1108	95733	CAT	56	1965-12-02	2036-03-25	115J15	
1109	95734	CAT	57	1965-12-02	2036-03-25	115J10	
1110	95735	CAT	58	1965-12-02	2036-03-25	115J10	
1111	95736	CAT	59	1965-12-02	2036-03-25	115J10	
1112	95737	CAT	60	1965-12-02	2036-03-25	115J10	
1113	95738	CAT	61	1965-12-02	2036-03-25	115J10	
1114	95739	CAT	62	1965-12-02	2036-03-25	115J10	
1115	Y 10701	JOE	97	1966-09-24	2036-03-25	115J10	
1116	Y 10704	JOE	100	1966-09-24	2036-03-25	115J10	
1117	Y 35582	MOUSE	161	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1118	Y 35583	MOUSE	162	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)

List of Casino Quartz Claims

District: Whitehorse
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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1119	Y 35584	MOUSE	163	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1120	Y 35585	LOST FR.	1	1969-06-25	2036-03-25	115J10	
1121	Y 35586	LOST FR.	2	1969-06-25	2036-03-25	115J10	
1122	Y 35587	LOST FR.	3	1969-06-25	2036-03-25	115J10	
1123	Y 36686	CAT	22	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1124	Y 36687	CAT	47	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1125	Y 36688	CAT	48	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1126	Y 36690	CAT	62	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1127	Y 39601	CAT	3	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1128	Y 39602	CAT	4	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1129	Y 39603	CAT	23	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1130	Y 51846	CAT	1	1970-03-29	2036-03-25	115J10	Partial Quartz fraction (<25 acres)
1131	Y 51847	CAT	2	1970-03-29	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1132	Y 51849	CAT	26	1970-03-30	2036-03-25	115J10	Partial Quartz fraction (<25 acres)
1133	YB37280	F	29	1992-08-30	2036-03-25	115J10	
1134	YB37282	F	31	1992-08-30	2036-03-25	115J10	
1135	YB37284	F	33	1992-08-30	2036-03-25	115J10	
1136	Y 36689	CAT	57	1969-08-12	2036-06-05	115J10	Full Quartz fraction (25+ acres)