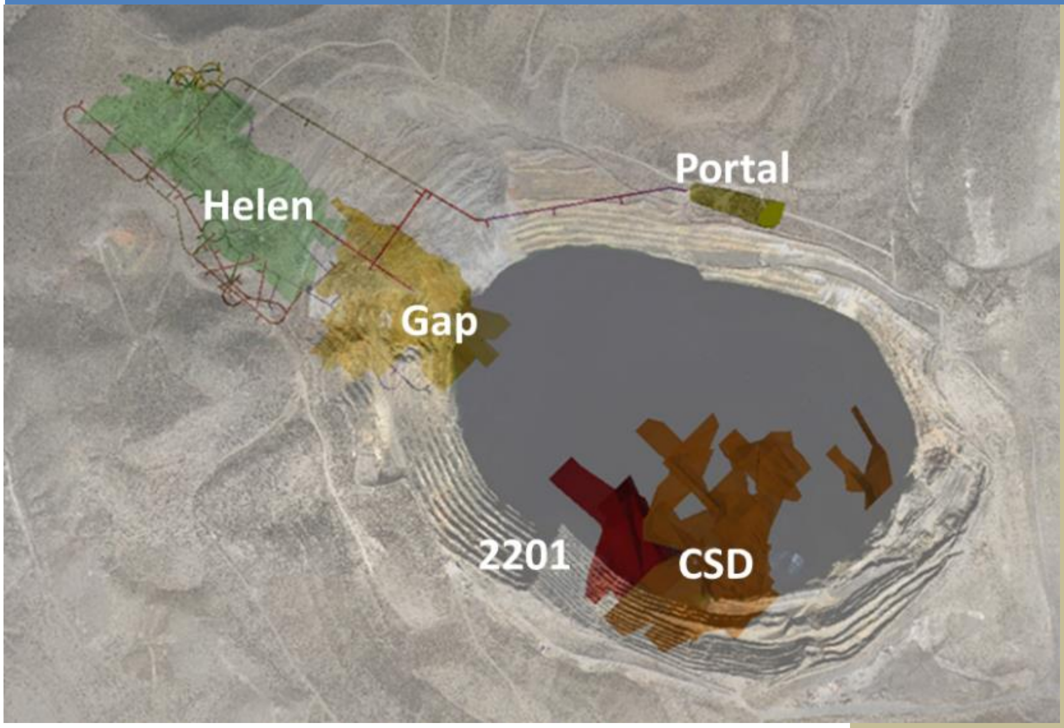
PREMIER
Gold Mines Limited

Preliminary Economic Assessment for the Cove Project, Lander County, Nevada



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The undersigned prepared this Technical Report (TR) report, titled: Preliminary Economic Assessment for the Cove Project, Lander County, Nevada, dated the 29th day of June 2018, with an effective date of March 31, 2018, in support of the public disclosure of Mineral Resource and Mineral Reserve estimates for the Cove Project. The format and content of the Technical Report have been prepared in accordance with Form 43-101F1 of National Instrument 43-101 – Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators.

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Table of Contents

Date and Signature Page ... iii

Table of Contents ... v

List of Tables ... xi

List of Figures ... xiv

List of Abbreviations ... xvi

1. Summary ... 17
   - 1.1. Introduction ... 17
   - 1.2. Property Description ... 18
   - 1.3. Geology and Mineral Resource ... 18
   - 1.4. Metallurgical Testing and Processing ... 20
   - 1.5. Mining, Infrastructure, and Project Schedule ... 23
   - 1.6. Economic Analysis ... 24
   - 1.7. Conclusions ... 26
   - 1.8. Recommendations ... 27
2. Introduction ... 30
   - 2.1. Terms of Reference and Purpose of this Technical Report ... 30
   - 2.2. Qualification of the Authors ... 30
   - 2.3. Sources of Information ... 31
   - 2.4. Units of Measure ... 31
   - 2.5. Coordinate Datum ... 31
3. Reliance on Other Experts ... 39
4. Property Description and Location ... 40
   - 4.1. Property Description ... 40
   - 4.2. Status of Mineral Titles ... 42
   - 4.3. Environmental Liabilities ... 46
   - 4.4. Permits/Licenses ... 47
5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography ... 48
   - 5.1. Accessibility ... 48

5.2. Climate ... 48
5.3. Local Resources ... 48
5.4. Infrastructure ... 48
5.5. Physiography ... 49
6. History ... 51
6.1. Previous Owners ... 51
6.2. Historic Exploration ... 51
6.3. Historic Resource Estimates ... 52
6.4. Historic Mining ... 53
7. Geologic Setting and Mineralization ... 55
7.1. Regional Geology ... 55
7.2. Local Geology ... 56
7.3. Structural Geology ... 62
7.4. Mineralization Controls ... 63
7.5. Post Mineral Faulting ... 64
7.6. Mineralization ... 64
8. Deposit Types ... 67
9. Exploration ... 68
10. Drilling ... 70
10.1. Historic Drilling Methodology ... 76
10.2. Current Drilling Methodology ... 77
10.2.1. Drill Hole Placement ... 77
10.2.2. Downhole Surveys ... 77
10.2.3. RC Drilling Procedures ... 77
10.2.4. Core Drilling Procedures ... 77
10.3. Sampling Methodology ... 78
10.4. Core Recovery ... 78
11. Sample Preparation, Analysis and Security ... 80
11.1. Pre-2012 ... 80
11.1.1. Sample Preparation Procedures ... 80

11.1.2. Laboratory Analysis Procedures ... 81
11.1.3. Security ... 82
11.2. Premier 2012-2018 ... 82
11.3. Quality Assurance and Quality Control ... 83
11.3.1. Standards and Blanks ... 83
11.3.2. Duplicate Assays ... 87
12. Data Verification ... 89
13. Mineral Processing and Metallurgical Testing ... 91
13.1. Historical Metallurgical Test Work ... 91
13.2. 2017 Metallurgical Test Work – Phase 1 ... 92
13.2.1. Composite Construction ... 93
13.2.2. Head Characterization ... 95
13.2.3. Mineralogy ... 101
13.2.4. Whole Ore Bottle Cyanidation Tests ... 103
13.2.5. Roasting and Calcine Cyanidation Tests ... 104
13.2.6. Pressure Oxidation and Cyanidation of Pressure Oxidation Residues ... 107
13.3. 2017 Metallurgical Test Work – Phase 2 ... 109
13.3.1. Rerun Calcine Diagnostic Leach ... 114
13.3.2. Roasting Optimization Testing ... 115
13.4. Future Metallurgical Testing ... 118
13.5. Conclusions and Recommendations: ... 119
13.5.1. Conclusions: ... 119
13.5.2. Recommendations ... 120
14. Mineral Resource Estimates ... 122
14.1. Introduction ... 122
14.2. Modeling of Lithology and Mineralization ... 127
14.3. Drill Data and Compositing ... 130
14.3.1. Drill Data Set ... 130
14.3.2. Compositing ... 131
14.4. Density ... 132

14.5. Statistics and Variography ... 133
14.6. Grade Capping ... 134
14.7. Block Model ... 135
14.8. Grade Estimation and Resource Classification ... 136
14.9. Mined Depletion and Sterilization ... 138
14.10. Model Validation ... 139
14.10.1. Model Smoothing Checks – Grade Tonnage Curves ... 152
14.11. Mineral Resources ... 155
15. Mineral Reserve Estimates ... 156
16. Mining Methods ... 157
16.1. Mine Development ... 157
16.1.1. Access Development ... 157
16.1.2. Ground Support ... 157
16.1.1. Ventilation and Secondary Egress ... 158
16.1.2. Dewatering ... 160
16.2. Mining Methods ... 160
16.2.1. Drift and Fill ... 161
16.3. Underground Labor ... 161
16.4. Mobile Equipment Fleet ... 162
16.5. Mine Plan ... 163
16.6. Mine Plan Reconciliation ... 167
17. Recovery Methods ... 169
17.1. Resource Processing ... 169
17.1.1. Feed Specifications ... 169
17.2. Projected Gold and Silver Recoveries Used for Metallurgical Zones ... 172
17.3. Composite and Metallurgical Zones ... 172
17.3.1. Payable Content ... 174
17.4. Conclusions and Recommendations: ... 175
17.4.1. Conclusions: ... 175
17.4.2. Recommendations: ... 175

18. Project Infrastructure ... 176
18.1. Dewatering ... 176
18.1.1. History ... 176
18.1.2. Pump Test PW17-101 ... 176
18.1.3. Analytical Dewatering Estimate ... 177
18.1.4. Rapid Infiltration Basins ... 179
18.1.5. Recommendations ... 180
18.2. Electrical Power ... 182
18.3. Mine Facilities ... 183
18.4. Backfill ... 184
19. Market Studies and Contracts ... 186
19.1. Precious Metal Markets ... 186
19.2. Contracts ... 187
19.3. Project Financing ... 187
20. Environmental Studies, Permitting and Social or Community Impact ... 188
20.1. Environmental Studies and Issues ... 188
20.2. Social or Community Impacts ... 188
20.1. Permitting ... 189
20.2. Closure and Reclamation Requirements ... 191
21. Capital and Operating Costs ... 193
21.1. Capital Costs ... 193
21.2. Closure and Reclamation ... 195
21.3. Operating Costs ... 196
21.4. Cutoff Grade ... 197
22. Economic Analysis ... 198
23. Adjacent Properties ... 204
24. Other Relevant Data and Information ... 205
25. Interpretation and Conclusions ... 206
26. Recommendations ... 208
26.1. Risks and Opportunities ... 210

27. Bibliography 212

Certification of Authors 214

List of Tables

Table 1-1 Cove Mineral Resources 19
Table 1-2 Financial Statistics[1] 25
Table 2-1 Qualified Professionals 30
Table 2-2 Units of Measure 31
Table 6-1 Historic Resource and Reserve Estimates 53
Table 6-2 Historic Cove and McCoy Mine Production 1986 through 2006 54
Table 10-1 List of Drilling by Operator 71
Table 10-2 Type of Drilling by Zone 75
Table 11-1 Gold Blank and Standard Summary Statistics 83
Table 12-1 Data Review Summary 90
Table 13-1 Helen Composites 94
Table 13-2 Gap Composites 94
Table 13-3 Helen Composite Precious Metal Assays 95
Table 13-4 Gap Composite Precious Metal Assays 96
Table 13-5 Helen Zone Hg, As, Sulfur Speciation, and Carbon Speciation 96
Table 13-6 Gap Zone Hg, As, Sulfur Speciation, and Carbon Speciation 97
Table 13-7 Helen Zone ICP Multi-Element Analyses 97
Table 13-8 Gap Zone ICP Multi-Element Analyses 98
Table 13-9 Helen Zone Comminution Properties 99
Table 13-10 Gap Zone Comminution Properties 99
Table 13-11 Toll Processing Feed Specifications 100
Table 13-12 QEMSCAN Mineral Compositions for Selected Helen & Gap Composites 102
Table 13-13 Summary Whole Ore Cyanidation Test Results 103
Table 13-14 Initial Batch Roast Test Summary 105
Table 13-15 Initial Direct Calcine Cyanidation Test Summary 106
Table 13-16 Pressure Oxidation Test Data 107
Table 13-17 Direct Cyanidation of POX Residue Test Data 108
Table 13-18 Rerun Roasting Data 110
Table 13-19 Roaster Rerun Calcine Cyanidation Data 111
Table 13-20 POX Rerun Test Data 111
Table 13-21 POX Rerun Residue Cyanidation Data 112
Table 13-22 Summary Rerun Calcine Diagnostic Leach 114
Table 13-23 Optimization Roast Test Conditions 115
Table 13-24 Optimization Roast Data 116
Table 13-25 Optimization Roast Calcine CIL Cyanidation Data 117
Table 14-1 Geology Codes 127
Table 14-2 Identification Codes for 3 g/t Grade Lenses 128

Table 14-3 Identification Codes for 0.2 g.t Grade Lenses 129
Table 14-4 Drill Hole Summary 131
Table 14-5 Composite Summary 131
Table 14-6 Density 132
Table 14-7 Gold Composite Statistics 133
Table 14-8 Silver Composite Statistics 134
Table 14-9 Composite Grade Capping 134
Table 14-10 Block Model Variables 135
Table 14-11 Estimation Parameters 137
Table 14-12 Classification Conditions 137
Table 14-13 Estimate Comparison for Gold versus a Nearest Neighbor at 0 Cutoff 139
Table 14-14 Estimate Comparison for Silver versus a Nearest Neighbor at 0 Cutoff 141
Table 14-15 Cove Mineral Resources 155
Table 16-1 Underground Workforce 2018 through 2020 162
Table 16-2 Peak Underground Workforce beginning 2021 162
Table 16-3 Underground Mobile Equipment and Support Equipment for Exploration Development Phase 162
Table 16-4 Underground Mobile Equipment and Support Equipment for Peak Production Mining 163
Table 16-5 Heading Productivity 164
Table 16-6 Annual Production and Development Plan Prior to Production Decision 164
Table 16-7 Annual Production and Development Following Positive Production Decision 164
Table 16-8 LOM Plan Reconciliation to Mineral Resource 167
Table 17-1 Toll Processing Feed Specifications 169
Table 17-2 Composites and Metallurgical Zones 173
Table 17-3 Gold and Silver Payable Recoveries 174
Table 18-1 Groundwater Monitoring during the 30-Day Constant Rate Test 177
Table 18-2 Dewatering Summery 178
Table 18-3 Backfill Scoping Tests 28-Day Unconfined Compressive Strength 184
Table 20-1 Cove Project Existing Permits 189
Table 21-1 Project Capital Costs ($M) 193
Table 21-2 Mine Development Unit Costs 193
Table 21-3 Mine Development Capital ($M) 194
Table 21-4 Dewatering Capital ($M) 194
Table 21-5 Facilities and Site General ($M) 194
Table 21-6 Electrical Transmission, Substation and Distribution ($M) 195
Table 21-7 Closure and Reclamation Costs ($M) 195
Table 21-8 Unit Operating Costs 196

Table 21-9 One Way Trucking Distance to Nevada Metallurgical Plants 196
Table 21-10 Operating Costs 196
Table 22-1 Income Statement 198
Table 22-2 Cash Flow Statement 199
Table 22-3 Financial Statistics[1] 199
Table 26-1 Project Risks 210
Table 26-2 Opportunities 210

List of Figures

Figure 1-1 Section View of Cove Mineralization looking NE 18
Figure 1-2 Project Timeline 24
Figure 4-1 Location Map 41
Figure 4-2 The McCoy Joint Venture Property and Cove Deposit 44
Figure 4-3 Mineral Claim Map 45
Figure 4-4 Carveout Boundary (Blue) and Summa Royalty (Red) 46
Figure 5-1 McCoy-Cove Project Area Looking Southeast 50
Figure 7-1 Regional Geology 58
Figure 7-2 Triassic Stratigraphy and Mineralization 59
Figure 9-1 Underground Exploration Potential 68
Figure 9-2 Joint Venture Exploration Targets outside the 100% Premier Carveout 69
Figure 10-1 Plan View of Drill Holes Used for the Current Analysis 70
Figure 10-2 Sample Section of CSD-Gap and Gap Hybrid Drilling 72
Figure 10-3 Sample Section of Helen Zone Drilling 73
Figure 10-4 Sample Section of CSD Zone Drilling 74
Figure 10-5 Sample Section of 2201 Zone Drilling 75
Figure 11-1 Blank Assay Results 84
Figure 11-2 SP 37 Standard Reference Material Results 85
Figure 11-3 CDN-GS-22 Certified Reference Material Results 85
Figure 11-4 CDN-GS-5H Certified Reference Material Results 86
Figure 11-5 GS912-1 Certified Reference Material Results 86
Figure 11-6 Prep Duplicates - ALS Reno 87
Figure 11-7 Lab Check Duplicates 88
Figure 14-1 Plan View of Cove Mineralized Zones 124
Figure 14-2 Section View of Cove Mineralized Zones looking NE 125
Figure 14-3 Section View of the Helen Zone looking NW 126
Figure 14-4 Section View of the 2201 Zone looking NW 127
Figure 14-5 Low Grade Envelope 129
Figure 14-6 Section Looking AZ 315 Showing Mineralized Lenses in the GAP Zone 130
Figure 14-7 Sterilization Surfaces 139
Figure 14-8 Comparison of Composite and Estimated Block Gold Grades, Helen Zone 144
Figure 14-9 Comparison of Composite and Estimated Block Gold Grades, Gap Zone 145
Figure 14-10 Comparison of Composite and Estimated Block Gold Grades, CSD Zone 146
Figure 14-11 Comparison of Composite and Estimated Block Gold Grades, 2201 Zone 146
Figure 14-12 Gold Swath Plots of Helen Zone 3103 147
Figure 14-13 Silver Swath Plots of Helen Zone 3103 148
Figure 14-14 Gold Swath Plots of Gap Zone 2208 149

Figure 14-15 Silver Swath Plots of Gap Zone 2208 150
Figure 14-16 Gold Swath Plots of CSD Zone 1106 150
Figure 14-17 Silver Swath Plots of CSD Zone 1106 151
Figure 14-18 Gold Swath Plots of 2201 Zone 1302 151
Figure 14-19 Silver Swath Plots of 2201 Zone 1302 152
Figure 14-20 Helen Zone Grade Tonnage Plots 153
Figure 14-21 Gap Zone Grade Tonnage Plots 153
Figure 14-22 CSD Zone Grade Tonnage Plots 154
Figure 14-23 2201 Zone Grade Tonnage Plots 154
Figure 16-1 Plan view showing portal, main haulages, and two raises to surface 157
Figure 16-2 Formation RQD 159
Figure 16-3 Long Section showing portal, main haulages, and two raises to surface 160
Figure 16-4 Gap and Helen Development and Production depicting dates of planned water drawdown 160
Figure 16-5 Depiction of Drift and Fill method 161
Figure 16-6 Production Profile 166
Figure 16-7 Long Section View of Helen Mine Plan by Year 166
Figure 16-8 Long Section View of Gap Mine Plan by Year 167
Figure 17-1 Helen and Gap Metallurgy Zones 173
Figure 18-1 Draw Down Isopleths and Monitoring Locations 177
Figure 18-2 Dewatering and Development Timing 178
Figure 18-3 Rapid Infiltration Basin During 2017 Pump Test 179
Figure 18-4 Proposed 2018 Hydrology Drill Plan 181
Figure 18-5 Electrical Demand 182
Figure 18-6 Electrical Site Plan 183
Figure 18-7 Mine Facilities Layout 184
Figure 19-1 Historical Monthly Average Gold and Silver Prices and 36 Month Trailing Average 186
Figure 21-1 Cutoff Grade vs. Recovery 197
Figure 22-1 Project Timeline 198
Figure 22-2 Gold Production and Unit Costs 201
Figure 22-3 Cash Flow Waterfall Chart Including Pre-Development (Sunk) Costs 201
Figure 22-4 NPV 5% Sensitivity 202
Figure 22-5 Profitability Index 5% Sensitivity 203
Figure 22-6 IRR Sensitivity 203

List of Abbreviations

A	Ampere	kA	kiloamperes
AA	atomic absorption	kCFM	thousand cubic feet per minute
A/m^2	amperes per square meter	Kg	Kilograms
AGP	Acid Generation Potential	km	kilometer
Ag	Silver	km2	square kilometer
ANFO	ammonium nitrate fuel oil	kWh/t	kilowatt-hour per ton
ANP	Acid Neutralization Potential	LoM	Life-of-Mine
Au	Gold	m	meter
AuEq	gold equivalent	m2	square meter
btu	British Thermal Unit	m3	cubic meter
°C	degrees Celsius	masl	meters above sea level
CCD	counter-current decantation	mg/L	milligrams/liter
CIL	carbon-in-leach	mm	millimeter
CoG	Cut off grade	mm2	square millimeter
cm	centimeter	mm3	cubic millimeter
cm^2	square centimeter	MME	Mine & Mill Engineering
cm^3	cubic centimeter	Moz	million troy ounces
cfm	cubic feet per minute	Mt	million tonnes
ConfC	confidence code	MTW	measured true width
CRec	core recovery	MW	million watts
CSS	closed-side setting	m.y.	million years
CTW	calculated true width	NGO	non-governmental organization
°	degree (degrees)	NI 43-101	Canadian National Instrument 43-101
dia.	diameter	oz	Troy Ounce
EA	Environmental Assesment	opt	Troy Ounce per short ton
EIS	Environmental Impact Statement	oz/ton	Troy Ounce per short ton
EMP	Environmental Management Plan	%	percent
FA	fire assay	PLC	Programmable Logic Controller
Ft	Foot	PLS	Pregnant Leach Solution
Ft2	Square foot	PMF	probable maximum flood
Ft3	Cubic foot	POO	Plan of Operations
g	Gram	ppb	parts per billion
g/L	gram per liter	ppm	parts per million
g-mol	gram-mole	QAQC	Quality Assurance/Quality Control
g/t	grams per metric tonne	RC	reverse circulation drilling
ha	hectares	ROM	Run-of-Mine
HDPE	Height Density Polyethylene	RQD	Rock Quality Description
HTW	horizontal true width	SEC	U.S. Securities & Exchange Commission
ICP	induced couple plasma	Sec	second
ID2	inverse-distance squared	SG	specific gravity
ID3	inverse-distance cubed	SPT	Standard penetration test
ILS	Intermediate Leach Solution	ton	US Short Ton
LOI	Loss On Ignition	Tonne	Metric Tonne

1. Summary

1.1. Introduction

Practical Mining LLC (Practical or PM) was engaged by Au-Reka Gold Corporation, a subsidiary of Premier Gold Mines Limited (Premier or the Company) to prepare a Preliminary Economic Assessment (PEA) on the McCoy Cove Project (Cove or the Project) in Lander County, Nevada. This Technical Report (TR) has been prepared in accordance with National Instrument 43-101 (NI43-101) of the Canadian Security Administrators, and follows the "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (CIM 2014). Mineral Resources classifications are in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" (CIM 2014).

This TR dated the 29th day of June 2018 with an effective date of March 31, 2018 updates the March 2017 Mineral Resource estimate and presents an underground mine plan, metallurgical testing, hydrogeologic summary, and financial analysis.

Cautionary Notes:

1. *The financial analysis contains certain information that may constitute "forward-looking information" under applicable Canadian securities legislation. Forward-looking information includes, but is not limited to, statements regarding the Company's achievement of the full-year projections for ounce production, production costs, AISC costs per ounce, cash cost per ounce and realized gold/silver price per ounce, the Company's ability to meet annual operations estimates, and statements about strategic plans, including future operations, future work programs, capital expenditures, discovery and production of minerals, price of gold and currency exchange rates, timing of geological reports and corporate and technical objectives. Forward-looking information is necessarily based upon a number of assumptions that, while considered reasonable, are subject to known and unknown risks, uncertainties, and other factors which may cause the actual results and future events to differ materially from those expressed or implied by such forward looking information, including the risks inherent to the mining industry, adverse economic and market developments and the risks identified in Premier's annual information form under the heading "Risk Factors". There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. All forward-looking information contained in this Presentation is given as of the date hereof and is based upon the opinions and estimates of management and information available to management as at the date hereof. Premier disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by law, and;*
2. *This PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.*

1.2. Property Description

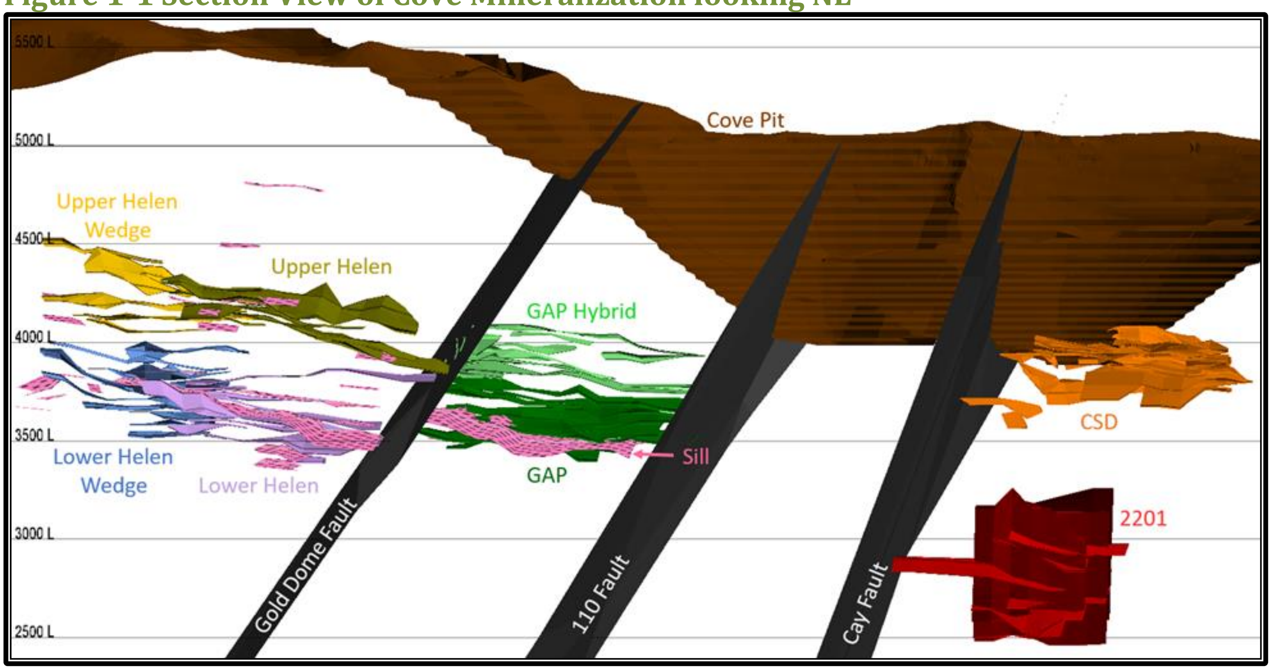
The Cove Project covers 28,218 acres and is located 32 miles south of the Town of Battle Mountain, in the Fish Creek Mountains of Lander County, Nevada. It is centered approximately at 40°22’ N and 117°13’ W and lies within the McCoy Mining District.

The Cove deposit consists of the Helen, Gap, CSD, and 2201 zones. They are located beneath the historically mined Cove open pit and extend approximately 2,000 feet northwest from the pit. The Cove deposit was mined by Echo Bay Mines Ltd. (Echo Bay) between 1987 and 2001, and produced 2.6 million ounces of gold and 100 million ounces of silver. Gold and silver production from heap leach pads continued until 2006.

1.3. Geology and Mineral Resource

The Cove Project contains four structurally controlled mineralized zones within the Triassic sedimentary package. The Helen and Gap zones are Carlin Style disseminated refractory gold deposits. The Cove South Deep (CSD) gold and silver mineralization is associated with disseminated sulfides and is characterized by Ag:Au ratios of 50:1 to over 100:1. The 2201 zone is comprised of disseminated sulfides within sheeted stockwork veins with high concentrations of lead and zinc (Figure 1-1).

Figure 1-1 Section View of Cove Mineralization looking NE



Mineral Resources are constrained to high-grade wireframe models constructed at a nominal 0.09 opt (3 g/t) grade shell. The Mineral Resource estimate relies on data from 387 core drill holes totaling 548,038 feet and 1,010 reverse circulation (RC) drill holes totaling 579,443 feet. From these drill holes, 3,146 samples were flagged within the high-grade wireframes to be used in grade estimation.

Parent block dimensions are 100 ft x 100 ft x 100 ft with sub-block dimensions as small as 1 ft x 1 ft x 1 ft. Block grades were estimated using Inverse Distance Cubed (ID^3) methods.

A block is classified as Indicated if there are at least two composites within an average distance of 100 feet or less and at least one of the samples is within fifty feet. A block is classified Inferred if there are at least two composites within 300 feet but more than 100 feet. Cove Mineral Resources as of March 31, 2018 are presented in Table 1-1.

Table 1-1 Cove Mineral Resources

	Tons (000)	Tonnes (000)	Au (opt)	Au g/t	Ag (opt)	Ag (g/t)	Au ozs (000)	Ag ozs (000)
Indicated Mineral Resource								
Helen	577	524	0.369	12.66	0.103	3.54	213	60
Gap	167	151	0.357	12.23	0.431	14.78	60	72
CSD	301	273	0.229	7.86	2.556	87.63	69	768
Total Indicated	1,045	948	0.327	11.21	0.861	29.53	342	900
Inferred Mineral Resource								
Helen	1,493	1,355	0.335	11.49	0.118	4.06	500	177
Gap	1,731	1,570	0.317	10.88	0.457	15.67	549	791
CSD	503	456	0.204	7.00	2.266	77.68	103	1,140
2201	310	282	0.546	18.72	1.127	38.65	169	350
Total Inferred	4,037	3,663	0.327	11.23	0.609	20.87	1,322	2,457

Notes:

1. *Mineral Resources have been estimated at a gold price of $1,400 per troy ounce;*
2. *Mineral Resources have been estimated using gold metallurgical recoveries of 79.5% and 85.2% for roasting and pressure oxidation respectively;*
3. *Mineral Resources have been estimated using a gold equivalent cutoff grade of 0.149 opt;*
4. *One ounce of gold is equivalent to 140 ounces of silver;*
5. *The effective date of the Mineral Resource estimate is March 31, 2018;*
6. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors; and*
7. *The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there is insufficient exploration to define these inferred Mineral Resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.*

1.4. Metallurgical Testing and Processing

Eleven composites from the Helen Zone and ten from the Gap were sent to SGS Canada Inc., Lakefield, Ontario, Canada in 2017. The primary objectives of the test program were as follows:

- Select drill holes and discrete intervals in the drill holes to obtain initial spatial representation vertically and with the length and breadth of the resources;
- Obtain head assays and tests to adequately characterize the physical and metallurgical properties of each resource required for processing by a third party;
- To develop initial metallurgical data to evaluate the resource targets at the project site based on potential metallurgical processing by a third party;
- Testing to project process precious metal extractions, and metal deportment, reagent consumptions, and track metals (Au, Ag, As, & Cu) by:
    - Whole cyanidation;
    - Roasting followed by calcine cyanidation, and;
    - Pressure Oxidation followed by cyanidation of neutralized slurry.
- Roaster and pressure oxidation test conditions used in the program were based on those provided by a potential toll processing operator, and;
- The program was not specifically designed to determine the optimal roasting or pressure oxidation conditions or develop design data for a new processing plant.

The roasting and calcine cyanidation tests indicated the following:

- The roasting effectively oxidized the sulfide content in both groups of composites with the Helen composites ranging from 85.9% to 97.0% sulfide oxidation while the Gap composites ranged from 87.9% to 98.1%;
- Carbonate oxidation in the Helen composites was generally low whereas the carbonate oxidation in the Gap composites was somewhat higher;
- The gold extractions by direct cyanidation of Helen composite calcines was variable ranging from 63.5% to 90.8%;
- The silver extractions by direct cyanidation of Helen composite calcines was variable ranging from 9.6% to 56.5%;
- The gold extractions by direct cyanidation of Gap composite calcines was variable ranging from 54.4% to 89.4%, and;
- The silver extractions by direct cyanidation of Gap composite calcines was also variable ranging from 23.1% to 77.0%.

The pressure oxidation and POX residue cyanidation tests indicated the following:

- The POX step effectively oxidized the sulfide content in both groups of composites;
- Carbonate removal in the Helen composites averaged 97.2% whereas the carbonate removal in the Gap composites averaged 82.5% however this average is skewed due to Gap composites 2 and 20 having very low head carbonate contents;
- The gold extractions by direct cyanidation of Helen composite POX residues were generally very low ranging from 0.3% to 96.6%;
- The silver extractions by direct cyanidation of Helen composite POX residues was variable ranging from 6.7% to 69.6%;
- The gold extractions by direct cyanidation of Gap composite POX residues was variable ranging from 5.7% to 73.6%;
- The silver extractions by direct cyanidation of Gap composite POX residues was also variable ranging from 52.5% to 81.7%, and;
- The data set was too small to establish any clear relations between mineralogy and metal head grade and extractions although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using pressure oxidation and residue cyanidation.

A second phase of testing was conducted to investigate the reasons for the low roaster and POX metal extractions observed in Phase 1 tests. The program first consisted of rerunning roasting and POX tests on selected composites from the Helen and Gap. The calcines and POX residues resulting from each of the rerun oxidation treatments were split in two. One half of each spilt was subjected to direct cyanidation as was performed in phase 1. The second half of each split was subjected to carbon-in-leach (CIL) cyanidation. The CIL leach was used as a means to partially diagnose if pregnant solution robbing was causing the low extractions.

The rerun pressure oxidation tests indicated the following:

- The Helen composites direct cyanidation of the rerun POX residues 48-hour gold extractions ranged from 0.6% to5.1% whereas the gold extractions for the 48-hour CIL tests ranged from 62.3% to 81.9%, significantly higher than the direct cyanidation;
- The Helen direct cyanidation of the rerun POX residues 48-hour silver extraction ranged from 36.2% to 86.9% whereas the silver extractions for the 48-hour CIL test ranged from 76.8% to 86.9% significantly higher than the direct cyanidation;
- The Gap composites direct cyanidation of the POX residues 48-hour gold extractions ranged from 1.6% to 77.8% whereas the average gold extractions for the 48-hour CIL tests ranged from 70.5% to 95.9%, significantly higher than the direct cyanidation;

- The Gap composites direct cyanidation of the rerun POX residues 48-hour silver extractions ranged from 19.9% to 84.1% whereas the gold extractions for the 48-hour CIL test ranged from 71.6% to 87.0%, significantly higher than the direct cyanidation, and;
- The rerun POX tests on the selected composites confirmed the supposition that pregnant solution robbing occurs in direct cyanidation of the calcines and that CIL cyanidation can increase gold extractions and silver extractions very significantly for both the Helen and Gap composites tested versus direct cyanidation.

The rerun roasting tests showed that the application of CIL cyanidation for the calcine leach could increase precious metal extractions, however, the gold extraction was still somewhat lower than expected. Diagnostic leaching of the rerun calcine cyanidation residue was conducted to investigate the distribution of gold in the leached calcine. The rerun calcine cyanide residue diagnostic leach indicated the following:

- The estimated amount of gold associated with the iron oxides, ferrites or calcite in the Helen composites leached calcine residues ranged from 8.2% to 17.9% and averaged 11.0%, with the remaining gold estimated to be in siliceous gangue which ranged from 9.2% to 18.0% and averaged 12.8%;
- The estimated amount of gold associated with the iron oxides, ferrites or calcite in the Gap composites leached calcine residues ranged from 11.7% to 35.9%, with the remaining gold estimated to be in siliceous gangue which ranged from 2.6% to 14.0%;
- The data for the composites tested indicated that the Helen likely has more gold associated with siliceous material than the Gap composites which showed a greater amount of gold associated with the iron oxides, ferrites, or calcite following roasting, and;
- The data also suggests that the specified roasting conditions from a potential toll roasting operation may not be optimal for the Helen or Gap material.

The premise for treating the material from the Helen and Gap resources is toll milling and treating by another mining company through either existing roasting and calcine cyanidation or existing pressure oxidation and residue cyanidation facilities.

Premier Gold solicited two items from a prospective toll operator with both roasting and pressure oxidation (POX) processes and their associated cyanidation processes for the respective calcines or POX residues.

The first item included the test protocols and test conditions for laboratory bench scale batch roasting and pressure oxidation tests conditions for the 2017 metallurgical testing. The conditions provided approximate the expected operating conditions in the prospective toll operator’s roasting and POX facilities.

The second item Premier Gold solicited was terms and conditions for toll milling and treating Helen resource material. Premier Gold provided a package of Helen metallurgical data for the roasting and POX tests from the 2017 test program to the prospective toll process operator for their consideration and as the basis for toll processing resource material through either the toll operator's roasting or POX facilities.

The test data indicates that the Helen composites were generally more amenable to roasting and calcine CIL cyanidation than POX and residue CIL cyanidation. The assay data for the Helen composites indicates that there may be some problems from some areas to meet roaster feed specifications. Onsite blending of Helen resource material to meet specifications prior to shipping to the toll processor provided that resource material is available for blending will likely be required.

Conversely, the Gap composite test data were generally more amenable to POX and residue CIL cyanidation. Again, blending would likely have to be used prior to shipping offsite to provide on specification material to the toll processor.

1.5. Mining, Infrastructure, and Project Schedule

Access to the mineralized zones will be through a portal located just north of the Cove Pit. Primary development totals 23,776 feet with gradients up to +/- 15%. Ventilation and secondary egress will be gained through ventilation boreholes located south of the deposit.

Drift and Fill mining with a minimum mining height of eight feet will be the primary method for extraction of the Helen and Gap Mineral Resource. Where the mineralized lenses thicken, breasting the sill or back can recover additional mineralization. Waste rock from development and waste reclaimed from historic dumps will be used for Cemented Rock Fill (CRF) or unconsolidated (GOB) fill as appropriate to achieve high levels of extraction. Development and production mining will be performed by a qualified mining contractor thus reducing the capital requirements for the Project.

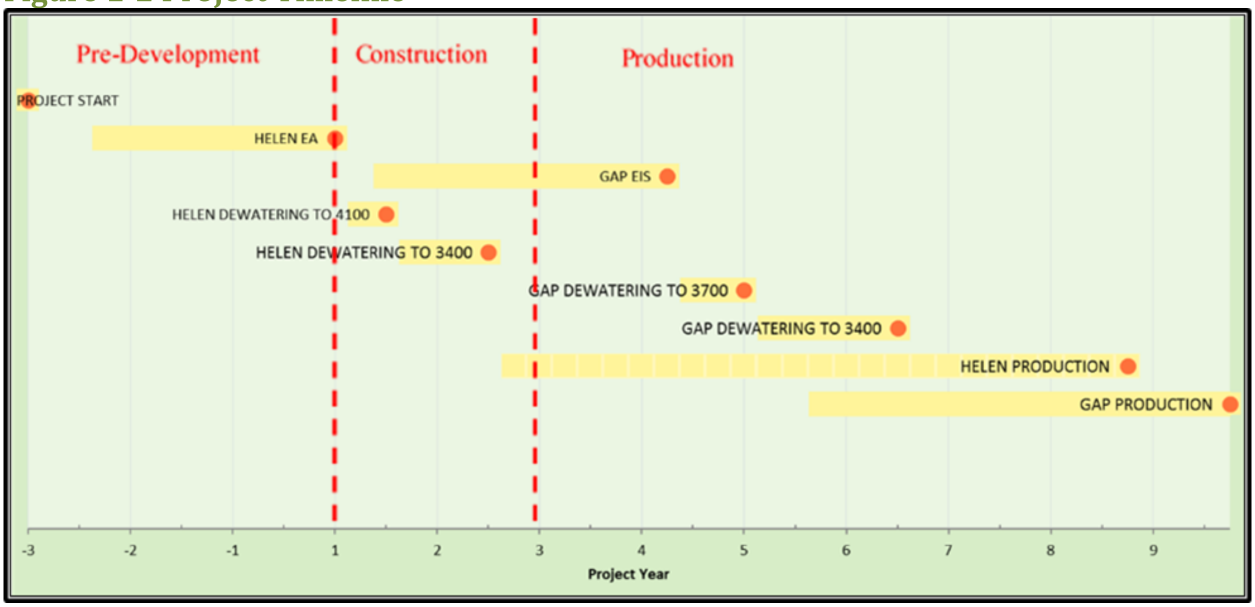
A trucking contractor will transport mineralization mined over local, state, and federal roads for processing at one of six roasting or pressure oxidation facilities in northeast Nevada.

Helen dewatering from up to five surface wells pumping at a combined rate of 10,500 gpm will be required prior to accessing the mineralization. Gap zone dewatering rates up to 26,000 gpm will be achieved from ten surface wells. Dewatering water will be piped to several Rapid Infiltration Basins (RIBs) constructed at the northern project boundary. The RIB locations have been selected to prevent recharge into the Cove hydrogeologic system.

Long term electrical power demands up to 12.5 MW will be supplied by NV Energy via an existing 120 kV transmission line which connects the Project site to NV Energy's Bannock substation. Power for initial mine development and underground delineation drilling will be provided from an existing 26.9kV distribution line that also terminates on the property. A new substation and 13.8kV distribution system will be constructed.

Permitting of the project is anticipated in three phases to gain early cash flow and maximize NPV. Approval of first phase permits for portal construction, initial decline development, and delineation drilling is anticipated in the first half 2018. The second phase of permitting will require an Environmental Assessment (EA) encompassing dewatering and mining of the Helen Zone. The final phase will require an Environmental Impact Statement (EIS) for dewatering and mining the Gap Zone. The overall project timeline is shown in Figure 1-2.

Figure 1-2 Project Timeline



1.6. Economic Analysis

Capital spending over the life of the project is subdivided into three categories. Pre-development spending of $25.8M encompasses portal construction, exploration decline and drill platform development, delineation drilling, baseline data collection, engineering, and permitting the Helen Zone. Construction capital is required for Helen Zone dewatering and mine development and is projected at $46.6M over a two-year period commencing in 2021. Sustaining capital includes the Gap dewatering system and mine development and totals $67.7M commencing in 2023.

Gold recovery will total 740,000 ounces over the eight-year mine life. Material mined for processing averages 0.305 Au opt. Production averages 1,270 tpd following a two-year buildup period and peaks in 2026 at 1,360 tpd.

The constant dollar financial analysis presented herein treats all pre-development capital as sunk capital and it is excluded from the cash flow and financial statistics of Table 1-2.

Table 1-2 Financial Statistics[1]

Gold price - base case (US$/oz)	$1,250.00
Silver price - base case (US$/oz)	$17.00
Mine life (years)	8.0
Maximum mining rate (tons/day)	1,360.0
Average grade (oz/t Au)	0.305
Average gold recovery (roaster %)	79%
Average gold recovery (autoclave %)	86%
Average annual gold production (koz)	92
Total recovered gold (koz)	740
Pre-development capital ($M)	$26
Mine construction capital ($M)	$47
Sustaining capital (M$)	$68
Development Decision Date	January 2021
Cash cost (US$/oz) [2]	$790
All-in sustaining cost (US$/oz) [2]	$924
Project after-tax $NPV_{5\%}$ (M$)	$142
Project after-tax IRR	48%
Payback Period	4.0 Years
Profitability Index $_{5\%}$ [3]	2.4

Notes:

1. *The financial data presented herein treats pre-development capital (planned expenditures prior to the development decision) as sunk costs and it is excluded from cost per ounce, NPV, IRR, payback period and profitability index calculations;*
2. *Net of byproduct sales;*
3. *Profitability index (PI), is the ratio of payoff to investment of a proposed project. It is a useful tool for ranking projects because it allows you to quantify the amount of value created per unit of investment. A profitability index of 1 indicates breakeven;*
4. *The Deferred Bullet Payment Consideration is not included in the cash-flow or financial calculations of this Technical Report.*
5. *The Newmont 1.5% NSR is not included in the cash-flow or financial calculations of this Technical Report.*

1.7. Conclusions

Metallurgical Testing

1. Head assaying for both the Helen and Gap indicated that the gold in the two resources will likely be finely disseminated and will not likely have a significant coarse or nugget gold content;
2. The mineralogy of the Helen and Gap resources differ in two significant areas, the first being that the Helen appears to be lower in arsenic content than the Gap resource and that the Gap resource appears to be lower on average in TCM and TOC than the Helen resource;
3. The Helen composite arsenic assays indicate the resource is lower in arsenic content that the Gap resource;
4. The Helen and Gap resources based on the composites tested appear to be generally refractory to conventional whole cyanidation and will need some type of oxidation process to significantly increase gold extractions over whole cyanidation;
5. Based on the composites tested the Helen appears to generally be more amenable to Roasting and CIL cyanidation, however, there may be areas that are more amenable or can only be treated using pressure oxidation and residue CIL cyanidation;
6. Based on the composites tested, the Gap resource appears to generally be more amenable to pressure oxidation followed by residue CIL cyanidation, however, there may be areas that are more amenable or can only be treated using roasting and calcine CIL cyanidation, and;
7. The data set was too small to establish any clear relations between mineralogy and metal head grade and extractions for either resource although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using either roasting and calcine cyanidation or pressure oxidation and residue cyanidation.

Toll Processing

1. The feed specifications appear to be somewhat rigid and could preclude some material being sent to the toll processor. Blending may allow shipment of some off-specification material provided appropriate material is available for onsite blending prior to shipping to the toll processor;
2. The terms appear to be consistent and typical with those encountered in the industry, and;
3. The recovery terms appear to be the result of analyzing the metallurgical data provided by Premier Gold.

Mining and Infrastructure

1. Mining conditions typical for sedimentary deposits in the north-east Nevada extensional tectonic environments are anticipated;
2. Helen Zone dewatering will require five wells and reach pumping rates of 10,500 gpm, and;
3. Gap Zone dewatering will require ten wells and reach pumping rates of 26,000 gpm for a total projected pumping rate of 36,500 gpm.

Financials

1. Capital requirements total $114.4M excluding $25.8M in sunk pre-development capital;
2. The project achieves NPV 5% of $142M and NPV 8% of $118M, and;
3. The estimated payback period is 4.0 years with an IRR of 48%.

1.8. Recommendations

Resource Delineation and Exploration

1. Portal construction and development of an underground drilling platform should proceed as soon as possible;
2. Resource delineation drilling from underground can be achieved with improved accuracy as compared to surface drill holes with depths approaching 2,000 feet and significant hole deviation;
3. The Cove Pit prohibits drilling the Gap extension area and portions of the Gap deposit. These are the most prospective nearby areas for adding significant Mineral Resources, and;
4. Expansion of the 2201 Zone could add high grade mineralization to the project which would be accessed through the Helen and Gap infrastructure.

Dewatering

1. PW 17-101 did not reach the targeted depth and pumping rates during the 30-day test were less than anticipated. Two additional wells and extended drawdown pumping tests need to be completed in the Helen and Gap zones during the 2018 season, and;
2. Complete detailed hydrogeologic modeling of the drawdown test results and update estimated dewatering requirements.

Mining

1. A geotechnical characterization program should be implemented along with resource delineation:
    a. The objectives of the program are to characterize the mining horizons using the Rock Mass Rating (RMR) system;
    b. Collect downhole Acoustic Tele Viewer (ATV) drill logs to collect joint orientation data for mine designs and accurately estimate ground support requirements, and;
    c. Collect full core samples for physical rock property testing.
2. Complete additional testing of potential back fill sources to optimize the Cemented Rock Fill (CRF) mix design, and
3. Complete a ventilation simulation to predict Diesel Particulate Matter (DPM), carbon monoxide, and other contaminate concentrations.

Metallurgical Testing

1. Additional metallurgical testing will be needed to thoroughly investigate the variability and viability of Helen and Gap resources to roasting and pressure oxidation with CIL cyanidation for which a program evaluating thirty to forty composites from each resource is suggested with objectives as follows:
    a. Assess variability of the responses to roasting and calcine cyanidation across the resources;
    b. Assess variability of the responses to pressure oxidation and residue cyanidation across the resources;
    c. Consider some POX optimization tests such as pre-acidulation ahead of the POX process;
    d. Testing should attempt to establish head grade and extraction relations for use in more detailed resource modelling;
    e. Mineralogy impacts need to be established and geologic domains within each resource need to be determined;
    f. Additional comminution data should be collected to assess variability within the resources.
2. In addition to evaluating resource process by a toll processing operator, consideration should be given to evaluate onsite processing;
3. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
4. Consider flotation tests to pre-float carbonates, and;

5. Consider other mill design tests as alternatives to toll processing. These would include roasting, POX optimization tests, and solid liquid separation tests.

Toll Processing

1. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll processing offsite is used and potentially for estimating extractions within the resources;
2. Additional metallurgical testing should be conducted to confirm the proposed payable recoveries are appropriate for the resources;
3. Development of a preliminary or conceptual onsite blending program is recommended to evaluate if on specification material can consistently be supplied to a toll processor, and:
4. The next phase metallurgical program should examine blending of out of specification resource materials to produce on specification material. The blending should be based on material projected to be mined in a given period, for example, blending of material that is available in the first six months of operation should not be tested with material projected to only be available in year three of mining.

Permitting and Development Decision

1. Baseline data collection in support of the Helen EA and GAP EIS should be done simultaneously to reduce the Project's critical path and bring forward production, and;
2. The project should proceed directly with a feasibility or pre-feasibility study to support a development decision.

2. Introduction

2.1. Terms of Reference and Purpose of this Technical Report

This TR updates the Cove Project Mineral Resources and provides a Preliminary Economic Assessment of the Project using indicated and inferred Mineral Resources. This TR was prepared in accordance with the requirements of NI 43-101 and Form 43-101F1 (43-101F1) for technical reports.

Mineral resource and mineral reserve definitions are set forth in "*Canadian Institute of Mining, Metallurgy and Petroleum (CIM) – Definition Standards for Mineral Resources and Mineral Reserves adopted by CIM Council on May 10, 2014*."

2.2. Qualification of the Authors

This TR includes technical evaluations from five independent consultants. The consultants are specialists in the fields of geology, exploration, and open pit and underground mining.

None of the authors has any beneficial interest in Premier or any of its subsidiaries or in the assets of Premier or any of its subsidiaries. The authors will be paid a fee for this work in accordance with normal professional consulting practices.

The QP's contributing to this report are listed in Table 2-1. The Certificates of Qualifications for each are provided in at the end of this report.

Table 2-1 Qualified Professionals

Company QP	Title	Discipline	Most Recent Personal Inspection	Responsible Sections
Practical Mining LLC				
Mark Odell	Manager	Mining and Mineral Resources	October 10, 2017	1.1 – 1.3, 1.5 – 1.8, 2 – 6, 11, 15, 16, 17.3, 18 - 26
Laura Symmes	Sr. Geologist	Geology and Mineral Resources	March 19, 2018	7-10, 12, 14
Sarah Bull	Mining Engineer	Mining	None	16
Adam Knight	Mining Engineer	Mining	June 5, 2018	16
Jacobs Engineering				
Rich Bohling	Technical Services Manager	Metallurgy	March 15, 2017	1.4, 1.7, 1.8, 13, 17.1, 17.2, 17.4, & 25

2.3. Sources of Information

Information sources are documented either within the text and cited in references, or are cited in references only. The authors believe the information provided by Premier to be accurate based on their work at the Project.

2.4. Units of Measure

The units of measure used in this report are shown in Table 2-2. U.S. Imperial units of measure are used throughout this document unless otherwise noted. The glossary of geological and mining related terms is also provided at the end of this section. Currency is expressed as United States Dollars unless otherwise noted.

Table 2-2 Units of Measure

US Imperial to Metric conversions
Linear Measure
1 inch = 2.54 cm
1 foot = 0.3048 m
1 yard = 0.9144 m
1 mile = 1.6 km
Area Measure
1 acre = 0.4047 ha
1 square mile = 640 acres = 259 ha
Weight
1 short ton (st) = 2,000 lbs = 0.9071 metric tons
1 lb = 0.454 kg = 14.5833 troy oz
Assay Values
1 oz per short ton = 34.2857 g/t
1 troy oz = 31.1036 g
1 part per billion = 0.0000292 oz/ton
1 part per million = 0.0292 oz/ton = 1g/t

2.5. Coordinate Datum

Spatial data utilized in analysis presented in this TR are projected to UTM Zone 11 North American Datum 1983 feet. All spatial measurements are in international survey feet.

Downhole surveys are collected with a True North seeking gyro. No correction for declination is needed.

Assay: The chemical analysis of mineral samples to determine the metal content.

Asbuilt: (plural asbuilts), a field survey, construction drawing, 3D model, or other descriptive representation of an engineered design for underground workings.

Composite: Combining more than one sample result to give an average result over a larger distance.

Concentrate: A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.

Crushing: Initial process of reducing material size to render it more amenable for further processing.

Cut-off Grade (CoG): The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.

Dilution: Waste, which is unavoidably mined with ore.

Dip: Angle of inclination of a geological feature/rock from the horizontal.

Fault: The surface of a fracture along which movement has occurred.

Footwall: The underlying side of a mineralized body or stope.

Gangue: Non-valuable components of the ore.

Grade: The measure of concentration of valuable minerals within mineralized rock.

Hanging wall: The overlying side of a mineralized body or stope.

Haulage: A horizontal underground excavation which is used to transport mined rock.

Igneous: Primary crystalline rock formed by the solidification of magma.

Kriging: A weighted, moving average interpolation method in which the set of weights assigned to samples minimizes the estimation variance.

Level: A main underground roadway or passage driven along a level course to afford access to stopes or workings and to provide ventilation and a haulage way for the removal of broken rock.

Lithological: Geological description pertaining to different rock types.

Milling: A general term used to describe the process in which the ore is crushed, ground and subjected to physical or chemical treatment to extract the valuable minerals in a concentrate or finished product.

Mineral/Mining Lease: A lease area for which mineral rights are held.

Mining Assets: The Material Properties and Significant Exploration Properties.

Sedimentary: Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.

Sill1: A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.

Sill2: The floor of a mine passage way.

Stope: An underground excavation from which ore has been removed.

Stratigraphy: The study of stratified rocks in terms of time and space.

Strike: Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.

Sulfide: A sulfur bearing mineral.

Tailings: Finely ground waste rock from which valuable minerals or metals have been extracted.

Thickening: The process of concentrating solid particles in suspension.

Total Expenditure: All expenditures including those of an operating and capital nature.

Variogram: A plot of the variance of paired sample measurements as a function of distance and/or direction.

Mineral Resources

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

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

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

'Modifying Factors' are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term ‘Mineral Reserve’ need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

‘Reference point’ refers to the mining or process point at which the Qualified Person prepares a Mineral Reserve. For example, most metal deposits disclose Mineral Reserves with a “mill feed” reference point. In these cases, reserves are reported as mined ore delivered to the plant and do not include reductions attributed to anticipated plant losses. In contrast, coal reserves have traditionally been reported as tonnes of “clean coal”. In this coal example, reserves are reported as a “saleable product” reference point and include reductions for plant yield (recovery). The Qualified Person must clearly state the ‘reference point’ used in the Mineral Reserve estimate.

Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

Proven Mineral Reserve (Proved Mineral Reserve)

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

Pre-Feasibility Study (Preliminary Feasibility Study)

The CIM Definition Standards requires the completion of a Pre-Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.

A Pre-Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Pre-Feasibility Study is at a lower confidence level than a Feasibility Study.

Feasibility Study

A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.

The term proponent captures issuers who may finance a project without using traditional financial institutions. In these cases, the technical and economic confidence of the Feasibility Study is equivalent to that required by a financial institution.

3. Reliance on Other Experts

The technical status for the claims and land holding is reliant on information provided by The US Bureau of Land Management and the Lander County Assessor's Office. The status of Premier's environmental program and the permitting activities were provided by Ms. Melissa Wendt, Principle Environmental Specialist, at Rubicon Environmental Services. Mr. Arnold Luther, Principal Hydrologist at Piteau Associates provided the hydrology model and analysis. These contributions have been reviewed by the authors and they are accurate portrayals of the Project at the time of writing this TR.

4. Property Description and Location

4.1. Property Description

The Cove Project covers 28,218 acres and is located 32 miles south of the Town of Battle Mountain, in the Fish Creek Mountains of Lander County, Nevada. It is centered approximately at 40°22’ N and 117°13’ W and lies within the McCoy Mining District (Figure 4-1).

The Cove deposit consists of the Helen, Gap, CSD, and 2201 zones. They are located beneath the historically mined Cove open pit and extend approximately 2,000 feet northwest from the pit. The historic McCoy open pit is located approximately 0.6 mi to the southwest. The Cove deposit was mined by Echo Bay Mines Ltd. (Echo Bay) between 1987 and 2001, and produced 2.6 million ounces of gold and 100 million ounces of silver. McCoy was mined between 1986 and 2001, and produced approximately 0.88 million ounces of gold and 3.0 million ounces of silver. Gold and silver production from heap leach pads continued until 2006.

The Project is located on federal land administered by the US Department of Interior - Bureau of Land Management (BLM) and patented mining claims.

Figure 4-1 Location Map

4.2. Status of Mineral Titles

The McCoy-Cove Project consists of 1,535 100%-owned unpatented claims and nine leased patented claims. The claim map provided by Premier is shown in Figure 4-3.

Unpatented claims have annual maintenance fees of $155 per claim payable to the Bureau of Land Management and a notice of intent to hold in the amount of $12 per claim payable to Lander County. The BLM LR2000 mining claim database shows all claim fees paid through September 2018. There are no additional work requirements for unpatented mining claims.

Patented claims are subject to property taxes and lease holding payments to the claim owner if applicable.

On June 15, 2006, Victoria Gold Corporation (Victoria) entered into a "Minerals Lease and Agreement" to lease a portion of the Project from Newmont. Under the terms of the Minerals Lease and Agreement, Victoria was subject to escalating yearly work commitments in the aggregate amount of $8.5 million over a period of seven years (consisting of $0.3 million, $0.7 million, $1.0 million, $1.25 million, $1.5 million, $1.75 million, and $2.0 million, respectively, in each year of the first seven years of the agreement dated June 15, 2006), of which $1.0 million was a firm obligation and was to be expended by June 15, 2008 (completed). Excess expenditures were allowed to be carried forward. Newmont acknowledged that Victoria spent over $9.1 million in exploration at the Project between June 15, 2006 and March 16, 2009, and satisfied the work commitment of Section 2(a) of the Minerals and Lease Agreement.

On June 14, 2012, Premier, through its wholly-owned subsidiary, Au-reka Gold Corporation (**Au-reka Gold**), acquired a 100% interest in the Cove portion of the Project from Victoria pursuant to an asset purchase agreement dated June 4, 2012 (**Cove Purchase Agreement**). In connection with the acquisition, Premier paid an aggregate of C$8,000,000 on closing, C$4,000,000 of which was paid in cash and the balance of which was satisfied by the issuance of 892,857 common shares of Premier. In addition, Premier issued a promissory note (**Cove Acquisition Promissory Note**) in the amount of C$20,000,000 payable in C$10,000,000 allotments on the first and second anniversary dates of the closing date of the acquisition. The Cove Promissory Note was repaid in full in June 2014. The Company also reimbursed Victoria in the amount of $1,206,277 in respect of exploration and related activities conducted on the Cove portion of the Project between March 15, 2012 and the closing of the transaction.

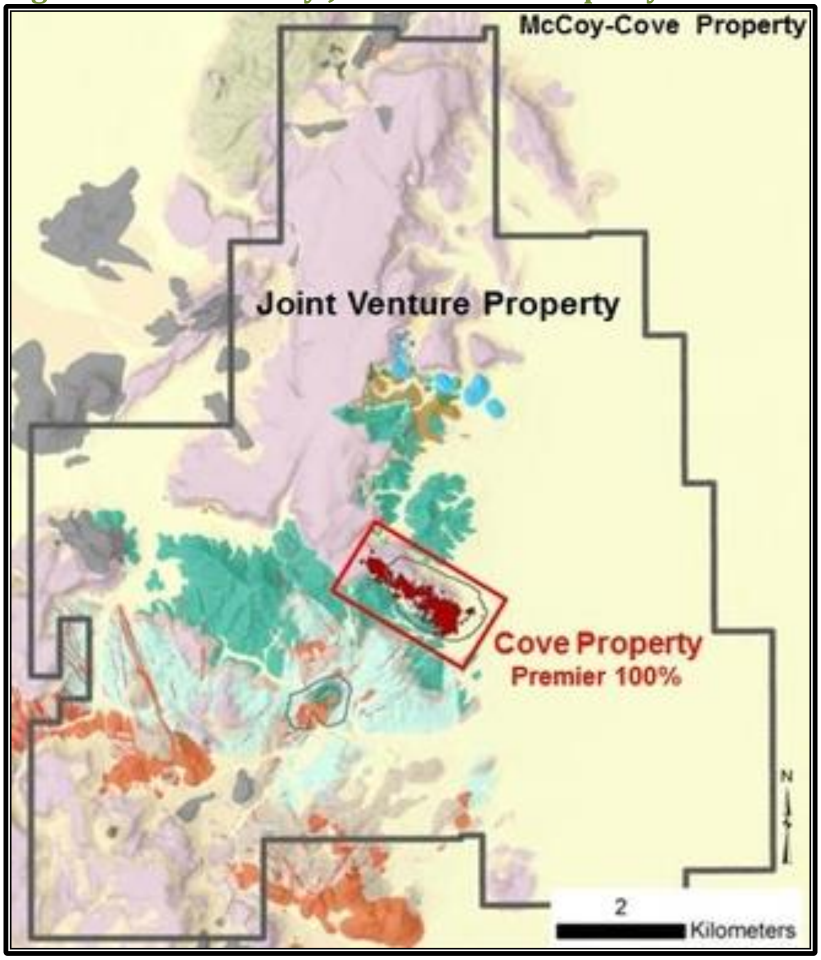
Pursuant to the Cove Purchase Agreement in the event of production from the Cove portion if the Project, Premier will make additional payments to Victoria in the aggregate amount of C$20,000,000 (consisting of cash and/or the equivalent value of Premier common shares, at Premier's option), payable in four installments of $5,000,000 each upon the cumulative production, to Premier's account, of 250,000, 500,000, 750,000 and 1,000,000 troy ounces of gold from the Cove potion of the Project (**Deferred Bullet Payment Consideration**). The Deferred Bullet Payment Consideration is not included in the cash-flow or financial calculations of this Technical Report.

In September 2014, Premier entered into an agreement with Newmont to acquire a 100% interest in the property. Upon closing of the transaction, Premier paid Newmont $15 million, replaced bonding of approximately $4 million via a surety policy, and transferred to Newmont all land sections that comprised the South Carlin Property. In addition, Premier made staged payments to Newmont over 18 months equal to $6 million. Additional details of the transaction included the elimination of Newmont's previous "back-in" rights to the Project, a 10-year good faith milling agreement for ores mined at McCoy-Cove and retention of a 1.5% NSR in the property.

Premier entered into an earn-in agreement (Barrick Earn-In Agreement) dated December 11, 2017, but effective January 8, 2018, with certain subsidiaries of Barrick Gold Corporation (Barrick).

Pursuant to the Barrick Earn-In Agreement, Barrick has an option to earn a 60% interest in the exploration portion of the Project (McCoy Joint Venture Property) by spending $22.5 million in exploration before June 30, 2022 (Barrick Earn-in). The McCoy Joint Venture Property excludes the "Cove Deposit" (being the claims within the "Carveout") portion of the Project which will be retained solely by Premier (Figure 4-2).

Figure 4-2 The McCoy Joint Venture Property and Cove Deposit



Following completion of the Barrick Earn-In, funding for the McCoy Joint Venture Property will be on a proportionate basis. Barrick will hold a right of first refusal over the "Cove Deposit" until the earlier of 5.5 years or one year following the completion of the Barrick Earn-In.

In addition, following completion of the Barrick Earn-In, with respect to the Deferred Bullet Payment Consideration due to Victoria: If any one of the payments is triggered by production from the McCoy Joint Venture Property and production from mining by Premier outside the McCoy Joint Venture Property, the payment would be proportionally split between the McCoy Joint Venture and Premier, on an ounce by ounce basis.

In June 2018, Premier provided notice to Newmont that it would exercise its right of first offer to acquire Newmont’s 1.5% net smelter return royalty in respect of the Project for a purchase price of $12,000,000. Premier has 30 days from the date of notice to complete the purchase. The 1.5% net smelter royalty is not included in the cash-flow or financial calculations of this Technical Report.

Figure 4-3 Mineral Claim Map

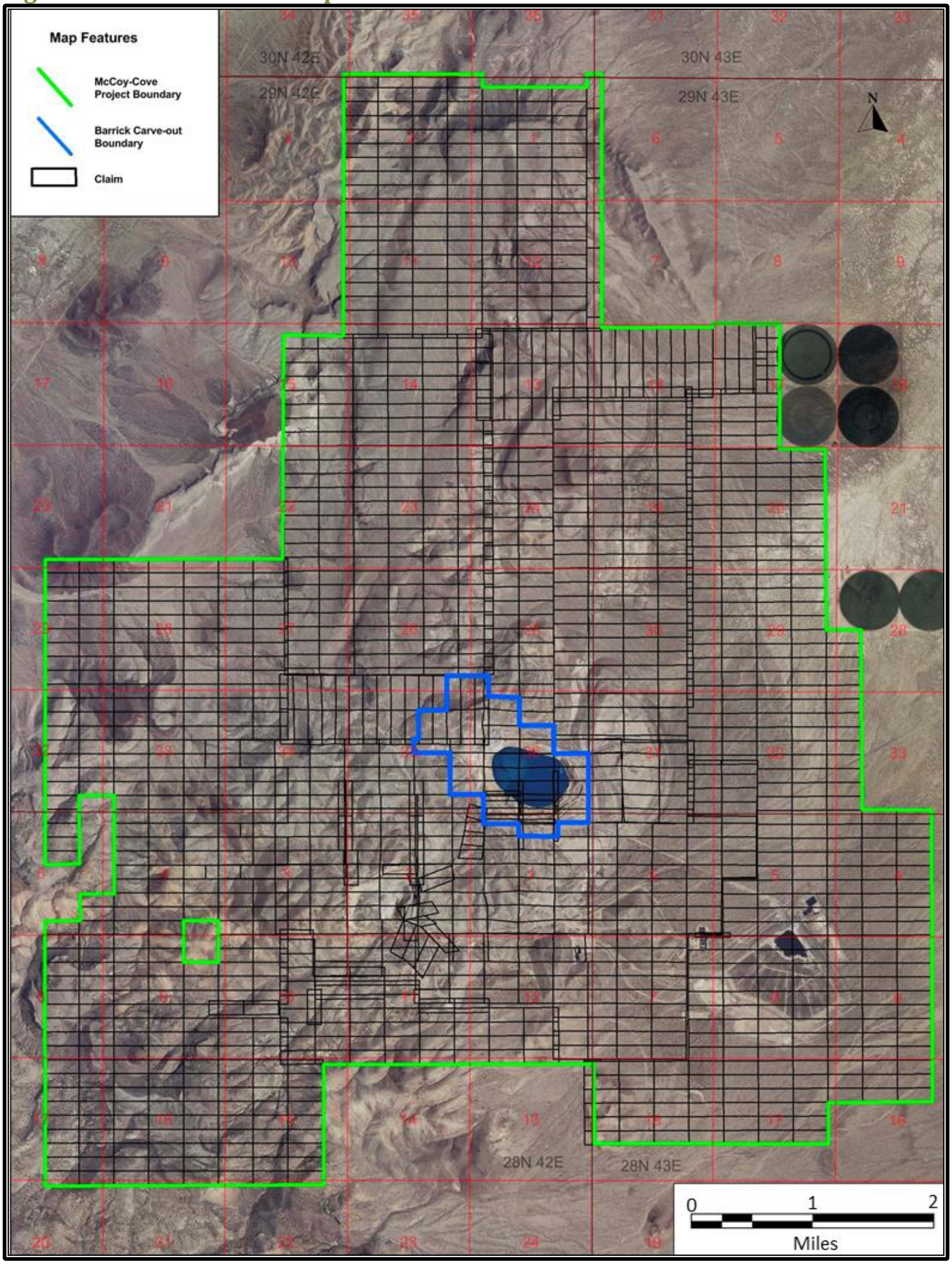
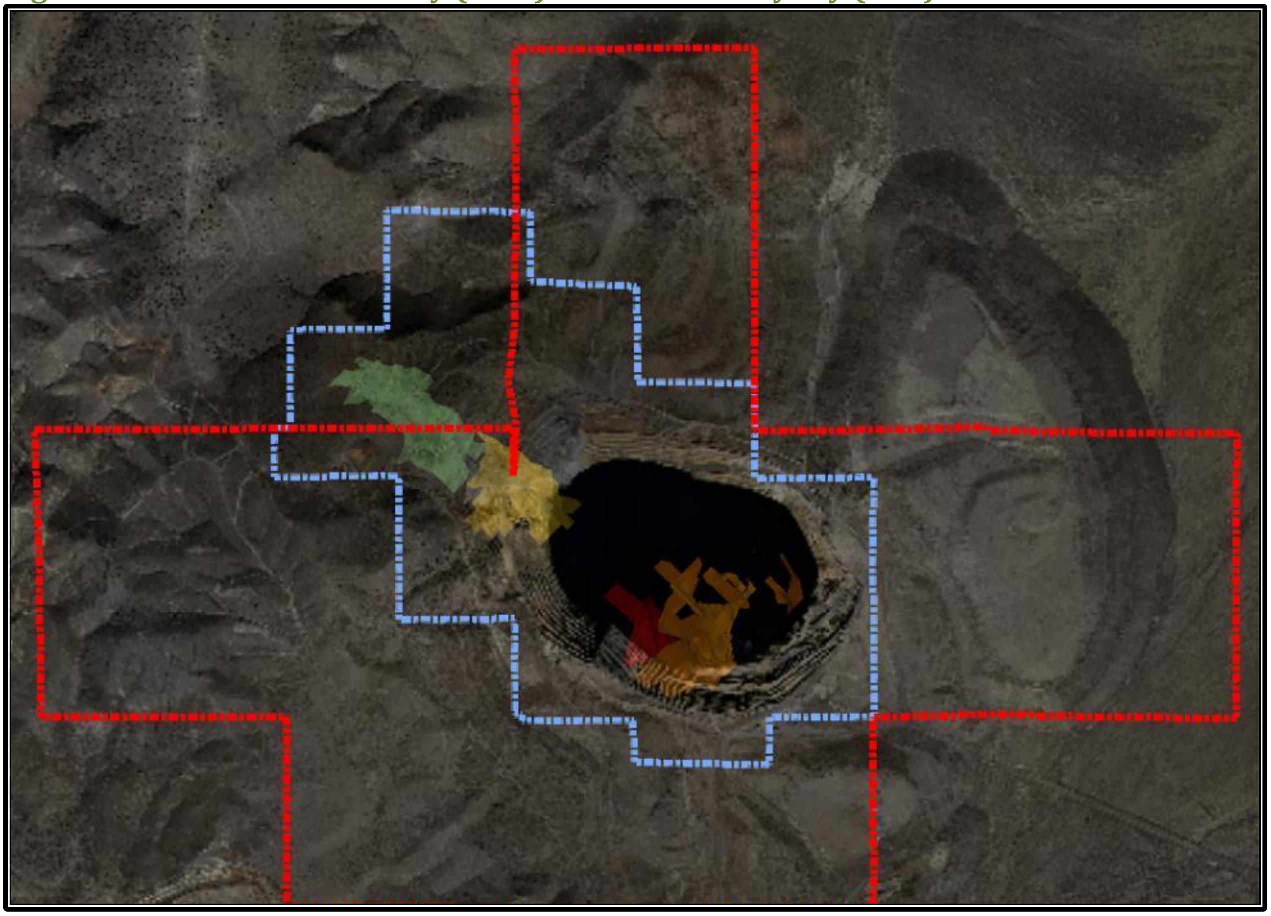


Figure 4-4 Carveout Boundary (Blue) and Summa Royalty (Red)



Not to scale.

4.3. Environmental Liabilities

The Project was under active reclamation by Newmont from 2003 to 2014. Activities include re-contouring and seeding of the dumps, leach pads, and tailings facility. All surface infrastructure outside of the maintenance shop and guard shack has been removed.

Premier is responsible for all environmental liabilities related to the closure of the McCoy-Cove Project as well as final clean-up of surface drill pads and minor drill roads. All closure activities other than evaporation of the tailings facility and water quality testing have been temporarily put on hold pending the potential for future production out of the Cove-Helen underground.

The authors are not aware of any additional environmental liabilities on the property. The authors are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

4.4. Permits/Licenses

Currently, Premier is working under the Cove-Helen Underground Exploration Plan of Operations (POO No. NVN-088795) approved in 2013. The POO authorizes Premier to complete up to 100 acres of surface exploration disturbance as well as an underground exploration decline and subsequent bulk sample of up to 120,000 tons.

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1. Accessibility

Access to the Project area is via State Highway 305, 30 miles south from the town of Battle Mountain, and then west approximately seven miles along the paved McCoy Mine Road. Battle Mountain is located on Highway 80, approximately 70 miles west of Elko, Nevada.

5.2. Climate

The climate in Lander County is typical of the high-desert environment. Average July temperatures range between 65°F and 75°F in the lower valleys and cooler in the higher elevations. Summer highs in the valleys are approximately the mid-90°F, with temperatures in the range of 50°F or 60°F at night. Winter temperatures average between 20°F and 30°F in the valleys with the possibility of frost from early September through June.

Average rainfall is 10 in to 15 in, with less than 10 in. of rain in the lowest areas and up to 20 in. occurring in the mountains. The majority of precipitation falls between November and May, with the possibility of summer thunderstorms.

Mining operations are able to continue year-round.

5.3. Local Resources

The McCoy Mining District has a long history of mining activity, and mining suppliers and contractors are locally available. Both experienced and general labor is readily available from the towns of Elko in Elko County (100 miles north and east of the Project) and Winnemucca in Humboldt County (83 miles north and west of the Project). Some services are also available in Battle Mountain (30 miles north of the Project). There are a number of mining operations in the area and as such there is always competition for employees.

5.4. Infrastructure

Dirt track access roads are located throughout the property for exploration access. The Project exploration facilities consist of a guard shack, mechanic shop and numerous shipping containers used as storage sheds in the laydown and core storage yards.

Nevada Energy (formerly Sierra Pacific) power lines run to the property at the McCoy-Cove Project. Power is available at the site from a 120 kv transmission line and a 24.9 kV distribution line.

Previous mining within the Project has left a legacy of:

- Cove open pit;
- Reclaimed leach pads;
- Tailings dam (partially reclaimed);
- Reclaimed dumps, and;
- Reclaimed infiltration basins.

All aforementioned facilities except for the tailings dam have been released by state and federal agencies and are considered reclaimed.

5.5. Physiography

The Project lies in the Basin and Range Province, a structural and physiographic province comprised of generally north to north-northeast trending, fault bounded mountain ranges separated by alluvial filled valleys.

The property is located on the northeastern side of the Fish Creek Mountains. Elevation in the McCoy Mining District ranges from about 4,800 feet to 6,900 feet above sea level. The valley in the Helen deposit area is at approximately the 4,800 feet elevation and the area overlying the deposit has an elevation of approximately 5,500 feet.

Vegetation is typical of the high desert; greasewood characterizes the salt flats, sagebrush dominates the alluvial fans, and piñon and juniper are found on the mountain slopes. Rabbit brush, white sage, and mountain mahogany are also present (Figure 5-1).

Figure 5-1 McCoy-Cove Project Area Looking Southeast

6. History

Gold was first discovered in the McCoy Mining District in 1914 by Joseph H. McCoy. Production through 1977 included approximately 10,000 ounces of gold plus minor amounts of silver, lead, and copper. Production in these early years came from placers and from gold-quartz veins that occurred in northeast striking faults and in intersections of northeast and northwest striking faults. Most of the non-placer production, however, came from argillized and oxidized skarn at what became the McCoy open pit mine.

6.1. Previous Owners

Summa Corporation (Summa), a Howard Hughes company, acquired most of the mining claims in the McCoy Mining District in the 1950s and 1960s. In 1977, Houston Oil and Minerals Corporation (Houston) purchased the McCoy-Cove Project. Gold Fields Mining Corporation (Gold Fields) leased the property in 1981 until September 1984, whereupon the property was returned to Tenneco Minerals Company (Tenneco), which had acquired Houston. Echo Bay Mines Ltd. (Echo Bay) purchased the precious metal holdings of Tenneco in October 1986. Newmont took ownership of the Cove and McCoy properties in February 2003 following the merger between TVX Gold Inc. (TVX), Echo Bay, and Kinross Gold Corporation (Kinross).

Victoria Gold Corp (Victoria) leased for the property in June 2006 as previously described in Section 4. In June 2012, Premier entered into an agreement to acquire the lease of the McCoy-Cove Project from Victoria and subsequently acquired a 100% interest in the land package from Newmont in September 2014.

6.2. Historic Exploration

Modern exploration for copper and gold in the McCoy Mining District started in the 1960s by Bear Creek Mining Company and Pilot Exploration drilling in 1967. Summa conducted extensive exploration on the McCoy skarn deposit from 1969 to 1977. Summa also undertook regional geologic mapping of 55 square miles (including the McCoy-Cove Project area) and extensive rock and chip surveys.

Houston explored the property in 1980, including geologic mapping, soil geochemical surveys, ground magnetic surveys, and drilling.

Gold Fields conducted an extensive induced polarization (IP) program, airborne magnetic surveys, detailed rock chip sampling, as well as limited geologic mapping and drilling between 1981 and 1984.

In 1985, Tenneco undertook drilling, metallurgical testing, and engineering and feasibility studies and began mining the McCoy deposit in February 1986. Tenneco also began systematic district-wide exploration in 1985 with the collection of 500 stream sediment samples from an eight-square mile area around the McCoy deposit. Evidence of what would become the Cove deposit was found in early 1986, when seven samples yielded gold values of between 15 ppb and 72 ppb with associated anomalous Ag, As, Hg, Sb, and Tl. Subsequent detailed geologic mapping identified jasperoid, manganiferous limestone, and outcrops of altered felsic dikes in the area of the anomalous samples. Surface rock chip samples of these rocks all contained significant gold mineralization. Tenneco's detailed mapping covered a large area that included both McCoy and Cove and extended to the north, west, and south. In September and October 1986, a total of 147 soil samples were collected from the B and C soil horizons over the altered area at Cove on a 100-foot by 200-foot grid.

Echo Bay continued the systematic district exploration program initiated by Tenneco that included stream sediment, soil, and rock chip sampling plus geologic mapping, exploration trenching using a bulldozer and drilling. Later soil sampling at Cove defined a gold anomaly measuring 2,800 feet long by 100 feet to 600 feet wide, with gold values ranging from 100 ppb to 2,600 ppb. Bulldozer trenching exposed ore grade rock over the entire length of this soil anomaly. Echo Bay discovered the Cove deposit with drilling in January 1987. By March 1987, Echo Bay had drilled 42 shallow exploration holes and development drilling began in late March. Echo Bay drilled 458 reverse circulation (RC) holes totaling 315,000 feet from January 1987 through June 1988 and 51 core holes totaling approximately 65,800 feet through 1989 (Briggs, 2001).

In 1999, Echo Bay drilled eight surface drill holes totaling 6,700 feet on the Cove South Deep deposit. This drilling, combined with bulk sampling from an underground exploration drift, confirmed the presence of a high-grade zone (0.25 opt Au) that could be mined by underground methods (Briggs, 2001). Detailed underground drilling of this deposit continued during 2000 as mining proceeded.

Newmont drilled 15 vertical holes on the property from 2004 to 2005. Victoria began exploring the property in 2006 resulting in the discovery of the Carlin-style Helen Zone immediately northwest of the Cove pit.

6.3. Historic Resource Estimates

Numerous estimates of historical "geological resources" and "proven and probable reserves" have been reported for the McCoy and Cove deposits. The estimates listed in Table 6-1 pre-date the introduction of NI 43-101 reporting standards and use classifications other than those set out in The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves. The assumptions, parameters, and methods used to create these historical estimates are unknown. A qualified person has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves and the issuer is not treating these historical estimates as current Mineral Resources or Mineral Reserves.

Table 6-1 Historic Resource and Reserve Estimates

Company	Date	Location	Tons (M)	Au Grade (opt)	Contained Au (000 oz)	Comments
Emmons and Coyle	1987	Cove & McCoy	50 to 70	0.065	3,000 to 5,000	Geological Resource
Kuyper et al.	1991	Cove & McCoy	53.7	0.054	2,900	Proven & Probable Reserves
Emmons and Eng	1995	Cove	-	-	3,600	Estimated in situ pre-mining Reserves
		McCoy	-	-	880	
Echo Bay Minerals Co.	1999	Cove & McCoy	11.8	0.043	500	1999 Year End Proven & Probable Reserves
Nevada Bureau of Mines & Geology	2000	Cove & McCoy	4.7	0.034	160	Proven & Probable Reserves
Nevada Bureau of Mines & Geology	2001	Cove & McCoy	0.4	0.031	12	Proven & Probable Reserves

6.4. Historic Mining

The earliest known significant mining was in the early 1930s at the Gold Dome mine, previously located on northeast side of the present McCoy open pit mine. This operation included a 250-foot shaft and five levels of workings at 50-foot intervals producing gold grades ranging between 0.25 opt and 2.0 opt.

Table 6-2 summarizes the annual production between 1986 and 2006 at the McCoy and Cove mines. Tenneco commenced mining at the McCoy open pit mine in 1986 and Echo Bay began open pit mining of the Cove deposit in 1988, accompanied by three phases of underground mining.

Underground access at the Cove Mine was via a decline with rubber-tire machines using a room and pillar mining method. From 1988 to 1993, underground mining was used to recover high grade, ore ahead of the pit. In 1999, additional underground mining at Cove South Deep (CSD) recovered approximately 300,000 tons of mineralization beyond the ultimate pit limits. The mineralization was relatively flat-lying from 10 feet to 80 feet thick. Longhole stoping and drift and fill methods were used with cemented rock fill. (CRF).

Conventional open pit mining methods were utilized at Cove open pit, with drilling and blasting of ore on 20 foot benches (double benched to 40 feet) and waste on 30 foot benches (double benched to 60 feet). The lower sulfide orebody was reached in late 1991.

Processing of low grade, run-of-mine heap leach ores from Cove began in 1992 and mining of high grade ores was completed in 1995. Open pit mining ended at Cove in October 2000.

In 1996, the mill facility was expanded from 7,500 stpd to 10,000 stpd, with milling of stockpiled ores from the Cove open pit beginning in the second half of 1997. Mill recoveries declined during the remaining life of the mine as lower grade, more refractory ores were processed. By October 2000, the mill was processing 11,369 stpd. As of that date, the gold grade was 0.055 opt Au and plant gold recovery was 51.8%; silver grade was 4.00 opt Ag and plant silver recovery was 71.5%.

The mill contained gravity, flotation, and cyanide leach circuits. Through 2006, a total of 3.41 million ounces of gold and 110.2 million ounces of silver were produced from Cove and McCoy, with the vast majority of both metals reportedly coming from the Cove deposit. Approximately 2.6 million ounces of gold were produced from the Cove open pit.

Table 6-2 Historic Cove and McCoy Mine Production 1986 through 2006

	Mineralized Material Processed			Oxide		Sulfide		Heap Leach			
Year	Milled Oxide Tons (000)	Milled Sulfide Tons (000)	Heap Leach Tons (000)	Au (opt)	Ag (opt)	Au (opt)	Ag (opt)	Au (opt)	Ag (opt)	Au Ounces	Ag Ounces
1986	-	-	1,851	-	-	-	-			34,035	na
1987	-	-	4,292	-	-	-	-	0.04	-	90,788	56,800
1988	-	-	2,994	-	-	-	-	0.053	1.14	104,009	764,116
1989	1,358	-	5,696	0.107	3.21			0.02	0.44	214,566	2,259,653
1990	2,004	201	5,709	0.084	0.82	0.227	6.17	0.021	0.2	255,044	1,982,455
1991	2,094	364	5,174	0.077	1.7	0.194	8.42	0.02	0.69	284,327	5,619,007
1992	1,483	990	9,029	0.075	2.54	0.163	7.57	0.014	0.6	301,512	7,921,496
1993	2,308	552	8,938	0.107	4.61	0.136	4.65	0.017	0.88	395,608	12,454,338
1994	506	2,304	7,892	0.126	6.71	0.143	4.91	0.013	0.48	359,360	10,443,151
1995	497	2,151	4,355	0.15	5.42	0.104	5.23	0.018	0.49	310,016	11,905,806
1996	-	3,287	6,068	-	-	0.086	3.14	0.018	0.27	271,731	7,102,348
1997	-	3,391	6,494	-	-	0.061	4.54	0.018	0.29	187,034	11,021,708
1998	-	4,306	4,112	-	-	0.046	2.95	0.021	0.26	167,494	9,412,823
1999	-	4,452	4,178	-	-	0.038	3.02	0.022	0.37	124,536	8,430,072
2000	-	4,172	1,809	-	-	0.053	3.71	0.024	0.93	162,784	12,328,297
2001	-	-	-	-	-					94,633	6,451,425
2002	-	-	-	-	-					33,142	1,987,421
2003	-	-	-	-	-					4,699	706
2004	-	-	-	-	-					8,454	64,335
2005	-	-	-	-	-					2,740	776
2006	-	-	-	-	-					2,939	596
Total	**10,250**	**26,170**	**78,591**	**0.10**	**2.93**	**0.08**	**3.98**	**0.02**	**0.48**	**3,409,451**	**110,207,329**

7. Geologic Setting and Mineralization

7.1. Regional Geology

The McCoy-Cove Project is located in the central Nevada portion of the Basin and Range Province, which underwent regional extension during the Tertiary that created the present pattern of alternating largely fault bounded ranges separated by alluvial filled valleys (Figure 7-1). Prior to this extension, central Nevada had been the site of numerous tectonic events, including at least two periods of regional compression. The property lies west of the central part of the Battle Mountain-Eureka Trend.

During the Paleozoic, central Nevada was the site of the generally north-northeast trending continental margin of North America, along which pre-orogenic rocks of Cambrian to Early Mississippian age were deposited. A carbonate platform sequence was deposited to the east along the continental margin, with siliceous and volcanic rocks deposited to the west. In Late Devonian to Early Mississippian time during the Antler Orogeny, rocks of the western assemblage moved eastward along the Roberts Mountains thrust, perhaps as much as 90 miles over the eastern assemblage carbonate rocks. A post-orogenic assemblage of coarse clastic sedimentary rocks of Mississippian to Permian age was shed eastward from an emerging highland to the west, overlapping the two earlier facies.

During Pennsylvanian and Permian time, chert, pyroclastic rocks, shale, sandstone, conglomerate, and limestone of the Havallah sequence were deposited in a deep eugeosynclinal trough to the west of the Antler orogenic belt. These rocks were thrust eastward along the Golconda thrust over the Antler overlap assemblage in Late Permian and Early Triassic time during the Sonoma Orogeny. The Golconda thrust is exposed to the west of the Roberts Mountains thrust.

Mesozoic rocks, primarily shallow water siliciclastic and carbonate units with minor volcanic and volcaniclastic rocks, are found in this part of Nevada. At least three additional tectonic events are recorded in late Paleozoic and Mesozoic time, including the formation of the late Jurassic Luning-Fencemaker fold and thrust belt in western and central Nevada. The most recent events in the Great Basin are widespread Cenozoic volcanism and extensional faulting. Late Jurassic (168-143 Ma), Cretaceous (128-90 Ma), and Eocene to Oligocene (43-30 Ma) intrusions have been reported from this part of Nevada.

7.2. Local Geology

The stratigraphy of the McCoy Mining District is well documented, and has been described in detail by Emmons and Eng (1995) and Johnston (2003). Generalized Triassic stratigraphy of the local area is presented in Figure 7-2 and the major lithological units are described below.

HAVALLAH FORMATION

The Permian Havallah Formation is the deepest drilled unit on the property and is composed of reddish-brown to green argillite and chert. Where it hosts veins, the Havallah displays alteration envelopes containing fine-grained quartz-illite/sericite. The total thickness of the Havallah across the property is unknown. Its contact with the overlying Dixie Valley Formation is sometimes offset by clearly defined reverse faulting and demarcated by the presence of an unconformable rhyodacite tuff (assumed to be Koipato Formation), while in other areas of the property, it is simply defined by the change from coarse-grained clastic conglomerates and sedimentary breccias to argillite.

KOIPATO FORMATION

Locally, at the contact between the Dixie Valley Formation and the Havallah, there is a maroon rhyodacite tuff assumed to be part of the Permo-Triassic Koipato sequence described by Silberling and Roberts (1962). The upper and lower contacts of this rhyodacite tuff are unconformities.

DIXIE VALLEY FORMATION

The early Middle Triassic Dixie Valley Formation consists primarily of coarse-grained conglomerates and intercalated dolomitic sandstones, as well as lesser fossiliferous limestone units generally restricted to the upper portion of the formation.

FAVRET FORMATION

The late Middle Triassic Favret Formation, approximately 750 feet thick, consists of an upper fossiliferous limestone unit containing ammonites and pelecypods, a middle unit of finely interbedded silty limestones and limestones (principal Carlin-style ore host), and a basal unit of debris flow fossil hash containing ammonites, pelecypods, and star-shaped crinoids.

AUGUSTA MOUNTAIN FORMATION – HOME STATION MEMBER

The late Middle Triassic Home Station Member is 100 feet to 150 feet thick and was previously described as a thicker unit consisting of massive calcareous and dolomitic limestone with lenses or beds of sandstone and conglomerate (Kuyper et al., 1991). Johnston (2003) however, classified this unit as silty dolostones based on exposures in the Cove open pit which displayed medium to dark grey, very thickly bedded (greater than 3 feet) dolostone consisting of three to 25 volume percent quartz grains (averaging 0.0016 in. diameter) in a recrystallized dolomite matrix. The clastic components of Kuyper et al.'s (1991) Home Station are now classified as Panther Canyon and the lower limestone is now considered the upper part of the Favret Formation. Although the contact between the Home Station Member and the overlying Panther Canyon Member was described as gradational by Kupyer et al. (1991), Johnston (2003) mapped the contact in the Cove open pit as sharp, and Premier geologists use a prominent lag gravel deposit (generally less than 15 feet to 20 feet thick) to mark this contact.

Figure 7-1 Regional Geology

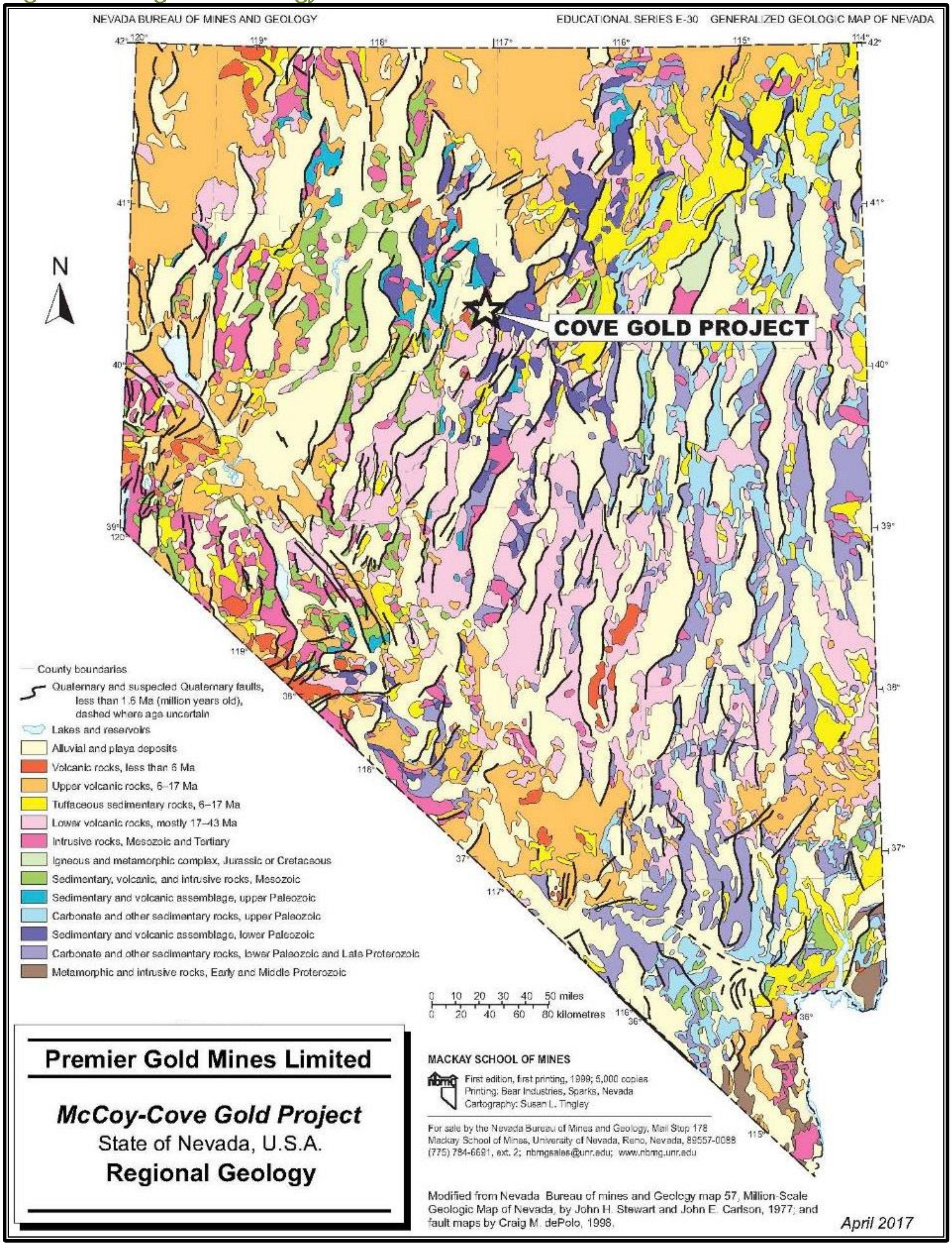
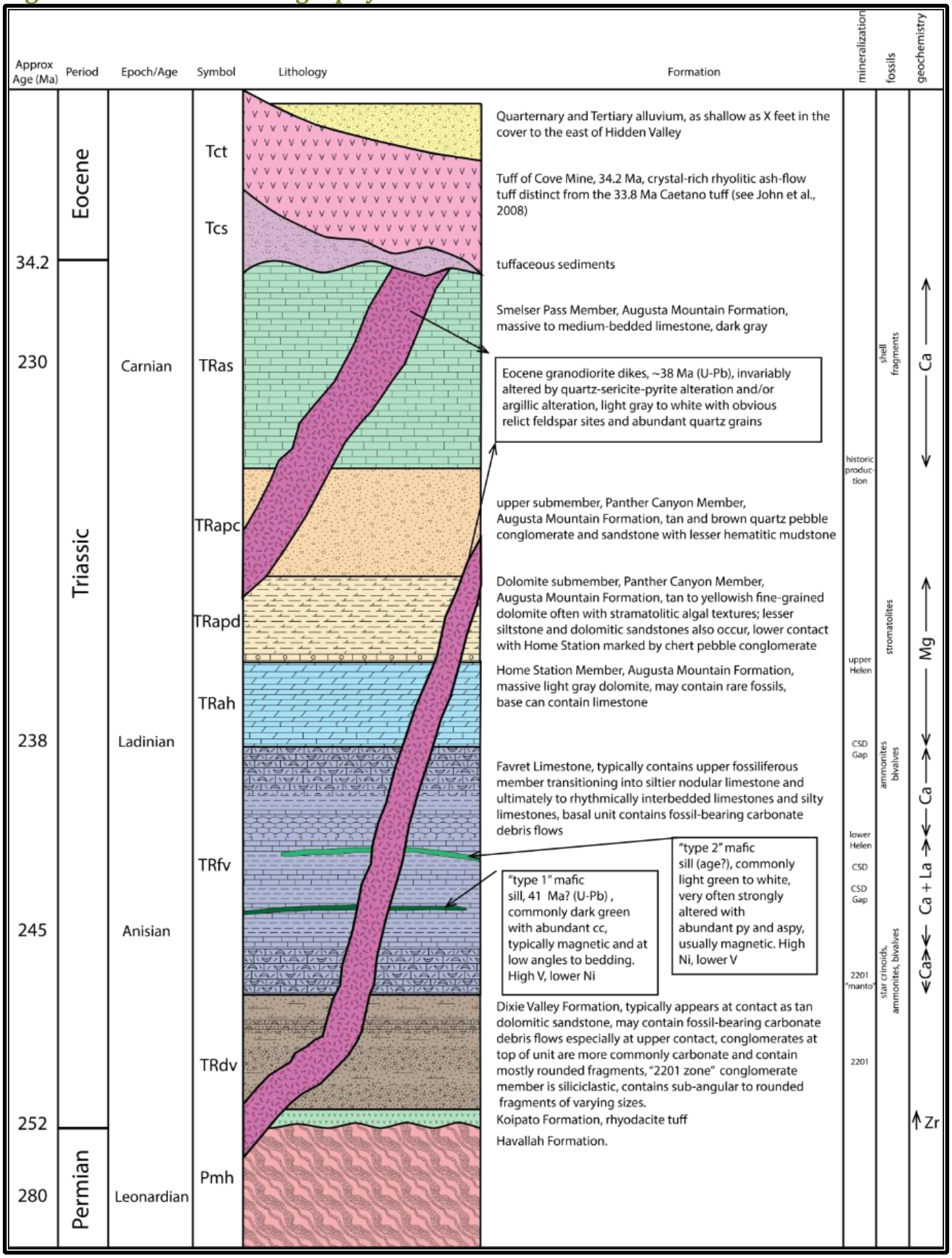


Figure 7-2 Triassic Stratigraphy and Mineralization



AUGUSTA MOUNTAIN FORMATION – PANTHER CANYON MEMBER

The Panther Canyon Member at Cove is divided into two informal units, the lower Dolostone Sub member and upper Transitional Sub member.

The lower Dolostone Sub member unit is generally 50 feet to 75 feet thick and consists of a well bedded, medium grey dolostone. Individual beds are typically less than three feet in thickness. This unit is a primary dolostone and commonly has stromatolitic algal textures (Emmons and Eng, 1995). Johnston (2003) noted that quartz grains (0.001 in. diameter) locally constitute up to 20 volume percent and that the contact with the overlying Transitional Sub member is very gradational over a distance of approximately 10 feet.

The upper Transitional Sub member is a 500 feet thick unit which coarsens upward, from a basal primary dolostone, through middle silty and sandy dolostone and carbonate cemented silt- and sandstone, to conglomerate near the top. The general transition is not smooth, however, as contrasting lithologies are interspersed throughout the unit at all levels, typically as lensoid bodies. This Transitional Sub member can be further separated into a lower carbonate rich and an upper clastic section as follows:

- Lithologies in the 165 feet thick lower carbonate rich section are highly variable. Although the strata are primarily made up of dolostone, lenses, and beds of carbonate cemented siltstone and very fine grained sandstone, coarser sandstone and conglomerate are abundant. The lower 80 feet of this section consists principally of massive dolostone. Typical strata in the upper 80 feet of this section consist of 0.001 in. to 0.003 in. diameter, subrounded, moderately sorted quartz grains. Individual beds are typically less than 3.3 feet in thickness. The diagenetic cement is calcite, but it has been dissolved and/or replaced by illite-sericite where hydrothermally altered.
- The 300 feet thick upper clastic section in the Transitional Submember generally consists of fine grained sandstone to cobble conglomerate. The thickness of bedding is highly variable, but the conglomerate beds are generally thicker (up to 16 feet thick) than the sandstone beds (up to 3.3 feet thick). Cross-bedding is common, and conglomeratic strata typically grade upwards from relatively coarse to relatively fine grained sediments. Detrital grains and cobbles consist of chert, quartzite, and quartz. These grains are rounded to subrounded and moderately sorted. Primary porosity, which was originally high, ranges up to 20 volume percent as observed by Johnston (2003). The contact with the overlying Smelser Pass Member is gradational over several tens of feet.

AUGUSTA MOUNTAIN FORMATION – SMELSER PASS MEMBER

The Smelser Pass Member unit is volumetrically the largest at Cove with a maximum thickness of just over 900 feet. The unit is predominantly a microcrystalline limestone with abundant recrystallized bioclasts, however, the upper 500 feet contain very minor thin interlaminated calcareous shale beds. The limestone is thick bedded to massive, with individual beds ranging from three feet to 16 feet in thickness. Macro allochemical remains consist of partial to complete brachiopods, pelecypods, gastropods, crinoids, corals, sponges, and ammonites, in decreasing order of abundance. The lowermost beds contain up to 15 volume percent of 0.0006 in. diameter quartz grains.

The Smelser Pass Member is separated from the overlying Oligocene tuffaceous sediments and Tuff of Cove Mine by an angular unconformity. Kuyper et al. (1991) determined that the upper 575 feet of the Smelser Pass were removed by erosion prior to deposition of the Oligocene units. More than 2,100 feet of the Triassic Cane Spring and Osobb Formations, which overlie the Smelser Pass Member elsewhere in the McCoy Mining District, are also missing at Cove. Much of the Smelser Pass Member has been subjected to supergene oxidation, giving the originally medium grey limestone an orange to brown appearance.

TUFF OF COVE MINE

The tuff of Cove Mine, previously thought to be the 33.8 Ma Caetano Tuff, has a maximum thickness of approximately 1,500 feet in the deepest parts of the paleovalley it filled. It consists of 0.016 in. to 0.276 in. long fragments of plagioclase, biotite, potassium-feldspar, and resorbed quartz phenocrysts in a glassy to devitrified matrix. Phenocrysts comprise 40 volume percent and matrix 60 volume percent of the rock. John et al. (2008) reported a 40Ar/39Ar age of approximately 34.2 Ma on a set of samples including some collected in the northern Fish Creek Mountains.

INTRUSIVE IGNEOUS ROCKS

Abundant dikes and sills are encountered in drilling at Cove, and historic convention at the property has been to classify them as ether “felsic” or “mafic.” The majority are “felsic” and can be mapped at surface associated with and occupying the main faults extending from the Eocene Brown stock at McCoy. Though commonly altered, their textural similarities to the unaltered granodioritic feldspar porphyry of the Brown Stock suggest that they were of similar composition. These dikes are light grey to white in colour due to sericitic or argillic alteration. Their porphyritic texture is preserved. They may be observed over drill hole intercepts ranging in length from less than 0.5 to 215 vertical feet and are usually steeply dipping. Less altered samples collected from the Cove open pit retain evidence for secondary biotite replacing hornblende suggesting a weak potassic alteration event that has been overprinted by lower temperature alteration events at depth. The Gold Dome is the most prominent "felsic" dike at the deposit and is cross-cut by both polymetallic veins and pervasively altered by weak Carlin-style mineralization.

As a result of the intense alteration, many occurrences of rocks of different composition have been incorrectly logged as "felsic." Multi-element geochemistry from Premier's data were used in 2016 to reclassify all igneous rocks by filtering for high occurrences of Cr, Ni, and V. When the reclassified lithologies were remodelled in 3D it became apparent that the mafic intrusive rocks are present as thin, laterally extensive, stacked sills that terminate down the northeast limb of the Cove anticline. As a result of that exercise, two distinct trends were discovered in the Ni and V concentrations of these mafic dikes and sills. Whole rock geochemistry and subsequent remodelling confirms the presence of two distinct mafic compositions. These are classified as "type 1" characterized by high V and lower Ni and "type 2" characterized by low V and higher Ni. "Type 1" in drill core is typically dark green in colour, contains abundant calcite, and may be magnetic. Though the "type 1" sills have a strong spatial association to Carlin-style mineralization across the deposit, they are rarely mineralized and can be devoid of As, Au, and Ag in direct contact with mineralized limestone. The "type 2" sills are generally light green to white in colour and can be difficult to distinguish from similarly altered "felsic" dikes. They appear to have been hornblende-biotite porphyries prior to alteration and commonly contain magnetite. They also share a spatial association to Carlin-style mineralization but, unlike "type 1" sills, are very commonly mineralized (up to 20 ppm Au, 20 ppm Ag). "Type 2" sills are less prevalent overall than "type 1" sills, and concrete cross-cutting relationships between any of these three intrusive rocks have thus far been elusive.

QUATERNARY ALLUVIUM

Emmons and Eng (1995) divided the Quaternary surficial units in the McCoy Mining District into alluvium, talus, and colluvium. Quaternary sediments exposed in the Cove open pit were not differentiated in this study. These sediments include unconsolidated sand and gravel, and are less than 215 feet thick.

7.3. Structural Geology

Deposits on the McCoy-Cove Project are related to specific structural features.

MAJOR DEFINING STRUCTURES

The major structure and control on fluid movement is the broad northwest-striking, gently southeast-plunging Cove anticline interpreted as a fault propagation fold over a deep northwest striking reverse fault identified in deep drill holes under the Cove pit. While the reverse fault can be identified in the 2201 zone, its presence at the Gap and Helen Zones is uncertain due to limited drilling in areas that would confirm its continuation. A northwest striking vertical dike called the Northwester Dike (classified as "type 2") extends from the Bay fault through the Gap and into the Helen. It appears to prohibit the flow of mineralizing fluids to the southwest in areas between the major northeast striking faults. Though there is no discernible separation on the dike, it may be related to a near vertical to steeply southwest dipping fault mapped in the pit by Echo Bay geologists called the Northwester fault.

The other major structures for fluid movement and mineralization are a number of northeast striking normal faults (Cay, Blasthole, Bay, 110, Gold Dome, and Norm). The northeast striking faults commonly host altered granodioritic dikes, the largest of which is the Gold Dome. The north-south striking Lighthouse fault also contains altered granodioritic dikes and is believed to have had both pre- and post-mineralization movement. Defining the northern extent of the Helen is a west- northwest striking fault called the B fault south of which is the east-west striking A fault. The A and B faults form a well mineralized "wedge" of high grade Au that requires additional testing along strike to fully realize the deposit's potential.

These faults and structures were defined and confirmed by:

- Surficial and open pit geologic mapping by Echo Bay, Victoria, and Premier;
- Offset observed during detailed cross section work by Premier in 2016, and;
- Oriented core measurements by Victoria and Premier, especially in the Helen and Gap.

7.4. Mineralization Controls

Carlin-style mineralization appears to be controlled by a combination of the axis of the Cove anticline, normal faults that cut the anticline, mafic sills and dikes throughout the property, and contacts between different sedimentary units. Generally, the highest grades are found where the rhythmically bedded unit of the Favret Limestone is cut by mafic dikes and sills along the axis of the anticline, and especially where this area is cut by apparent small-scale, unmapped faults. Lower-grade (0.05 opt to 0.25 opt Au) Carlin-style mineralization in the Helen and Gap zones is typically found along the Favret-Home Station contact and the contact between the Panther Canyon's upper conglomerate unit and lower dolomite unit.

The northeast striking faults commonly contain quartz-sericite-pyrite and argillic altered granodioritic dikes that carry low to anomalous values of Au and Ag. Carlin-style mineralization in the Favret and other units is typically bounded by these northeast structures with higher grades focused in the axis of the anticline and lower grades with associated pathfinder elements (As, Sb, Tl, Hg, etc.) typically along the margins of the anticline as well as immediately adjacent to these major structures.

In the 2201 zone, structural controls are poorly defined, however, vein-bearing Au occurrences do trend northwest and may be related to structures formed in the hanging wall of the deep-seated reverse fault or to the near vertical to steeply southwest dipping Northwester fault. .

7.5. Post Mineral Faulting

There is at least one instance of significant post-mineral faulting. The Striper Splay is believed to be a splay off of the Lighthouse fault which is known to have both pre- and post-mineralization movement. It dips steeply northeast and strikes approximately 320° along the northeast limb of the Cove anticline causing significant post-mineral normal displacement before terminating against the Bay/110 fault complex. The overlying volcanics are not significantly faulted, as defined by holes NW-1, NW-2 & 2A, and NW-3.

7.6. Mineralization

There are four distinct mineralization types known on the property: Carlin-style, polymetallic sheeted veins, carbonate replacement (Manto) and skarn. The Helen, Gap and CSD deposits are Carlin-style deposits while the 2201 zone is comprised of steeply dipping polymetallic sheeted veins.

CARLIN-STYLE (AU-AG)

> The gold in Carlin-style deposits is usually sub-micron in size and generally occurs in pyrite and arsenical pyrite. An envelope characterized by decalcification, silicification, and argillization accompanied by anomalous amounts of silver, arsenic, antimony, thallium, and mercury often accompanies mineralization. The Carlin-style mineralization at Cove is relatively rich in silver compared to similar deposits elsewhere in northern Nevada (Johnston, 2003). When Carlin-style mineralization occurs in the silty limestones and packstones of the Favret Formation and Home Station Dolomite, decarbonatization replaces fine-grained calcite and/or dolomite with quartz and forms very fine-grained illite and pyrite. Diagenetic pyrite was probably present in the Helen Zone before Carlin-style mineralization based on the abundant presence of subhedral pyrite grains that bear no arsenian rims. The arsenic-bearing pyrite precipitated as a product of Carlin-style mineralization in the Helen are fine-grained (~10 microns) patchy, anhedral "fuzzy" pyrite generally smaller than the diagenetic pyrite grains. In the CSD zone, most pyrite grains in high-grade samples are larger (~20 microns), display spectacular, sharp geochemical

zonations, and are rimmed with arsenian pyrite or stoichiometric arsenopyrite. The few samples studied from the Gap under the SEM suggest it shares more in common with the CSD zone though its silver content is lower overall.

POLYMETALLIC SHEETED VEINS (AU-AG±PB-ZN)

The polymetallic veins in the 2201 zone are enveloped by a zone of illitic of the conglomerate matrix detected by sodium cobaltinitrate staining and confirmed by scanning electron microscope (SEM) analysis. Minor silicification is relatively common, especially in the conglomerate, however, it is not present everywhere and not always directly associated with mineralization.

CARBONATE REPLACEMENT (AG-PB-ZN±AU)

Carbonate replacement mineralization occurs as local pods of manto-style mineralization characterized by massive sulfide (pyrite-sphalerite-galena) replacing basal limestone at the Dixie Valley/Favret contact. Mineralization is discontinuous and generally defined by high-grade Ag-Zn-Pb±Au.

SKARN (AU-AG±-CU)

Skarn mineralization at the historic McCoy pit occurs as both endoskarn and exoskarn mineralization characterized by a predominantly garnet-diopside-magnetite mineral assemblage.

The Carlin-style mineralization across the deposit appears to represent an evolving system from a “primary” endmember represented by the CSD zone with higher Ag/Au, coarser-grained pyrite, and a close proximal relationship to Ag-Pb-Zn-(Au) mineralization to the “evolved” endmember represented by the Helen Zone with lower Ag/Au, very fine-grained pyrite, and weak spatial association with any other styles of mineralization. The Gap can be considered a “transition” zone between the two endmembers until more petrography is conducted on the recently discovered Gap to test this hypothesis. Helen Zone geochemistry is distinct from the CSD zone in many ways. For samples greater than 1 ppm Au, less than or equal to 100 ppm Ag, and confirmed to be Carlin-style mineralization by core photo review, the Helen Zone has an average Ag/Au ratio of approximately 0.85 whereas the CSD zone is 2.25. Gold in both the Helen and CSD zones correlates with As, Sb, and Hg, however, Au correlates moderately (0.52 correlation coefficient) with Ag in the CSD zone but more weakly (0.3652 correlation coefficient) in the Helen Zone.

Like the geochemistry, the mineralization in the Helen and CSD is also distinct. The As-bearing (assumed to also be Au-bearing) pyrite in the Helen are generally finer-grained, less euhedral, and more poorly zoned than the As-bearing CSD zone pyrite. Helen pyrite overall have lower As content – ranging from just at detection limit (~0.3 wt% to 0.5 wt%) to 2.1 wt% – than the CSD zone which contains pyrite with arsenic contents ranging from detection limit to 6 wt%. The SEM-EDS system first detected trace elements such as Te, Tl, Hg, Sb, and even Au and Ag in CSD zone pyrite, while electron microprobe analysis confirmed the presence of Au, Ag, As, Tl, Hg, Sb, and Pb in CSD mineralization. Other pyrite in the CSD zone contain fewer trace elements but still display complex elemental zoning and growth patterns visible only in backscatter electron imaging. The complicated nature of the mineralized pyrite at the CSD zone is suggestive of a more complex and long-lasting mineralizing event in comparison to the seemingly simple Helen mineralization.

In the 2201 zone, Au correlates with Ag, As, Cu, Fe, Pb, Sb, and Zn – a distinctly different grouping of elements from the CSD, Gap, and Helen Zones. The 2201 zone veins typically occur as sheeted veins and range in thickness from 0.1 cm to 6.5 cm and contain both quartz and carbonate minerals as gangue. Generally, the calcite and dolomite-dominant veins are shallower and thinner whereas the quartz (-carbonate)-bearing veins are deeper and can reach widths of 15 cm. The sulfides are mostly pyrite, sphalerite, and galena with arsenopyrite, chalcopyrite, and pyrrhotite also locally present. Visible gold is mostly limited to the thicker veins and is always observed along the margins with coarse-grained quartz. When microscopic, the gold is present as electrum with approximately 15 wt% Ag (measured on SEM-EDS) and hosted within sulfides such as chalcopyrite or arsenopyrite. Galena may also carry up to 10 wt% Ag. An oriented hole drilled in 2014 (PG14-23) provided some structural data for the vein-type mineralization. There were no trends for veins grouped by gangue or thickness, however, when grouped by depth, the data show that veins shallower than 1,750 feet generally strike northeast-southwest with varying dips and veins deeper than 1,900 feet generally strike northwest-southeast and dip steeply in both directions.

8. Deposit Types

The Cove-Helen deposit consist of two mineralization styles, Carlin-style and polymetallic sheeted veins, as outlined in Section 7 of the report. The Carlin-style mineralization within the Helen, Gap, and CSD zones comprises approximately 85% of the existing resource with high gold and silver grades occurring as both stratabound and structurally controlled mineralization at the intersection of the Cove anticline and favorable lithologic beds, structures, intrusive dikes and sills.

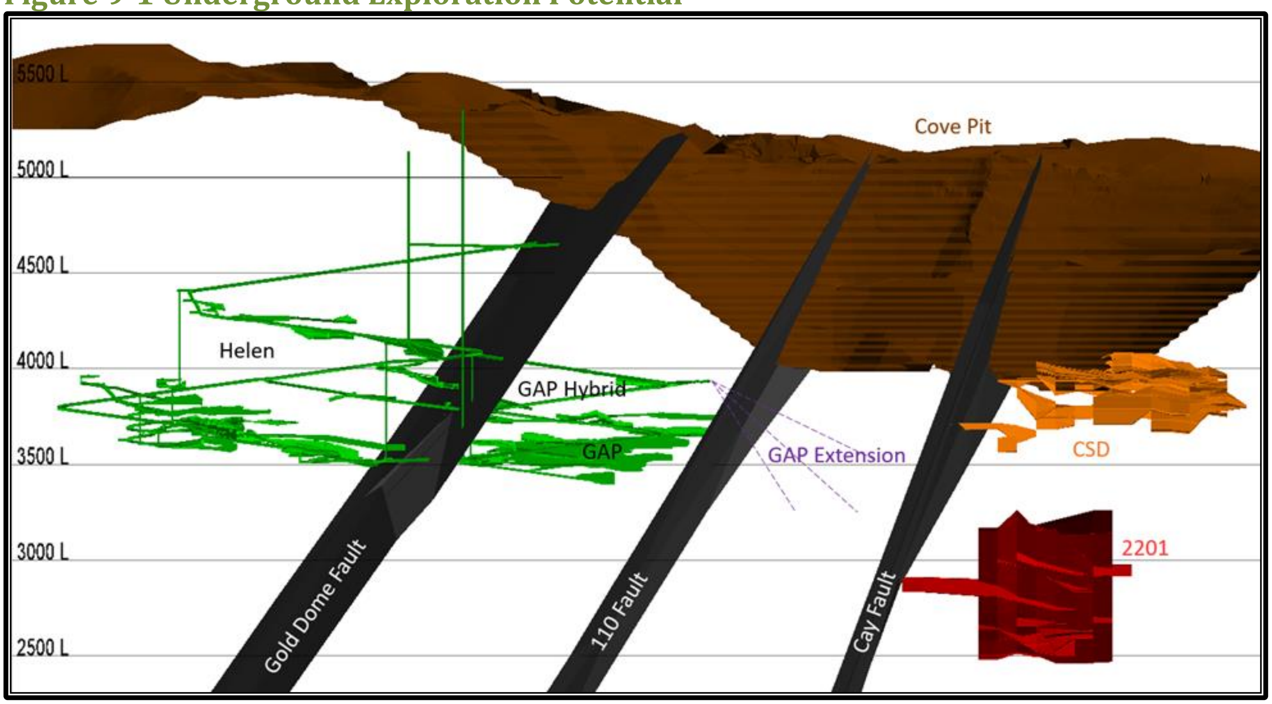
The polymetallic 2201 zone is a separate deposit from the shallower Carlin-style mineralization and is believed to be a structurally controlled sheeted vein system. Veining is oriented northwest, with vein geometry being controlled by a deeper northwest striking reverse fault. Due to its depth, the 2201 zone has seen limited drilling since its original discovery in late 2013, however, additional infill and step-out drilling in the future will help to better define deposit potential and mineralization controls.

9. Exploration

The larger McCoy-Cove claim block area is an advanced-stage property with an extensive history of exploration and production. Historical exploration from the 1960's to 2012 included stream sediment (silt) sampling, soil sampling, rock chip sampling, geophysical surveys, and geologic mapping. Since 2012, the structural geometry of the "plunge tube" model as proposed by Victoria has been disproven and replaced with the litho-structural model proposed by Premier. The re-interpretation of the litho-structural model resulted in expansion of the Cove-Helen deposit's known mineralized zones. The updated litho-structural model has guided property-wide target generation which Premier has investigated using soil sampling, field mapping, and geophysics. Highlights of exploration from 2013 to present include the discovery of the 2201 and Gap zones as well as the re-interpretation of the Helen Zone resulting in improved continuity throughout the zone. The 2018 exploration program has consisted of select infill drilling as well as large step-out exploration holes on the Windy Point and Lakeside/Lighthouse targets.

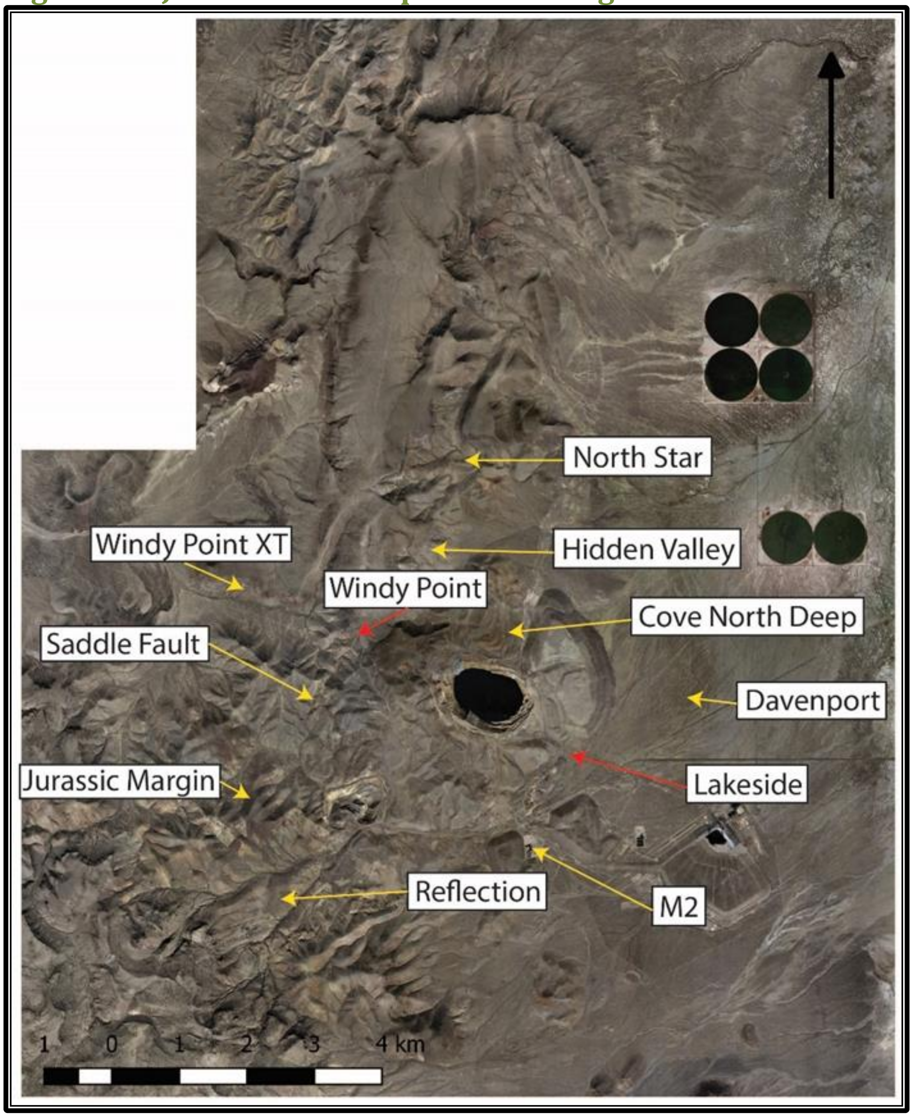
Underground mine development will provide drill platforms for infill and exploration drilling, including access to the difficult-to-reach prospective Gap Extension target under the Cove pit (Figure 9-1).

Figure 9-1 Underground Exploration Potential



In January 2018, Premier and Barrick entered into an exploration and production agreement which includes a significant exploration budget commitment from Barrick to be spent on the McCoy-Cove property. Exploration on the Joint Venture Property began in mid-2018 and will include detailed surface mapping, soil sampling, gravity survey, and drilling to primarily test the Helen XT/Windy Point and Lighthouse XT/Lakeside targets.

Figure 9-2 Joint Venture Exploration Targets outside the 100% Premier Carveout



10. Drilling

The McCoy-Cove drill hole database is large, containing many holes drilled across the large land package. For the current resource estimate, the drill data was filtered to contain only holes within and near the Helen, CSD, CSD-Gap, Gap Hybrid and 2201 Zones. A total of 1,397 holes totaling 1,127,481 feet of drilling were included in the current estimate. Holes were drilled using both core and reverse circulation (RC) methods. Premier drilled 123 of the holes, and the remainder were drilled by Victoria, Newmont and Echo Bay. Figure 10-1 shows a plan view of the drill holes, and Table 10-1 lists the type and extent of drilling completed by each operator.

Figure 10-1 Plan View of Drill Holes Used for the Current Analysis

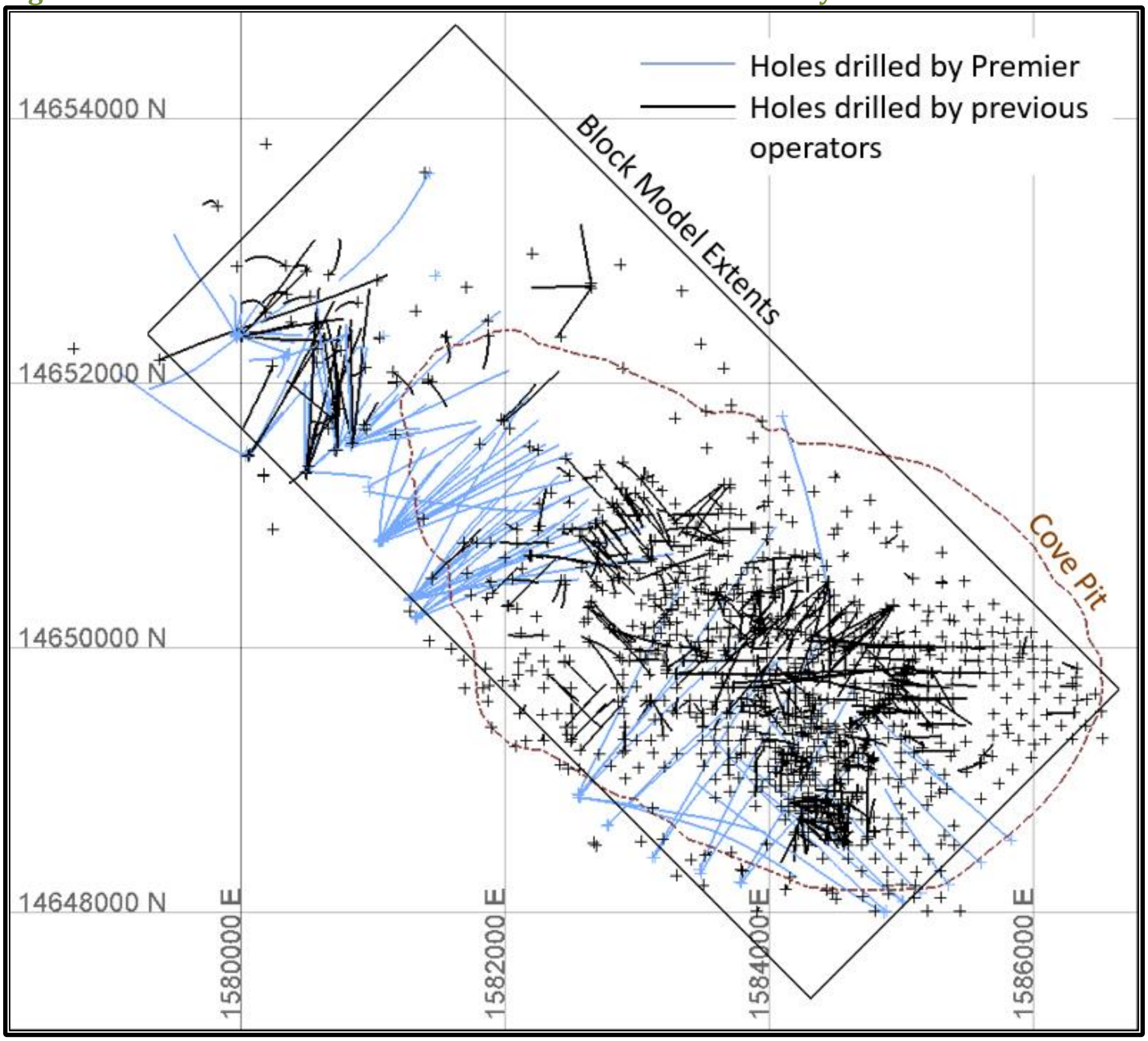


Table 10-1 List of Drilling by Operator

Year	Drill Hole Type	Operator	Number of Holes	Length Drilled (ft)
1985-2000	Reverse Circulation	Echo Bay	788	520,194
1999-2000	Cubex (RC)	Echo Bay	201	22,829
1987-2000	Diamond Drill	Echo Bay	251	216,059
2004-2005	Reverse Circulation	Newmont	13	22,080
2006-2009	Diamond Drill	Victoria	21	47,118
2013-2017	Reverse Circulation	Premier	8	14,340
2012-2018	Diamond Drill	Premier	115	284,862
Total			**1,397**	**1,127,481**

Figure 10-2 through Figure 10-5 show 100-foot thick sample sections of drilling in the CSD-Gap, Helen, CSD and 2201 zones. Holes drilled by Premier are labeled and shown with thicker traces. Models of lithologic surfaces and 3-gram grade polygons are shown for reference.

Figure 10-2 Sample Section of CSD-Gap and Gap Hybrid Drilling

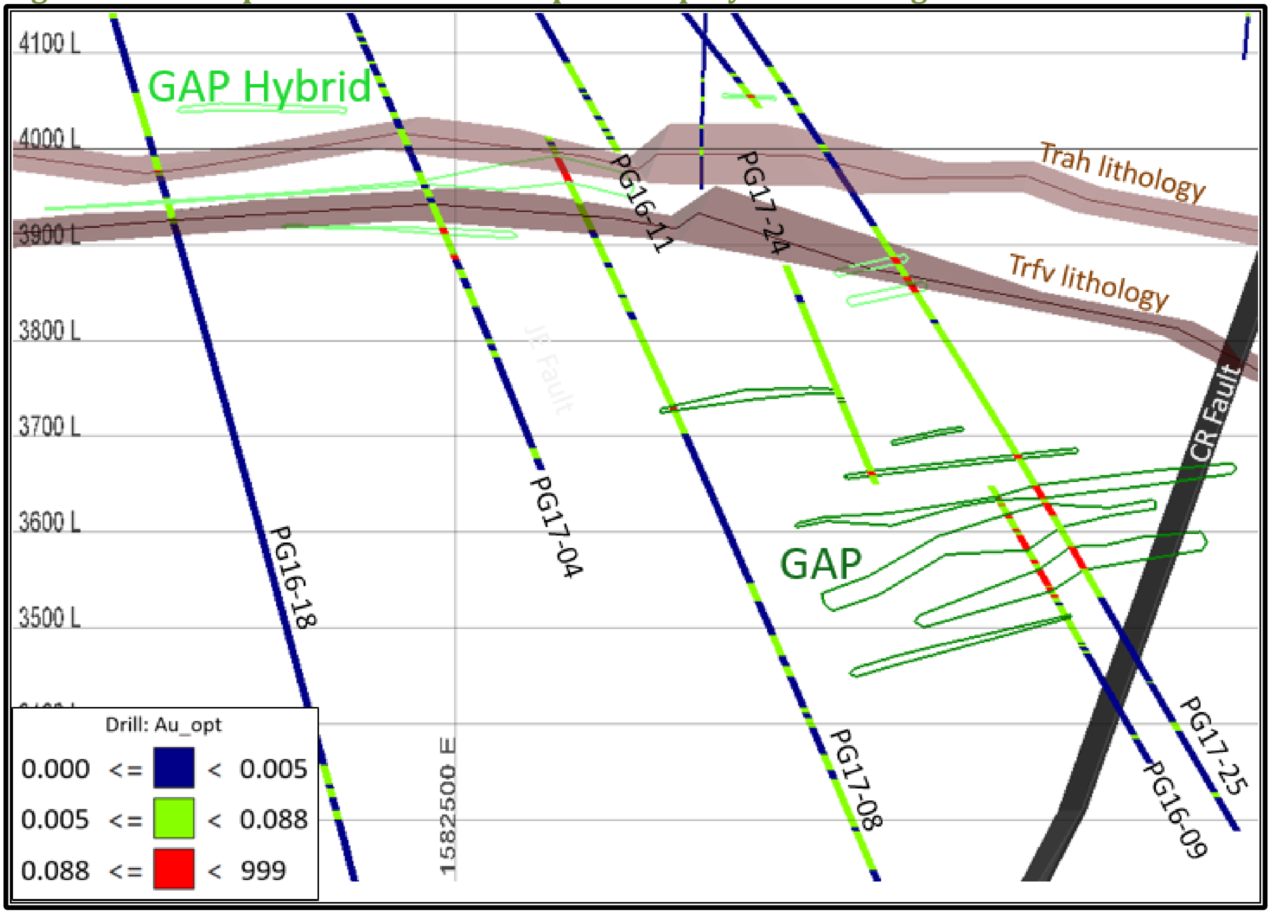


Figure 10-3 Sample Section of Helen Zone Drilling

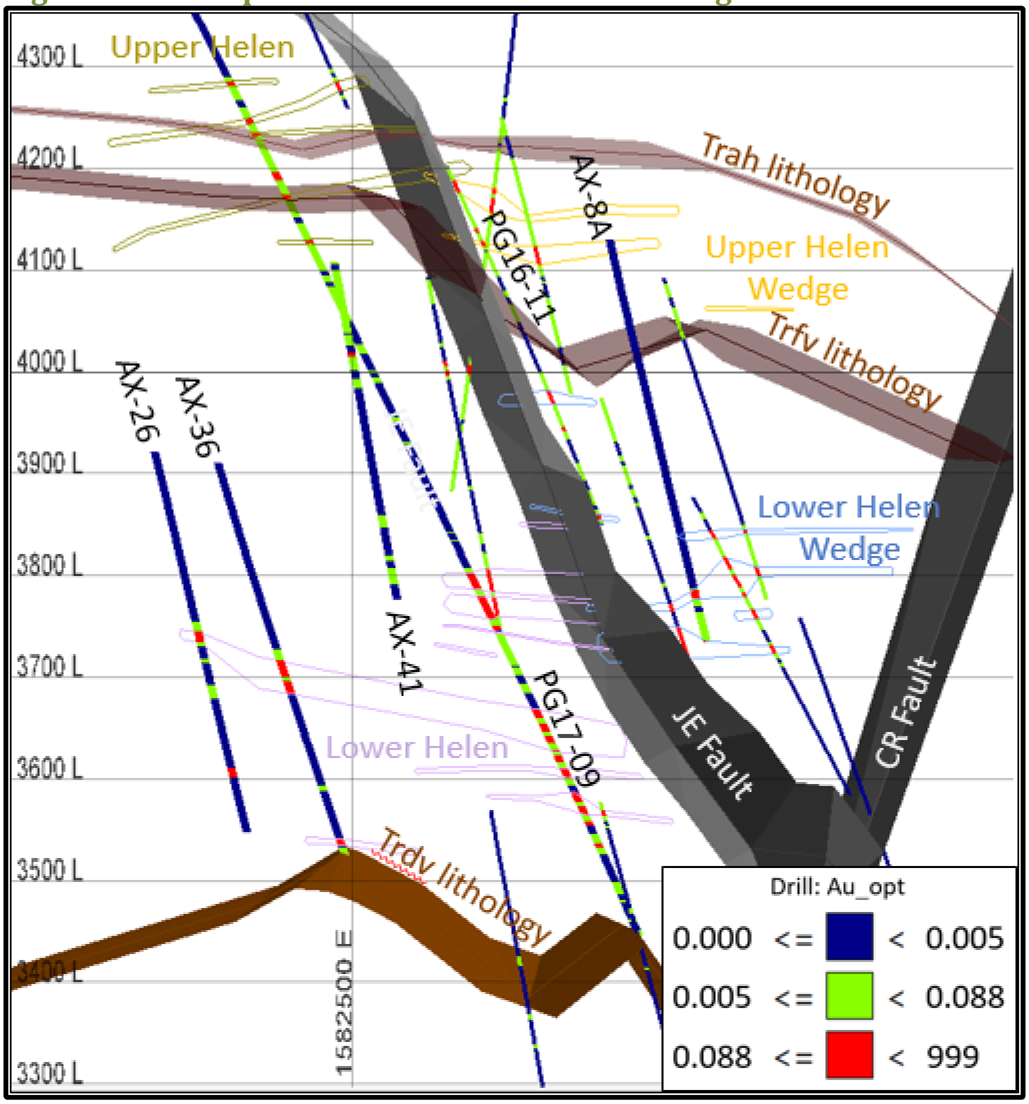


Figure 10-4 Sample Section of CSD Zone Drilling

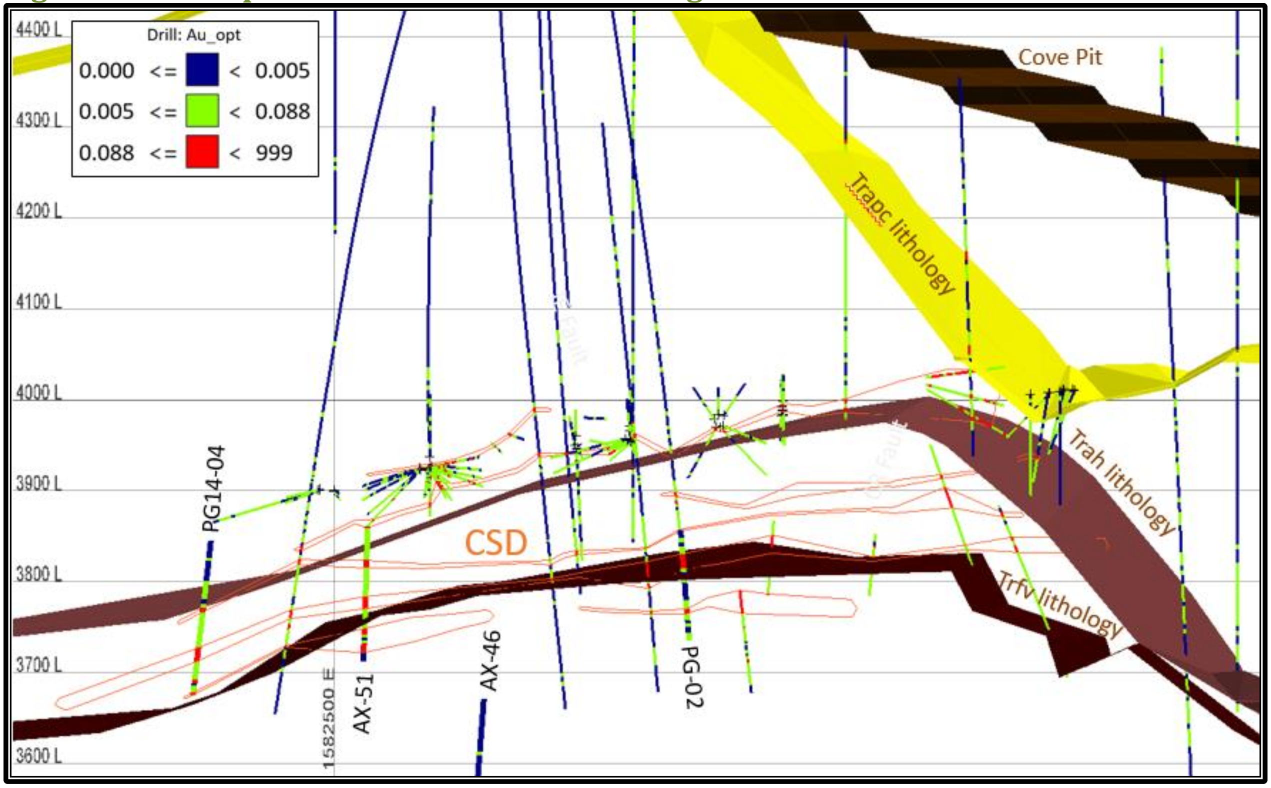
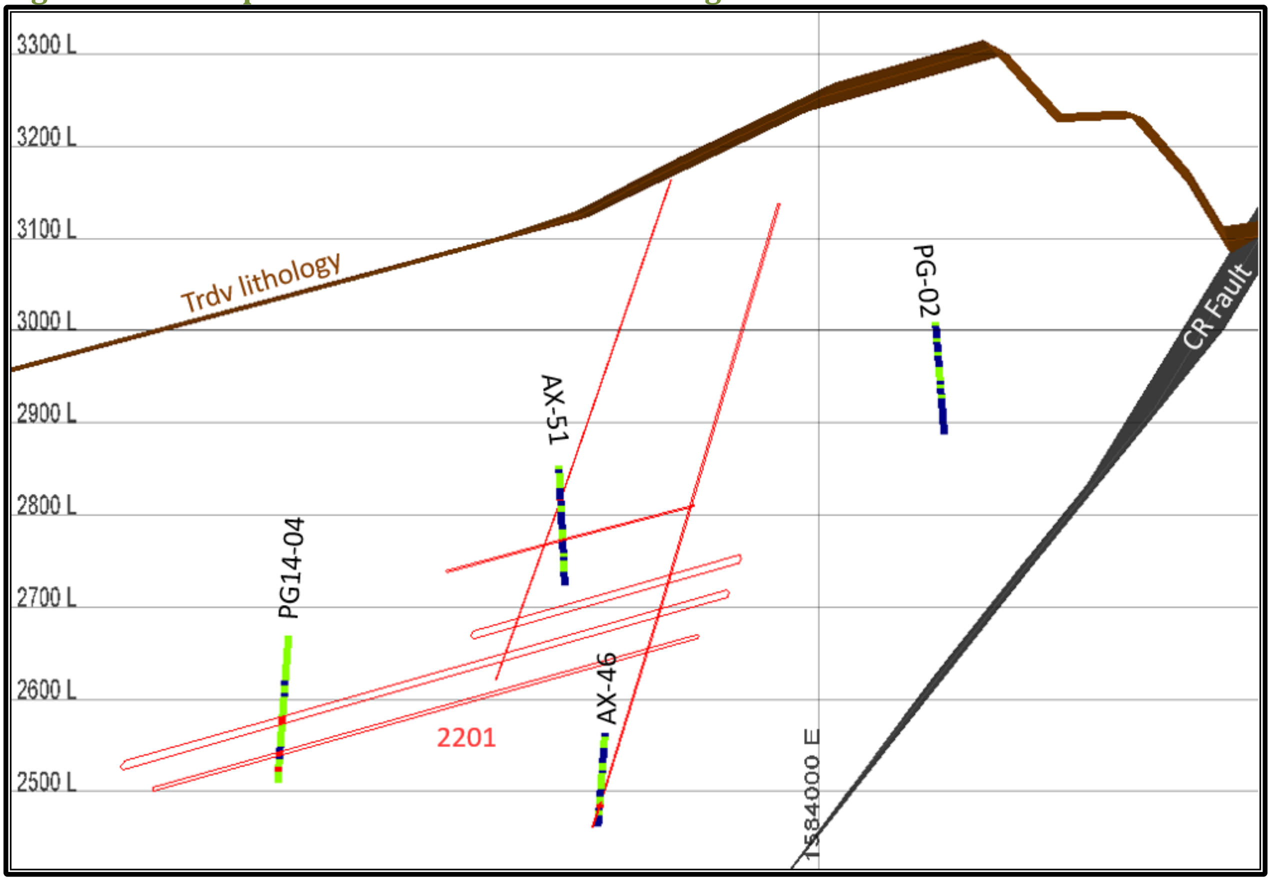


Figure 10-5 Sample Section of 2201 Zone Drilling



Recent drill projects have predominantly been completed by coring, while RC drilling was used extensively to delineate historic pit and underground resources. Accordingly, the recently discovered Helen, 2201 and CSD-Gap zones were modeled almost exclusively using core holes, while the pit-proximal CSD Zone and low-grade lenses were modeled using a mix of RC and core. Table 10-2 details the proportion of core drilling used to model each zone. The authors have carefully reviewed the data and consider both core and RC data to be reliable.

Table 10-2 Type of Drilling by Zone

Zone	Mineral Lens Codes	Number of Holes	RC Composites	Core Composites	Total Composites	% Core
CSD_Gap	220X	27	0	327	327	100
GAP Hybrid	500X	27	1	132	133	99
CSD	110X	269	1,276	699	1,975	35
Helen	310X, 320X, 330X, 340X	65	23	871	894	97
2201	130X, 140X	8	0	53	53	100

Zone	Mineral Lens Codes	Number of Holes	RC Composites	Core Composites	Total Composites	% Core
Low_CSD_Gap and Low_Gap Hybrid	22000	157	3,897	6,153	10,050	61

10.1. Historic Drilling Methodology

Evan et al., (2011) described drilling protocols for Victoria:

"Victoria diamond drill holes NW-01 to NW-09, inclusive, were spotted by hand-held GPS. This included collar, foresight and backsight. Drill holes NW-10 to NW-15, inclusive, were surveyed by All Points North, registered Nevada Land Surveyors. A Brunton compass was used to set the drill head.

"All diamond drill holes were proposed and collared based on the property grid, which was referenced in a historical digital terrain map (DTM) created prior to full scale mining and reclamation." (page 74)

"All Victoria diamond drilling was completed from surface retrieving whole core. The holes were collared HQ size and reduced down to NQ size dependent upon ground or drilling conditions. Drill muds were utilized to ensure consistent core recovery." (pg. 71)

"Victoria downhole surveys were completed using a North Seeking Gyro (NSG) by Major Technical Services and International Directional Services. NSGs eliminated the need for sighting on surface (gyro-compass alignment) and offered high accuracy. Generally NSG surveys were performed twice, once at mid-hole and again at hole completion. Readings of dip and azimuth were taken at nominal 50 ft intervals.

"RPA notes that no directional tests were taken during regular drilling operations. Holes NW-02 and NW-09A were unable to be downhole surveyed as the holes had to be abandoned due to poor ground conditions." (Page 74)

Formal records of Newmont and Echo Bay drill procedures have not been located, but methods are assumed to have followed industry standard.

10.2. Current Drilling Methodology

10.2.1. Drill Hole Placement

Initial surface collar locations are based on drill plan targeting – collar locations are marked in the field by a geologist using a hand held global positioning system (GPS) device loaded with coordinates from drill plans in either Gemcom or MapInfo project files. A wooden collar picket is marked with both the azimuth and dip designations. The azimuth is also painted in a line on the ground directly in-line with the collar picket allowing the drill rig to line up on the correct bearing from the collar location. The geologist re-confirms both azimuth and dip once the rig is lined up on the drill pad using a Brunton compass. After drilling is complete, holes are abandoned and marked with a metal tag cemented into the collar. A final collar location survey is performed by a professional contract surveyor. The project uses UTM NAD83 Zone 11N coordinate system.

10.2.2. Downhole Surveys

International Directional Services (IDS) of Elko performs downhole surveys on all drill holes. Holes are surveyed on 50-foot intervals using a north-seeking gyroscopic downhole survey tool.

10.2.3. RC Drilling Procedures

Holes are drilled using industry standard RC drilling equipment. Samples are collected on five-foot intervals using a cyclone sample collector. The sample interval is written on the sample bag using permanent marker. Drilling advances are paused at the end of each sample run to ensure the complete sample has been collected and avoid contamination of the following sample. The optimum sample size collected is approximately one quarter to one half of a 17-inch by 22-inch sample bag (about 4.5 to 9 kg or 10-20 pounds.)

10.2.4. Core Drilling Procedures

Core holes are drilled using HQ (about 3-inch diameter) core. Holes may be reduced to NQ (about 2.4-inch diameter) to permit continuation of a hole in difficult drill conditions. Premier has used both standard and triple-tube tooling. Triple-tube is preferable in broken ground because it facilitates placement of core into the core box, allowing the sample to remain more intact. Drilled material is placed in wax-impregnated core boxes. Drillers label the end of the core run to the nearest half of a foot, and measure and record the recovery in feet on wooden blocks, which are placed in the core box at the end of each drilled interval. Core boxes are labeled with company name, property, bore hole identifying number (BHID), box number and drilled interval. The authors believe the drilling procedures are adequate.

10.3. Sampling Methodology

Boxed core is delivered to the Battle Mountain core logging facility by Premier geologists or geotechnicians. The core is washed, photographed, and RQD logged. Detailed geology logs are completed. Data is entered directly into LogChief, a Maxwell software logging module loaded on a laptop.

Sample intervals are chosen by the geologist based on detailed geology observations. Sample intervals may range from ten feet to a minimum of one foot. The geologist marks sample intervals on the core and staples a sample ticket double-stub in the core box at the end of the sample interval. Sample IDs are automatically generated in LogChief starting with a number the geologist enters from a printed fifty-sample booklet. Logged core boxes are stacked on a wooden pallet prior to being moved into the adjoining warehouse for sample cutting.

The geologist prints a cut-sheet from "LogChief" software with the sample numbers and intervals and gives the cut-sheet to the geotechnician. The geotechnician puts one sample bag in a five-gallon plastic bucket on the floor next to the core saw. The core is sawed in half, and the left piece is placed into the bag on the floor; right piece goes back into the core box. In the case of broken core, the sampler does his best to divide the sample equally. Once the interval is split, the geotechnician takes one part of the double sample stub from the core box and staples it to the sample bag. The remaining sample stub remains in the core box for future reference. The geotechnician then ties the sample bag shut and marks the sample off the cut-sheet. The tied sample bags are stored in a sample bin for the lab driver to pick up.

The geologist assigns five QAQC samples per 50 samples. The geotechnician places the blanks and duplicates with their sample tags in the sample bin with the regular core samples. The standards are placed in a smaller box on a desk next to the large sample bin.

The geologist completes a sample submittal sheet. The lab driver picks up the samples directly from Premier's warehouse location and is given a chain of custody form with sample ID's for the shipment. An electronic copy of the sample submittal form is emailed to the lab.

Drill hole status, such as splitting, sample dispatch date, batch ID, and dates of both preliminary and final results, are tracked on a white board in the geology office.

The authors believe the sampling procedures are adequate.

10.4. Core Recovery

Historic core recovery was described by (Evan et al., 2011):

“Overall core recovery for Victoria’s diamond drilling at the Cove Project is estimated at 90%.

“In RPA’s opinion, these values are likely to be overestimated based on the broken nature of the samples retrieved.” (Page 86)

The average recovery for core drilled by Premier is about 90%, which is consistent with historic recovery measurements. Recovery is calculated by measuring the length of material between blocks in the core box, and dividing that length by the drilled interval length. It is difficult to measure length accurately for a broken interval of core, and the tendency is to over-estimate recovery in broken intervals. This is a typical problem for drilling in Northern Nevada, and the authors believe that 90% is a reasonable estimate of recovery. Although any sample with less than 100% recovery is sub-optimal, the authors believe the samples provide a reasonable representation of the rock package.

11. Sample Preparation, Analysis and Security

11.1. Pre-2012

Of the 21 Echo Bay RC holes, only seven were presented with assay results. RPA was unable to determine the sample preparation laboratory or procedures for the Echo Bay and Newmont RC holes. RPA assumes that they were prepared to industry standards at the time either in-house or at a commercial facility. The Echo Bay samples were analyzed by Rocky Mountain Geochemical Corp. in West Jordan, Utah. The Newmont samples were analyzed by ALS Chemex in Sparks, Nevada. As per the ALS Chemex certificates, pulp samples were received and a 50-element aqua regia inductively coupled plasma (ICP) analytical package (ME-MS41) was run. The ICP elements, and their ranges in ppm or percent, are listed below:

Ag	0.01-100	Cu	0.2-10,000	Na	0.01%-10%	Ta	0.01-500
Al	0.01%-25%	Fe	0.01%-50%	Nb	0.05-500	Te	0.01-500
As	0.1-10,000	Ga	0.05-10,000	Ni	0.2-10,000	Th	0.2-10,000
B	10-10,000	Ge	0.05-500	P	10-10,000	Ti	0.005%-10%
Ba	10-10,000	Hf	0.02-500	Pb	0.2-10,000	Tl	0.02-10,000
Be	0.05-1,000	Hg	0.01-10,000	Rb	0.1-10,000	U	0.05-10,000
Bi	0.01-10,000	In	0.005-500	Re	0.001-50	V	1-10,000
Ca	0.01%-25%	K	0.01%-10%	S	0.01%-10%	W	0.05-10,000
Cd	0.01-1,000	La	0.2-10,000	Sb	0.05-10,000	Y	0.05-500
Ce	0.02-500	Li	0.1-10,000	Sc	0.1-10,000	Zn	2-10,000
Co	0.1-10,000	Mg	0.01%-25%	Se	0.2-1,000	Zr	0.5-500
Cr	1-10,000	Mn	5-50,000	Sn	0.2-500		
Cs	0.05-500	Mo	0.05-10,000	Sr	0.2-10,000		

ppm unless otherwise indicated

Fire Assay (FA) with an atomic absorption (AA) finish was utilized for gold assays (Au-AA23 package). Any gold FA values over 3 ppm were rerun by gravimetric methods (Au-GRA21). The detection limit for both gold assaying methods was 0.005 ppm. (Roscoe Postle Associates Inc., 2017)

Victoria's Cove samples were all prepared and analyzed by the Inspectorate assay laboratory located in Sparks, Nevada. The following discussion relates specifically to Victoria's samples.

11.1.1. Sample Preparation Procedures

Upon receipt by Inspectorate the core samples were reviewed, sorted, and oven dried (230°F). The samples were crushed to +80% passing 10 mesh by jaw crusher and pulverized to +90% passing

150 mesh by ring and puck. The samples were then split by a splitter; one half of the sample was set aside as the "reject" and the remaining half sample split again. This process was continued until the sample equalled three-fourths of the volume of a pulp envelope. The total rejects were tied, tagged, and palletized.

11.1.2. Laboratory Analysis Procedures

Gold assays were first run by FA with an AA finish with a detection limit of 5 ppb. Any gold FA values over 3 ppm were rerun by gravimetric methods. Silver assays were also run by FA/AA with a detection limit of 0.1 ppm.

A summary of Inspectorate's FA method is described below:

- Samples are received from weigh-room in labelled envelopes;
- Crucibles are set up in trays of twenty by numbers assigned from Laboratory Information Management System (LIMS);
- Crucibles are charged with the appropriate type and amount of flux;
- Samples are transferred from the envelopes to the appropriately labelled crucible, copper spikes are inserted, and inquarting is conducted;
- Additional reagents are added to the crucible if needed and sample and flux is mixed with cover flux added on to the top of charge;
- Crucibles in sets of 80 charges are then loaded into pre-heated gas fusion furnace and fusion is conducted for one hour at 2,100°F;
- Upon completion of fusion, molten lead-slag is poured into numbered conical moulds. Unsatisfactory fusions are submitted back to the weighing room for reweigh;
- Fusions are allowed to cool and the moulds are transferred in order to the slagging station. Slag is removed with hammer, and lead buttons are cubed and placed in numbered trays;
- MgO cupels are heat treated in the cupel furnace at 1,800°F for a minimum of five minutes to drive off moisture. Cupels are then carefully evaluated for cracks or erosion and are discarded accordingly;
- Lead buttons are loaded into cupels in order and the set is then loaded with a fork into an electric oven set at 1,800°F;
- Upon full cupellation (lead adsorption), the cupels are allowed to cool and the resulting Ag ± Au prills are placed into numbered trays;
- For AA finish, the prills are dissolved in aqua regia and analyzed on the ICP, and;
- For gravimetric finish, the prills are placed in parting cups, approximately two-thirds full with 20% Nitric Acid to dissolve the silver, and then heated on a hotplate at 125°F until parted. The gold bead is then allowed to cool, transferred to cups, rinsed with cold de-

ionized water, and allowed to dry. The cups are fired at 1,560°F for approximately five minutes, and then allowed to cool. The resulting doré bead is weighed on a microbalance.

A multi-element ICP analytical package was also run for all samples. The ICP elements determined including their detection limits in ppm are presented below:

Ag	0.1-100	Co	1-10,000	Mg	100-100,000	Sc	1-10,000
Al	100-100,000	Cr	1-10,000	Mn	5-10,000	Se	0.2-1,000
As	5-10,000	Cu	1-10,000	Mo	1-10,000	Sr	0.2-10,000
B	10-10,000	Fe	100-100,000	Na	100-100,000	Ti	100-100,000
Ba	10-10,000	Ga	0.05-10,000	Ni	1 -10,000	Tl	10-100,000
Bi	2-10,000	Hg	3-100,000	P	10-50,000	V	1-10,000
Ca	100-100,000	K	100-100,000	Pb	2-10,000	W	10-5,000
Cd	0.5-1,000	La	2 -10,000	Sb	2-10,000	Zn	2-10,000

11.1.3. Security

Security measures taken to ensure the validity and integrity of the samples collected included:

- Chain of custody of drill core from the drill site to the core logging area;
- Buildings were kept locked when not in use;
- Core sampling was undertaken by technicians under the supervision of Victoria geologists;
- All intersections were kept in the Reno office, and;
- Inspectorate was storing all the rejects and pulps indefinitely.

11.2. Premier 2012-2018

Drill hole samples collected by Premier were sent for assay analyses to three independent laboratories:

- American Assay laboratory, Sparks, Nevada, accredited ISO/IEC 17025:2005;
- Inspectorate America Corporation, Sparks, Nevada, accredited ISO 9001:2008 and ISO/IEC 17025:2005, and;
- ALS Minerals, Vancouver, British Columbia, accredited ISO/IEC 17025:2005.

From 2012 until end of 2014, samples were sent for analyses to Inspectorate laboratories. Starting with 2015, samples were sent to ALS. The pulp sample checks were sent to the American Assay laboratory.

The sample preparation and gold FA procedures for the Premier 2012-2016 drilling programs at all the laboratories are essentially the same as described above except that gold FA results greater than 10 g/t Au are re-assayed by FA/gravimetric.

In addition to the fire assay analysis, the current program includes analysis of gold and silver by screen metallic methods when visible gold is noted in the polymetallic sheeted veins intercepted in the 2201 zone.

The current program also incorporates a 42-element, four-acid, ICP-mass spectrometry, ultra-trace level analysis.

The sample preparation, analysis, and security procedures at the Project are adequate for use in the estimation of Mineral Resources.

11.3.Quality Assurance and Quality Control

11.3.1. Standards and Blanks

A total of 69 different blank and gold standard reference materials have been used at Cove. Table 11-1 presents the results of the most frequently assayed materials. The null hypothesis test compares the calculated t-statistic to the t value for a 95% confidence level. Acceptance of the test indicates that the lab mean is within the 95% confidence limit of the standard value. A rejection result from the test does not necessarily mean the data is not representative of the expected value but rather that the test was inconclusive. Groups which have a high out limit frequency are not necessarily reject by the t-test if the standard deviation for the group is not excessively high.

Table 11-1 Gold Blank and Standard Summary Statistics

ID	Count	Lab Mean PPM	Lab Std Dev	Out of Limit Rate	Std. Value PPM	t-statistic	$t_{\alpha/2}$	Null Hypothesis Test
Blank	1880	0.054	0.626	14%	0.005	3.380	1.646	Reject
CDN-GS-P8C	166	0.777	0.108	3%	0.784	(0.843)	1.974	Accept
CDN-GS-P4E	392	0.519	0.412	3%	0.493	1.256	1.966	Accept
SP37	283	17.892	2.560	0%	18.140	(1.631)	1.968	Accept
CDN-ME-1301	150	0.550	0.434	20%	0.437	3.180	1.976	Reject
CDN-GS-22	144	22.600	2.523	14%	22.940	(1.618)	1.977	Accept
CDN-GS-5L	193	4.747	0.551	10%	4.740	0.180	1.972	Accept
OREAS 503b	116	0.695	0.013	0%	0.695	(0.086)	1.981	Accept
G912-1	109	7.356	0.112	0%	7.290	6.132	1.982	Reject
OxJ120	107	2.365	0.044	45%	2.365	0.033	1.983	Accept
CDN-GS-5H	99	4.004	1.930	48%	3.840	0.847	1.984	Accept
CDN-GS-2M	85	2.917	3.865	22%	2.210	1.686	1.989	Accept
CDN-GS-12	77	9.423	1.770	32%	9.980	(2.760)	1.992	Reject

ID	Count	Lab Mean PPM	Lab Std Dev	Out of Limit Rate	Std. Value PPM	t-statistic	$t_{\alpha/2}$	Null Hypothesis Test
CDN-GS-11	63	3.398	0.877	21%	3.400	(0.020)	1.999	Accept
CDN-GS-4D	47	3.839	0.408	19%	3.810	0.483	2.013	Accept
G307-7	32	7.964	0.102	0%	7.750	11.837	2.040	Reject
SQ48	45	30.229	0.327	22%	30.250	(0.433)	2.015	Accept
OXI81	43	1.815	0.126	28%	1.807	0.418	2.018	Accept
OXD87	34	0.412	0.024	18%	0.417	(1.162)	2.035	Accept
CDN-GS-30	33	33.553	0.786	3%	33.500	0.390	2.037	Accept
G909-4	33	7.496	0.176	0%	7.520	(0.770)	2.037	Accept
CDN-GS-15B	31	15.619	2.179	13%	15.980	(0.924)	2.042	Accept

Figure 11-1 Blank Assay Results

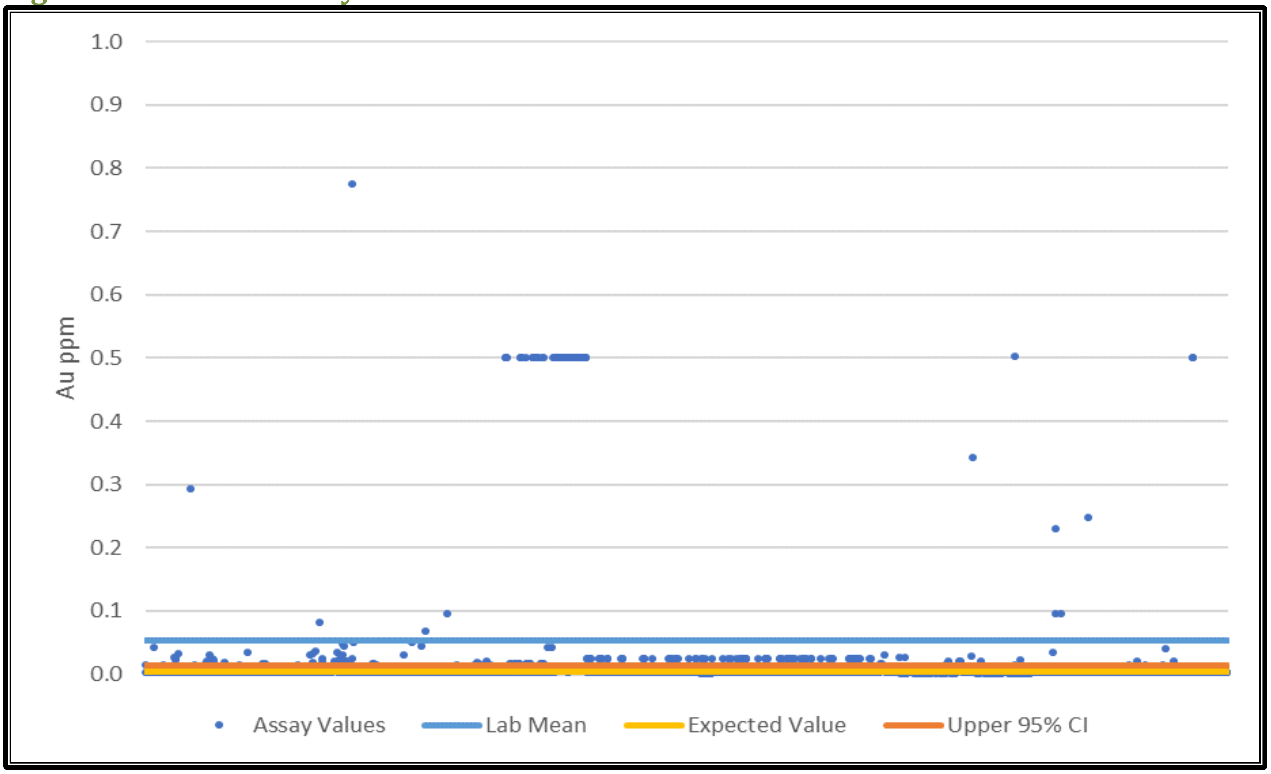


Figure 11-2 SP 37 Standard Reference Material Results

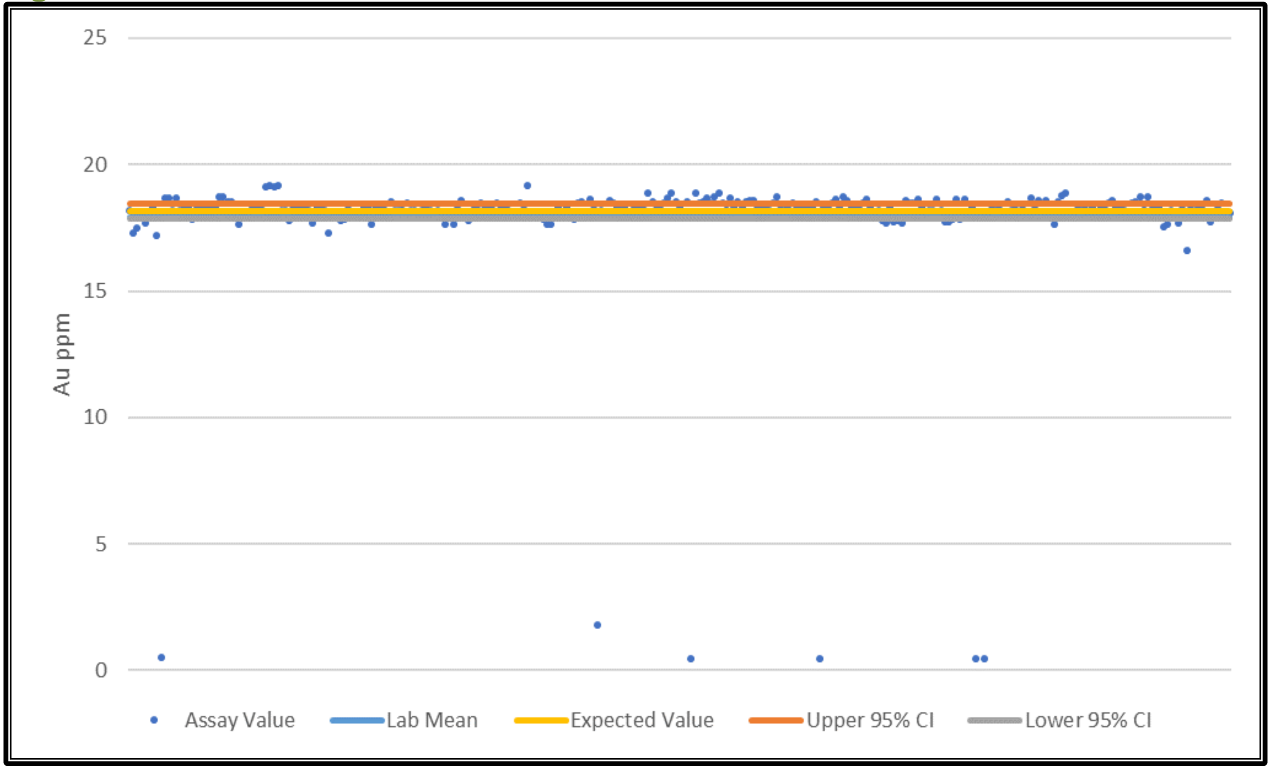


Figure 11-3 CDN-GS-22 Certified Reference Material Results

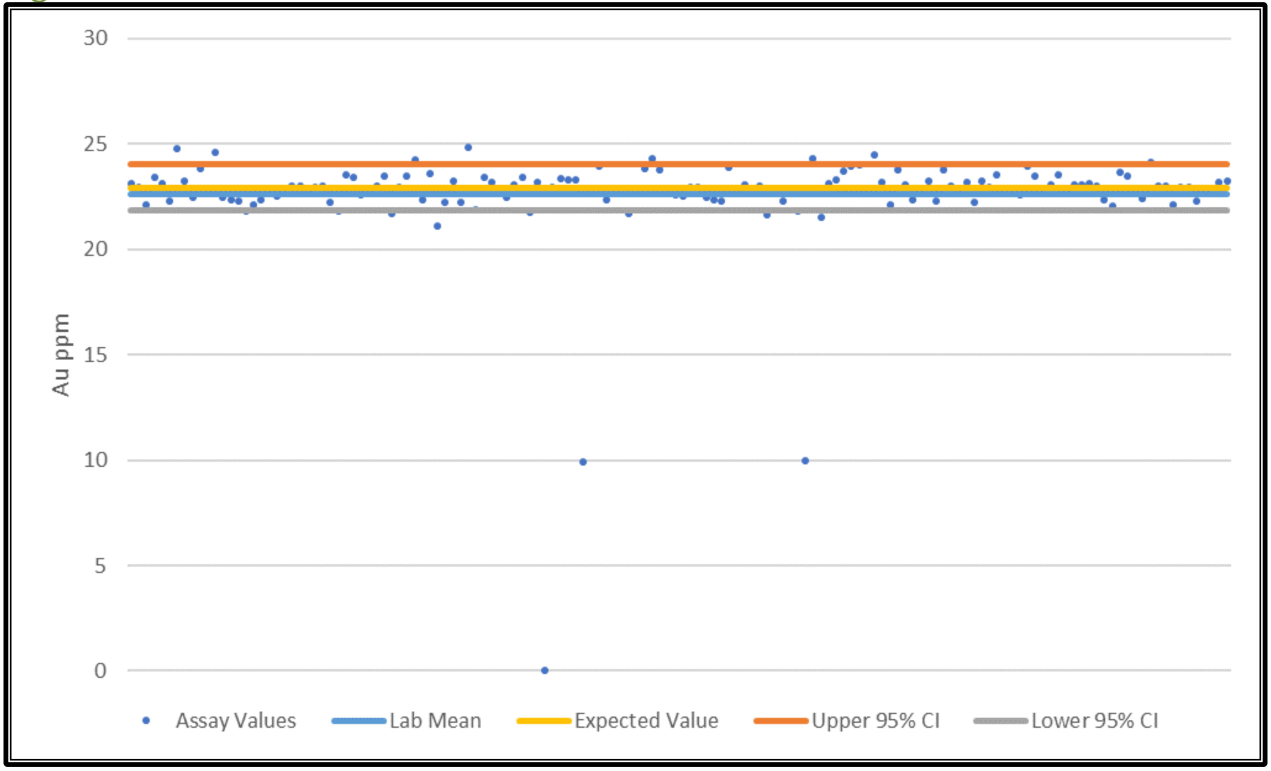


Figure 11-4 CDN-GS-5H Certified Reference Material Results

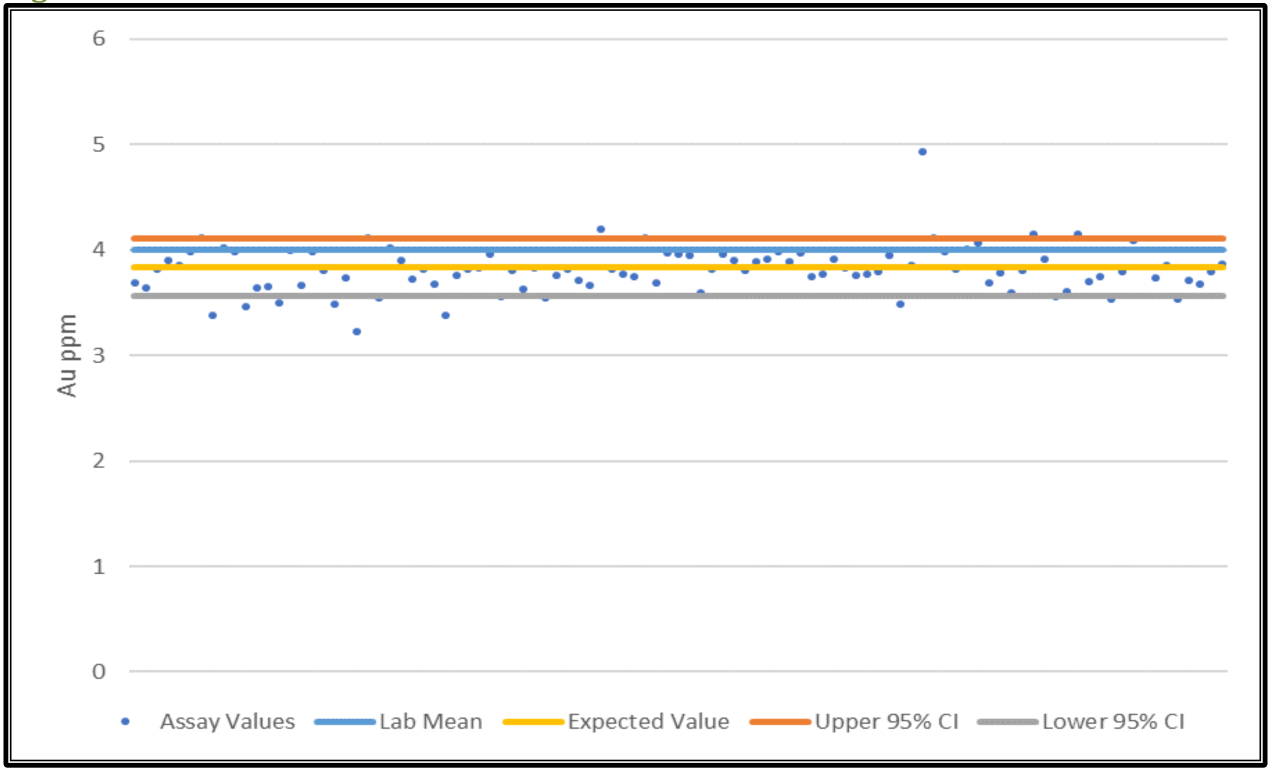
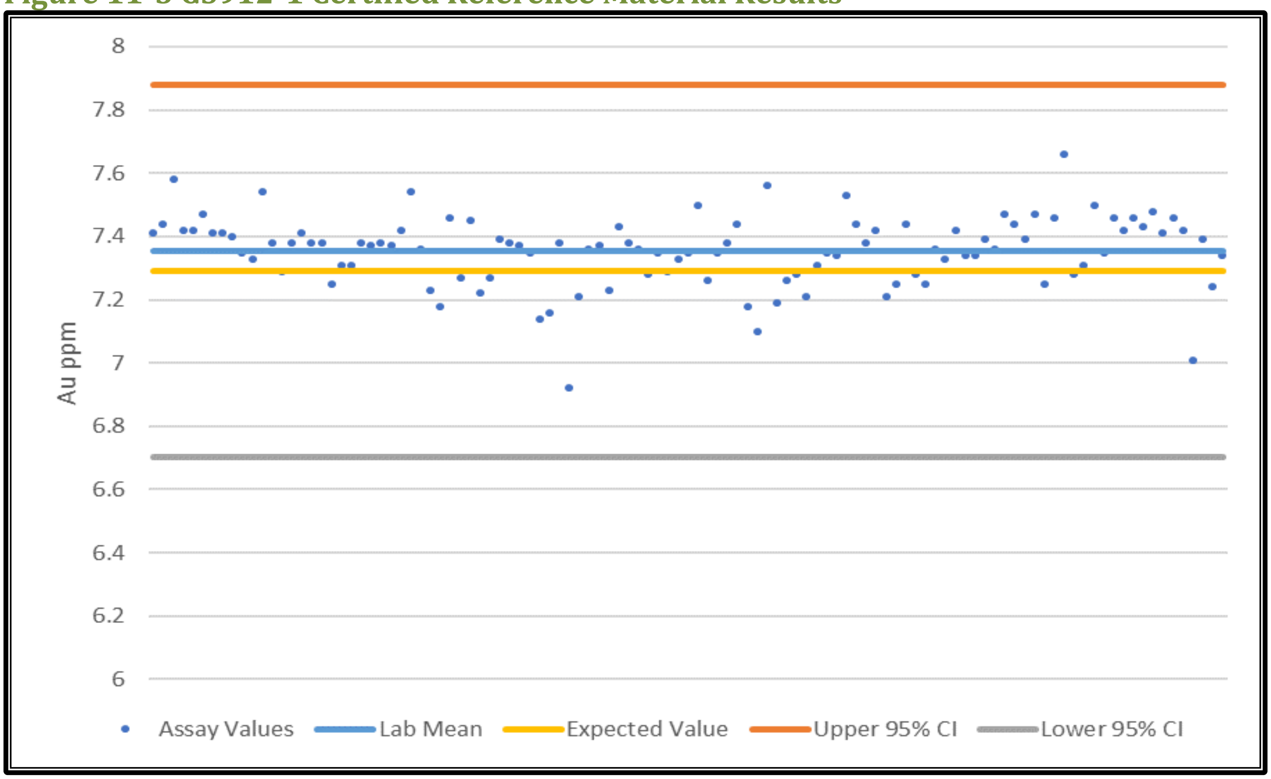


Figure 11-5 GS912-1 Certified Reference Material Results



11.3.2. Duplicate Assays

Duplicate assays are performed under two scenarios. The geologist can instruct the lab to duplicate the pulp of a specified sample (Figure 11-6) or the lab can send a pulp to another lab for check assay (Figure 11-7). Both types of duplicates show good replication of assay values.

Figure 11-6 Prep Duplicates - ALS Reno

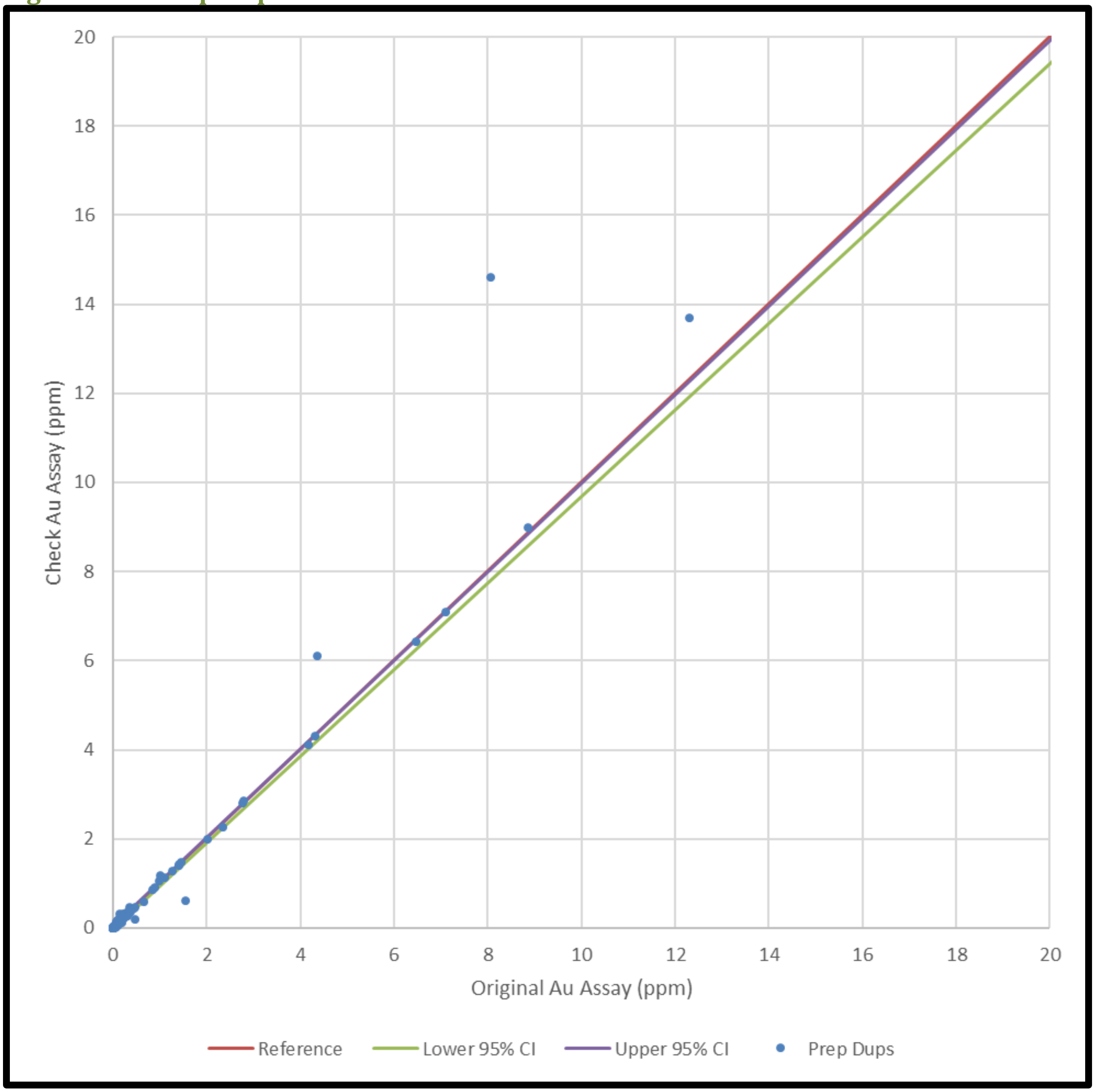
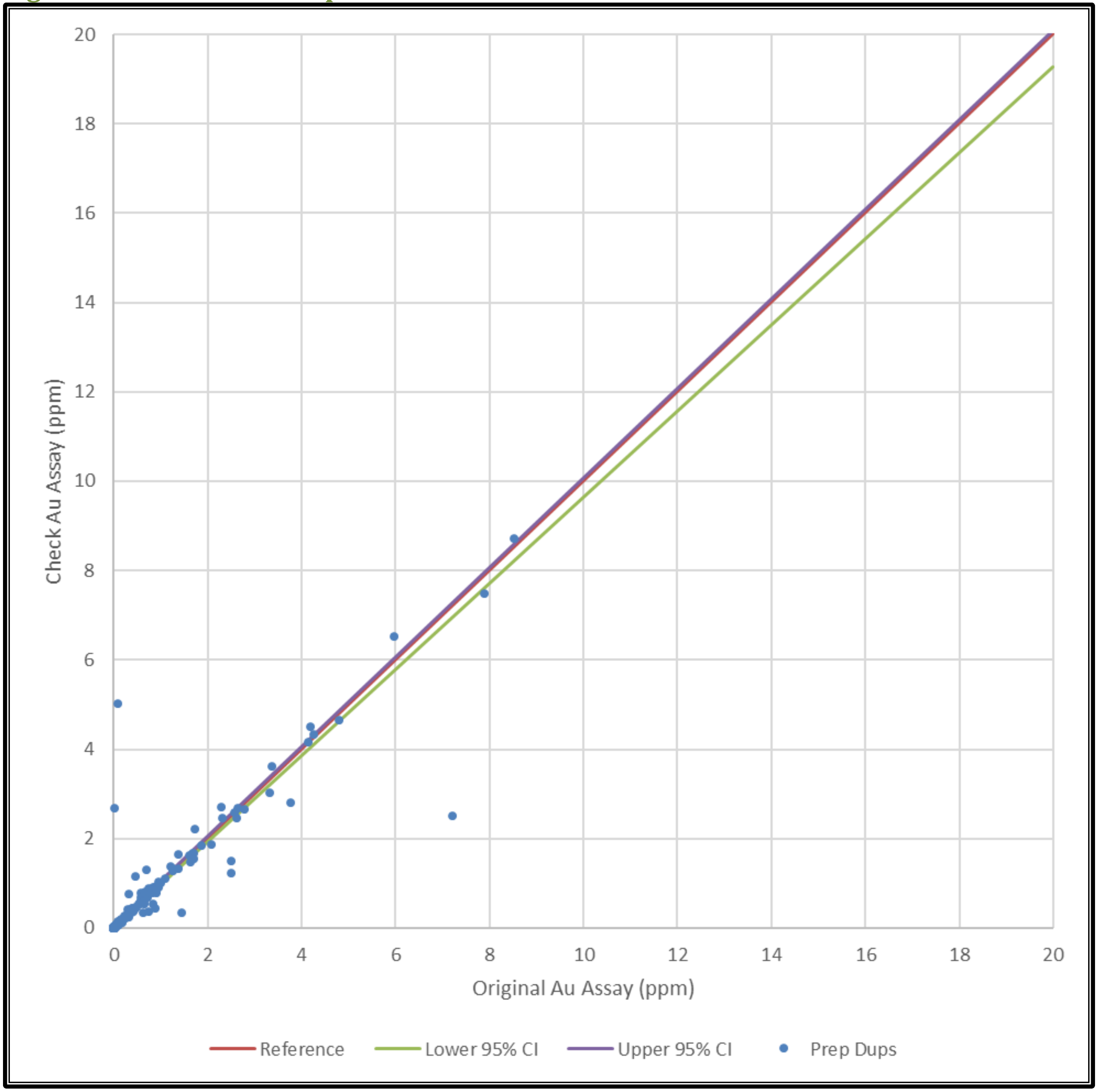


Figure 11-7 Lab Check Duplicates



It is the authors opinion that the sample preparation, security, and analytical procedures are adequate for the estimation of Mineral Resources.

12. Data Verification

Practical Mining received the McCoy Cove drill hole database from Mia O'Neal, Premier Senior Geologist. Premier manages the data using Maxwell Geoservices software, and Ms. O'Neal exported the data as csv files for Practical Mining. The authors imported the data into Maptek Vulcan software and identified holes within the resource area. The authors selected 5 percent of holes from the resource dataset for detailed review. The selected holes are a spatial and temporal sampling of the data, the majority consisting of holes drilled by Victoria and Premier because most older holes are in the mined area and supported by past production. Ms. O'Neal supplied copies of the raw data for the selected holes to the authors.

The authors compared the raw data with the corresponding records in the database. Records reviewed include assay values for gold and silver, collar location surveys, and downhole deviation surveys. The authors observed no significant problems with the data, and conclude the data is suitable for use in the resource estimation.

The authors did not observe any mismatches between assay certificates and the database. Minor inconsistencies in the handling of missing data were noted. Sampled intervals which lack assay data typically have a blank cell in the assay column of the csv, but holes AX-10 and AX-22 contained negative values. Those holes were subsequently corrected by re-importing into Maxwell Geoservices software. All missing data cells were assigned -99 for use in Vulcan software, including holes AX-10 and AX-22, so the database inconsistency did not affect the estimation.

Collar surveys are collected by professional land surveyors and reported to Premier in Excel spreadsheets. Collar surveys are occasionally duplicated on subsequent surveyor visits, and surveys will vary slightly due to limits in precision. The authors noted one collar with a slight mismatch between the surveyor's spreadsheet and the database, however the small difference in distance has an insignificant effect on hole placement and may be attributed to multiple surveys of the same collar.

The authors did not observe any mismatches between downhole survey reports and the database. Table 12-1 summarizes the scope of the detailed drill data review.

Table 12-1 Data Review Summary

	Holes in Data Set	Holes Audited	Collar Survey Coordinates Reviewed	Downhole Surveys Reviewed	Assay Certificates Reviewed
Number of Drill Holes	1,397	70	70	70	88
Percent of Population Reviewed		5%	5%	5%	6%

All holes were checked for overlapping intervals using Vulcan, and there were none. Hole traces were viewed in Vulcan to confirm there were no extreme survey deviations. Lithology was viewed in Vulcan to confirm that the geology model conforms to the geology data.

In summary, the authors observed no significant problems with the data, and conclude the data is suitable for use in the resource estimation.

13. Mineral Processing and Metallurgical Testing

13.1. Historical Metallurgical Test Work

The historical metallurgical testing reviewed in regard to the Cove Project includes the following:

- Kappes Cassiday & Associates (KCA) for Victoria Gold – 2008 – Whole Ore Leaching and Flotation Tests, and;
- Kappes Cassiday & Associates (KCA) for Victoria Gold – 2009 – Roasting and Cyanidation of Calcine, Hot Lime Treatment, and Flotation Tests on Rejects from 2008 Program

The 2008 KCA test program was conducted on nine (9) composites from the Helen Zone. The testing included:

- Bottle Roll direct cyanidation of each composite;
- Bottle roll Carbon-In-Leach (CIL) cyanidation of each composite, and;
- Rougher and Scavenger Flotation on each composite.

The whole ore cyanidation tests gave generally poor gold extractions ranging from 1% to 23%.

The CIL cyanidation tests gave higher gold extractions ranging from 49% to 82%.

The difference between the whole ore cyanidation and the CIL cyanidation indicates a pregnant solution robbing factor in the composites tested.

The flotation tests gave gold recoveries into a concentrate ranging from 24% to 59%. The corresponding concentrate weight recoveries ranged from 9 to 13%.

The flotation tests gold recoveries were low and did not demonstrate a strong amenability towards flotation.

Based on the suspicion that the relatively poor cyanidation results from the 2008 testing were due to the carbonaceous content of the materials tested the 2009 KCA test program investigated three types of processes to mitigate the effects of carbonaceous matter. Testing was conducted using a composite constructed from the composite remains from the 2008 program. The testing included:

- Head Characterization for Carbonaceous and Sulfide Material in 2008 Drill hole interval samples used in 2008 composite construction;

- Roasting followed by cyanidation of calcine using both direct cyanidation and CIL cyanidation of the calcines;
- Hot Lime treatment of the Composite, and;
- Flotation.

The head analyses indicated organic carbon contents ranging from 0.03% to 0.96% with an average of 0.44%. The sulfide sulfur content ranged from 0.15% to 1.79% with an average of 1.02%. The assays confirmed the presence of carbonaceous material as well as potential refractory aspects related to sulfide sulfur content.

Roasting tests were conducted using a 650°C for two hours. The gold extraction for the direct cyanidation of the calcine was 87% while the extraction using CIL cyanidation of the calcine was 90% which indicates that after calcination there are still active pregnant solution robbing factors.

The hot lime treatment was conducted on a sample of the composite ground to 80% passing 200mesh to which a lime addition of 100lbs/ton was made. The slurry was heated to 100°C and agitated for 8 hours. The slurry was then leached with cyanide. The gold extraction for the hot lime test was 40%.

Two flotation tests were conducted, the first using a rougher, scavenger, cleaner simulation, the second simulating four stages of rougher flotation. The gold recovery from the first test was 54% into a concentrate with a 17.6% weight recovery. The second test gave a gold recovery of 31% into a concentrate with a weight recovery of 20.7%.

The 2009 tests confirmed the presence of carbonaceous material, the likely cause of pregnant solution robbing observed in the whole cyanidation tests.

The tests indicated that roasting and calcine cyanidation may be an effective treatment for the material tested. The hot lime treatment and flotation tests did not match the roast and calcine cyanidation gold recoveries.

13.2.2017 Metallurgical Test Work – Phase 1

A preliminary metallurgical scoping test program was conducted in 2017 to support a PEA of the Cove Project Helen and Gap Deposit resource targets.

Three concepts formed the basis for the test program as follows:

- First it was recognized, based on the historical testing, that the material within target resources would likely be refractory in various degrees to direct or CIL cyanidation and that some type of oxidation process would be required to treat the materials prior to cyanidation, and;

- Second, the historical test work indicated that there may be a significant amount of variability within the resources and that testing to initially assess metallurgical variability within the resources was needed, and;
- Third, toll processing of the material by a second party will be used initially to place the property into production as quickly as possible.

The primary objectives of the test program were as follows:

- Select drill holes and discrete intervals in the drill holes to obtain initial spatial representation vertically and with the length and breadth of the resources, and;
- Obtain head assays and tests to adequately characterize the physical and metallurgical properties of each resource required for processing by a third party.

To develop initial metallurgical data to evaluate the resource targets at the project site based on potential metallurgical processing by a third party.

Testing to project process precious metal extractions, and metal deportment, reagent consumptions, and track metals (Au, Ag, As, & Cu) by:

- Whole cyanidation;
- Roasting followed by calcine cyanidation;
- Pressure Oxidation followed by cyanidation of neutralized slurry, and;
- Roaster and pressure oxidation test conditions used in the program were based on those provided by a potential toll processing operator.

The program was not specifically designed to determine the optimal roasting or pressure oxidation conditions or develop design data for a new processing plant.

SGS Canada Inc., Lakefield, Ontario, Canada was selected to perform the test program.

13.2.1. Composite Construction

Premier Gold Mines and Jacobs personnel met in the Premier Gold Mines Battle Mountain office on March 15 and 16, 2017 to select drill holes and intervals to construct composites for the test program. The objectives in selecting drill holes and drill hole intervals for the composites were as follows:

- Begin developing the basis for any assumptions or predictions regarding recovery within the resources;
- Test samples that are representative of the various types and styles of mineralization within the mineral deposits;
- Determine processing factors or deleterious elements that could have a significant effect on potential economic evaluation, and;
- Select samples generally conforming to projected specifications by prospective toll processing operations for processing either through a roaster or via pressure oxidation.

The following Table 13-1 and Table 13-2 shows summaries of the composites selected for the test program. The assays shown are weighted averages based on interval assays for the respective intervals.

Table 13-1 Helen Composites

Comp	HOLE ID	Interval length ft	Au opt	Ag opt	CU PPM	PB PPM	AS PPM	CD PPM	MN PPM	SB PPM	HG PPM
5	Ax-12	28	0.269	0.079	40.7	8.7	531	0.7	257.3	50.7	2.7
6	AX-18	27	0.327	0.083	12.7	1.0	344	0.3	339.5	22.5	1.9
20	PG17-07	18	0.158	0.080	16.5	9.5	165	0.4	397.0	35.6	0.0
21	PG17-07	22.5	0.291	0.082	23.6	7.6	1252	1.1	726.8	82.4	12.3
14	PG17-07	18	1.103	0.316	28.3	6.7	2104	1.2	922.5	74.9	14.8
22	PG17-07	22.5	0.205	0.099	20.1	5.3	350	1.4	691.8	92.0	40.9
15	AX-27	20.5	0.142	0.049	10.0	1.5	106	0.3	2259.6	17.9	5.6
16	AX-27	23.5	0.269	0.057	30.0	7.3	513	0.4	99.9	61.1	10.5
17	AX-27	22	0.254	0.088	19.3	3.6	144	0.5	560.7	24.0	1.9
18	AX-27	25.7	0.358	0.138	21.9	3.7	108	0.4	568.2	23.6	5.1
19	AX-27	46	0.425	0.157	20.2	5.0	180	0.9	575.5	64.3	13.2

Table 13-2 Gap Composites

Comp	HOLE ID	Interval length ft	Au opt	Ag opt	CU PPM	PB PPM	AS PPM	CD PPM	MN PPM	SB PPM	HG PPM
2	PG16-02	34	0.175	1.3	32.3	18.5	1221	0.2	73.6	967.3	4.0
9	PG16-06	37.3	0.491	0.2	19.0	8.1	2080	0.2	207.5	282.4	4.4
10	PG16-11	19.5	0.465	0.1	39.2	11.8	1915	0.1	402.1	1137.1	4.3
11	PG16-11	19.7	0.291	0.2	26.3	7.9	3339	0.2	889.9	261.2	4.0
12	PG16-11	17.3	0.148	0.1	24.7	6.2	1101	0.2	1421.2	221.4	1.8
13	PG16-11	17	0.565	1.6	31.2	7.0	1239	0.7	713.1	72.7	0.9
15	PG16-12	35	0.932	0.2	16.9	5.4	2705	0.1	332.0	1033.6	2.2
16	PG16-12	19.5	0.932	0.4	19.3	4.6	1920	0.6	832.5	56.3	1.0
17	PG16-12	22	0.262	0.5	27.2	14.4	990	0.7	392.9	45.8	1.4
20	PG16-16	32	0.232	0.1	27.0	13.2	2403	0.2	52.3	89.0	5.3

Comp	HOLE ID	Interval length ft	Au opt	Ag opt	CU PPM	PB PPM	AS PPM	CD PPM	MN PPM	SB PPM	HG PPM
2	PG16-02	34	0.175	1.3	32.3	18.5	1221	0.2	73.6	967.3	4.0

13.2.2. Head Characterization

The following analyses were performed as recommended by a potential toll process plant operator:

- Fire Assay in Triplicate for Au and Ag;
- Screen Metallic Assay for Au and Ag;
- Cyanide Soluble Au and Ag;
- High precision Assay for Hg and AS;
- Sulfur Speciation – Total S, Sulfide Sulfur, Sulfate, Native Sulfur;
- Carbon Speciation – Total Carbon, Graphite, Total Organic Carbon, Total Carbonaceous Carbon, CO_3, and CO_2, and;
- ICP Multi-element Analysis – Al, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Sn, Sr, Ti, U, V, Y, Zn;
- Comminution Properties

The following Table 13-3 and Table 13-4 show the precious metals assays for the Helen and Gap Composites.

Table 13-3 Helen Composite Precious Metal Assays

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Au (FA) cut 1	g/t	7.62	8.49	31.9	5.94	10.8	6.80	11.9	10.3	4.44	9.29	7.52	10.45
Au (FA) cut 2	g/t	7.52	8.49	31.7	5.94	10.7	6.75	11.8	10.3	4.35	9.32	7.61	10.41
Au (FA) cut 3	g/t	7.52	8.43	33.2	6.00	10.7	6.66	11.9	10.4	4.53	9.38	7.68	10.58
Au (FA) Avg.	g/t	7.55	8.47	32.3	5.96	10.7	6.74	11.9	10.3	4.44	9.33	7.60	10.48
Au (SM)	g/t	-	8.81	32.2	-	-	-	21.1	10.4	-	9.17	-	
Au CN Soluble	g/t	1.9	0.2	2.0	0.6	0.3	< 0.2	0.3	< 0.2	3.8	< 0.2	< 0.2	1.3
Ag (FA) cut 1	g/t	2.3	2.0	8.1	1.6	1.8	2.6	5.2	3.4	2.0	2.1	3.4	3.1
Ag (FA) cut 2	g/t	2.2	2.0	8.2	1.8	1.8	2.3	5.3	3.5	2.0	2.0	3.5	3.1
Ag (FA) cut 3	g/t	2.4	2.1	8.4	1.6	1.9	2.6	4.9	3.6	2.2	2.1	3.6	3.2

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Ag (FA) Avg.	g/t	2.3	2.0	8.2	1.7	1.8	2.5	5.1	3.5	2.1	2.1	3.5	3.2
Ag (SM)	g/t		< 2.68	< 8.88				< 5.10	< 3.65		< 2.26		
Ag CN Soluble	g/t	1.08	1.06	4.20	0.92	1.07	0.52	2.48	1.75	1.12	0.47	0.75	1.40

Table 13-4 Gap Composite Precious Metal Assays

Parameter	Hole Comp	PG16-02 #2	PG16-06 #9	PG16-11 #10	PG16-11 #11	PG16-11 #12	PG16-11 #13	PG16-12 #15	PG16-12 #16	PG16-12 #17	PG16-16 #20	
Au (FA) cut 1	g/t	4.46	15.5	17.7	8.46	5.60	24.0	36.7	34.7	11.3	7.17	16.56
Au (FA) cut 2	g/t	4.43	15.6	17.7	8.47	5.54	23.1	36.6	35.0	11.2	7.11	16.48
Au (FA) cut 3	g/t	4.37	15.2	17.6	8.53	5.61	22.2	37.7	34.7	11.4	7.16	16.45
Au (FA) Avg.	g/t	4.42	15.4	17.7	8.49	5.58	23.1	37.0	34.8	11.3	7.15	16.49
Au (SM)	g/t	-	16.3	-	-	-	20.8	38.4	36.7	-	-	
Au CN Soluble	g/t	0.3	0.3	4.2	< 0.2	< 0.2	0.7	0.2	0.6	< 0.2	0.2	0.9
Ag (FA) cut 1	g/t	37.3	7.4	4.4	4.7	3.3	43.4	11.3	12.8	11.3	3.0	13.9
Ag (FA) cut 2	g/t	38.7	7.6	4.5	4.7	3.3	41.4	11.0	12.6	12.2	3.2	13.9
Ag (FA) cut 3	g/t	37.0	7.6	4.4	4.7	3.3	39.6	11.2	11.9	13.5	3.2	13.6
Ag (FA) Avg.	g/t	37.7	7.5	4.4	4.7	3.3	41.5	11.2	12.4	12.3	3.1	13.8
Ag (SM)	g/t		7.43				46.2	10.9	< 12.4			
Ag CN Soluble	g/t	18.8	2.44	2.30	2.20	0.84	30.2	3.90	7.72	6.47	1.06	7.59

The following Table 13-5 and Table 13-6 show the mercury, arsenic, sulfur speciation, and carbon speciation assays for the Helen and Gap Composites.

Table 13-5 Helen Zone Hg, As, Sulfur Speciation, and Carbon Speciation

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Hg	g/t	3.9	3.0	13.6	7.9	11.3	13.0	6.3	9.6	4.2	11.5	49.7	12.2
As	%	0.12	0.057	0.20	0.031	0.14	0.059	0.072	0.076	0.014	0.12	0.039	0.084
As	ppm	1200	570	2000	310	1400	590	720	760	140	1200	390	844
ST	%	1.29	0.72	1.03	0.67	1.98	1.00	0.81	1.16	0.35	1.17	1.17	1.03
S=	%	1.26	0.68	0.92	0.54	1.60	0.87	0.71	0.96	0.35	1.02	1.00	0.90
SO4	%	0.1	0.1	0.1	0.2	0.8	0.2	0.1	0.1	0.1	0.2	0.3	0.2
S°	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	<0.05
CT	%	3.10	6.27	4.30	5.55	1.27	4.93	6.12	4.73	6.47	2.78	5.80	4.67

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Cg	%	< 0.05	0.12	0.18	0.17	0.11	0.18	0.14	0.28	< 0.05	0.16	0.34	0.162
TOC (leco)	%	0.13	0.43	0.56	0.33	1.03	0.47	0.47	0.43	0.16	0.77	0.67	0.495
Corg	%	< 0.08	0.31	0.38	0.16	0.92	0.29	0.33	0.15	< 0.11	0.61	0.33	0.334
TCM	%	2.60	5.16	3.00	4.15	0.16	3.55	5.01	3.30	6.16	1.55	3.95	3.51
CO3	%	13.0	25.8	15.0	20.8	0.81	17.8	25.0	16.5	30.8	7.76	19.8	17.5
CO2	%	10.4	20.6	12.0	16.6	0.65	14.2	20.0	13.2	24.6	6.21	15.8	14.0

Table 13-6 Gap Zone Hg, As, Sulfur Speciation, and Carbon Speciation

Parameter	Hole Comp	PG16-02 #2	PG16-06 #9	PG16-11 #10	PG16-11 #11	PG16-11 #12	PG16-11 #13	PG16-12 #15	PG16-12 #16	PG16-12 #17	PG16-16 #20	
Hg	g/t	4.1	4.6	4.5	3.6	1.9	0.9	2.2	1.0	1.5	6.1	3.0
As	%	0.066	0.20	0.22	0.29	0.11	0.15	0.32	0.20	0.13	0.23	0.19
As	ppm	660	2000	2200	2900	1100	1500	3200	2000	1300	2300	1916
ST	%	1.39	1.14	1.30	1.40	1.38	1.05	1.00	3.52	1.16	2.34	1.57
S=	%	1.22	1.06	1.11	1.29	1.34	0.99	0.91	3.15	1.06	1.90	1.40
SO4	%	0.3	0.1	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.4	0.2
S°	%	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	<0.05
CT	%	0.41	0.74	1.31	3.50	4.33	2.79	1.89	2.37	1.09	0.52	1.90
Cg	%	0.05	0.08	0.10	0.18	0.18	0.25	0.06	0.24	0.22	0.08	0.144
TOC (leco)	%	0.50	0.24	0.28	0.51	0.28	0.25	0.35	0.42	0.37	0.53	0.373
Corg	%	0.45	0.16	0.18	0.33	0.1	0.00	0.29	0.18	0.15	0.45	0.229
TCM	%	0.01	0.47	0.93	2.36	3.18	1.88	1.33	1.50	0.48	< 0.01	< 1.22
CO3	%	0.06	2.35	4.66	11.8	15.9	9.40	6.63	7.49	2.40	< 0.06	< 6.07
CO2	%	0.05	1.88	3.73	9.43	12.7	7.52	5.30	5.99	1.92	< 0.05	< 4.86

The following Table 13-7 and Table 13-8show the ICP multi-element analyses for the Helen and Gap Composites.

Table 13-7 Helen Zone ICP Multi-Element Analyses

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Al	g/t	29800	27000	37200	18300	45100	33600	29800	31500	16000	39500	27000	30436
Ba	g/t	232	213	215	215	677	224	214	185	1000	277	184	331
Be	g/t	0.56	0.76	0.88	0.64	1.06	0.88	0.76	0.78	0.46	0.98	0.78	1
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Ca	g/t	55000	175000	102000	153000	22400	124000	173000	111000	116000	57100	135000	111227
Cd	g/t	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	<10
Co	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Cr	g/t	72	64	83	52	91	70	70	80	69	95	77	75
Cu	g/t	36.1	17.3	25.4	9.9	32.5	21.6	19.2	23.1	10.9	23.0	20.9	22
Cu	%	0.004	0.002	0.003	0.001	0.003	0.002	0.002	0.002	0.001	0.002	0.002	0
Fe	g/t	14800	10300	14100	7130	20300	13700	12500	13000	7960	15600	12700	12917
K	g/t	13000	13200	15600	8500	22900	15200	13200	14500	6320	19100	12500	14002

Parameter	Hole Comp	AX-12 #5	AX-18 #6	PG17-07 #14	AX-27 #15	AX-27 #16	AX-27 #17	AX-27 #18	AX-27 #19	PG17-07 #20	PG17-07 #21	PG17-07 #22	
Li	g/t	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	<40
Mg	g/t	30600	8480	9890	3980	4310	10400	10000	11300	67000	12100	11700	16342
Mn	g/t	382	510	856	2670	94.6	802	777	965	378	669	771	807
Mo	g/t	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	<10
Na	g/t	545	356	334	272	470	376	360	332	471	443	331	390
Ni	g/t	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	<50
P	g/t	1970	3030	3920	5040	6760	3940	4030	3570	1090	6660	4100	4010
Pb	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Sb	g/t	64.5	< 50	62.6	< 50	95.8	57.9	< 50	79.4	< 50	81.6	94.9	77
Se	g/t	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	<40
Sn	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	<30
Sr	g/t	44.4	619	399	222	177	366	580	480	94.0	242	603	348
Ti	g/t	1650	1660	2100	1050	2560	1830	1770	1820	933	2330	1560	1751
Tl	g/t	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	<50
U	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	<30
V	g/t	63	49	64	28	81	60	57	67	37	73	59	58
Y	g/t	13.9	23.2	25.9	25.5	32.9	23.9	22.4	26.2	8.6	34.0	31.5	24
Zn	g/t	200	56	78	49	63	58	50	82	42	93	100	79

Table 13-8 Gap Zone ICP Multi-Element Analyses

Parameter	Hole Comp	PG16-02 #2	PG16-06 #9	PG16-11 #10	PG16-11 #11	PG16-11 #12	PG16-11 #13	PG16-12 #15	PG16-12 #16	PG16-12 #17	PG16-16 #20	
Al	g/t	31800	43000	40700	40900	39700	37900	37300	19500	33400	58400	38260
Ba	g/t	233	566	507	214	201	217	195	126	190	443	289
Be	g/t	0.44	0.78	0.60	1.00	0.92	0.68	0.58	0.46	0.76	0.50	0.67
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Ca	g/t	11600	26400	42800	85200	109000	76300	47400	63000	20700	14900	49730
Cd	g/t	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Co	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Cr	g/t	94	90	96	104	97	118	98	102	145	102	104.6
Cu	g/t	27.3	20.0	24.1	24.8	22.7	29.2	29.2	16.0	26.8	27.6	24.77
Cu	%	0.003	0.002	0.002	0.002	0.002	0.003	0.003	0.002	0.003	0.003	0.002
Fe	g/t	13200	13700	14200	18000	17700	18600	12400	35200	13100	22200	17830
K	g/t	14500	23000	19300	20400	18000	13200	16900	8270	14600	24200	17237
Li	g/t	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	<40
Mg	g/t	2200	6200	3430	11900	13600	5300	9810	4920	6720	2760	6684
Mn	g/t	61.5	182	333	915	1490	752	337	629	661	46.9	540.74
Mo	g/t	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	<10
Na	g/t	354	329	406	353	329	856	282	204	243	428	378
Ni	g/t	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	<50
P	g/t	4740	5360	4190	5300	3220	4060	4570	5080	3810	6330	4666
Pb	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Sb	g/t	849	255	810	76.0	312	50.8	739	51.3	< 50	94.7	359.8
Se	g/t	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40	< 40
Sn	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30
Sr	g/t	75.9	134	131	241	213	146	106	213	89.2	91.1	144.0
Ti	g/t	1780	2440	2090	2280	2180	2670	1760	923	1880	2580	2058.3
Tl	g/t	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50	< 50
U	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30
V	g/t	43	65	45	76	72	87	52	33	63	82	62
Y	g/t	20.6	29.0	21.3	29.2	23.2	22.6	24.7	20.1	22.3	28.7	24.2
Zn	g/t	122	61	23	26	45	85	19	119	105	20	62.5

JKTech SMC testing was performed on selected composites from the Helen Zone and Gap. The following Table 13-9 and Table 13-10 shows a summary of the comminution properties for selected composites from the Helen and Gap zones.

Table 13-9 Helen Zone Comminution Properties

Comp	HOLE-ID	A	b	A x b	Hardness Percentile	ta	DWI (kWh/m3)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	SCSE (kWh/t)	Relative Density
6	AX-18	68.6	0.52	35.7	71.0	0.4	7.4	21.4	16.1	8.3	10.3	2.7
16	AX-27	71.7	2.48	178.0	4.0	2.2	1.2	6.2	3.3	1.7	6.5	2.1
17	AX-27	57.8	0.93	53.8	38.0	0.6	4.5	15.9	10.8	5.6	8.6	2.4
19	AX-27	62.2	1.30	80.9	18.0	0.8	3.3	11.2	7.2	3.7	7.4	2.7

Table 13-10 Gap Zone Comminution Properties

Comp	HOLE-ID	A	b	A x b	Hardness Percentile	ta	DWI (kWh/m3)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	SCSE (kWh/t)	Relative Density
2	PG16-02	81.3	2.22	180.0	4.0	2.2	1.2	6.1	3.2	1.7	6.3	2.2
9	PG16-06	67.3	0.63	42.4	56.0	0.5	5.6	19.1	13.6	7.0	9.5	2.4
15	PG16-12	66.7	0.67	44.7	52.0	0.5	5.5	18.3	13.0	6.7	9.2	2.5
20	PG16-16	74.6	1.93	144.0	6.0	1.9	1.4	7.5	4.0	2.1	7.1	2.0

The head analyses indicate the following:

- The triplicate head fire assaying for both the Helen and Gap composites were reproducible and indicates finely disseminated gold and silver values and were consistent with the screen metallic assays which did not indicate the presence of significant native metals or a nugget effect;
- The mercury assays showed that the Helen composites ranged from 3.9 to 49.7 ppm The Gap composites ranged from 0.9ppm to 6.1 ppm;
- The arsenic assays showed that the Helen composites ranged from 140 ppm to 2000 ppm. The Gap composites ranged from 660ppm to 3200 ppm;
- The sulfur speciation indicated that bulk of the sulfur present in both the Helen and Gap composites is present as sulfide sulfur ranging from 0.35% to 1.60% in the Helen composites and 1.06% to 3.15% in the Gap composites;
- The carbon speciation for the Helen indicates the presence of carbonaceous material and significant amount of organic carbon. The total carbonaceous material (TCM) content in the Helen composites ranged from 0.16% to 6.16%. The total organic carbon (TOC by LECO) ranged from 0.13% to 1.03%;
- The carbon speciation for the Gap composites differed from Helen with a lower amount of TCM and a slightly lower amount of TOC. The total carbonaceous material (TCM) content

in the Gap composites ranged from <0.01% to 3.18%. The total organic carbon (TOC by LECO) ranged from 0.24% to 0.53%, and;

- Initial terms from a prospective toll processing plant indicated the following constituents in Table 13-11 will be included for process feed specifications:

Table 13-11 Toll Processing Feed Specifications

Constituent	Maximum Acceptable Level	Unit of Measure
Mercury	25	ppm
Arsenic	1200	ppm
Lead	100	ppm
Zinc	200	ppm
Total Copper	0.25	%
Cyanide Soluble Copper	250	ppm
Selenium	1	ppm
Barium	500	ppm
Chromium	100	ppm
Cobalt	100	ppm
Nickel	100	ppm
Cadmium	1	ppm
Free Gold	Any visible amount	

- The ICP multi-element analyses and the mercury and arsenic analyses indicated the following in regard to the specifications in Table 13.2.11:
    - Mercury:
        - Helen - Only one Helen Comp (22) had Hg higher than 25 ppm;
        - Gap - No composite in Gap had Hg higher than proposed spec.
    - Arsenic:
        - Helen – Two of the eleven composites (14 & 16) exceeded the proposed specification with two more at the maximum level. (5 and 21);
        - Gap - Eight of the ten composites exceed the proposed maximum.
    - Lead:
        - Helen – None of the composite exceed the lead limit;
        - Gap – None of the composites exceed the lead limit..
    - Zinc:
        - Helen – Only one composite (5) is at the zinc limit, all others are below he limit;
        - Gap – None of the composites exceed the zinc limit.
    - Total Copper:
        - Helen – None of the composite exceed the copper limit;

        - Gap – None of the composites exceed the copper limit.
    - Cyanide Soluble Copper:
        - Helen – None of the composite would exceed the cyanide soluble copper limit as the total copper is well below the soluble limit;
        - Gap – None of the composites exceed the cyanide soluble copper limit as the total copper is well below the soluble limit.
    - Selenium:
        - Helen – Assays were not performed to a 1 ppm limit so at present it cannot if be determined if composites meet the Se limit;
        - Gap – Assays were not performed to a 1 ppm limit so at present it cannot if be determined if composites meet the Se limit.
    - Barium:
        - Helen – Two of the Helen composites exceed the Barium limit (16 and 20);
        - Gap – Two of Gap composites exceed the Barium limit, (9 and 10).
    - Chromium:
        - Helen – None of the Helen composites exceed the Chromium limit.;
        - Gap – Five of the Gap composites exceed the Chromium limit, (11, 13, 16, 17, and 20).
    - Cobalt:
        - Helen – None of the Helen composites exceed the Nickel limit;
        - Gap – None of the Gap composites exceed the Nickel limit.
    - Nickel:
        - Helen – None of the Helen composites exceed the Cobalt limit;
        - Gap – None of the Gap composites exceed the Cobalt limit.
    - Cadmium:
        - Helen - Assays were not performed to a 1 ppm limit so at present it is not possible to determine if the composites meet the Cd limit;
        - Gap – Assays were not performed to a 1 ppm limit so at present it is not possible to determine if the composites meet the Cd limit.
- The SMC comminution testing indicated that the hardness of the Helen and Gap composites was variable ranging from soft to medium hard but were within normal ranges per the JKTech data base.

13.2.3. Mineralogy

QEMSCAN mineralogy tests were performed on selected composites from the Helen (5, 14, 16, & 21) and Gap (9, 11, 12, & 15) targeting sulfide and gangue mineralization. The following Table 13-12 summarizes the QEMSCAN mineral compositions for selected composites.

Table 13-12 QEMSCAN Mineral Compositions for Selected Helen & Gap Composites

Mineralogy Table	Units	5	14	16	21	Averages
Pyrite	%	5.00	1.49	3.23	2.10	2.96
Arsenopyrite	%	0.33	0.05	0.00	0.19	0.14
Quartz	%	58.10	49.60	62.50	58.40	57.15
Sericite/Muscovite	%	8.28	13.80	21.90	8.05	13.01
Dolomite	%	24.80	12.10	0.50	3.00	10.10
Calcite	%	0.04	11.30	4.50	3.00	4.71
Apatite	%	1.57	2.90	4.50	4.91	3.47
Barite	%	0.12	0.17	0.23	0.15	0.17
Fe-Oxides	%	0.09	0.09	0.04	0.08	0.08
K-Feldspar	%	1.01	2.32	3.19	3.07	2.40
Plagioclase	%	0.22	1.89	0.52	0.92	0.89
Rutile	%	0.38	0.52	0.55	0.58	0.51
Gap Composites		**9**	**11**	**12**	**15**	**Averages**
Pyrite	%	1.77	3.45	3.27	2.39	2.72
Arsenopyrite	%	0.49	0.71	0.32	0.80	0.58
Quartz	%	63.70	49.70	45.60	61.30	55.08
Sericite/Muscovite	%	18.00	16.00	15.00	14.70	15.93
Dolomite	%	2.48	10.50	14.20	6.22	8.35
Calcite	%	1.25	6.76	9.44	3.57	5.26
Apatite	%	4.09	3.91	2.44	3.31	3.44
Barite	%	0.25	0.16	0.15	0.12	0.17
Fe-Oxides	%	0.05	0.08	0.08	0.09	0.08
K-Feldspar	%	5.20	4.04	4.22	3.97	4.36
Plagioclase	%	0.23	0.46	0.68	0.39	0.44
Rutile	%	0.48	0.47	0.48	0.44	0.47

The mineral compositions show the following:

- Quartz is the dominant gangue mineral in the Helen and Gap composites and is on average slightly higher in the Helen composites;
- Sericite/Muscovite and Dolomite are the next most abundant gangue minerals;
- Pyrite was present in both groups of composites and averaged slightly high for the Helen composites at 2.96% but more variable than Gap composites which averaged 2.72%, and;
- The Helen composites contained some arsenopyrite averaging 0.14% as compared to an average of 0.58% for the Gap composites.

13.2.4. Whole Ore Bottle Cyanidation Tests

Baseline whole ore bottle roll cyanidation tests were performed on each of the Helen and Gap composites. The bottle roll conditions are summarized as follows:

- Twenty-four (24) hour duration with five solution monitoring periods;
- 0.5kg charge ground to 74 µm P_{80}, 50% solids w/w;
- 0.5 gpl NaCN, and;
- pH 10.5 to 11.0.

Table 13-13 shows a summary of the baseline whole ore cyanidation test results.

Table 13-13 Summary Whole Ore Cyanidation Test Results

Zone	Drill Hole	Composite	Grind Size P80, µm	Reagent Consumption kg/t of CN Feed NaCN	CaO	Au % Extraction (CN) 24 h PLS	Head Au, g/t CN Calc	Direct RST	Ag % Recovery (CN) 24 h PLS	Head Ag, g/t CN Calc	Direct
Helen	AX-12	5	79	0.21	1.12	25.3	7.35	7.55	20.1	2.6	2.3
	AX-18	6	78	0.32	1.17	0.2	8.22	8.47	6.2	< 2.6	2.0
	PG17-07	14	63	0.53	1.96	0.1	31.3	32.3	1.8	< 9.0	8.2
	AX-27	15	73	0.32	1.25	0.7	5.81	5.96	9.2	1.8	1.7
	AX-27	16	46	1.34	5.62	0.2	10.5	10.7	8.4	< 2.0	1.8
	AX-27	17	82	0.41	1.50	0.3	6.72	6.74	5.9	< 2.8	2.5
	AX-27	18	67	0.37	1.61	0.2	11.5	11.9	3.1	< 5.2	5.1
	AX-27	19	66	0.36	1.52	0.2	10.2	10.3	4.5	< 3.7	3.5
	PG17-07	20	74	0.14	0.89	90.8	4.42	4.44	46.1	2.2	2.1
	PG17-07	21	99	0.51	1.80	0.2	9.24	9.33	6.5	< 2.6	2.1
	PG17-07	22	64	0.63	1.62	0.3	7.29	7.60	4.2	< 4.0	3.5

Zone	Drill Hole	Composite	Grind Size P80, µm	Reagent Consumption kg/t of CN Feed		Au % Extraction (CN)	Head Au, g/t		Ag % Recovery (CN)	Head Ag, g/t	
				NaCN	CaO	24 h PLS	CN Calc	Direct RST	24 h PLS	CN Calc	Direct
Gap	PG16-02	2	68	1.08	2.20	1.8	3.24	4.42	5.7	39.3	37.9
	PG16-06	9	74	0.42	1.42	0.9	14.0	15.4	19.5	7.5	7.5
	PG16-11	10	75	0.85	2.29	5.5	17.6	17.7	33.9	4.5	4.4
	PG16-11	11	73	0.40	1.73	0.2	8.27	8.48	3.3	< 5.0	4.7
	PG16-11	12	70	0.42	1.27	0.4	5.63	5.58	4.5	< 3.6	3.3
	PG16-11	13	81	0.34	2.11	0.3	21.9	23.1	4.3	39.0	41.5
	PG16-12	15	73	0.80	1.34	0.1	37.6	37.0	2.1	< 11.3	11.2
	PG16-12	16	69	0.66	1.49	0.1	34.3	34.8	1.8	< 12.3	12.4
	PG16-12	17	71	0.65	1.51	0.2	10.5	11.3	2.6	< 12.7	12.3
	PG16-16	20	82	1.16	4.32	0.3	6.79	7.15	8.9	< 1.9	3.1

The whole ore bottle roll cyanidation tests showed the following:

- The gold and silver extractions were generally poor for both the Helen and Gap composites.
- The Helen composites gold and silver extractions ranged as follows:
  - Gold ranged from 0.1% to 90.8%
  - Silver ranged from 1.8% to 46.1%.
- The Gap composites gold and silver extractions ranged as follows:
  - Gold ranged from 0.1% to 5.5%.
  - Silver ranged from 1.8% to 33.9%.

13.2.5. Roasting and Calcine Cyanidation Tests

Batch roasting and direct cyanidation of the calcine tests were performed on each of the Helen and Gap composites.

The roasting conditions, based on those provided from a potential toll processing operator, are summarized as follows:

- Dry grind 0.5kg to a P_{80} of 74 microns;
- 2 stage roast;
- Heating rate 5°C/minute;
- Stage 1, 530°C for 30 min with O2/CO2 ratio of 60:40 at 2L/min sparged over bed, and;
- Stage 2, 570°C for 15 min with O2 at 2L/min sparged over bed.

The calcine bottle roll conditions are summarized as follows:

- Twenty-four (24) hour duration with five solution monitoring periods;
- 0.5 kg charge ground to a P_{80}74 µm, and 50% solids w/w;
- 35% slurry density;
- 0.5 gpl NaCN, and;
- pH 10.5 to 11.0.

Table 13-14 shows a summary of the roasting results and Table 13-15 shows the summary results for the direct cyanidation of the calcines.

Table 13-14 Initial Batch Roast Test Summary

Zone	Drill Hole	Composite	Test	Weight Loss %	S= Oxidation %	CO3 Oxidation %	Head Au, g/t	
							Calc	Direct
Helen	AX-12	5	RST-3	1.3	93.7	-1.7	7.27	7.55
	AX-18	6	RST-4	1.1	92.7	-1.2	8.42	8.47
	PG17-07	14	RST-5	2.6	94.7	1.3	32.2	32.3
	AX-27	15	RST-6	1.0	90.8	-1.8	5.96	5.96
	AX-27	16	RST-7	2.8	97.0	5.2	10.7	10.7
	AX-27	17	RST-8	1.3	93.2	-1.5	6.83	6.74
	AX-27	18	RST-9	1.7	93.1	-3.4	11.6	11.9
	AX-27	19	RST-10	1.7	94.9	-0.1	10.1	10.3
	PG17-07	20	RST-11	1.0	85.9	-5.1	4.50	4.44
	PG17-07	21	RST-12	3.5	94.3	11.4	9.54	9.33
	PG17-07	22	RST-13	3.0	95.1	5.4	7.41	7.60
Gap	PG16-02	2	RST-14	1.5	96.0	17.9	4.39	4.42
	PG16-06	9	RST-15	1.6	95.4	12.5	16.1	15.4
	PG16-11	10	RST-16	1.5	95.6	2.4	17.9	17.7
	PG16-11	11	RST-17	2.2	96.2	3.1	8.56	8.49
	PG16-11	12	RST-18	1.8	96.3	1.2	5.74	5.58
	PG16-11	13	RST-19	1.9	95.0	1.8	21.2	23.1
	PG16-12	15	RST-20	1.1	94.6	-0.1	37.8	37.0
	PG16-12	16	RST-21	1.7	98.1	22.9	34.5	34.8
	PG16-12	17	RST-22	1.5	95.4	23.3	11.1	11.30
	PG16-16	20	RST-23	3.1	97.5	19.3	7.02	7.15
	PG16-07	23	RST-24	1.6	87.9	18.4	7.01	7.63

Table 13-15 Initial Direct Calcine Cyanidation Test Summary

Zone	Drill Hole	Composite	Grind Size P80, µm	Reagent Consumption kg/t of CN Feed		Au % Extraction (CN)	Head Au, g/t		Ag % Recovery (CN)	Head Ag, g/t	
				NaCN	CaO	24 h PLS	CN Calc	Direct RST	24 h PLS	CN Calc	Direct
Helen	AX-12	5	71	0.16	4.48	74.5	7.63	7.27	53.7	1.7	2.3
	AX-18	6	58	0.15	1.23	67.8	8.47	8.42	56.5	1.6	2.0
	PG17-07	14	113	0.16	3.10	80.9	33.8	32.2	15.0	6.1	8.2
	AX-27	15	79	0.15	1.32	71.6	5.83	5.96	47.6	1.2	1.7
	AX-27	16	46	0.29	6.78	88.0	11.5	10.7	50.8	1.0	1.8
	AX-27	17	62	0.13	3.00	63.7	6.69	6.83	25.1	1.6	2.5
	AX-27	18	65	0.16	1.79	65.5	11.7	11.6	23.2	3.5	5.1
	AX-27	19	110	0.18	3.33	63.5	10.1	10.1	42.5	2.1	3.5
	PG17-07	20	76	0.13	0.87	92.8	4.52	4.50	9.6	4.5	2.1
	PG17-07	21	125	0.16	4.87	65.6	9.19	9.54	30.4	1.3	2.1
	PG17-07	22	64	0.18	1.65	90.8	7.47	7.41	53.8	1.7	3.5
Gap	PG16-02	2	84	0.40	6.08	89.4	4.36	4.39	56.0	39.1	37.9
	PG16-06	9	96	0.19	4.01	68.1	16.0	16.1	29.5	4.5	7.5
	PG16-11	10	70	0.36	3.77	71.9	18.0	17.9	60.5	4.1	4.4
	PG16-11	11	74	0.17	3.00	75.8	8.59	8.56	23.1	2.5	4.7
	PG16-11	12	72	0.15	2.79	72.1	5.64	5.74	31.9	2.2	3.3
	PG16-11	13	78	0.22	3.45	74.9	22.9	21.2	77.0	34.7	41.5
	PG16-12	15	70	0.19	2.66	54.4	39.1	37.8	41.1	9.0	11.2
	PG16-12	16	75	0.18	4.77	80.7	34.7	34.5	42.8	9.6	12.4
	PG16-12	17	74	0.18	5.68	58.0	10.8	11.1	23.3	6.5	12.3
	PG16-16	20	73	0.41	9.90	88.3	7.63	7.02	27.5	2.5	3.1

The roasting and calcine cyanidation tests indicated the following:

- The roasting effectively oxidized the sulfide content in both groups of composites with the Helen composites ranging from 85.9% to 97.0%sulfide oxidation while the Gap composites ranged from 87.9% to 98.1%;
- Carbonate oxidation in the Helen composites was generally low whereas the carbonate oxidation in the Gap composites was somewhat higher;
- The gold extractions by direct cyanidation of Helen composite calcines was variable ranging from 63.5% to 90.8%;

- The silver extractions by direct cyanidation of Helen composite calcines was variable ranging from 9.6% to 56.5%;
- The gold extractions by direct cyanidation of Gap composite calcines was variable ranging from 54.4% to 89.4%;
- The silver extractions by direct cyanidation of Gap composite calcines was also variable ranging from 23.1% to 77.0%, and;
- The data set was too small to establish any clear relations of between mineralogy and metal head grade and extractions although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using roasting and cyanidation.

13.2.6. Pressure Oxidation and Cyanidation of Pressure Oxidation Residues

Batch pressure oxidation (POX) and direct cyanidation of the pressure oxidation residue tests were performed on selected composites from the Helen (5, 14, 16, 21, & 22) and all of the Gap composites.

The POX conditions, based on those provided from a potential toll processing operator, are summarized as follows:

- Wet Grind P80 of 74 microns;
- 30% solids;
- 60°C acidulation with H_2SO_4 at pH 1.8, and;
- 225°C for 60 min with100psi of O2 overpressure;

The following Table 13-16 summarizes the pressure oxidation test data and Table 13-17 summarizes the direct cyanidation test results to date for the Helen and Gap composites.

Table 13-16 Pressure Oxidation Test Data

Sample	Acid Addition kg/t	Average Temp °C	Average %O2 Off Gas	POX Pulp pH (units)	Residue Assays ST %	S= %	CT %	CO3 %
Helen Comp #5	258	223	96	1.31	4.42	0.08	0.04	0.05
Helen Comp #14	247	224	90	1.70	7.43	0.08	0.71	0.32
Helen Comp #16	12	223	94	1.36	1.44	0.05	1.09	0.17
Helen Comp #21	135	224	94	1.74	4.02	0.08	1.02	0.32
Helen Comp #22	313	224	87	6.59	9.95	0.34	1.06	0.7
Gao #2	6	224	97	1.24	0.65	< 0.05	0.41	< 0.05
Gap #9	58	224	94	1.28	1.74	< 0.05	0.20	< 0.05
Gap #10	76	223	93	1.69	3.14	< 0.05	0.24	< 0.05

Sample	Acid Addition kg/t	Average Temp °C	Average %O2 Off Gas	POX Pulp pH (units)	Residue Assays ST %	S= %	CT %	CO3 %
Gap #11	223	225	93	1.28	6.1	0.07	0.56	< 0.05
Gap #12	277	224	95	1.42	7.71	0.08	0.42	0.08
GAP #13	170	224	90	1.54	5.49	0.07	0.47	< 0.05
GAP #15	126	224	94	1.25	3.46	0.05	0.32	< 0.05
GAP #16	143	224	95	0.98	4.62	0.10	0.48	< 0.05
GAP #17	58	224	91	1.23	1.34	< 0.05	0.47	< 0.05
GAP #20	8	225	92	1.25	0.96	< 0.05	0.48	< 0.05

Table 13-17 Direct Cyanidation of POX Residue Test Data

Zone	Drill Hole	Composite	CN Residue Size P80, µm	Reagent Consumption kg/t of CN Feed		Au % Extraction (CN)	Head Au, g/t		Ag % Recovery (CN)	Head Ag, g/t	
				NaCN	CaO	24 h PLS	CN Calc	Direct RST	24 h PLS	CN Calc	Direct
Helen	AX-12	5	53	0.13	1.20	96.6	6.78	8.58	69.6	1.5	2.6
	PG17-07	14	29	0.16	1.53	6.6	29.9	32.3	45.8	6.6	8.2
	AX-27	16	21	0.12	2.77	2.8	9.72	11.4	30.0	2.4	1.9
	PG17-07	21	41	0.14	2.18	0.6	8.49	9.91	13.4	2.9	2.2
	PG17-07	22	42	0.24	1.45	0.3	7.01	7.49	6.7	3.3	3.4
GAP	PG16-02	2	49	0.12	1.47	24.2	3.81	4.60	81.7	34.7	39.3
	PG16-06	9	64	0.11	1.30	72.2	13.2	16.5	81.7	4.6	8.0
	PG16-11	10	51	0.11	1.12	73.6	14.3	18.1	62.7	1.8	4.5
	PG16-11	11	38	0.16	1.69	6.9	7.58	8.56	52.5	3.3	4.7
	PG16-11	12	38	0.10	1.81	5.7	5.12	5.64	33.1	1.3	3.3
	PG16-11	13	46	0.22	1.07	38.0	19.8	23.3	64.7	32.9	41.9
	PG16-12	15	51	0.21	1.56	49.3	29.6	38.3	66.2	10.3	11.6
	PG16-12	16	54	0.13	1.92	57.3	27.7	36.2	76.3	10.2	12.9
	PG16-12	17	50	0.15	1.65	13.6	10.1	12.0	62.6	8.1	13.0
	PG16-16	20	42	0.11	4.42	19.5	5.90	7.59	52.5	1.2	3.3

The pressure oxidation and POX residue cyanidation tests indicated the following:

- The POX step effectively oxidized the sulfide content in both groups of composites ;
- Carbonate removal in the Helen composites averaged 97.2% whereas the carbonate removal in the Gap composites averaged 82.5% however this average is skewed due to Gap composites 2 and 20 having very low head carbonate contents;

- The gold extractions by direct cyanidation of Helen composite POX residues generally very low ranging from 0.3% to 96.6%;
- The silver extractions by direct cyanidation of Helen composite POX residues was variable ranging from 6.7% to 69.6%;
- The gold extractions by direct cyanidation of Gap composite POX residues was variable ranging from 5.7% to 73.6%;
- The silver extractions by direct cyanidation of Gap composite POX residues was also variable ranging from 52.5% to 81.7%, and;
- The data set was too small to establish any clear relations of between mineralogy and metal head grade and extractions although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using pressure oxidation and residue cyanidation.

13.3.2017 Metallurgical Test Work – Phase 2

The Phase 1 2017 roasting and POX cyanidation testing were lower than anticipated. The conjectured reasoning for the low and variable gold extractions by the two processes are summarized as follows:

- The roasting step effectively removed the sulfide content in both the Helen and Gap composites, however, the roasting step did not appear to effectively treat the pregnant solution robbing factors, such as the carbonaceous content and in particular the organic carbon content, to make them inert, and;
- Pressure oxidation step effectively removed the sulfide content and most of the carbonate content but as with roasting did not render pregnant solution robbing factors inert.

A second phase of testing was conducted to investigate the reasons for the low roaster and POX metal extractions observed in Phase 1 tests. The program first consisted of rerunning roasting and POX tests on selected composites from the Helen and Gap. The calcines and POX residues resulting from each of the rerun oxidation treatments were split in two. One half of each spilt was subjected to direct cyanidation as was performed in phase 1. The second half of each split was subjected to carbon-in-leach (CIL) cyanidation. The CIL leach was used as a means to partially diagnose if pregnant solution robbing was causing the low extractions.

Helen Composites 6, 14, 19, and 2 and Gap composites 10, 11, 15, 16, and 20 were chosen for rerun testing.

The same roasting and pressure oxidation conditions were used for the rerun tests. The cyanidation conditions were modified to extend the leach time to 48 hours instead of 24 hours in Phase 1.

Solution monitoring at the 24-hour mark was included to have a comparison reference to Phase 1 tests.

The roasting data for the reruns in comparison to the initial roasting data is shown in Table 13-18. The roasting rerun cyanidation data is summarized in Table 13-19.

The POX data for the reruns in comparison to the initial POX data is shown in Table 13-20. The POX rerun cyanidation data is summarized in Table 13-21.

Table 13-18 Rerun Roasting Data

Zone	Drill Hole	Comp.	Test	Weight Loss %	S= Oxidation %	CO3 Oxidation %	TOC Oxidation %	Au, g/t	
								Residue	Direct
Helen	AX-18	6	RST-4	1.1	92.7	-1.2		8.42	8.47
	AX-18	6	RST-26	1.3	92.7	0.6	77.1	8.54	8.47
	PG17-07	14	RST-5	2.6	94.7	1.3		32.2	32.3
	PG17-07	14	RST-27	2.3	94.7	0.3	89.5	32.6	32.3
	AX-27	19	RST-10	1.7	94.9	-0.1		10.1	10.3
	AX-27	19	RST-28	2.0	94.9	1.4	84.0	9.97	10.3
	PG17-07	21	RST-12	3.5	94.3	11.4		9.54	9.33
	PG17-07	21	RST-29	3.3	95.3	7.1	90.0	9.38	9.33
Gap	PG16-11	10	RST-16	1.5	95.6	2.4		17.9	17.7
	PG16-11	10	RST-30	1.2	95.5	0.1	78.8	17.9	17.7
	PG16-11	11	RST-17	2.2	96.2	3.1		8.56	8.49
	PG16-11	11	RST-31	1.4	96.2	1.4	86.5	8.30	8.49
	PG16-12	15	RST-20	1.1	94.6	-0.1		37.8	37.0
	PG16-12	15	RST-32	2.2	94.6	1.2	72.1	37.6	37.0
	PG16-12	16	RST-21	1.7	98.1	22.9		34.5	34.8
	PG16-12	16	RST-33	1.8	97.8	20.2	64.9	34.0	34.8
	PG16-16	20	RST-23	3.1	97.5	19.3		7.02	7.15
	PG16-16	20	RST-34	3.0	97.4	19.2	90.9	6.95	7.15

Table 13-19 Roaster Rerun Calcine Cyanidation Data

Zone	Drill Hole	Comp	CN Test	Grind Size P80 µm	Reagent Cons.kg/t of CN Feed		Au % Ext CN		Au % Ext CIL	Head Au g/t			Ag % Recovery (CN)		Ag % Ext CIL	Head AG g/t		
					NaCN	CaO	24 h PLS	48 h PLS	48 h	CN Calc	CIL Calc	Direct RST	24 h PLS	48 h PLS	48 h	CN Calc	CIL Calc	Direct Roast
Helen	AX-18	6	CN-62	53	0.12	1.30	68	69.8		8.46	-	8.54	30	30.6		2.7	-	2.8
			CIL-1		0.52	1.33	-	-	72.4	-	8.54		-	-	38.8	-	2.8	
	PG17-07	14	CN-63	96	0.22	3.98	65	67.9	-	30.9	-	32.6	19	20.0		8.4	-	9.5
			CIL-2		0.67	3.93	-	-	72.9	-	32.1		-	-	19.0	-	9.4	
	AX-27	19	CN-64	100	0.18	3.33	77	75.4	-	10.0	-	10.0	24	26.8		2.6	-	3.4
			CIL-3		0.64	3.22	-	-	77.9	-	10.1		-	-	23.6	-	3.9	
	PG17-07	21	CN-65	101	0.27	4.79	79	78.7	-	8.96	-	9.38	17	18.4		1.7	-	2.5
			CIL-4		0.83	5.03	-	-	81.6	-	9.34		-	-	23.0	-	2.0	
GAP	PG16-11	10	CN-66	70	0.19	3.03	69	70.8	-	17.7	-	17.9	54	54.1		3.9	-	4.5
			CIL-5		0.86	3.31	-	-	73.4	-	17.8		-	-	55.3	-	4.3	
	PG16-11	11	CN-67	71	0.24	3.46	74	69.2	-	7.50	-	8.30	20	20.8		3.4	-	5.1
			CIL-6		0.71	3.46	-	-	74.1	-	8.49		-	-	19.2	-	5.0	
	PG16-12	15	CN-68	71	0.65	2.80	54	58.6	-	35.5	-	37.6	33	34.0		9.6	-	11.7
			CIL-7		0.64	2.95	-	-	60.3	-	37.1		-	-	34.1	-	10.9	
	PG16-12	16	CN-69	73	0.29	5.31	76	76.0	-	30.7	-	34.0	48	53.0		10.8	-	13.5
			CIL-8		0.73	5.29	-	-	79.9	-	34.2		-	-	56.1	-	13.0	
	PG16-16	20	CN-70	103	0.33	10.46	84	84.0	-	6.65	-	6.95	41	42.1		1.7	-	2.2
			CIL-9		1.06	10.01	-	-	85.7	-	6.96		-	-	44.1	-	2.1	

Table 13-20 POX Rerun Test Data

Sample	Acid Addition kg/t	Average Temp °C	Average O_2 % Off Gas	POX Pulp pH (units)	Residue Assays			
					ST %	S= %	CT %	CO_3 %
Helen Comp #6	431	223	85	1.93	12.2	< 0.05	0.38	< 0.05
Helen Comp #14	247	224	90	1.70	7.43	0.08	0.71	0.32
Helen Comp #14	243	224	95	2.61	7.56	0.09	0.68	0.24
Helen Comp #19	261	224	90	5.14	8.26	0.34	0.70	0.28
Helen Comp #21	135	224	94	1.74	4.02	0.08	1.02	0.32
Helen Comp #21	128	224	85	1.98	4.11	0.39	1.00	0.13
Gap #10	76	223	93	1.69	3.14	< 0.05	0.24	< 0.05
Gap #10	79	225	91	1.24	3.10	0.10	0.22	< 0.05
Gap #11	223	225	93	1.28	6.1	0.07	0.56	< 0.05
Gap #11	203	224	86	1.58	6.18	0.07	0.57	< 0.05
Gap #15	126	224	94	1.25	3.46	0.05	0.32	< 0.05
Gap #15	132	223	95	1.21	3.31	0.07	0.31	< 0.05

Sample	Acid Addition kg/t	Average Temp °C	Average O_2 % Off Gas	POX Pulp pH (units)	Residue Assays			
					ST %	S= %	CT %	CO_3 %
Gap #16	143	224	95	0.98	0.96	< 0.05	0.48	< 0.05
Gap #16	134	223	92	1.02	1.08	0.11	0.48	< 0.05
Gap #20	8	225	92	1.25	0.96	< 0.05	0.48	< 0.05
Gap #20	8	225	92	1.05	1.08	0.11	0.48	< 0.05

Table 13-21 POX Rerun Residue Cyanidation Data

Zone	Drill Hole	Comp	CN Test	Grind Size P80 µm	Reagent Cons.kg/t of CN Feed		Au % Ext CN		Au % Ext CIL	Head Au g/t			Ag % Recovery (CN)		Ag % Ext CIL	Head AG g/t		
					NaCN	CaO	24 h PLS	48 h PLS	48 h	CN Calc	CIL Calc	Direct	24 h PLS	48 h PLS	48 h	CN Calc	CIL Calc	Direct Roast
Helen	AX-18	6	CN-71	< 38	0.11	1.88	1.7	1.5	-	7.36	-	7.69	53	55.3	-	2.7	-	1.8
			CIL-10		0.44	1.92	-	-	68.9	-	7.78		-	-	77.8	-	2.7	
	PG17-07	14	CN-72	< 38	0.14	2.33	6.7	5.1	-	28.7	-	31.1	-	62.0	-	5.8	-	7.9
			CIL-11		0.33	2.79	-	-	81.9	-	30.7		-	-	86.4	-	8.8	
	AX-27	19	CN-73	< 38	0.17	2.25	1.1	0.6	-	9.5	-	10.4	40	40.2	-	3.7	-	3.5
			CIL-12		0.60	2.23	-	-	63.5	-	9.7		-	-	86.9	-	4.6	
	PG17-07	21	CN-74		0.13	2.15	0.6	0.4	-	9.10	-	9.75	35	36.2	-	4.4	-	2.2
			CIL-13		0.57	2.26	-	-	62.3	-	9.31		-	-	76.6	-	4.3	
GAP	PG16-11	10	CN-75		0.07	6.75	77	77.8	-	13.0	-	18.2	87	78.7	-	3.8	-	4.5
			CIL-14		0.42	1.71	-	-	95.0	-	17.5		-	-	82.4	-	4.5	
	PG16-11	11	CN-79		0.10	1.81	2.3	1.6	-	7.90	-	8.49	25	19.9	-	3.4	-	4.7
			CIL-18		0.61	1.76	-	-	70.5	-	7.95		-	-	72.2	-	4.0	
	PG16-12	15	CN-76		0.13	1.73	57	54.3	-	25.4	-	38.4	72	73.1	-	8.2	-	11.6
			CIL-15		0.48	1.99	-	-	94.0	-	38.1		-	-	73.1	-	10.4	
	PG16-12	16	CN-78		0.22	2.20	54	43.3	-	26.5	-	35.2	87	84.1	-	10.1	-	12.5
			CIL-17		1.06	2.29	-	-	88.5	-	32.8		-	-	87.0	-	12.3	
	PG16-16	20	CN-77		0.14	2.31	8.5	7.3	-	8.39	-	7.75	55	52.1	-	1.0	-	3.4
			CIL-16		0.45	2.65	-	-	85.8	-	7.13		-	-	71.6	-	1.8	

The rerun roasting tests indicated the following:

- The weight loss, sulfur oxidation, and CO3 oxidation were similar to the initial tests;
- The TOC oxidation for the rerun showed that the major portion of the organic carbon content was oxidized;
- The Helen composites direct cyanidation of the rerun calcines 24-hour gold extraction ranged from 67.7% to 78.7 % at 48 hours whereas the gold extractions for the 48-hour CIL tests ranged from 72.4% to 81.6% about 3% higher than the direct cyanidation for each of the respective tests;
- The Helen Zone direct cyanidation of the rerun calcines 48-hour silver extraction ranged from 18.4% to 30.6% whereas the gold extractions for the 48-hour CIL tests ranged from 19.0% to 38.8% or about 1 to 2% higher than the direct cyanidation;
- The Gap composites direct cyanidation of the rerun calcines 48- hour gold extraction ranged from 58.6% to 84.0% whereas the gold extractions for the 48-hour CIL test ranged from 60.3% to 85.7% or 3.0% higher for each of the respective composites;
- The Gap composites direct cyanidation of the rerun calcines 48-hour silver extraction ranged from 20.8% to 54,1% whereas the gold extractions for the 48-hour CIL test ranged from 19.2% to 56.1% or about 1.0% higher than the direct cyanidation for each of the respective composites, and;
- The rerun roasting tests on the selected composites confirmed the supposition that pregnant solution robbing occurs in direct cyanidation of the calcines and that CIL cyanidation can increase gold extractions and silver extractions versus direct cyanidation.

The rerun pressure oxidation tests indicated the following:

- The Helen composites direct cyanidation of the rerun POX residues 48-hour gold extractions ranged from 0.6% to5.1% whereas the gold extractions for the 48-hour CIL tests ranged from 62.3% to 81.9% %, significantly higher than the direct cyanidation;
- The Helen direct cyanidation of the rerun POX residues 48-hour silver extraction ranged from 36.2% to 86.9% whereas the silver extractions for the 48-hour CIL test ranged from 76.8% to 86.9% significantly higher than the direct cyanidation;
- The Gap composites direct cyanidation of the POX residues 48-hour gold extractions ranged from 1.6% to 77.8% whereas the average gold extractions for the 48-hour CIL tests ranged from 70.5% to 95.9%, significantly higher than the direct cyanidation;
- The Gap composites direct cyanidation of the rerun POX residues 48-hour silver extractions ranged from 19.9% to 84.1% whereas the gold extractions for the 48-hour CIL test ranged from 71.6% to 87.0%, significantly higher than the direct cyanidation, and;

- The rerun POX tests on the selected composites confirmed the supposition that pregnant solution robbing occurs in direct cyanidation of the calcines and that CIL cyanidation can increase gold extractions and silver extractions very significantly for both the Helen and Gap composites tested versus direct cyanidation.

13.3.1. Rerun Calcine Diagnostic Leach

The rerun roasting tests showed that the application of CIL cyanidation for the calcine leach could increase precious metal extractions, the gold extraction was still somewhat lower than expected. Diagnostic leaching of the rerun calcine cyanidation residue was conducted to investigate the distribution of gold in the leached calcine. The procedure employed was as follows:

- Perform a hydrochloric acid leach on a CIL residue to attempt to liberate gold possibly associated with any pyrrhotite, calcites, ferrites, dolomite, galena and hematite;
- Perform a cyanide leach of acid leach residue to determine gold liberated from the acid leach, and;
- The gold remaining in the cyanide residue is estimated to be locked or associated with siliceous gangue.

The calcine diagnostic leach results are summarized in Table 13.3.5 as follows:

Table 13-22 Summary Rerun Calcine Diagnostic Leach

Zone	Drill Hole	Composite	Test	Estimated Au Distribution In Ferrites, Sulfides, Or Carbonates/Whole Ore Basis	Estimated Au Distribution in Siliceous Gangue/ Whole Ore Basis
Helen	AX-18	6 rerun	RST-26	9.6	18.0
	PG17-07	14 rerun	RST-27	17.9	9.2
	AX-27	19 rerun	RST-28	8.2	13.9
	PG17-07	21 rerun	RST-29	8.2	10.2
Gap	PG16-11	10 rerun	RST-30	23.2	3.4
	PG16-11	11 rerun	RST-31	11.9	14.0
	PG16-12	15 rerun	RST-32	35.9	3.8
	PG16-12	16 rerun	RST-33	12.1	8.0
	PG16-16	20 rerun	RST-34	11.7	2.6

The rerun calcine cyanide residue diagnostic leach indicated the following:

- The estimated amount of gold associated with the iron oxides, ferrites or calcite in the Helen composites leached calcine residues ranged from 8.2% to 17.9% and averaged 11.0%, with the remaining gold estimated to be in siliceous gangue which ranged from 9.2% to 18.0% and averaged 12.8%

- The estimated amount of gold associated with the iron oxides, ferrites or calcite in the Gap composites leached calcine residues ranged from 11.7% to 35.9%, with the remaining gold estimated to be in siliceous gangue which ranged from 2.6% to 14.0%;
- The data for the composites tested indicated that the Helen Zone likely has more gold associated with siliceous material than the Gap composites which showed a greater amount of gold associated with the iron oxides, ferrites, or calcite following roasting, and;
- The data also suggests that for the roasting conditions from a potential toll roasting operation may not be optimal for the Helen or Gap material.

13.3.2. Roasting Optimization Testing

The rerun leached calcine diagnostic test results plus the variable and somewhat low roaster and calcine gold extractions indicate that the roasting conditions may not be optimal for the Helen and Gap composites tested. A short roasting optimization program was conducted to determine if there were more optimal conditions than the projected toll roaster conditions obtained from a prospective toll processor.

Two composites from the Helen Zone (6 and 19) and one from the Gap (15) were selected to perform the roast optimization tests.

Four sets of roasting conditions were chosen for tests summarized in Table 13-23:

Table 13-23 Optimization Roast Test Conditions

Parameter	Conditions #1	Conditions #2	Conditions #3	Conditions #3
Stage 1 Roast				
Temperature ° C	530	530	530	650
CO2 Flow - lpm	2	2.4	2.4	
O2 flow - lpm	3.6	3.6	3.6	6
Air flow -lpm	--	--	--	6
Time - minutes	30	30	30	120
Stage 2 Roast				
Temperature ° C	570	590	610	
O2 flow - lpm	6	6	6	
Time - minutes	30	30	30	
Temp. Ramp Up - ° C per min	5	5	5	
Calcine CIL Cyanidation				
NaCN -gpl	0.5			5
pH	10.5 to 11.5	10.5 to 11.5	10.5 to 11.5	10.5 to 11.5
Leach Time - hours	48	48	48	48

The first set of conditions are those used for the phase 1 tests and are those per the prospective toll roasting operator roaster. Condition sets 2 and 3 are variations on the condition 1 conditions and though to be achievable in the prospective toll roasting operator roaster. The fourth set of conditions are those used by KCA in the Victoria Gold testing and are thought to be a set extreme conditions and probably not achievable in the prospective toll roasting operator roaster.

Table 13-24 shows the roasting data for the optimization roasts. Table 13-25 shows the calcine CIL cyanidation data for the optimization roast test.

Table 13-24 Optimization Roast Data

Zone	Drill Hole	Comp	Test	Roast Condition #	Weight Loss %	TCM Oxidation %	TOC Oxidation %	S= Oxidation %	FE Species		
									Fe %	Fe+2 %	Fe+3 %
Helen	AX-18	6	RST-37	1	1.6	97.5	84.0	92.8	1.09	0.27	0.80
			RST-40	2	1.5	97.3	86.2	92.8	1.08	0.31	0.80
			RST-43	3	1.5	96.4	86.3	92.8	1.08	0.33	0.80
			RST-46	4	3.3	96.6	86.5	92.9	1.06	0.21	0.90
Helen	AX-27	19	RST-35	1	2.2	96.1	84.1	94.9	1.34	0.17	1.20
			RST-38	2	2.2	95.3	84.1	94.9	1.34	0.17	1.20
			RST-41	3	2.3	94.1	84.1	94.9	1.38	0.17	1.20
			RST-44	4	4.5	94.5	84.4	95.0	1.41	< 0.15	1.40
GAP	PG16-12	15	RST-36	1	1.6	89.6	85.9	94.6	1.36	0.39	1.00
			RST-39	2	1.7	89.7	86.0	94.6	1.36	0.35	1.00
			RST-42	3	2.0	84.5	86.0	94.6	1.37	0.33	1.00
			RST-45	4	3.0	84.0	86.1	94.7	1.40	0.18	1.20

Table 13-25 Optimization Roast Calcine CIL Cyanidation Data

Zone	Drill Hole	Comp	Roast Test	Roast Test Weight Loss %	Reagent Consumption kg/t of CN Feed		Au % Extraction (CN)	Au % Extraction (CIL)	Head Au, g/t			Ag % Extraction (CN)	Ag % Extraction (CIL)	Head Ag, g/t		
					NaCN	CaO	48 h	48 h	CN Calc	CIL Calc	Direct RST	48 h	48 h	CN Calc	CIL Calc	Direct RST
Helen	AX-18	6	37	1.6	0.05 0.30	0.00 0.02	**71.1** -	- **72.8**	8.61 -	- 9.19	8.61	**31.1** -	- **30.2**	0.9 -	- 1.2	2.0
			40	1.5	0.58 0.46	0.33 0.38	**70.6** -	- **73.5**	8.08 -	- 8.77	8.60	**45.1** -	- **32.8**	0.9 -	- 1.3	2.0
			43	1.5	0.03 0.48	0.42 0.53	**68.8** -	- **68.6**	8.71 -	- 8.74	8.60	**41.9** -	- **41.2**	0.9 -	- 1.2	2.0
			46	3.3	1.39 2.11	0.00 0.00	**79.3** -	- **84.3**	10.5 -	- 8.75	10.7	**59.1** -	- **36.3**	2.7 -	- 1.3	3.6
Helen	AX-27	19	35	2.2	0.11 0.59	2.70 2.67	**76.6** -	- **79.5**	9.79 -	- 10.80	10.5	**28.2** -	- **34.4**	2.2 -	- 2.5	3.6
			38	2.2	0.49 1.26	2.67 2.81	**80.2** -	- **82.1**	9.67 -	- 10.7	10.5	**25.5** -	- **32.2**	1.9 -	- 1.8	3.6
			41	2.3	0.08 0.51	2.21 2.39	**77.6** -	- **79.0**	10.0 -	- 10.6	10.5	**65.4** -	- **36.7**	3.5 -	- 2.4	3.6
			44	4.5	1.24 2.24	0.00 0.00	**87.9** -	- **91.7**	10.3 -	- 10.8	10.8	**6.1** -	- **33.8**	1.0 -	- 1.2	3.7
Gap	PG16-12	15	36	1.6	0.18 0.65	2.85 3.01	**67.1** -	- **64.1**	37.2 -	- 39.0	37.6	**29.2** -	- **24.4**	7.2 -	- 9.1	11.4
			39	1.7	0.52 0.66	2.94 2.82	**60.8** -	- **64.8**	38.0 -	- 40.1	37.6	**39.2** -	- **33.7**	7.7 -	- 8.1	11.4
			42	2.0	0.13 0.55	2.44 2.50	**65.1** -	- **66.1**	36.4 -	- 38.6	37.8	**32.3** -	- **34.8**	6.7 -	- 6.9	11.4
			45	3.0	1.26 2.03	0.04 0.00	**82.7** -	- **83.5**	43.4 -	- 39.7	38.1	**22.5** -	- **32.4**	4.5 -	- 4.1	11.5

The optimization roasting shows the following:

- Roasting conditions 1 to 3 gave very similar results in regard to weight, TCM, TOC, sulfide oxidation and iron speciation;
- The condition 4 roasting weight loss and iron speciation differed significantly from those achieved using conditions 1 to 3. The condition 4 weight loss for each the composites was about double that produced using conditions 1 to 3. Additionally, the amount of Fe^{+2} from condition 4 was about 30% to 50% lower than for condition 1 to 3 and the Fe^{+3} was higher than observed using conditions 1 to 3 for each composite;
- The calcine cyanidation data shows that gold and silver extractions were very similar using conditions 1 to 3. The gold and silver extractions were significantly higher for all three composites using the condition 4 roasting and cyanidation parameters, and;
- The higher gold and silver extractions for the condition 4 parameters likely is due to increased conversion of sulfide iron to the Fe +3 state resulting in a more permeable iron oxide matrix for cyanidation.

13.4. Future Metallurgical Testing

The initial metallurgical test program indicated that the Helen and Gap Resources appear to be amenable in various degrees to roasting and pressure oxidation followed by CIL cyanidation using conditions at a potential toll processing operation. Additionally, the optimization tests indicated a potential to increase roasting gold extractions using more extreme conditions than the prospective toll processing operator. Such conditions could possibly be considered should some type of roasting facility be considered for the project. The initial group of composites were chosen to investigate metallurgy spatially within the two resources. Additional work will be required to advance the project to the next phase.

The additional work should continue to investigate variability of metallurgy within the resources with the major objectives as follows:

- Assess variability of the responses to roasting and calcine cyanidation across the resources;
- Assess variability of the responses to pressure oxidation and residue cyanidation across the resources;
- Testing should attempt to establish head grade and extraction relations for use in more detailed resource modelling;
- Mineralogy impacts need to be established and geologic domains within each resource need to be determined, and;
- Additional comminution data should be collected to assess variability within the resources.

The suggested next phase of metallurgical investigations, for preliminary planning purposes, is described as follows:

- Identify thirty to forty drill hole intervals within each resource for metallurgical testing which represent significant tonnages or grades within the resources:
    - Perform head analyses on each composite as follows;
    - Fire Assay in Triplicate for Au and Ag;
    - Screen Metallics Assay for Au and Ag;
    - Cyanide Soluble Au and Ag;
    - High precision Assay for Hg and AS;
    - Sulfur Speciation – Total S, Sulfide Sulfur, Sulfate, Native Sulfur;
    - Carbon Speciation – Total Carbon, Graphite, Total Organic Carbon, Total Carbonaceous Carbon, CO_3, and CO_2;
    - ICP Multi-element Analysis – Al, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Sn, Sr, Ti, U, V, Y, Zn.
- Comminution Properties – perform on 10 to 20 composites from each resource;
- Perform roasting and CIL calcine cyanidation on each composite;
- Perform pressure oxidation and residue CIL cyanidation on each composite;
- Perform mineralogy on five to ten composites from each resource;
- Consider flotation and concentrate leaching on three to five composites from each resource, and;
- Consider alternative oxidation processes such as the Albion process on three to five composites from each resource.

The estimated cost for the suggested next phase metallurgical program is $640,000 to $850,000 based on pricing obtained for the 2017 test work. These costs do not include the flotation or alternative oxidation process investigation. Initial investigations for those processes could add an additional $100,000 or more to the next phase work.

13.5. Conclusions and Recommendations:

The following are the major conclusions and recommendations from the 2017 Helen and Gap composite metallurgical test program:

13.5.1. Conclusions:

1. Head assaying for the both the Helen Zone and Gap indicated that the gold in the two resources will likely be finely disseminated and will not likely have a significant coarse or nugget gold content;

2. The mineralogy of the Helen and Gap resources differ in two significant areas, the first being that the Helen appears to be lower in arsenic content than the Gap resource and that the Gap resource appears to be lower on average in TCM and TOC than the Helen resource;
3. The Helen composite arsenic assays indicate the resource lower in arsenic content that the Gap resource;
4. The Helen and Gap resources based on the composites tested appear to be generally refractory to conventional whole cyanidation and will need some type of oxidation process to significantly increase gold extractions over whole cyanidation;
5. Based on the composites tested the Helen Zone appear to generally be more amenable to Roasting and CIL cyanidation, however, there may be areas that are more amenable or can only be treated using pressure oxidation and residue CIL cyanidation;
6. Based on the composites tested, the Gap resource appears to generally be more amenable to pressure oxidation followed by residue CIL cyanidation, however, there may be areas that are more amenable or can only be treated using roasting and calcine CIL cyanidation;
7. The data set was too small to establish any clear relations of between mineralogy and metal head grade and extractions for either resource although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using either roasting and calcine cyanidation or pressure oxidation and residue cyanidation.

13.5.2. Recommendations

1. Additional metallurgical testing will be needed to thoroughly investigate the variability and viability of Helen and Gap resources to roasting and pressure oxidation with CIL cyanidation for which a program evaluating thirty to forty composites from each resource is suggested with objectives as follows:
    - Assess variability of the responses to roasting and calcine cyanidation across the resources;
    - Assess variability of the responses to pressure oxidation and residue cyanidation across the resources;
    - Consider some POX optimization tests such as pre-acidulation ahead of the POX process;
    - Testing should attempt to establish head grade and extraction relations for use in more detailed resource modelling;
    - Mineralogy impacts need to be established and geologic domains within each resource need to be determined, and;
    - Additional comminution data should be collected to assess variability within the resources.
2. In addition to evaluating resource process by a toll processing operator, consideration should be given to evaluate onsite processing;

3. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
4. Consider flotation tests to pre-float carbonates, and;
5. Consider other mill design tests as alternative to toll processing. These would include roasting, POX optimization tests, and solid liquid separation tests.

14. Mineral Resource Estimates

14.1. Introduction

The mineral resource estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 (Canadian Securities Administrators, 2011) and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (Canadian Instirute of Mining, Metallurgy, and Petroleum, 2014 A). Mineral resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" (Canadian Institute of Mining Metallurgy and Petroleum, 2014 B)

- Measured Mineral Resource: "A 'Measured Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough to confirm both geological and grade continuity."

- Indicated Mineral Resource: "An 'Indicated Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."

- Inferred Mineral Resource: "An 'Inferred Mineral Resource' is that part of a mineral resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes."

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

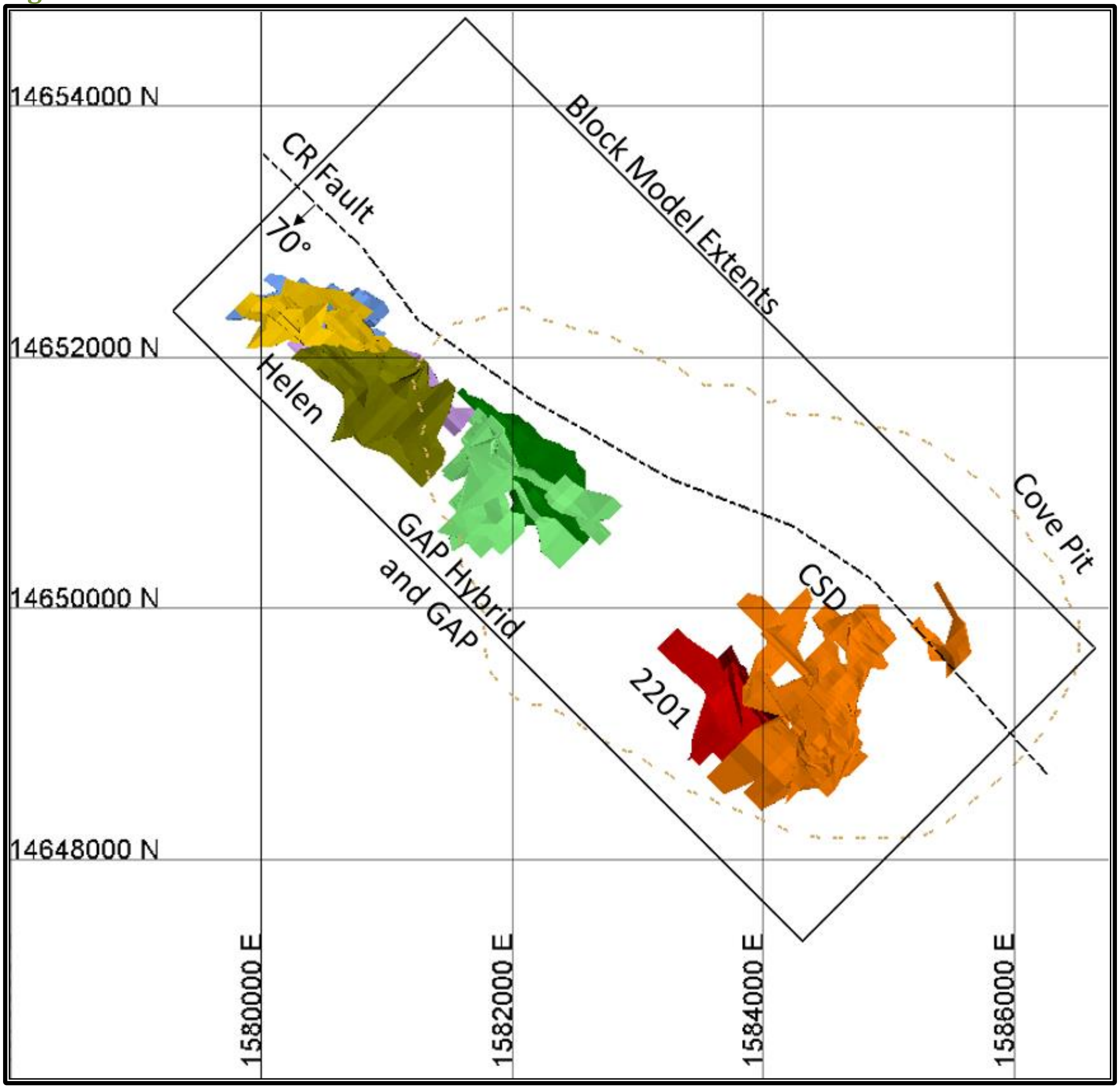
All mineral resource estimation work reported herein was carried out by Laura Symmes, Senior Geologist for Practical Mining. Section 14 is an update of the previous technical report, there were no material changes to the methodologies or assumptions within the estimation process from the previous Technical Report.

The effective date of this mineral resource estimate is March 31, 2018. The purpose of this estimate is to include the new drill holes, which were drilled since the last mineral resource estimate, dated April 15, 2017. All data coordinates are measured in the NAD83 feet and quantities are given in imperial units unless indicated otherwise.

The gold and silver mineralization at the Project was estimated using Vulcan versions 9.1.8 and 10.1.5 modeling software using the Inverse Distance Cubed (ID3) estimation method. A Nearest Neighbor method was run for comparison. The estimate was performed by Practical Mining LLC.

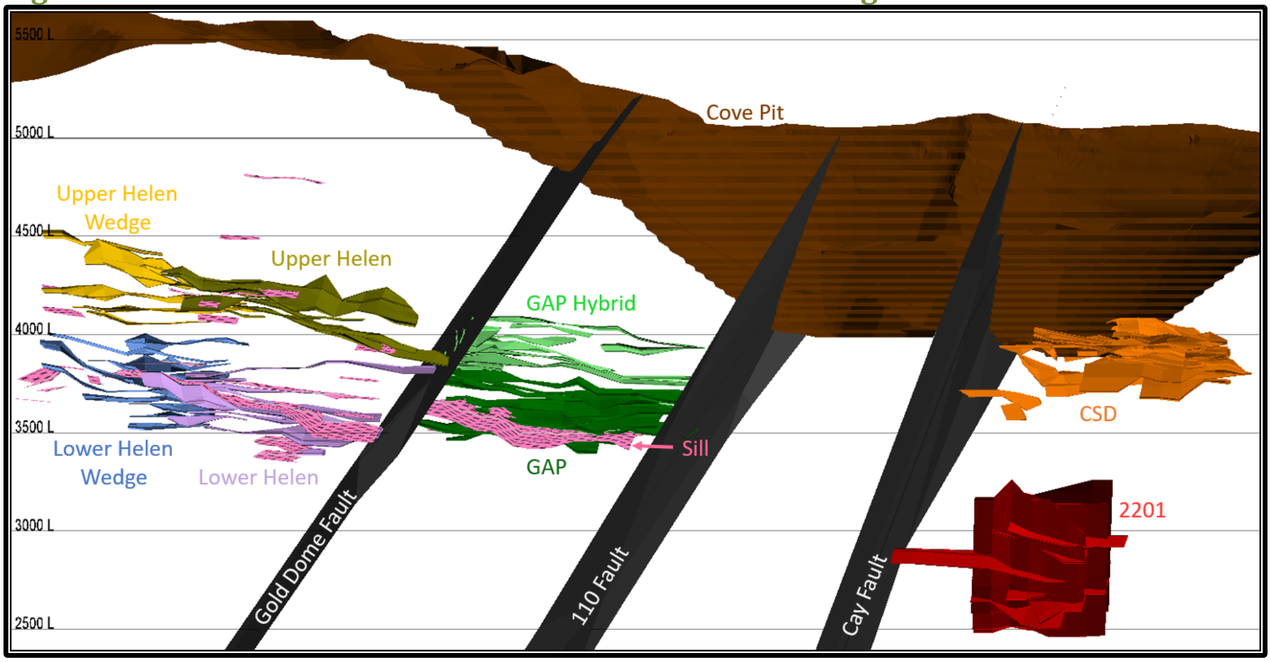
The Cove area includes four distinct mineralized zones: CSD, GAP, Helen, and 2201. The mineralized zones follow a southeast to northwest trend beginning below the historic Cove pit and extending over 6,000 feet to the northwest. Figure 14-1 shows the location of the zones.

Figure 14-1 Plan View of Cove Mineralized Zones



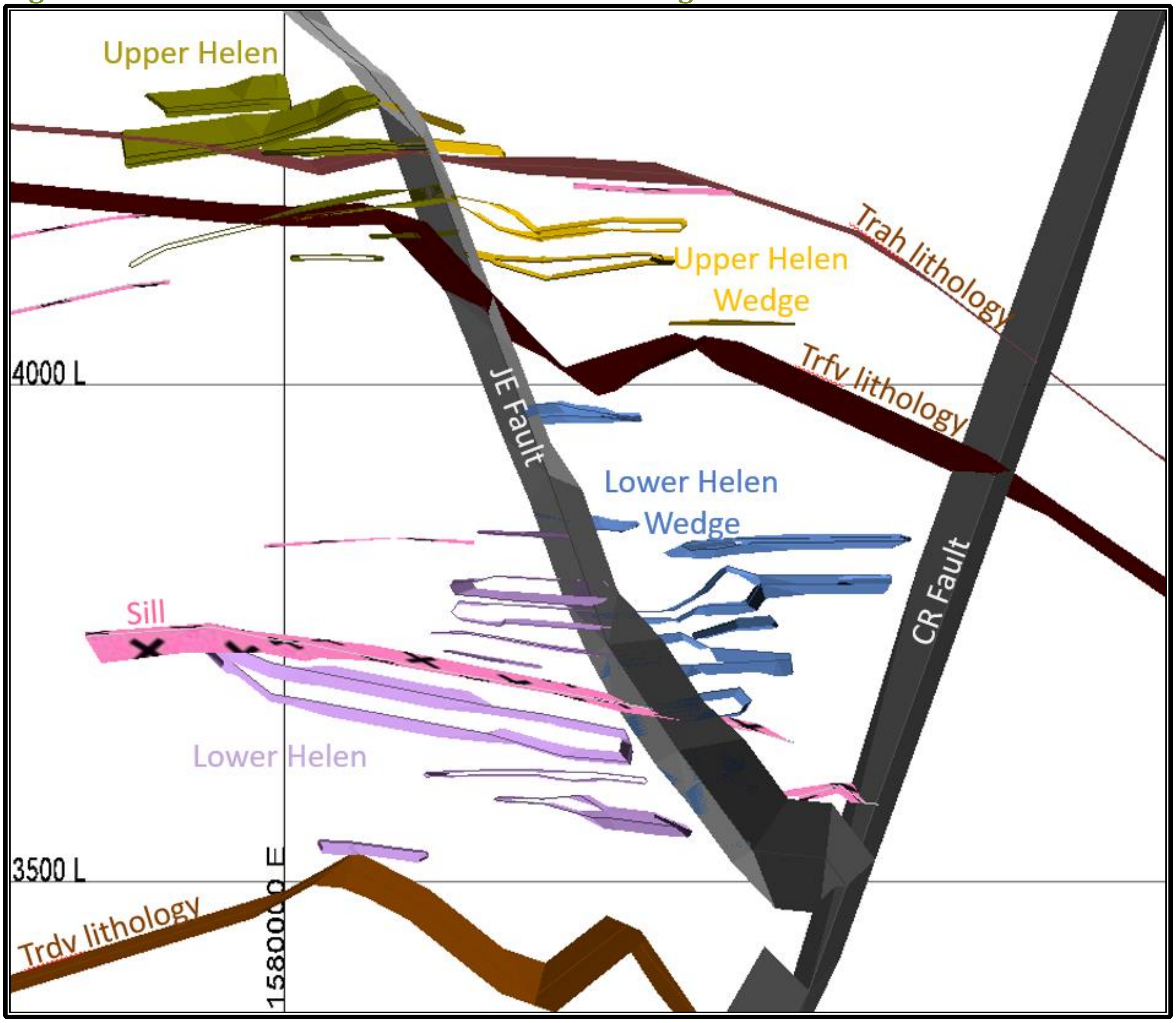
Zones are bounded by fault blocks. The Helen Zone lies north of the Gold Dome fault. The GAP GAP zone lies between the Gold Dome and 110 Faults. A prospective, unmodeled zone lies between the 110 and Cay faults. The CSD and 2201 Zones lie south of the Cay fault. All zones are bounded by the CR fault to the northeast. Figure 14-2 shows the faults bounding the mineralized zones.

Figure 14-2 Section View of Cove Mineralized Zones looking NE



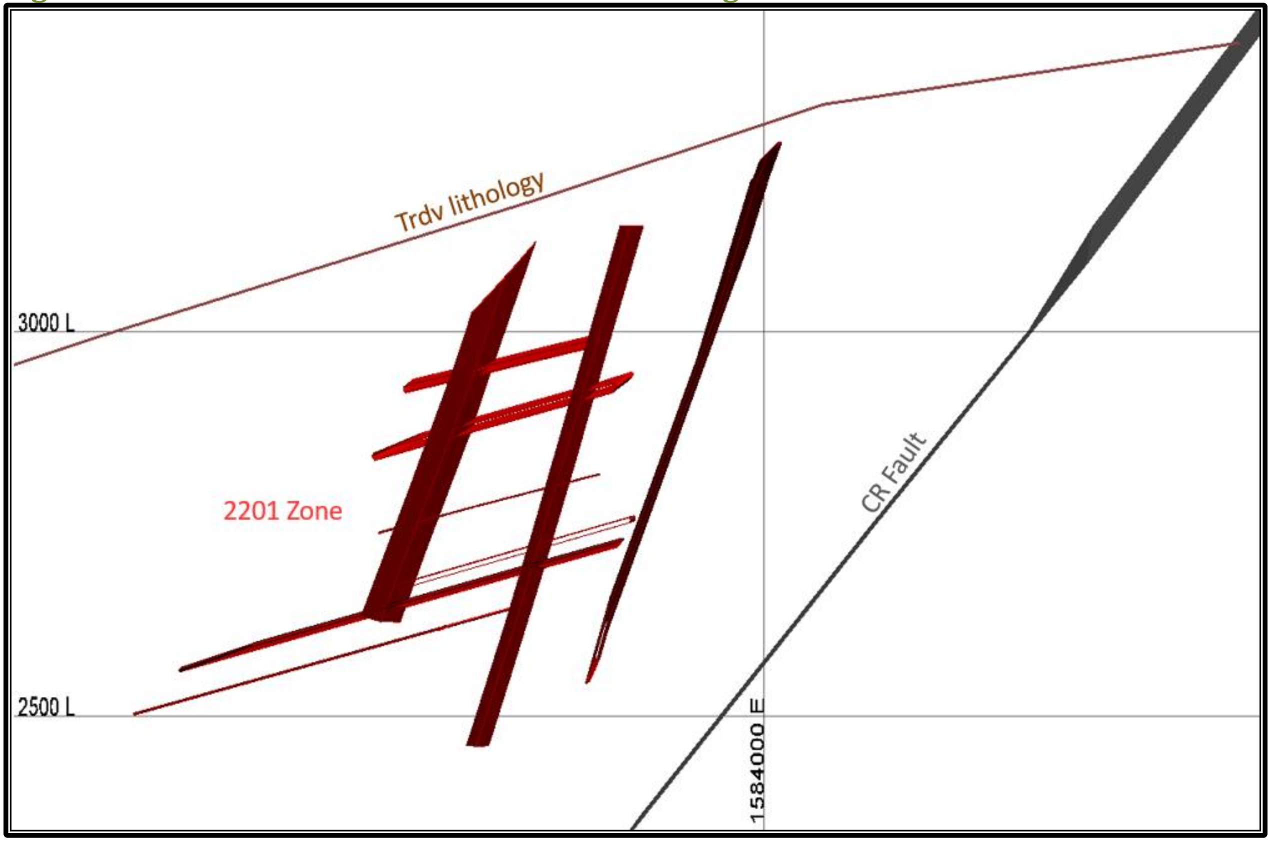
The Helen Zone is divided into four sub-zones: Upper Helen, Lower Helen, Upper Helen Wedge, and Lower Helen Wedge. The Upper Helen Zone is situated in the Home Station and Favret Formations, while the Lower Helen Zone is in the Dixie Valley Formation. The JE Fault cuts through the northern one-third of the Helen Zone striking east-west and dipping 68° N. The offset forms a wedge of mineralized material between the JE and CR faults. The Helen sub-zones are shown in Figure 14-3.

Figure 14-3 Section View of the Helen Zone looking NW



Mineralization included in the current estimate is characterized predominantly as disseminated Carlin style, except for the 2201 zone, which is polymetallic. Some polymetallic mineralization has been observed in the Gap Hybrid zone. Mineralization is controlled both lithologically and structurally. Disseminated mineralization tends to occur in lenticular geometries following both favorable bedding and T1-type sills, which are generally low angle. The sills are depicted in Figure 14-2 and Figure 14-3. Polymetallic vein mineralization is also lithologically and structurally controlled, generally with higher grades occupying narrow high angle structures with adjacent moderate grades lying along favorable low angle bedding. Figure 14-4 shows bedding parallel mineralization with high-angle mineralized structures in the 2201 zone.

Figure 14-4 Section View of the 2201 Zone looking NW



14.2. Modeling of Lithology and Mineralization

Premier geologists prepared preliminary geologic and grade shell models for the Cove area based on drill hole logging, assay data and geologic mapping. The Premier models served as the basis for resource modelling performed by Laura Symmes, Senior Geologist Practical Mining LLC.

Lithologic contacts were modeled by connecting corresponding logged drill hole intercepts in adjacent holes to form a surface representing each geologic formation. Surfaces were also created for significant lithologies associated with mineralization, including mafic sills. Faults were modeled using drill hole intercepts and by interpreting offset of lithologic surfaces. While structural interpretation is ongoing, the authors find the current lithology and structure models to be reasonable, and applied no significant edits to Premier's work. Table 14-1 lists the database codes for the modeled lithologic surfaces.

Table 14-1 Geology Codes

Formation Name	Abbreviation
Tuff of Cove Mine	Tc

Formation Name	Abbreviation
Augusta- Smelser Pass member	Tras
Augusta- Panther Canyon member	Trap
Augusta- Home Station member	Trah
Favret	Trfv
Dixie Valley	Trdv
T1 Mafic Sill	T1
T2 Mafic Sill	T2

Gold mineralization was modelled on 100-foot vertical sections facing azimuth 315. Polygons were digitized around drill hole intercepts with significant gold assay values. Intercepts in adjacent holes were connected within a polygon so that the polygons lie generally parallel with bedding and sills. Using the lithology model as a guide, polygons on adjacent sections at similar stratigraphic depths were connected to create mineral lenses. The mineral lenses were then trimmed against ore controlling faults. To model the higher grades in the 2201 zone, very high-grade intercepts were connected with high angle polygons oriented sub-parallel to the CR fault. The remaining moderate grade intercepts were digitized parallel to bedding.

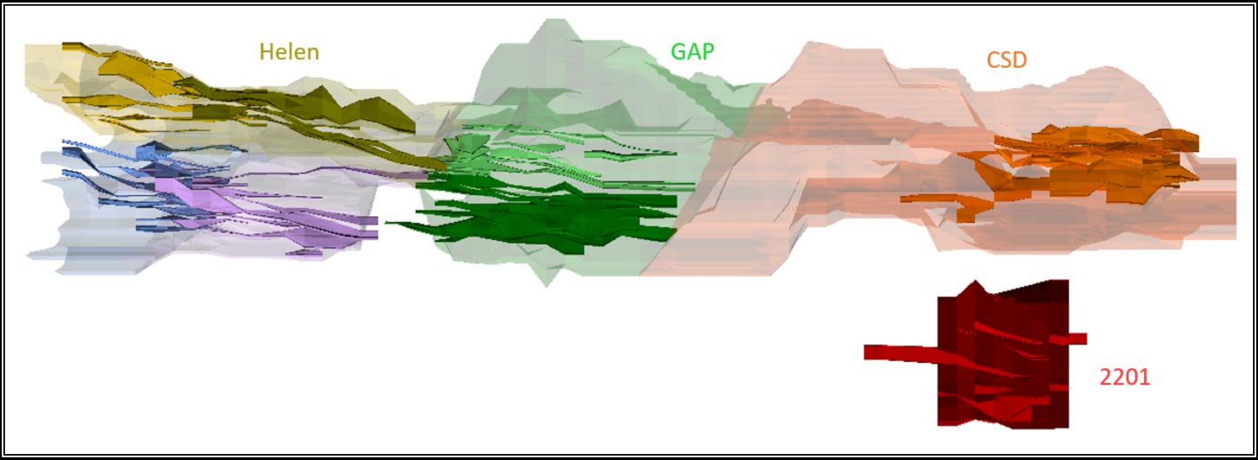
Premier's grade model conformed to a strict 3 g/t cutoff. PM modified this to allow lower grades locally in order to maintain continuity so long as the composite grade of the interval remained above 3 g/t. Each mineral lens was assigned a unique numerical code as listed in Table 14-2.

Table 14-2 Identification Codes for 3 g/t Grade Lenses

Zone	Mineral Lens Codes
CSD_GAP	2203, 2204, 2205, 2206, 2207, 2208, 2209
GAP Hybrid	5001, 5002, 5003, 5004, 5005, 5006, 5007, 5008, 5009, 5010, 5011
CSD	1101, 1104, 1105, 1106, 1107, 1108, 1109, 1112, 1113, 1114, 1115, 1116, 1117, 1118, 1119, 1120
Upper Helen	3101, 3102, 3103, 3104, 3105, 3106, 3107
Upper Helen Wedge	3301, 3302, 3303, 3304, 3305, 3306
Lower Helen	3202, 3203, 3204, 3205, 3206, 3207, 3208, 3209, 3210, 3211
Lower Helen Wedge	3400, 3401, 3402, 3403, 3404, 3405, 3406, 3407, 3408, 3409
2201 high grade	1301, 1302, 1303
2201	1400, 1401, 1403, 1404, 1405, 1406

Practical Mining also digitized a low grade mineral envelope at an approximate 0.2 g/t cutoff which surrounds all the zones except 2201. The later does not have sufficient data to construct a low-grade envelope. The low-grade envelope was divided by zone and assigned codes. In Figure 14-5, the low-grade halo is translucent with the 3 g/t lenses visible inside.

Figure 14-5 Low Grade Envelope



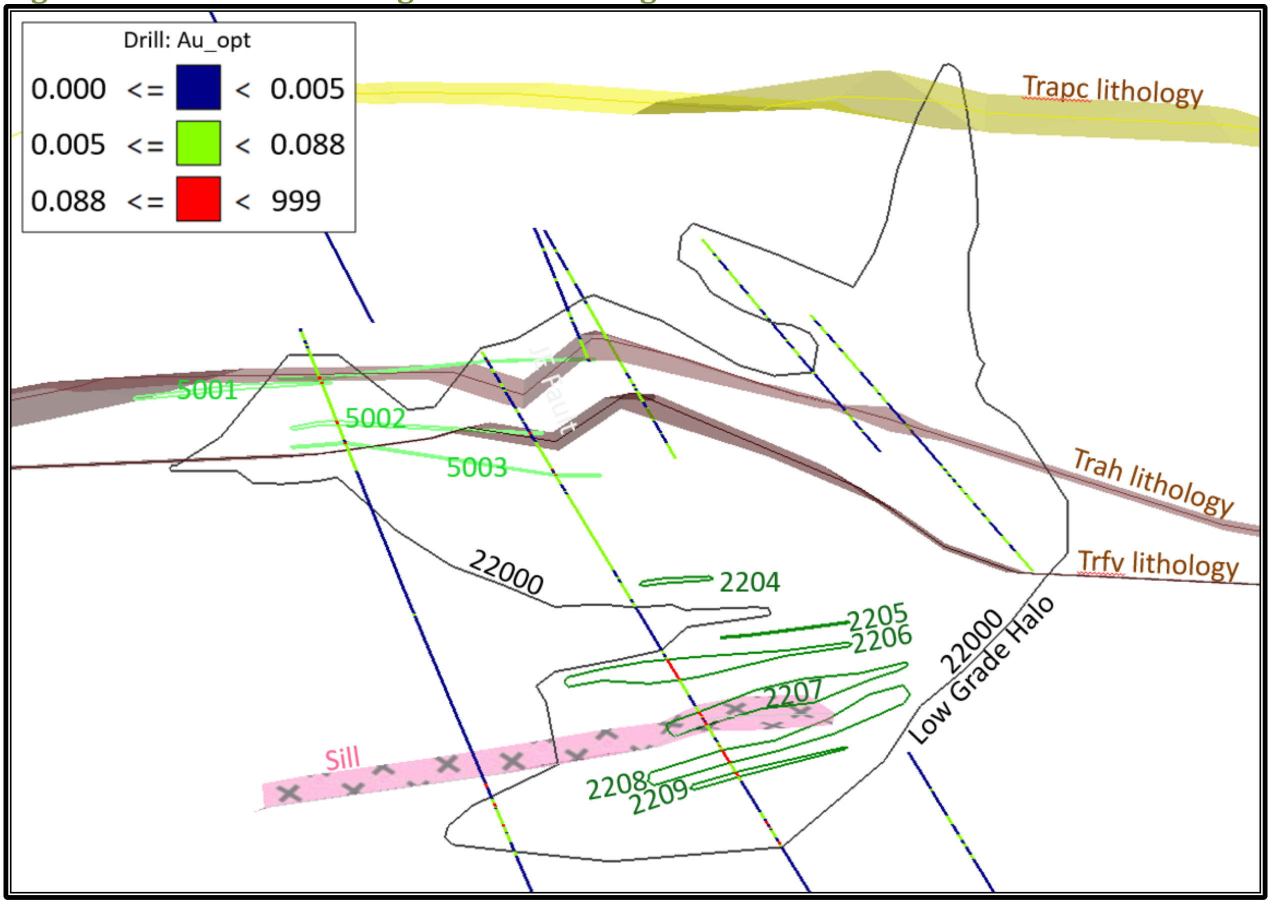
Several areas of low grade mineralization outside the low grade mineral envelope were identified and modelled and assigned codes including suffix X. Low grade codes are listed in Table 14-3.

Table 14-3 Identification Codes for 0.2 g.t Grade Lenses

Zone	Mineral Lens Codes
Low_CSD_GAP and Gap Hybrid	22000
Low_CSD	11000
Low_Upper Helen	31000
Low_Upper Helen Wedge	33000
Low_Lower Helen	32000
Low_Lower Helen Wedge	34000
Low_NE of CR Fault	10000
Low_Other	1100X, 2200X, 3300X, 3400X

Figure 14-6 shows the modeled grade lenses and low-grade halo on a section in the GAP zone.

Figure 14-6 Section Looking AZ 315 Showing Mineralized Lenses in the GAP Zone



14.3. Drill Data and Compositing

14.3.1. Drill Data Set

The drill data set used for the resource estimation contains 1,397 drill holes totaling 1,127,481 feet of drilling, of which 579,443 feet is RC and 548,038 feet is core. Premier has identified a subset of RC holes drilled prior to 2012 which may be affected by grade contamination, and those holes were excluded from the data set used for the estimation. The remaining RC holes correlate well with the surrounding core holes. One hole, NW-9A was excluded due to lack of survey data.

Premier provided the drill data to Practical Mining as csv files. Gold and silver assays were converted from g/t to opt by dividing by 34.2857 in Excel, and blank values were assigned the value -99. The CSV files were then imported into a Vulcan ISIS database. A flag field was added to the ISIS database to contain numerical code of modeled lenses. Samples within the grade model polygons were flagged with the corresponding mineral lens code using the Vulcan flagging utility. The 3 g/t polygons take precedence over the low-grade polygons for lens code flagging. Of the 1,397 holes analyzed, 1,204 intersect at least one modeled grade polygon. Of these, 370 were flagged by the 3 g/t polygons and 1,195 were flagged by the low-grade polygons. An overview of drill hole and sample statistics is shown in Table 14-4.

Table 14-4 Drill Hole Summary

Data Population		Core	RC	Total
All Holes	No. Holes	387	1,010	1,397
	Length Drilled (ft)	547,787.0	579,694.0	1,127,481.0
	No. Samples	74,913	101,637	176,550
	Length Sampled (ft)	430,357.5	572,744.0	1,003,101.5
Holes with Flags for 3g Lenses	No. Holes	195	175	370
	Length Drilled (ft)	329,645.2	49,160.0	378,805.2
	No. Flagged Samples	1,957	1,189	3,146
	Length Flagged Samples (ft)	8,941.2	5,950.0	14,891.2
Holes with Flags for Low Grade Lenses	No. Holes	370	825	1,195
	Length Drilled (ft)	513,909.9	463,031.0	976,940.9
	No. Flagged Samples	32,094	36,946	69,040
	Length Flagged Samples (ft)	161,085.2	187,141.5	348,226.7

14.3.2. Compositing

Gold and silver assay values were composited into 5-foot lengths beginning at the drill hole collar. Compositing intervals were truncated and a new compositing interval was begun where the drill hole intersects a modeled grade polygon. Only samples with like flags may be composited together. If the intercept length within the grade polygon is less than 5 feet, the composite consists of only that length contained within the polygon. The numerical lens flag was recorded with each composite in the composite database for use in the mineral resource estimation.

The total flagged composite length is 361,800 feet from 1,204 drill holes. Composite statistics by zone are shown in Table 14-5.

Table 14-5 Composite Summary

Zone	Mineral Lens Codes	Number of Holes	Number of Composites	Length of Composites (ft)
CSD_GAP	220X	27	327	1,429.4
GAP Hybrid	500X	27	133	500.0

Zone	Mineral Lens Codes	Number of Holes	Number of Composites	Length of Composites (ft)
CSD	110X	269	1,975	8,820.3
Upper Helen	310X	31	191	824.2
Upper Helen Wedge	330X	26	141	615.9
Lower Helen	320X	22	168	736.0
Lower Helen Wedge	340X	30	394	1,784.0
2201 high angle	130X	7	28	100.2
2201 low angle	140X	6	25	90.0
Low_CSD_GAP and Gap Hybrid	22000	157	10,050	48,039.6
Low_CSD	11000	596	33,785	166,141.7
Low_Upper Helen	31000	50	1,917	9,251.4
Low_Upper Helen Wedge	33000	43	2,278	11,052.7
Low_Lower Helen	32000	26	728	3,350.5
Low_Lower Helen Wedge	34000	25	802	3,732.5
Low_NE of CR fault	10000	295	8,518	41,468.7
Low_Other	1100X, 2200X, 3300X, 3400X	619	13,057	63,863.5

14.4. Density

Premier augmented their density data set in 2017 by submitting 29 samples from modeled 3 g/t zones to ALS for analysis. The new data include 23 samples from the Helen and Gap zones, five samples from the 2201 zone and one sample from the CSD zone. The density data set was filtered by analysis method, and only samples analyzed using the water displacement method with wax coating were used. The data were then sorted by zone and grade, and results were averaged by zone. Results indicate that densities are similar for samples from 3 g/t grade shells and samples from low grade shells. The densities calculated for each zone are listed in Table 14-6.

Table 14-6 Density

Zone	Density (ton/ft^3)	Number Samples
Helen	0.0691	29
GAP and GAP Hybrid	0.0708	17

Zone	Density (ton/ft^3)	Number Samples
CSD	0.0772	25
2201 Veins	0.0826	7
2201 Replacement	0.0984	13
East of CR Fault	0.0677	Default value

14.5.Statistics and Variography

Univariate statistics for gold and silver composites within the 3 g/t grade shells and low-grade shells are presented in Table 14-7 and Table 14-8 below. The composite data are not closely spaced enough to permit construction of valid variograms.

Table 14-7 Gold Composite Statistics

Zone	No. Samples	Median	Max	Min	Mean	Std. Dev.	Variance	Upper 95% CI	Lower 95% CI
CSD	1444	0.089	20.602	0.000	0.188	0.739	0.035	0.226	0.149
GAP	342	0.180	1.616	0.007	0.288	0.276	0.079	0.318	0.259
Gap Hybrid	154	0.111	1.182	0.002	0.158	0.173	0.064	0.186	0.131
Upper Helen	225	0.133	1.570	0.002	0.205	0.228	0.053	0.235	0.175
Upper Helen Wedge	190	0.112	0.752	0.001	0.131	0.104	0.012	0.145	0.116
Lower Helen	180	0.196	3.943	0.002	0.338	0.445	0.142	0.403	0.273
Lower Helen Wedge	375	0.201	2.330	0.000	0.315	0.325	0.108	0.348	0.282
2201 Vein	103	0.119	5.320	0.006	0.315	0.597	1.041	0.430	0.200
2201 Replacement	25	0.163	1.224	0.08	0.306	0.312	0.097	0.428	0.184
Low_CSD	17010	0.009	20.602	0	0.026	0.21	0.001	0.029	0.023
Low_GAP and GAP Hybrid	6260	0.011	0.593	0	0.019	0.025	0.001	0.019	0.018
Low_Upper Helen	1822	0.012	0.835	0	0.021	0.032	0.001	0.022	0.019
Low_Upper Helen Wedge	1661	0.015	0.378	0	0.022	0.024	0.001	0.024	0.021
Low_Lower Helen	828	0.007	0.285	0	0.016	0.025	0.001	0.018	0.014
Low_Lower Helen Wedge	1280	0.011	1.347	0	0.026	0.06	0.001	0.029	0.022

Table 14-8 Silver Composite Statistics

Zone	No. Samples	Median	Max	Min	Mean	Std. Dev.	Variance	Upper 95% CI	Lower 95% CI
CSD	521	0.852	48.673	0.000	2.396	4.391	19.281	2.773	2.019
GAP	304	0.131	5.163	0.004	0.261	0.483	0.233	0.315	0.207
Gap Hybrid	119	0.184	42.351	0.007	1.972	5.813	33.788	3.016	0.927
Upper Helen	189	0.050	4.887	0.007	0.173	0.503	0.253	0.245	0.102
Upper Helen Wedge	126	0.029	3.702	0.000	0.110	0.369	0.136	0.175	0.046
Lower Helen	523	0.058	1.397	0.000	0.091	0.115	0.013	0.101	0.081
Lower Helen Wedge	353	0.044	0.549	0.001	0.075	0.088	0.008	0.084	0.066
2201 Vein	26	1.027	2.920	0.099	1.122	0.848	0.719	1.448	0.796
2201 Replacement	25	0.653	3.987	0.102	1.043	1.142	1.304	1.49	0.595
Low_CSD	17012	0.19	71.667	0	0.64	1.891	0.002	0.669	0.612
Low_GAP and GAP Hybrid	6262	0.069	20.816	0	0.397	0.983	0.002	0.421	0.372
Low_Upper Helen	1749	0.01	4.947	0	0.046	0.177	0.002	0.054	0.038
Low_Upper Helen Wedge	1433	0.008	4.44	0	0.046	0.242	0.002	0.058	0.033
Low_Lower Helen	803	0.007	0.802	0	0.02	0.04	0.002	0.023	0.017
Low_Lower Helen Wedge	1233	0.005	1.031	0.001	0.016	0.057	0.002	0.019	0.012

14.6.Grade Capping

Cap grades were applied to composites with values above a statistically determined threshold. Cap grade values were determined individually for each zone, set according to the upper 95% Cl listed in Table 14-8. For the estimation, composite values exceeding the cap grade were set to the cap grade. Of the composites within the 3 g/t grade shells 6.3% were capped. Grade capping details are listed in Table 14-9.

Table 14-9 Composite Grade Capping

Zone	Grade Cap		Composites Above Cap		Number of Comps.	Capped %		Average Grade Before Capping	
	Au opt	Ag opt	Au	Ag		Au	Ag	Au opt	Ag opt
Lower Helen	1.07	0.29	14	15	168	8.3	8.9	1.61	0.49
Lower Helen Wedge	0.96	0.26	18	23	394	4.6	5.8	1.41	0.46
Upper Helen	0.65	0.49	11	10	191	5.8	5.2	0.98	1.48
Upper Helen Wedge	0.36	0.38	5	7	141	3.5	5.0	0.52	1.20
GAP Hybrid	0.48	14.93	6	7	133	4.5	5.3	1.05	23.23
GAP	0.84	0.82	16	16	327	4.9	4.9	1.20	1.86
CSD	0.43	8.84	138	199	1,975	7.0	10.1	1.61	7.30
2201 Vein	1.56	2.70	2	2	28	7.1	7.1	3.46	2.86
2201 Replacement	0.87	3.61	2	2	25	8.0	8.0	1.05	3.82
Low_GAP	0.09	1.86	305	1,251	10,050	3.0	12.4	0.25	5.98

Zone	Grade Cap		Composites Above Cap		Number of Comps.	Capped %		Average Grade Before Capping	
	Au opt	Ag opt	Au	Ag		Au	Ag	Au opt	Ag opt
Low_Upper Helen	0.09	0.16	24	94	1,917	1.3	4.9	0.16	1.48
Low_Lower Helen	0.09	0.06	8	26	728	1.1	3.6	0.15	0.13
Low_Upper Helen Wedge	0.09	0.10	27	96	2,278	1.2	4.2	0.13	0.67
Low_Lower Helen Wedge	0.09	0.05	29	25	802	3.6	3.1	0.22	0.17
Low_CSD	0.09	2.56	1,333	3,745	33,785	3.9	11.1	0.37	7.30

14.7.Block Model

The block model origin was set at coordinate 1584315.0, 14647350.0, 3300.0 with bearing 45° to match the northwest trend of the deposit. The plunge and dip were both set to zero. The model extends 7,100 feet to the northwest, 2,400 feet to the northeast, and is 2,400 feet thick. The parent block size is 100 ft x 100 ft x 100 ft with a sub block size of 5 ft x 5 ft x 2.5 ft. The 2201 zone was assigned a sub block size of 1 ft x 1 ft x 1ft.

Variables were assigned to the model to contain gold and silver estimation values and other assigned values. The block model variables are listed in Table 14-10.

Table 14-10 Block Model Variables

Variables	Default	Type	Description
density	0	float	density
au_opt	-99	float	Gold - Grade Estimate (Ounces per Ton)
au_flag	0	byte	Gold - Estimation Flag
au_ndh	0	byte	Gold - Number Drill Holes
au_dist	0	float	Gold - Average Distance to Samples
au_ns	0	byte	Gold - Number of Samples
au_opt_nn	-99	float	Gold - Nearest Neighbor (Ounces per Ton)
au_nn_dist	0	float	Distance to nearest sample
ag_opt	-99	float	Silver - Grade Estimate (Ounces per Ton)
ag_flag	0	byte	Silver - Estimation Flag
ag_ndh	0	byte	Silver - Number Drill Holes
ag_dist	0	float	Silver - Average Distance to Samples
ag_ns	0	byte	Silver - Number of Samples

Variables	Default	Type	Description
ag_opt_nn	-99	float	Silver - Nearest Neighbor (Ounces per Ton)
ag_nn_dist	0	float	Distance to nearest sample
aueq	-99	double	Gold Equivalence (Ounces per Ton)
agau	-99	double	Silver:Gold Ratio
mined	insitu	name	Block Status (insitu, sterile, mined)
classname	none	name	Classification (meas, ind, inf)
mii	0	byte	1 eq meas, 2 eq ind, 3 eq inf
aueng	0	float	Au Engineering
ageng	0	float	Ag Engineering
aueqeng	0	float	AuEq Engineering
zone	none	name	Zone
volume	-	predefined	
xlength	-	predefined	
ylength	-	predefined	
zlength	-	predefined	
xcentre	-	predefined	
ycentre	-	predefined	
zcentre	-	predefined	
xworld	-	predefined	
yworld	-	predefined	
zworld	-	predefined	

14.8.Grade Estimation and Resource Classification

The gold and silver variables in the block model were estimated using inverse distance cubed (ID^{3}) and nearest neighbor methods. The estimations were completed with one pass.

Anisotropic search parameters were set to the average orientation of each zone. Average orientation was determined by loading the modeled 3 g/t lenses by zone in Vulcan and visually estimating average dip and dip direction. Distances were selected based on the drill spacing of samples intercepting the lenses and on the general orientation and shape of the interpreted solids. Blocks inside of the modelled 3 g/t lenses were estimated using only composites flagged with the corresponding lens code. Blocks outside of the 3 g/t lenses were estimated using composites with the corresponding low-grade flag. Blocks lying outside the low-grade halo were not estimated. The estimation search parameters are listed in Table 14-11.

Table 14-11 Estimation Parameters

Zone	Mineral Lens Code	Bearing	Plunge	Dip	Major Search Axis (ft)	Semi-major Search Axis (ft)	Minor Search Axis (ft)	Min Samp	Max Samp
CSD_GAP	220X	315	4.8	10.3	300	300	100	1	3
GAP Hybrid	500X	315	5.0	0.0	300	300	100	1	3
CSD	110X	315	6.3	7.1	300	300	100	1	3
Upper Helen	310X	315	10.7	7.0	300	300	100	1	3
Upper Helen Wedge	330X	315	6.2	-7.4	300	300	100	1	3
Lower Helen	320X	315	0.0	0.0	300	300	100	1	3
Lower Helen Wedge	340X	315	-1.5	0.0	300	300	100	1	3
2201 high angle vein 1	1301	342	0.0	73.6	300	300	300	1	3
2201 high angle vein 2	1302	342	0.0	73.6	300	300	300	1	3
2201 high angle vein 3	1303	322	0.0	73.6	300	300	300	1	3
2201 low angle	140X	315	7.6	15.7	300	300	100	1	3
Low_CSD_GAP and Gap Hybrid	22000	315	4.8	10.3	300	300	100	1	3
Low_CSD	11000	315	6.3	7.1	300	300	100	1	3
Low_Upper Helen	31000	315	10.7	7.0	300	300	100	1	3
Low_Upper Helen Wedge	33000	315	6.2	-7.4	300	300	100	1	3
Low_Lower Helen	32000	315	0.0	0.0	300	300	100	1	3
Low_Lower Helen Wedge	34000	315	-1.5	0.0	300	300	100	1	3
Low_NE of CR fault	10000	315	0.0	0.0	300	300	100	1	3
Low_Other_1100X	1100X	315	6.3	7.1	300	300	100	1	3
Low_Other_2200X	2200X	315	4.8	10.3	300	300	100	1	3
Low_Other_3300X	3300X	315	6.2	-7.4	300	300	100	1	3
Low_Other_3400X	3400X	315	-1.5	0	300	300	100	1	3

A script was run on the estimated block model to populate the classification variable. The classification categories are indicated, inferred and none. Classification was determined based on three block model variables: au_dist, au_ndh and au_nn_dist. These three variables represent, respectively, the average distance to the composites used to estimate the grade of the block, the number of drill holes contributing to the grade of the block, and the distance to the nearest composite. The default value was defined as 'none', which was over-written by indicated or inferred where the required conditions were satisfied. The conditions of the classification script are listed in Table 14-12.

Table 14-12 Classification Conditions

Class	Script Condition	au_dist (ft)	au_ndh	au_nn_dist (ft)
Indicated	if	<100	at least 2	50 or less
Inferred	elseif	<=300	at least 2	

Class	Script Condition	au_dist (ft)	au_ndh	au_nn_dist (ft)
None	default			

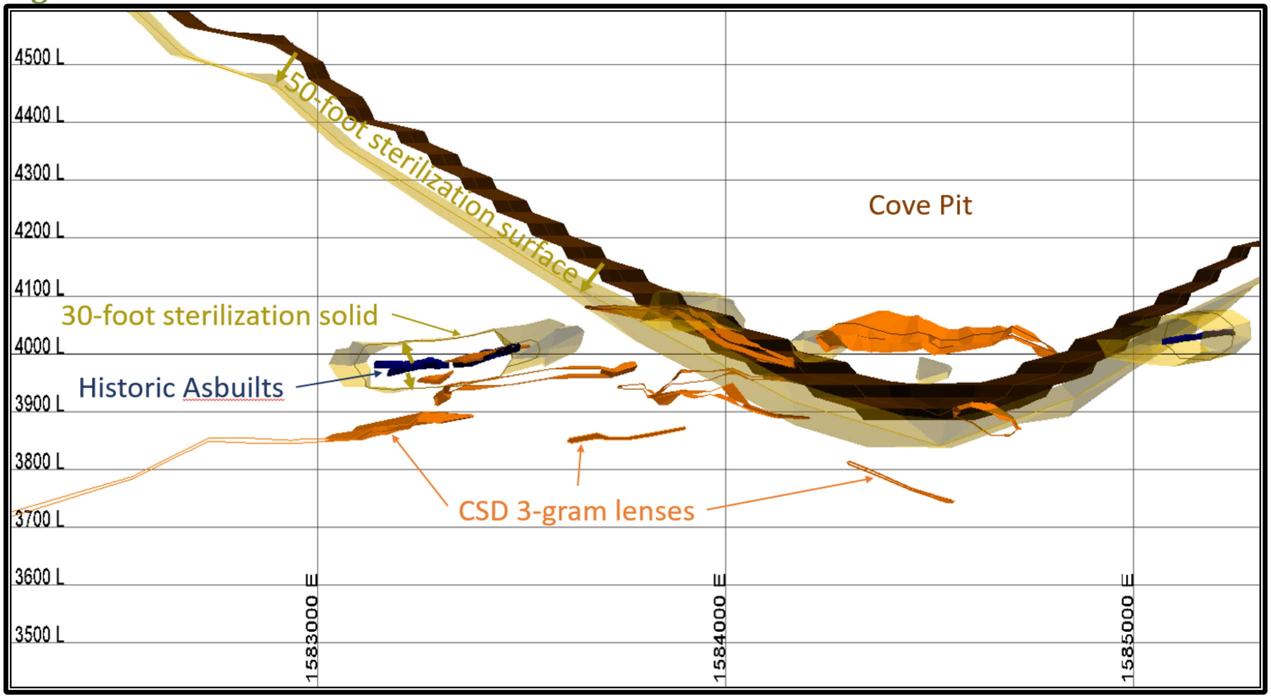
Significant parameters used in the gold and silver estimations included:

1. Only composites with a value of greater than or equal to zero were used;
2. A minimum of one and maximum of three composites were used;
3. Only one composite per drill hole was allowed;
4. Composites were selected using anisotropic distances oriented to the local dip and dip direction of the zone;
5. Only composites within a lens were used to estimate blocks within the lens;
6. Grades were capped using a top cut method, and;
7. Gold and silver for blocks outside modelled 3 g/t and low-grade shells were not estimated.

14.9. Mined Depletion and Sterilization

The CSD zone lies adjacent to the historic Cove pit and was historically exploited using underground cut-and-fill and stoping methods. Part of the modelled CSD zone has been mined, and areas of insitu material near historically mined areas are rendered inaccessible. The block model includes a ‘mined’ variable which stores information identifying each block as insitu, mined or sterile. The default value is insitu. Blocks lying above the ultimate pit topography or inside the underground mine asbuilt are defined as mined. Sterile blocks were defined using two shapes modelled in Vulcan. The first is a surface digitized 50-feet below deepest mined topography, and the second is a solid shape digitized around the underground mine asbuilt representing a 30-foot buffer zone. Blocks lying between the 50-foot surface and the ultimate pit topographic surface are sterile, and blocks within 30 feet of the historic underground mine are sterile. Figure 14-7 shows the sterilization surfaces in a section view of the CSD zone.

Figure 14-7 Sterilization Surfaces



14.10. Model Validation

The mean gold grades for each lens were compared against a nearest neighbor (representing declustered composites) in Table 14-13. Individual lens comparisons vary depending on sample support and grade variability. Overall the ID3 estimate is slightly lower than the nearest neighbor. Table 14-14 represents the same data for silver which shows the same general relationships.

Table 14-13 Estimate Comparison for Gold versus a Nearest Neighbor at 0 Cutoff

	ID3 Estimate						Nearest Neighbor						Mean
Vein	Mean	Std. Dev.	Max	Q3	Q1	Min	Mean	Std. Dev.	Max	Q3	Q1	Min	Diff
1101	0.186	0.056	0.284	0.245	0.136	0.089	0.193	0.077	0.27	0.27	0.102	0.051	-3.6%
1104	0.143	0.076	0.416	0.188	0.089	0.01	0.148	0.102	0.416	0.216	0.081	0.007	-3.4%
1105	0.172	0.079	0.43	0.218	0.12	0.002	0.172	0.105	0.43	0.245	0.105	0.002	0.0%
1106	0.147	0.085	0.43	0.188	0.092	0.003	0.15	0.111	0.43	0.194	0.09	0.002	-2.0%
1107	0.184	0.09	0.43	0.235	0.119	0.01	0.186	0.117	0.43	0.274	0.096	0.01	-1.1%
1108	0.147	0.05	0.43	0.151	0.116	0.028	0.15	0.061	0.43	0.16	0.114	0.022	-2.0%
1109	0.177	0.057	0.428	0.222	0.129	0.027	0.181	0.071	0.43	0.237	0.126	0.026	-2.2%
1112	0.109	0.031	0.429	0.11	0.09	0.089	0.108	0.041	0.43	0.1	0.089	0.089	0.9%
1113	0.136	0.092	0.39	0.186	0.06	0.011	0.149	0.136	0.391	0.34	0.031	0	-8.7%
1115	0.115	0.101	0.43	0.166	0.037	0	0.116	0.126	0.43	0.164	0.018	0	-0.9%
1116	0.134	0.068	0.43	0.164	0.092	0.006	0.129	0.088	0.43	0.161	0.073	0.006	3.9%

Vein	ID3 Estimate Mean	Std. Dev.	Max	Q3	Q1	Min	Nearest Neighbor Mean	Std. Dev.	Max	Q3	Q1	Min	Mean Diff
1117	0.174	0.095	0.43	0.266	0.1	0.009	0.148	0.116	0.43	0.17	0.074	0.009	17.6%
1118	0.22	0.099	0.424	0.281	0.14	0.035	0.225	0.138	0.424	0.35	0.113	0.034	-2.2%
1120	0.106	0.094	0.43	0.14	0.034	0.002	0.108	0.106	0.43	0.118	0.037	0	-1.9%
2203	0.146	0.076	0.347	0.151	0.099	0.097	0.145	0.09	0.347	0.138	0.097	0.097	0.7%
2204	0.263	0.166	0.773	0.305	0.172	0.068	0.259	0.204	0.773	0.211	0.146	0.068	1.5%
2204	0.304	0.157	0.84	0.361	0.2	0.022	0.29	0.205	0.84	0.376	0.139	0.012	4.8%
2205	0.244	0.12	0.609	0.338	0.147	0.101	0.24	0.139	0.609	0.376	0.13	0.101	1.7%
2206	0.263	0.192	0.84	0.267	0.156	0.039	0.272	0.221	0.84	0.273	0.145	0.039	-3.3%
2207	0.342	0.215	0.84	0.479	0.16	0.031	0.339	0.266	0.84	0.538	0.127	0.031	0.9%
2208	0.282	0.173	0.84	0.375	0.147	0.006	0.286	0.216	0.84	0.423	0.115	0.006	-1.4%
2209	0.199	0.073	0.363	0.244	0.142	0.119	0.199	0.089	0.363	0.25	0.137	0.119	0.0%
3101	0.257	0.089	0.648	0.278	0.227	0.094	0.253	0.113	0.65	0.281	0.197	0.094	1.6%
3102	0.177	0.073	0.318	0.261	0.125	0.09	0.181	0.081	0.318	0.258	0.125	0.09	-2.2%
3103	0.192	0.135	0.65	0.216	0.109	0.002	0.192	0.169	0.65	0.201	0.097	0.002	0.0%
3104	0.167	0.077	0.449	0.178	0.13	0.003	0.168	0.098	0.449	0.195	0.104	0.003	-0.6%
3105	0.407	0.172	0.65	0.561	0.284	0.106	0.406	0.236	0.65	0.65	0.222	0.106	0.2%
3106	0.159	0.061	0.374	0.192	0.115	0.046	0.158	0.072	0.374	0.188	0.112	0.045	0.6%
3107	0.128	0.069	0.319	0.136	0.09	0.023	0.132	0.103	0.319	0.124	0.088	0.023	-3.0%
3202	0.254	0.124	0.482	0.365	0.133	0.083	0.266	0.158	0.484	0.399	0.082	0.082	-4.5%
3202	0.256	0.133	0.734	0.329	0.159	0.082	0.243	0.155	0.735	0.248	0.114	0.082	5.3%
3203	0.405	0.151	1.069	0.464	0.329	0.11	0.416	0.223	1.07	0.458	0.299	0.11	-2.6%
3204	0.256	0.033	0.336	0.273	0.229	0.197	0.249	0.05	0.336	0.272	0.222	0.197	2.8%
3205	0.6	0.344	1.07	0.954	0.257	0.118	0.625	0.445	1.07	1.07	0.138	0.118	-4.0%
3206	0.418	0.28	1.07	0.608	0.168	0.004	0.413	0.358	1.07	0.649	0.116	0.004	1.2%
3207	0.271	0.177	0.612	0.405	0.128	0.114	0.285	0.204	0.612	0.612	0.116	0.114	-4.9%
3208	0.159	0.038	0.256	0.17	0.139	0.1	0.163	0.054	0.256	0.177	0.101	0.1	-2.5%
3210	0.157	0.043	0.209	0.203	0.109	0.1	0.156	0.055	0.209	0.209	0.1	0.1	0.6%
3211	0.16	0.065	0.5	0.178	0.123	0.073	0.157	0.077	0.5	0.197	0.112	0.073	1.9%
3301	0.14	0.057	0.36	0.17	0.104	0.03	0.134	0.079	0.36	0.155	0.093	0.03	4.5%
3302	0.102	0.069	0.298	0.167	0.018	0.006	0.111	0.091	0.305	0.138	0.018	0.006	-8.1%
3303	0.125	0.054	0.34	0.126	0.099	0.031	0.121	0.064	0.34	0.133	0.087	0.025	3.3%
3304	0.169	0.061	0.36	0.216	0.127	0.022	0.169	0.089	0.36	0.236	0.114	0.001	0.0%
3305	0.142	0.066	0.36	0.179	0.096	0.024	0.143	0.081	0.36	0.202	0.091	0.024	-0.7%
3306	0.129	0.022	0.174	0.146	0.118	0.044	0.13	0.028	0.177	0.158	0.114	0.043	-0.8%
3400	0.224	0.171	0.727	0.283	0.112	0.017	0.239	0.22	0.729	0.322	0.109	0.017	-6.3%
3401	0.214	0.108	0.658	0.273	0.143	0.018	0.21	0.13	0.659	0.272	0.118	0.018	1.9%
3402	0.313	0.236	0.96	0.512	0.134	0	0.322	0.272	0.96	0.574	0.106	0	-2.8%
3403	0.277	0.173	0.96	0.329	0.162	0.001	0.28	0.217	0.96	0.412	0.149	0.001	-1.1%
3404	0.365	0.237	0.96	0.461	0.204	0.098	0.364	0.291	0.96	0.416	0.143	0.092	0.3%
3405	0.182	0.082	0.367	0.25	0.115	0.114	0.184	0.084	0.367	0.255	0.115	0.114	-1.1%

Vein	ID3 Estimate Mean	Std. Dev.	Max	Q3	Q1	Min	Nearest Neighbor Mean	Std. Dev.	Max	Q3	Q1	Min	Mean Diff
3406	0.36	0.205	0.96	0.482	0.222	0.116	0.363	0.246	0.96	0.495	0.22	0.116	-0.8%
3407	0.267	0.17	0.96	0.378	0.136	0.056	0.273	0.226	0.96	0.42	0.104	0.056	-2.2%
3408	0.146	0.048	0.224	0.177	0.113	0.008	0.141	0.052	0.224	0.167	0.099	0.008	3.5%
3409	0.112	0.021	0.158	0.125	0.107	0.057	0.112	0.028	0.166	0.129	0.114	0.057	0.0%
5001	0.3	0.068	0.407	0.322	0.239	0.202	0.302	0.078	0.407	0.322	0.202	0.202	-0.7%
5002	0.165	0.092	0.48	0.193	0.109	0.055	0.167	0.115	0.48	0.209	0.108	0.055	-1.2%
5003	0.179	0.04	0.413	0.191	0.153	0.114	0.177	0.055	0.415	0.179	0.153	0.104	1.1%
5004	0.14	0.067	0.4	0.179	0.097	0.041	0.145	0.089	0.4	0.181	0.097	0.041	-3.4%
5005	0.202	0.144	0.48	0.202	0.107	0.093	0.195	0.154	0.48	0.217	0.098	0.092	3.6%
5006	0.186	0.096	0.48	0.201	0.119	0.091	0.18	0.128	0.48	0.159	0.094	0.091	3.3%
5007	0.165	0.095	0.465	0.207	0.104	0.022	0.166	0.118	0.466	0.206	0.099	0.02	-0.6%
5008	0.18	0.085	0.479	0.21	0.113	0.078	0.184	0.112	0.48	0.201	0.111	0.077	-2.2%
5009	0.108	0.01	0.133	0.114	0.101	0.095	0.107	0.013	0.133	0.103	0.1	0.095	0.9%
5010	0.17	0.079	0.447	0.196	0.123	0.068	0.167	0.107	0.448	0.153	0.11	0.068	1.8%
5011	0.299	0.035	0.348	0.321	0.286	0.148	0.293	0.073	0.382	0.382	0.245	0.075	2.0%

Table 14-14 Estimate Comparison for Silver versus a Nearest Neighbor at 0 Cutoff

Vein	ID3 Estimate Mean	Std. Dev.	Max	Q3	Q1	Min	Nearest Neighbor Mean	Std. Dev.	Max	Q3	Q1	Min	Mean Diff
1101	1.759	1.219	5.8	2.449	0.854	0.07	1.422	1.628	8.165	2.416	0.07	0.07	23.7%
1104	3.145	2.067	8.84	4.673	1.413	0	2.988	2.595	8.84	4.832	0.832	0	5.3%
1105	1.21	1.643	8.84	1.676	0.05	0	1.213	2.028	8.84	1.521	0.04	0	-0.2%
1106	2.329	2.498	8.84	4.026	0.216	0	2.42	2.891	8.84	3.653	0.245	0	-3.8%
1107	0.33	0.289	1.437	0.537	0.056	0.004	0.335	0.34	1.437	0.5	0.047	0.004	-1.5%
1108	2.817	1.191	6.683	3.566	1.991	0.261	2.968	1.558	6.683	3.652	1.657	0.214	-5.1%
1109	2.871	1.698	8.653	4.016	1.282	0.065	2.831	1.922	8.653	3.907	1.15	0.06	1.4%
1112	1.627	2.105	8.659	1.799	0.445	0.12	1.74	3.026	8.662	0.455	0.455	0.12	-6.5%
1113	2.098	1.383	5.621	2.726	0.968	0.203	1.807	1.874	5.63	1.801	0.65	0.2	16.1%
1115	1.882	2.733	8.84	1.782	0.252	0	1.852	2.921	8.84	1.651	0.166	0	1.6%
1116	0.905	1.61	8.834	0.742	0.094	0	0.887	1.806	8.84	0.39	0.071	0	2.0%
1117	0.949	0.975	8.84	0.824	0.484	0.152	0.807	1.075	8.84	0.961	0.311	0.151	17.6%
1118	1.731	1.98	8.453	1.635	0.638	0.098	1.744	2.754	8.453	1.1	0.3	0	-0.7%
1120	1.732	1.696	8.681	2.328	0.625	0.01	1.705	2.401	8.84	2.395	0.14	0.01	1.6%
2203	0.051	0.022	0.087	0.07	0.041	0.007	0.053	0.025	0.087	0.087	0.044	0.007	-3.8%
2204	0.264	0.194	0.805	0.336	0.114	0.012	0.262	0.215	0.805	0.332	0.107	0.012	0.8%
2204	0.108	0.082	0.369	0.12	0.056	0.009	0.104	0.108	0.369	0.106	0.04	0.007	3.8%
2205	0.236	0.217	0.82	0.347	0.067	0.016	0.245	0.254	0.82	0.305	0.04	0.016	-3.7%

	ID3 Estimate						Nearest Neighbor						Mean
Vein	Mean	Std. Dev.	Max	Q3	Q1	Min	Mean	Std. Dev.	Max	Q3	Q1	Min	Diff
2206	0.126	0.089	0.506	0.152	0.069	0.005	0.128	0.101	0.506	0.168	0.059	0.005	-1.6%
2207	0.236	0.186	0.82	0.294	0.107	0.016	0.237	0.222	0.82	0.317	0.076	0.016	-0.4%
2208	0.214	0.171	0.82	0.263	0.094	0.007	0.217	0.211	0.82	0.298	0.062	0.007	-1.4%
2209	0.49	0.2	0.82	0.688	0.365	0.075	0.503	0.269	0.82	0.82	0.356	0.075	-2.6%
3101	0.062	0.031	0.197	0.094	0.044	0.02	0.061	0.038	0.207	0.097	0.04	0.02	1.6%
3102	0.08	0.035	0.176	0.113	0.046	0.02	0.078	0.044	0.176	0.115	0.04	0.02	2.6%
3103	0.111	0.107	0.49	0.159	0.029	0.007	0.112	0.139	0.49	0.159	0.02	0.007	-0.9%
3104	0.08	0.089	0.49	0.086	0.03	0.007	0.081	0.108	0.49	0.096	0.026	0.007	-1.2%
3105	0.22	0.115	0.417	0.316	0.111	0.047	0.222	0.158	0.417	0.361	0.055	0.047	-0.9%
3106	0.157	0.111	0.49	0.211	0.072	0.007	0.15	0.14	0.49	0.222	0.035	0.007	4.7%
3107	0.216	0.195	0.49	0.439	0.016	0.007	0.205	0.234	0.49	0.49	0.009	0.007	5.4%
3202	0.017	0.02	0.143	0.021	0.007	0.001	0.006	0.034	0.265	0.001	0	0	183.3%
3202	0.028	0.023	0.106	0.039	0.012	0	0.028	0.028	0.106	0.058	0.008	0	0.0%
3203	0.094	0.07	0.29	0.138	0.033	0.012	0.091	0.093	0.29	0.1	0.012	0.012	3.3%
3204	0.088	0.027	0.142	0.112	0.063	0.047	0.084	0.043	0.16	0.125	0.047	0.047	4.8%
3205	0.126	0.071	0.29	0.153	0.071	0.044	0.128	0.091	0.29	0.123	0.05	0.044	-1.6%
3206	0.144	0.068	0.29	0.192	0.09	0.007	0.143	0.088	0.29	0.23	0.071	0.007	0.7%
3207	0.121	0.068	0.251	0.166	0.076	0.015	0.129	0.081	0.251	0.251	0.093	0.015	-6.2%
3208	0.085	0.05	0.213	0.094	0.055	0.035	0.086	0.061	0.213	0.09	0.058	0.035	-1.2%
3210	0.02	0.016	0.041	0.038	0.002	0	0.02	0.021	0.041	0.041	0	0	0.0%
3211	0.046	0.013	0.086	0.052	0.038	0.025	0.046	0.015	0.086	0.05	0.041	0.025	0.0%
3301	0.046	0.021	0.122	0.063	0.026	0	0.046	0.028	0.122	0.063	0.017	0	0.0%
3302	0.025	0.015	0.048	0.041	0.01	0	0.025	0.018	0.049	0.043	0.007	0	0.0%
3303	0.096	0.109	0.38	0.104	0.025	0.006	0.094	0.12	0.38	0.093	0.014	0.006	2.1%
3304	0.072	0.078	0.38	0.128	0.007	0	0.076	0.099	0.38	0.152	0.001	0	-5.3%
3305	0.053	0.05	0.38	0.069	0.018	0	0.054	0.065	0.38	0.077	0.009	0	-1.9%
3306	0.108	0.141	0.38	0.251	0.005	0.001	0.111	0.152	0.38	0.306	0.001	0.001	-2.7%
3400	0.023	0.024	0.129	0.037	0.003	0	0.025	0.03	0.129	0.044	0.002	0	-8.0%
3401	0.052	0.049	0.245	0.07	0.012	0.001	0.05	0.056	0.245	0.064	0.007	0.001	4.0%
3402	0.07	0.069	0.26	0.134	0.009	0	0.075	0.081	0.26	0.152	0.007	0	-6.7%
3403	0.056	0.036	0.26	0.061	0.034	0.001	0.054	0.044	0.26	0.061	0.023	0.001	3.7%
3404	0.109	0.063	0.26	0.135	0.071	0.001	0.109	0.08	0.26	0.136	0.052	0.001	0.0%
3405	0.046	0.005	0.057	0.047	0.04	0.04	0.046	0.005	0.057	0.05	0.04	0.04	0.0%
3406	0.074	0.051	0.26	0.087	0.045	0.003	0.074	0.058	0.26	0.082	0.053	0.003	0.0%
3407	0.084	0.063	0.26	0.125	0.035	0.009	0.087	0.083	0.26	0.143	0.02	0.009	-3.4%
3408	0.044	0.024	0.102	0.053	0.021	0.007	0.042	0.028	0.102	0.044	0.017	0.007	4.8%
3409	0.039	0.037	0.22	0.037	0.021	0.009	0.04	0.053	0.26	0.035	0.015	0.009	-2.5%
5001	0.335	0.204	0.639	0.543	0.133	0.077	0.339	0.267	0.639	0.639	0.121	0.077	-1.2%
5002	0.671	0.776	3.091	0.732	0.167	0.064	0.67	0.96	3.091	0.612	0.153	0.059	0.1%
5003	1.721	2.195	7.508	2.337	0.091	0.055	1.812	2.654	7.508	1.491	0.088	0.038	-5.0%

Vein	ID3 Estimate						Nearest Neighbor						Mean
	Mean	Std. Dev.	Max	Q3	Q1	Min	Mean	Std. Dev.	Max	Q3	Q1	Min	Diff
5004	1.671	3.361	14.597	1.464	0.153	0.044	1.73	4.021	14.93	0.449	0.125	0.043	-3.4%
5005	1.595	3.085	14.929	0.842	0.076	0.016	1.256	3.774	14.93	0.292	0.029	0.016	27.0%
5006	0.425	0.738	4.802	0.427	0.07	0.038	0.377	0.916	4.87	0.56	0.062	0.038	12.7%
5007	1.023	0.586	3.419	1.465	0.544	0.081	0.976	0.962	3.439	1.123	0.181	0.08	4.8%
5008	1.594	3.097	14.93	0.838	0.109	0.053	1.462	3.343	14.93	0.72	0.108	0.053	9.0%
5009	0.088	0.065	0.237	0.144	0.024	0.008	0.093	0.081	0.237	0.174	0.013	0.007	-5.4%
5010	0.318	0.322	1.576	0.329	0.137	0.031	0.317	0.462	1.578	0.226	0.12	0.031	0.3%
5011	1.979	0.473	2.945	2.369	1.596	1.003	2.092	1.646	3.501	3.501	0.174	0.164	-5.4%

On a local scale, model validation can be confirmed by the visual comparison of block grades to composite grades. Figure 14-8 through Figure 14-11 show typical cross sections where the block and drill color schemes are identical.

Figure 14-8 Comparison of Composite and Estimated Block Gold Grades, Helen Zone

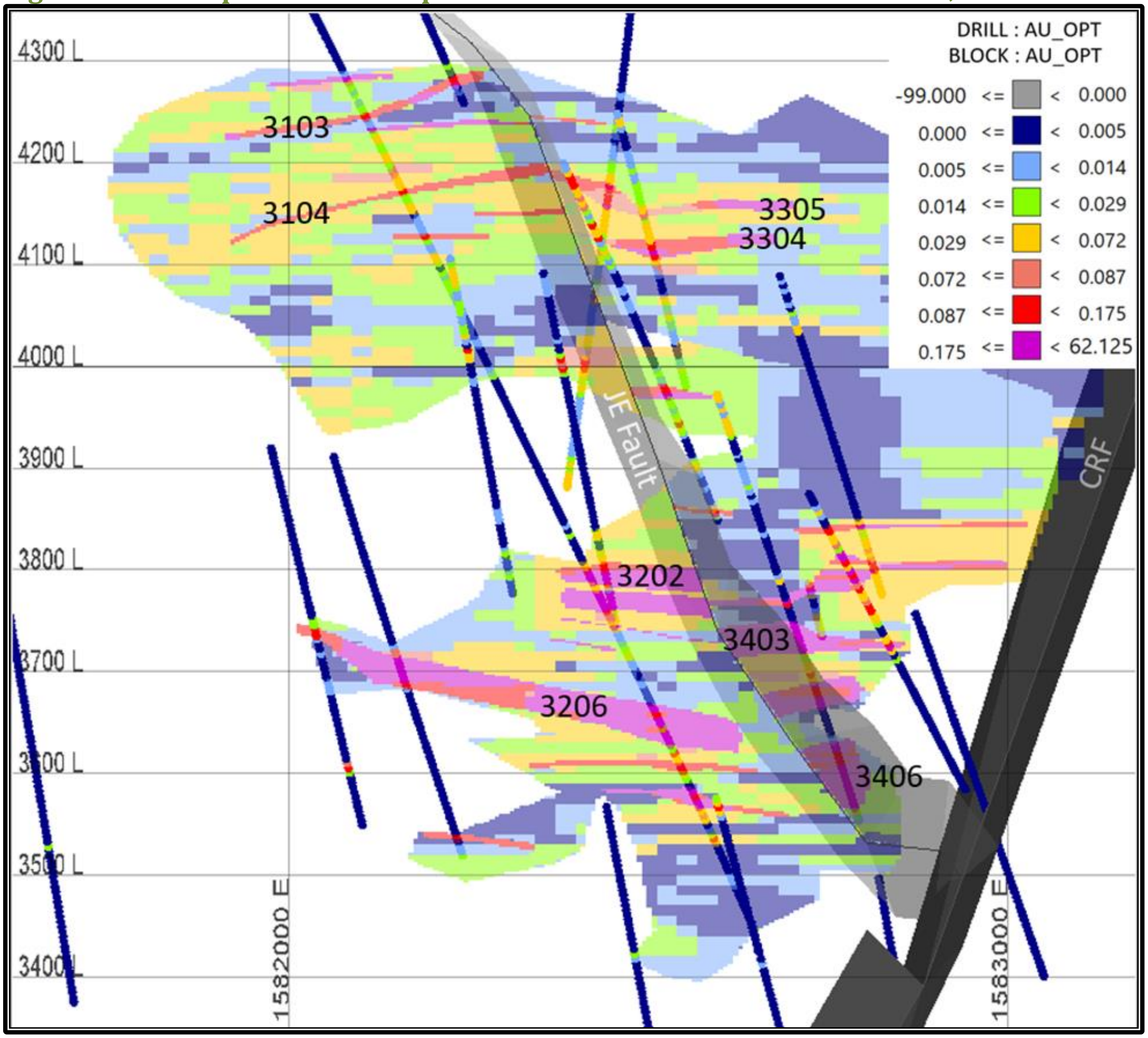


Figure 14-9 Comparison of Composite and Estimated Block Gold Grades, Gap Zone

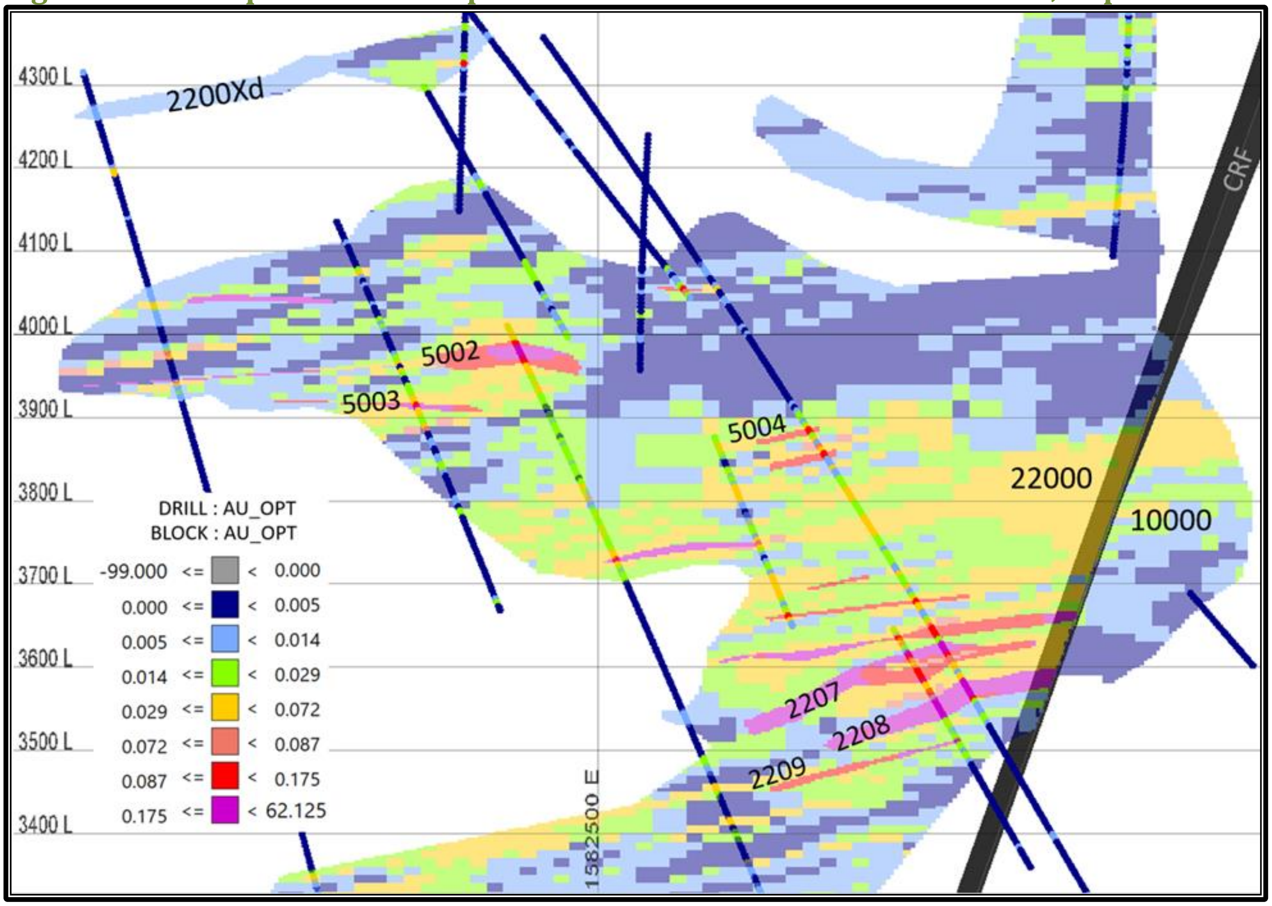


Figure 14-10 Comparison of Composite and Estimated Block Gold Grades, CSD Zone

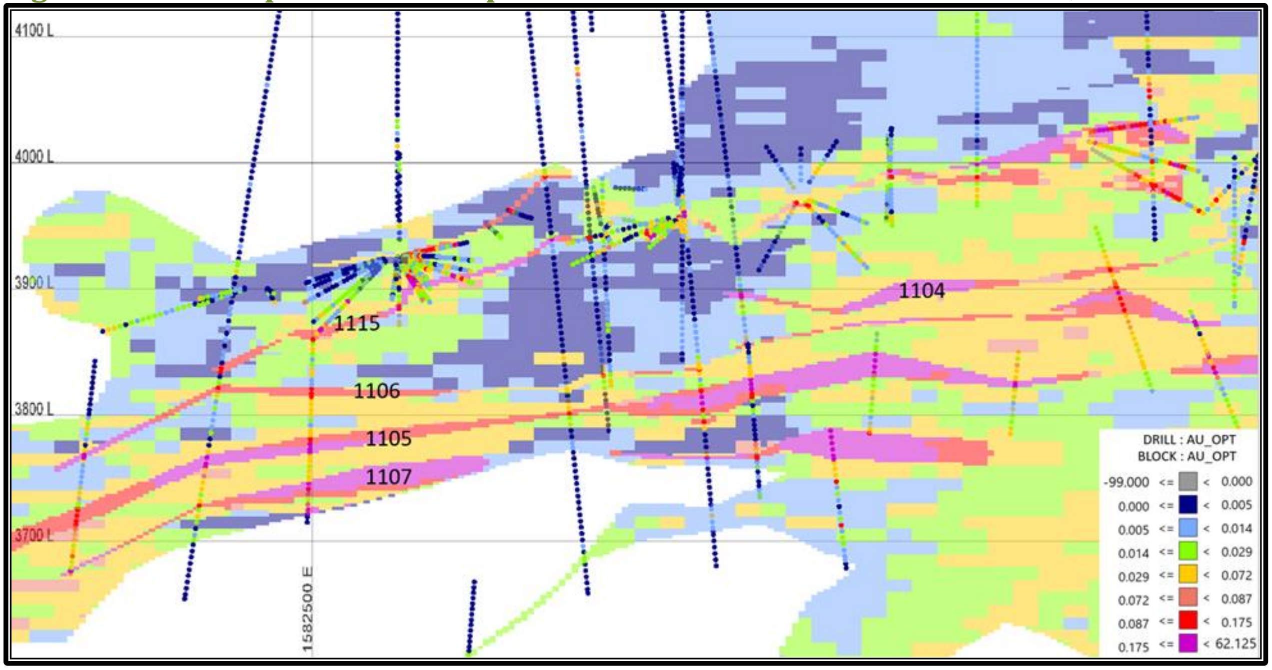
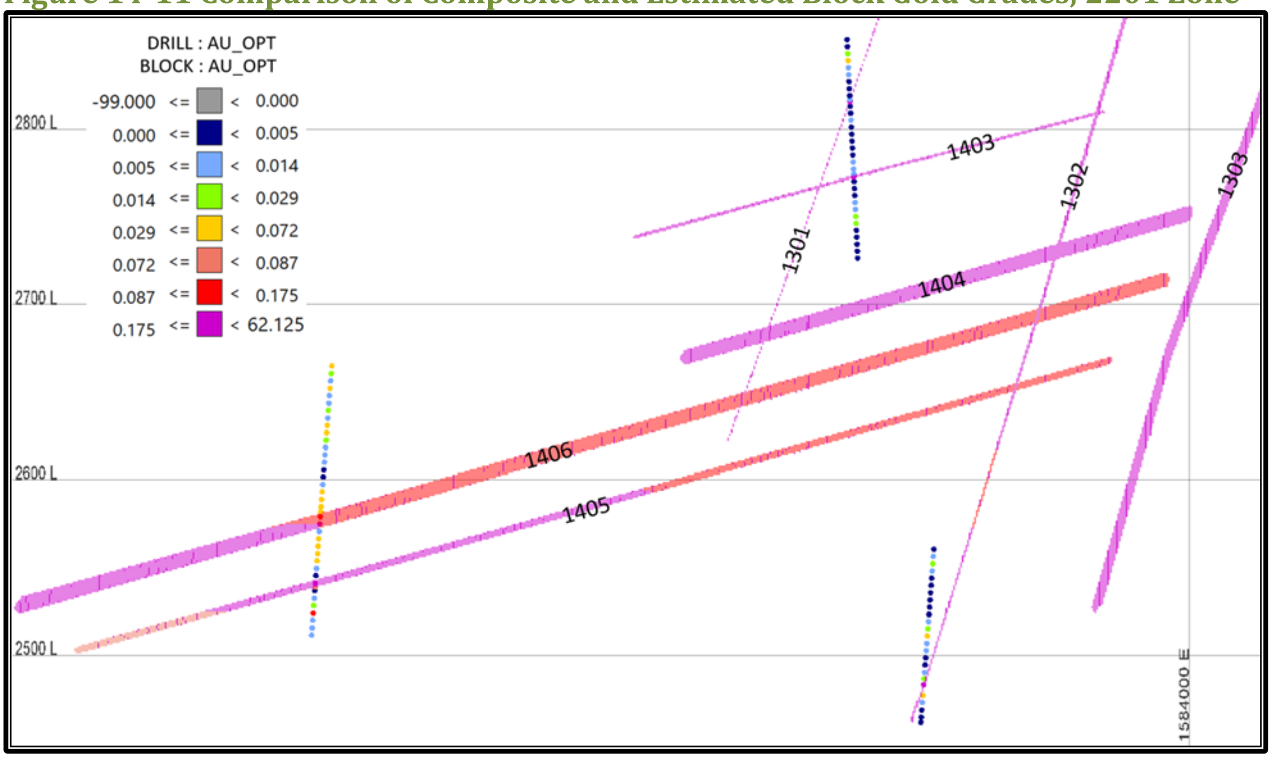


Figure 14-11 Comparison of Composite and Estimated Block Gold Grades, 2201 Zone



Further spatial model validation is provided by swath plots of individual lenses. Swath plots for a typical lens from each zone are presented in Figure x through Figure x. These plots compare the average grade from the estimation to the NN from within regularly spaced swaths or slices through the lens in three dimensions (along strike, along width and vertically). Examination of the swath plots shows good agreement among the gold and silver estimation values.

Figure 14-12 Gold Swath Plots of Helen Zone 3103

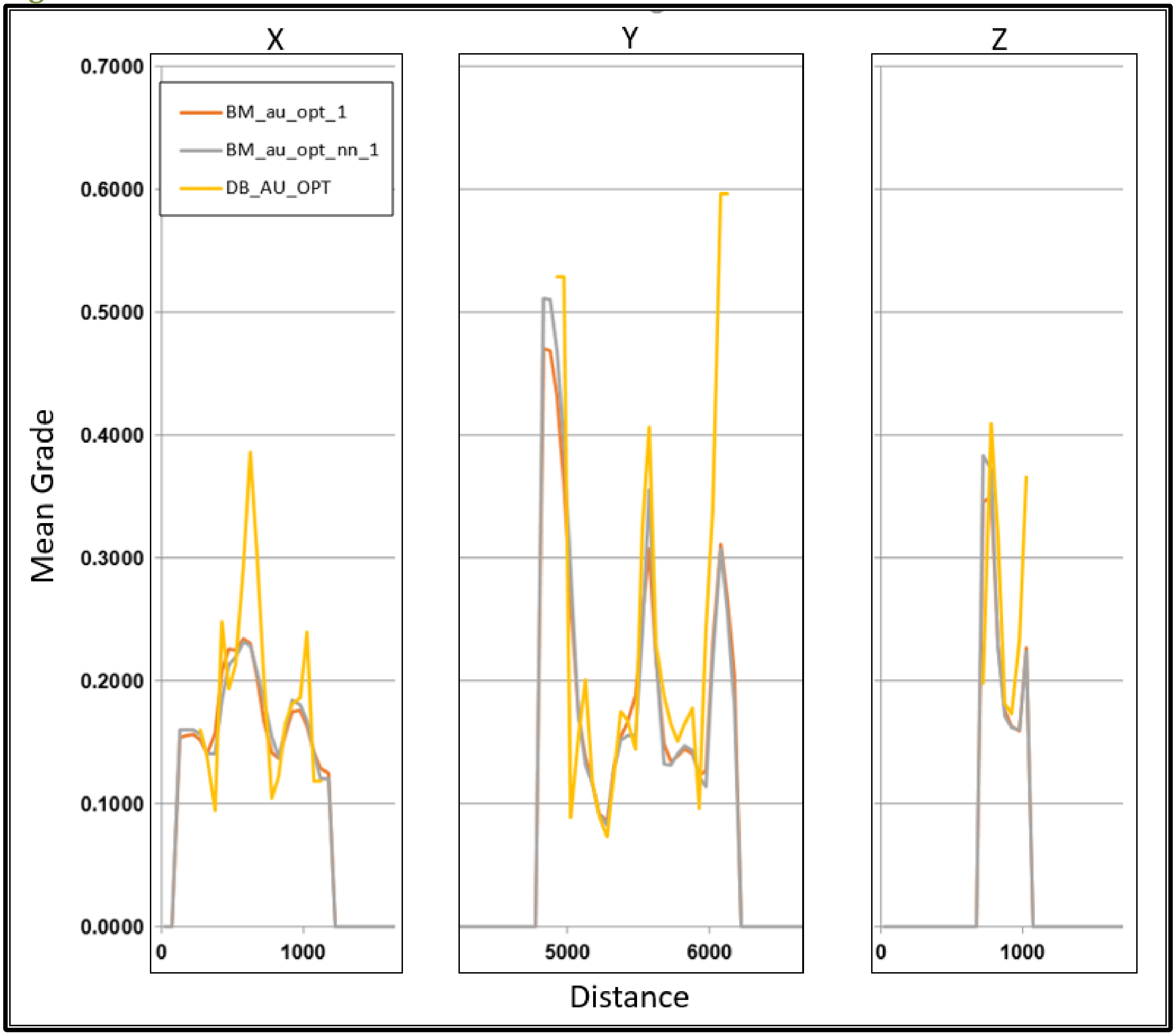


Figure 14-13 Silver Swath Plots of Helen Zone 3103

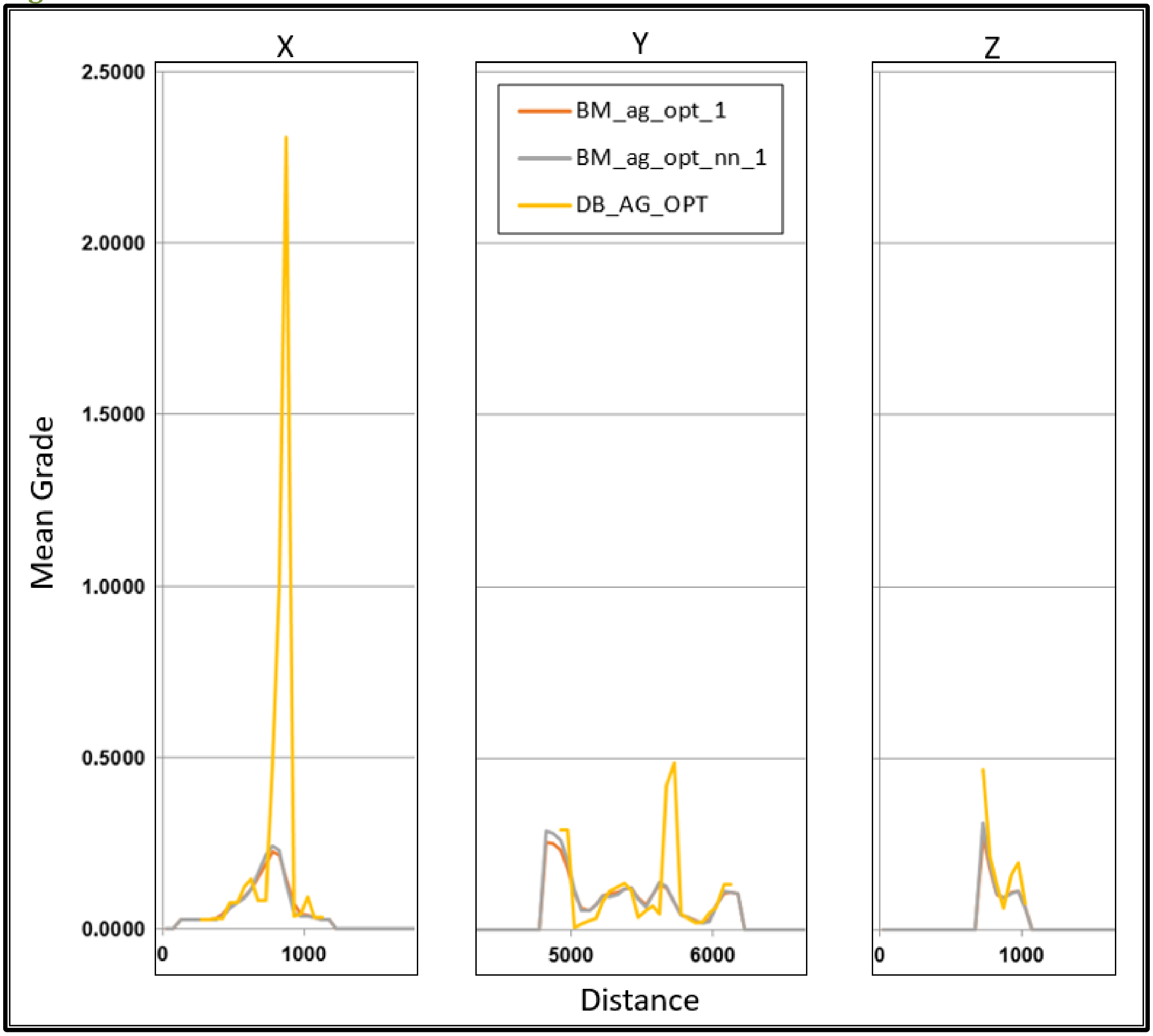


Figure 14-14 Gold Swath Plots of Gap Zone 2208

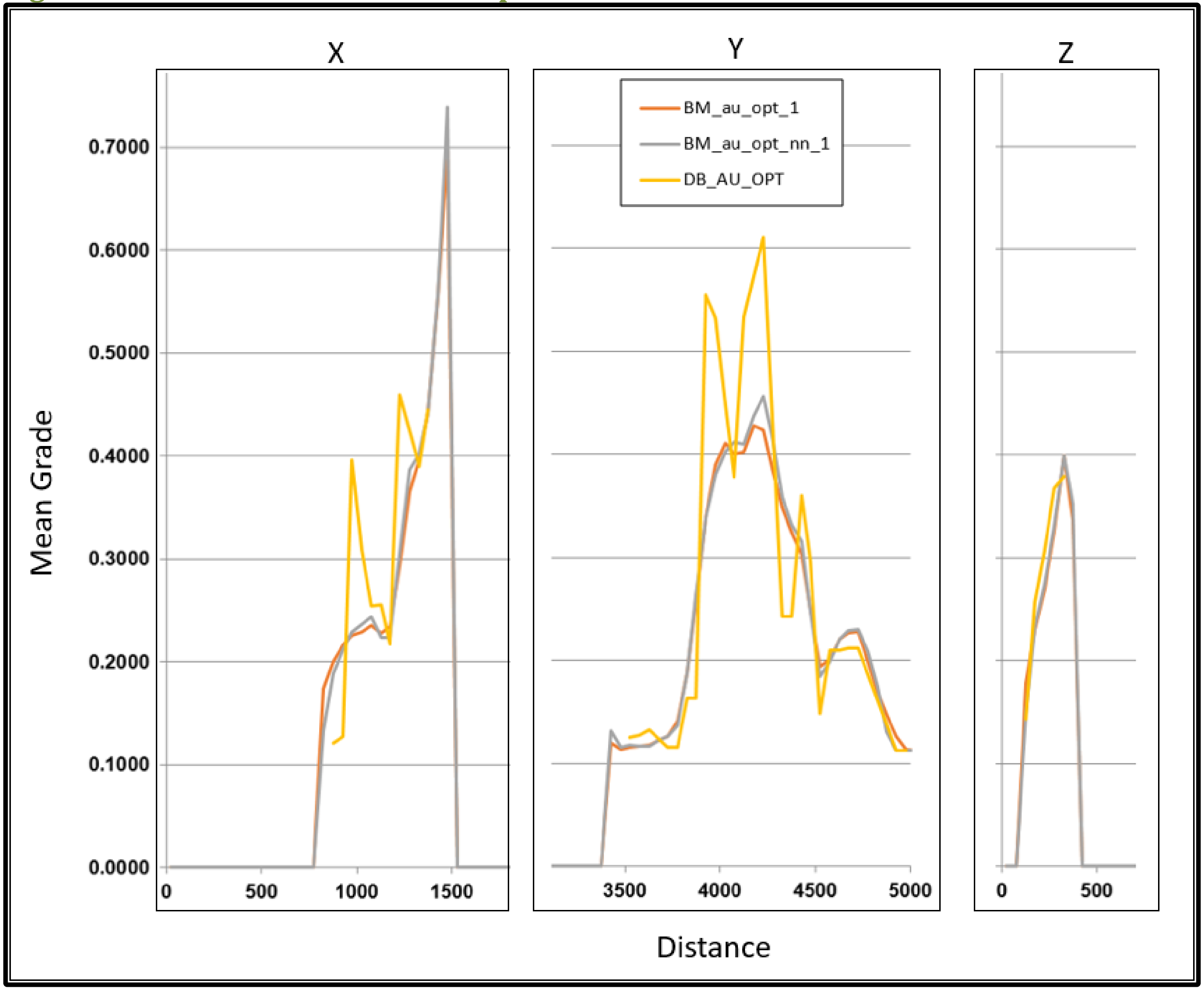


Figure 14-15 Silver Swath Plots of Gap Zone 2208

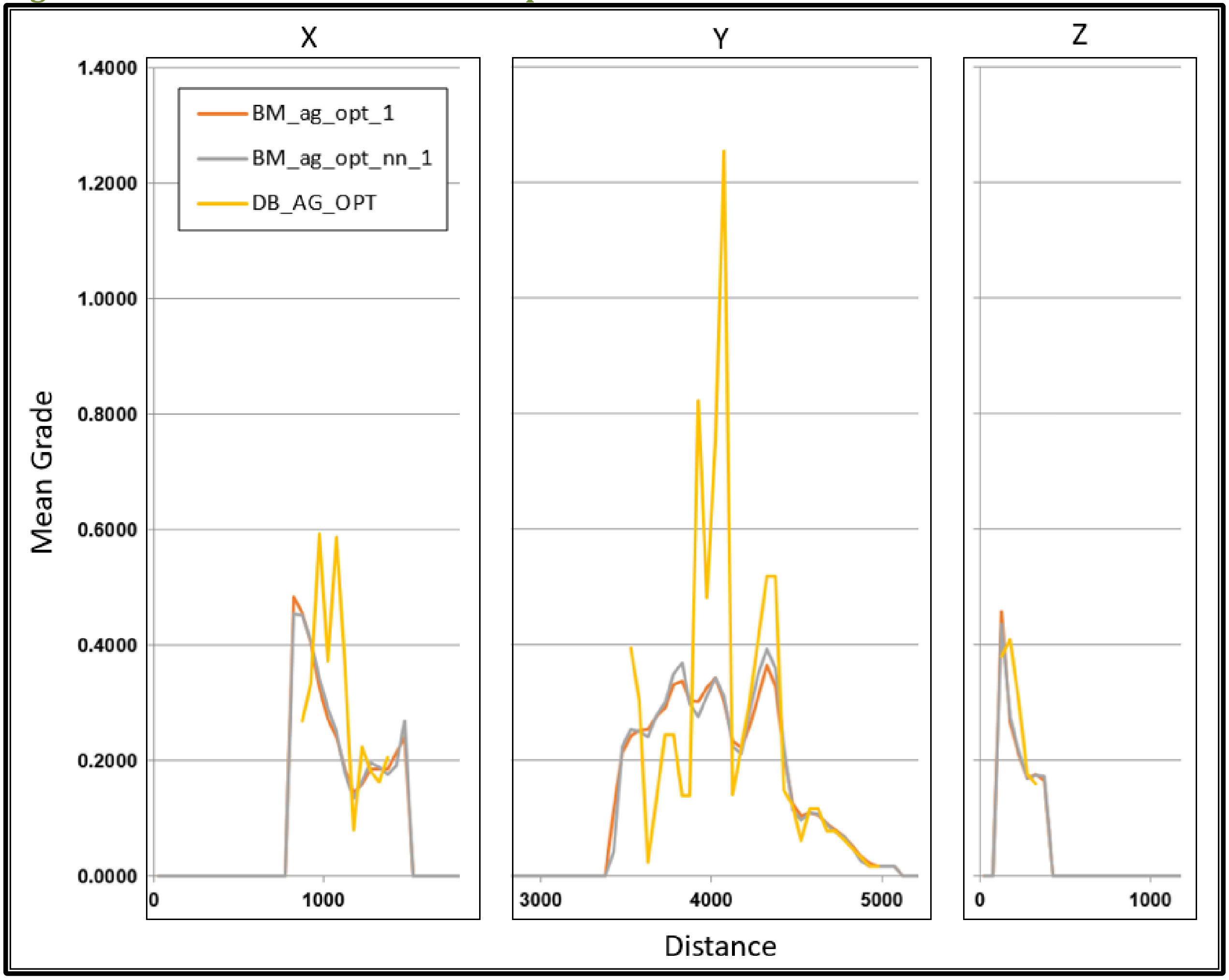


Figure 14-16 Gold Swath Plots of CSD Zone 1106

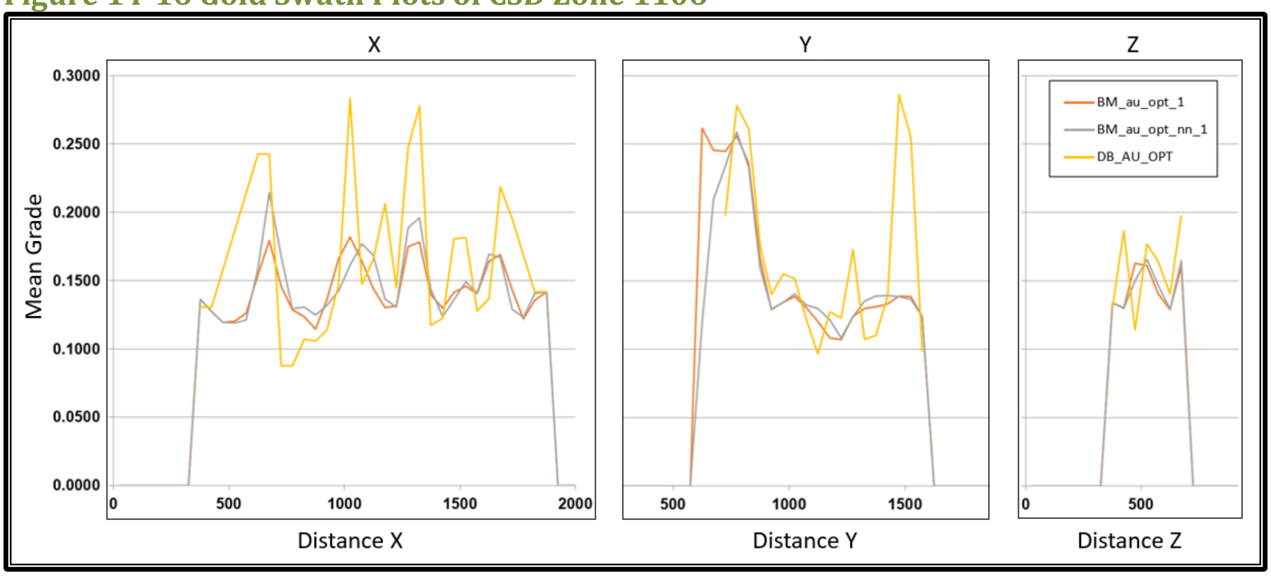


Figure 14-17 Silver Swath Plots of CSD Zone 1106

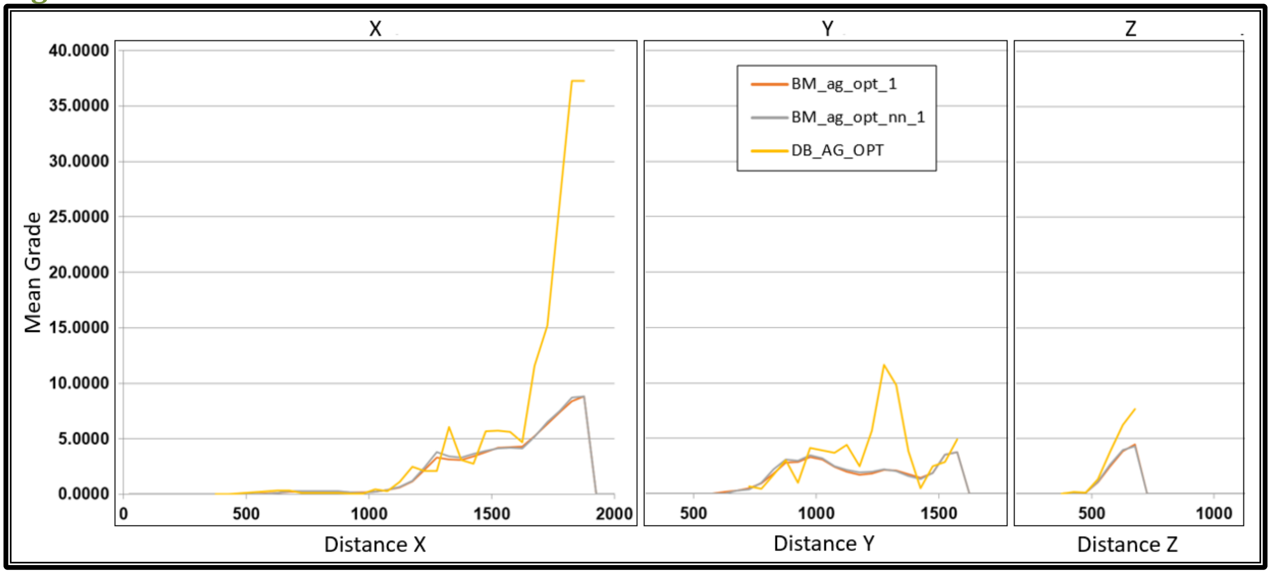


Figure 14-18 Gold Swath Plots of 2201 Zone 1302

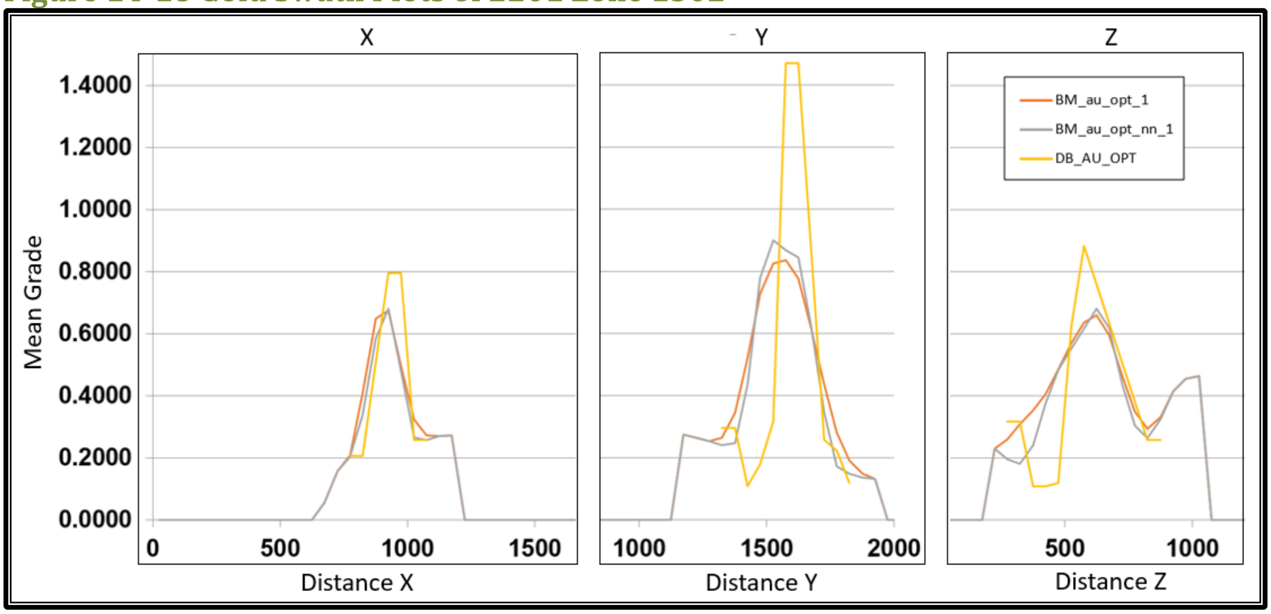
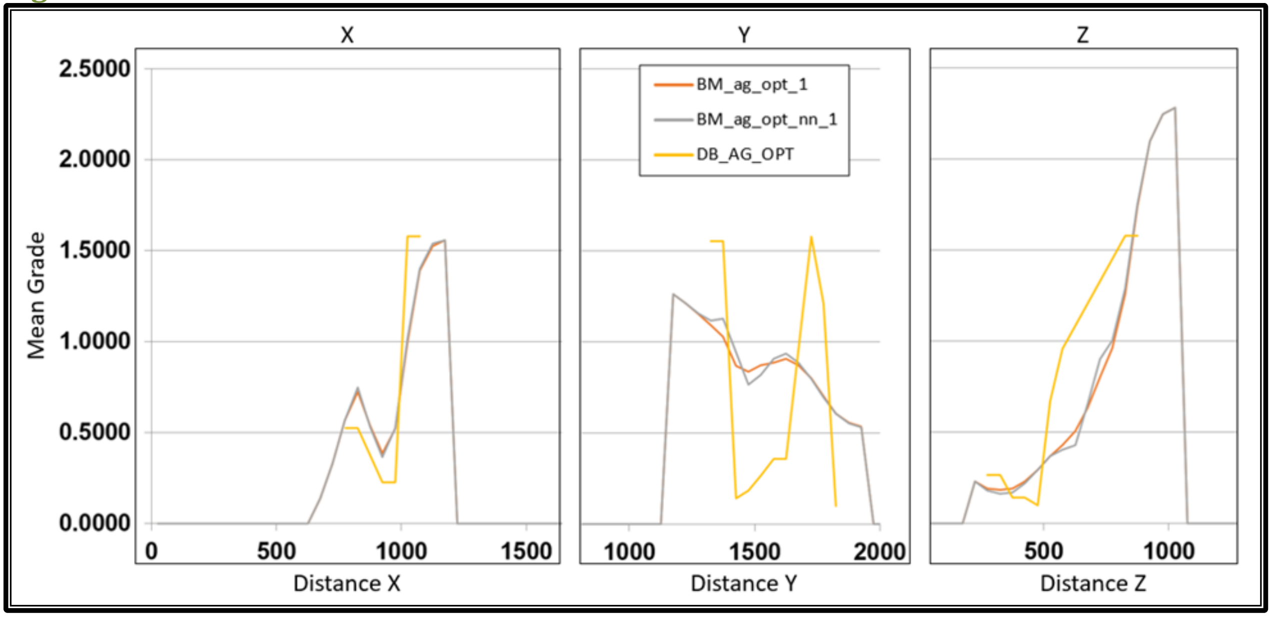


Figure 14-19 Silver Swath Plots of 2201 Zone 1302



14.10.1. Model Smoothing Checks – Grade Tonnage Curves

A final model validation check can be made by examining the grade tonnage distribution for the estimation, which is illustrated in Figure 14-20 through Figure 14-23. The grade tonnage curve is used to describe the tons and grade that may be present above a cutoff for mining. Smoothing in the estimate, the spacing of the informing samples, and the continuity of grades within the vein all affect the shape of the estimated grade tonnage curve. Above a 0.1 opt gold cut-off grade the curve shows gradually increasing grade and decreasing tonnage as the cut-off grade is increased.

Figure 14-20 Helen Zone Grade Tonnage Plots

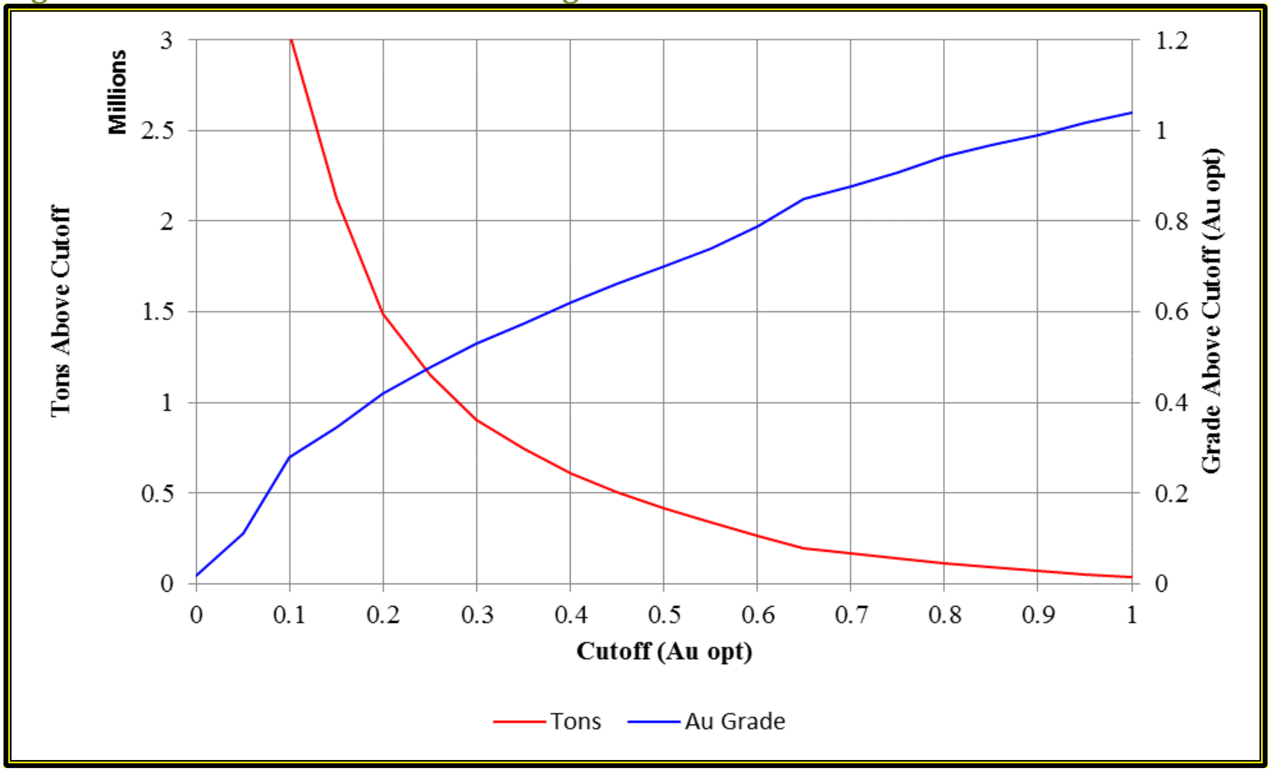


Figure 14-21 Gap Zone Grade Tonnage Plots

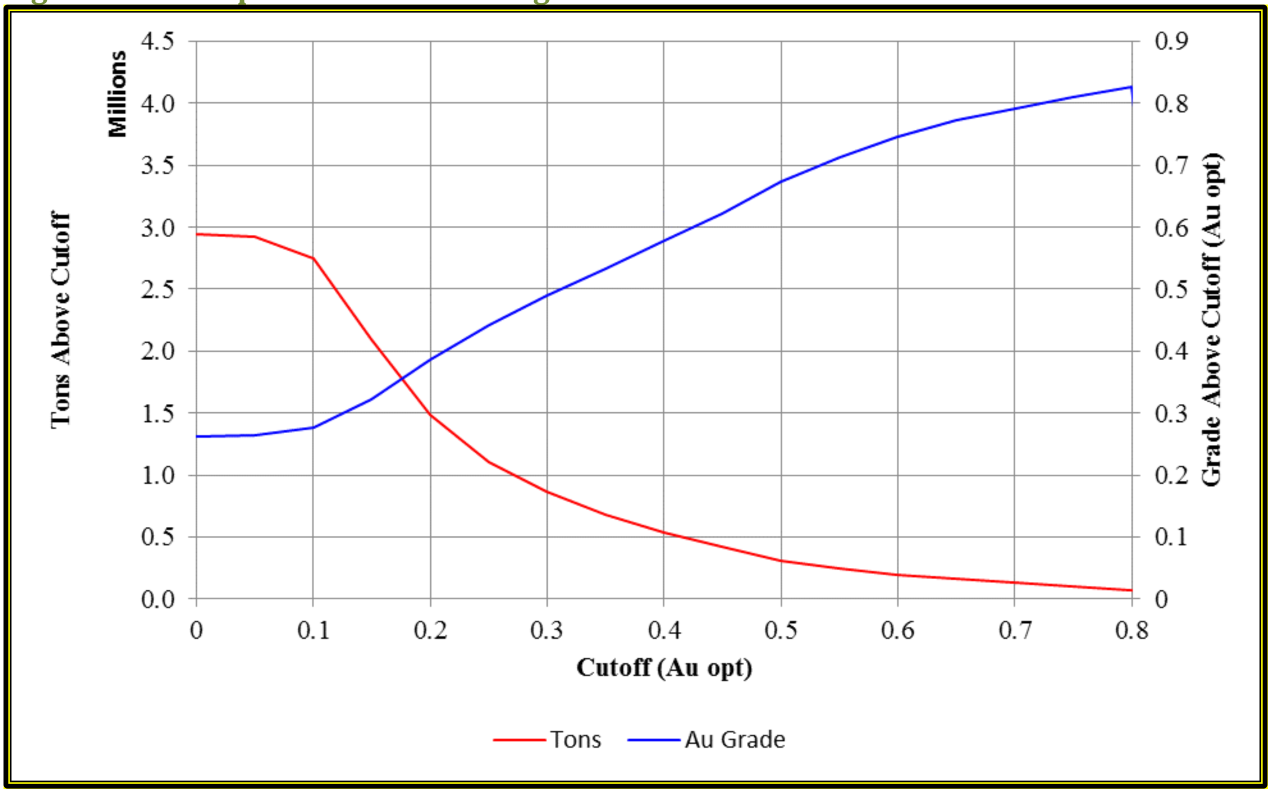


Figure 14-22 CSD Zone Grade Tonnage Plots

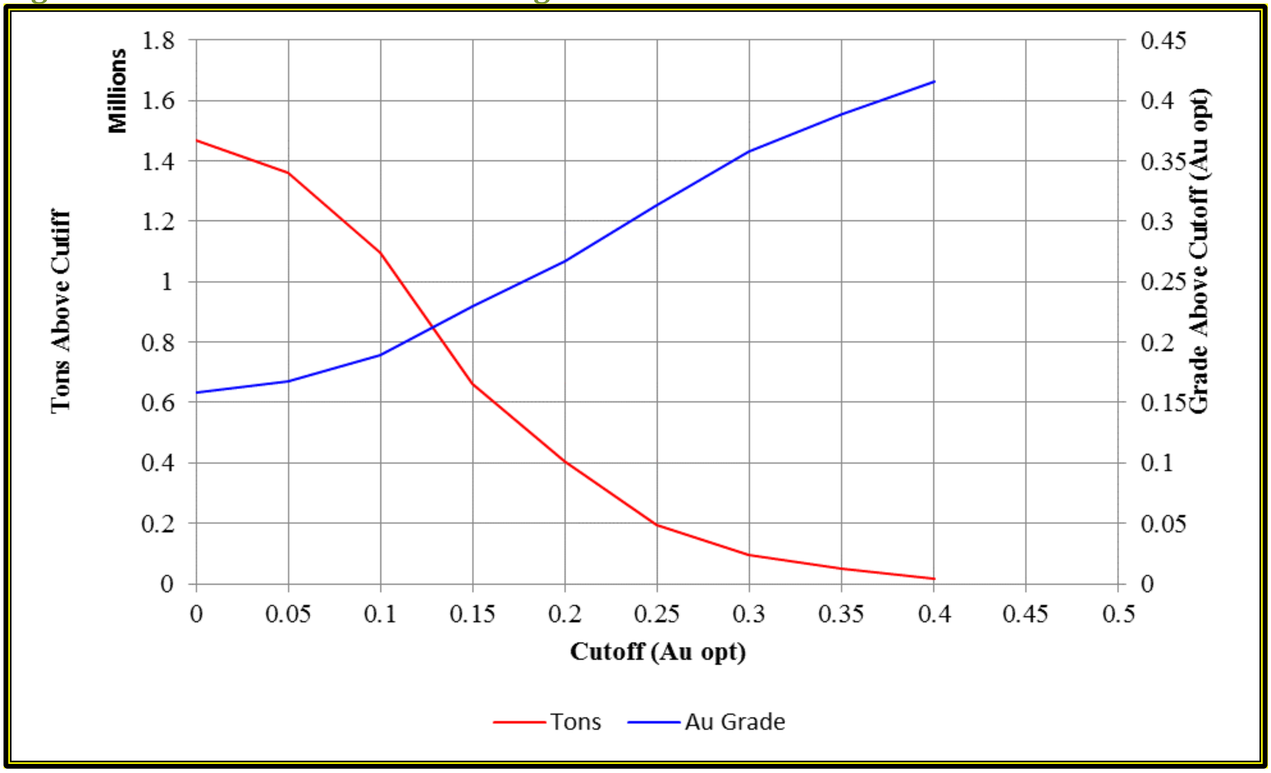
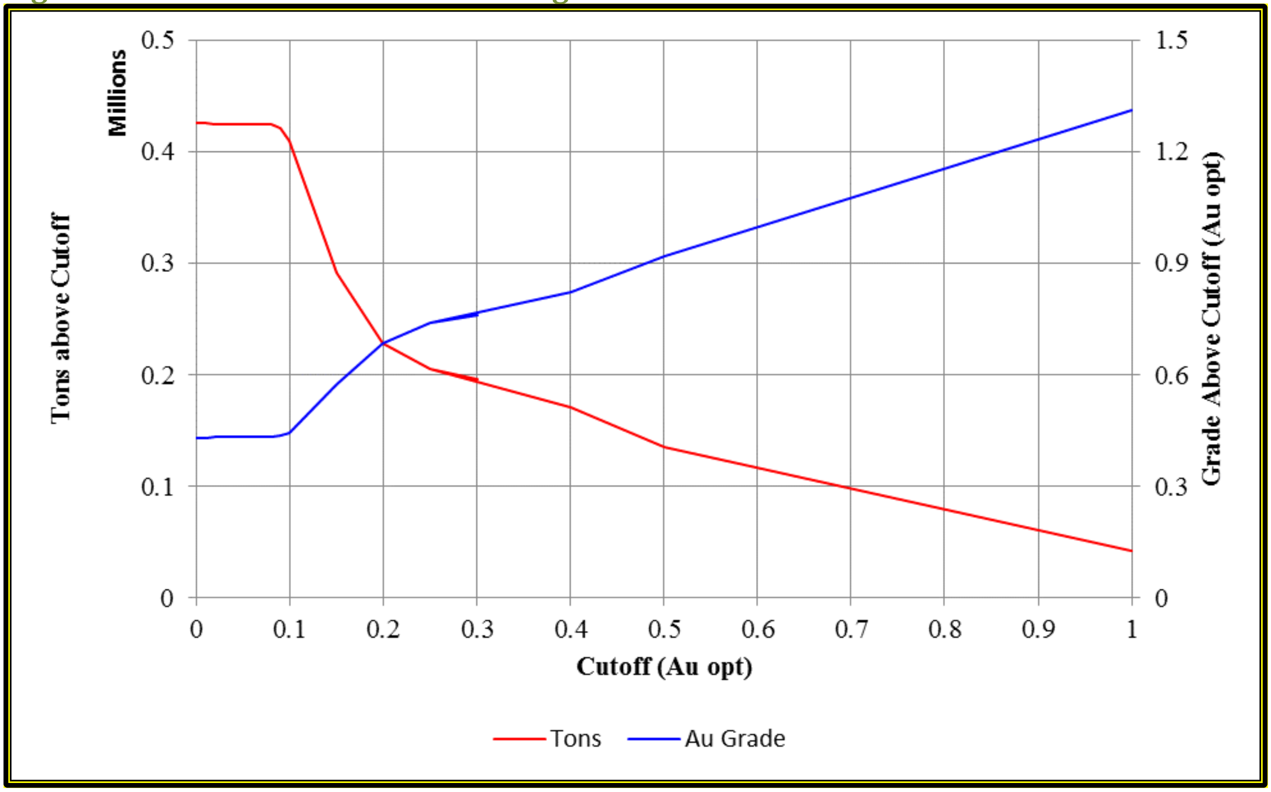


Figure 14-23 2201 Zone Grade Tonnage Plots



14.11. Mineral Resources

Table 14-15 Cove Mineral Resources

	Tons (000)	Tonnes (000)	Au (opt)	Au g/t	Ag (opt)	Ag (g/t)	Au ozs (000)	Ag ozs (000)
Indicated Mineral Resource								
Helen	577	524	0.369	12.66	0.103	3.54	213	60
Gap	167	151	0.357	12.23	0.431	14.78	60	72
CSD	301	273	0.229	7.86	2.556	87.63	69	768
Total Indicated	1,045	948	0.327	11.21	0.861	29.53	342	900
Inferred Mineral Resource								
Helen	1,493	1,355	0.335	11.49	0.118	4.06	500	177
Gap	1,731	1,570	0.317	10.88	0.457	15.67	549	791
CSD	503	456	0.204	7.00	2.266	77.68	103	1,140
2201	310	282	0.546	18.72	1.127	38.65	169	350
Total Inferred	4,037	3,663	0.327	11.23	0.609	20.87	1,322	2,457

Notes:

1. *Mineral Resources have been estimated at a gold price of $1,400 per troy ounce;*
2. *Mineral Resources have been estimated using gold metallurgical recoveries of 79.5% and 85.2% for roasting and pressure oxidation respectively;*
3. *Mineral Resources have been estimated using a gold equivalent cutoff grade of 0.149 opt;*
4. *One ounce of gold is equivalent to 140 ounces of silver;*
5. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors; and*
6. *The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there is insufficient exploration to define these inferred Mineral Resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.*

15. Mineral Reserve Estimates

The Cove Project does not have any Mineral Reserves.

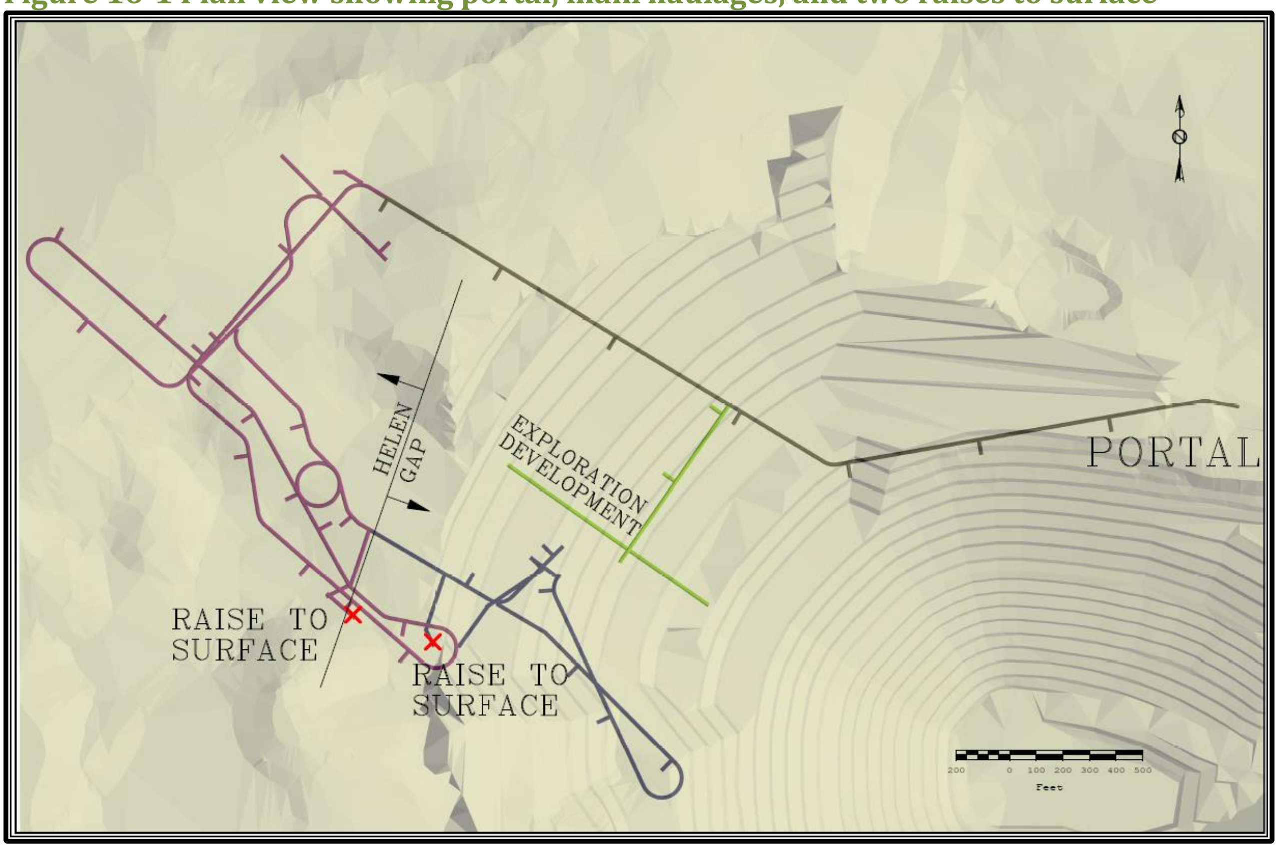
16. Mining Methods

16.1.Mine Development

16.1.1. Access Development

Underground access to the mining areas will begin with a portal on the North side of the existing pit and ramp down. Primary access drifts are designed 15 feet wide and 17.5 feet high to permit 30-ton haulage trucks and provide a large cross section for ventilation. Drift gradients will vary from – 15% to + 15% to reach the desired elevation. Secondary drifts, spiral ramps and vertical raises will connect the haulage drifts to provide a pathway for ventilation to the surface and serve as a secondary escape way. (Figure 16-1)

Figure 16-1 Plan view showing portal, main haulages, and two raises to surface



16.1.2. Ground Support

The ground conditions at the Project are typical of the northern Nevada extensional tectonic environment. Joint spacing varies from a few inches to a foot or more. It is expected that Swellex rock bolts along with welded wire mesh will be able to control all conditions encountered during decline development and stoping. Shotcrete will also be liberally applied as needed to prevent long-term deterioration of the rock mass. Under more extreme conditions, resin anchor bolts, or cable bolts can be used to supplement the primary support. Steel sets and spiling may also be used to support areas with the most severe ground conditions.

Project geologists have recorded core recovery and Rock Quality Designation (RQD) as part of their normal core logging process. Figure 16-2 summarizes RQD for each formation in the mining horizon. RQD values from 30% to the low 40% range are typical for mines in the area. RQD values are also dependent on drill orientation relative to the major joint sets and can vary widely.

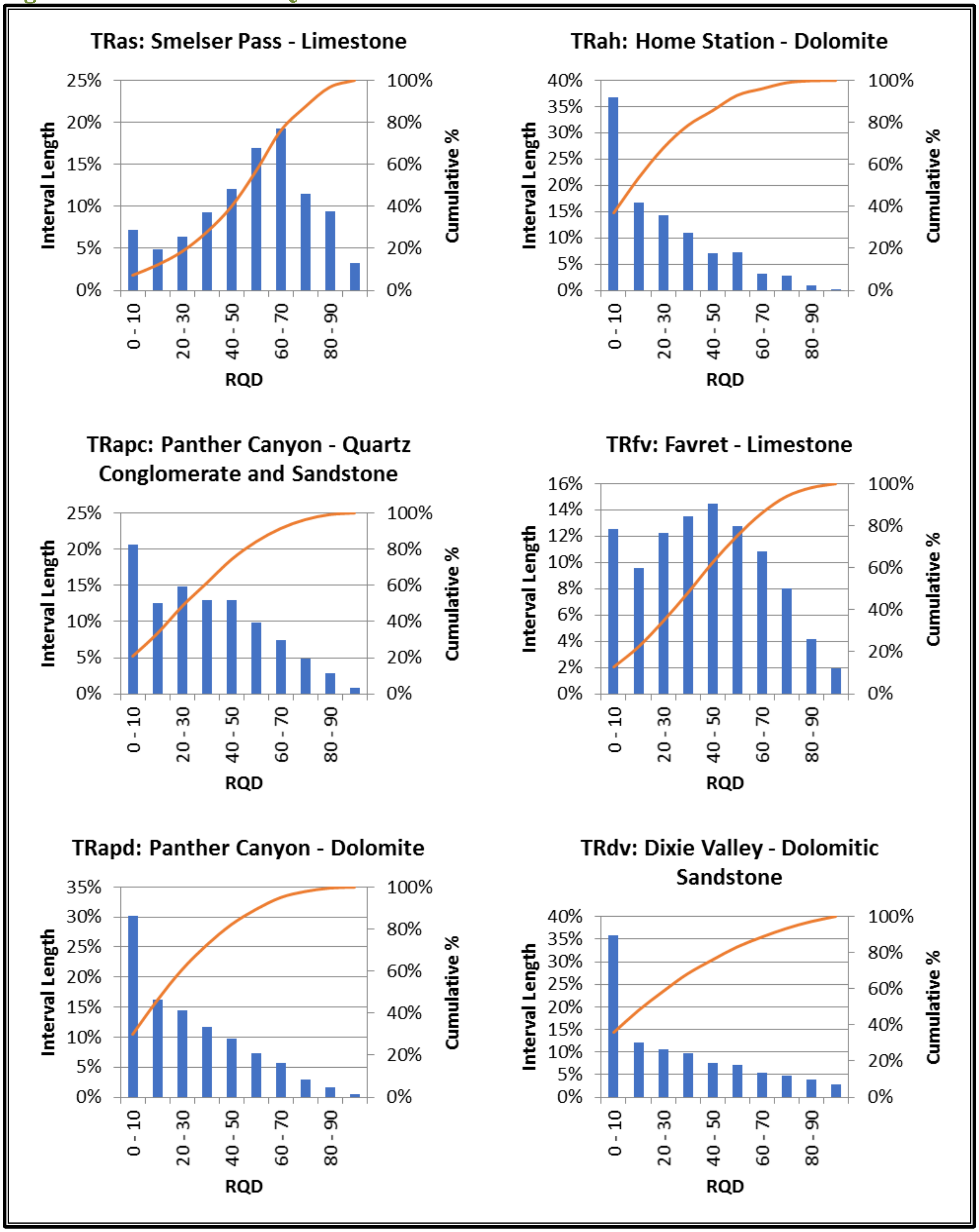
The Modified Rock Mass Rating system proposed by Jakubec and Laubscher (2000) provides for additional characteristics to be considered in addition to RQD. These include filling material, joint waviness, alteration, weathering and the presence of water. A selection of core holes in the resource delineation program should be logged with the MRMR system to allow comprehensive classification of the rock mass.

Joint set orientation relative to the mine opening geometry is the most significant factor in opening stability in north-east Nevada. Also in conjunction with the resource delineation program, Acoustic Tele Viewer logging should be obtained to determine joint orientation for each domain to optimize mine opening orientation and estimate support requirements.

16.1.1. Ventilation and Secondary Egress

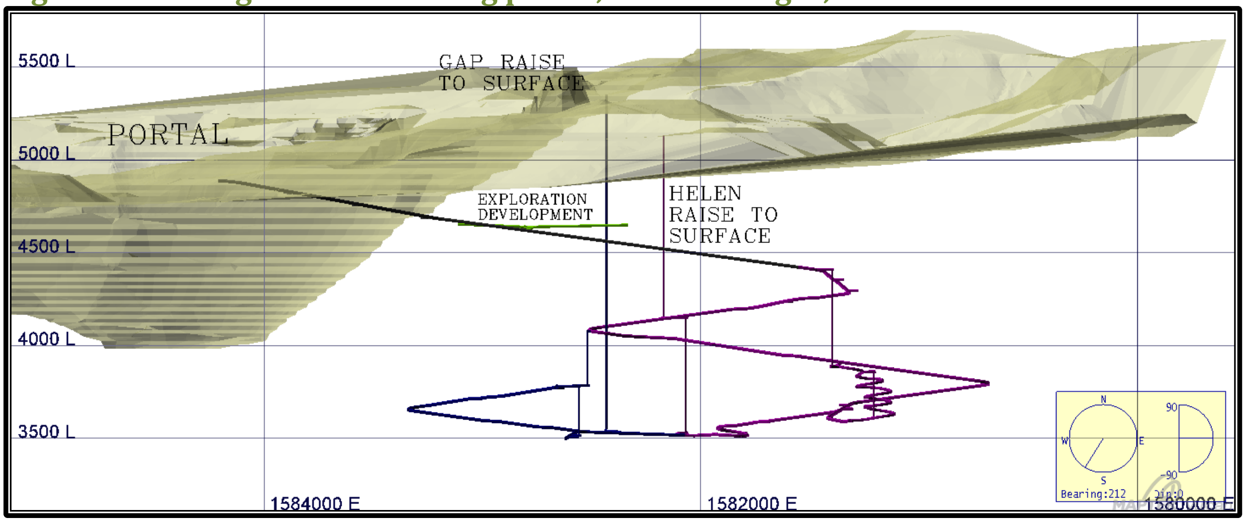
Underground mining relies heavily on diesel equipment to extract the mineralized material and waste rock and to transport backfill to the stopes. Diesel combustion emissions will require substantial amounts of fresh ventilation air to remove the diesel exhaust and maintain a healthy working environment. A combination of the main access drifts and vertical raises to the surface are arranged in a manner to provide a complete ventilation circuit capable of supplying the mine with 500,000 cubic feet per minute (CFM) of fresh air. The mine portal can be used as either an intake or an exhaust. Air movement is facilitated by primary ventilation fans placed at the surface and underground in strategic locations. Small auxiliary fans and ducting will draw primary ventilation air directly into the working faces.

Figure 16-2 Formation RQD



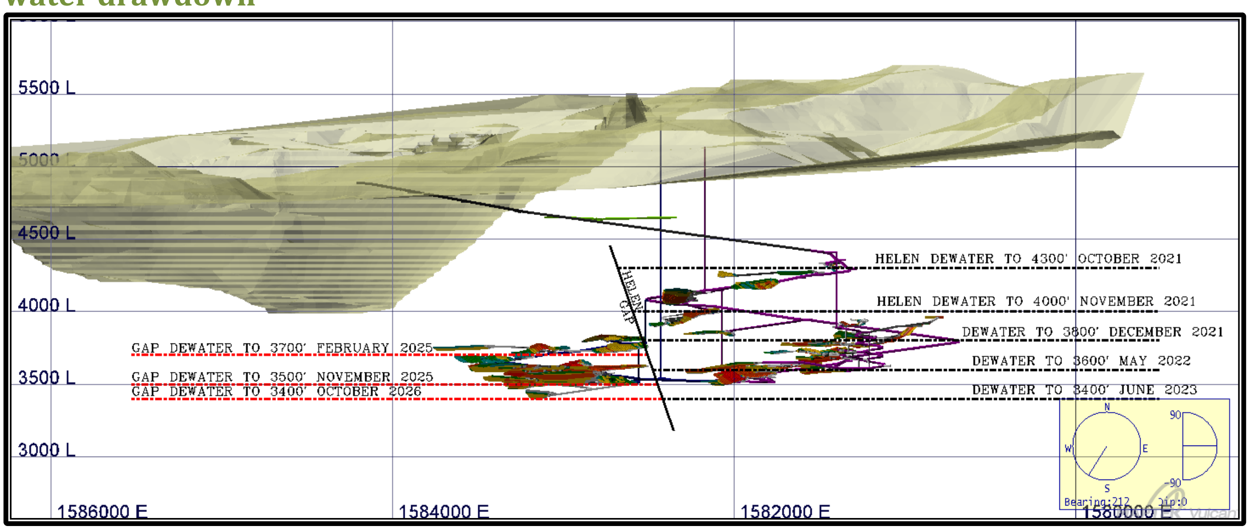
Each zone will have its own designated ventilation raise connecting the main decline to the surface. Since the vertical extent of each of the raises exceeds the maximum 300 feet permitted for a continuous ladder way, each raise will be equipped with an automatic hoist and personnel capsule for evacuating the mine in the event of an emergency. (Figure 16-3 and Figure 16-4)

Figure 16-3 Long Section showing portal, main haulages, and two raises to surface



16.1.2. Dewatering

Figure 16-4 Gap and Helen Development and Production depicting dates of planned water drawdown



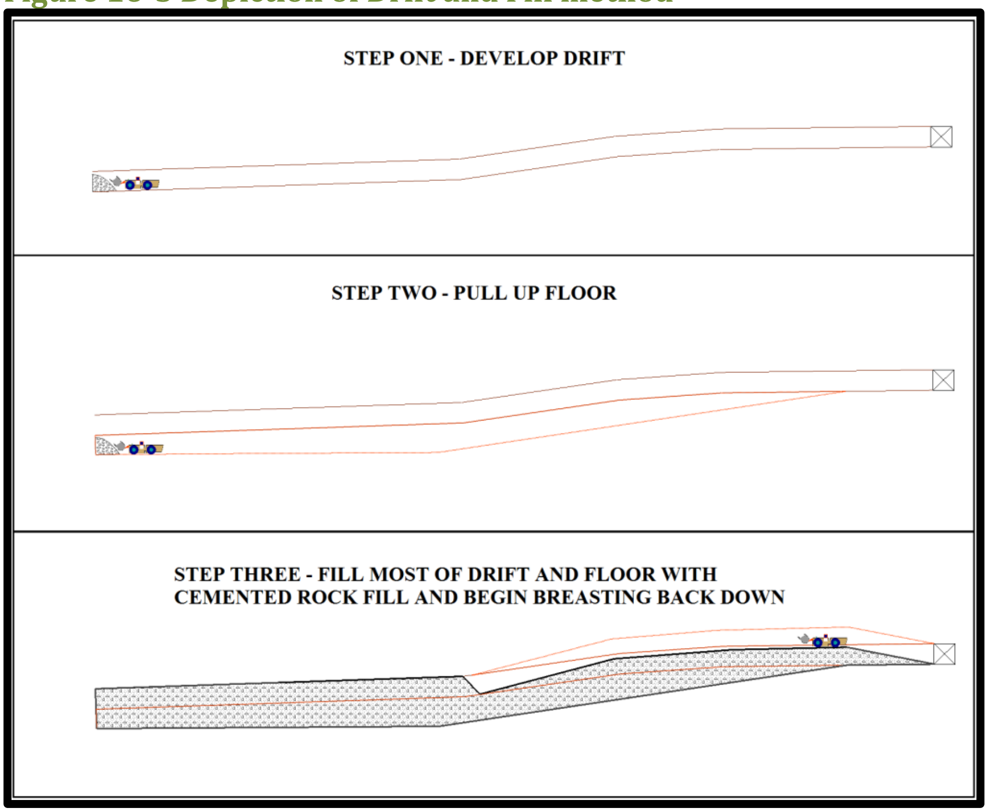
16.2. Mining Methods

Due to the mostly flat geometry of the ore lenses, all planned production mining will be completed using drift and fill mining. The final choice of mining method will depend upon the geometry of the stope block, proximity to main access ramps, ventilation and escape routes, the relative strength or weakness of the mineralized material and adjacent wall rock, and finally the value or grade of the mineralized material. The choice of mining method will not be finalized until after the stope delineation and definition drilling is completed. The drift and fill method is discussed briefly in the following paragraphs.

16.2.1. Drift and Fill

Drift and Fill is a very selective mining method. A drift and fill stope is initiated by driving a waste crosscut from the access ramp to the ore. The initial ore drift is driven at planned 13-feet wide by 13-feet high dimensions, with gradient varying between +/-20% to follow the geometry of the mineralization. The minimum cut and fill drift height is eight feet to minimize dilution on the thinner mineralized lenses. Once the initial drift is driven, floor may be pulled and/or back may be breasted down to capture the full thickness of the lens. Where mining is planned adjacent to the drift, it will be backfilled with CRF prior to mining the subsequent drifts. (Figure 16-5)

Figure 16-5 Depiction of Drift and Fill method



16.3. Underground Labor

Approximately 4,000 feet of development will be undertaken in the second half of 2018 to provide access for underground delineation and exploration drilling. Underground workforce requirements for this early development phase of the Project are estimated in Table 16-1. Following a positive production decision in 2021, production will increase and peak underground workforce requirements for the Project are presented in Table 16-2. This estimate was prepared using productivity rates typical for large-scale mechanized mining in North America. The Project will operate 24 hours per day seven days per week. Project operations workforce will be divided into four crews scheduled to work 14 out of every 28 days.

Table 16-1 Underground Workforce 2018 through 2020

Job Classification	Count
Miners	8
Mechanics	4
Supervision	2
Technical Staff	8
Manager	1
Total	23

Table 16-2 Peak Underground Workforce beginning 2021

Job Classification	Count
Miners	80
Mechanics/Electricians	20
Supervision	8
Technical Staff	16
Manager	1
Total	125

16.4.Mobile Equipment Fleet

During the early exploration phase, capital development drifting will average 10-15 feet per day from 2018 through early 2019. Following a positive production decision, ore production will begin to approach 500 tpd until water levels draw down enough in 2024 to achieve maximum production of 1360 tpd, a rate that will hold steady until all the ore is exhausted. Table 16-3 lists the mining fleet necessary to achieve the development and production goals during the test mining phase. Table 16-4 lists the mining fleet necessary to achieve the development and production goals for peak mining levels.

Table 16-3 Underground Mobile Equipment and Support Equipment for Exploration Development Phase

Description	Quantity
6-Yd LHD	1

Description	Quantity
30-T Haul Truck	1
Jumbo Drill	1
Bolter	1
Fork Lift	1
Lube Truck	1
Grader	1
Emergency Rescue	1
Tractor	2
UTV	1

Table 16-4 Underground Mobile Equipment and Support Equipment for Peak Production Mining

Description	Quantity
6-Yd LHD	6
30-T Haul Truck	8
Jumbo Drill	4
Bolter	4
Remix Truck	2
Cement Pump	2
Fork Lift	2
Lube Truck	1
Grader	1
Emergency Rescue	1
Heavy Duty Pickup	1
Tractor	3
UTV	4

16.5. Mine Plan

The productivities of Table 16-5 were used to develop the production plan. The production plan is limited by overall production rates. Assuming a positive production decision in 2021, development and production rates will increase as headings become available, eventually reaching a maximum rate of 100 total feet per day and 1,360 tons of ore production per day. At these rates, the mine plan is exhausted in 2029. The mine plan is depicted in Figure 16-7 through Figure 16-8 and Table 16-6. The production profile ove the life of mine is shown in Figure 16-6

Table 16-5 Heading Productivity

Heading Type	Units	Daily Rate
Primary Capital Development Drift	Feet/Day	12
Secondary Capital Development Drift	Feet/Day	10
Raise Bore	Feet/Day	10
Drop Raise	Feet/Day	15
Ore Drift Development	Feet/Day	10
Floor Pulls	Ton/Day	300
Breast Downs	Ton/Day	100
Long Hole Stoping	Ton/Day	500
Backfill	Ton/Day	500

Table 16-6 Annual Production and Development Plan Prior to Production Decision

Calendar Year	2018	2019	2020	Total
Waste Mining				
Expensed Waste Drifting (Feet)	-	-	-	-
Expensed Waste (000's Tons)	-	-	-	-
Primary Capital Drifting (Feet)	1,107	972	-	2,079
Secondary Capital Drifting (Feet)	100	1,816	-	1,916
Capital Raising (Feet)	-	-	-	-
Capitalized Mining (000's Tons)	23.8	51.2	-	75.0
Mining Rate (tpd)	130	140	-	137

Table 16-7 Annual Production and Development Following Positive Production Decision

Calendar Year	2021	2022	2023	2024	2025	2026	2027-2029	Total
Ore Mined								
Indicated Ore Mined (000's Tons)	0.1	19.8	19.2	199.1	109.9	97.7	137.9	583.8
Gold Grade (Ounce/Ton)	0.031	0.325	0.325	0.315	0.329	0.340	0.321	0.324
Silver Grade (Ounce/Ton)	0.016	0.112	0.122	0.075	0.097	0.238	0.201	0.139
Contained Gold (000's Ounces)	-	6.4	6.2	62.7	36.1	33.2	44.3	189.0
Contained Silver (000's Ounces)	-	2.2	2.3	15.0	10.6	23.3	27.7	81.2
Inferred Ore Mined (000's Tons)	1.4	144.4	153.7	290.7	364.2	401.1	1,010.8	2,366.2
Gold Grade (Ounce/Ton)	0.130	0.250	0.290	0.328	0.323	0.325	0.284	0.300
Silver Grade (Ounce/Ton)	0.029	0.100	0.110	0.081	0.110	0.333	0.220	0.191
Contained Gold (000's Ounces)	0.2	36.0	44.5	95.3	117.7	130.2	286.9	710.8

Calendar Year	2021	2022	2023	2024	2025	2026	2027-2029	Total
Contained Silver (000's Ounces)	-	14.5	16.9	23.5	40.1	133.7	222.4	451.2
Total Ore Mined (000's Tons)	1.5	164.2	172.9	489.8	474.0	498.8	1,148.7	2,950
Gold Grade (Ounce/Ton)	0.121	0.259	0.293	0.322	0.325	0.327	0.288	0.305
Silver Grade (Ounce/Ton)	0.028	0.102	0.111	0.079	0.107	0.315	0.218	0.180
Contained Gold (000's Ounces)	0.2	42.5	50.7	157.9	153.9	163.3	331.3	899.8
Contained Silver (000's Ounces)	-	16.7	19.3	38.5	50.7	157.0	250.2	532.4
Production Mining								
Total Ore Mined (000's Tons)	1.5	164.2	172.9	489.7	473.9	498.6	1,148.5	2,949.3
Ore Production Rate (tpd)	28[1]	450	474	1,338	1,298	1,366	1,048	702
Backfill								
Total Backfill (000's Tons)	1.0	107.7	113.5	321.4	311.0	327.2	753.7	1,935.5
Waste Mining								
Expensed Waste (000's Tons)	2.3	34.8	13.5	46.1	45.2	27.0	80.3	249.3
Primary Capital Drifting (Feet)	3,635	5,717	4,037	1,859	4,799	265	1,386	21,698
Secondary Capital Drifting (Feet)	600	794	855	574	829	39	396	4,086
Capital Raising (Feet)	-	934	578	241	264	-	1,727	3,744
Capitalized Mining (000's Tons)	81.0	132.8	95.1	43.1	107.7	4.9	49.8	514.4
Total Tons Mined (000's Tons)	84.9	331.8	281.5	579.0	626.8	530.5	1,278.5	3,713
Mining Rate (tpd)	309[1]	909	771	1,582	1,717	1,453	1,199	1,161

Figure 16-6 Production Profile

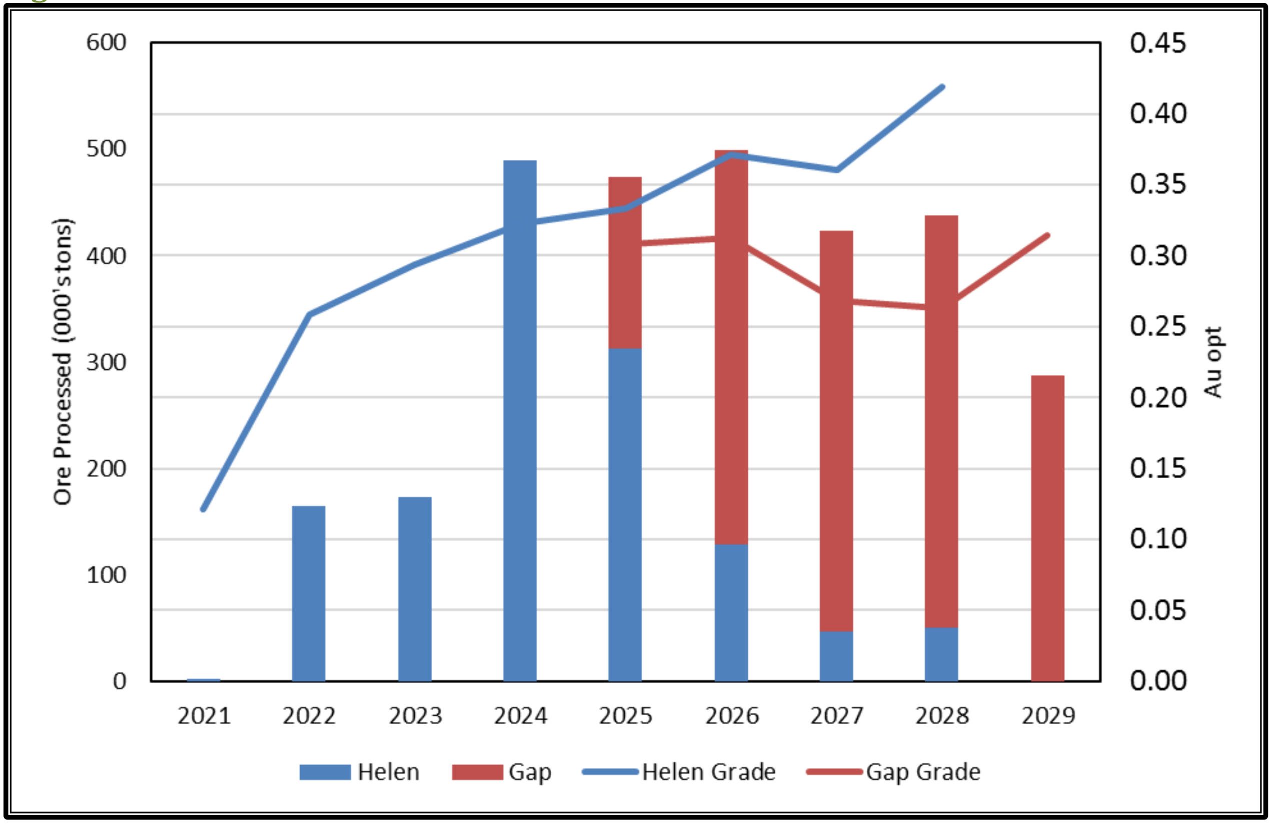


Figure 16-7 Long Section View of Helen Mine Plan by Year

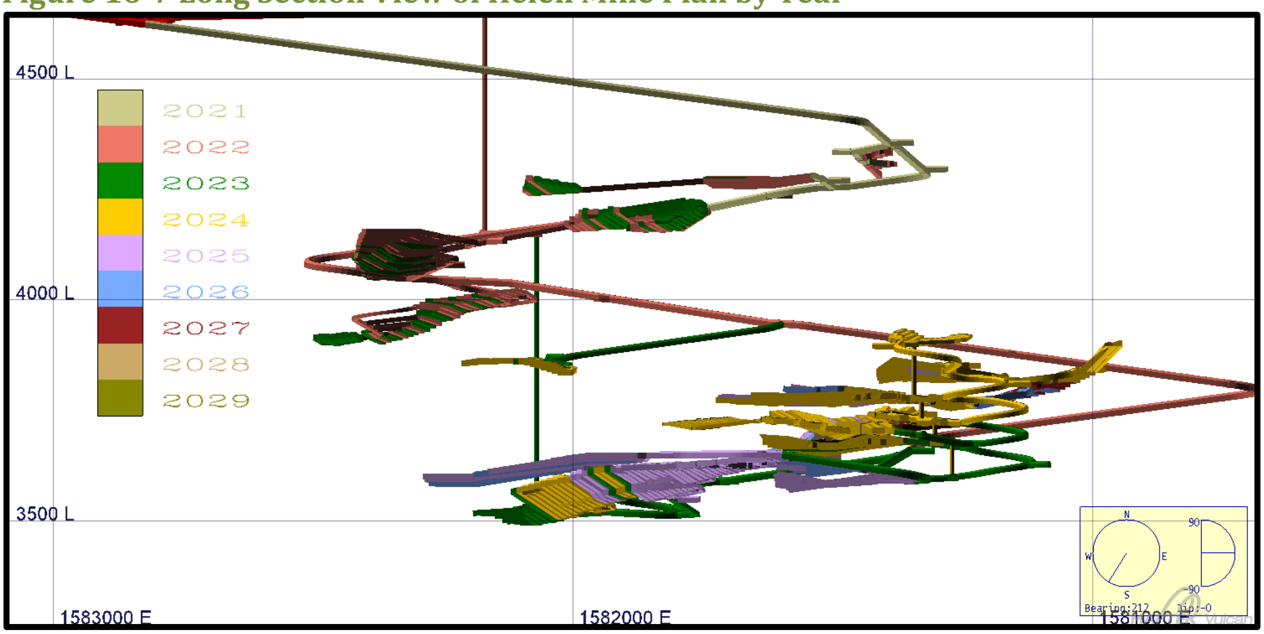
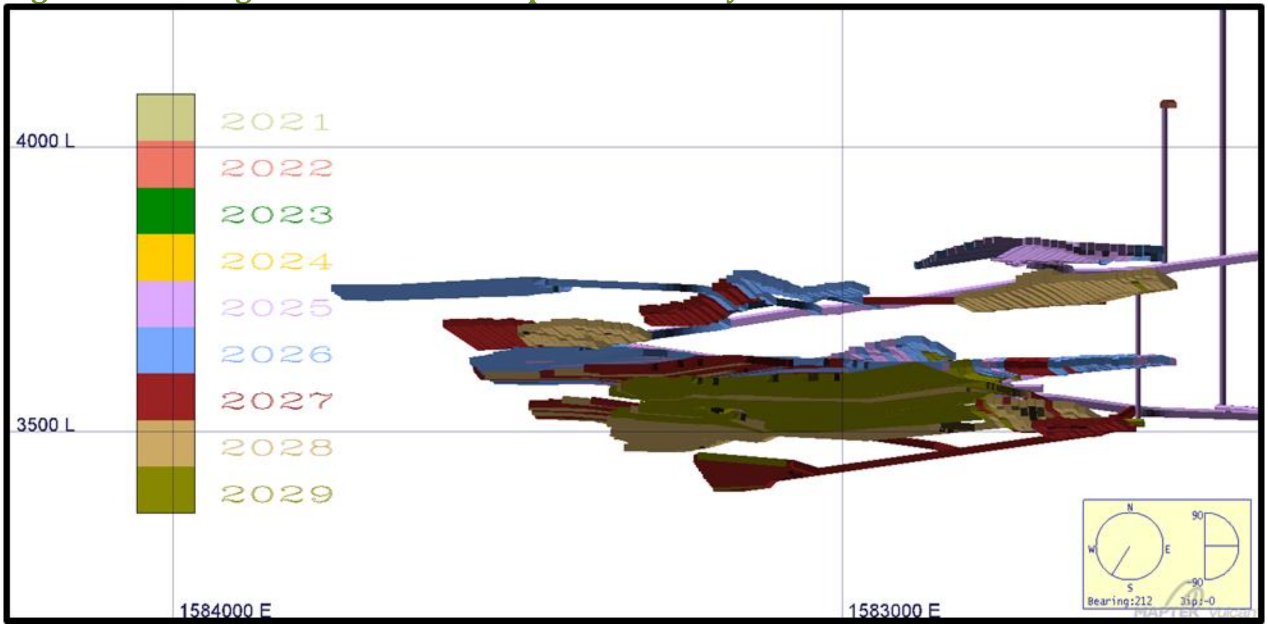


Figure 16-8 Long Section View of Gap Mine Plan by Year



16.6.Mine Plan Reconciliation

Table 16-8 reconciles the mine plan to the Mineral Resource. The mine plan extracts slightly less than ½ of the Helen and Gap Mineral Resource. Low-grade dilution peripheral to the mineralized lenses accounts for 34% of the material mined for processing and grades 0.050 Au opt.

Table 16-8 LOM Plan Reconciliation to Mineral Resource

		Indicated			Inferred		
		tons	grade	koz	tons	grade	koz
Helen	Resource	577	0.369	213	1,493	0.335	500
	Mined PEA Plan	284	0.486	138	637	0.471	300
	Unmined	294	0.257	75	857	0.234	200
	Dilution/Losses	116	0.010	1	330	0.006	2
	PEA Plan	400	0.347	139	967	0.313	302
Gap	Resource	167	0.357	60	1,731	0.317	549
	Mined PEA Plan	106	0.423	45	928	0.395	366
	Unmined	61	0.241	15	803	0.228	183
	Dilution/Losses	77	0.066	5	471	0.089	42
	PEA Plan	184	0.272	50	1,399	0.292	409
CSD & 2201	Resource	301	0.229	69	813	0.335	272

		Indicated			Inferred		
		tons	grade	koz	tons	grade	koz
Total	Resource	1045	0.327	342	4,037	0.328	1,322
	Mined PEA Plan	390	0.469	183	1,564	0.426	667
	Unmined	655	0.243	159	2,468	0.266	655
	Dilution/Losses	194	0.032	6	802	0.055	44
	PEA Plan	584	0.324	189	2,366	0.300	711

17. Recovery Methods

17.1. Resource Processing

The premise for treating the material from the Helen and Gap resources is toll milling and treating by another mining company through either existing roasting and calcine cyanidation or existing pressure oxidation and residue cyanidation facilities.

Premier Gold solicited two items from a prospective toll operator with both roasting and pressure oxidation (POX) processes and their associated cyanidation processes for the respective calcines or POX residues.

The first item were the test protocols and test conditions for laboratory bench scale batch roasting and pressure oxidation tests were obtained from the prospective toll operator for the 2017 metallurgical testing described in Section 13 of this technical report. The conditions provided approximate the expected operating conditions in the prospective toll operators roasting and POX facilities.

The second item Premier Gold solicited was terms and conditions for toll milling and treating Helen Zone resource material. Premier Gold provided a package of Helen Zone metallurgical data, for the roasting and POX tests, from the 2017 test program, to the prospective toll process operator for their consideration and as the basis for toll processing resource material through either the toll operator's roasting or POX facilities.

The prospective toll process operator provided terms and conditions for processing the Helen Zone and Gap resource material through their existing operations. The key terms and specifications are summarized as follows:

17.1.1. Feed Specifications

Table 17-1 shows proposed feed specifications applicable to both roaster and pressure oxidation feed.

Table 17-1 Toll Processing Feed Specifications

Constituent	Maximum Acceptable Level	Unit of Measure
Mercury	25	ppm
Arsenic	1200	ppm
Lead	100	ppm
Zinc	200	ppm
Total Copper	0.25	%

Constituent	Maximum Acceptable Level	Unit of Measure
Cyanide Soluble Copper	250	ppm
Selenium	1	ppm
Barium	500	ppm
Chromium	100	ppm
Cobalt	100	ppm
Nickel	100	ppm
Cadmium	1	ppm
Free Gold	Any visible amount	

The ICP multi-element analyses and the mercury and arsenic analyses indicated the following parameters may limit or require blending at the Premier Gold Cove mine site in order to meet the proposed feed specifications:

- Mercury:
  - Helen - Only one Helen Comp (22) had Hg higher than 25 ppm;
  - Gap - No composite in Gap had Hg higher than proposed spec.
- Arsenic:
  - Helen – Two of the eleven composites (14 & 16) exceeded the proposed specification with two more at the maximum level. (5 and 21);
  - Gap - Eight of the ten composites exceed the proposed maximum.
- Lead:
  - Helen – None of the composite exceed the lead limit;
  - Gap – None of the composites exceed the lead limit.
- Zinc:
  - Helen – Only one composite (5) is at the zinc limit, all others are below he limit;
  - Gap – None of the composites exceed the zinc limit.
- Total Copper:
  - Helen – None of the composite exceed the copper limit;
  - Gap – None of the composites exceed the copper limit.
- Cyanide Soluble Copper:
  - Helen – None of the composite would exceed the cyanide soluble copper limit as the total copper is well below the soluble limit;
  - Gap – None of the composites exceed the cyanide soluble copper limit as the total copper is well below the soluble limit.
- Selenium
  - Helen – Assays were not performed to a 1 ppm limit so at present it cannot be determine if the composites meet the Se limit;

  - Gap – Assays were not performed to a 1 ppm limit so at present it cannot be determined of composites meet the Se limit.
- Barium:
  - Helen – Two of the Helen composites exceed the Barium limit (16 and 20);
  - Gap – Two of Gap composites exceed the Barium limit, (9 and 10).
- Chromium:
  - Helen – None of the Helen composites exceed the Chromium limit;
  - Gap – Five of the Gap composites exceed the Chromium limit, (11, 13, 16, 17, and 20).
- Cobalt:
  - Helen – None of the Helen composites exceed the Nickel limit;
  - Gap – None of the Gap composites exceed the Nickel limit.
- Nickel:
  - Helen – None of the Helen composites exceed the Cobalt limit;
  - Gap – None of the Gap composites exceed the Cobalt limit.
- Cadmium:
  - Helen - Assays were not performed to a 1 ppm limit so at present it is not possible to determine if the composites meet the Cd limit.
  - Gap – Assays were not performed to a 1 ppm limit so at present it is not possible to determine if the composites meet the Cd limit.

The proposed specifications for Roaster Feed are summarized with comments in regard to assays for the Helen and Gap resources as follows:

- CO_3– Two of the Helen composites were below this specification whereas 7 of the Gap composites were below the specification
- TCM - In the Helen ten of eleven composites exceed the TCM spec. Five CSD Composites exceeded this specification;
- Sulfide Sulfur – Seven Helen composites are below this specification. Two Gap composites were below this specification;
- Moisture between 3% and 7%

The proposed specifications for Pressure Oxidation Feed for either acid autoclaves or alkaline autoclaves are summarized with comments in regard to assays for the Helen and Gap resources as follows:

- **Acid Autoclave -** Generally the Helen Zone composites do not meet these specifications and would likely be roaster feed instead, conversely, most of the Gap composites meet this specification and would likely be POX feed.
- **Alkaline Autoclave -** Some of the Gap composites meet this specification and would be directed to the alkaline system**:**

The test data indicates that the Helen Zone composites were generally more amenable to roasting and calcine CIL cyanidation than POX and residue CIL cyanidation. The assay data for the Helen composites indicates that there may be some problems from some areas to meet roaster feed specifications. Onsite blending of Helen resource material to meet specifications prior to shipping to the toll processor provided that resource material is available for blending will likely be required.

Conversely, the Gap composite test data were generally more amenable to POX and residue CIL cyanidation. Again blending would likely have to be used prior shipping offsite to provide on specification material to the toll processor.

Even though that it appears that the Helen resource may generally be more amenable to roasting and calcine CIL cyanidation, it is likely that during mining of the resource for toll processing, that there will be areas that can be directed to POX and residue CIL cyanidation. The reverse would be likely for the Gap resource where areas within the resource could be directed to roasting instead of POX.

17.2.Projected Gold and Silver Recoveries Used for Metallurgical Zones

Roaster and pressure oxidation recoveries assuming CIL processing were projected based on the SGS composite testing for use in the Mineral Resource lens modelling. These are initial projections and further sampling, assaying, and testing will be needed to confirm the projections and increase the understanding of recoveries by roasting or pressure oxidation within the metallurgical zones. Note some projections were extrapolated for composites where the only direct cyanidation of calcines or POX residues was performed based on a function developed from the composite data where both types of cyanidations were performed. The testing showed that CIL cyanidation significantly increased metal extractions over direct cyanidation of calcines and POX residues. These relations need further investigation as the project progresses. Typical extractions are shown in Table 17-2.

17.3.Composite and Metallurgical Zones

The source of the composites tested by SGS were referenced to the Mineral Resource lenses in Table 17-2 by Practical Mining. These lenses were grouped into the metallurgical zones shown in

Figure 17-1 for selection of the preferred processing method and estimation of gold and silver recoveries (Table 17-3.

Table 17-2 Composites and Metallurgical Zones

Hole ID	From	To	Length	Composite ID	Met. Zone	Head Au (g/t)	Head Ag (g/t)	Projected Roaster CIL Recovery	Projected POX CIL Recovery
AX-12	1265	1293	28	HELEN5	G1	7.55	2.3	77.5	96.6
PG16-11	1816.5	1836	19.5	CSDGAP10	G1	17.7	4.4	72.9	95.0
PG16-12	1967.5	2002.5	35	CSDGAP15	G1	37	11.2	67.2	94.0
PG16-02	2004	2038	34	CSDGAP 2	G2	4.42	37.7	95	83.5
PG16-06	2204	2241.3	37.3	CSDGAP9	G2	15.4	7.5	68.7	72.2
PG16-11	2060	2079.7	19.7	CSDGAP11	G2	8.49	4.7	70.4	70.5
PG16-11	2079.7	2097	17.3	CSDGAP12	G2	5.58	3.3	74.9	74.9
PG16-11	2110	2127	17	CSDGAP13	G2	23.1	41.5	79.3	79.3
PG16-12	2057.5	2080	22.5	CSDGAP16	G2	34.8	12.4	81.0	88.5
PG16-12	2123	2145	22	CSDGAP17	G2	11.3	12.3	53.1	80.7
PG17-07	1423.8	1442.5	18.7	HELEN20	H1	4.44	2.1	93.2	
PG16-16	1830	1862	32	CSDGAP20	H1	7.15	3.1	93.4	85.8
AX-27	1709	1729.5	20.5	HELEN15	H2	5.96	1.7	75.0	
AX-27	1729.5	1753	23.5	HELEN16	H2	10.7	1.8	89.1	72.8
AX-27	1840	1862	22	HELEN17	H3	6.74	2.5	68.3	
AX-27	1870.5	1896.2	25.7	HELEN18	H3	11.9	5.1	69.8	
PG17-07	1952.5	1976	23.5	HELEN21	H3	9.33	2.1	81.6	63.0
AX-18	1876.5	1903.5	27	HELEN6	H4	8.47	2	72.4	67.5
PG17-07	2066	2084	18	HELEN14	H4	32.3	8.2	71.9	82.2
AX-27	1932	1978	46	HELEN19	H4	10.3	3.5	77.9	63.8
PG17-07	2132	2147	15	HELEN22	H5	7.6	3.5	91.5	62.6

Figure 17-1 Helen and Gap Metallurgy Zones

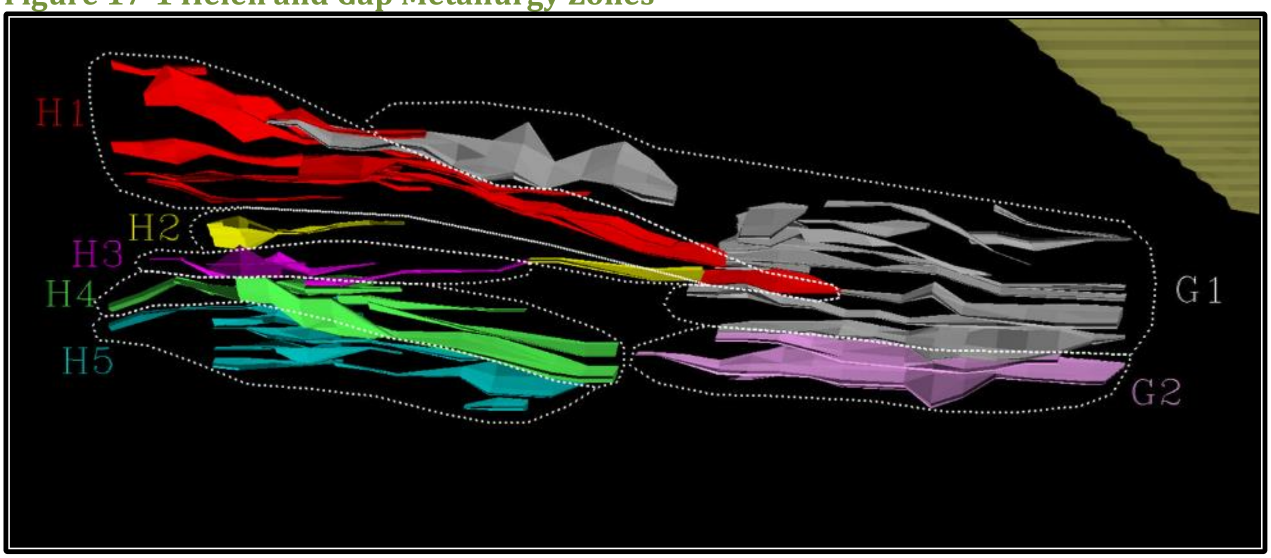


Table 17-3 Gold and Silver Payable Recoveries

Zone	Process	Au Recovery	Ag Recovery	As Penalty	Au Payable Recovery	Ag Payable Recovery
G1	POX	95.1	78	0.0	95	25.0
G2	POX	78.5	78	0.0	78.5	25.0
H1	Roast	93.3	30.7	1.1	92.2	10.0
H2	Roast	82.6	30.7	1.1	81.5	10.0
H3	Roast	73.2	30.7	1.1	72.1	10.0
H4	Roast	75.1	30.7	1.1	74.0	10.0
H5	Roast	91.5	30.4	1.1	90.4	10.0

17.3.1. Payable Content

The proposed toll processing terms from the prospective toll processor contained terms for determining recoverable metals by roasting and POX processes summarized. Payable metal content was generally based on feed head grades of gold and silver. Note that the proposed terms are based on the Helen Zone Data package only and is presumed to apply to toll processing Gap resource material also.

Additionally, the proposed terms state that at month's end for each period covered by a potential contract, the recoverable gold will be adjusted based on the toll processor's actual plant recoveries and proportions of toll resource to the processor's own materials processed. This could be a positive or negative adjustment to the recovery estimated per the recovery equation.

As with the proposed roaster terms the POX recovery will be adjusted at month's end for each period covered by a potential contract, the recoverable gold will be adjusted based on the toll processor's actual plant recoveries and proportions of toll resource to the processor's own materials processed. This could be a positive or negative adjustment to the recovery estimated per the recovery equation

The same end of month recovery adjustment also applies to the alkaline POX recovery.

The proposed terms indicate that the recoverable silver will be 10 to 20% and will be adjusted at months end in a similar manner as for gold. Silver recovery in roasting and POX operations is typically low. The 2017 test work indicates that the Helen and Gap resource material may yield higher silver extractions however the proposed terms will likely pay for lower amounts unless the toll processor's silver recovery are higher when processing the Helen or Gap resource materials.

17.4. Conclusions and Recommendations:

The following are the major conclusions and recommendations from the 2017 Helen and Gap composite metallurgical test program:

17.4.1. Conclusions:

1. The feed specifications appear to be somewhat rigid and could preclude some material being sent to the toll processor. Blending may allow shipment of some off-specification material provided appropriate material is available for onsite blending prior to shipping to the toll processor;
2. The terms appear to be consistent and typical with those encountered in the industry, and;
3. The recovery terms appear to be the result of analyzing the metallurgical data provided by Premier Gold.

17.4.2. Recommendations:

1. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
2. Additional metallurgical testing should be conducted to confirm the proposed payable recoveries are appropriate for the resources;
3. Development of a preliminary or conceptual onsite blending program is recommended to evaluate if on specification material can consistently be supplied to a toll processor;
4. The next phase metallurgical program should examine blending of out of specification resource materials to produce on specification material. The blending should be based on material projected to mined in a given period, for example, blending of material that is available in the first six months of operation should not be tested with material projected to only be available in year three of mining.

18. Project Infrastructure

18.1. Dewatering

18.1.1. History

Dewatering of the Cove Pit occurred from 1988 until mid-2001 utilizing surface dewatering wells, sumps, and horizontal drains. Water pumped from the dewatering wells was piped to a series of rapid infiltration basins (RIBs) located north of the pit, where the water was infiltrated into the alluvium of the Reese River Valley. All wells constructed for dewatering purposes have been abandoned in accordance with Nevada Division of Water Resources regulations as part of the mine's closure plan. Following cessation of dewatering activities, a pit lake began forming in 2001 and has reached an elevation of approximately 4,626 ft. (Piteau Associates USA Ltd., 2018)

The pit reached the ground water level in 1991. The pumping rate peaked at 19.000 gpm in 1994 and 1995. By the year 2000, the last full year of mining it had declined to 13,400 gpm. The infrastructure required to move this volume of water included 23 pumping wells and two in pit pumping stations. (Echo Bay Minerals Company, 2002)

18.1.2. Pump Test PW17-101

The plan for drilling of PW17-101 was to drill and complete a test well at 14-inch diameter to a depth of 2,000 ft below surface. This would allow testing and analysis of the entire saturated section of the resource that was planned for mining and which would require dewatering. As drilling advanced, repeated sloughing and lost circulation events slowed progress. Following several remediation attempts the holewas completed to a TD of 1,465 ft and testing initiated.

Pumping tests of PW17-101 consisted of an initial four stage step test followed by a 30-day constant rate test. Groundwater levels were monitored at locations across the site via VWPs set in exploration core holes and HE holes in the resource area.

Details of the constant rate test are as follows:

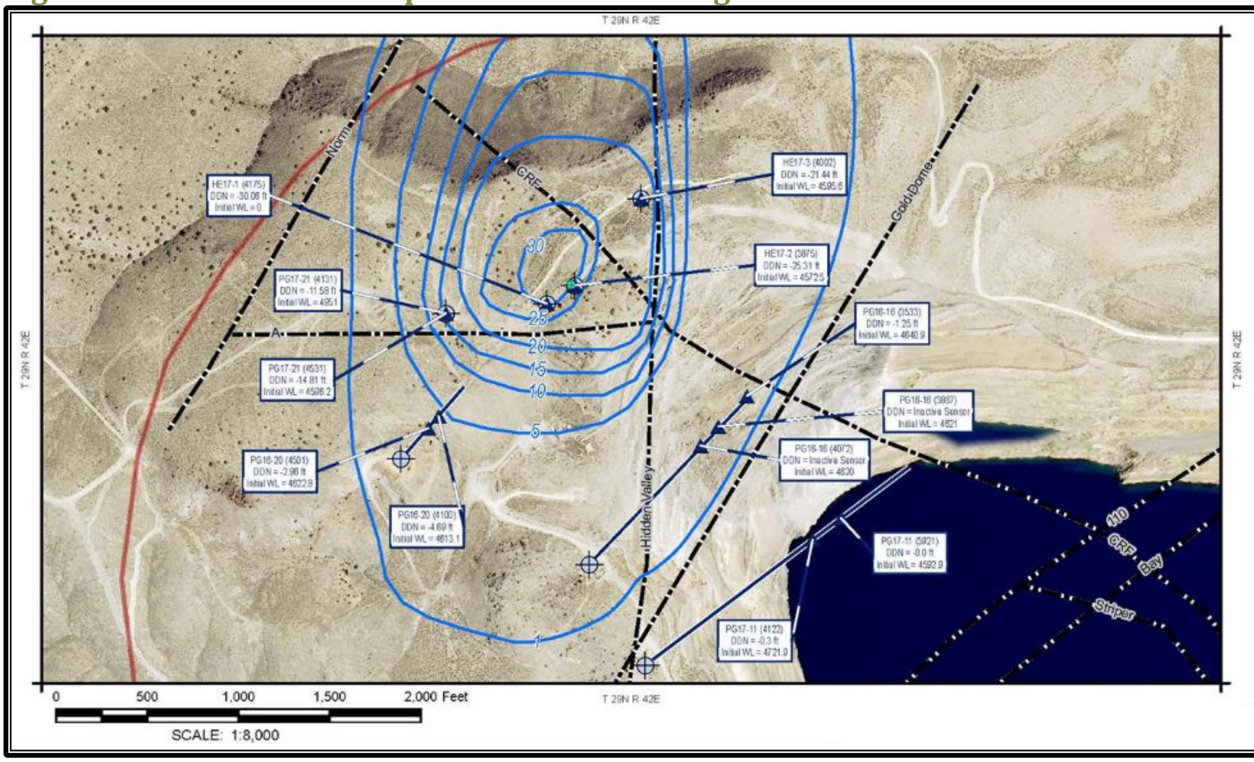
- Constant rate testing was conducted at 220 gpm over the 30-day testing period;
- Initial depth to water was measured at 959 ft below ground surface (bgs) or 4611 ft amsl prior to testing;
- Discharge head pressures ranged between 27 and 38 psi, and;
- Variable frequency drive was held steady at 47 htz for the duration of the test.

Groundwater monitoring data during the test is shown im Table 18-1 and Figure 18-1. (Piteau Associates USA Ltd., 2018)

Table 18-1 Groundwater Monitoring during the 30-Day Constant Rate Test

VWP monitoring points	Location with respect to PW17-101 and sensor elevation (amsl)	Maximum Observed Draw down (ft)
HE17-01	200 ft SW of the pumping well at 4131 ft amsl	30.1
HE17-02	23 ft NE of the PW17-101 at 3875 ft amsl	25.3
HE17-03	750 ft NW, North of CRF Structure at 4002 ft amsl	21.4
PG16-16	1,000 ft SE, east of the Hidden Valley structure at 3533 ft amsl	1.3
PG16-20, Shallow	1,200 ft SW south of the "A" structure at 4501 ft amsl	3.0
PG16-20, Deep	1,200 ft SW south of the "A" structure at 4100 ft amsl	4.7
PG17-11, Shallow	2,000 ft SE east of the Gold Dome Structure at 4122 ft amsl	0.3
PG17-11, Deep	2,000 ft SE east of the Gold Dome Structure at 3921 ft amsl	0.0
PG17-21, Shallow	750 ft west at 4531 ft amsl at 4531 ft amsl	14.8
PG17-21, Deep	750 ft west at 4131 ft amsl at 4100 ft amsl	11.6

Figure 18-1 Draw Down Isopleths and Monitoring Locations



18.1.3. Analytical Dewatering Estimate

Future dewatering estimates utilized input parameters obtained via the 30-day pumping test, mine planning, and from the observed and reported rock mass characteristics. These parameters are:

- Transmissivity: A transmissivity value of 750 ft2/d was selected for the analysis from an average of HE holes. Transmissivity values from these locations are believed to represent the hydrologic block's bulk transmissivity. Sensitivity dewatering estimates use the maximum (897 ft2/d) and minimum transmissivity values (631 ft2/d);
- Transmissivity values for the Gap deposit were assumed to be 3000 ft2/d (K=2.5 ft/d), rather than using calculated values from PG16-16. This was done because conductivity values at PG16-16 are believed to be overestimated considering that the analysis doesn't account for the hydraulic boundary effect of the Gold Dome fault between the pumping well and PG16-16;
- Radius: Effective radii of 492 ft and 629 ft were calculated from the area of the Helen to the 4100 and Helen's footprint. These footprints were used to simulate the draw down from the Theis equation. The Gap radius was estimated to be 550 ft;
- Storage: A storage value of 0.01 was used to reflect a rock mass with 1% drainable porosity. Rock mass storativity was considered negligible relative to drainable porosity.

A Theis analysis was completed to estimate the future pumping required to dewater the underground resources and meet the development advance shown in Figure 18-2 and the results are presented in Table 18-2.

Figure 18-2 Dewatering and Development Timing

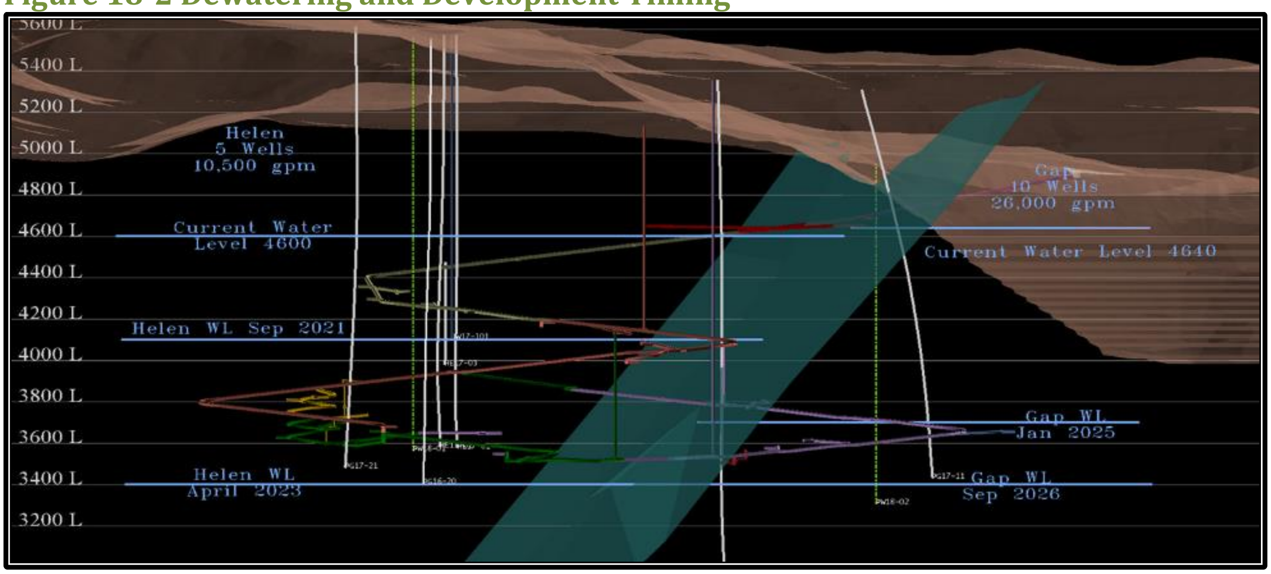


Table 18-2 Dewatering Summery

	Helen	Gap
Current Water Elevation	4600	4640
First Ore Elevation	4300	3800

Time to Dewater First Ore	4 months	8 months
Lowest Ore Elevation	3500	3400
Time to Dewater Lowest Ore	22 months	18 months
Number of Wells	5	10
Mean Pumping Rate	10,500 gpm	26,000 gpm
Minimum Rate	9,200 gpm	12,300 gpm
Maximum Rate	17,000 gpm	35,000 gpm

18.1.4. Rapid Infiltration Basins

Figure 18-3 Rapid Infiltration Basin During 2017 Pump Test

The RIBs should be located and designed to infiltrate water into the alluvial sediments of the Reese River Valley and located in a manner that will minimize re-circulation to the Cove Pit lake. Infiltration of dewatering water to a series of Rapid Infiltration Basins has been used at the McCoy Cove site in the past to re-introduce dewatering discharge into the groundwater system.

Over the past decade regulatory action has lowered the NDEP Profile I reference values for Arsenic (As) from 0.05 mg/l to 0.01 mg/l, making permitting of new RIBs more complex. Since the concentrations of As and Iron (Fe) were found to be above NDEP Profile I reference values in discharge water produced from PW17-101, some additional work will be needed to obtain approval for disposal of dewatering discharge via new Rapid Infiltration Basins (RIB) at Cove Helen. Subsequently, addressing elevated As and Fe in waters planned for infiltration will require an attenuation study aimed at demonstrating the ability of native soils to remove As and Fe. A study of the attenuation capacity of native soils at the RIB site should be undertaken to evaluate the ability of local soils to remove As and Fe as water is infiltrated to the alluvial soils of the Lower Reese River Valley. (Piteau Associates USA Ltd., 2018)

18.1.5. Recommendations

Further hydrogeologic characterization of the Cove Helen resource should focus on three areas where additional work is needed to advance permitting and development of the project. These areas are:

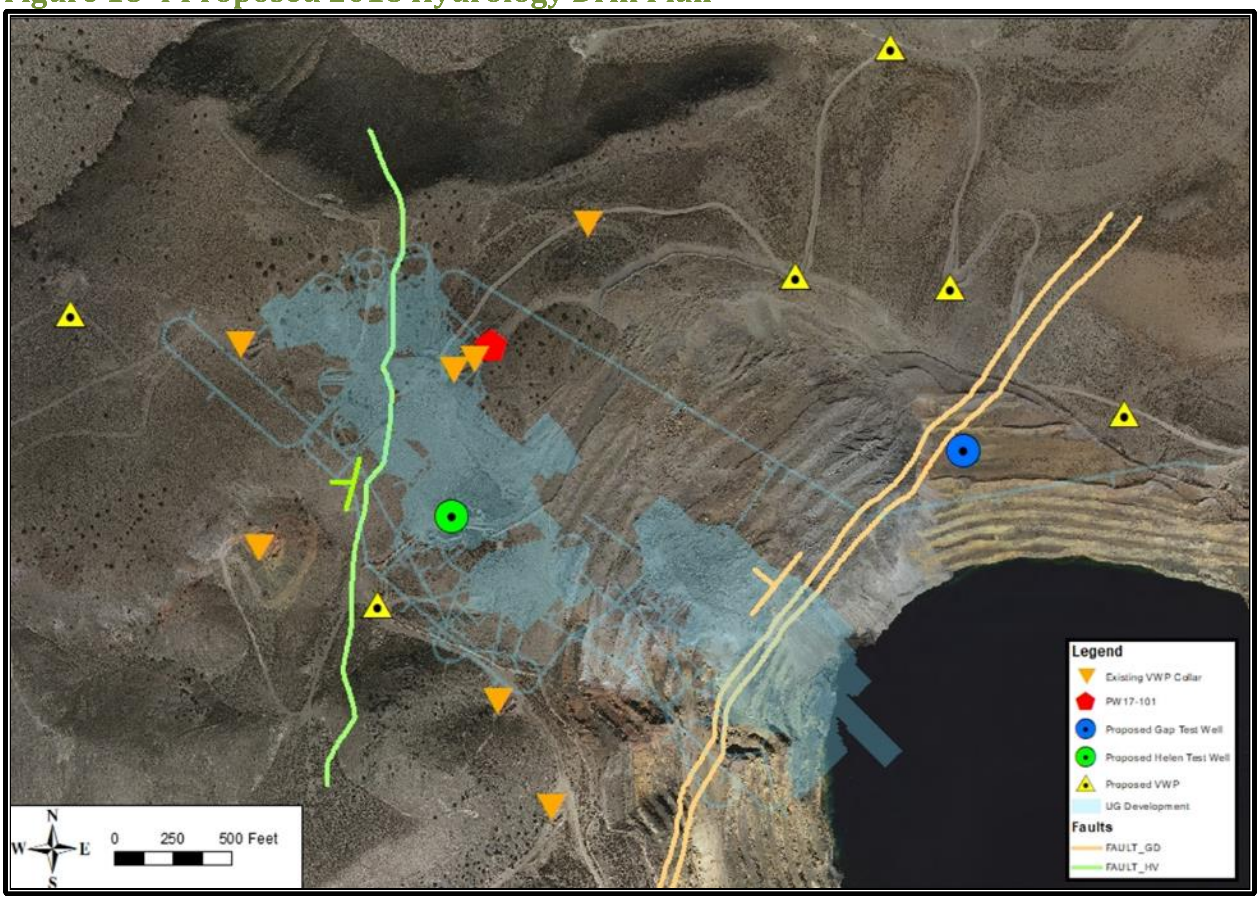
- Hydrogeologic characterization. Additional analysis is needed to fully characterize the Helen and Gap Zones in preparation for NEPA permitting. Characterization activities include the installation of monitoring wells to evaluate groundwater quality in the resource area, monitoring of seeps and perennial streams of Fish Creek and Dais Creek in the project area as well as VWPs in select locations to support baseline data development. Drilling and testing of a test well in the deep Helen is needed to define draw down response outside the structural wedge containing the Helen mineralization. Similarly, another test well is needed in the Gap Zone to evaluate the existence of suspected groundwater flow barriers and connectivity with the Cove Pit Lake;
- Operational support. In 2018 Premier plans to develop a decline that will allow additional drilling and evaluation of the Gap and Helens. The decline is planned to stop above the water table and continue on grade to provide access to a series of bays needed for additional exploration drilling. Opportunistic VWP installations in specific exploration core holes represents a significant savings in drilling cost for sensor installation. Groundwater data in this area of the resource is needed to support the development of the baseline resource assessment studies supporting permit approval;
- Permitting support, baseline studies and numerical model development. NEPA and other regulatory permitting actions planned for Cove Helen in 2018 will require the development of supporting hydrogeological baseline studies. Baseline resource assessment studies of this type require the characterization of surface and groundwater across the project area and will include a field program for the installation of monitoring wells both up and down gradient of the resource;
- Further development of the numerical model will be needed to assess potential impacts of dewatering in support regulatory approval for mining of the resource. Data collected

during the hydrogeologic characterization and operational support tasks will be integrated into the numerical model along with other groundwater monitoring data as it becomes available;

- NDEP permitting of infiltration an attenuation capacity study will be needed to support approval for RIB development. The study will use test pits, shallow drill holes, column testing and groundwater quality evaluation in the Reese River alluvium to demonstrate the potential impact resulting from infiltration of dewatering discharge in terms of degradation of waters of the state.

The proposed work plan to achieve these goals in shown in Figure 18-3. (Piteau Associates USA Ltd., 2018)

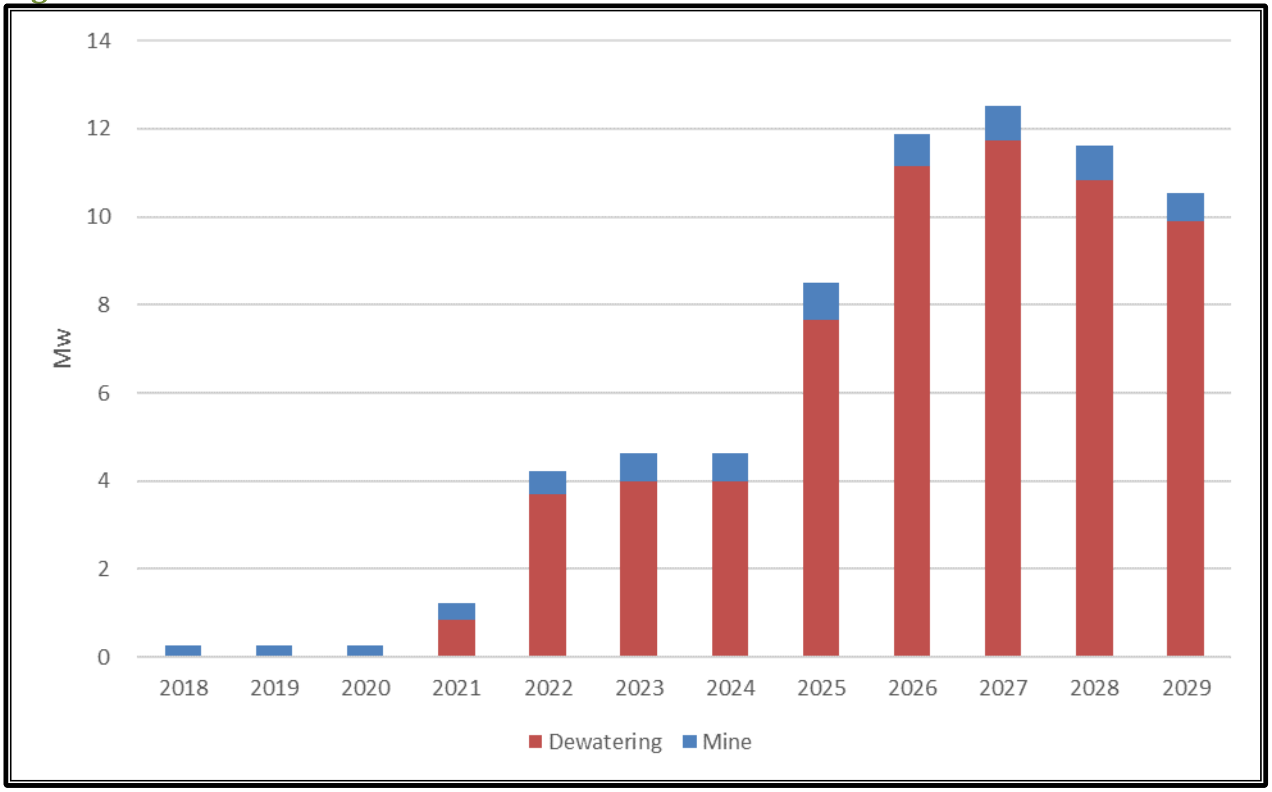
Figure 18-4 Proposed 2018 Hydrology Drill Plan



18.2. Electrical Power

Dewatering constitutes 90% of electrical power demand over the Project's duration. Demand for dewatering was estimated from projected water elevations and pumping rates and peak demand of 12.5 megawatt (Mw) occurs in 2027 (Figure 18-5).

Figure 18-5 Electrical Demand



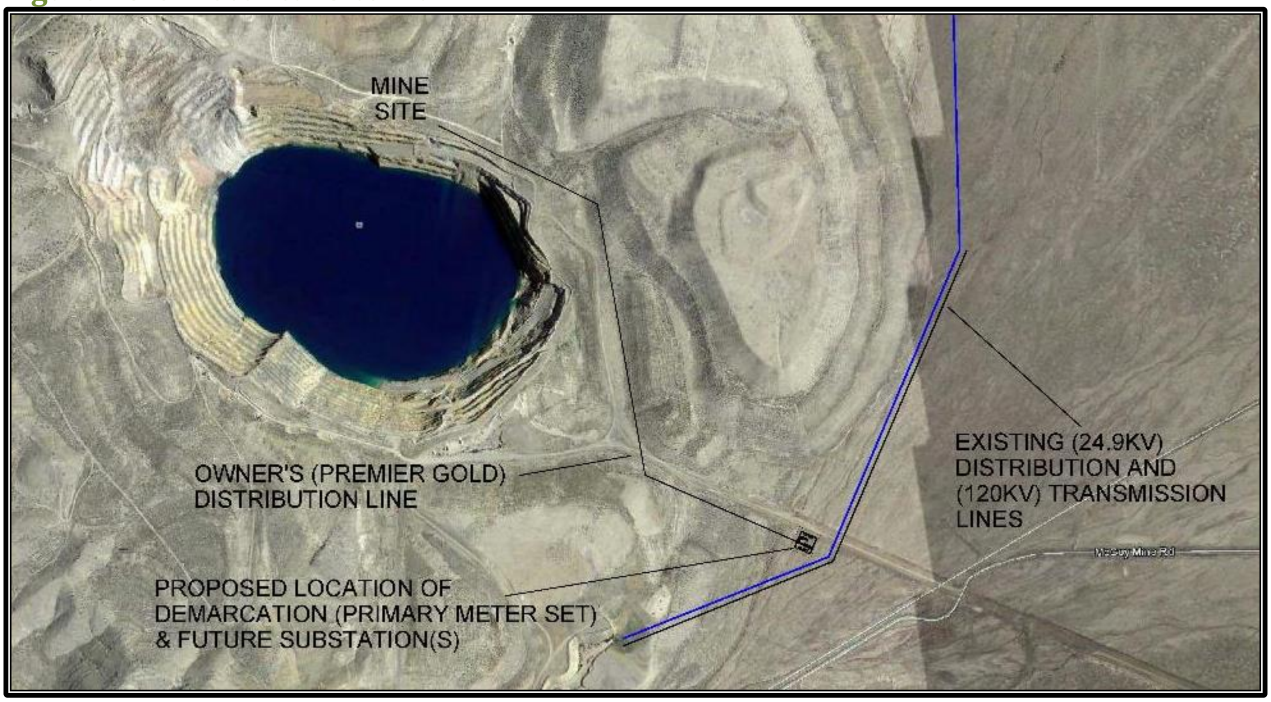
An existing NV Energy 24.9 kilovolt (kV) distribution line and meter will provide one (Mw) to the Cove Project during the initial decline development and underground drilling program. Permanent power for the project will be supplied by an existing 120kV transmission line. This line previously powered the Cove Project and extends approximately 9 ½ miles from NV Energy's Bannock substation to and terminates at the Cove Project. The line is in good condition and will not require any repairs.

The Bannock substation serves the Phoenix Mine and a geothermal power plant located in Jersey Valley. The substation has ample capacity to provide the estimated 12Mw of power required by the Cove Project. Prior to reconnecting the line to the grid NV Energy requires updating the switchgear at the substation to a ring configuration as a result of new standards implemented since the line was taken out of service after the cessation of activities at Cove by Echo Bay. The full cost of these upgrades will be borne by the Cove Project.

Where the lines cross the project access road a new substation will be constructed. Initially it will contain a 24/9/13.8kv 1,500 kilovolt-ampere pad mounted transformer and related equipment. Approximately 7,500 feet of distribution line will connect the substation to the portal site and related surface facilities (Figure 18-6).

The substation will be upgraded with a 120/13.8kv transformer when permanent power is being connected that will feed the distribution line to the portal. As the dewatering wells are completed additional distribution lines will be added to connect the wells.

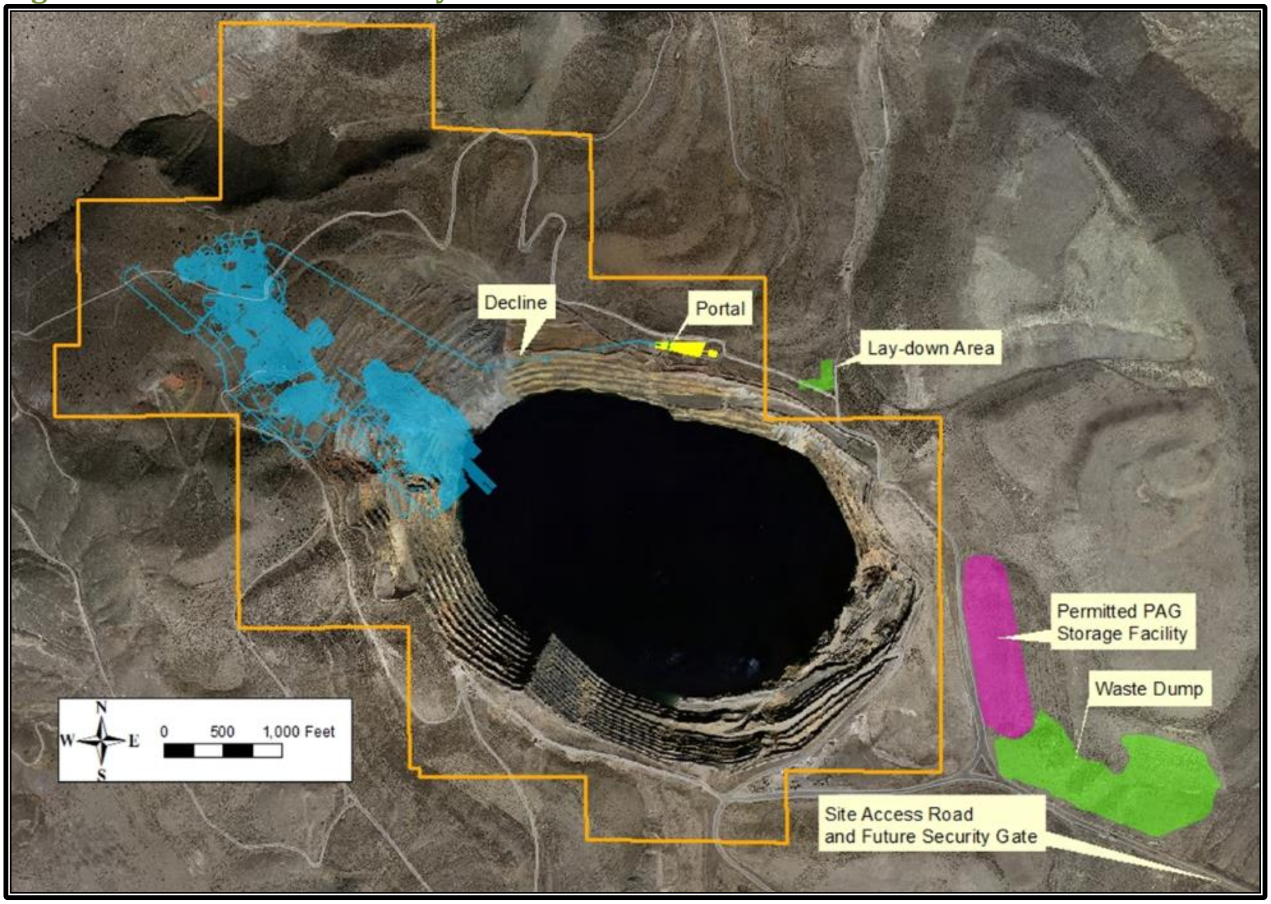
Figure 18-6 Electrical Site Plan



18.3. Mine Facilities

The proposed location of mine facilities is shown in Figure 18-7. The laydown area will contain the mine office, maintenance shop, equipment wash down bay, fuel and oil storage, employee dry facilities and warehouse.

Figure 18-7 Mine Facilities Layout



18.4.Backfill

Backfill material for unconsolidated waste fill (GOB) can be obtained from any suitable source such as development waste, open pit waste dumps, or leach pads.

Backfill material for Cemented Rock Fill (CRF) will need to meet specifications designed to achieve minimum Uniaxial Compressive Strength (UCS) specifications. This specification is designed to provide the pillar strength needed to maintain stability of adjacent underground excavations and may require screening and/or crushing. The results of backfill testing for six types of material available at Cove are shown in Table 18-3. CRF material will be mixed at a backfill plant located near the portal and transported underground using the same truck fleet used to remove mineralized material and waste from the mine.

Table 18-3 Backfill Scoping Tests 28-Day Unconfined Compressive Strength

Cement Content	4%	6%	8%
Aggregate Source			
Waste	440	510	830

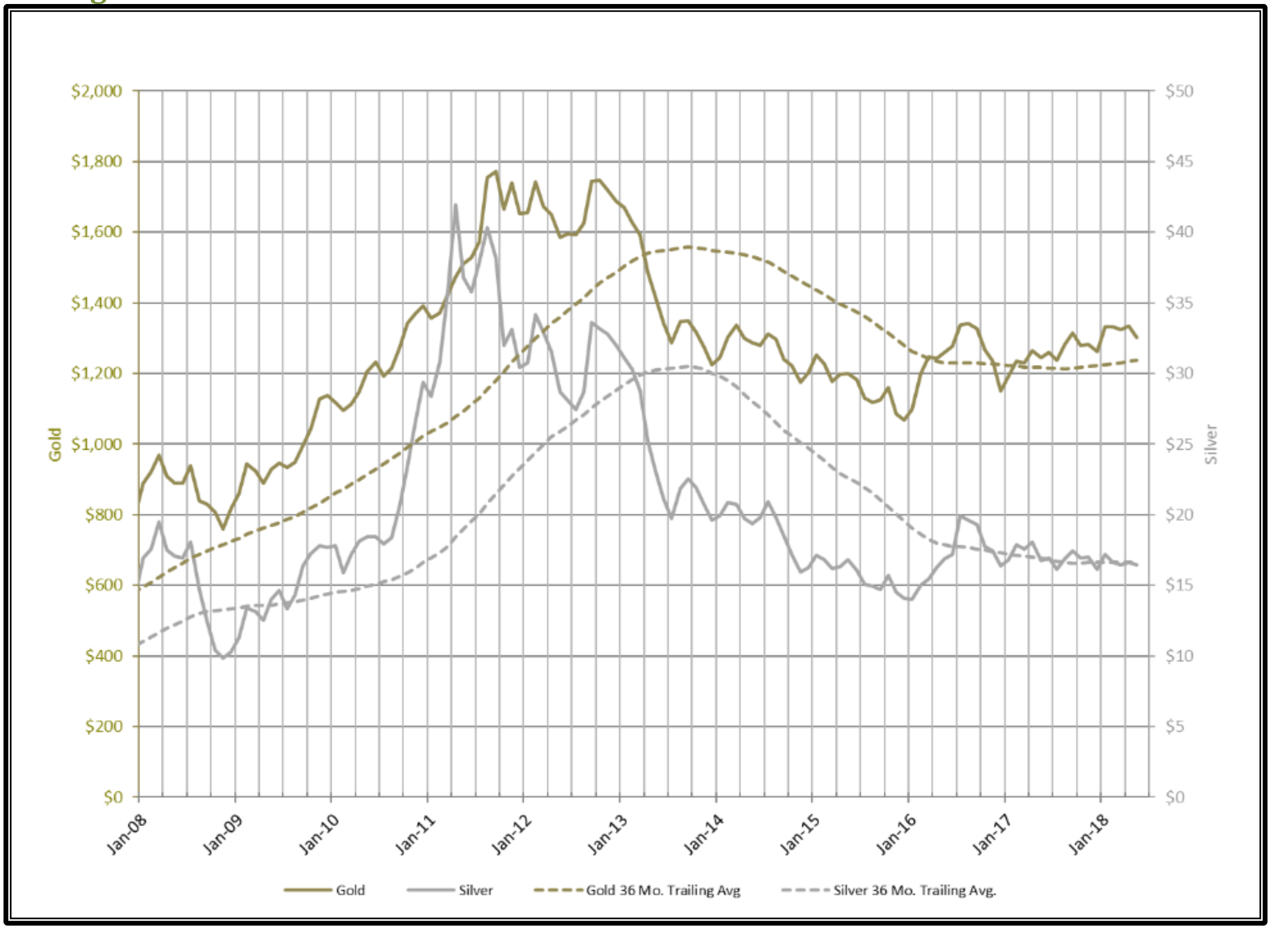
Cement Content	4%	6%	8%
Aggregate Source			
Tails	60	90	120
Tuff	90	140	240
Pad 3	360	590	500
Pad 2	190	200	260
Mill Rejects	210	510	810

19. Market Studies and Contracts

19.1. Precious Metal Markets

Gold and silver markets are mature with reputable smelters and refiners located throughout the world. Following several years of increases, gold and silver prices declined from 2012 through 2015 but have been increasing since. As of April 2018, the 36-month trailing average gold price was $1,231 per ounce while the average price during March 2018 was $1,325 per ounce. The silver price trend shows similar behavior with the 36-month trailing average of $16.62. Historical prices for both are shown in Figure 19-1.

Figure 19-1 Historical Monthly Average Gold and Silver Prices and 36 Month Trailing Average



19.2.Contracts

Premier's contracts with Newmont and Barrick were discussed in Section 4. From time to time the company enters into other contracts for goods and services as a routine course of business.

19.3.Project Financing

Project financing arrangements will be determined during the feasibility study and financing costs have not been included in this evaluation.

20. Environmental Studies, Permitting and Social or Community Impact

Au-Reka Gold Corporation (AGC), a wholly owned subsidiary of Premier Gold Mines Limited, is the designated operator on all Cove Project permits. AGC currently conducts mineral exploration activities in compliance with all applicable environmental protection legislation. The Cove Project is primarily located on public lands administered by the Bureau of Land Management (BLM) and subject to both Federal and State permitting requirements. AGC is unaware of any existing environmental issues or compliance problems that have the potential to impede production at the Cove Project. AGC is working closely with both State and Federal regulators to ensure that the permitting and compliance strategies are acceptable and will not cause delays in production or mine development. At this time, there are no community or social impact issues regarding work being completed at the Project and AGC has been coordinating with local stakeholders.

The Cove Project site is located within a previously mined area and most activities are currently being conducted or are planned on existing previously disturbed or mined areas, thereby limiting the potential environmental impacts to the site. All necessary studies and permits are in place to support the permitted exploration and test mining activities at the site. Some supplemental studies are being conducted to update permits to optimize operations under the existing authorizations. Additional studies will be conducted to support the full-scale mining operations for the Cove Project to supplement the existing studies as required by Federal and State regulatory agencies.

20.1. Environmental Studies and Issues

The following environmental studies are underway or planned:

- Hydrology including pumping rates, well locations, water quality determination, water treatment requirements and infiltration basin design;
- Water rights acquisition;
- Waste Rock Characterization;
- Air quality modelling;
- Updated biological surveys, and;
- Updated cultural resource inventories.

There are no known Greater Sage Grouse leks near the project.

20.2. Social or Community Impacts

Premier is committed to involving local ranchers and Tribal officials in the progress of activities and potential impacts from the Cove Project. Opposition to date is limited to a water rights protest on the part of Pershing County farmers. The company has submitted evidence of no impact to the Humboldt River and inter basin transfers and anticipates the protest will be dismissed.

20.1.Permitting

Permitting of the project is currently planned in three phases, pending studies and coordination with the regulators. These phases are structured to provide early cash flow while maintaining permitting flexibility. The first phase will encompass portal construction and initial underground development, underground delineation and exploration drilling, hydrological testing and baseline data collection. Premier has submitted a Plan of Operations/Reclamation Permit Amendment and an Engineering Design Change (EDC) to optimize construction and operations under the existing design and authorizations, which include relocating the underground portal opening to a more stable location outside of the Cove Pit, modifying the design of the waste rock disposal facility to accommodate more waste material and optimize water management from the facility, and rerouting the distribution powerline at the site on a more efficient route along an existing access road to limit disturbance. This modification request also included a RCE update to bring online in the bond for the Project all of the remaining surface support facilities and additional surface exploration acreage, which would bring the RCE and bond total to approximately $5.1M. The permit modifications are currently under review at the BLM and State and approval, including an update of the bond, is expected late in the second quarter 2018 to allow construction to start in August 2018. Table 20-1 lists the major permits currently in place.

Table 20-1 Cove Project Existing Permits

Permit Name	Number	Agency	Description
Plan of Operations	NVN-088795	BLM	Plan of Operations is required for all mining and processing activities and exploration exceeding 5 acres of surface disturbance on public lands managed by the BLM. The BLM approves the plan and determines the required environmental studies, usually an Environmental Assessment (EA) or an Environmental Impact Statement (EIS) based on the requirements outlined in the National Environmental Policy Act (NEPA).
National Environmental Policy Act - Environmental Assessment (EA), Decision Record (DR) Findings of No Significance (FONSI);)	EA#DOI-BLM-NV-B010-2011-0040-EA	BLM	A Decision Record (DR) and Findings of No Significance (FONSI) are issued when an EA document is accepted demonstrating no significant impacts to the environment based on Project design and environmental protection measures committed by the proponent. The Cove Project currently is operating under a DR/FONSI for test mining issued following an EA. A Record of Decision (ROD) in the United States is the formal decision on an EIS document that the BLM issues to disclose potential impacts to the environment with applicable mitigation measures to prevent undue and unnecessary degradation to public lands. It is assumed an EIS and ROD will be required for full-production mining.

Water Pollution Control Permit (Facilities)	NEV2010 102.01	NDEP, BMRR - Regulation Branch	Mines operating in the State of Nevada are required to have a Water Pollution Control Permit (WPCP) to ensure protection of waters of the State during mining activities. The current permit is a Small Mine Permit authorizing the extraction of 120,000 tons of ore over the life of the Project. The permit can be modified to remove the ore tonnage cap and other facility design changes as the Project moves forward.
Water Pollution Control Permit (Rapid Infiltration Basins)	NEV2010 107	NDEP, BMRR - Regulation Branch	Water Pollution Control Permit (WPCP) for infiltration of water from the underground mine operations into Rapid Infiltration Basins (RIBs). The current discharge rate allowed under this permit is 2,500 gallons per minute, but this permit can be modified with additional studies to increase the discharge rate as needed. Contingency RIBs are conceptually included in this permit to facilitate a quicker permit modification process should additional discharge be needed to accommodate mining.
Water Rights	80341/803 42	NDWR	Water rights are issued by the Nevada Division of Water Resources and State Engineer based on Nevada water law which allocated rights based on appropriation and beneficial use within the water basin. Prior appropriation (also known as "first in time, first in right") allows for the orderly use of the state's water resources by granting priority to parties with senior water rights. This concept ensures the senior uses are protected, even as new uses for water are allocated. Mining water rights are considered Temporary in nature. The current water rights for the Cove Project cover the 2,500 gpm dewatering and additional water for dust control and operations from the Cove Pit Lake. An application has been submitted to the State Engineer and is under review to acquire additional water rights for the project
Nevada Reclamation Permit	#0342	NDEP, BMRR - Reclamation Branch	The BMRR Reclamation Branch works in coordination with the BLM for projects on public land to establish reclamation guidelines and a reclamation cost estimate to support project bonding. This permit and associated bond ensures land disturbed by mining activities are reclaimed to safe and stable conditions to promote safe and stable post-mining land use. A permit is required for any disturbance over 5 acres. The reclamation cost estimate (RCE) is financially secured with a posted security. The posted surety amount provides assurance that reclamation will be pursuant to the approved reclamation plan in the event that the State has to perform reclamation or is held until reclamation has been successfully conducted.
Air Quality Operation Permit	AP1041-2774	NDEP, BAPC	An owner or operator of any proposed stationary source must submit an application for and obtain an appropriate operating permit before commencing construction or operation. Class II Air Permit - Typically for facilities that emit less than 100 tons per year for any one regulated pollutant and emit less than 25 tons per year of total Hazardous Air Pollutants (HAP's) and emit less than 10 tons per year of any one HAP. The current air quality operations permit for the Project covers emissions from back-up generators at the site.
Air Quality Surface Area	AP1041-2192.02	NDEP, BAPC	A Surface Area Disturbance Permit (SAD) is required for any project that disturbs more than 25 acres of ground. Annual updates show what areas have been disturbed.

Disturbance Permit			
Industrial Artificial Pond Permit	S-407174	Nevada Division of Wildlife	The NDOW oversees wildlife management of artificial ponds at mine sites. The ponds are required to have wildlife protection design standards and quarterly mortality reports are submitted to document any deceased wildlife discovered in the ponds.
Storm Water Control Permit	NVR 3000000	NDEP Bureau of Water Pollution Control	Storm water runoff from waste rock piles, haul roads, milling facilities and other mine areas that have not mixed with process solutions or other contaminant sources. Typical pollutants include suspended and dissolved solids and minerals eroded from exposed surfaces.

Phase two will obtain all operating permits necessary for full scale mining of the Helen including necessary infrastructure and facilities. It is anticipated this will require a new Environmental Assessment (EA) as many of the permitted facilities and operations will be used; however, the BLM will ultimately determine the level of NEPA required. Under an EA, Phase II of permitting is expected to be completed by Q1 2021.

The third and final phase is assumed to require an Environmental Impact Statement (EIS) to obtain all necessary permits required for mining the Gap deposit. The EIS will be necessary due to the anticipated scope of the dewatering effort required and potential impacts to the Cove Pit lake. Engineering and baseline data collection to support the EIS will occur concurrently with phase two.

The U.S. Department of the Interior recently released guidance documents for streamlining the NEPA process, including expedited timeframes and limiting the number of pages in NEPA documents. The timeframes discussed above include the upfront baseline studies and engineering required, prior to permit submittal and NEPA.

20.2.Closure and Reclamation Requirements

Premiers’s last amendment to the Reclamation Cost Estimate (RCE) included construction of Rapid Infiltration Basins (RIBs) and a test well for dewatering discharge, in addition to the previously bonded exploration disturbance, existing site infrastructure, and some roads and buildings. The total of the RCE is calculated using the State of Nevada’s Standard Reclamation Cost Estimator (SRCE), which is adjusted for inflation. The SRCE was developed in a cooperative effort between the NDEP, Bureau of Mining Regulation and Reclamation, BLM, and the Nevada Mining Association to facilitate accuracy, completeness, and consistency in the calculation of costs for mine site reclamation. AGC is required to update the total RCE for the Cove Project every three years or as necessary to bring online phased project disturbance and infrastructure. The next

RCE update is scheduled for 2018; however, the recent RCE and bond update submittal may satisfy this requirement.

RCE costs for reclamation currently include the following categories: roads; exploration roads and drill pads; RIBs; ponds; electrical infrastructure; buildings and equipment; portal and vent raise plugging; re-vegetation; and contractor management. The most current RCE was approved by BLM and NDEP in March 2017 in the amount of $2,442,461.

21. Capital and Operating Costs

21.1. Capital Costs

Costs were generated from estimates provided by local suppliers and contractors and from similar work performed at other area mines. All cost estimates include Lander County and Nevada sales taxes of 7.1%, freight, contractor mobilization and demonization, engineering procurement and construction management. Capital cost estimates for the project are summarized in Table 21-1 and detailed in Table 21-2 through Table 21-6.

Table 21-1 Project Capital Costs ($M)

Category	Pre-Development			Construction		Sustaining					Total
	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	
Mine Development	1.9	4.0		6.6	12.1	8.8	4.2	9.3	0.5	6.2	53.6
Dewatering	4.2			6.7	5.4	1.4	11.2	12.5	4.0	2.3	47.7
Facilities and Administration	3.9	2.3	2.2	7.1	1.9	1.4	0.5	0.2	0.1	0.2	19.8
Electrical Service and Power Line	1.0			3.1							4.1
Delineation Drilling		1.1	3.3								4.4
Contingency[1]	1.3	0.3	0.3	2.5	1.1	0.4	1.8	1.9	0.6	0.4	10.6
Total	12.3	7.7	5.8	26.0	20.5	12.0	17.7	23.9	5.2	9.1	140.2
	25.8			46.5		67.9					

1. *15% Contingency added to Dewatering, Facilities, and Electrical Service and Power Line.*

Th mine development unit costs shown in Table 21-2 are typical contractor costs in northern Nevada. These combined with the mine development schedule presented in Section 16 yield the development capital shown in Table 21-3

Table 21-2 Mine Development Unit Costs

Description	$/ft
Primary Drifting (15 ft x 17 ft)	$1,600
Secondary Horizontal Access (15 ft x 15 ft)	$1,350
Raise Bore (10 ft dia.)	$2,000

Table 21-3 Mine Development Capital ($M)

	Pre-Development			Construction		Sustaining					
	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Total
Helen											
Primary Drifting	1.8	1.6		5.8	9.1	6.5	3.0	2.2			29.9
Secondary Drifting	0.2	2.9		0.6	1.3	1.4	0.9	0.2			7.5
Raising					1.5	0.9	0.4	4.2			2.8
Gap											
Primary Drifting								5.5	0.4	2.2	8.1
Secondary Drifting								1.1	0.1	0.6	2.1
Raising								0.4		2.8	3.2
Total	1.9	4.0		6.6	12.1	8.8	4.2	9.3	0.5	9.2	53.6

Dewatering capital includes 5 pumping wells in the Helen and ten in the Gap. Well drilling and completion costs are approximately $2M per well. Costs include drilling, completion, and pumping equipment. Dewatering capital costs are listed in Table 21-4.

Table 21-4 Dewatering Capital ($M)

	Pre-Development			Construction		Sustaining					
	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Total
Helen											
Wells	2.1			4.0	4.0						10.0
Ribs and Pipe Line				2.3	1.4	1.4	1.3				6.4
Electrical				0.4							0.4
Gap											
Wells	2.1						9.9	7.9			19.9
Ribs and Pipe Line								4.3	3.8	2.3	10.3
Electrical								0.3	0.2		0.5
Dewatering Total	4.2			6.7	5.4	1.4	11.2	12.5	4.0	2.3	47.5

(Piteau 2018)

Table 21-5 Facilities and Site General ($M)

	Pre-Development			Construction		Sustaining					
	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Total
Environmental & Permitting	1.0	0.5	0.5	0.5	1.0	0.5	0.5				4.5
Metallurgical Testing and Feasibility Study	0.3	0.3	0.3								0.9
Portal	0.3										0.3
WRDA				2.6							2.6

	Pre-Development			Construction		Sustaining					
	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	Total
Shop, Office & Dry	1.0			1.0							2.0
Backfill Plant				1.0							1.0
Escape Hoist					0.5	0.5					1.0
Fans and Load Centers				0.3	0.4	0.3		0.2	0.1	0.2	1.5
Property Holding Costs	0.2	0.3	0.2	0.3							1.0
Electrical Power	0.1	0.2	0.2	0.7							1.2
Administration & Management	1.0	1.0	1.0	1.0							4.0
Total	3.9	2.3	2.2	7.4	1.9	1.3	0.5	0.2	0.1	0.2	20.0

Table 21-6 Electrical Transmission, Substation and Distribution ($M)

	Pre-Development			Construction	Total
	2018	2019	2020	2021	Total
25.9/13.8 kV Substation	0.4				0.4
Distribution Line to Portal	0.3				0.3
Portal Switchgear and Transformers	0.2				0.2
120kV/13.8kV Sub				0.8	0.8
Bannock Sub Upgrades				2.3	2.3
Total	1.0			3.1	4.1

(Quantum 2017)

21.2. Closure and Reclamation

Reclamation bonding requirements are estimated at $20M each for the Helen and Gap zones. Regulatory bonding requirements will be satisfied by the purchase of surety for an annual cost of 2% per year. Estimated reclamation costs net of salvage total $5.0M. Post closure monitoring is forecast to continue for 10 years following final reclamation at a cost of $0.4M per annum. The monitoring costs discounted at 8% per year equate to $2.7M and are charged to the cash flow in the first year following final reclamation. Closure and reclamation costs on a per unit basis total $17.29 per gold ounce.

Table 21-7 Closure and Reclamation Costs ($M)

	2018 - 2020	2121- 2024	2025 - 2028	2029	2030	2031	Total
Reclamation Bonding	0.1	0.4	0.8				5.1
Reclamation				2.5	2.5		5.0
Closure and Monitoring						2.7	2.7 [1]
Total	0.1	0.5	0.8	2.5	2.8	2.7	12.8

1. *Closure and Monitoring-10 years x $0.4M per year @8% = $2.7M in 2031*

21.3.Operating Costs

The unit mining costs presented in Table 21-8 are typical contractor costs for the anticipated conditions at Cove.

Table 21-8 Unit Operating Costs

Item	Unit Cost	Units
Stope Development	$75	$/ ton
Production	$55	$ /ton
Cemented Backfill	$30	$ /fill ton
Gob Fill	$10	$ /fill ton
Expensed Waste	$75	$ /waste ton
Ore Hauling	$0.21	$ /ton-mile
Toll Roasting	$45	$ /ton
Toll Pressure Oxidation	$55	$ /ton

Table 21-9 One Way Trucking Distance to Nevada Metallurgical Plants

Name and Description	Distance (miles)
Barrick Goldstrike Roaster	107
Barrick Goldstrike Autoclave	106
Jerritt Canyon Roaster	150
Newmont Gold Quarry Roaster	87
Newmont Twin Creeks Autoclave	101
Newmont Lone Tree Autoclave (Idle)	55

Table 21-10 Operating Costs

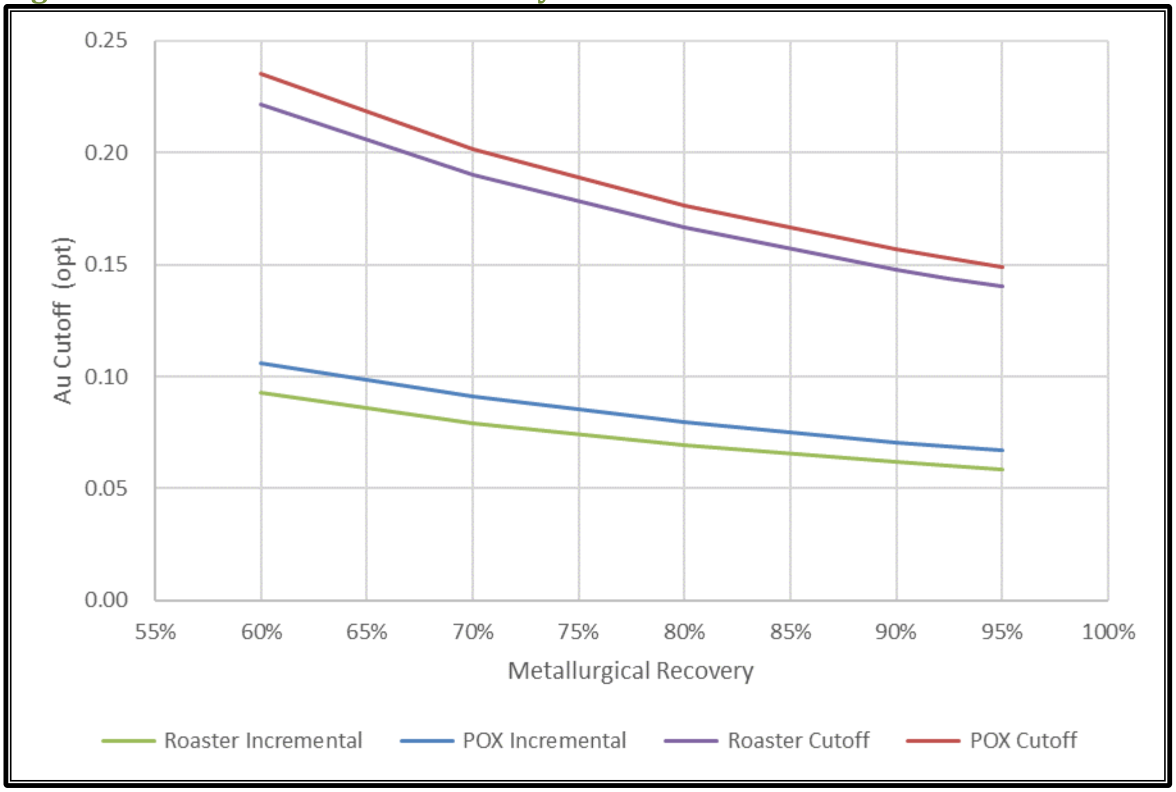
Category	Total Cost ($M)	$/ore ton	$/Au oz
Mining	270	92	365
Roasting	57	19	77
Pressure Oxidation	92	23	124
Ore Haulage	68	23	92
G&A, Royalties and Net Proceeds Tax	96	33	130
By Product Credits	(2)	(1)	(3)
Total Operating Costs	584	199	790
Closure and Reclamation	13	4	17
Income Tax	19	6	25

Category	Total Cost ($M)	$/ore ton	$/Au oz
Sustaining Capital	68	23	92
All in Sustaining Costs	684	233	924

21.4.Cutoff Grade

Cut off Grades were calculated using the operating costs presented above for each process at variable recoveries. (Figure 21-1)

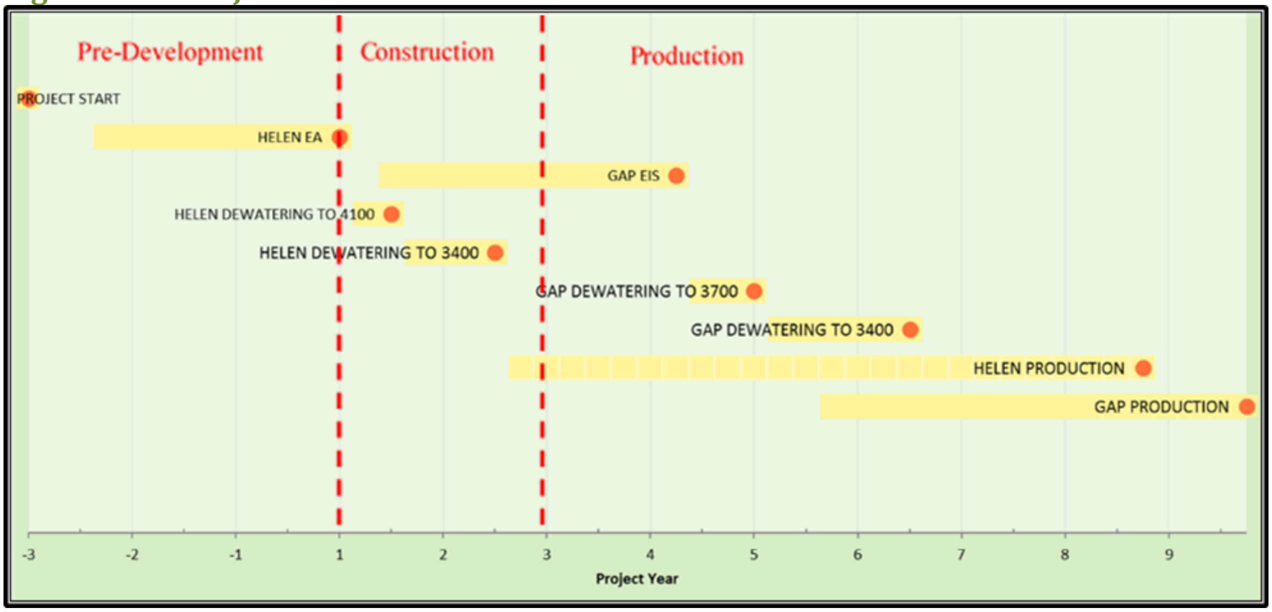
Figure 21-1 Cutoff Grade vs. Recovery



22. Economic Analysis

The project timeline is shown in Figure 22-1. The pre -development work is necessary to reach a production decision. All costs during this period are being treated as sunk costs and they have been excluded from the financial analysis.

Figure 22-1 Project Timeline



Constant dollar cash flow analysis is presented in Table 22-1 through Table 22-3 and graphically in Figure 22-2 and Figure 22-3. Royalties include both the 1 ½ % Newmont NSR and the 2% Summa Corporation NSR now held by Kinross. The Summa royalty does not apply to all the mineralization in the mine plan.

Federal income taxes of 21% apply to taxable income after appropriate deductions for depreciation and depletion. The gold percentage depletion rate is 15%.

Table 22-1 Income Statement

	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
Gold Sales	$0.2	$49.8	$56.7	$157.2	$132.4	$184.1	$128.0	$127.3	$88.7	$924.5
Silver Sales	$0.0	$0.1	$0.1	$0.1	$0.1	$0.6	$0.5	$0.4	$0.2	$1.9
Total Revenue	$0.2	$49.9	$56.7	$157.2	$132.5	$184.8	$128.4	$127.7	$89.0	$926.4
Mining Cost	($0.3)	($16.8)	($15.8)	($44.7)	($44.3)	($44.4)	($38.5)	($40.2)	($24.9)	($539.8)
Haulage and Processing	($0.1)	($12.1)	($12.7)	($33.4)	($29.5)	($39.2)	($33.7)	($33.7)	($22.5)	($433.9)
Electrical Power	$0.0	($2.2)	($2.4)	($2.4)	($4.3)	($6.0)	($6.3)	($5.8)	($5.3)	($34.6)
Site Administration	$0.0	($3.8)	($3.8)	($3.8)	($3.8)	($3.8)	($3.8)	($3.8)	($3.8)	($30.4)

	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
Refining and Sales	($0.0)	($0.2)	($0.2)	($0.6)	($0.5)	($0.7)	($0.5)	($0.5)	($0.4)	($3.7)
Royalties	($0.0)	($0.9)	($1.1)	($0.6)	($1.9)	($3.4)	($2.7)	($2.7)	($2.1)	($15.4)
Nevada Net Proceeds	$0.0	($0.5)	($0.8)	($3.3)	($2.1)	($4.0)	($1.8)	($1.7)	($1.2)	($15.4)
Total Cash Cost	($0.4)	($36.5)	($36.9)	($88.8)	($86.4)	($101.5)	($87.4)	($88.5)	($60.1)	($586.4)
Cash Cost per Ounce[1] ($/oz)	$2,471	$913	$812	$706	$815	$685	$849	$865	$843	$790
EBITA	($0.2)	$13.4	$19.9	$68.4	$46.1	$83.2	$41.1	$39.2	$28.9	$340.0
Reclamation Accrual	($0.0)	($0.7)	($0.8)	($2.2)	($1.8)	($2.5)	($1.8)	($1.8)	($1.2)	($12.8)
Depreciation	($0.0)	($3.9)	($5.2)	($17.8)	($19.8)	($29.4)	($23.8)	($23.7)	($16.5)	($140.1)
Total Cost	($0.4)	($41.0)	($42.8)	($108.8)	($108.1)	($133.5)	($112.9)	($113.9)	($77.8)	($739.4)
Income Tax	$0.0	($0.9)	($1.3)	($5.3)	($2.3)	($5.1)	($1.4)	($1.3)	($1.1)	($18.6)
Net Income	($0.2)	$8.0	$12.6	$43.1	$22.1	$46.2	$14.1	$12.5	$10.1	$168.4

1. *N et of Byproduct Sales*

Table 22-2 Cash Flow Statement

	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
Net Income	($0.2)	$8.0	$12.6	$43.1	$22.1	$46.2	$14.1	$12.5	$10.1	$0.0	$0.0	$168.4
Depreciation	$0.0	$3.9	$5.2	$17.8	$19.8	$29.4	$23.8	$23.7	$16.5	$0.0	$0.0	$140.1
Reclamation	($0.4)	$0.3	$0.4	$1.8	$1.0	$1.7	$1.0	$1.0	($1.3)	($2.5)	($2.7)	$0.3
Working Capital	($0.0)	($4.2)	($0.0)	($6.0)	$0.3	($1.7)	$1.6	($0.1)	$3.3	$6.9	$0.0	$0.0
Operating Cash Flow	($0.7)	$8.0	$18.1	$56.8	$43.2	$75.6	$40.5	$37.0	$28.6	$4.4	($2.7)	$308.8
Capital Costs	($26.2)	($20.4)	($11.8)	($17.7)	($23.9)	($5.2)	($9.0)	$0.0	$0.0	$0.0	$0.0	($114.4)
Net Cash Flow	($26.9)	($12.4)	$6.3	$39.0	$19.3	$70.4	$31.4	$37.0	$28.6	$4.4	($2.7)	$194.5
AISC [1,2] ($/oz)	$4,794	$945	$1,111	$892	$1,070	$761	$959	$886	$893	$0	$0	$924

1. *N et of Byproduct Sales*
2. *Note: AISC Exclusive of Corporate Costs*

Table 22-3 Financial Statistics[1]

Gold price - base case (US$/oz)	$1,250.00
Silver price - base case (US$/oz)	$17.00
Mine life (years)	8.0
Maximum mining rate (tons/day)	1,360.0
Average grade (oz/t Au)	0.305
Average gold recovery (roaster %)	79%
Average gold recovery (autoclave %)	86%
Average annual gold production (koz)	92
Total recovered gold (koz)	740
Pre-development capital ($M)	$26
Mine construction capital ($M)	$47
Sustaining capital (M$)	$68
Development Decision Date	January 2021

Cash cost (US$/oz)	$790
All-in sustaining cost (US$/oz)	$924
Project after-tax $NPV_{5\%}$ (M$)	$142
Project after-tax IRR	48%
Payback Period	4.0 Years
Profitability Index $_{5\%}$[3]	2.4

Notes:

1. *The financial data presented herein treats pre-development capital (planned expenditures prior to the development decision) as "sunk" costs and it is excluded from cost per ounce, NPV, IRR, payback period and profitability index calculations;*
2. *Net of byproduct sales;*
3. *Profitability index (PI), is the ratio of payoff to investment of a proposed project. It is a useful tool for ranking projects because it allows you to quantify the amount of value created per unit of investment. A profitability index of 1 indicates breakeven;*
4. *The Deferred Bullet Payment Consideration is not included in the cash-flow or financial calculations of this Technical Report.*
5. *The Newmont 1.5% NSR is not included in the cash-flow or financial calculations of this Technical Report.*
6. *The financial analysis contains certain information that may constitute "forward-looking information" under applicable Canadian securities legislation. Forward-looking information includes, but is not limited to, statements regarding the Company's achievement of the full-year projections for ounce production, production costs, AISC costs per ounce, cash cost per ounce and realized gold/silver price per ounce, the Company's ability to meet annual operations estimates, and statements about strategic plans, including future operations, future work programs, capital expenditures, discovery and production of minerals, price of gold and currency exchange rates, timing of geological reports and corporate and technical objectives. Forward-looking information is necessarily based upon a number of assumptions that, while considered reasonable, are subject to known and unknown risks, uncertainties, and other factors which may cause the actual results and future events to differ materially from those expressed or implied by such forward looking information, including the risks inherent to the mining industry, adverse economic and market developments and the risks identified in Premier's annual information form under the heading "Risk Factors". There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. All forward-looking information contained in this Presentation is given as of the date hereof and is based upon the opinions and estimates of management and information available to management as at the date hereof. Premier disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by law, and;*
7. *This PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.*

Figure 22-2 Gold Production and Unit Costs

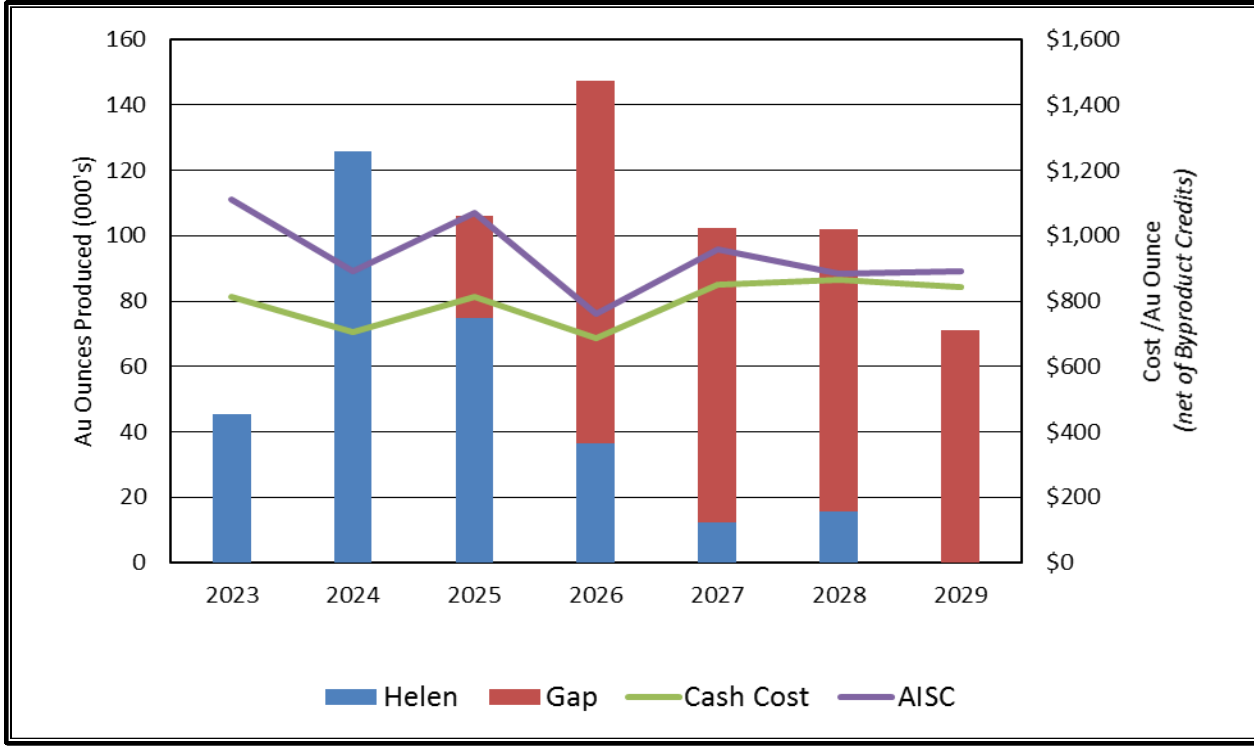
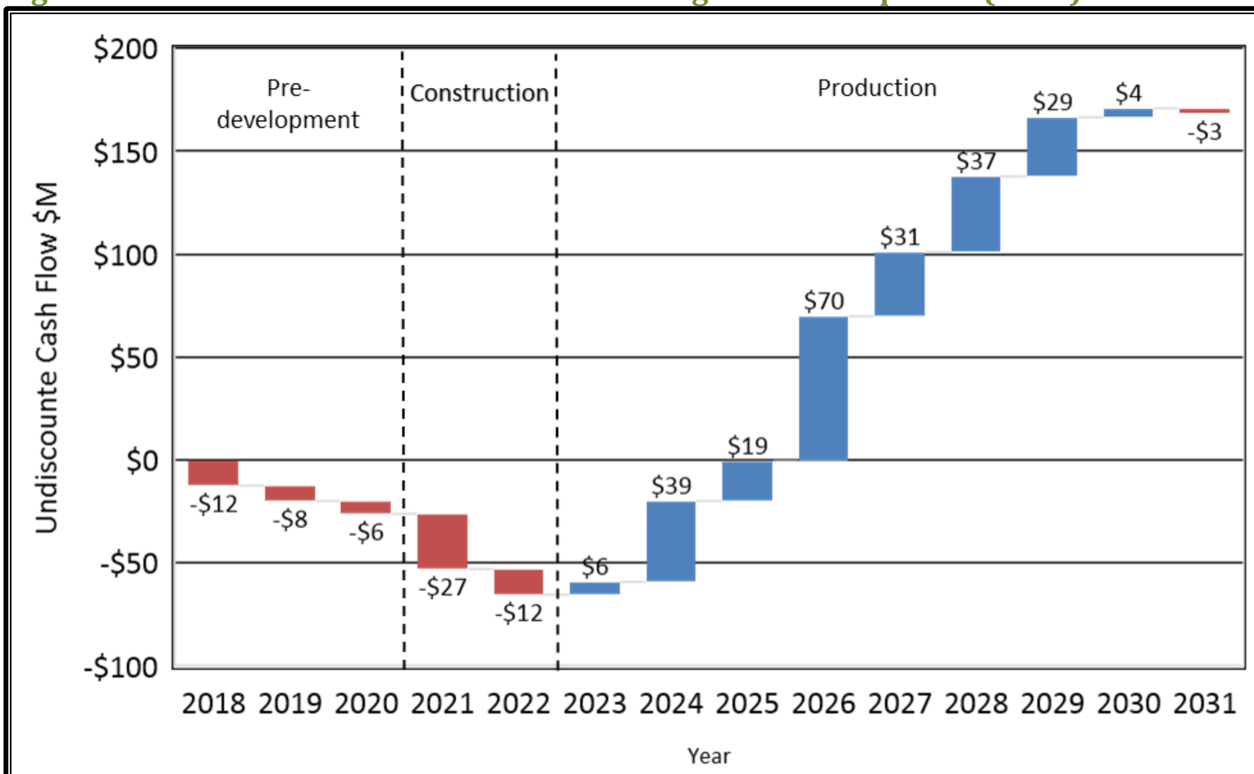


Figure 22-3 Cash Flow Waterfall Chart Including Pre-Development (Sunk) Costs



The project's sensitivity to -40% to +40% variations in commodity pricing, operating costs, and capital costs is presented in Figure 22-4 through Figure 22-6. The project is breakeven with a 20% negative variation of gold price to $1000 per ounce or with a 40% negative variation in operating costs.

Figure 22-4 NPV 5% Sensitivity

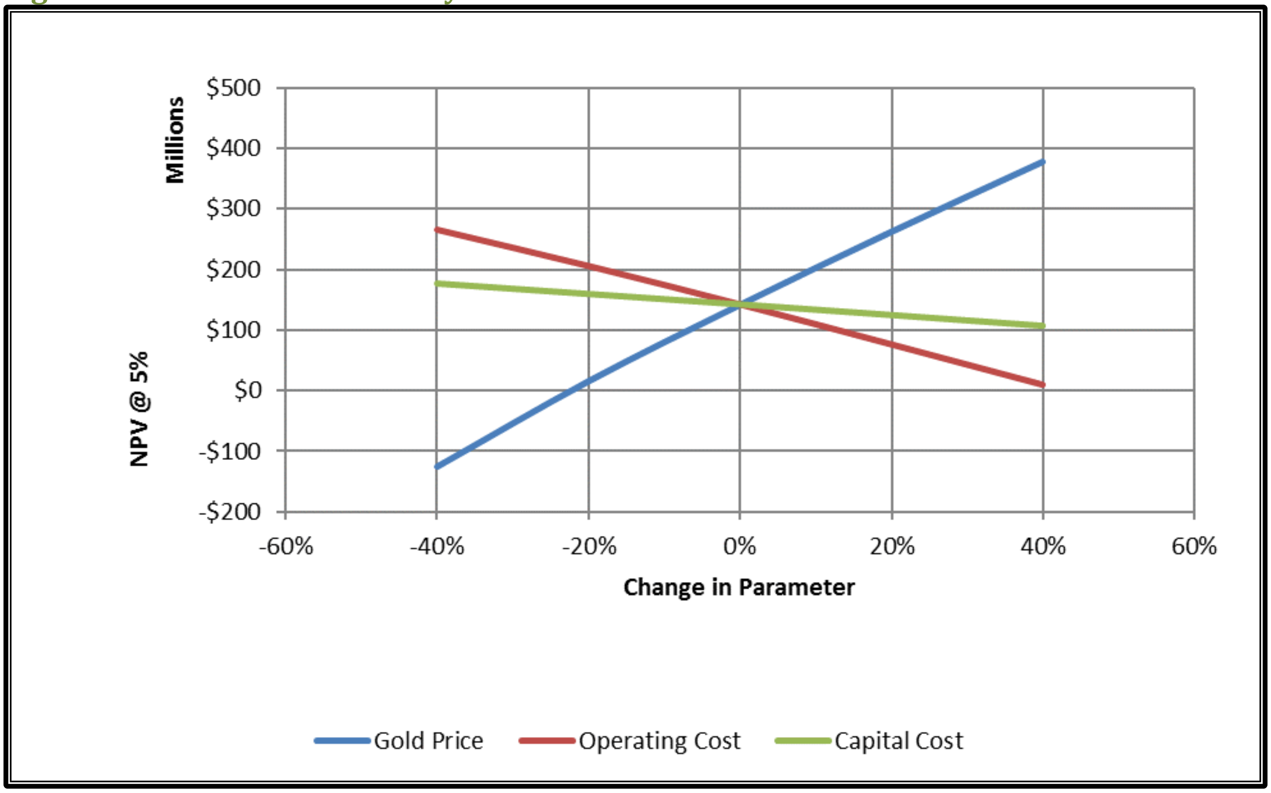


Figure 22-5 Profitability Index 5% Sensitivity

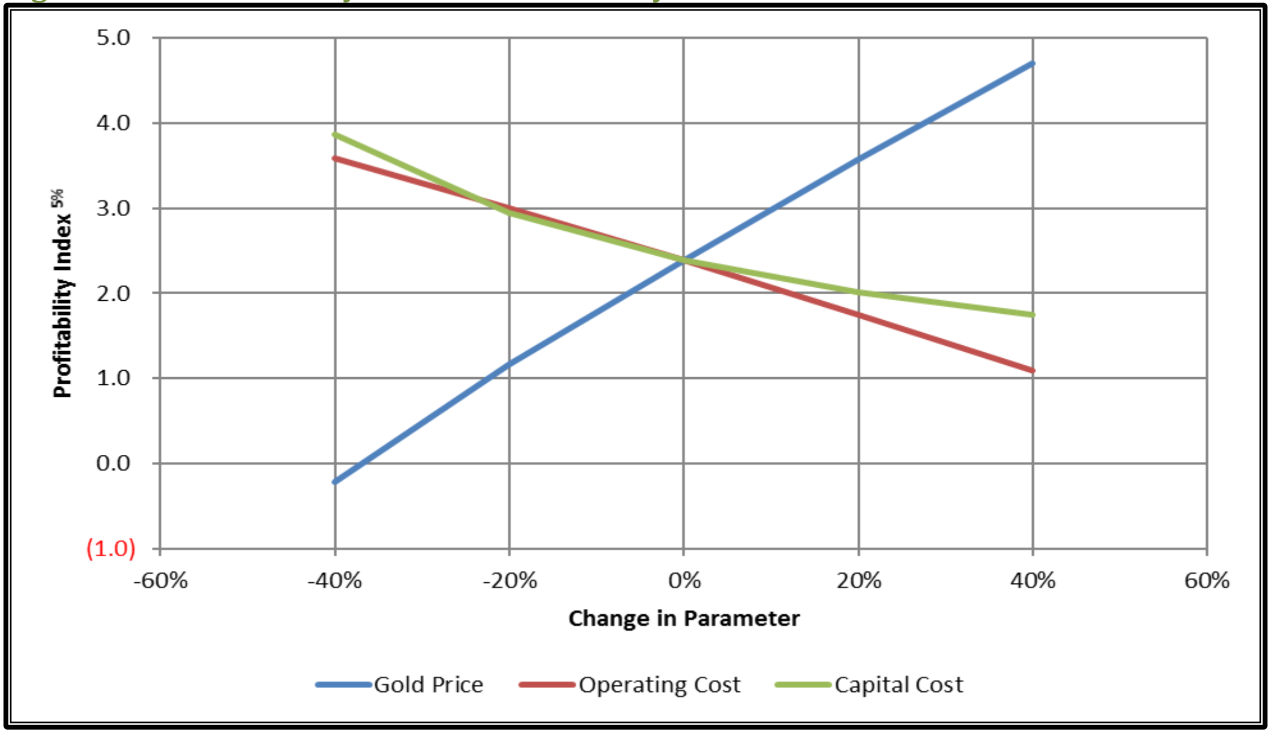
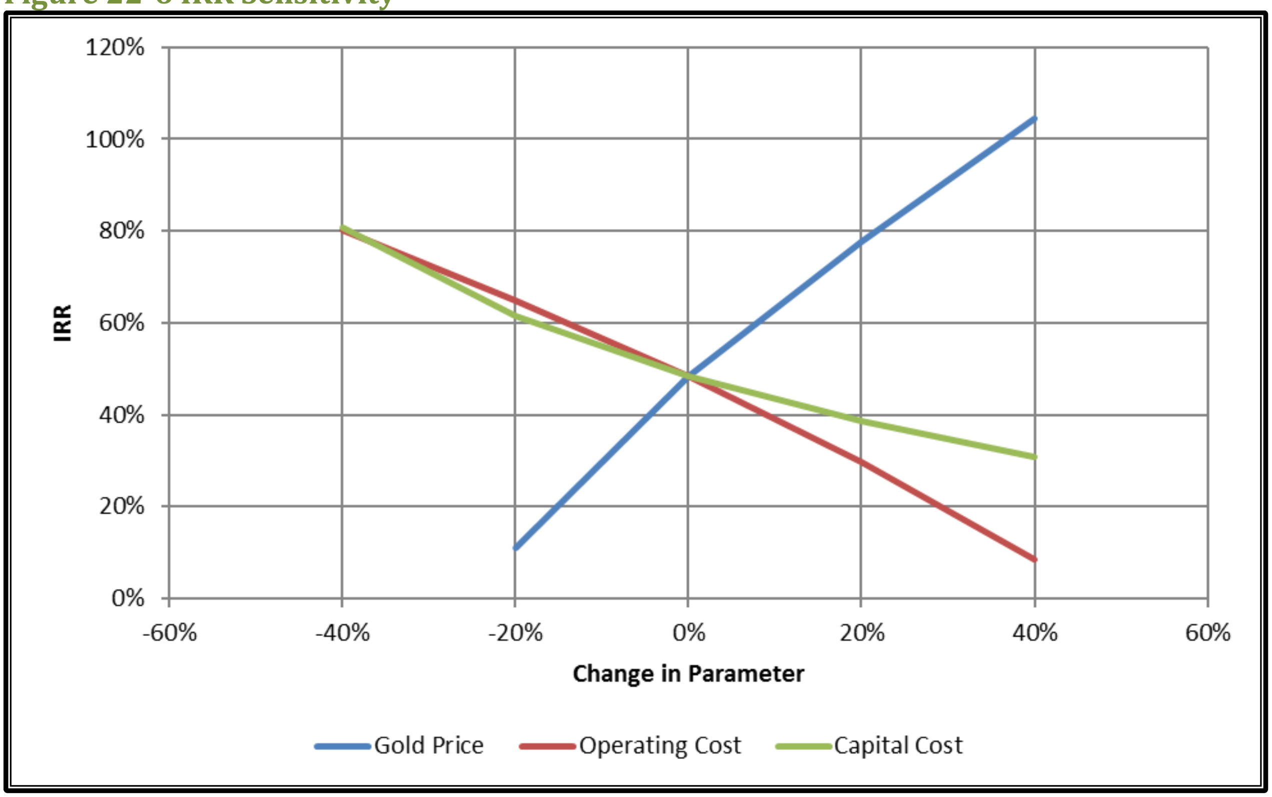


Figure 22-6 IRR Sensitivity



23. Adjacent Properties

There are no adjacent properties with a similar geologic setting to McCoy Cove.

24. Other Relevant Data and Information

The authors are not aware of any other relevant technical data or information pertaining to the Cove Project necessary to make this Technical Report understandable and not misleading.

25. Interpretation and Conclusions

The Cove Project is in a politically stable mining friendly jurisdiction with a long history of Mineral Resource extraction. The Project is potentially economic. Results from this PEA indicate a life of mine NPV 5% of $142M and IRR of 48%. The project should proceed immediately with a pre-feasibility or feasibility study in support of a development decision.

Metallurgical Testing

1. Head assaying for the both the Helen and Gap indicated that the gold in the two resources will likely be finely disseminated and will not likely have a significant coarse or nugget gold content;
2. The mineralogy of the Helen and Gap resources differ in two significant areas, the first being that the Helen appears to be lower in arsenic content than the Gap resource and that the Gap resource appears to be lower on average in TCM and TOC than the Helen resource;
3. The Helen composite arsenic assays indicate the resource lower in arsenic content that the Gap resource;
4. The Helen and Gap resources based on the composites tested appear to be generally refractory to conventional whole cyanidation and will need some type of oxidation process to significantly increase gold extractions over whole cyanidation;
5. Based on the composites tested the Helen Zone appear to generally be more amenable to Roasting and CIL cyanidation, however, there may be areas that are more amenable or can only be treated using pressure oxidation and residue CIL cyanidation;
6. Based on the composites tested, the Gap resource appears to generally be more amenable to pressure oxidation followed by residue CIL cyanidation, however, there may be areas that are more amenable or can only be treated using roasting and calcine CIL cyanidation, and;
7. The data set was too small to establish any clear relations of between mineralogy and metal head grade and extractions for either resource although it is clear that mineralogy factors such as arsenic content and TCM or TOC are influencing extractions using either roasting and calcine cyanidation or pressure oxidation and residue cyanidation.

Toll Processing

1. The feed specifications appear to be somewhat rigid and could preclude some material being sent to the toll processor. Blending may allow shipment of some off-specification material provided appropriate material is available for onsite blending prior to shipping to the toll processor;
2. The terms appear to be consistent and typical with those encountered in the industry, and;

3. The recovery terms appear to be the result of analyzing the metallurgical data provided by Premier Gold.

Mining and Infrastructure

1. Mining conditions typical for sedimentary deposits in the north-east Nevada extensional tectonic environments are anticipated;
2. Helen Zone dewatering will require five wells and reach pumping rates of 10,500 gpm, and;
3. Gap Zone dewatering will require ten wells and reach pumping rates of 26,000 gpm for a total projected pumping rate of 36,500 gpm.

Financials

1. Capital requirements total $114.4M excluding $25.8M in sunk pre-development capital;
2. The project achieves NPV 5% of $142M and NPV 8% of $118M, and;
3. The estimated payback period is 4.0 years with an IRR of 48%.

26. Recommendations

The project pre-feasibility or feasibility study should address the following components. The work should be planned to minimize the permitting time required to achieve positive cash flow.

Resource Delineation and Exploration

1. Portal construction and development of an underground drilling platform should proceed as soon as possible;
2. Resource delineation drilling from underground can be achieved with improved accuracy as compared to surface drill holes with depths approaching 2,000 feet and significant hole deviation;
3. The Cove Pit prohibits drilling the Gap extension area and portions of the Gap deposit. These are the most prospective nearby areas for adding significant Mineral Resources, and;
4. Expansion of the 2201 Zone could add high grade mineralization to the project which would be accessed through the Helen and Gap infrastructure.

Dewatering

1. PW 17-01 did not reach the targeted depth and pumping rates during the 30-day test were less than anticipated. Two additional wells and extended drawdown pumping tests need to be completed in the Helen and Gap zones during the 2018 season, and;
2. Complete detailed hydrogeologic modeling of the drawdown test results and update estimated dewatering requirements.

Mining

1. A geotechnical characterization program should be implemented along with resource delineation:
    a. The objectives of the program are to characterize the mining horizons using the Rock Mass Rating (RMR) system;
    b. Collect downhole Acoustic Tele Viewer (ATV) drill logs to collect joint orientation data for mine designs and accurately estimate ground support requirements, and;
    c. Collect full core samples for physical rock property testing.
2. Complete additional testing of potential back fill sources to optimize the Cemented Rock Fill (CRF) mix design, and
3. Complete a ventilation simulation to predict Diesel Particulate Matter (DPM), carbon monoxide, and other contaminate concentrations.

Metallurgical Testing

1. Additional metallurgical testing will be needed to thoroughly investigate the variability and viability of Helen and Gap resources to roasting and pressure oxidation with CIL cyanidation for which a program evaluating thirty to forty composites from each resource is suggested with objectives as follows:
    a. Assess variability of the responses to roasting and calcine cyanidation across the resources;
    b. Assess variability of the responses to pressure oxidation and residue cyanidation across the resources;
    c. Consider some POX optimization tests such as pre-acidulation ahead of the POX process;
    d. Testing should attempt to establish head grade and extraction relations for use in more detailed resource modelling;
    e. Mineralogy impacts need to be established and geologic domains within each resource need to be determined;
    f. Additional comminution data should be collected to assess variability within the resources.
2. In addition to evaluating resource process by a toll processing operator, consideration should be given to evaluate onsite processing;
3. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
4. Consider flotation tests to pre-float carbonates, and;
5. Consider other mill design tests as alternative to toll processing. These would include roasting, POX optimization tests, and solid liquid separation tests.

<u>Toll Processing</u>

1. The resource model should be advanced to include arsenic, TCM, TOC, mercury, lead, zinc, total copper selenium, barium, cobalt, nickel, and cadmium as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
2. Additional metallurgical testing should be conducted to confirm the proposed payable recoveries are appropriate for the resources;
3. Development of a preliminary or conceptual onsite blending program is recommended to evaluate if on specification material can consistently be supplied to a toll processor, and:
4. The next phase metallurgical program should examine blending of out of specification resource materials to produce on specification material. The blending should be based on

material projected to mined in a given period, for example, blending of material that is available in the first six months of operation should not be tested with material projected to only be available in year three of mining.

Permitting and Development Decision

1. Baseline data collection in support of the Helen EA and GAP EIS should be done simultaneously to reduce the Project's critical path and bring forward production, and;
2. The project should proceed directly with a feasibility or pre-feasibility study to support a development decision.

26.1. Risks and Opportunities

The authors have identified the following risks and opportunities to the project.

Table 26-1 Project Risks

Risks	Impact	Mitigation Measure
Agencies may require full EIS for Helen Mining rather than an EA	Project delays	Proceed with baseline data collection and engineering to support both possibilities
Dewatering rates may increase	Additional facilities required.	Complete hydrology testing in 2018
Water quality levels above Tier I standards for infiltration	Water treatment required, increased capital costs.	Geochemical study of RIB's to ascertain the possibility of attenuation
Water rights Availability	Project delays and increased costs	Continue water rights acquisition and seek agreements with local ranches

Table 26-2 Opportunities

Opportunities	Impact
Senior level government initiative to streamline the permitting process	Earlier production and increased NPV
Resource additions in the Gap Extension area	Increased ounce production and improved project economics
2201 Zone could add higher grade mineralization to the mine plan utilizing common infrastructure	Increased ounce production and improved project economics

Opportunities	Impact
Develop grade thickness mineralization model	Optimize mine design

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Certification of Authors

CERTIFICATE of QUALIFIED PERSON

Re: *Preliminary Economic Assessment for the Cove Project, Lander County, Nevada,* dated the 29th day of June 2018, with an effective date of March 31, 2018 (the "Technical Report"):

I, Mark A. Odell, P.E., do hereby certify that:

As of June 29, 2018, I am a consulting mining engineer at:
Practical Mining LLC
495 Idaho Street, Suite 205
Elko, Nevada 89801
775-345-3718

1) I am a Registered Professional Mining Engineer in the State of Nevada (# 13708), and a Registered Member (#2402150) of the Society for Mining, Metallurgy and Exploration (SME).

2) I graduated from The Colorado School of Mines, Golden, Colorado with a Bachelor of Science Degree in Mining Engineering in 1985. I have practiced my profession continuously since 1985.

3) Since 1985, I have held the positions of mine engineer, chief engineer, mine superintendent, technical services manager and mine manager at underground and surface metal and coal mines in the western United States. The past 12 years, I have worked as a self-employed mining consultant with clients located in North America, Asia and Africa. My responsibilities have included the preparation of detailed mine plans, geotechnical engineering, reserve and resource estimation, preparation of capital and operating budgets and the economic evaluation of mineral deposits.

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 experience and qualifications and good standing with proper designation within a recognized professional organization fully meet the criteria as a Qualified Person as defined under NI 43-101.

5) I am a contract consulting engineer for the Issuer and Project owner: Premier Gold Mines Limited and last inspected the Cove Project on October 11, 2017.

6) I am responsible for preparation of all sections of the Technical Report.

7) I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.

8) I was paid a daily rate for consulting services performed in evaluation of the Cove Project for Premier Gold Mines Limited and do not have any other interests relating to the project. I do not have any interest in adjoining properties in the Cove area.

9) I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in accordance with that instrument and form.

10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of June 2018,

"Signed" *Mark A. Odell*

Mark A. Odell, P.E.
Practical Mining LLC
markodell@practicalmining.com

CERTIFICATE OF AUTHOR

Re: *Preliminary Economic Assessment for the Cove Project, Lander County, Nevada*, dated the 29th day of June 2018, with an effective date of March 31, 2018 (the "Technical Report").

I, Laura M. Symmes, SME, do hereby certify that:

As of June 29, 2018, I am a geologist at:

Practical Mining, LLC
495 Idaho Street, Suite 205
Elko, NV 89801

1) I graduated with a Bachelor of Science degree in Geology from Utah State University in 2003.
2) I am a registered member of the Society for Mining, Metallurgy & Exploration (SME) #4196936.
3) I have worked as a geologist for a total of 14 years since my 2003 graduation from university. My experience has been focused on exploration and production of gold deposits, including planning and supervision of drill projects, generating data from drilled materials and making geologic interpretations, data organization, geologic mapping, building digital models of geologic features and mineral resources, and grade control of deposits in production.
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 on NI 43-101.
5) I am a contract consulting geologist for the Issuer and Project owner: Premier Gold Mines Limited.
6) I am responsible for sections 4 -12, and 14 of the Technical Report. I last visited the Cove Project on March 19, 2018.
7) I am independent of Premier Gold Mines Limited. within the meaning of Section 1.5 of National Instrument 43-101.
8) I was paid a daily rate for consulting services performed in evaluation of the Cove Project and do not have any other interests relating to the project. I do not have any interest in adjoining properties in the Cove area.
9) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this29th day of June 2018,

"Signed" Laura M. Symmes

Laura M. Symmes, SME **SME No. 4196936**

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CERTIFICATE OF AUTHOR

Re: *Preliminary Economic Assessment for the Cove Project, Lander County, Nevada*, dated the 29th day of June 2018, with an effective date of March 31, 2018 (the "Technical Report").

I, Sarah M Bull, P.E., do hereby certify that:

As of June 29, 2018, I am a consulting mining engineer at:

Practical Mining LLC
495 Idaho Street, Suite 205
Elko, Nevada 89801
775-345-3718

1) I am a Registered Professional Mining Engineer in the State of Nevada (# 22797).

2) I am a graduate of The University of Alaska Fairbanks, Fairbanks, Alaska with a Bachelor of Science Degree in Mining Engineering in 2006.

3) Since my graduation from university I have been employed as a Mine Engineer at an underground gold mining operation and as Senior Mine Engineer for a consulting engineering firm. My responsibilities have included mine ventilation engineering, stope design and mine planning.

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 experience and qualifications and good standing with proper designation within a recognized professional organization I fully meet the criteria as a Qualified Person as defined under the terms of NI 43-101.

5) I am a contract consulting engineer for the issuer and Project owner: Premier Gold Mines Limited

6) I am responsible for preparation of section 16 of the Technical Report. I have not made a personal inspection of the Cove Project.

7) I am independent of Premier Gold Mines Limited. within the meaning of Section 1.5 of NI 43-101.

8) I was paid a daily rate for engineering consulting services performed in evaluation of the Cove Project for Premier Gold Mines Limited and do not have any other interests relating to the project. I do not have any interest in adjoining properties in the Cove Project area.

9) I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in accordance with that instrument and form.

10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of June 2018.

"Signed" *Sarah Bull*

Sarah M Bull, P.E.

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CERTIFICATE of QUALIFIED PERSON

Re: *Preliminary Economic Assessment for the Cove Project, Lander County, Nevada*, dated the 29th day of June 2018, with an effective date of March 31, 2018 (the "Technical Report"):

I, Adam S. Knight, P.E., do hereby certify that:

As of June 29, 2018, I am a consulting mining engineer at:
Practical Mining LLC
495 Idaho Street, Suite 205
Elko, Nevada 89801
775-345-3718

1) I am a Registered Professional Mining Engineer in the State of Nevada (# 15796).

2) I graduated with a Bachelor of Science degree in Mining Engineering from University of Nevada Reno in 1997.

3) Since 1993, I have worked as Mine Surveyor, Mine Engineer, Mine Manager, Consulting Engineer, and Mining and Milling General Manager. Positions have been held in the US and Africa. Commodities worked include gold, silver, molybdenum and tungsten. I have 20 years' experience in surface and underground mines, including seven years' supervising and managing mineral processing operations.

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 experience and qualifications and good standing with proper designation within a recognized professional organization fully meet the criteria as a Qualified Person as defined under NI 43-101.

5) I am a contract consulting engineer for the Issuer and Project owner: Premier Gold Mines Limited and last inspected the Cove Project on June 5, 2018.

6) I am responsible for preparation of section 16, of the Technical Report.

7) I am independent of the Issuer within the meaning of Section 1.5 of NI 43-101.

8) I was paid a daily rate for consulting services performed in evaluation of the Cove Project for Premier Gold Mines Limited and do not have any other interests relating to the project. I do not have any interest in adjoining properties in the Cove area.

9) I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in accordance with that instrument and form.

10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of June 2018,

"Signed" *Adam S. Knight*

Adam S. Knight, P.E.
Practical Mining LLC
adamknight@practicalmining.com

JACOBS

CERTIFICATE OF QUALIFIED PERSON

I, Richard A. Bohling, as a co-author of the Preliminary Economic Assessment, do hereby certify that:

1. I am currently Technical Services Manager for:

   Jacobs Engineering Group
   9191 South Jamaica Street
   Englewood, Colorado, USA, 80112

2. This certificate applies to the technical report prepared for Premier Gold Mines Limited entitled Preliminary Economic Assessment for the Cove Project, Lander County, Nevada with an effective date of March 31, 2018 and a filing date of June 29, 2018.
3. I graduated from the Colorado School of Mines in 1975 with a Bachelor of Science degree in Metallurgical Engineering. I have practiced my profession continuously for forty-three years since graduation and have experience in the engineering, operation, and management of mining, mineral processing, metallurgical, water treatment plants and support facilities. I have been responsible for feasibility studies, operating cost estimates, financial evaluations, metallurgical and environmental testing and monitoring, project management, and project supervision. I have provided mineral processing, metallurgical, and general process expertise for the evaluation, design, and construction of new and existing facilities, and have been involved in the permitting, startup and daily operations of several mills, heap leach operations, tailings facilities, and water treatment plants. I have experience with environmental permitting, permit compliance, and the operation of remediation facilities.
4. I am a Registered Professional Engineer in the state of Colorado (19639) and am a member of the Society for Mining, Metallurgy, and Exploration (SME).
5. I visited the Cove project site on March 15th, 2017 for 1 day.
6. I am responsible for Sections 1.4, 1.7 and 1.8 – Metallurgical Testing and Toll Processing, 13, 17.1, 17.2, 17.3.1, 17.4, and 25 – Metallurgical Testing and Toll Processing of the Technical Report.
7. I am independent of the technical report issuer per Section 1.5 of NI 43-101.
8. I was involved in the initial startup and operation of the Cove Mill in 1987 as an employee of Echo Bay Mine and have had no involvement since 1987 or have no interest in the property that is the subject of this technical report.
9. I have read NI-43-101 and the sections of the Technical Report under my responsibility have been prepared in compliance with the instrument.
10. Those, as the effective date of the Technical Report, to best of my knowledge, information, and belief, the sections of the Technical report under my responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Dated: June 29, 2018

Richard A. Bohling, PE

Technical Services Manager