



Feasibility Study Copperwood Project Michigan, USA

Prepared for:

Highland Copper Company Inc.

EFFECTIVE DATE : June 14, 2018

SIGNATURE DATE : July 31, 2018

Prepared by:

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Feasibility Study – Copperwood Project

Revision #0

Michigan, USA

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Table of Content

1. SUMMARY	1-1
1.1 Introduction	1-1
1.2 Reliance on Other Experts	1-1
1.3 Property Description and Location	1-2
1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography	1-2
1.4.1 Accessibility	1-2
1.4.2 Climate	1-3
1.4.3 Local Resources	1-3
1.4.4 Existing Infrastructure	1-3
1.4.5 Physiography	1-3
1.5 History	1-4
1.6 Geological Setting	1-6
1.7 Mineralization	1-6
1.8 Deposit Types	1-6
1.9 Exploration	1-7
1.10 Drilling	1-8
1.11 Data Verification	1-9
1.12 Mineral Processing and Metallurgical Testing	1-9
1.13 Mineral Resources Estimate	1-10
1.14 Mineral Reserves Estimate	1-13
1.15 Mining	1-14
1.15.1 Mining Method	1-14
1.15.2 Mine Access	1-14
1.15.3 Rock Mechanics and Geotechnical Design Criteria	1-14
1.15.4 Mine Design	1-16
1.15.5 Mine Production Schedule	1-18
1.15.6 Mine Operations	1-22
1.15.7 Mine Services	1-23
1.16 Recovery Methods	1-23
1.16.1 Crushed Ore Reclaim	1-24
1.16.2 Grinding and Classification	1-24
1.16.3 Rougher Flotation	1-24
1.16.4 Regrind	1-24
1.16.5 Cleaner Flotation	1-24
1.16.6 Concentrate Thickening and Filtration	1-25
1.16.7 Tailings Pumping	1-25
1.17 Project Infrastructure	1-25
1.18 Market Studies and Contracts	1-26
1.19 Environmental Studies and Permitting	1-27
1.19.1 Environmental Studies	1-27

1.19.2	Permitting	1-27
1.20	Capital and Operating Costs	1-28
1.21	Economic Analysis	1-30
1.22	Adjacent Properties	1-32
1.23	Other Relevant Data and Information.....	1-32
1.24	Interpretation and Conclusions.....	1-32
1.24.1	Conclusions	1-32
1.24.2	Risks and Opportunities	1-33
1.25	Recommendations.....	1-33
2.	INTRODUCTION	2-1
2.1	Scope of Work	2-1
2.2	Sources of Information and Data.....	2-2
2.3	Qualifications and Experience	2-3
2.4	Site Visits	2-4
2.5	Units of Measure, Abbreviations and Nomenclature	2-5
3.	RELIANCE ON OTHER EXPERTS	3-1
4.	PROPERTY DESCRIPTION AND LOCATION	4-1
4.1	Location	4-1
4.2	Mineral Tenure	4-1
4.3	Surface Rights	4-3
4.4	Agreements, Royalties and Encumbrances	4-5
4.4.1	Mining Leases	4-5
4.4.2	Options to Lease	4-7
4.4.3	Encumbrances.....	4-8
4.4.4	Osisko Royalty.....	4-8
4.5	Environmental Liabilities	4-9
4.6	Permitting	4-9
4.6.1	Exploration.....	4-9
4.6.2	Development	4-10
4.7	Socio-Economic.....	4-11
5.	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY	5-1
5.1	Accessibility	5-1
5.2	Climate.....	5-1
5.3	Local Resources.....	5-3
5.4	Infrastructure	5-3
5.5	Physiography	5-5
6.	HISTORY.....	6-1
6.1	Exploration History	6-1
6.2	Production History	6-4
6.3	Environmental History	6-4
6.4	Historical Resources.....	6-4

6.4.1	USMR and AMAX Historical Resource Estimates	6-5
6.4.2	Orvana – AMEC Historical Resource Estimates	6-6
6.4.3	Orvana – Marston Historical Resource Estimate	6-7
6.4.4	Highland – GMSI Resource Estimate.....	6-8
6.4.5	Highland – GMSI Resource Update October 2017	6-10
7.	GEOLOGICAL SETTING AND MINERALIZATION.....	7-1
7.1	Regional Geology	7-1
7.2	Project Area Geology	7-5
7.2.1	Lower Copper Bearing Sequence	7-6
7.2.2	Upper Copper Bearing Sequence	7-9
7.2.3	Nonesuch Undivided and Freda Formations.....	7-10
7.2.4	Structure	7-12
7.3	Mineralization	7-19
7.4	Comparison to White Pine Deposit	7-22
8.	DEPOSIT TYPES	8-1
9.	EXPLORATION.....	9-1
9.1	Exploration History	9-1
9.2	Orvana Exploration Programs	9-1
9.3	Highland Exploration Program	9-1
9.4	Airborne Geophysical Studies	9-2
9.5	Geochemical Surveys.....	9-2
10.	DRILLING.....	10-1
10.1	Drilling History	10-1
10.2	Drilling Procedures	10-3
10.2.1	Collar Surveys	10-5
10.2.2	Down-Hole Surveys.....	10-5
10.2.3	Core Logging	10-5
10.2.4	Core Storage	10-6
10.3	Sampling Method and Approach	10-6
11.	SAMPLE PREPARATION, ANALYSES AND SECURITY	11-1
11.1	Sample Preparation and Reduction	11-1
11.1.1	Analysis	11-1
11.1.2	Quality Control.....	11-2
11.1.3	Blanks and Assessment of Contamination.....	11-3
11.1.4	Duplicate Sample Performance.....	11-5
11.2	Performance of Standards	11-9
11.3	Security.....	11-17
11.4	Conclusions	11-18
12.	DATA VERIFICATION.....	12-1
12.1	Database	12-1
12.2	GMSI Data Verification	12-3
12.3	Drill Hole Collar Location	12-4
12.4	QA/QC Validation	12-6

12.5	Conclusions	12-6
13.	MINERAL PROCESSING AND METALLURGICAL TESTING.....	13-1
13.1	Early Metallurgical Testing (before 2012).....	13-1
13.2	2013 Locked Cycle Flotation Tests	13-5
13.2.1	Head Assay	13-6
13.2.2	NaHS Dosage Series	13-6
13.2.3	Locked Cycle Flotation	13-7
13.2.4	2017 Grindability Tests – Main Zone.....	13-9
13.2.5	Sample Preparation and Testing Matrix	13-11
13.2.6	JK Drop-Weight and SMC Tests	13-14
13.2.7	Bond Rod Mill Grindability Test.....	13-15
13.2.8	Bond Ball Mill Grindability Test.....	13-16
13.2.9	Bond Abrasion Test.....	13-18
13.3	2017 Grindability Tests – Zones 5 and 6.....	13-19
13.3.1	Sample Preparation.....	13-19
13.3.2	SMC Tests.....	13-22
13.3.3	Bond Ball Mill Grindability Test.....	13-22
13.3.4	Bond Abrasion Test.....	13-23
13.3.5	Special Jar Mill Grindability Test and SMD Lab Test	13-24
13.3.6	High Intensity Grindability Test.....	13-25
13.4	2017 / 2018 Flotation Optimization Tests.....	13-26
13.4.1	Test Program Summary	13-26
13.4.2	Sample Location and Composite Definition	13-26
13.4.3	Head Assays	13-29
13.4.4	Bench Scale Rougher Flotation Tests.....	13-31
13.4.5	Bench Scale Cleaner Flotation Tests	13-33
13.4.6	Flash Flotation Tests	13-37
13.4.7	Locked Cycle Flotation Tests	13-37
13.4.8	Variability Testwork	13-39
13.4.9	MLA Analysis.....	13-42
13.4.10	QEMSCAN Assay Reconciliation	13-42
13.4.11	Modal Mineralogy	13-43
13.4.12	Copper Sulphides and Silicate Association.....	13-46
13.4.13	Copper Deportment.....	13-47
13.4.14	Locked Cycle Concentrate Specifications.....	13-49
13.4.15	Testwork Discussion and Recommendations	13-51
13.5	Key Process Design Criteria	13-53
14.	MINERAL RESOURCE ESTIMATE	14-1
14.1	Data	14-1
14.1.1	Drill Hole Spacing	14-2
14.1.2	Data Conditioning	14-3
14.2	Modelling Approach.....	14-4
14.2.1	Structural Model	14-5
14.2.2	Lithology Model - LCBS.....	14-6
14.2.3	Weathering Wireframes.....	14-8
14.2.4	Topography Surface.....	14-9

14.3	Statistical Analysis	14-9
14.3.1	Statistics of the Raw Assays	14-9
14.3.2	Contact Analysis	14-11
14.3.3	Compositing	14-13
14.3.4	Statistics of the Composites	14-13
14.4	Bulk Density Data	14-16
14.5	Variography	14-18
14.6	Block Modelling	14-19
14.7	Grade Estimation Methodology	14-22
14.8	Classification and Resource Reporting	14-25
14.9	Grade Estimation Validation	14-30
14.9.1	Visual Validation	14-30
14.9.2	Statistical Validation	14-30
14.9.3	Quantile: Quantile Plots	14-32
14.9.4	Local Statistical Validation - Swath Plots	14-33
14.9.5	Discussion on Block Model Validation	14-34
14.10	Global Resources	14-34
14.10.1	Grade Dilution	14-34
14.10.2	Constrained Underground Mineral Resources Sensitivity - LCBS	14-36
14.10.3	Constrained Underground Mineral Resources Sensitivity – UCBS	14-39
14.11	Underground Constrained Resources	14-42
14.11.1	Underground Optimization Parameters	14-42
14.11.2	Underground Mineral Resource Estimate	14-43
15.	MINERAL RESERVE ESTIMATES	15-1
15.1	Estimation Procedures	15-2
15.2	Cut-Off Grade	15-4
15.3	Ore Recovery and Mine Dilution	15-6
15.4	Minimum Mining Height	15-8
15.5	Factors Possibly Affecting Mineral Reserves	15-9
15.6	Comments	15-10
16.	MINING METHODS	16-1
16.1	Introduction	16-1
16.2	Geotechnical Considerations	16-3
16.2.1	Geotechnical Background	16-3
16.2.2	Geotechnical Characterization	16-5
16.2.3	Available Geotechnical Data	16-6
16.2.4	Intact Rock Strength	16-6
16.2.5	Rock Structure	16-8
16.2.6	Rock Quality	16-9
16.2.7	Rock Mass Strength	16-9
16.2.8	Basal Gouge	16-10
16.2.9	Other Design Considerations	16-11
16.2.10	Pillar Design	16-12
16.2.11	Regional Pillars	16-14

16.2.12	Superior Lake Protection	16-14
16.2.13	Ground Support	16-14
16.2.14	Subsidence	16-15
16.2.15	Hydrogeological Considerations.....	16-15
16.3	Selection of Mining Method	16-16
16.3.1	Development Design	16-16
16.3.2	Stope Entry	16-17
16.3.3	Exhaust Ventilation Raise	16-18
16.3.4	Stope Design	16-18
16.4	Mine Operations	16-22
16.4.1	Stoping	16-22
16.4.2	Ore Handling System	16-27
16.4.3	Mining Equipment.....	16-27
16.5	Development Schedule	16-28
16.5.1	Pre-production Objectives	16-29
16.5.2	Production	16-29
16.6	Production Schedule	16-30
16.7	Manpower and Working Schedule	16-35
16.8	Mine Services	16-40
16.8.1	Ventilation	16-40
16.8.2	Water Supply	16-41
16.8.3	Power	16-42
16.8.4	Dewatering	16-42
16.8.5	Compressed Air	16-44
16.8.6	Fuel Storage and Distribution	16-44
16.8.7	Communications.....	16-45
16.8.8	Explosives Storage and Handling	16-45
16.8.9	Personnel and Underground Material Transportation	16-45
16.8.10	Underground Construction and Mine Maintenance.....	16-45
16.8.11	Equipment Maintenance.....	16-46
16.9	Safety Measures.....	16-46
16.9.1	Industrial Hygiene.....	16-46
16.9.2	Emergency Exits.....	16-46
16.9.3	Refuge Stations	16-46
16.9.4	Fire Protection	16-47
16.9.5	Mine Rescue.....	16-47
16.9.6	Emergency Stench System	16-47
16.9.7	Dust Control.....	16-47
17.	RECOVERY METHODS.....	17-1
17.1	Process Design	17-1
17.1.1	Selected Process Flowsheet	17-1
17.1.2	Key Process Design Criteria	17-4
17.2	General Process Description.....	17-6
17.3	Crushed Ore Reclaim	17-6
17.4	Grinding and Classification Circuit	17-6

17.5	Rougher Flotation	17-7
17.6	Regrind	17-8
17.7	Cleaner Flotation	17-9
17.8	Concentrate Thickening and Filtration.....	17-10
17.9	Tailings Handling	17-10
17.10	Raw Water, Potable Water and Process Water	17-11
17.11	Reagents	17-12
17.11.1	Frother (MIBC/D-250).....	17-12
17.11.2	Sodium Isobutyl Xanthate (SIBX).....	17-12
17.11.3	Sodium Silicate/Carboxymethyl Cellulose Sodium (SS/CMC).....	17-12
17.11.4	N-Dodecyl Mercaptan (NDM)	17-13
17.11.5	Flocculant	17-13
17.11.6	Sodium Hydrosulphide (NaSH)	17-14
17.11.7	Aeroflot 249 (A-249)	17-14
17.11.8	Hydrated Lime	17-14
17.11.9	Anti-scalant.....	17-14
17.12	Services and Utilities	17-15
17.12.1	On-stream Analysis System	17-15
17.12.2	High and Low-pressure Air	17-15
18.	PROJECT INFRASTRUCTURE	18-1
18.1	General	18-1
18.2	Public Access Road.....	18-4
18.3	Communications.....	18-4
18.4	Site Roads	18-5
18.4.1	Main Access Road.....	18-5
18.4.2	Site Roads	18-6
18.5	Box-Cut and Ore Stockpile	18-6
18.5.1	Box-Cut.....	18-6
18.5.2	Ore Stockpile Pad.....	18-7
18.6	Water Management.....	18-8
18.6.1	Sewage Treatment	18-8
18.6.2	Water Filtration	18-8
18.6.3	Water Treatment Plant	18-8
18.6.4	Water Treatment Plant Design	18-10
18.6.5	Influent Design Basis.....	18-10
18.6.6	Water Treatment Plant Process	18-11
18.7	Potable Water – Lake Superior In-Take	18-13
18.8	Diesel Fuel Storage	18-13
18.9	Power Supply and Distribution	18-13
18.10	Fire Protection	18-13
18.11	Security.....	18-14
18.12	Explosive Magazine.....	18-14
18.13	On-site Buildings	18-14
18.13.1	Truckshop, Warehouse and Related Offices	18-14

18.13.2 Mine Dry	18-15
18.13.3 Construction Offices	18-17
18.13.4 Met Lab and Mill Offices	18-17
18.13.5 Off-Site Buildings	18-17
18.13.6 Administration Building and Assay Laboratory	18-17
18.13.7 Transload Facility	18-18
18.14 Site Vehicles and Mobile Equipment	18-19
18.15 Tailings Storage Facility	18-19
18.15.1 General Arrangement and Development	18-19
18.15.2 Embankment Disposition	18-20
18.15.3 Decant System and Tailings Management	18-27
19. MARKET STUDIES AND CONTRACTS	19-1
19.1 Metal Prices	19-1
19.2 Market Studies	19-3
19.2.1 Copper Concentrate	19-3
19.3 Realization Costs	19-5
19.3.1 Concentrate Transportation	19-5
19.3.2 Insurance	19-6
19.3.3 Losses	19-7
19.4 Contracts	19-7
20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	20-1
20.1 Introduction	20-1
20.2 Environmental Studies	20-1
20.3 Description of Site Features	20-3
20.3.1 Topography and Drainage	20-3
20.3.2 Soil Conditions	20-3
20.3.3 Acid-Rock Drainage Potential of Site Geology	20-4
20.3.4 Hydrogeology	20-4
20.3.5 Streams, Wetlands, Ponds, and other Surface Water Bodies	20-5
20.3.6 Surface Water (Stream) Flow	20-6
20.3.7 Water Balance	20-6
20.3.8 Floodplains	20-7
20.3.9 Great Lakes Shoreline	20-7
20.3.10 Wetlands Near the Mine Site	20-7
20.3.11 Natural Rivers and Wild & Scenic Rivers	20-7
20.3.12 Air Quality	20-8
20.3.13 Existing Infrastructures and Utilities	20-8
20.3.14 Historical and Current Land Use	20-9
20.3.15 Flora and Fauna	20-9
20.3.16 Plant and Animal Species of Special Concern	20-10
20.3.17 Threatened or Endangered Species	20-11
20.3.18 Species of Special Concern	20-12
20.3.19 Non-Native or Invasive Species	20-13
20.3.20 Identified Ecological Systems	20-15
20.3.21 Archaeological and Historical Resources	20-16

20.4	Potential Impacts to Site Features and Mitigations	20-16
20.4.1	Topography and Drainage.....	20-16
20.4.2	Surface Water Flow	20-16
20.4.3	Water Balance	20-17
20.4.4	Surface Water Quality	20-18
20.4.5	Wetlands.....	20-18
20.4.6	Great Lakes Shorelines	20-19
20.4.7	Flora and Fauna	20-20
20.4.8	Invasive Species.....	20-23
20.4.9	Air Quality Impacts	20-23
20.4.10	State Park Impacts	20-23
20.4.11	Aesthetic Impacts	20-24
20.4.12	Acoustic Impacts	20-24
20.4.13	Seismic Impacts	20-25
20.5	Tailings Management	20-25
20.6	Cumulative and Additive Impacts	20-25
20.7	Environmental Management Plan	20-26
20.8	Reclamation Plan and Financial Assurance	20-27
20.9	Legal Framework.....	20-29
20.10	Permitting Process	20-29
20.11	Permits to Obtain.....	20-29
21.	CAPITAL AND OPERATING COSTS	21-1
21.1	Capital Expenditures	21-1
21.1.1	Infrastructures.....	21-2
21.1.2	Power Supply and Communications	21-3
21.1.3	Water and Tailings Disposal Management	21-4
21.1.4	Mobile Equipment.....	21-5
21.1.5	Mine Infrastructure.....	21-6
21.1.6	Process Plant and Related Infrastructures.....	21-7
21.1.7	Construction Indirect Costs	21-10
21.1.8	General Services	21-10
21.1.9	Pre-production and Commissioning Expenditures	21-11
21.2	Sustaining Capital.....	21-12
21.3	Closure Costs and Salvage Value.....	21-15
21.4	Operating Costs.....	21-17
21.4.1	Mining Costs.....	21-19
21.4.2	Processing Costs.....	21-21
21.4.3	General and Administration	21-24
22.	ECONOMIC ANALYSIS	22-1
22.1	Assumptions	22-1
22.1.1	Metal Prices.....	22-1
22.1.2	Fuel.....	22-1
22.1.3	Exchange Rates	22-2
22.2	Metal Production and Revenue	22-2
22.3	Capital Expenditures	22-5

22.3.1	Initial Capital Expenditures	22-5
22.3.2	Sustaining Capital Expenditures	22-6
22.3.3	Closure and Reclamation	22-7
22.3.4	Working Capital	22-7
22.4	Operating Cost Summary	22-7
22.5	Taxes and Royalties	22-9
22.5.1	Income Tax	22-9
22.5.2	Michigan Severance Tax	22-9
22.5.3	Royalties	22-10
22.6	Economic Model Results	22-10
22.7	Sensitivity Analysis	22-14
23.	ADJACENT PROPERTIES	23-1
24.	OTHER RELEVANT DATA AND INFORMATION	24-1
24.1	Project Implementation	24-1
24.2	Project Development Organization	24-1
24.3	Project Engineering	24-5
24.4	Early Works	24-6
24.5	Detailed Engineering/Procurement	24-7
24.6	Capital Procurement and Logistics	24-7
24.7	Project Controls	24-9
24.8	Quality and Design Standards	24-9
24.9	Quality Management	24-10
24.10	Commissioning	24-10
24.11	Project Schedule	24-11
25.	INTERPRETATION AND CONCLUSIONS	25-1
25.1	Conclusions	25-1
25.1.1	Geology and Mineral Resources	25-1
25.1.2	Mining and Mineral Reserves	25-3
25.1.3	Metallurgical Testing and Mineral Processing	25-4
25.1.4	Infrastructure	25-5
25.1.5	Environmental and Permitting	25-6
25.1.6	CAPEX, OPEX and Economic Analysis	25-7
25.2	Risks and Opportunities	25-8
25.2.1	Risks	25-8
25.2.2	Opportunities	25-10
26.	RECOMMENDATIONS	26-1
26.1	Project Recommendations	26-1
26.2	Recommended Work Programs	26-1
27.	REFERENCES	27-1

List of Figures

Figure 1.1: Mining Column and Pillar Stratigraphy	1-15
Figure 1.2: Mine Design General Arrangement	1-17
Figure 1.3: Mine Production Schedule	1-18
Figure 1.4: After-Tax Annual Project Cash Flow	1-31
Figure 1.5: After-Tax Project Cash Flow Waterfall	1-31
Figure 4.1: Project Location and Infrastructure	4-2
Figure 4.2: Project Location with Lease Information - Surface and Mineral Rights	4-4
Figure 5.1: Project Location and Infrastructure	5-2
Figure 5.2: Michigan Natural Gas Pipeline Map	5-4
Figure 5.3: Michigan Gas Utility Service Areas	5-4
Figure 7.1: Location of the Midcontinent Rift System	7-2
Figure 7.2: Stratigraphic Column of Regional Geology	7-3
Figure 7.3: Regional Geology and Project Location	7-4
Figure 7.4: Copperwood Deposit Stratigraphy	7-7
Figure 7.5: Mineralized Domino Subunit Drill Core Sample	7-8
Figure 7.6: Stratigraphic Column of the Project Area Geology	7-11
Figure 7.7: Thrust Fault and Basal Gouge Thickness	7-13
Figure 7.8: Cross Section Showing the LCBS – South West-North East Fence Diagram – Western Copperwood Deposit	7-14
Figure 7.9: Cross Section Showing the LCBS – South-North Fence Diagram – Western Copperwood Deposit	7-15
Figure 7.10: Cross Section Showing the LCBS – South-North Fence Diagram – Central Copperwood Deposit	7-16
Figure 7.11: Cross Section Showing the LCBS – South-North Fence Diagram – East Copperwood and Satellite Deposits	7-17
Figure 7.12: Longitudinal Section Showing the LCBS – West-East Fence Diagram – Copperwood and Satellite Deposits	7-18
Figure 7.13: Strip Log Showing Typical Distribution of Copper (red) and Silver (blue) in the LCBS	7-21
Figure 7.14: Comparison of Copperwood and White Pine North Stratigraphy	7-25
Figure 10.1: Plan View of the Historical Drilling (2017/18 Highland drilling excluded)	10-3
Figure 10.2: Winter Drilling at the Section 5 Area	10-4
Figure 10.3: Core Saw Station at White Pine Site	10-7
Figure 10.4: Core Logging at White Pine Site	10-8

Figure 10.5: Core Photography Setup at White Pine Site	10-8
Figure 10.6: Point Load Testing (Bemek Rock tester)	10-9
Figure 10.7: Specific Gravity Station.....	10-9
Figure 10.8: Wrapped Metallurgical Core Samples from Wedge	10-10
Figure 10.9: Top of LCBS Showing Marked Intervals for Assay Sampling	10-10
Figure 10.10: Bottom of UCBS Showing Marked Intervals for Assay Sampling.....	10-11
Figure 11.1: Highland-inserted Blank Material Analytical Results (coarse CRM) for Cu (top) and Ag (bottom)	11-3
Figure 11.2: Internal Laboratory Blank Material Analytical Results for Copper and Silver	11-4
Figure 11.3: Core Duplicate Performance for Cu (top) and Ag (bottom)	11-6
Figure 11.4: Crush Duplicate Performance for Cu (top) and Ag (bottom)	11-7
Figure 11.5: Analytical Pulp Performance for Cu (top) and Ag (bottom)	11-8
Figure 11.6: Performance of Control Reference Material OREAS 162 for Cu (top) and Ag (bottom) ...	11-10
Figure 11.7: Performance of Control Reference Material OREAS 97 for Cu (top) and Ag (bottom)	11-11
Figure 11.8: Performance of Control Reference Material OREAS 930 for Cu (top) and Ag (bottom) ...	11-12
Figure 11.9: Performance of Control Reference Material CCU-1D for Cu (top) and Ag (bottom)	11-14
Figure 11.10: Performance of Control Reference Material CZN-4 for Cu (top) and Ag (bottom)	11-15
Figure 11.11: Performance of Control Reference Material MP-1b for Cu (top) and Ag (bottom)	11-16
Figure 11.12: Performance of Control Reference Material 14P for Cu.....	11-17
Figure 12.1: Drill Hole Collar Example in Section 6 - CW17-195	12-5
Figure 12.2: Drill Hole Collar Example from Section 5 - CW-17-184.....	12-6
Figure 13.1: Main Zone Sample Location	13-3
Figure 13.2: Zone 6 Sample Locations	13-4
Figure 13.3: CBS 2 Sample Locations	13-6
Figure 13.4: Locked Cycle Flowsheet.....	13-8
Figure 13.5: Main Zone Samples Location Map	13-13
Figure 13.6: Bond Ball Mill Work Index Database	13-16
Figure 13.7: Bond Mill Work Index Comparison	13-17
Figure 13.8: Bond Abrasion Index Comparison	13-18
Figure 13.9: Zones 5 and 6 Samples Location Map	13-22
Figure 13.10: Bond Ball Mill Work Index Comparison	13-23
Figure 13.11: Bond Abrasion Index Comparison	13-24
Figure 13.12: Sample Location Main Zone (in blue).....	13-27
Figure 13.13: LCT Samples Drill Holes Map.....	13-29
Figure 13.14: Batch Rougher Flotation Tests – Mass Pull vs. Recovery	13-33
Figure 13.15: Batch Cleaner Flotation Test Matrix	13-35
Figure 13.16: Modified Locked Cycle Testwork No.8 Block Flow Diagram	13-39

Figure 13.17: Locked Cycle Testwork Results over the Copperwood Deposit.....	13-42
Figure 13.18: QEMSCAN Assay Reconciliation	13-43
Figure 13.19: Copper Sulphides Association.....	13-46
Figure 13.20: Silicates Associations	13-47
Figure 13.21: Elemental Department of Copper	13-48
Figure 13.22: Chalcocite Recover vs. Particle Size	13-49
Figure 13.23: Flowsheet Comparison between Design and Final Located Cycle Test	13-52
Figure 14.1: Drill Status Plan as of April 12 th , 2018	14-3
Figure 14.2: Orthogonal View (looking NE) Showing the Thrust Fault in Yellow.....	14-5
Figure 14.3: Section 270375 mE showing displacement of the UCBS (vertical exaggeration x 3).....	14-6
Figure 14.4: Modelling of the Stratigraphy and Associated Rock Codes	14-7
Figure 14.5: Drill Holes PC-16 and Subsequent Diversion of the LCBS Interpretation	14-8
Figure 14.6: Overlaid Cumulative Probability Graphs of Cu % Raw Assays for units of the LCBS (left) and the UCBS (right).....	14-10
Figure 14.7: Overlaid Cumulative Probability Graphs of Ag g/t Raw Assays for units of the LCBS (left) and the UCBS (right).....	14-11
Figure 14.8: Contact Analysis Plots of the Basal Contact of the Domino (upper image) and Basal Contact of the Red Laminated (lower image).....	14-12
Figure 14.9: Overlaid Cumulative Probability Graphs of Cu % Composites for Units of the LCBS (left) and the UCBS (right).....	14-15
Figure 14.10 Overlaid Cumulative Probability Graphs of Ag g/t Composites for Units of the LCBS (left) and the UCBS (right).....	14-15
Figure 14.11: Composites from the Domino Unit Colored by Ag with Leasing Outlines. Note the Sub-population in the Northern Area (within the sparse drilling)	14-16
Figure 14.12: Variogram Model Cu% for the Grey Laminated subunit of the LCBS	14-19
Figure 14.13: Interpolation Passes with April 2018 Mineral Resource Categories – Copperwood Deposit	14-28
Figure 14.14: Resource Categories - Copperwood Project – April 2018.....	14-29
Figure 14.15: Resource Categories - Copperwood Project – October 2017	14-29
Figure 14.16: Quantile: Quantile Plots of Cu % distributions for the Domino (23) and Grey Laminated (25) Subunits of the LCBS.....	14-32
Figure 14.17: Quantile:Quantile plots of Ag g/t Distributions for the Domino (23) and Grey Laminated (25) Subunits of the LCBS.....	14-33
Figure 14.18: Swath Plot of Cu % for the Domino (23) by Easting.....	14-34
Figure 14.19: Copper Grade Distribution (diluted to 2 m) in the LCBS with Mineral Resource Classification	14-35

Figure 14.20: Copper Grade Distribution in the UCBS (2 m minimum thickness) with Mineral Resource Classification	14-36
Figure 14.21: Grade-Tonnage Curve of Measured + Indicated Resources for the LCBS at the Copperwood Deposit	14-38
Figure 14.22: Grade-Tonnage Curve of Inferred Resources for the LCBS at the Copperwood Deposit ..	14-38
Figure 14.23: Grade-Tonnage Curve of Inferred Resources for the LCBS at the Satellite Deposits	14-39
Figure 14.24: Grade-Tonnage Curve of Measured + Indicated Resources for the UCBS at the Copperwood Deposit	14-41
Figure 14.25: Grade-Tonnage Curve of Inferred Resources for the UCBS at the Copperwood Deposit..	14-41
Figure 14.26: Grade-Tonnage Curve of Inferred Resources for the UCBS at the Satellite Deposits....	14-42
Figure 15.1: Conversion from Resources M+1 to Mining Reserves Proven and Probable	15-3
Figure 15.2: Economic Outline (orange line)	15-4
Figure 15.3: Typical Room and Pillar Shape (maximum floor slope <6°)	15-7
Figure 15.4: Minimum Mining Height	15-8
Figure 16.1: Mine Configuration.....	16-2
Figure 16.2: Idealized Pillar Stratigraphy	16-6
Figure 16.3: Domino 2018 Laboratory Strength Testing Results.....	16-8
Figure 16.4: Basal Gouge 2018 Laboratory Strength Testing Results	16-11
Figure 16.5: Development Design	16-17
Figure 16.6: Panel Division – Western Section.....	16-19
Figure 16.7: Panel Division – Eastern Section.....	16-20
Figure 16.8: Room and Pillar Stope Configuration	16-22
Figure 16.9: Particle Size Distribution Red Massive with ANFO Explosives	16-23
Figure 16.10: Round Cycle Time	16-24
Figure 16.11: Drift, Room and Pillar before Recovery	16-25
Figure 16.12: Drift, Room and Pillar after Recovery	16-26
Figure 16.13: Pre-production Development Period	16-29
Figure 16.14: Panel Sequence	16-34
Figure 16.15: Ventilation Layout during Production Period (western side).....	16-40
Figure 16.16: Ventilation Layout during Production Period (eastern side)	16-41
Figure 16.17: Dewatering Circuit	16-43
Figure 16.18: Pumping Requirement over LoM in L/min Operating @ 100%	16-44
Figure 17.1: Overall Process Flow Diagram	17-3
Figure 18.1: Copperwood Project Site General Arrangement	18-2
Figure 18.2: Copperwood Project Plant Site Area General Arrangement	18-3

Figure 18.3: Main Road Cross-section	18-5
Figure 18.4: Box-cut Design.....	18-7
Figure 18.5: Water Balance Schematic	18-9
Figure 18.6: Process Flow Diagram of Water Treatment Plant	18-12
Figure 18.7: Gatehouse	18-14
Figure 18.8: Maintenance Shop and Warehouse	18-15
Figure 18.9: Plan View – Dry	18-16
Figure 18.10: Transload Building Cross-section	18-18
Figure 18.11: Transload Building Plan View	18-19
Figure 18.12: Tailings Disposal Facility – Stage 1	18-22
Figure 18.13: Tailings Disposal Facility – Stage 2	18-23
Figure 18.14: Tailings Disposal Facility – Stage 3	18-24
Figure 18.15: Embankment and Basin Details (1 of 2)	18-25
Figure 18.16: Embankment and Basin Details (2 of 2)	18-26
Figure 19.1: 10-year Historical Copper Prices	19-2
Figure 19.2: 10-year Historical Silver Prices	19-2
Figure 19.3: CN Rail – Ashland Sub Line	19-6
Figure 20.1: Proposed Mining Area Boundary.....	20-2
Figure 21.1: Operating Cost per lb of Payable Copper.....	21-18
Figure 22.1: LoM Payable Metal Profile	22-4
Figure 22.2: After-Tax Annual Project Cash Flow	22-11
Figure 22.3: After Tax Project Cash Flow Waterfall.....	22-12
Figure 22.4: After-Tax NPV8% Sensitivity	22-16
Figure 22.5: After-Tax IRR Sensitivity.....	22-16
Figure 24.1: Project Team Organization for Engineering and Construction Phases	24-4
Figure 24.2: Engineering Process.....	24-5
Figure 24.3: Project Level 1 Schedule	24-12

List of Tables

Table 1.1: Summary of Copperwood Exploration Activity	1-5
Table 1.2: Key Process Design Criteria	1-10
Table 1.3: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade April 30 th , 2018	1-12
Table 1.4: Mineral Reserves Estimate - Copperwood Project	1-13
Table 1.5: Pillar Size Recommendations	1-16
Table 1.6: Mine Production Schedule Summary	1-20
Table 1.7: Mill Production Schedule Summary	1-21
Table 1.8: Metal Price Assumptions	1-26
Table 1.9: Concentrate Marketing Assumptions	1-27
Table 1.10: Initial Capital Expenditure Summary	1-29
Table 1.11: Sustaining Capital Expenditure Summary	1-29
Table 1.12: Operating Cost Summary	1-30
Table 1.13: Project Risks and Opportunities	1-33
Table 2.1: Summary of Qualified Persons	2-4
Table 2.2: List of Main Abbreviations	2-5
Table 4.1: Copperwood Mineral Tenure	4-3
Table 4.2: KLA and Sage 2008 Mining Lease Payment Schedules	4-6
Table 4.3: Chesbrough 2010 Mining Lease Payment Schedule	4-6
Table 4.4: KLA 2017 Mining Lease Payment Schedule	4-7
Table 4.5: Payment Schedule on Option to Lease Agreement	4-8
Table 6.1: Summary of Copperwood Exploration Activity	6-1
Table 6.2: USMR and AMAX Historical Resource Estimates for Copperwood	6-6
Table 6.3: AMEC Historical Resource Estimates for Copperwood Deposit	6-7
Table 6.4: Marston 2011 Historical Mineral Resource Estimate Presented by Area	6-8
Table 6.5: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade – April 15, 2015	6-9
Table 6.6: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade - Oct. 2017	6-11
Table 10.1: Drilling Statistics by Company and Exploration Campaign	10-2
Table 10.2: Specific Gravity Summary for the LCBS	10-12
Table 10.3: Specific Gravity Summary for the UCBS	10-12
Table 11.1: Overview of QA/QC Sampling	11-2
Table 12.1: Drill Holes Available in the Database for Resource Estimation	12-2
Table 12.2: Drill Holes Randomly Selected from the Database for Lithology Validation	12-4
Table 13.1: Composites No.4 and No.5 Assays	13-2

Table 13.2: Details of Composites No. 4 and Composite No.5	13-2
Table 13.3: LCT Composite No.4 Results	13-4
Table 13.4: LCT Composite No.5 Results	13-5
Table 13.5: NaHS Dosage Series Rougher Flotation Test Results Summary	13-7
Table 13.6: CBS2 Locked Cycle Flotation Test Results Summary.....	13-9
Table 13.7: Grindability Test Summary.....	13-10
Table 13.8: Grindability Test Statistics.....	13-11
Table 13.9: Sample Preparation Information	13-12
Table 13.10: Sample Received Weights and Testing Matrix	13-14
Table 13.11: JK Drop Weight and SMC Test Results Summary	13-14
Table 13.12: Bond Rod Mill Grindability Test Results Summary	13-15
Table 13.13: Bond Ball Mill Grindability Test Results Summary.....	13-17
Table 13.14: Bond Abrasion Test Results Summary	13-18
Table 13.15: Grindability Results Summary for Zones 5 and 6	13-19
Table 13.16: Zones 5 and 6 Test Matrix	13-20
Table 13.17: Sample Inventory and Preparation Summary.....	13-21
Table 13.18: SMC Test Results Summary.....	13-22
Table 13.19: Bond Ball Mill Grindability Test Results Summary.....	13-23
Table 13.20: Bond Abrasion Test Results Summary	13-24
Table 13.21: Metso Grinding Testwork Results	13-25
Table 13.22: Main Zone Sample Preparation Summary.....	13-27
Table 13.23: Flotation Sample Grinding	13-28
Table 13.24: Overall Drill Holes	13-28
Table 13.25: Heady Assay for Main Zone Composites	13-30
Table 13.26: Batch Rougher Flotation Test Matrix	13-32
Table 13.27: Batch Cleaner Flotation Tests – Copper Recovery vs. Concentrate Grade	13-37
Table 13.28: Overall Locked Cycle Testwork Results	13-40
Table 13.29: Locked Cycle Testwork Results by Zone.....	13-41
Table 13.30: Model Mineral Concentrations for Test F39R.....	13-44
Table 13.31: Mean Grain Size by Frequency (micron) for Test F39R.....	13-45
Table 13.32: Elemental Department of Copper	13-48
Table 13.33: Copper Concentrate Specification	13-50
Table 13.34: Key Process Design Criteria	13-54
Table 14.1: Legacy Drill Holes by Company.....	14-2
Table 14.2: Average Vertical Thicknesses of the LCBS Units.....	14-7
Table 14.3: Length-weighted Statistics of the Copper Raw-Assays.....	14-9
Table 14.4: Length-weighted Statistics of the Silver Raw-Assays.....	14-10

Table 14.5: Statistics of the Copper Composites	14-14
Table 14.6: Statistics of the Silver Composites	14-14
Table 14.7: Statistics of the Specific Gravity Measurements Presented by Years of Collection for the LCBS	14-17
Table 14.8: Statistics of the Specific Gravity Measurements Presented by Lithology	14-17
Table 14.9: Specific Gravity Averages Used in the Resource Estimation	14-17
Table 14.10: Variogram Models for the Copper and Silver Composites of Zone	14-18
Table 14.11: Block Model Parameters - Copperwood Project.....	14-20
Table 14.12: Rock Codes Used in the Rock Type Model	14-21
Table 14.13: List of Attributes Found in the Block Model	14-22
Table 14.14: Interpolation Profile Settings for Resource Estimation - Copperwood Project	14-24
Table 14.15: Sample Search Ellipsoid Settings for Resource Estimation - Copperwood Project	14-25
Table 14.16: Comparative Statistics for Cu (%) Between Composites and Blocks Grouped by Rock Code	14-31
Table 14.17: Comparative Statistics for Ag (g/t) Between Composites and Blocks Grouped by Rock Code	14-31
Table 14.18: LCBS Constrained Mineral Resource Sensitivity – Measured and Indicated.....	14-37
Table 14.19: LCBS Constrained Mineral Resource Sensitivity - Inferred	14-37
Table 14.20: LCBS Constrained Mineral Resource Sensitivity – Satellite Inferred	14-37
Table 14.21: UCBS Constrained Mineral Resource Sensitivity – Measured and Indicated	14-40
Table 14.22: UCBS Constrained Mineral Resource Sensitivity - Inferred	14-40
Table 14.23: UCBS Constrained Mineral Resource Sensitivity – Satellite Inferred.....	14-40
Table 14.24: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade – April 30 th , 2018	14-44
Table 15.1: Mineral Reserve Estimate - Copperwood Project.....	15-1
Table 15.2: Resource Model versus Mining Model.....	15-2
Table 15.3: Underground Room and Pillar Mining Method.....	15-5
Table 15.4: Mining Recovery and Dilution Summary.....	15-9
Table 16.1: UCS Testing Result Summary	16-7
Table 16.2: Rock Mass Strength Parameters	16-10
Table 16.3: Ground Parameters	16-13
Table 16.4: Pillar Size Recommendations	16-14
Table 16.5: Mine Design Summary.....	16-21
Table 16.6: Stope Pillar Size and Mining Recovery.....	16-21
Table 16.7: Mine Equipment Requirements.....	16-28
Table 16.8: Ramp-up Summary	16-31
Table 16.9: Productivity per Mining Panel	16-32

Table 16.10: Mine Production Schedule Summary	16-33
Table 16.11: Operating Shift Assumptions	16-35
Table 16.12: Production Working Schedule.....	16-37
Table 16.13: Mine Manpower Requirements	16-38
Table 16.14: Mine LoM Manpower Requirements	16-39
Table 16.15: Equipment Water Consumption	16-41
Table 17.1: Key Process Design Criteria	17-4
Table 18.1: TDF Capacity	18-20
Table 19.1: Metal Price Assumptions	19-1
Table 19.2: Concentrate Marketing Assumptions	19-3
Table 19.3: Concentrate Specifications	19-4
Table 19.4: Concentrate Transportation Cost (Mine to Horne Smelter)	19-5
Table 21.1: Capital Expenditures Summary	21-2
Table 21.2: Infrastructures CAPEX.....	21-3
Table 21.3: Power Supply and Electrical Capital Expenditures	21-4
Table 21.4: Tailings & Water Capital Expenditures	21-5
Table 21.5: Mobile Equipment Capital Expenditures	21-6
Table 21.6: Mine Infrastructure Capital Expenditures.....	21-7
Table 21.7: Processing Capital Expenditures	21-9
Table 21.8: Construction Indirect Capitals	21-10
Table 21.9: General Services Expenditures	21-11
Table 21.10: Pre-Production and Commissioning Expenditures	21-12
Table 21.11: Summary of Sustaining Capital Costs	21-13
Table 21.12: Sustaining Capital Costs	21-14
Table 21.13: Closure Cost & Salvage Value.....	21-16
Table 21.14: LoM Operating Cost Summary	21-17
Table 21.15: First 5-year Operating Cost Summary	21-17
Table 21.16: Annual Operating Costs	21-18
Table 21.17: Mining Operating Cost Summary	21-19
Table 21.18: Annual LoM Mining OPEX	21-20
Table 21.19: Process Operating Cost Summary	21-21
Table 21.20: Annual LoM Processing OPEX	21-22
Table 21.21: Process Plant Reagent Consumption	21-23
Table 21.22: Grinding Media and Liner Consumption	21-23
Table 21.23: Mill Power Consumption by Area.....	21-24
Table 21.24: General Management and Administration Cost Summary	21-25
Table 21.25: Annual LoM G&A OPEX	21-26

Table 22.1: Off-Highway Diesel Fuel Price Assumption	22-2
Table 22.2: Exchange Rate Assumptions	22-2
Table 22.3: Metal Production	22-3
Table 22.4: Mill Commissioning and Ramp-up	22-5
Table 22.5: Initial Capital Expenditure Summary	22-6
Table 22.6: Sustaining Capital Expenditure Summary	22-6
Table 22.7: Closure and Reclamation Cost Estimate by Stage	22-7
Table 22.8: Operating Cost & Summary	22-8
Table 22.9: Life-of-Mine C1 & C3 Cost Summary	22-8
Table 22.10: First 5-Year C1 & C3 Cost Summary	22-9
Table 22.11: Economic Results Summary	22-11
Table 22.12: After-Tax Annual Cash Flow Summary	22-13
Table 22.13: Pre-Tax Sensitivity Results	22-14
Table 22.14: After-Tax Sensitivity Results	22-15
Table 25.1: Copperwood Project Risks	25-9
Table 25.2: Copperwood Project Opportunities	25-11
Table 26.1: Recommended Work Programs	26-2

1. SUMMARY

1.1 Introduction

G Mining Services Inc. (“GMSI”) and other engineering consultants were retained by Highland Copper Company Inc. (“Highland” or the “Company”) to produce a Feasibility Study (the “FS” or “Study”) for its Copperwood Project located in the western Upper Peninsula of Michigan, USA and to prepare a technical report (the “Report”) in accordance with Canadian National Instrument 43-101 (“NI 43-101”) *Standards of Disclosure for Mineral Projects* to support the results of the FS as disclosed in Highland’s press release entitled “*Highland Copper Announces Positive Feasibility Study Results for its Copperwood Project in Michigan*” dated June 15, 2018.

The major contributors for the Study and the Report and their respective areas of responsibility are as follows:

- GMSI – overall Report and FS coordination, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, mineral resource estimates, mineral reserves, mining methods, economic analysis, operating costs, infrastructure, power supply, capital cost estimate and project execution plan;
- SGS Canada Inc. (Lakefield) (“SGS”) – mineral test work;
- Lycopodium Limited (“Lyco”) – flow sheet, mass balance, recovery methods, mineral process plant design and input to operating and capital cost estimates for the process plant;
- Golder Associates Ltd. (“Golder”) – rock mechanics and underground geotechnical assessment, water balance, water treatment design, and tailings disposal facility (“TDF”) design;
- Foth Infrastructure & Environment (“Foth”) – environmental, permitting and social aspects.

1.2 Reliance on Other Experts

Certain sections of this Report rely on reports and statements from legal and technical experts who are not Qualified Persons (“QP”) as defined by NI 43-101. The QPs responsible for the preparation of this Report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound and are acceptable for use in this Report.

1.3 Property Description and Location

The Copperwood Project is located within Gogebic County near Ironwood and Wakefield townships in northwestern Michigan, USA.

The Copperwood Project comprises the Copperwood Deposit and the Satellite Deposits. The Copperwood Deposit includes three zones referred to in this Report as the Main Zone, the Section 5 (or Zone 5) and the Section 6 (or Zone 6).

The Copperwood Project consists of four metallic and non-metallic mineral leases totaling 1,904 contiguous hectares under two 20-year lease agreements with Keweenaw Minerals, LLC (formerly Keweenaw Land Association Limited) ("KLA"), a 20-year lease agreement with Sage Minerals Inc. ("Sage") and a 30-year mineral lease agreement with A. M. Chesbrough LLC ("Chesbrough"). Each lease was executed by Copperwood Resources Inc. ("CRI"), formerly known as Orvana Resources US Corp., a wholly-owned subsidiary of Highland.

In addition to annual lease payments, CRI must pay a sliding scale net smelter return royalty on production from its leases to the mineral right owners (KLA, Sage and Chesbrough). The royalty rate ranges from 2% to 4% on a sliding scale based on adjusted copper prices.

Moreover, Osisko Gold Royalties Ltd ("Osisko") has acquired a 3.0% net smelter return royalty on all metals produced from the Copperwood Project provided, however, that upon final closing of the acquisition by Highland of the White Pine Project, Highland will grant Osisko a 1.5% NSR royalty on all metals from the White Pine North Project and Osisko's royalty on the Copperwood Project will be reduced to 1.5%.

CRI owns approximately 700 ha of land that provides full access rights to the Copperwood Project and provides surface infrastructure space for the future mine site.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.4.1 Accessibility

The Copperwood Project property is located approximately 22.5 km by road to the north of the town of Wakefield in Gogebic County, Michigan, and is also located approximately 40 km by road from the town of Ironwood, also in Gogebic County. Wakefield and Ironwood have populations of 2,300 and 6,800 respectively.

The main access to the Copperwood Project property is by way of the paved north-south County Road 519, which branches off State Highway M-28 just east of Wakefield. Future mining activities at the Copperwood Project will require an upgrade of the paved County Road 519 to an all-season level and an upgrade of the dirt road from County Road 519 to the Copperwood site.

1.4.2 Climate

The Copperwood Project property is situated immediately south of the Lake Superior shoreline where the local climate consists of four seasons typical of mid-latitude temperate climates. The annual precipitation is approximately 890 mm of rain equivalent (rain and snow) with the greatest monthly precipitation of about 100 mm. Mean annual total snowfall is approximately 4.5 m with the maximum monthly mean snow depth of about 0.6 m.

1.4.3 Local Resources

The workforce for any current and future mining activity could be sourced from a combination of the local area or from external areas. Unemployment is high in Gogebic County; both skilled and unskilled labour forces are available for work.

1.4.4 Existing Infrastructure

The only infrastructure on the Copperwood Project property is a network of dirt roads, logging roads and trails. The main dirt roads are in good condition.

There is an 88 kilovolt ("kV") power line located 18 km from the Copperwood Project; however, this is a unique voltage that may be obsolete before long. Xcel Energy owns the nearest transmission lines, which are located approximately 32 km south of the property.

Onsite power generation is also an option. Natural gas is available from two major pipeline companies; TransCanada through their Great Lake Gas Transmission ("GLGT") subsidiary and Northern Natural Gas ("NNG").

1.4.5 Physiography

The land surface at the Copperwood Project property slopes northwest toward the Lake Superior shoreline. The ground surface elevation along the southern edge of the site is approximately 288 mamsl as compared to the approximate elevation of 198 mamsl at the top of the bluff along the Lake Superior shoreline.

1.5 History

Exploration history on Copperwood dates back to 1954.

Several historical resource estimates for the Copperwood deposit have been issued:

- USMR – Covering larger area that included the Copperwood Project area, prepared in 1959;
- AMAX – Covering larger area that included the Copperwood Project area, prepared in 1974;
- Orvana Minerals (AMEC) – Copperwood area, published April 2010, effective date of April 30, 2010;
- Orvana Minerals (AMEC) – Satellite Deposits, published January 2011, effective date of January 24, 2011;
- Orvana Minerals (Marston) – Copperwood areas, published March 2011, effective date of January 25, 2011;
- Highland (GMSI) – Copperwood Deposit, published June 25, 2015, effective date of April 15, 2015;
- Highland (GMSI) – Copperwood Deposit, published December 5, 2017, effective date of October 18, 2017.

Table 1.1 summarizes the history of exploration completed in the Copperwood area.

Table 1.1: Summary of Copperwood Exploration Activity

Company	Activity	Year
U.S. Geological Survey	Economic geology publication demonstrates potential of Western Syncline.	1954
USMR	Leased 1,552 ha in Western Syncline area (Cox, 2003).	1956
USMR	Drilled 26 holes focused on margin of Western Syncline and discovered Copperwood.	1956
USMR	Drilled 135 holes throughout the Western Syncline.	1958
AMAX	Sank 71 m vertical exploration shaft and advanced 635 m of exploration drifts, including three small stopes.	1957-1958
BCR	Drilled 23 holes in the Satellite deposits. BCR terminated leases in the early 1960's.	1959
AMAX	Internal engineering and economic study that ended activities by USMR.	1959
AMAX	Engineering and economic review concluded deposit was mineable.	1974
AMAX	Terminated Western Syncline leases.	1983
Orvana	Leased 712 ha at Copperwood and options 1,559 ha in Western Syncline.	2008
Orvana	Began environmental studies with five drill holes intersecting copper mineralization.	2008
Orvana	Drilled 82 holes.	2009
Orvana	Leased 229 ha covering Section 6.	2010
Orvana	Drilled 38 holes. Completed Mineral Resource estimate.	2010
Orvana	Completed Mineral Resource estimate.	2011
Orvana	Completed Prefeasibility Study.	2011
Orvana	Completed Feasibility Study.	2012
Orvana	Mining Permit Approved by Michigan Department of Environmental Quality.	2012
Orvana	Drilled 21 holes for metallurgical and geotechnical studies.	2013
Highland	Drilled 40 holes and 13 wedges for resource estimate, metallurgical and geotechnical studies.	2017
Highland	Drilled 8 holes and 1 wedge as infill for Feasibility Study.	2018

1.6 Geological Setting

The Copperwood Project is situated on the flank of the 2,200 km long Mesoproterozoic mid-continent rift system of North America and is hosted in the Nonesuch Formation; a package of lacustrine and fluvial sediments, which form part of the Oronto Group post-rifting basin fill. Mineralization is hosted within two sedimentary sequences termed the Lower Copper Bearing Sequence ("LCBS") and Upper Copper Bearing Sequence ("UCBS") at the base of the Nonesuch Formation.

1.7 Mineralization

The LCBS is composed of the Domino, Red Massive and the Gray Laminated units. The Domino unit is the principal copper host at Copperwood and is characterized by black shale with a mean thickness of 1.6 m. The Red Massive sub-unit comprises siltstone to sandstone and has a mean thickness of 0.3 m. The Gray Laminated sub-unit is a gray laminated siltstone and has a mean thickness of 1.0 m.

The UCBS is composed of the Upper Transition, Thinly, Brown Massive and Upper Zone of Values units. The Upper Transition unit comprises thinly bedded siltstone to sandstone and black shale with a mean thickness of 1.0 m. The Thinly unit is characterized by black shale with a mean thickness of 0.1 m. The Brown Massive unit is characterized by a brownish-red siltstone with a mean thickness of 1.1 m and one- to two-centimeter thick calcareous nodules. The Upper Zone of Values unit is composed of laminated, greenish black, shaley siltstone with a mean thickness of 0.5 m. The UCBS is separated from the LCBS by thinly to medium-bedded red siltstone, grey siltstone, and sandstone. The thickness between the UCBS and the LCBS gradually decreases from 6.0 m in the western part of the Deposit to 0.5 m in the eastern part of the Deposit.

The LCBS and UCBS at Copperwood have been delineated by drilling over an area of approximately 5,600 m east-west and 1,700 m north-south. The Copperwood and Satellite Deposits are hosted within the limbs of the broad, gently northwest-plunging Presque Isle Syncline. The LCBS dips gently and subcrops beneath 20 to 35 m of unconsolidated glacial sediments along the southern edge of the Copperwood Project area.

1.8 Deposit Types

The mineralization at Copperwood has been interpreted as a sediment-hosted stratiform copper deposit of the reduced facies class. Well known reduced-facies sediment-hosted stratiform copper deposits include most of the deposits within the Central African Copperbelt and the Kupferschiefer (Poland and Germany), Redstone (Canada) and nearby White Pine (Michigan).

Sediment-hosted stratiform copper deposits consist of copper and copper-iron sulphide minerals hosted by siliciclastic or dolomitic rocks in which a relatively thin copper-bearing zone is mostly conformable with stratification of the host sedimentary rocks. Copper in chalcocite occurs as disseminations and seams along bedding planes. Chalcocite is the only observed copper sulphide bearing mineral present at Copperwood.

1.9 Exploration

Historical exploration at Copperwood has been completed through surface drilling programs conducted in 1956, 1957, 1959, 2008, 2009, 2010, 2011, 2013 and 2017. In 1958, AMAX sunk an exploration shaft and completed test mining from a 620 m exploration drift.

To date, there have been no surface geochemical exploration programs nor have there been any surface or airborne geophysical exploration programs conducted on the Copperwood Project.

Historical exploration drilling on the Copperwood Project property and surrounding leases was completed during two separate phases of activity; the first phase by USMR and Bear Creek Mining (“BCM”) was performed from 1956 to 1959, while the second phase was performed by Orvana Minerals Corp. (“Orvana”) starting in 2008 and completed in 2013.

Between 1956 and 1959, USMR and BCM drilled 184 core holes in the Western Syncline area. Some 96 of these drill holes were drilled in the Copperwood Deposit area. USMR drilled 42 holes in the “Main” area and 31 holes in Section 5 from 1956 to 1958. BMC drilled 23 holes in Section 6 in 1959. USMR drilled 88 drill holes in the Satellite Deposits from 1956 to 1957. The core diameter for these holes was between 3.01 cm (AX size core) and 4.20 cm (BX size core).

The second phase of drilling at Copperwood commenced in 2008, with Orvana US drilling five core holes for environmental purposes. These drill holes intersected significant copper mineralization. Orvana subsequently completed 82 drill holes in 2009. Orvana US drilled 24 additional core holes during 2010 to firm up the resource, to collect metallurgical and geotechnical data and to investigate a suspected fault. Another 15 core holes were drilled during 2010 to verify copper mineralization in the Section 6 area. In 2013, Orvana drilled 21 core holes to collect additional metallurgical and geotechnical data. The core diameter for the Orvana drill holes was 4.80 cm (NQ size core) for the 2008 to 2010 drilling and 6.35 cm (HQ size core) for the 2013 drilling program.

The third phase of drilling at Copperwood was by Highland, where 35 HQ diameter (plus 13 wedges) and five PQ-diameter drill holes for a total of 7,666 m of core were drilled in 2017. This drilling was to upgrade Mineral Resources in Sections 5 and 6, and to provide samples for metallurgical studies. In 2018, Highland

completed a drilling program of eight NQ-diameter holes and one wedge as well as finishing one HQ-diameter hole which was collared before abandoning during spring break-up in 2017. This drilling was designed to upgrade Mineral Resources in Section 5.

1.10 Drilling

Only diamond drilling has been conducted at Copperwood, with drill core diameters varying from 45 mm to 85 mm. Historical drilling in the 1950's was undertaken using AX or BX drill rod sizes, with later drilling by Highland and Orvana using NQ, HQ or PQ drill rods sizes depending on the purpose of the drilling (infill resources, extensional resources, metallurgy, etc.). Drilling is usually undertaken in winter to minimise environmental impacts and to facilitate access. Core recovery is considered excellent, with minimal core-loss observed.

A Highland geologist supervised the extraction of the mineralized intervals from the drill casing to ensure recovery and correct orientation during boxing. Each core box containing the mineralized core was sealed with shrink wrap and a sticker initialed by the driller's helper and the on-site geologist. A chain of custody form for the mineralized core boxes was filled out with a signature from the driller. Core boxes were immediately transported by the geologist via pick-up truck to a secured building in White Pine.

Sampling by Highland comprised half and quarter-split core samples collected from the 2017 and 2018 surface diamond-drill program. Sample intervals were variable and honoured logged lithologic intervals. Extensive specific gravity measurements and core recovery observations and measurements were collected.

Activation Laboratories Ltd. ("Actlab") in Thunder Bay, Ontario, Canada was used as the primary laboratory for the final preparation of samples and assays for the Highland program. Actlab is accredited by the Standards Council of Canada and conforms to requirements of CAN P 1579 (ISO/IEC 17025:2005). Accreditation includes the analytical procedures used for the samples.

All samples for geotechnical and metallurgical testing were shipped to specialized laboratories. For an improved understanding of the ore geotechnical characteristics, 19 holes were televised and subsequently cemented.

GMSI reviewed all available QA/QC data (standards, blanks, field duplicates, check assays) and found no significant issues. Highland uses an external database consultant which employs rigorous QA/QC protocols to ensure database integrity.

1.11 Data Verification

GMSI has reviewed the available data used in the Mineral Resource estimate, including drill logs, assay certificates, downhole surveys, and additional information sources. Approximately 50% of the entire assay database was investigated against the original assay certificates for possible typographical errors, wrong sample numbers or duplicates in 2015. Additionally, 76 drill holes were randomly selected to compare with original lithological logs. Very few minor errors were found in less than half of a percent of the data investigated. Drill hole collars from 2017 were visited, and drill core was viewed during November 2017 by the GMSI QP and Highland representatives. Assay certificates from the 2018 drilling campaign were checked against the database to ensure accuracy. GMSI's QP is of the opinion that the drill hole database is in good condition and could be used with confidence in the Mineral Resource estimate.

1.12 Mineral Processing and Metallurgical Testing

Comprehensive metallurgical testwork programs have been done on Copperwood ores over the years with variable results. During the last testwork program in 2017 and 2018, the main objective was to evaluate the process performance selected in the 2012 Feasibility Study and to improve the performance and verify the variability of the ore over the deposit.

Alternative reagents were tested but finally the reagents used in the METCON testwork appeared to deliver better performance for the samples processed. However, modification to the process flowsheet. grind size target combined with modified reagents additions and dosage delivered better performance.

The major modifications consisted of finer primary grind (40 microns), finer regrind (15 microns). Recirculation of the first cleaner scavenger concentrate to regrind and recirculation of the first cleaner tailings to rougher scavenger. The flotation time for most circuits increased which will require further investigation in a next testwork program. Closing the first cleaner circuit with recirculation of the first cleaner scavenger concentrate to regrind with the same conditions appeared to increase the copper recovery by 3%.

The primary observation of variability testwork showed that the copper recovery varies from 77% up to ~ 90% with a concentrate grade from 20% up to 29% Cu. The overall average Cu recovery was at 86% with an average Cu concentrate grade of 24.5%.

The key process design criteria listed in Table 1.2 form the basis of the detailed process design criteria and mechanical equipment list. The design criteria were selected based on the best information available at the

time of completion of the Study and will have to be adjusted during detailed engineering based on the final testwork results.

Table 1.2: Key Process Design Criteria

Parameter	Units	Value	Value LCT-8	Source
Plant Throughput	mtpd	6,600	-	Highland
Head Grade - LoM	% Cu	1.35		Highland
	g/t Ag	3.41		Highland
Plant Availability	%	91.3		Lycopodium
Bond Crusher Work Index (CWi)	kWh/t	20.3		Consultant
Bond Ball Mill Work Index (BWi)	kWh/t	16.2		Testwork
SMC Axb ¹		34.5		Consultant
Bond Abrasion Index (Ai)	G	0.014		Testwork
Grind Size (P ₈₀)	µm	45	40-45	Testwork
Rougher Residence Time – Lab	Min	50	75	Testwork
Cleaner 1 Residence Time – Lab	Min	6	20	Testwork
Cleaner 1st Scavenger Residence Time – Lab	Min	10	20	Testwork
Cleaner 2 Residence Time – Lab	Min	5	10	Testwork
Cleaner 3 Residence Time – Lab	Min	3	5	Testwork
Regrind Mill Product Size (P ₈₀)	µm	20	15	Testwork
Concentrate Production Rate	t/h	15.1		Calc
Concentrate Thickener Solids Loading	t/m ² .h	0.20		Lycopodium
Filter Solids Loading	kg/m ² .h	160		Lycopodium

1. Design A x b value derived from the 85th percentile ranking of specific energies determined for each individual ore type.

1.13 Mineral Resources Estimate

The estimate was conducted in a block model characterised by three key units of the LCBS (LCBS: Gray Laminated, Red Massive, and Domino beds) and a single unit representing the UCBS. Lithological solids were built in Leapfrog GEO™ for each unit of the LCBS, and a single unit with a minimum thickness of 2.0 m was created for the UCBS. Hanging wall and footwall dilutions zones were also incorporated into the block model. Uncapped raw assays were composited to produce a single composite per unit, per drill hole. Variography studies highlighted a near horizontally isotropic distribution of copper and a low nugget effect

on copper and silver grades. Block sizes of 20 m x 20 m horizontally, with a 2.5 m height were used in the block model. Bulk density was assigned based on rock type, derived from core measurements. Copper and silver grades were estimated using the Ordinary Kriging ("OK") interpolation method in three successive passes, using ellipse ranges of 175 m, 250 m, and 350 m. Grade estimates were validated using on-section visual comparison, swath plots, Q:Q plots and global descriptive statistics.

To define resource categories, GMSI outlined groups of globally similar interpolation passes. Measured Mineral Resources thus constitute the bulk of the Mineral Resources in the Copperwood deposit area and include blocks interpolated generally in the first pass. Indicated Mineral Resources are located at the periphery of the measured category where blocks are generally interpolated in the second pass. All other interpolated blocks are categorized in the Inferred Mineral Resource category, including all blocks in the Satellite Deposits.

Resources are reported using a cut-off grade of 1.0% Cu, based on an underground "room and pillar" mining scenario. Mineral Resources were classified according to the CIM Definition Standards on Mineral Resources and Mineral Reserves. Grade dilution was applied where the combined thickness of the LCBS was less than 2.0 m, using grades estimated in the hanging wall and footwall.

The Copperwood deposit total underground Measured and Indicated Mineral Resources are reported at 49.3 Mt grading an average 1.54% Cu and 3.76 g Ag/t containing 1.68 billion pounds of copper and 5.9 Moz Ag using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined. Inferred Mineral Resources are reported at 1.6 Mt grading an average 1.18% Cu and 1.55 g Ag/t containing 43 million pounds of copper and 0.1 Moz Ag using a cut-off grade of 1.0% Cu.

The Satellite deposits total underground Inferred Mineral Resources are reported at 49.9 million tonnes grading 1.15% Cu and 3.42 g Ag/t containing 1.27 billion pounds of copper and 5.5 M oz Ag using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined.

Table 1.3 reports Mineral Resources for an underground room-and-pillar mining scenario for the Copperwood and Satellite Deposits by resource categories. All parameters used in the calculations are also presented in the table's notes.

**Table 1.3: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade
April 30th, 2018**

Deposits	Resource Category	Tonnage (Mt)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs)	Silver Contained (M oz)
LCBS	Measured	27.3	1.68	4.58	1,009	4.0
	Indicated	14.9	1.46	2.47	479	1.2
	M + I	42.2	1.60	3.84	1,488	5.2
	Inferred	1.6	1.18	1.55	43	0.1
UCBS	Measured	-	-	-	-	-
	Indicated	7.1	1.21	3.26	189	0.7
	M + I	7.1	1.21	3.26	189	0.7
	Inferred	-	-	-	-	-
Satellite LCBS	Inferred	34.4	1.17	2.29	888	2.5
Satellite UCBS	Inferred	15.5	1.12	5.92	384	3.0

Notes on Mineral Resources:

- 1) Mineral Resources are reported using a copper price of US\$ 3.00/lb and a silver price of US\$ 18/oz.
- 2) A payable rate of 96.5% for copper and 90% for silver was assumed.
- 3) The Copperwood Feasibility Study reported metallurgical testing with recovery of 86% for copper and 73.5% for silver.
- 4) Cut-off grade of 1.0% copper was used, based on an underground "room and pillar" mining scenario.
- 5) Operating costs are based on a processing plant located at the Copperwood site.
- 6) Assuming a \$3.00/lb Cu price, a sliding scale 3.0% NSR royalty on the Copperwood Project is payable to leaseholders. Assuming closing of the acquisition of the White Pine Project, a 3% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd is reduced to a 1.5% NSR royalty.
- 7) Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.
- 8) No mining dilution and mining loss were considered for the Mineral Resources.
- 9) Rock bulk densities are based on rock types.
- 10) Classification of Mineral Resources conforms to CIM definitions.
- 11) The qualified person for the estimate is Mr. Réjean Sirois, P.Eng., Vice President Geology and Resources for GMSI. The estimate has an effective date of 30th April 2018.
- 12) Mineral Resources that are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- 13) LCBS: Lower Copper Bearing Sequence.
- 14) UCBS: Upper Copper Bearing Sequence.
- 15) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.

1.14 Mineral Reserves Estimate

The Mineral Reserves for the Copperwood Project are estimated at 25.4 Mt, at an average grade of 1.43% Cu and 3.83 g/t Ag, as summarized in Table 1.4. The mine design targets mineralization above a 1% copper grade which generates an NSR near the breakeven cost of US\$ 48/t of ore which includes provisions for sustaining capital.

The Mineral Reserve is net of all pillars including those in the mine panels, the Lake Superior 30 m offset, a crown pillar providing for 25 m vertical of rock above openings and a 15 m barrier pillar around the historical test mine openings. A 0.3 m skin of gray laminated is left in place to provide for a more competent back. The Mineral Reserve includes planned dilution and unplanned dilution allowances. The planned dilution consists of imposing a 2.1 m minimum mining height and sloping sections of floor to have a maximum 6° cross slope. The unplanned dilution or overbreak allowance includes 0.25 m in the back and 0.10 m from the floor. The overall mining dilution is estimated at 34.8% with an overall mining recovery of 71% for pillars left between stopes and development headings.

A 3% ore loss is assumed to calculate the final Proven and Probable Mineral Reserve.

Table 1.4: Mineral Reserves Estimate - Copperwood Project

Reserve by Category	Tonnes (Mt)	Cu Grade (%)	Ag Grade (g/t)	Cu contained (M lb)	Ag contained (M oz)
Proven	17.5	1.50	4.43	579.6	2.5
Probable	7.9	1.28	2.5	222.2	0.6
Proven & Probable	25.4	1.43	3.83	801.8	3.1

Notes on Mineral Reserves:

- 1) The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.
- 2) Mineral Reserves are estimated at a cut-off grade of 1% Cu. The cut-off will vary depending on the economic context and the operating parameters.
- 3) Mineral Reserves are estimated using a long-term copper price of US\$ 3.00/lb and a silver price of US\$ 16.00/oz.
- 4) Assuming a \$3.00/lb Cu price, a sliding scale 3.0% NSR royalty on the Copperwood Project is payable to leaseholders. Assuming closing of the acquisition of the White Pine Project, a 3% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd is reduced to a 1.5% NSR royalty.
- 5) Mineral Reserves are estimated using an ore loss of 3%, a dilution of 0.1 m for the floor and a 0.25 m for the back of the stope and the development.
- 6) The economic viability of the mineral reserve has been demonstrated.
- 7) A minimum mining height of 2.1 m was used.
- 8) The copper recovery was estimated at 86%.
- 9) The qualified person for the estimate is Mr. Carl Michaud, Eng., Underground Engineering Manager for GMSI. The estimate has an effective date of May 25, 2018
- 10) The number of metric tonnes was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.

1.15 Mining

1.15.1 Mining Method

The deposit is relatively sub-horizontal with a thickness that varies from 1.6 m to 3.7 m. It is proposed to mine the deposit with a conventional drill and blast room-and-pillar mining method that is highly mechanized.

The method consists of the extraction of a series of entries and cross-cuts in the ore, leaving pillars in place to support the back. The entries, cross cuts and pillars have been sized using geotechnical analysis of the rock, and experience from other mines sharing similar ground conditions.

1.15.2 Mine Access

The mine will be accessed via a covered box-cut to establish a portal at the mine entrance from the surface. From the surface portal, only 2 drifts are excavated, and expand to 4 drifts at a depth of 35 m. The mine consists of 2 mining sectors: West and East. The mine development is designed with four drifts per main access including: fresh air intake drift, ore conveyor drift, hauling drift and return air drift. The main access drifts will be in the ore from the box-cut. The conveyor drift will be reinforced with shotcrete.

The drift width is set at 6.1 m, and the height varies from a minimum of 3 m to a maximum of 6 m. At the intersection of conveyor drifts, the size will be 6 m high to allow the installation of a transfer point between the two conveyors.

Barrier pillars between the main access and the stopes will be kept in place until the stope area is mined out. These barrier pillars are designed to be recovered, but they will respect Golder's recommended pillar size.

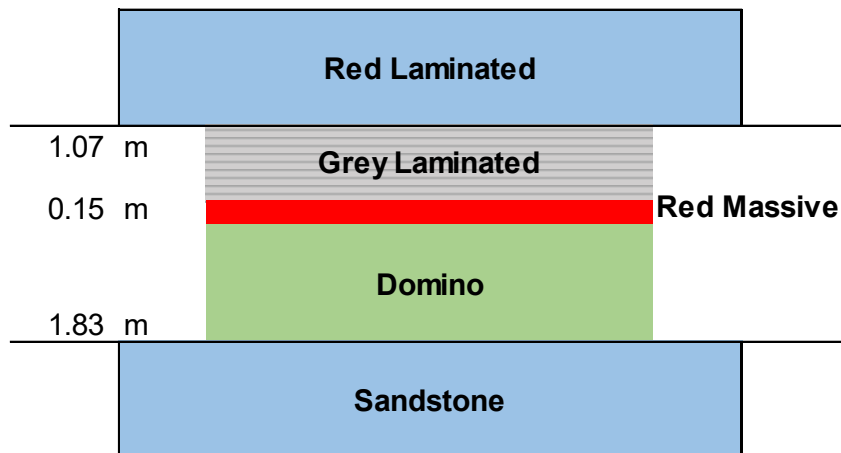
1.15.3 Rock Mechanics and Geotechnical Design Criteria

A detailed geotechnical evaluation of the Copperwood deposit was completed by Golder in 2018 and established many of the mine design criteria, in particular the pillar design which affects the mine recovery factor.

The strength of the pillars is governed by the strength and behaviour of the geological units in the pillars and in the immediate roof. The conceptualized stratigraphy in the ore and surrounding rock mass is presented in Figure 1.1. The mining column, referred to as the LCBS, consists of three bedding units

referred to as the Domino, Red Massive and Grey Laminated. The Red Massive unit is thin with low-grade generally below the cut-off grade but is mined as internal dilution in the mining column. The Domino unit in the footwall is the higher-grade seam and lies above a competent sandstone. The Grey Laminated is of medium grade and lies beneath a Red Laminated unit that would form part of the roof or back.

Figure 1.1: Mining Column and Pillar Stratigraphy



Golder supplemented available geotechnical data with additional investigations in 2017 consisting of geotechnical drilling which included vertical and inclined drill holes to collect structural data as well as core samples for characterization and laboratory testing. The pillar dimensioning based on numerical modeling is summarized in Table 1.5. The pillar dimensions are specific to the East and West mine where square pillar dimensions are a function of depth from surface and room height.

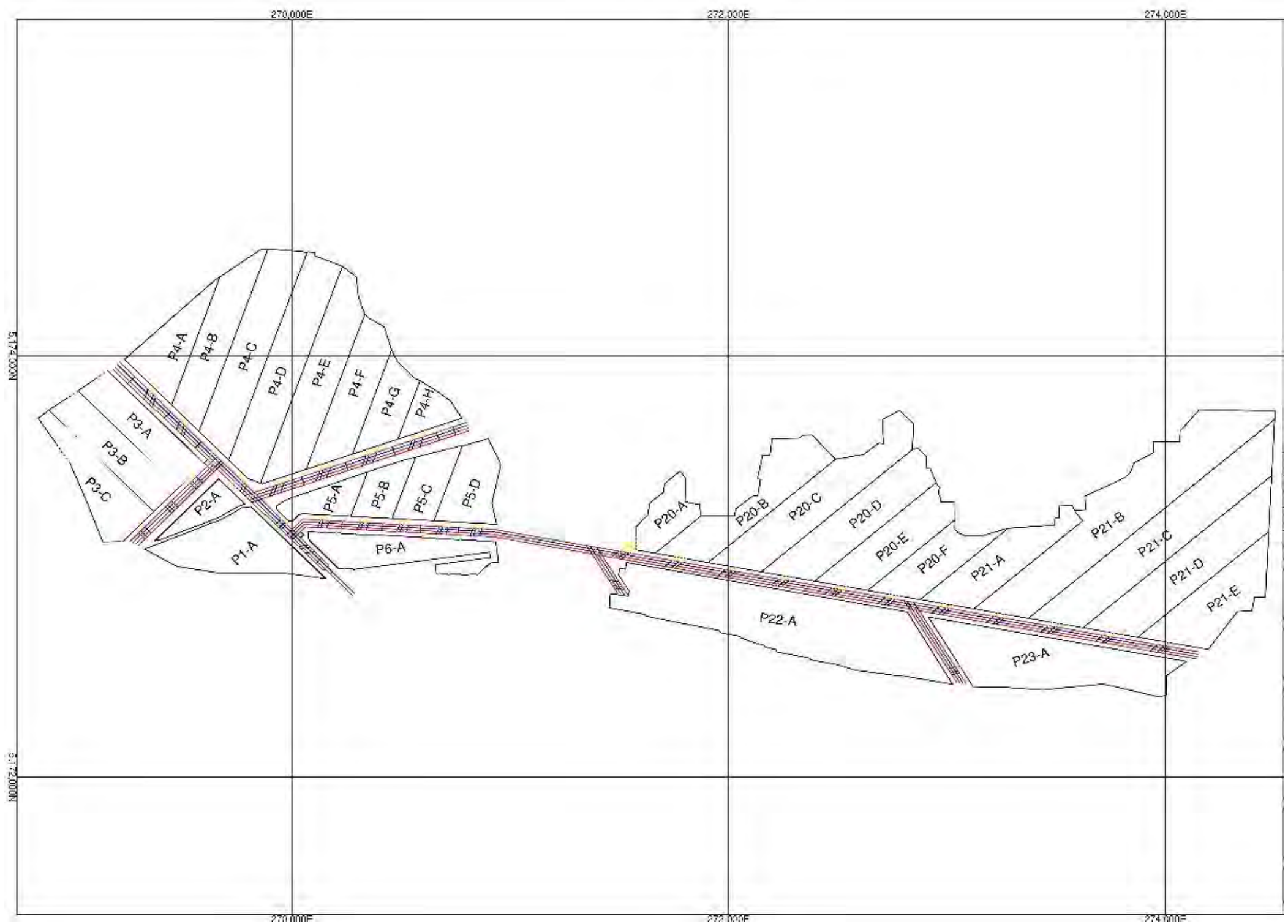
Table 1.5: Pillar Size Recommendations

Orebody	Panel	Depth (m)	Assumed Pillar Height (m)	Recommended Pillar Dimensions (m)
East	20	183	2.3	5.8 x 5.8
		274	2.9	7.6 x 7.6
	21	183	2.3	6.1 x 6.1
		274	2.3	7.6 x 7.6
	22	122	3.0	4.9 x 4.9
	23	122	2.9	5.2 x 5.2
West	1 to 6	91	3.0	5.5 x 5.5
		183		7.3 x 7.3
		274		9.4 x 9.4

1.15.4 Mine Design

The mine is divided into two sectors; the eastern part and the western part. The western part contains higher grades and a thicker mineralized zone. For these reasons, mining will start in the western part which is subdivided into 6 extraction panels as detailed in Figure 1.2. The eastern part is subdivided into 4 extraction panels; panels 20 to 23. The mining direction will generally follow the dip of the orebody, but in some areas the dip is too steep to follow. In the areas where the dip is too steep, the mining will be done at an angle to the dip direction.

Figure 1.2: Mine Design General Arrangement



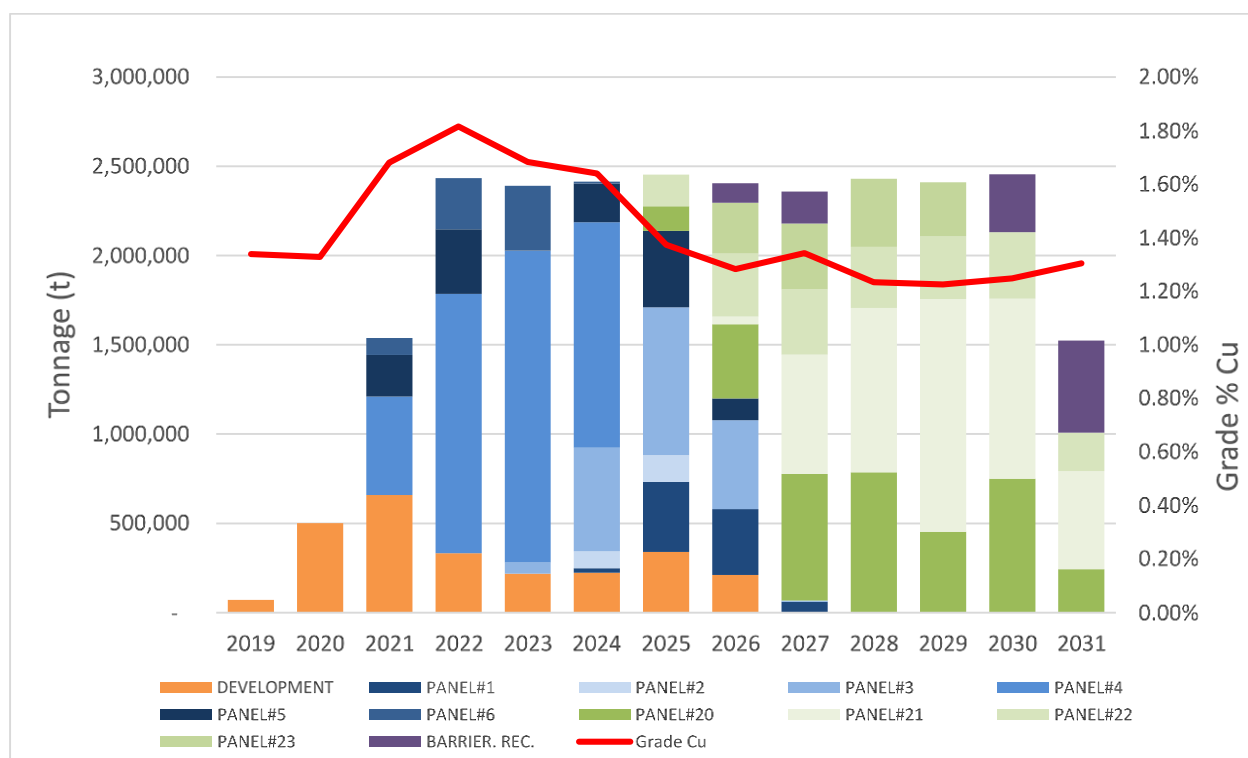
1.15.5 Mine Production Schedule

Development mining during the pre-production period is planned to start in August 2019 once the box-cut has been completed. Development initially consists of two headings with an advance rate of 5.6 m/d from the box-cut entrance and splits into 4 headings at which point an additional development team is planned and the advance rate increases to 9.0 m/d. Development mining will be ongoing at different rates until 2026 or until the east part of the mine is fully developed (Figure 1.3).

Development ore will be stockpiled at surface on a designated ore stockpile pad for rehandling into a hopper feeding the main conveyor to the ore bins. This stockpile will serve as buffer as the mine stoping production ramps-up.

Stoping activities are initiated in March 2021 ahead of the start of commercial production. Commissioning and plant ramp-up take place during the first quarter of 2021 using development ore. The mine production schedule is presented in Table 1.6.

Figure 1.3: Mine Production Schedule



Payable copper produced over the life of mine ("LoM") is 300 kt (660 M lb) with an annual average of 28 kt (61.7 M lb) over the 10.7-year life which includes 3 months of commissioning and ramp-up. The average

payable copper payable rate is 95.8% which includes a 0.2% concentrate loss. Payable silver production over LoM is 1.08 M oz with an annual average of 100 k oz at an average payable rate of 46.9% which is affected by low payable rates in the second half of the LoM when the silver concentrate grade often falls below the minimum payable of 30 g/dmt. The metal production is presented on an annual basis in Table 1.7.

Table 1.6: Mine Production Schedule Summary

Mine Production		Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Development Mining															
Tonnage	kt	2,552	71	501	659	332	217	223	339	210	-	-	-	-	-
Cu Head Grade	% Cu	1.07	1.34	1.33	1.39	0.86	0.72	0.75	0.81	0.75	-	-	-	-	-
Ag Head Grade	g/t	3.21	4.66	4.40	4.47	2.60	1.91	1.45	1.94	2.11	-	-	-	-	-
Cu Contained Metal	kt	27	0.9	6.7	9.2	2.9	1.6	1.7	2.8	1.6	-	-	-	-	-
Ag Contained Metal	k oz	263	10.6	70.9	94.8	27.8	13.4	10.4	21.2	14.3	-	-	-	-	-
Production Mining															
Tonnage	Kt	22,837	-	-	880	2,102	2,174	2,191	2,115	2,196	2,359	2,430	2,411	2,456	1,524
Cu Head Grade	% Cu	1.47	-	-	1.90	1.97	1.78	1.73	1.46	1.33	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.90	-	-	6.31	6.82	6.23	6.06	4.35	3.32	2.90	2.17	2.10	2.19	2.21
Cu Contained Metal	kt	337	-	-	17	41	39	38	31	29	32	30	30	31	20
Ag Contained Metal	k oz	2,866	-	-	178	461	435	427	296	235	220	169	163	173	108
Total Mining															
Tonnage	kt	25,389	71	501	1,539	2,434	2,391	2,414	2,454	2,406	2,359	2,430	2,411	2,456	1,524
Cu Head Grade	% Cu	1.43	1.34	1.33	1.68	1.82	1.68	1.64	1.37	1.28	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.83	4.66	4.40	5.52	6.24	5.84	5.63	4.02	3.22	2.90	2.17	2.10	2.19	2.21
Cu Contained Metal	kt	364	1	7	26	44	40	40	34	31	32	30	30	31	20
Ag Contained Metal	k oz	3,129	11	71	273	489	449	437	317	249	220	169	163	173	108

Table 1.7: Mill Production Schedule Summary

Mill Production		Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Tonnage Processed	kt	25,389			2,089	2,409	2,409	2,416	2,409	2,409	2,409	2,416	2,409	2,409	1,606
Cu Head Grade	% Cu	1.43			1.59	1.81	1.69	1.64	1.38	1.28	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.83			5.23	6.23	5.84	5.64	4.04	3.25	2.91	2.17	2.10	2.19	2.22
Concentrate (dry)	dmt	1,264			115.3	151.7	141.1	137.7	115.3	107.5	112.3	103.6	102.7	104.3	72.7
Concentrate (wet)	wmt	1,389			126.7	166.7	155.0	151.3	126.7	118.1	123.4	113.9	112.8	114.6	79.9
Cu Contained Metal	kt	364			33	44	41	40	33	31	32	30	30	30	21
Cu Contained Metal	M lb	802			73	96	90	87	73	68	71	66	65	66	46
Ag Contained Metal	k oz	3,129			352	483	452	438	313	251	225	169	163	169	114
Cu Recovery	%	86.00			86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0
Ag Recovery	%	73.40			73.4	73.4	73.4	73.4	73.4	73.4	73.4	73.4	73.4	73.4	73.4
Cu Metal Production	kt	313			28.5	37.5	34.9	34.1	28.5	26.6	27.8	25.6	25.4	25.8	18.0
Cu Metal Production	M lb	690			62.9	82.8	77.0	75.1	62.9	58.7	61.3	56.5	56.0	56.9	39.7
Ag Metal Production	k oz	2,296			258	354	332	321	230	184	166	124	119	124	84
Cu Payable Rate	%	95.76			95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76
Ag Payable Rate	%	46.91			56.91	58.69	59.00	58.66	51.57	43.80	34.55	19.23	17.08	19.10	16.52
Cu Payable Metal	kt	300			27.3	35.9	33.4	32.6	27.3	25.5	26.6	24.6	24.3	24.7	17.2
Cu Payable Metal	M lb	660			60.2	79.3	73.7	71.9	60.2	56.2	58.7	54.1	53.6	54.5	38.0
Ag Payable Metal	k oz	1,077			146.8	207.9	195.8	188.4	118.4	80.8	57.2	23.8	20.4	23.8	13.9

1.15.6 Mine Operations

Mining operations are planned with two 10-hour shifts per day, 360 days per year to achieve a production target of 2.4 Mtpa, or 6,600 mtpd. To achieve this production, a total of 7 to 9 panels must be in production at any given time.

The mine is split into panels which consist of at least 12 rooms that provide multiple headings where all activities of the mining cycle can be done in parallel to achieve high productivities as opposed to activities in series as is the case in a single heading. The mining cycle consists of:

- Drilling;
- Explosives loading and blasting;
- Mucking;
- Scaling;
- Bolting.

The mining operation begins with drilling of the working face which is accomplished with two-boom hydraulic-electric jumbo drills. Each round is drilled 4.25 m (14 ft) in length with an effective break of 4.00 m. The rooms are 6.1 m wide with a height that varies according to the ore column thickness. The height dimension dictates the productivity which varies from panel to panel. The drill penetration rate is evaluated at 1.85 m/s for an average drill time per round of 3.5 hours.

Explosives loading will be done with a mixture of ANFO and emulsion where water is present. A decoupled explosive charge is recommended to pre-split the back. Blasting will be done at shift ends with a period of 2 hours planned to vent blast fumes.

Mucking will be done with 10 t load-haul-dump ("LHD") units that will load muck at the mine face and transport it to the conveyor loading point established for the production panel. The LHD performance will be a function of dip of the stope and distance. The conveyor loading points will be regularly moved as production advances in the panel to be less than 250 m from the headings. A total of 67 loading point moves are planned over the LoM.

Scaling of the rooms is planned with a smaller low-profile LHD unit equipped with a scaling arm that rubs the roof to remove any loose rocks.

Bolting will be done by a mechanized bolter to install roof support and wall bolts. In the stopes, 1.8 m rebar bolts are required on a 1.2 m by 1.2 m pattern with wire mesh. In addition, 1.8 m friction bolts are planned in the pillars (i.e. walls) on a 1.5 m by 1.5 m pattern with wire mesh. At room intersection rebar bolt length

is increased to 2.4 m. For rooms with heights inferior to 2.4 m, connectable bolts are planned. Due to the quantity of ground support to install, the ratio of bolters to jumbos is 1.5 on average.

1.15.7 Mine Services

Mine services to support mine production include ventilation, dewatering and materials handling.

Ventilation during the pre-production period will be supplied by two 300 HP 54 in. (1.4 m diam.) parallel van axial fans on surface. These fans will generate about 115,000 CFM each and will be operational until the main intake fans are commissioned.

The permanent ventilation system will consist of a push system with two 1,250 HP 101.5 in (2.60 m diam.) parallel main fans installed at surface each providing 425,000 CFM. These fans will push heated air through a 5 m diameter ventilation raise from which air will be distributed using ventilation regulators, auxiliary fans, doors, and bulkheads. Two 5 m exhaust ventilation raises for each side of the mine will be equipped as emergency egresses.

The dewatering system will consist of six pumping stations capable of evacuating 2,220 l/min of underground water inflow and mine water.

1.16 Recovery Methods

The process plant design for the Copperwood Project is based on a metallurgical flowsheet designed to produce copper concentrate. The process plant has been designed for a nominal throughput of 6,600 mtpd. The overall flowsheet includes the following steps:

- Crushed ore reclaim;
- Grinding and classification;
- Rougher flotation;
- Rougher concentrate regrinding;
- Cleaner flotation, using three stages of cleaning;
- Concentrate thickening and filtration;
- Tailings pumping.

1.16.1 Crushed Ore Reclaim

Crushed ore from the underground mine will be conveyed to a crushed ore transfer conveyor that will discharge onto a bidirectional/reversible conveyor which in turn feeds the crushed ore bins. The two 1,200 t crushed ore bins will be equipped with two pan feeders, each to reclaim material to feed the SAG mill feed conveyor.

1.16.2 Grinding and Classification

The grinding circuit will receive ore at a nominal top size of 203 mm with an 80% passing size of 150 mm. The circuit will consist of a SAG mill in closed circuit with a screen and a ball mill in closed circuit with a cyclone cluster. The target primary grind size is 40 microns.

The SAG mill will be a 7.92 m diameter x 4.21 m EGL mill with a 5,500 kW motor. The SAG mill discharge will be screened with oversize recycled back to the SAG mill and the undersize will gravitate to the cyclone feed pump box where it will be further diluted to achieve the required cyclone feed density.

Cyclone underflow will gravitate to the ball mill, while cyclone overflow will gravitate to the trash screen. The ball mill will be a 5.80 m diameter x 9.86 m EGL overflow mill, with a 5,500 kW fixed speed motor.

1.16.3 Rougher Flotation

Screen undersize will gravitate to the rougher conditioner tank. The rougher flotation cells will consist of eight 130 m³ forced air tank cells in series. Rougher concentrate will gravitate into the regrind cyclone feed hopper.

1.16.4 Regrind

Rougher concentrate and second cleaner tailings will report to the regrind cyclone feed pump box. The slurry will be pumped to the regrind cyclone cluster by the regrind cyclone feed pumps. The regrind mill will be a vertical mill and grinding will be achieved via attrition and abrasion of the particles in contact with steel media.

1.16.5 Cleaner Flotation

Cleaner flotation will consist of three stages of closed circuit cleaning. The final arrangement includes recirculation of the first cleaner scavenger concentrate and tailings to the regrinding/first cleaner circuit and

rougher last cells (scavenger) respectively. The number of cleaning stages and regrinding arrangement will remain unchanged.

The first cleaner flotation cells will consist of six 18.0 m³ trough cells in series. First cleaner concentrate will gravitate to the first cleaner concentrate, while the first cleaner tailings will gravitate to the first cleaner scavenger flotation cells.

The second cleaner flotation cells will consist of six 8 m³ trough cells in series.

The third cleaner flotation cells will consist of six 2 m³ trough cells in series. Third cleaner concentrate will be collected in a pump box and will be pumped to the concentrate thickener.

1.16.6 Concentrate Thickening and Filtration

Final concentrate will be pumped to a 16 m diameter high rate thickener. Thickened concentrate will be pumped in batch to the concentrate filter press (1,500 mm x 1,500 mm x 40 m) with a target moisture of 9%.

1.16.7 Tailings Pumping

Rougher and first cleaner scavenger tailings will be combined in a mixing box from where a final flotation sampler will take a sample to the OSA for metallurgical and process control purposes. Flotation tailings will be pumped to the TDF.

1.17 Project Infrastructure

The Copperwood Project requires several infrastructure elements to support the mining and processing operations. The infrastructure planned for the Project includes the following:

- County Road 519 upgrade under responsibility of the Michigan Department of Transportation;
- Site access road (4.1 km) from the entrance of CR 519;
- Grid power connection requiring 25 mi of 115 kV line between the Norrie substation in Ironwood and main substation at Copperwood under the responsibility of utility company;
- Site electrical distribution at 13.8 kV;
- Communications infrastructure (fiber optic link and LTE communications network);
- Covered box cut for the mine entry (250 m long ramp at 15%);

- Ore stockpile pad at surface (65,000 m² area with HDPE liner);
- Water intake in Lake Superior with a capacity of 500 USGPM (31.5 L/s) for fresh water make-up and potable water supply;
- Sewage treatment using stabilization ponds;
- Fuel storage (10,000 l);
- Gatehouse to control site access;
- Explosives depot;
- Truck shop (5 bays including one wash bay), warehouse (20 m x 25 m) and related offices;
- Mine dry for 375 workers;
- Metallurgical laboratory and mill offices;
- Transload facility for concentrate handling (located in Park Falls);
- Administration and assay laboratory (located in Wakefield);
- TDF constructed with cut and fill approach in three stages with HDPE liner;
- Effluent water treatment plant for 275 USGPM (17.3 L/s) constructed in 2025;
- Event pond ditches for surface water management at mill site.

1.18 Market Studies and Contracts

The metal prices selected for the economic evaluation in this Report are presented in Table 1.8. Higher near-term copper prices are assumed reflecting commodity price forecasts from analysts and reverting to a lower long-term price of US\$ 3.10/lb. The silver price has been assumed constant at US\$ 16.00/oz over the LoM.

Table 1.8: Metal Price Assumptions

Metal Price Scenario	Yr 1 (2021)	Yr 2 (2022)	Yr 3 (2023)	Yr 4+ (2024+)
Copper (US\$/lb)	3.40	3.25	3.15	3.10
Silver (US\$/oz)	16.00	16.00	16.00	16.00

The copper concentrate produced from Copperwood will require downstream smelting and refining to produce marketable copper and silver metal. Concentrate transportation charges will be a function of the final destination.

The concentrate from Copperwood will be loaded into heavy-duty dump trailers with a cover and transported by truck to rail trainload facility located in Park Falls, Wisconsin. The truck configuration consists of 5 axles and will transport approximately 20 t per shipment. Park Falls is a preferred trainload location as it is currently served by the Canadian National ("CN") railroad and is approximately 80 miles from the mine site.

A summary of the copper concentrate marketing assumptions is summarized in Table 1.9.

Table 1.9: Concentrate Marketing Assumptions

Copper Concentrate Marketing Assumptions	
Copper Payable Rate	96.5% payment of Cu in concentrate >22% Cu and <32% Cu subject to a 1% minimum deduction
Silver Payable Rate	90% payment of Ag subject to 30 g/dmt minimum deduction
Copper Treatment & Refining Charge (TC/RC)	TC = US\$ 70/dmt of concentrate, RC = \$0.070/lb of Cu
Silver Refining Charge	RC = US\$ 0.50/oz of Ag

1.19 Environmental Studies and Permitting

1.19.1 Environmental Studies

Environmental baseline studies were initiated for the Copperwood Project in late 2008 through the spring of 2011. These studies were used to identify potential siting of infrastructures based on an environmental management and permit approvals perspective.

An Environmental Impact Assessment ("EIA") was prepared to comply with the State of Michigan requirements of Rule 425.202 of Part 632 of Act No. 451 of Public Acts of 1994 as amended. This document outlines the baseline monitoring and studies conducted for the Copperwood Project. This includes characterization of the natural, social, economic, cultural, and historical aspects of the environment that may be potentially impacted by the Copperwood Project design.

1.19.2 Permitting

To start construction and begin operation of this Project a number of permits must be obtained and agreed upon between Highland and the regulators, at both the state and federal levels. The major environmental permits required include:

- Part 632 Non-Ferrous Metallic Mining Permit;

- Part 31 National Pollutant Discharge Elimination System Permit;
- Part 55 Air Permit to Install;
- Part 301 Inland Lakes and Streams Permit;
- Part 303 Wetland Permit;
- Part 315 Dam Safety Permit;
- Part 325 Bottomlands Permit;
- Section 10 US Army Corps of Engineers Water Intake Permit.

Other minor and local permits are also required to start construction and mine operation that include:

- Local building and zoning permits;
- Explosives handling permit from the US Bureau of Alcohol, Tobacco, and Firearms;
- Storage tank permits;
- Mine Safety and Health Administration registration.

1.20 Capital and Operating Costs

The capital expenditure ("CAPEX") for Project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be US\$275 M. The total initial Project capital includes a contingency of US\$22.9 M, which is 9.1% of the total CAPEX before contingency, and excludes pre-production revenue of US\$30.35 M. Net of pre-production revenue, the initial CAPEX is estimated at US\$244.6 M as presented in Table 1.10. The initial Project CAPEX is spent over a period of 27 months starting in January 2019 and ending in March 2021.

Table 1.10: Initial Capital Expenditure Summary

Initial CAPEX		US\$ k
000 - General		1,150
100 - Infrastructure		36,650
200 - Power & Electrical		5,156
300 - Water & TDF Mgmt.		22,875
400 - Mobile Equipment		27,240
500 - Mine Infrastructure		53,529
600 - Process Plant		45,771
700 - Construction Indirects		27,609
800 - General Services & Owner's Costs		22,251
900 - Pre-Production, Commissioning		9,838
Sub-Total Before Contingency		252,069
Contingency	9.1%	22,899
Total Incl. Contingency		274,968
Less: Pre-Production Revenue		(30,348)
Total Incl. Contingency & Pre-Prod. Revenue		244,619

Sustaining capital expenditures during operations are required for additional mine equipment purchases, mine development work, tailings storage expansion for Stages 2 and 3, and the water treatment plant ("WTP"). The total LoM sustaining CAPEX is estimated at US\$156.5 M with the breakdown presented in Table 1.11.

Table 1.11: Sustaining Capital Expenditure Summary

Sustaining CAPEX	LoM (US\$M)	\$/t ore	US\$/lb Cu Payable
Tailings Disposal Facility Expansions	28.4	1.14	0.04
Water Treatment Plant	6.1	0.25	0.01
Mine Equipment Purchases	43.7	1.75	0.07
Mine Development Expenditures	78.2	3.13	0.12
Total Sustaining CAPEX	156.5	6.26	0.24

Note: Ore tonnage and payable copper unit costs during operations period only

Operating expenditures (“OPEX”) include mining, processing, G&A services, concentrate transportation and concentrate treatment and refining charges. The concentrate transportation, treatment and refining charges are deducted from gross revenues to calculate the NSR. The NSR for the Project during operations is estimated at US\$1,821 M excluding US\$30.35 M of NSR generated during pre-production and treated as pre-production revenue. The average NSR over the LoM is US\$2.80/lb of payable copper. Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivity. The operating costs are detailed in Section 21 of this Report. The average OPEX over the LoM is US\$39.84/t of ore or US\$1.53/lb of payable copper with mining representing 53.4% of the total OPEX, or US\$ 21.26/t of ore. A summary of operating cash flow and operating costs is presented in Table 1.12.

Table 1.12: Operating Cost Summary

Operating Cash Flow	LoM (US\$M)	US\$/t ore	US\$/lb Cu Payable
Cu Revenue	2,047	81.92	3.15
Ag Credits	17	0.67	0.03
Revenue	2,064	82.59	3.17
Concentrate Transportation Costs	94	3.75	0.14
Treatment & Refining Charges	149	5.96	0.23
Net Smelter Return	1,821	72.88	2.80
Royalties	85	3.39	0.13
Mining Costs	531	21.26	0.82
Processing Costs	308	12.31	0.47
G&A Costs	72	2.88	0.11
Total OPEX	996	39.84	1.53
Operating Cash Flow	826	33.03	1.27

Note: Ore tonnage and payable copper unit costs during operations period only

1.21 Economic Analysis

The undiscounted after-tax cash flow is estimated at US\$ 316 M for the Copperwood Project. The pre-tax net present value at 8% (“NPV_{8%}”) is estimated at US\$ 162.1 M with an 21.1% internal rate of return (“IRR”) and 2.9 y payback period. Similarly, the after-tax NPV_{8%} is estimated at US\$ 116.8 M with an 18.0% IRR and 3.2 y payback period.

The annual cash flow is summarized in Figure 1.4 and a cash flow waterfall for the Copperwood Project is presented in Figure 1.5.

Figure 1.4: After-Tax Annual Project Cash Flow

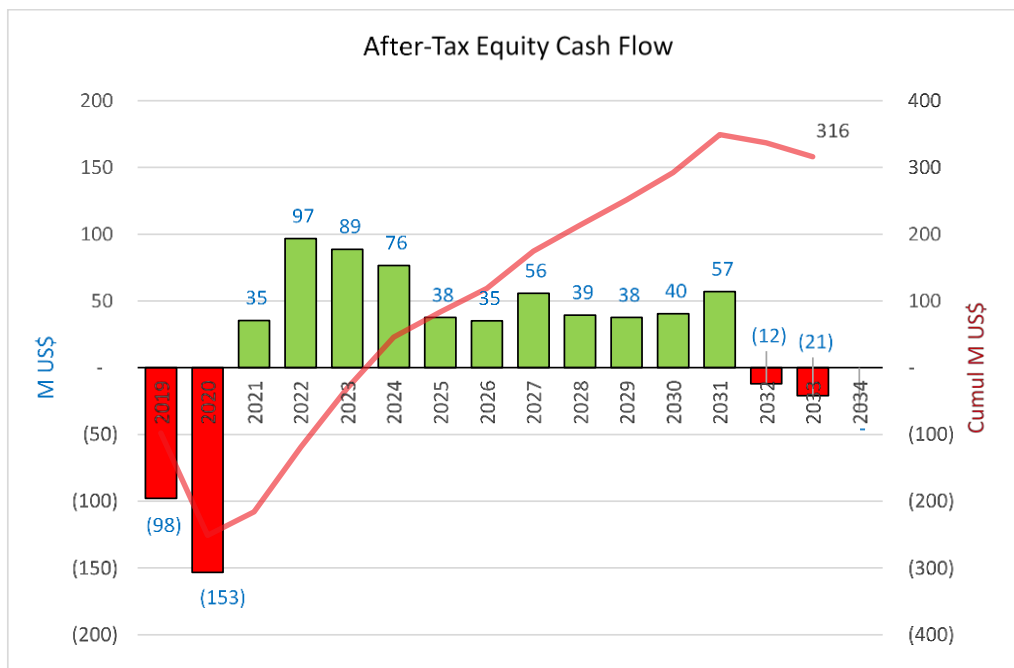
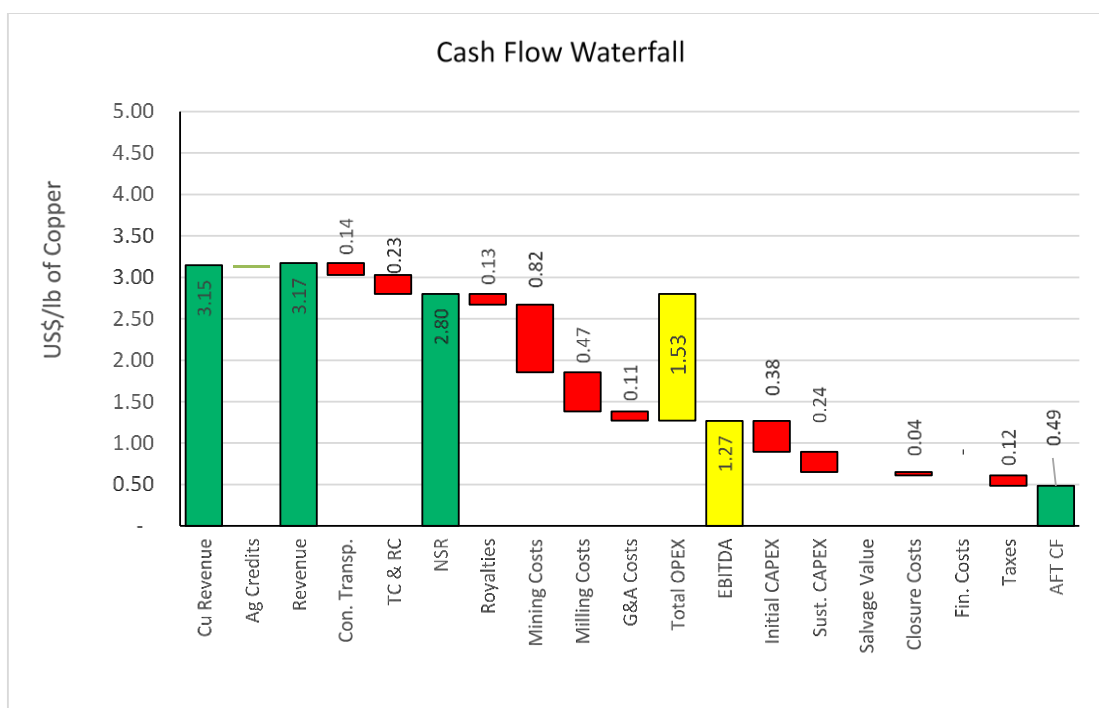


Figure 1.5: After-Tax Project Cash Flow Waterfall



1.22 Adjacent Properties

There are no other mineral exploration or development projects adjacent to the Copperwood Project.

1.23 Other Relevant Data and Information

Execution will essentially be an “owner managed” construction consisting of a team of Highland personnel and GMSI personnel with local contractors hired to perform the work on an hourly rate basis. However, several aspects of the Project will be turnkey type constructions.

Engineering will be managed by GMSI and aspects such as mining, infrastructure with other components to be accomplished by other external firms such as the process plant engineering, power supply and TDF.

The Project schedule milestones are:

- Start of detailed engineering: Jan 2019;
- Early works ground breaking: March 2019;
- Box-cut completion: June 2019;
- Mining equipment delivery and start of development: June 2019;
- TDF Phase 1 construction start: August 2019;
- Start process plant construction: August 2019;
- Grinding mills delivery: June 2020;
- Powerline commissioning complete: December 2020;
- Plant commission start date: December 2020.

Highland notes that the timeline of activities described above and completion of such activities is subject at all times to matters that are not within the exclusive control of Highland. These factors include the ability to obtain, on terms applicable to Highland, financing and required permits.

1.24 Interpretation and Conclusions

1.24.1 Conclusions

- The Copperwood deposit presents little geological risk given its excellent lateral continuity;

- The room-and-pillar mining method is well suited for the deposit geometry and is a highly mechanized mining method allowing for high production rates;
- The currently defined mineral reserves of 25.4 Mt allow for a 10.7 yr LoM (excluding commissioning and ramp-up) based on a 6,600 mtpd nominal milling rate. The mine design criteria are based on a geotechnical assessment completed by Golder;
- The process flow sheet has been validated and optimized with additional metallurgical testwork completed in 2017 and 2018 which has resulted in finer grinding, optimized reagent dosages, and increased flotation time;
- The construction of the project is planned over a 27-month period which is essentially dictated by the power line, permitting and construction schedule and to a lesser extent by the mine development.

1.24.2 Risks and Opportunities

The risks and opportunities identification and assessment process are iterative and have been applied throughout the FS phase. The following risks and opportunities are summarized in Table 1.13.

Table 1.13: Project Risks and Opportunities

Project Risks	Project Opportunities
Permit acquisition or delays	Additional mineral reserves
Ability to attract experienced professionals	Using a continuous mining equipment
Declining metal prices	Ground support design criteria and mining height
	Underground tailings disposal
Development and construction start date	Metallurgical recovery improvements
Faults creating offsets to the mineralization	Copper concentrate leaching
Reduction in grant for County Road 519 upgrade	Rising metal prices

1.25 Recommendations

Based on the positive results of the FS, GMSI recommends that the Copperwood Project move forward to the next phase which would include the following:

- Secure project financing;
- Complete environmental permitting process;
- Initiate critical detailed engineering to support critical item purchases;

- Finalize and implement an early works program in anticipation of construction release;
- General detailed engineering of process plant and other project components;
- Implement an ERP to facilitate project management and controls;
- Project construction.

2. INTRODUCTION

Highland acquired all rights, title and interests in the Copperwood Project through the acquisition of all the outstanding shares of CRI from Orvana in June 2014. Most of the exploration work on Copperwood was done by Orvana. Throughout this Report, unless otherwise indicated, activities performed before June 17, 2014, refer to events and work performed during the period Orvana owned the Copperwood Project. Activities performed after June 17, 2014, refer to events and work performed during the period after Highland acquired CRI.

2.1 Scope of Work

GMSI was retained by Highland to lead and coordinate a FS and prepare a Report in accordance with the NI 43-101 for the Copperwood Project located in the western Upper Peninsula of Michigan, USA.

This Report supports the results of the FS as disclosed in Highland's press release entitled "*Highland Copper Announces Positive Feasibility Study Results for its Copperwood Project in Michigan*" dated June 15, 2018.

This Report has a number of cut-off dates for information:

- The effective date of the Current Mineral Resource is April 30, 2018
- The effective date of the Mineral Reserve is May 25, 2018
- The effective date of this Report is June 14, 2018

The FS is focused on the extraction and processing of the Mineral Reserves from the Copperwood Project contained within the Main Zone, Sections 5 and 6. The Mineral Resource update includes the Satellite zones but the resources estimated on the Satellite zones are not included in the mine plan or economic evaluation.

The FS scope includes the following main aspects:

- Drilling for the collection of metallurgical samples and geotechnical investigations;
- Mineral Resource drilling focused on Zones 5 and 6 and complete Mineral Resource update of LCBS and UCBS for entire property;
- Geotechnical assessment and updated mine design criteria;

- Updated mine engineering, including mine design and production schedule;
- Metallurgical testing with additional composites from all mining zones;
- Revised plant design and support infrastructure;
- Power supply options evaluation;
- Lake Superior water intake design;
- Design modifications of the tailings disposal facility and capacity adjustment for increased tailings volume;
- Updated water balance model;
- Estimation of OPEX and CAPEX for the Project.

2.2 Sources of Information and Data

The information and data contained in this Report were obtained from Highland; sources included the previously published NI 43-101 technical reports and references cited in those reports. The most recent technical report stating a Mineral Resource estimate for the Copperwood Project was written by GMSI in 2017. Previous technical reports include Marston and Marston Inc. (now part of Golder) in March 2011 and Golder, in 2014, in connection with the acquisition of CRI, which only reported historical estimates for the Copperwood Deposit.

GMSI has sourced information from previous technical reports and appropriate reference documents as cited in the text and summarized in Section 27 of this Report. GMSI has relied upon other experts in the fields of mineral tenure, surface rights, permitting and environment as outlined in Section 3.

- Orvana issued several NI 43-101 reports regarding the Copperwood Project.
- AMEC produced a Mineral Resource estimate as part of a NI 43-101 technical report in April 2010. The April 2010 AMEC technical report addressed the resource in the Project area on lands covering portions of Sections 1 and 2 of Township 49N, R46W and Sections 35 and 36 of Township 50N Range 46W. The April 2010 AMEC technical report concluded that there was a NI 43-101 compliant resource for the Copperwood Project with both Measured Mineral Resources and Indicated Mineral Resources. The technical report had an effective date of April 30, 2010.
- A second NI 43-101 Mineral Resource estimate technical report was prepared in 2011 by AMEC, covering an additional 229 ha from the nearby Section 6 property and surrounding Satellite Deposits, was issued in January 2011. The resources on the Satellite Deposits, including Section 6,

were evaluated by AMEC in a NI 43-101 technical report published on January 27, 2011. The technical report had an effective date of January 24, 2011.

- Another NI 43-101 Mineral Resource estimate technical report was prepared in March 2011 by Marston and covered what was called the Copperwood Main, Bridge and Section 6 areas. The technical report had an effective date of January 25, 2011.
- In addition to these NI 43-101 Mineral Resource estimate technical reports issued, Orvana also issued:
 - ✓ A Scoping Study (effective date of September 24, 2010, authored by AMEC);
 - ✓ A Prefeasibility Study (effective date of July 29, 2011, authored by KD Engineering, Marston and Knight Piesold);
 - ✓ A Feasibility Study (effective date of March 21, 2012, authored by KD Engineering, Golder and Milne and Associates Inc.) for the Copperwood Project.
- Golder prepared a NI 43-101 technical report in March 2014 for Highland in connection with the TSX Venture Exchange acceptance of Highland's acquisition of the Copperwood Project. The Golder technical report reported the mineral resources as historical estimates for the Copperwood Project. The Golder technical report has an effective date of March 17, 2014.
- GMSI prepared a NI 43-101 technical report in June 25, 2015, for Highland as a review of the Copperwood Project resources using then current market conditions and included recommendations of further work. This GMSI technical report had an effective date of April 15, 2015.

2.3 Qualifications and Experience

The major contributors for the Study and the Report and their respective areas of responsibility are as follows:

- GMSI – overall Report and FS coordination, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, Mineral Resource estimates, Mineral Reserves, mining methods, economic analysis, operating costs, infrastructure, power supply, capital cost estimate and project execution plan;
- SGS – mineral test work;
- Lyco – flow sheet, mass balance, recovery methods, mineral process plant design and input to operating and capital cost estimates for the process plant;

- Golder – rock mechanics and underground geotechnical assessment, water balance, water treatment design, and tailings disposal facility design;
- Foth – environmental, permitting and social aspects.

A summary of the QPs responsible for each section of the Report is detailed in Table 2.1.

Table 2.1: Summary of Qualified Persons

	Qualified Person	Company	Report Sections
1	Louis-Pierre Gignac, M.A.Sc., P. Eng.	GMSI	1, 2, 3, 4, 5, 19, 21 (excluding 21.1, 21.4.1, 21.4.2), 22, 23, 25, 26, 27
2	Réjean Sirois, P. Eng.	GMSI	6, 7, 8, 9, 10, 11, 12, 14
3	Carl Michaud, P. Eng.	GMSI	15, 16 (excluding 16.2), 21.4.1
4	Paul Murphy, P. Eng.	GMSI	18, 20, 21.1, 24
5	Manochehr Oliazadeh, Ph.D, P. Eng.	Lyco	13, 17, 21.4.2
6	Ross D. Hammett, Ph.D., P. Eng.	Golder	16.2

2.4 Site Visits

Mr. Sirois met with Highland personnel, including Mr. Carlos H. Bertoni, Vice President, Exploration at the Project Office in Calumet, Michigan between January 13th and January 17th, 2014, to discuss the Copperwood Project. The purpose of the visit was to familiarize the QP with the general geology of the area and detailed geology of the Copperwood Project property, to review the Project exploration history, to review available information and to discuss procedures and methods applied during the past exploration programs. A second site visit was performed from November 6th to November 9th, 2017, by Mr. Réjean Sirois, P. Eng. and Mr. James Purchase of GMSI. The purpose of the second site visit was to examine new drill hole sites and review new drill cores.

Mr. Carl Michaud, Mr. Robert Marchand and Mr. Pong Mony Khuon of GMSI visited the Copperwood site and core shack with Highland personnel to discuss the rock units found in the mining column and to discuss the rock mechanics as well as the geotechnical investigation program. Discussions regarding the historical mining at White Pine were also held with Mr. Jack Parker who formerly worked at the mine and Stan Vitton, professor at Michigan Tech. Members of the Golder team included: Ross Hammett, Karen Moffit and Dan SaintDon.

2.5 Units of Measure, Abbreviations and Nomenclature

Unless otherwise indicated, this Report uses Canadian English spelling, USA dollar currency and System International (metric) units. Coordinates in this Report are presented in metric units metres (m) or kilometres (km) using the Universal Transverse Mercator (“UTM”) projection (UTM Zone 16, NAD83 datum). Elevations are reported as metres above mean sea level (mamsl).

The previous Copperwood Project technical reports used a combination of metric and imperial units; however, to reduce confusion and avoid the use of mixed measurement units, GMSI has converted imperial units from these reports to metric wherever possible.

The previous Copperwood Project technical reports presented coordinates using State Plane coordinates (Michigan North Zone, NAD83) in international feet, and elevations were derived using GEOID03 and NAVD88. These coordinates were converted by Coleman Engineering Co. of Ironwood, Michigan, contracted by Highland. In the current Report, GMSI has used these coordinates in metric units and the UTM projection (UTM Zone 16, NAD83 datum).

A list of the main abbreviations and terms used throughout this Report is presented in Table 2.2

Table 2.2: List of Main Abbreviations

Abbreviations	Full Description
Actlab	Activation Laboratories Ltd.
AX	AX Size Core; Core Diameter 3.01 cm
G	Billion
Ga	Billion years
BCM	Bank Cubic Meter
BSZ	Basic Shear Zone / Basal Gouge Zone
BX	BX Size Core; Core Diameter 4.20 cm
CAPEX	Capital Expenditures
CBS	Copper Bearing Sequence
cm	Centimetre
CN	Canadian National
CIM	Canadian Institute of Mining Metallurgy and Petroleum
CFM	Cubic foot per minute
CoV	Coefficient of variation
CPG	Certified Professional Geologist
Chesbrough	A.M. Chesbrough LLC
CIM	Canadian Institute of Mining Metallurgy and Petroleum

Abbreviations	Full Description
CRI	Copperwood Resources Inc. (formerly known as Orvana Resources US Corp.)
CRM	Control Reference Material
CSA	Canadian Securities Administrators
CSF	Confinement Strength Factor
Cu	Copper
°	Degrees (Azimuth or Dip)
°C	Degrees Celsius
Dmt	Dry metric tonne
E	East
EIA	Environmental Impact Assessment
Eng	Engineering
ERP	Enterprise Resource Planning
FS	Feasibility Study
ft	Feet
FEMA	Federal Emergency Management Agency
Fe-O	Iron Oxide
G&A	General & Administration
GMSI	G Mining Services Inc.
Golder	Golder Associates Ltd.
GLGT	Great Lake Gas Transmission
g	Grams
g/t	Grams per Tonne
ha	Hectares
HDPE	High Density Polyethylene
Highland	Highland Copper Company Inc.
HQ	HQ Size Core; Core Diameter 6.35 cm
ICP OES	Inductively Coupled Plasma Optical Emission Spectrometry
IDB	Influent Design Basis
IEC	International Electrotechnical Commission
IRR	Internal Rate of Return
IRS	Internal Revenue Service
ISO	International Organization for Standardization
KLA	Keweenaw Minerals, LLC (formerly Keweenaw Land Association Limited)
Kg	Kilogram
k/t	Kilogram per tonne
km	Kilometre
kV	Kilovolt

Abbreviations	Full Description
LAN	Local Area Network
LCCS	Low Cost Country Sourcing
LiDAR	Light Detection and Ranging
l	Litre
LHD	Load Haul Dump
LCBS	Lower Copper Bearing Sequence
LLC	Limited Liability Company
LoM	Life of Mine
Lyco	Lycopodium Limited
METCON	Metcon Research
m	Metre
m/d	Metres per day
mamsl	Metres above mean sea level
MDEQ	Michigan Department of Environmental Quality
MDNR	Michigan Department of Natural Resources
MDOT	Michigan Department of Transportation
MST	Nonferrous Metallic Minerals Extraction Severance Tax
µm	Micron
mm	Millimetre
Mt	Million Tonnes
Mtpa	Million tonnes per annum
MACRS	Modified Accelerated Cost Recovery System
GEOID03	National Geodetic Survey Geoid 03
N	North
NAD83	North American Datum 1983
NAVD88	North American Vertical Datum 1988
NI 43-101	National Instrument 43-101
NI 43-101CP	National Instrument 43-101 Companion Policy
NI 43-101F1	National Instrument 43-101 Form 1
NNG	Northern Natural Gas
NPV	Net Present Value
NQ	NQ Size Core; Core Diameter 4.80 cm
NREPA	Natural Resources and Environmental Protection Act, Act 451 of the Public Acts 1994, as amended
NSR	Net Smelter Return
NCNST	North Country National Scenic Trail
OK	Ordinary Kriging
OPEX	Operating Expenditures
PMWSP	Porcupine Mountains Wilderness State Park

Abbreviations	Full Description
Osisko	Osisko Gold Royalties Ltd
Orvana	Orvana Minerals Corp.
lb	Pound(s)
%	Percent
PE	Professional Engineer
Project	Copperwood Project
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
REI	Resource Exploration Inc
R&P	Room and Pillar
Ag	Silver
S	South
Sage	Sage Minerals Inc.
SG	Specific Gravity
SGS	SGS Lakefield
SGCN	Michigan Species of Greatest Conservation Need
km ²	square kilometre
TC/RC	Transportation Costs & Smelter Conversion Charges
TDF	Tailings Dam Facility
TSF	Tailings Storage Facility
TDM	Tailings & Water Disposal Management
3D	Three Dimensional
t	Tonnes
tpa	Tonnes per annum
mtpd	Metric tonnes per day
UCBS	Upper Copper Bearing Sequence
US\$	United States Dollars
USA	United States of America
USGPM	US Gallon per minute
USG	US Gallon
USFWS	U.S. Fish and Wildlife Service
USMR	United States Metals Refining Company
UTM	Universal Transverse Mercator
WBS	Work Breakdown Schedule
WC	Working Capital
WTP	Water Treatment Plant
WWTP	Waste Water Treatment Plant
W	West

Abbreviations	Full Description
wt.%	Weight Percent
yr	Year

3. RELIANCE ON OTHER EXPERTS

This Report has been prepared by GMSI for Highland. The information, conclusions, opinions, and estimates contained herein are based on:

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

Certain sections of the Report rely on reports and statements from legal and technical experts who are not QPs as defined by NI 43-101. The QPs responsible for preparation of this Report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound and are acceptable for use in this Report.

The following companies and consultants have been retained by Highland to prepare some aspects of this Report. Their involvements are listed below upon which GMSI has relied:

- GMSI has relied upon information provided by Highland including lease agreements and legal opinions concerning Highland's mineral and surface rights prepared by Kendricks, Bordeaux, Keefe, Seavoy & Larsen, P.C., a Michigan law firm;
- Concept Consulting LLC conducted a rail transportation study for the Copperwood Project. GMSI relied on this Report for concentrate transportation costs and the selection of a trainload facility location;
- GMSI has relied on input from KPMG LLP regarding the taxation model and estimates used to estimate after-tax cash flows in the economic model;
- GMSI has relied on geotechnical input from Dr. Stanley Vitton of Michigan Technological University for foundation design criteria;
- GMSI has relied on Golder for design revisions to the tailings disposal facility;
- GMSI has relied on elements from Golder and Mr. Stephen Daughney a water specialist for the water balance model and the effluent water treatment plant;
- GMSI has relied on Coleman Engineering from Michigan for wetland area surveys, surface water drainage design and the water intake design and capital cost estimate.

This Report is intended to be used by Highland as a technical report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes contemplated under provincial securities laws, any other use of this Report by any third party is at the party's sole risk.

Permission is given to use portions of this Report to prepare advertising, press releases and publicity material, provided such advertising, press releases and publicity material does not impose any additional obligations upon, or create liability for GMSI.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Copperwood Project is located within Gogebic County, Ironwood and Wakefield townships northwestern Michigan, USA, as shown in Figure 4.1.

Surface and mineral rights in Michigan are located and described with reference to a grid established by the federal government as part of the Public Lands Survey System. Townships are squares of 36 mi² (93 km²) comprising 6 x 6 arrays of 36 sections, named according to distance and direction from a principal meridian and baseline. Sections are 36 mi² (93 km²), and can be divided into quarters, labelled NE, NW, SE, and SW. Each quarter can also be split into halves or quarters, which are labelled according to the side or corner of the quarter section they encompass (e.g., NE 1/4 of the NW 1/4).

4.2 Mineral Tenure

The Copperwood Project comprises the Copperwood Deposit and the Satellite Deposits. The Copperwood Deposit consists of four metallic and non-metallic mineral leases totalling contiguous 1,904 contiguous ha under two 20-year lease agreements with KLA, a 20-year lease agreement with Sage and a 30-year mineral lease agreement with Chesbrough. The mineral rights' boundaries and lease details are summarized in Figure 4.1. The sections, surveyed as part of the Public Lands Survey System, are identified at corners with federal monuments. The Satellite Deposits consist of options to convert an additional 595 ha into mineral leases on mineralized zones adjacent to the Copperwood Deposit.

In Michigan, as with many other states, mineral rights are distinct from surface rights. Mineral rights may be sold or retained separately from the surface rights, in which case, the mineral rights are said to be severed. The Copperwood Deposit mineral rights are severed.

Figure 4.1: Project Location and Infrastructure

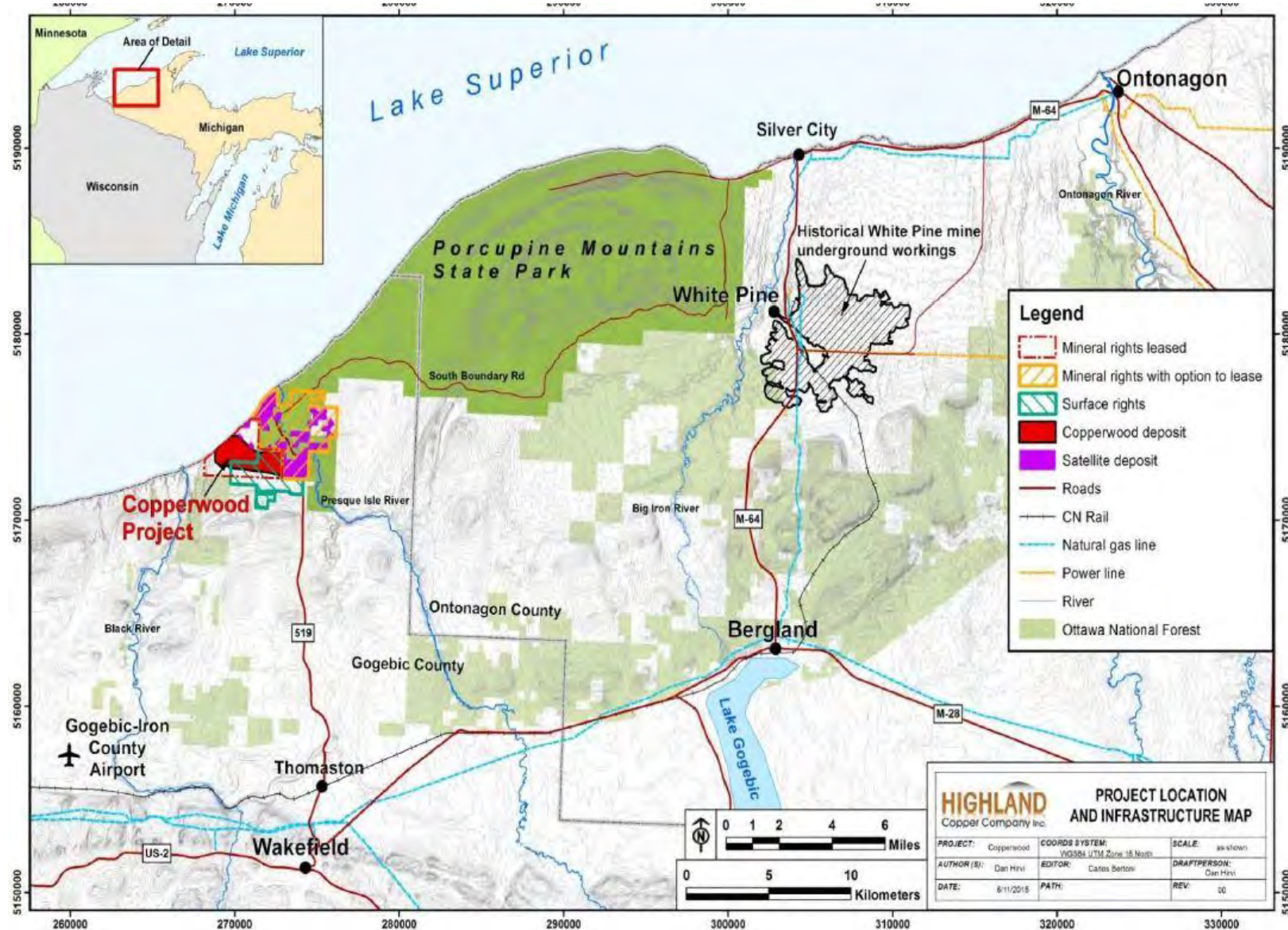


Table 4.1: Copperwood Mineral Tenure

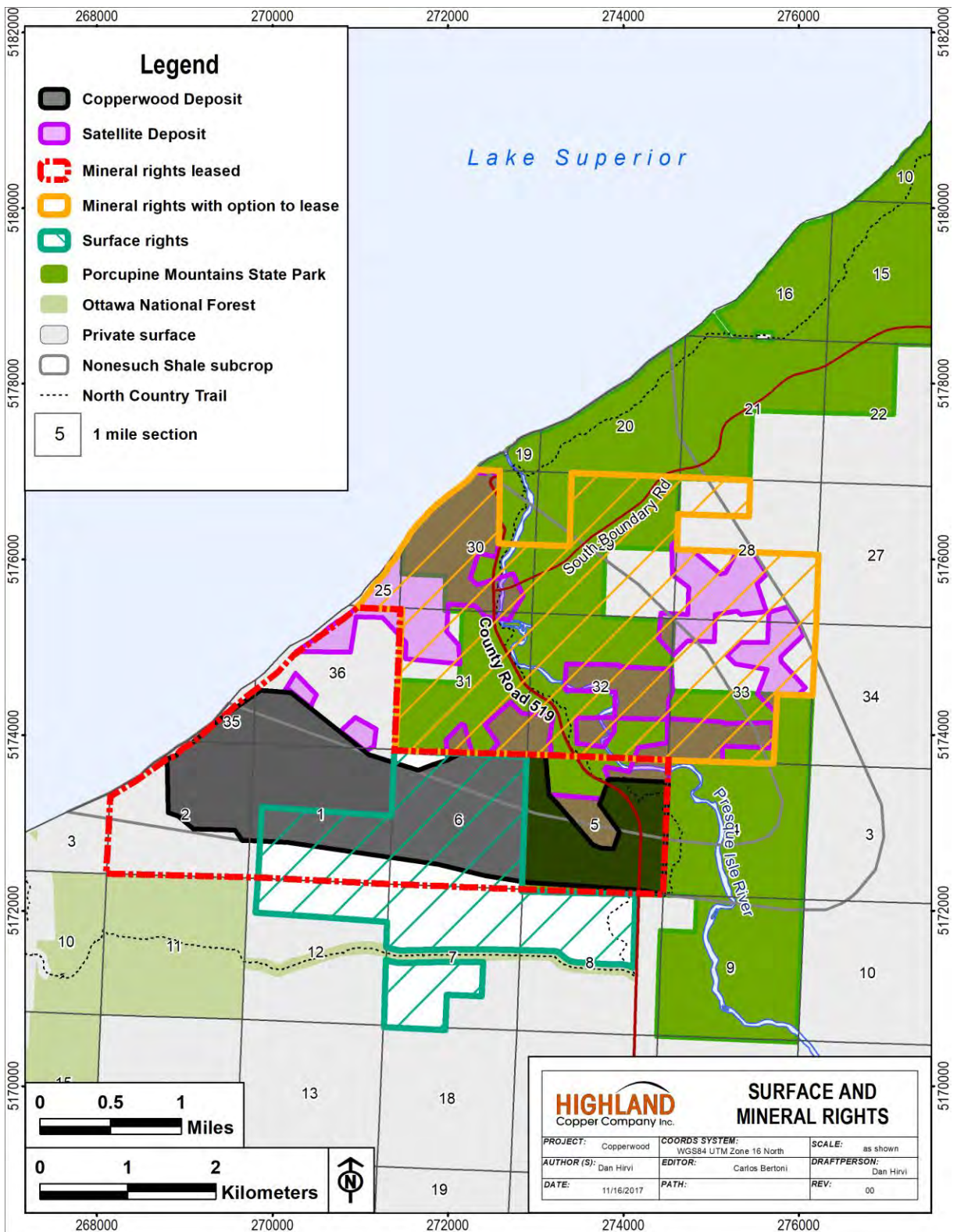
Township & Range	Sections	Area (ha)	Status
50N 46W	36	214.5	20-Year Lease ending in 2028
49N 46W	2	221.8	
50N 46W	35	28.3	20-Year Lease ending in 2028
49N 46W	1	247.3	
49N 45W	6	229.0	30-Year Lease ending in 2036
49N 45W	5	247.0	20-year Lease ending in 2037
50N 45W	29 (fraction)	226.6	
50N 45W	31	243.2	
50N 45W	33 (fraction)	226.6	
50N 46W	25 (fraction)	20.5	
50N 45W	28, 30, 32	595	Option to Lease

4.3 Surface Rights

CRI owns approximately 700 ha of land that provides full access rights to the Copperwood Project and provides space for surface infrastructure for the potential future mine site. These lands are described below and depicted in Figure 4.2:

- The entire Section 6, Township 49 North, Range 45 West, Wakefield Township;
- The North Half, the Southwest Quarter, and the Northeast Quarter of the Southeast Quarter, Section 7, Township 49 North, Range 45 West, Wakefield Township;
- The North Half of Section 8, Township 49 North, Range 45 West, Wakefield Township, except the portion lying East of the County Road 519 right of way;
- The North Half of the North Half, Section 12, Township 49 North, Range 46 West, Ironwood Township;
- The South Half of Section 1, Township 49 North, Range 46 West, Ironwood Township, Gogebic County, Michigan;
- A 200 x 300 feet (61 x 91 m) parcel in Government Lot 2, Section 2, Township 49 North, Range 46 West, Ironwood Township, Gogebic County, Michigan;
- An easement for ingress, egress, utilities and underwater pipe installation over Government Lot 2, Section 2, Township 49 North, Range 46 West, Ironwood Township, Gogebic County, Michigan.

Figure 4.2: Project Location with Lease Information - Surface and Mineral Rights



4.4 Agreements, Royalties and Encumbrances

The Copperwood Project consists of four metallic mineral leases totalling 1,188 ha, as well as one option to lease up to an additional approximate 595 ha.

4.4.1 Mining Leases

Mining Lease between CRI and KLA dated September 10, 2008, concerning:

- Section 1, Township 49 North, Range 46 West, Ontonagon Township, Gogebic County;
- Section 35, Township 50 North, Range 46 West, Ontonagon Township, Gogebic County;

Mining Lease between CRI and Sage Minerals Inc. (Sage) dated October 16, 2008, concerning:

- Section 2, Township 49 North, Range 46 West, Ontonagon Township, Gogebic County;
- Section 36, Township 50 North, Range 46 West, Ontonagon Township, Gogebic County.

Mining Lease between CRI and Chesbrough dated September 30, 2010, concerning:

- Limit pre-production mining to upper bench levels unaffected pit water;
- Section 6, Township 49 North, Range 45 West, Wakefield Township, Gogebic County.

Mining Lease between CRI and KLA (March 31, 2016), concerning the following properties located in Ironwood and Wakefield Townships, Gogebic County, State of Michigan:

- Section 5, T49N, R 45W;
- The Entire (except the W/2 of the NW/4) Section 29, T50N, R 45W;
- Section 31, T50N, R 45W;
- The Entire (except the E/2 of the SE/4) Section 33, T50N, R 45W;
- The Entire Fractional Section 25, T50N, R 46W.

To maintain its rights under the leases, CRI must pay an annual rent as shown in Table 4.2, Table 4.3 and Table 4.5.

In addition to the lease payments, CRI must pay to the mineral right owners (Sage, KLA and Chesbrough) a sliding scale NSR royalty on production from its leases. The royalty rate ranges from 2% to 4% on a sliding scale based on adjusted copper prices. Initially the royalty will be:

- 2% NSR for an invoiced copper price below a lower benchmark price;
- 4% NSR for an invoiced copper price above an upper benchmark price.

Table 4.2: KLA and Sage 2008 Mining Lease Payment Schedules

Date	Amount (US\$)
Commencement Date	10,000
1 st Anniversary of Commencement Date	15,000
2 nd Anniversary of Commencement Date	20,000
3 rd Anniversary of Commencement Date	25,000
4 th Anniversary of Commencement Date	30,000
5 th through 10 th Anniversary of Commencement Date	40,000
11 th through 15 th Anniversary of Commencement Date	50,000
16 th through 20 th Anniversary of Commencement Date	90,000

Table 4.3: Chesbrough 2010 Mining Lease Payment Schedule

Date	Amount (US\$)
Commencement Date	12,500
1 st through 4 th Anniversary of Commencement Date	9,000
5 th through 10 th Anniversary of Commencement Date	11,250
11 th through 15 th Anniversary of Commencement Date	15,000
16 th through 20 th Anniversary of Commencement Date	18,750
21 st through 25 th Anniversary of Commencement Date	22,500
26 th through 30 th Anniversary of Commencement Date	26,250

Table 4.4: KLA 2017 Mining Lease Payment Schedule

Date	Amount (US\$)
Commencement Date	35,000
1 st Anniversary of Commencement Date	52,500
2 nd Anniversary of Commencement Date	70,000
3 rd Anniversary of Commencement Date	87,500
4 th Anniversary of Commencement Date	105,000
5 th through 10 th Anniversaries of Commencement Date	140,000
11 th through 15 th Anniversaries of Commencement Date	175,000
16 th and later Anniversaries of Commencement Date	315,000

For an invoiced copper price greater than the lower benchmark price and less than the upper benchmark price, the following equation is used:

$$\frac{2\% * \text{Invoiced Copper Price}}{\text{Lower Benchmark Copper Price}}$$

Invoiced copper is the price per pound of copper shown on a concentrate invoice. The lower and upper benchmark prices are subject to adjustment for inflation on a quarterly basis based on the Producer Price Index – Finished Goods, prepared by the USA Department of Labour. Benchmark prices are initially set at US\$2/lb Cu and US\$4/lb Cu, respectively.

All lease payments may be applied as a credit against the royalties during production.

4.4.2 Options to Lease

CRI is party to an option to lease agreement with Sage covering approximately 595 ha located within Wakefield Township, Gogebic County, Michigan, with an effective date of October 16, 2008. The option is for a twenty-year term (subject to termination in whole or in part by CRI on 60 days' notice and termination in whole by the option or for breach of the optional agreement) and provide for option payment as described in Table 4.5.

Table 4.5: Payment Schedule on Option to Lease Agreement

Date	Amount (US\$)
On Effective Date	6.18/ha
On 1 st Through 5 th Anniversaries of Effective Date	6.18/ha
On 6 th Through 10 th Anniversaries of Effective Date	12.36/ha
On 11 th Through 15 th Anniversaries of Effective Date	18.53/ha
On 16 th and Later Anniversaries of Effective Date	24.71/ha

CRI has the right to exercise the Sage option at any time during the term and to enter into a mining lease and net smelter return royalty agreements in respect of the covered mineral hectares. The sliding scale NSR royalty is on the same terms as those applicable to the mining leases set out above.

4.4.3 Encumbrances

As security for the payment and performance of obligations under agreements with Osisko including a net smelter royalty deed, CRI has granted to Osisko a security interest in CRI's right, title and interest in and to (i) the above-mentioned mineral leases; and (ii) all profits and income that at any time arise from the mineral leases or from the sale of minerals that are located in, on or under the leased area.

There are no other known encumbrances affecting the mineral rights that are subject to the mining leases.

4.4.4 Osisko Royalty

On June 30, 2016, the Company and Osisko agreed to amend the terms of their agreement entered into in December 2014 and to convert the US\$10 M deposit on sale of royalty into a 3.0% NSR royalty on all metals produced from the mineral rights and leases associated with the Copperwood Project. The amendment also provides that upon final closing of the acquisition of the White Pine Project, the Company will grant Osisko a 1.5% NSR royalty on all metals from the White Pine North Project, and Osisko's royalty on the Copperwood Project will be reduced to 1.5%. Osisko retains security over all of the Company's assets. On June 30, 2016, the amount of US\$10 M was recorded as a reduction of the carrying amount of the related exploration and evaluation assets.

In December 2014, the Company also granted to Osisko an option to purchase for US\$26 M a 100% NSR on any future silver production from the Company's projects, including White Pine, Copperwood and Keweenaw. Osisko may elect to exercise the option to purchase the silver production by paying US\$26 M

to the Company within 60 days following the delivery to Osisko of a Feasibility Study on the Michigan Projects.

4.5 Environmental Liabilities

Environmental work performed by CRI identified potential localized surface water impacts resulting from the surface rock piles from the 1950's exploration shaft excavation; some of this excavated material was also used in historic road and culvert construction on the property. As part of the permitting process CRI proposed mitigation in the form of removing this material from the rock pile site, roads and culverts and storing it in the planned Copperwood Tailings Disposal Facility. No other known environmental liabilities exist on the Copperwood Project property.

4.6 Permitting

The Michigan Department of Environmental Quality ("MDEQ") is responsible for enforcing state laws for protecting natural resources. Michigan's environmental regulations are compiled under the NREPA. Mining of nonferrous metals is regulated under Part 632 of NREPA.

4.6.1 Exploration

The drilling, operating, plugging, and site restoration of test wells (drill holes) are regulated under Part 625, Mineral Wells of NREPA. In addition, test wells must meet the requirements of other parts of the NREPA to prevent damage to water, air, soil, wetlands, and other environmental values. In most areas of the state, Part 625 requires a permit for a test well that penetrates 15 m (50 ft) or more into bedrock or below the deepest fresh water aquifer. However, a permit is not required for test wells where the bedrock is Precambrian in age, although these wells must meet all other requirements of Part 625. A test well must be plugged promptly after abandonment, following procedures specified by the MDEQ. A well is considered abandoned if it is inactive for one year, unless an extension is granted by the MDEQ based on the owner showing a good reason to keep the well open. Wells must be plugged in a manner that seals off and confines any fluids in the formations penetrated by the well and prevents any surface water or other materials from entering the well. Removal of overburden and extraction of limited amounts of materials for exploration to the extent necessary to determine the location, quantity, or quality of a mineral deposit on land that does not become a part of a mining operation within two years must be graded and revegetated.

All drilling at the Copperwood Project is in Precambrian bedrock and therefore no permits for drilling is required.

4.6.2 Development

Mining of nonferrous metals is regulated under Part 632 of NREPA. Part 632 covers all aspects of nonferrous metal mining including transportation, storage, treatment, and disposal of ore, waste rock, and other materials. A permit application under Part 632 must include an environmental impact assessment that describes baseline conditions, expected impacts to the mined area and surrounding affected areas, and alternatives. An application must also include a detailed plan for mining and reclamation that would minimize impacts of the proposed operation, and a contingency plan for dealing with any accidents or failures.

Part 632 provides extensive opportunities for public input, including a public meeting on an application and a public hearing on a proposed permit decision. A permit can be granted only if the applicant demonstrates that the mining operation will not pollute, impair, or destroy the air, water, or other natural resources or the public trust in those resources in accordance with the Michigan Environmental Protection Act. Upon completion of mining, the mine site and associated lands must be reclaimed to achieve a self-sustaining ecosystem that does not require perpetual care. Post-closure monitoring of water quality must be continued for at least 20 years, subject to modification after public review. Part 632 requires a mining company to maintain financial assurance throughout the mining operation and the post-closure monitoring period. The financial assurance must cover the cost for the MDEQ to conduct any necessary reclamation and remediation measures and must be updated at least every three years. Funds to cover the costs for the MDEQ to administer the law comes from permit fees and from annual operating fees based on mass of material mined.

CRI obtained the following permits from the MDEQ:

- April 30, 2012 – Part 632 Mining Permit for Copperwood Project, Upper Peninsula, Michigan, USA;
- November 13, 2012 - National Pollutant Discharge Elimination System permits for treated sanitary and process wastewater related to the proposed Copperwood copper mine, Upper Peninsula, Michigan, USA;
- July 17, 2012 – Air Quality Division Permit to Install 180-11;
- February 22, 2013 - Wetlands Part 303 and the Inland's Lakes and Streams Part 301 permits for the proposed Copperwood copper mine;
- June 24, 2013 – Part 315 Dam Safety Permit.

Highland is currently considering the amendment of some permits in relation to the ongoing Feasibility Study. The amendment of permits is further described in Section 20.

4.7 Socio-Economic

The State of Michigan, and particularly the Upper Peninsula, has a long-mining history, primarily for copper and iron. The large-scale underground White Pine copper mine in Ontonagon County began operation in 1953 and ended in 1996. Exploration programs and mining operations in Michigan are governed by modern mining and environmental laws. The workforce of the western Upper Peninsula of Michigan is currently experiencing high unemployment levels. Many experienced miners and locally owned firms also exist in the region with necessary mining support capabilities. The Copperwood Project has received local and Michigan State bipartisan support.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Copperwood Project property is located approximately 22.5 km by road to the north of the town of Wakefield in Gogebic County, Michigan, and is also located approximately 40 km by road from the town of Ironwood, also in Gogebic County. Wakefield and Ironwood have populations respectively of 2,300 and 6,800.

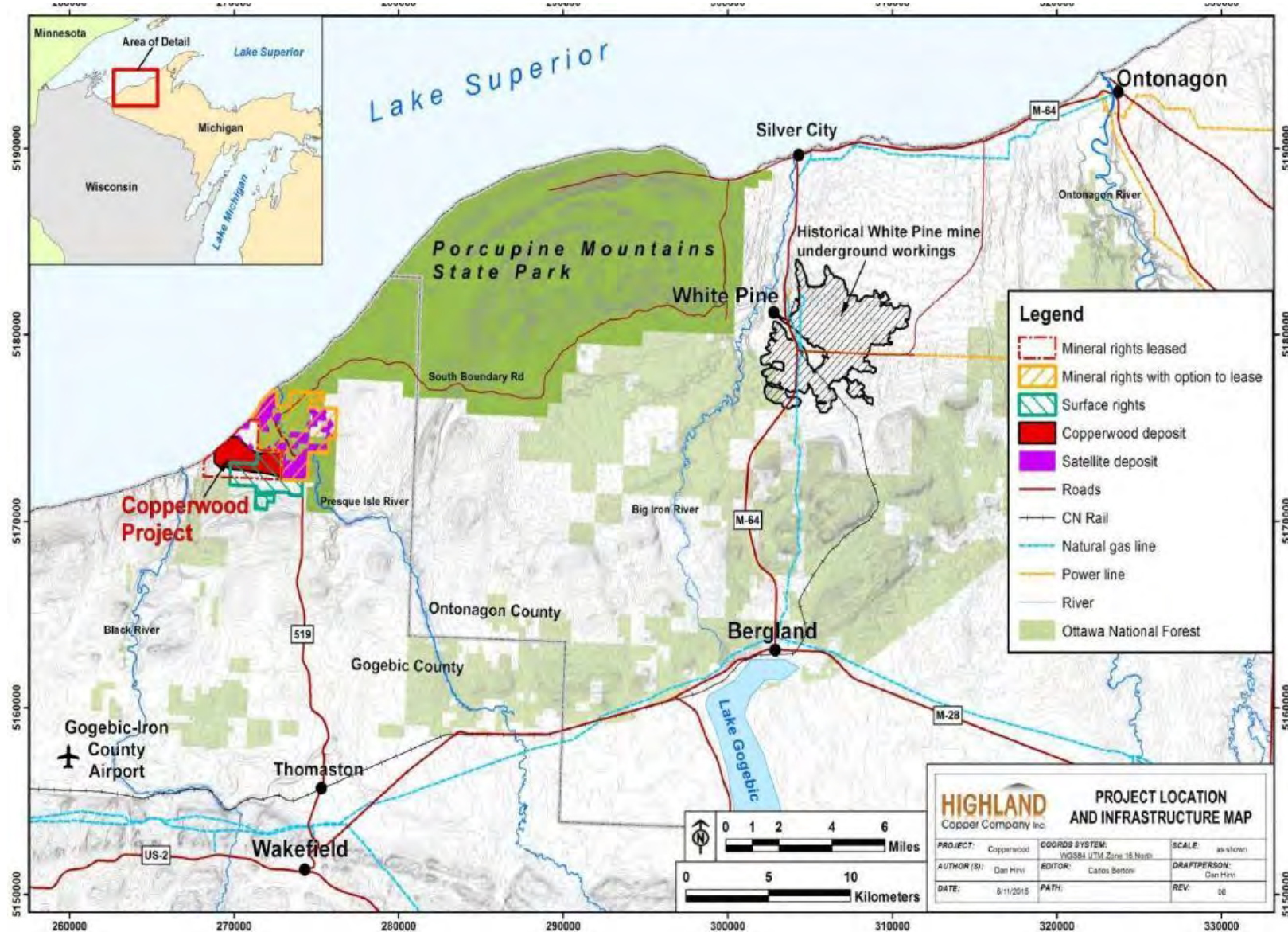
The main access to the Copperwood Project property is by way of the paved north-south County Road 519, which branches off State Highway M-28 just east of Wakefield. The Project property is transected by a series of dirt roads and drill trails allowing access for exploration activities.

During inclement weather, four-wheel drive vehicles are required for accessing the Project property. Future mining activities at the Copperwood Project will require an upgrade of the paved County Road 519 to an all-season level and an upgrade of the dirt road from County Road 519 to the Copperwood site. Site access is shown in Figure 5.1.

5.2 Climate

The Copperwood Project property is situated immediately south of the Lake Superior shoreline where the local climate consists of four seasons typical of mid-latitude temperate climates. The maximum mean monthly temperature in the summer months is approximately 18°C and about –12°C in the winter months. The annual precipitation is approximately 890 mm of rain equivalent (rain and snow) with the greatest monthly precipitation of about 100 mm and least monthly precipitation of about 30 mm of rain equivalent. Mean annual total snowfall is approximately 4.5 m with the maximum monthly mean snow depth of about 0.6 m. Wind at the Copperwood site is predominantly from the east-southeast and west-northwest directions with peak gusts of about 60 km/hr. Weather measurements are from a local meteorological station operating at the Copperwood Project property and from the Ironwood, Michigan meteorological station.

Figure 5.1: Project Location and Infrastructure



5.3 Local Resources

A Canadian National Railway Company (“CN”) rail line is located at Thomaston about 18 km south from the Copperwood site via County Road 519. There was an existing loading station at Thomaston, which was used for timber. Additionally, there is an old railway spur bed that passes immediately adjacent to the property; laying tracks along this bed would provide rail access right to the Copperwood Project site. Access by way of air travel is accomplished through the Gogebic-Iron County Airport located 6 km north of Ironwood.

The workforce for any current and future mining activity could be sourced from a combination of the local area after training as appropriate or from external areas. Unemployment is high in Gogebic County; both skilled and unskilled labour forces are available for work.

5.4 Infrastructure

The only infrastructure on the Copperwood Project property is a network of dirt roads, logging roads and drill trails. The main dirt roads are in good condition.

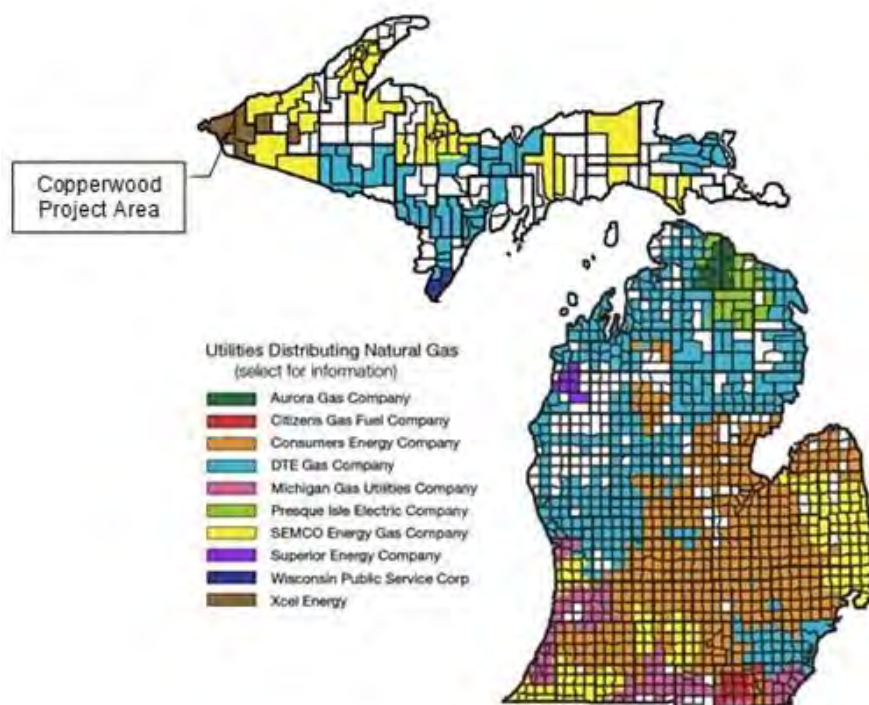
There is an 88 kV power line located 18 km from the Copperwood Project; however, this is a unique voltage that may be obsolete before long. Xcel Energy owns the nearest transmission lines, which are located approximately 32 km south of the property.

Onsite power generation is also an option. Natural gas is available from two major pipeline companies; TransCanada through their subsidiary Great Lake Gas Transmission (“GLGT”) and Northern Natural Gas (“NNG”). Both companies have pipelines and stations in Wakefield (Figure 5.2). Gas supply to site must be provided by a local distributor. Xcel Energy is the local gas distributor for the Copperwood Project area (Figure 5.3).

Figure 5.2: Michigan Natural Gas Pipeline Map



Figure 5.3: Michigan Gas Utility Service Areas



There are no aquifers beneath the surface of the property capable of yielding sufficient water for the process plant. Potable and process water for any planned mining operation is planned from a water intake from Lake Superior. However, during normal course operations it is planned to recycle water from the tailings disposal facility back to the process plant.

Current site communications comprise cell phone service available via a repeater tower at Indianhead ski area.

The Copperwood Project property and surface rights is of sufficient extent for all needed surface infrastructure including a processing facility, maintenance, surface equipment storage, fuel storage, explosives storage, administrative offices, water treatment plant, and, storage for waste rock, top soil and snow.

5.5 Physiography

The land surface at the Copperwood Project property slopes northwest toward the Lake Superior shoreline. The ground surface elevation along the southern edge of the site is approximately 288 mamsl as compared to the approximate elevation of 198 mamsl at the top of the bluff along the Lake Superior shoreline. Mean elevation of the Lake Superior shoreline is approximately 184 mamsl. The topographic contours across the area are generally parallel to the Lake Superior shoreline with the ground surface sloping at a rate of approximately 19 m/km to the northwest. The gently undulating planar surface is transected by small intermittent streams that flow northwest towards Lake Superior. The larger of these streams form steep-walled valleys in glacial deposits that are 3 to 5 m deep in the upper reaches and as much as 12 m deep nearing Lake Superior.

Vegetation at the Copperwood Project is characterized by immature mixed deciduous forest. Wetlands occur onsite in the base of drainage channels and stream corridors that direct surface runoff. Wetlands are also established in depressions or small isolated basins on gently sloping plateaus between the drainage channels and stream corridors. Commercial logging and hunting cabins are the current land uses within, and in direct vicinity of, the Copperwood Project.

6. HISTORY

6.1 Exploration History

Exploration history is well documented by Golder in the March 2014 NI 43-101 technical report and it is repeated here as referenced. Table 6.1 summarizes the history of exploration completed in the Copperwood area.

Table 6.1: Summary of Copperwood Exploration Activity

Company	Activity	Year
USGS.	Economic geology publication demonstrates potential of Western Syncline.	1954
USMR	Leased 1,552 ha in Western Syncline area (Cox, 2003).	1956
USMR	Drilled 26 holes focused on margin of Western Syncline and discovered Copperwood.	1956
USMR	Drilled 135 holes throughout the Western Syncline.	1958
AMAX	Sank 71 m vertical exploration shaft and advanced 635 m of exploration drifts, including three small stopes.	1957-1958
BCR	Drilled 23 holes in the Satellite properties. BCR terminated leases in the early 1960's.	1959
AMAX	Internal engineering and economic study that ended activities by USMR.	1959
AMAX	Engineering and economic review concluded deposit was mineable.	1974
AMAX	Terminated Western Syncline leases.	1983
Orvana	Leased 712 ha at Copperwood and option 1,559 ha in Western Syncline.	2008
Orvana	Began environmental studies with five drill holes intersecting copper mineralization.	2008
Orvana	Drilled 82 holes.	2009
Orvana	Leased 229 ha covering Section 6.	2010
Orvana	Drilled 38 holes. Completed NI 43-101 compliant Mineral Resource estimate.	2010
Orvana	Completed NI 43-101 compliant Mineral Resource estimate.	2011
Orvana	Completed NI 43-101 compliant Prefeasibility Study.	2011
Orvana	Completed NI 43-101 compliant Feasibility Study.	2012
Orvana	Mining Permit Approved by Michigan Department of Environmental Quality.	2012
Orvana	Drilled 21 holes for metallurgical and geotechnical studies.	2013
Highland	Drilled 40 holes and 13 wedges for resource estimate, metallurgical and geotechnical studies.	2017
Highland	Drilled 8 holes and 1 wedge as infill for Feasibility Study.	2018

Archaeological evidence suggests that native copper was first extracted by natives on the Keweenaw Peninsula about 7,000 years ago. From 1610 to 1845, the presence of Lake Superior copper attracted early European and American interest. From 1845 to 1968, the mines of the Keweenaw Peninsula produced approximately 5 million tonnes ("Mt") of refined copper from 380 Mt of ore hosted by tops of sub-aerial lava flows, interflow clastic sedimentary beds and vein systems. Native copper represented over 99% of the metallic minerals in the mined ore bodies of the Keweenaw Peninsula. Copper mineralization at the base of the Nonesuch Formation was first recognized in the 1850s in the White Pine area about 30 km northeast of Copperwood (Ensign et al., 1968). From 1915 to 1921, native copper was economically extracted along the White Pine fault, from the base of the Nonesuch Formation.

Subsequent exploration led to the discovery and the 1953 opening by Copper Range Company of the White Pine Mine. The construction of the White Pine Mine, mill, smelter, refinery and power plant was financed by the U.S. Government. Approximately 2 Mt Cu and 128 million grams of silver, with a mean grade of 1.14 wt.% Cu and 7 g/t Ag, were produced from 1954 until its closure in 1996. Chalcocite accounted for 85% to 90% of the copper with the remainder as native copper.

From about 1948 to 1954, geologists Walter White and James Wright of the U.S. Geological Survey ("USGS") conducted a major study of the Nonesuch Formation at the White Pine Mine and surrounding area. In a paper summarizing their work (White and Wright, 1954), the Western Syncline is clearly shown. Although there is no comment on copper mineralization in the Western Syncline, they concluded, "*The environment favorable for deposition of sediment's similar to those at White Pine therefore existed over an area many times larger than that of the White Pine copper deposit itself.*" This publication led to the leasing of the Western Syncline area by the USMR. This syncline is also known as Presque Isle Syncline in the literature.

In 1956, the United States Metals Refining Company ("USMR") secured an option from KLA and Sage (timber companies who had retained the mineral rights after selling the surface rights) to lease mineral rights over and proximal to the Western Syncline. USMR drilled a total of 161 vertical holes between August 1956 and November 1958. The first 26 holes were drilled to define the margin of the syncline and to sample the base of the Nonesuch Formation. One hundred thirty-five holes were then completed at 660 or 330 m spacing. Forty-two of these holes, the deepest of which reached 337 m, were drilled within the area of the Copperwood leased mineral rights. This drilling led to the discovery of the Copperwood deposit.

An underground exploration program was initiated by AMAX in July 1958. A vertical exploration shaft was sunk 71 m through 28 m of glacial overburden, 39 m of the Nonesuch Formation and 4 m of the Copper Harbour Formation sandstones. Exploration drifts were driven along strike 373 m to the east and 262 m to

the west, and three small stopes were driven up-dip to assess rock mechanic characteristics and the nature of the mineralized zone. The exploration shaft was refilled from the surface upon completion.

During a proposed merger of the Copper Range Company, the operator of the White Pine Mine and AMAX in 1974, an independent consultant completed an engineering study and review of existing data, including a resource estimate for the Western Syncline Deposit (Parker, 1974). The U.S. Government disallowed the proposed merger and in 1983, due to corporate financial issues, AMAX terminated the Western Syncline mineral lease agreements.

No further work was conducted on the Copperwood Project between 1983 and 2008.

Beginning in 2008, Orvana conducted a series of exploration drilling programs at Copperwood (2008, 2009, 2010 and 2011) culminating in 126 drill holes (17,480 m total of drilling). Additionally, Orvana commissioned several independent technical reports for the Copperwood and "Satellite Deposit" areas in 2010 and 2011.

In 2013, Orvana drilled 21 drill holes to collect samples for metallurgical and geotechnical studies (2,781 m total of drilling); 11 holes were drilled primarily for metallurgical purposes and seven holes were drilled primarily for geotechnical purposes with one hole drilled for both metallurgical and geotechnical purposes.

Details of the Orvana exploration, drilling, sampling and analytical programs are expanded upon in Sections 9,10 and 11 of this Report.

In 2017, Highland carried out a drilling program comprising of 35 HQ diameter (plus 13 wedges) and five PQ-diameter drill holes for a total of 7,666 m of core. The 2017 drill program was designed to upgrade the Mineral Resources of the eastern section of the deposit, obtain metallurgical samples and carry out geotechnical studies to refine the mining plan.

In 2018, Highland completed a drilling program of eight NQ-diameter holes and one wedge as well as finishing one HQ-diameter hole which was collared before abandoning during spring break-up in 2017. This program consisted of 2,925 m of core drilling and was carried out as infill drilling in Section 5 with the purpose of upgrading Inferred Resources to the Indicated category.

6.2 Production History

The Copperwood Project property has not had any production. The vertical shaft, exploration drifts and stopes developed by AMAX in 1958 were purely for exploration and test mining purposes.

6.3 Environmental History

In September 2008, Orvana contracted STS to conduct the base line studies for an Environmental Impact Assessment (“EIA”) covering the Copperwood Project area. STS was subsequently purchased by AECOM and the environmental studies were continued with AECOM.

In January 2009, the EIA’s initial phase of surface and subsurface water sampling was completed. This is the first step in the two-year-long process of developing a seasonal and long-term characterization of the site. In completing this phase of the assessment, 20 holes (totaling 1,239 m) were drilled, packer-tested, and completed as groundwater monitoring wells. These drill holes encountered between 21 to 33 m of fine-grained, unconsolidated glacial sediments overlying the bedrock. Fourteen drill holes were completed in bedrock above the copper-bearing interval and six holes intersected the copper-bearing interval. Also, 14 shallow water monitoring wells were completed.

A meteorological and air quality monitoring station was installed on the Copperwood Project site and data collection commenced in December 2008.

Other studies required as part of the EIA, including studies of the site’s ecosystem, habitat features and terrestrial and aquatic flora and fauna, have also been done.

An environmental geochemical examination was completed on eight reject samples of mineralization, hanging wall, and footwall rocks from three historical drill holes. Interpretation of the geochemical test results by Geochimica, Inc. indicates that Copperwood rocks are unlikely to be acid generating and, consequently, may be characterized as non-reactive under Michigan mining laws. In addition, the rock pile created by the extraction of copper-bearing rock from the underground exploration activity in the 1950s was trenched and sampled after being subjected to approximately 50 years of wet, oxidizing conditions. Based on visual observations, the rocks appear to be non-reactive.

6.4 Historical Resources

As discussed previously, several historical resource estimates for the Copperwood deposit have been issued:

- USMR – Covering larger area that included the Copperwood Project area, prepared in 1959;
- AMAX – Covering larger area that included the Copperwood Project area, prepared in 1974;
- Orvana (AMEC) – Copperwood area, published April 2010, effective date of April 30, 2010;
- Orvana (AMEC) – Satellite Deposits, published January 2011, effective date of January 24, 2011;
- Orvana (Marston) – Copperwood areas, published March 2011, effective date of January 25, 2011;
- Highland (GMSI) – Copperwood Deposit, published June 25, 2015, effective date of April 15, 2015;
- Highland (GMSI) – Copperwood Deposit, published December 5, 2017, effective date of October 18, 2017.

The United States Metals Refining Company (“USMR”) and AMAX estimates predated the introduction of NI 43-101 (2001) guidelines, while the 2010, 2011, 2015 and 2017 estimates were prepared in accordance with NI 43-101 guidelines in place at the time of preparation.

6.4.1 USMR and AMAX Historical Resource Estimates

An internal engineering and economic study of the entire Western Syncline (or Presque Isle Syncline) was completed in 1959 by USMR. The study reported an estimated Mineral Resource of 136.9 Mt at 1.07 wt.% Cu at a 1 wt.% Cu cut-off in some areas and a copper cut-off of 0.8 wt.% in others. The USMR Mineral Resource estimate also included mineralization in the “upper shale unit”, or UCBS. This mineralization was not included in the later historical resource estimates. The Copperwood portion of this historical resource estimate was 23.8 Mt at 1.46 wt.% Cu. USMR planned to mine the deposit by applying a room-and-pillar mining method. The USMR study concluded it would be necessary to extract barren siltstone hanging wall to reach a stable back. This resulted in excessive dilution and unfavorable economics.

During a proposed merger of the Copper Range Company, the operator of the White Pine mine, and AMAX in 1974, an independent consultant (J. Parker, 1974) completed an engineering study and review of existing data and concluded that the back could be controlled by using resin bolts, which had been recently employed at the White Pine mine. By controlling the back, the problem of excessive dilution would be eliminated, and the economics of mining the Western Syncline Deposit were deemed favorable. An independent historical, non-compliant Mineral Resource estimate for the Western Syncline Deposit was completed in 1974 that included Mineral Resources of 92.3 Mt at 1.27 wt.% Cu at a 0.9 wt.% cut-off and a minimum mining height of 1.83 m using the same raw data as used by USMR. The Copperwood deposit portion of this historical resource estimate was 21.9 Mt at 1.68 wt.% Cu.

USMR and AMAX historical Mineral Resource estimates for the Copperwood deposit portion of the Western Syncline are summarized in Table 6.2.

Table 6.2: USMR and AMAX Historical Resource Estimates for Copperwood

Historical Resource	Tonnage (Mt)	Copper Grade (wt.%)	Copper Cut-off (wt.%)	Minimum Thickness (m)
1959 USMR Engineering and Economic Study	23.8	1.46	1.0	2.6
1974 Independent Consultant Engineering and Economic Review	21.9	1.68	1.0	2.0

Note: The historical estimate cited herein has no equivalent category under CIM Definition Standards (2005). These estimates are of unknown quality and should not be relied upon.

6.4.2 Orvana – AMEC Historical Resource Estimates

In 2008, Orvana leased the Copperwood Project area from KLA and Sage and initiated an EIA as required by Michigan's Nonferrous Metallic Mining Regulations. In the fall of 2008, groundwater monitoring wells were completed. Five of these water-monitoring holes intersected the mineralized zone of the Copperwood deposit. In 2009, Orvana completed 82 exploration drill holes. On March 22, 2010, Orvana announced an NI 43-101 compliant resource estimate for the Copperwood deposit. This was followed by an NI 43-101 compliant resource estimate for the Section 6 and Satellite zones (north limb of Western Syncline) in January 2011. Both resource estimates were completed by AMEC. The AMEC historical resource estimates are summarized in Table 6.3.

Table 6.3: AMEC Historical Resource Estimates for Copperwood Deposit

Historical Resource Estimates	Tonnage (Mt)	Copper Grade (wt.%)	Copper Cut-off (wt.%)	Minimum Thickness (m)
2010 AMEC Copperwood “Main” Domino				
Measured	7.79	2.56	1	1.66
Indicated	2.48	2.39	1	1.22
Measured and Indicated	10.27	2.52	1	1.53
Inferred	1.30	2.29	1	0.95
2010 AMEC Copperwood “Main” Upper Layer				
Measured	6.35	1.15	1	1.35
Indicated	2.85	1.07	1	1.39
Measured and Indicated	9.20	1.13	1	1.36
Inferred	1.97	0.96	1	1.43
2010 AMEC Copperwood “Main” Combined Domino and Upper				
Measured	14.15	1.93	1	3.01
Indicated	5.33	1.69	1	2.60
Measured and Indicated	19.47	1.86	1	2.89
Inferred	3.27	1.49	1	2.38
2011 AMEC Section 6 Area				
Indicated	8.41	1.42	1	1.89
Inferred	0.46	1.29	1	1.54

6.4.3 Orvana – Marston Historical Resource Estimate

In March 2011, Marston completed an update to the Copperwood Main and Section 6 resource estimates (Table 6.4). The model used in the resource estimate update was built by Peter DuBois, PE, in Marston's St. Louis office under the supervision of Michael B. Ward, CPG, Senior Geological Consultant, for Marston. The Mineral Resource estimates were completed using Ventyx (formerly Mincom) Stratmodel and Block Model software.

Marston adhered to the Canadian Institute of Mining Metallurgy and Petroleum (“CIM”) definitions of resources and reserves as referenced in NI 43-101. Mineral Resources were confined by the software to the appropriate stratigraphic units. Mineral Reserves were not estimated as part of the 2011 Marston technical report as a preliminary feasibility study had not been completed. The Marston 2011 historical Mineral Resource estimates are summarized in Table 6.4 (the “Main”, “Bridge” and “Section 6” areas are equivalent to the Copperwood Deposit in this Report, except for Section 5).

Table 6.4: Marston 2011 Historical Mineral Resource Estimate Presented by Area

Copperwood “Main”			
Historical Resource Category	Tonnage (Mt)	Copper Grade (wt.%)	Silver Grade (g/t)
Measured	17.0	1.84	5.75
Indicated	3.6	1.62	4.57
Measured and Indicated	20.7	1.80	5.54
Inferred	2.6	1.06	2.02
“Bridge” Area (between “Main” and Section 6)			
Historical Resource Category	Tonnage (Mt)	Copper Grade (wt.%)	Silver Grade (g/t)
Measured	0.6	1.1	1.63
Indicated	0.2	1.1	1.84
Measured and Indicated	0.8	1.1	1.67
Inferred	0.0	-	-
Section 6 Area			
Historical Resource Category	Tonnage (Mt)	Copper Grade (wt.%)	Silver Grade (g/t)
Measured	5.6	1.38	1.96
Indicated	3.0	1.24	1.17
Measured and Indicated	8.6	1.34	1.69
Inferred	0.1	1.35	1.53
Total (Copperwood “Main, Bridge and Section 6” Combined)			
Historical Resource Category	Tonnage (Mt)	Copper Grade (wt.%)	Silver Grade (g/t)
Measured and Indicated	30.1	1.65	4.34
Inferred	2.9	1.07	2.01

6.4.4 Highland – GMSI Resource Estimate

In April 2015, GMSI completed an update to the Copperwood Main and Section 6 Resource Estimates. Réjean Sirois, Eng., built the model used in the resource estimate update at GMSI’s Brossard Office, Quebec, Canada. GMSI adhered to the CIM definitions of resources and reserves as referenced in NI 43-101.

The estimate was conducted in a block model limited by a single mineralized domain, interpreted as the LCBS. Hanging wall and footwall surfaces of the LCBS were modelled and merged to create the mineralization solid. The footwall surface was adjusted beforehand to keep a minimum thickness of 2.2 m throughout the deposit, acting as the minimum mining height. Uncapped raw assays were composited into zone composites (one composite per drill hole) with a minimum thickness of 2.2 m. Block sizes of 10 m x 10 m horizontally, with a 2.5 m height were used in the block model. A uniform bulk density of 2.7 g/cm³ was used for all rock sequences in the model. Copper and silver grades were estimated using the Ordinary Kriging interpolation method in three successive passes, using ellipse ranges of 175 m, 250 m, and 350 m.

To define resource categories, GMSI outlined groups of globally similar interpolation passes. Measured Mineral Resources thus constituted the bulk of the Mineral Resources in the Copperwood Deposit (as defined in the Report) area and include blocks interpolated generally in the first pass. Indicated Mineral Resources were located at the periphery of the Measured category where blocks are generally interpolated in the second pass and are limited to the Copperwood Deposit. All other interpolated blocks were categorized in the Inferred Mineral Resource category, including all blocks in the Satellite Deposits. A summary of Mineral Resource estimates is presented in Table 6.5.

Table 6.5: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade – April 15, 2015

Deposits	Resource Category	Tonnage (Mt)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs)	Silver Contained (M oz)
Copperwood	Measured	22.5	1.73	5.08	861	3.7
	Indicated	6.6	1.37	2.56	200	0.5
	M + I	29.1	1.65	4.51	1,061	4.2
	Inferred	1.9	1.24	2.37	52	0.1
Satellite	Inferred	38.6	1.23	2.09	1,050	2.6

Notes on Mineral Resources:

- 1) Mineral Resources are reported using a copper price of US\$3.00/lb and a silver price of US\$20/oz.
- 2) A payable rate of 96.5% for copper and 90% for silver was assumed.
- 3) The Copperwood Feasibility Study reported metallurgical testing with recovery of 86% for copper and 50% for silver.
- 4) Cut-off grade of 1.0% Cu was used.
- 5) Operating costs are estimated at US\$ 49/t of ore including ore transportation to a plant at the White Pine site.
- 6) An NSR sliding scale royalty is applicable and equivalent to 3.0% at US\$3.00/lb.
- 7) Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.
- 8) No mining dilution and mining loss were considered for the Mineral Resources.
- 9) Rock bulk densities are based on rock types, % Cu and average of specific gravity measurements.
- 10) Classification of Mineral Resources conforms to CIM definitions.

- 11) The qualified person for the estimate is Mr. Réjean Sirois, Eng., Vice President Geology and Resources for GMSI. The estimate has an effective date of April 15, 2015.
- 12) Mineral Resources that are not mineral reserves do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues may materially affect the estimate of Mineral Resources.
- 13) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.

6.4.5 Highland – GMSI Resource Update October 2017

In October 2017, GMSI completed an updated resource estimate based on the 35 additional drill holes completed that year, including Section 5. The Mineral Resource estimate was based on a block model characterised by two separate copper-bearing sequences, the LCBS (including the Gray Laminated, Red Massive, and Domino units), and the Upper Copper Bearing Sequence (“UCBS”). Individual units within the LCBS were modelled and estimated separately according to the logged geological units. Uncapped raw assays were composited into separate geological units (Domino, Red Massive and Grey Laminated), with one composite per drill hole produced for each unit. For the UCBS, a grade-based modelling approach was adopted where a single layer was modelled based on assays greater than 1% Cu. This approach was applied due to a lack of historical logging and some ambiguity regarding the UCBS position in the stratigraphy. Variography studies undertaken on each geological unit highlighted strong continuity of copper and silver grades, with a low nugget effect observed. A bulk density of 2.7 g/cm³ was applied to Domino and Red Massive units, and 2.72 g/cm³ was applied to the Grey Laminated and UCBS units. Copper and silver grades were estimated using the ordinary kriging (OK) interpolation method in three successive passes, using ellipse ranges of 175 m, 250 m, and 350 m. To address the currently accepted minimum mining height of 2 m, copper and silver grades were diluted in areas where the LCBS is less than 2 m in height. Dilution grades were derived from a grade estimation of the hanging wall sediments (Red Laminated unit), which was modelled as a 50 cm buffer zone situated directly above the LCBS.

To define resource categories, GMSI outlined groups of globally similar interpolation passes. Measured Mineral Resources thus constitute the bulk of the Mineral Resources in the Copperwood Deposit area and include blocks interpolated generally in the first pass. Indicated Mineral Resources are located at the periphery of the Measured category where blocks are generally interpolated in the second pass. All other interpolated blocks are categorized in the Inferred Mineral Resource category, including all blocks in the satellite deposits. A summary of the October 2017 Resource Estimate can be found in Table 6.6 below.

Table 6.6 Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade - Oct. 2017

Deposits	Resource Category	Tonnage (Mt)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs)	Silver Contained (M oz)
LCBS	Measured	26.8	1.69	4.59	1,000	4.0
	Indicated	11.6	1.50	2.68	383	1.0
	M + I	38.4	1.63	4.02	1,383	5.0
	Inferred	4.6	1.36	1.69	138	0.3
UCBS	Measured	-	-	-	-	-
	Indicated	4.1	1.19	3.33	107	0.4
	M + I	4.1	1.19	3.33	107	0.4
	Inferred	0.3	1.05	3.23	8	0.0
Satellite LCBS	Inferred	33.2	1.21	2.37	885	2.5
Satellite UCBS	Inferred	6.1	1.15	4.75	155	0.9

Notes on Mineral Resources:

- 1) Mineral Resources are reported using a copper price of US\$3.00/lb and a silver price of US\$18/oz.
- 2) A payable rate of 96.5% for copper and 90% for silver was assumed.
- 3) The 2012 Copperwood Feasibility Study by Orvana reported metallurgical testing with recovery of 86% for copper and 50% for silver.
- 4) Cut-off grade of 1.0% Cu was used, based on an underground "room and pillar" mining scenario.
- 5) Operating costs are based on a processing plant located at the Copperwood site
- 6) An NSR sliding scale royalty is applicable and equivalent to 3.0% at US\$3.00/lb.
- 7) Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.
- 8) No mining dilution and mining loss were considered for the Mineral Resources.
- 9) Rock bulk densities are based on rock types.
- 10) Classification of Mineral Resources conforms to CIM definitions.
- 11) The qualified person for the estimate is Mr. Réjean Sirois, Eng., Vice President Geology and Resources for GMSI. The estimate has an effective date of October 18, 2017.
- 12) Mineral Resources that are not mineral reserves do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- 13) LCBS: Lower Copper Bearing Sequence.
- 14) UCBS: Upper Copper Bearing Sequence.

The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.

7. GEOLOGICAL SETTING AND MINERALIZATION

Geological descriptions for the Copperwood Project area are based on several authors including Cannon *et al.*, 1989; Elmore, 1984; Elmore *et al.*, 1989; Hieshima and Pratt, 1991; Davis and Paces, 1990; Bornhorst *et al.*, 1988; Cannon, 1992; Bornhorst, 1997; Cannon, 1994; Swenson *et al.*, 2004; White, 1968; Stoiber and Davidson, 1959; Bornhorst and Robinson, 2004; Catacosinos, 2001; Bornhorst and Lankton, 2009; and, Bornhorst and Williams, 2013.

7.1 Regional Geology

The Copperwood Project area is situated along the southeast flank of the 2,200 km long Mesoproterozoic mid-continent rift system of North America within the Keweenaw Copper province as shown in Figure 7.1. The rocks of this rift system consist of a package of volcanic and clastic sedimentary rocks that are up to 30 km thick called the Keweenaw Supergroup. They are only exposed in the Lake Superior region. The rocks range from about 1.15 Ga to 1.03 Ga in age and include active rift-phase rocks of the Bergland Group and the post rift clastic sedimentary rocks of the Oronto and Bayfield Groups. These groups are shown in the stratigraphic column in Figure 7.2.

The Bergland Group consists of tholeiitic flood basalts with minor interbedded red conglomerate and sandstone of the Portage Lake Lava Series. This sequence hosts native copper deposits that yielded five million tonnes of the metal between 1845 and 1969. A significant amount of silver was produced as a by-product. In the Copperwood area, the Porcupine Mountain volcanic rocks lie at the top of the Bergland Group. The lowest exposed portion of the Bergland Group lies along the Keweenaw fault as shown in Figure 7.3.

Following the active rifting phase, the basin continued to subside and clastic sedimentary rocks of the Oronto and Bayfield Groups were deposited. The Oronto Group directly overlies the Bergland Group. It is subdivided into three formations: the Copper Harbor Formation, the Nonesuch Formation and the Freda Formation. The Nonesuch Formation hosts the mineralization at both the Copperwood Project area and the White Pine mine, as shown in Figure 7.3.

The Copper Harbor Formation is composed of red-brown conglomerates and sandstones with lesser siltstone and these sedimentary rocks were fluvial deposits in coalescing alluvial fans. They are upward and basinward-fining.

Figure 7.1: Location of the Midcontinent Rift System

The Nonesuch Formation interfingers with and conformably overlies the Copper Harbor Formation. This unit consists of a package of lacustrine and fluvial black-to-gray-to-green-red siltstone and shale with minor carbonate laminates, and sandstone lenses that is up to 300 m thick. Black to dark-gray shale, deposited in anoxic lacustrine conditions favorable for the preservation of organic carbon and pyrite, are common in the lower 30 m of the formation. The Nonesuch Formation is thought to have been deposited in a marine environment.

The Freda Formation is gradational with and conformably overlies the Nonesuch Formation. It consists of red-brown fine to very fine sandstone, siltstone and mudstone, deposited by shallow meandering rivers, resulting in fining-upward sequences on a scale of meters.

The last developmental phase of the mid-continent rift system, from 1.07 Ga to 1.05 Ga, was characterized by a partial inversion of the original graben-bounding normal faults into major reverse faults, accompanied by the deposition of mature clastic sedimentary rocks of the Bayfield Group. This event was likely caused by continental collision along the Grenville Front to the east. The present-day dip of Keweenawan Supergroup strata is a result of syn-depositional sagging and tilting related to faults and folds associated with this compression event. Figure 7.3 shows the Keweenaw fault separating the older Bergland and

Oronto Group rocks to the northwest that have been thrust over the younger Jacobsville sandstone of the Bayfield Group to the southeast.

Figure 7.2: Stratigraphic Column of Regional Geology

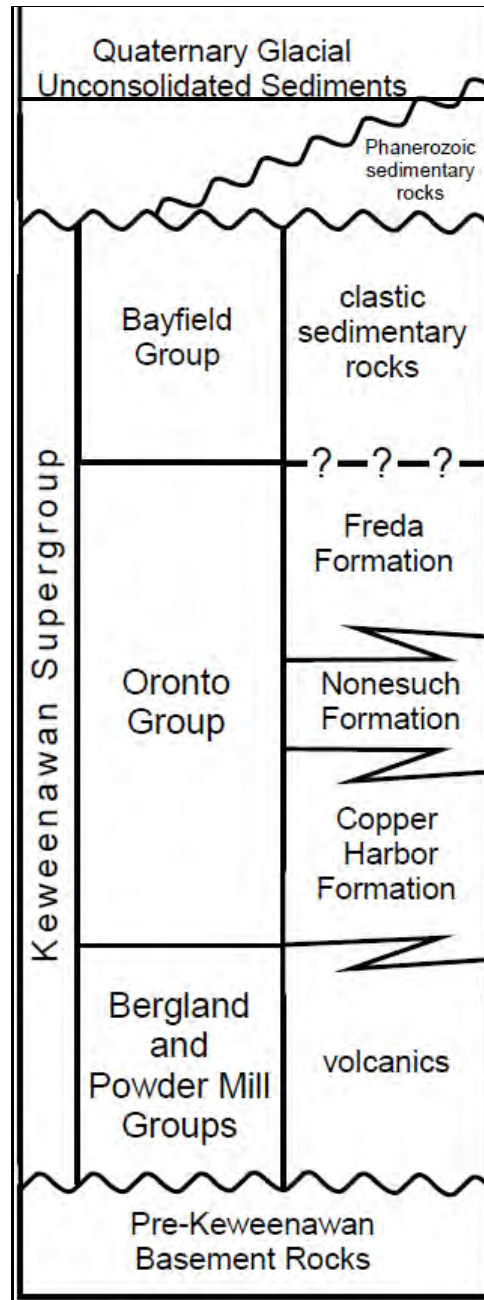
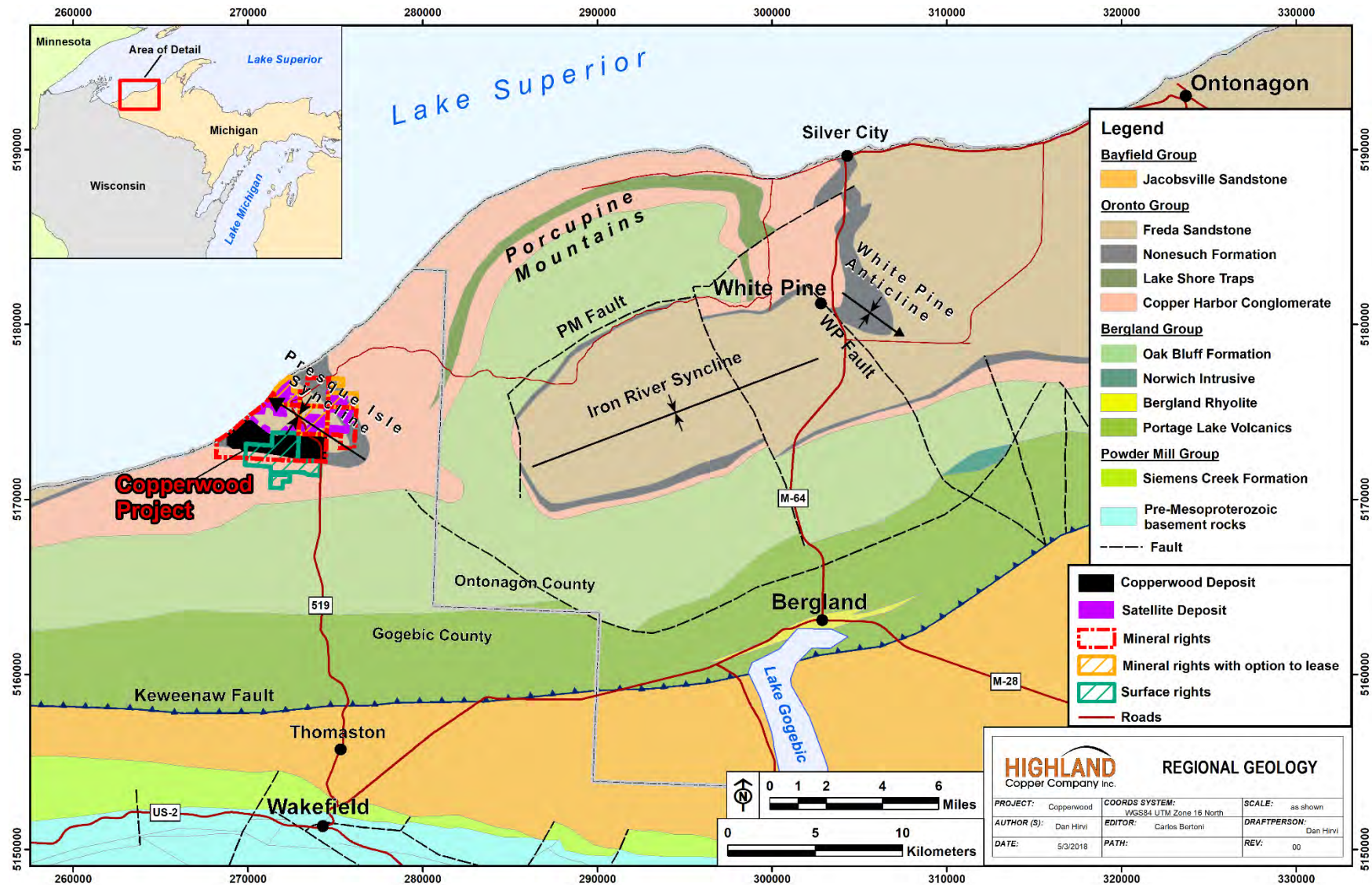


Figure 7.3: Regional Geology and Project Location



Evidence of pervasive alteration by metamorphic fluid is shown in the rift-phase volcanic rocks. These metamorphic fluids moved through a network of faults and fractures developed during late rift compression and are likely responsible for deposition of native copper in the volcanic-dominated strata of the Keweenaw Peninsula rocks in the base of the Nonesuch shale.

Multiple kilometers of bedrock were eroded following the late rift compression event. As a result, the copper deposits were exposed. These Precambrian copper deposits were likely subjected to a long period of downward percolating ground waters followed by marine submergence during the Phanerozoic. The rift rocks were subsequently buried by Phanerozoic sedimentary rocks beginning in the late Cambrian and ending in the middle Jurassic. Deposition of the Phanerozoic rocks was followed by another period of erosion and non-deposition from the middle Jurassic to the Pleistocene. The Phanerozoic rocks were removed by erosion from Precambrian rocks of the western Upper Peninsula by Pleistocene continental glaciers beginning about two million years ago.

The last retreating glaciers left behind unconsolidated gravels, sands and muds deposited in glacial, glaciofluvial and glacial lacustrine cover about 10,000 years ago.

7.2 Project Area Geology

Clastic sediments of the Oronto Group, including the Copper Harbor, Nonesuch and Freda Formations, underlay the entire Copperwood Project area. Mineralization is hosted at the base of the Nonesuch Formation on the limbs of the northwest-plunging Presque Isle Syncline as shown in Figure 7.3, (also known as Western Syncline). A complete stratigraphic section up to about 220 m thick of the Nonesuch Formation occurs in the northern part of the Copperwood Project mineral lease area. Moving to the south, the upper contact is missing due to erosion. The Nonesuch disappears where the basal contact subcrops near the southern boundary of the mineral lease.

The lowest part of the stratigraphy at the Copperwood Project is the Copper Harbor Formation. Although the unit is normally characterized by a conglomerate facies, the upper portion of the unit intersected by drilling at Copperwood consists mostly of red-brown sandstone. At the contact with the Nonesuch Formation, there is a thin, red-brown siltstone, ranging from about 10 cm up to 0.5 m in thickness. Regionally, the Copper Harbor Formation is up to 2,000 m thick, but the unit is thinner at Copperwood because of the proximity to the Porcupine volcanic center, which was a topographic high at the time of deposition of the Copper Harbor Formation conglomerates and sandstones.

The Nonesuch Formation marks a dramatic change from the oxidized red-colored Copper Harbor Formation to a gray- to black-colored fine-grained clastic sedimentary section. The change to a more reducing

depositional environment played an important role in the location of the mineralized horizons. The basal portion of the Nonesuch Formation is termed the LCBS. The LCBS is a group of subunits of the Nonesuch Formation that host the bulk of the copper and silver mineralization at Copperwood. The UCBS is a second group of subunits that contain copper mineralization at Copperwood, higher in the stratigraphy. The UCBS and the LCBS are separated by grey sandstones with thinly bedded, dark reddish-brown siltstones and shales. These intervals gradually decrease in thickness from 8 m in the westernmost part of the deposit to 1.8 m in the easternmost part of the deposit, as shown in Figure 7.4. Above the UCBS, the Nonesuch Formation consists of shale, mudstone and siltstone with almost no mineralization.

7.2.1 Lower Copper Bearing Sequence

The LCBS at the Copperwood Deposit is subdivided into the Domino, Red Massive and Gray Laminated subunits. This sequence directly overlies the red sandstone and siltstone of the Copper Harbor Formation, as shown in Figure 7.4.

The Domino subunit, the principal copper host at Copperwood, lies immediately above the Copper Harbor Formation and is characterized by laminated dark gray to black shale and siltstone. A mineralized sample of the Domino subunit is shown in Figure 7.5. Red-brown layers are present throughout in varying frequency. There are occasionally very fine-grained gray sandstone beds with thickness of a few centimeters within the upper half of Domino. A thin, typically less than 0.1 m thick zone of brecciated shale/siltstone is often, but not always, present at or near the base. The Domino ranges in thickness from 0.0 to 2.3 m and has a mean thickness of 1.6 m.

Figure 7.4: Copperwood Deposit Stratigraphy

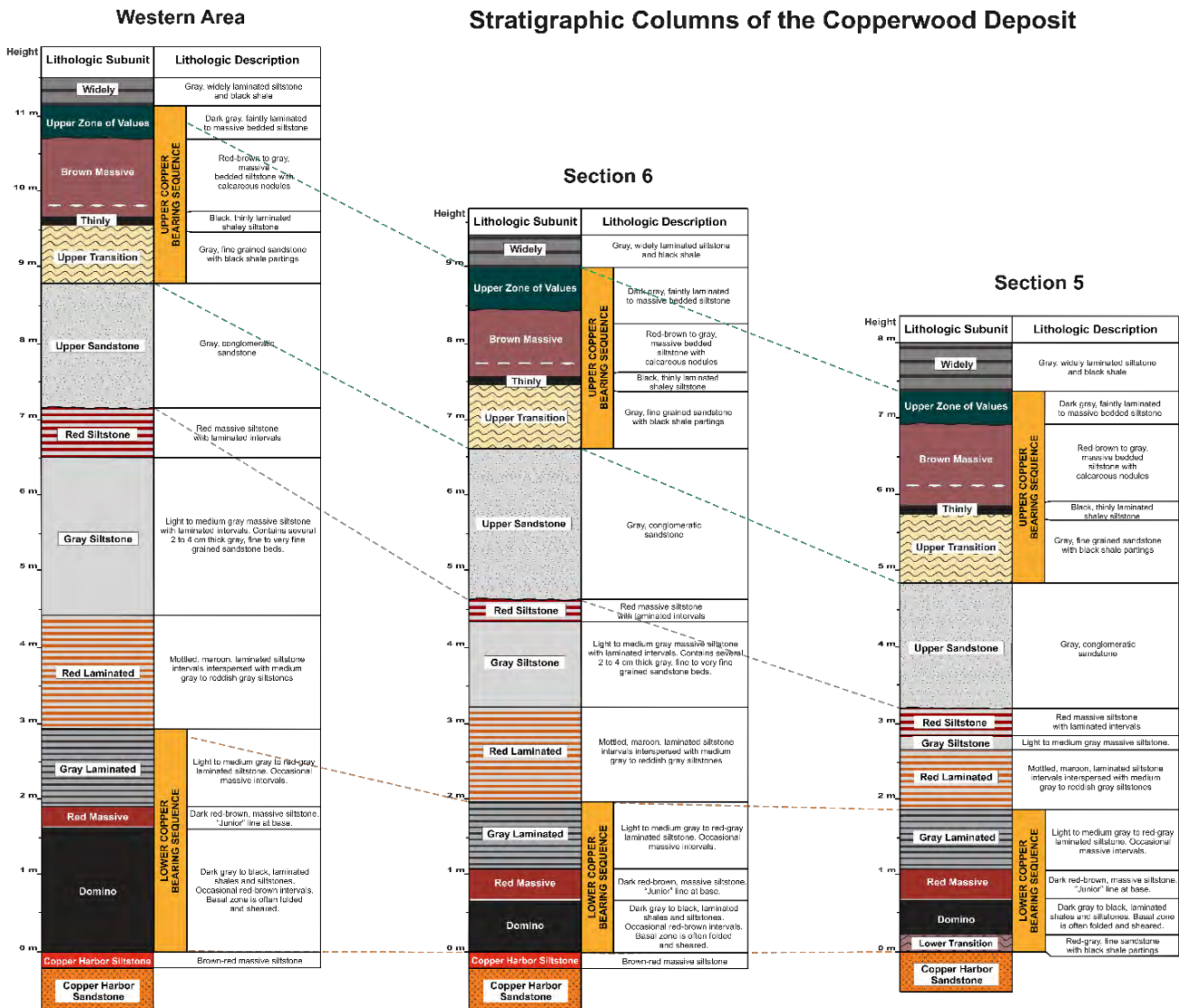
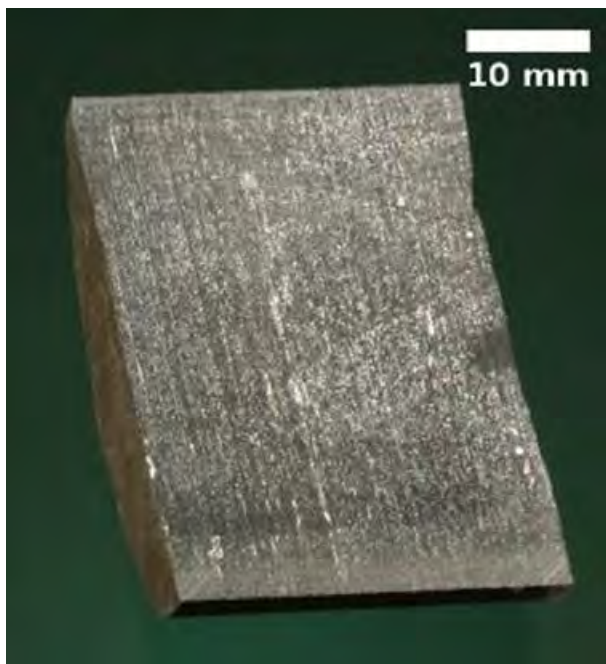


Figure 7.5: Mineralized Domino Subunit Drill Core Sample

The Red Massive subunit overlies the Domino consisting of massive dark red-brown siltstone with beds of fine-grained sandstone. The contact with the Domino is sharp and easily recognized in drill core as an abrupt change from the dark-gray or black color of Domino to the red-brown of Red Massive. Towards the top of the Red Massive, the color changes from red-brown to reddish-gray. The upper contact is placed where the color changes from reddish gray to gray. This upward color change typically occurs over a thickness of a few centimeters. The Red Massive is weakly mineralized and has a mean thickness of 0.3 m and ranges from 0.0 to 1.2 m thick.

The Gray Laminated subunit contact with the underlying Red Massive is gradational. This subunit consists of light to medium-gray to reddish-gray, laminated to locally massive siltstone. Brownish layers are occasionally present in parts of the Gray Laminated interval. A 10 to 50 cm thick zone of calcareous nodules in gray siltstone occurs in all holes near the base of Gray Laminated. The upper contact is placed where the color changes from dominantly gray to mixed maroon and gray. The transition zone is typically on the order of 0.1 m thick. The Gray Laminated is mineralized and has a mean thickness of 1.0 m and ranges from 0.0 to 2.6 m thick.

The LCBS is overlain by the following subunits: Red Laminated, Gray Siltstone, Red Siltstone and Upper Sandstone. These subunits are not mineralized except the Red Laminated where copper-rich mineralization occurs in the lower 0.3 m of the subunit.

The Red Laminated subunit overlies the Gray Laminated. This subunit is characterized by laminated siltstone with a bimodal color distribution of maroon to red-brown and gray. Typical Red Laminated has mottled or wavy maroon intervals interspersed with medium gray to reddish gray siltstone. The Red Laminated sub-unit has a mean thickness of 1.4 m and ranges from 0.0 to 3.1 m thick.

The Gray Siltstone and Red Siltstone subunits overlie the Red Laminated. The Gray Siltstone consists of a laminated, light and dark gray siltstone. The Red Siltstone is a red-gray to red-brown siltstone.

Most minerals in the siltstone-dominated lithologies of the sequence are too fine-grained to be identified in drill core using only the aid of a hand lens. An exception is calcite, which fills thin single millimeter-scale healed fractures that cut across bedding typically at high angles. At least a few calcite-healed fractures are found in the sequence of every hole. The non-sulfide mineralogy of the sequence is consistent with low-temperature and low-pressure metamorphism.

This sequence of rocks is overlain by the Upper Sandstone subunit of the Nonesuch Formation. The contact is sharp. The Upper Sandstone consists of generally massive gray siltstones and sandstones, with minor gray conglomeratic, white sandstone and red-brown siltstone lenses.

7.2.2 Upper Copper Bearing Sequence

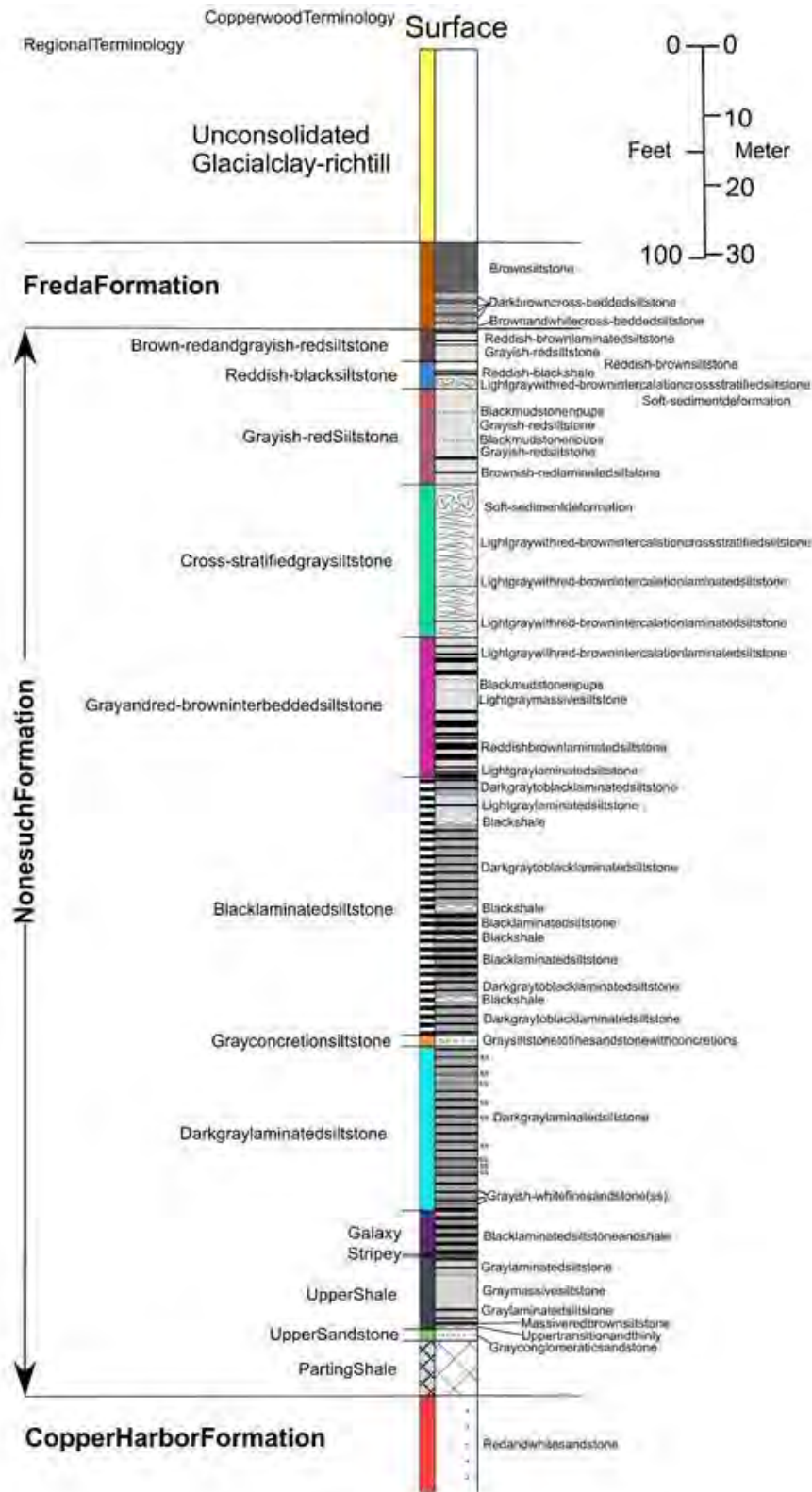
The UCBS, which lies above the Upper Sandstone subunit, is comprised of the following subunits: Upper Transition, Thinly, Brown Massive and Upper Zone of Values.

The Upper Transition subunit is composed of finely interbedded coarse grey siltstone with dark grey shaley siltstone and is approximately 0.6 to 1.2 m thick. It is overlain with a sharp contact by the Thinly subunit, composed of thin, black laminated shale, typically 6 to 10 cm thick. There is a gradational contact to the Brown Massive subunit, composed of massive, brownish red siltstone 0.6 to 1.6 m thick and contains oval shaped calcareous nodules 2 cm thick. The uppermost subunit of the UCBS is the Upper Zone of Values, composed of faintly laminated, greenish black shaley siltstone 0.1 to 1.0 m thick, and is less distinct than at White Pine. The bottom contact is very gradational with splotchy shale partings.

7.2.3 Nonesuch Undivided and Freda Formations

Above the UCBS, subunits of the Nonesuch Formation have not been formally named. They include a series of siltstone and shale horizons shown in Figure 7.6. Their color varies from light to dark gray and black with lesser amounts of reddish brown, oxidized zones. There are variable amounts of calcareous material occurring as disseminations, blebs and veinlets. The Freda Formation at Copperwood consists mainly of reddish brown to brown siltstone and fine sandstone.

Figure 7.6: Stratigraphic Column of the Project Area Geology



7.2.4 Structure

All the units on the southwestern limb of the Presque Isle Syncline dip gently to the north and vary from 12° in the south near the interface with overburden to 8° in the north near the synclinal axis. The lower contact of the Nonesuch Formation subcrops beneath 20 to 35 m of unconsolidated glacial sediments and is approximately 275 m beneath the bedrock surface about 1.3 km to the north.

Figure 7.8 through Figure 7.11 present a series of cross sections within the Copperwood Project area. The cross sections show the constant gentle dip of the LCBS across an east-west distance of 1,220 m. Figure 7.12 presents a longitudinal view of the Copperwood Deposit.

Highland has delineated a low angle reverse fault that dips 23 degrees to the north-northwest in the western, thicker part of the Copperwood Deposit, as shown in Figure 7.7.A The average vertical displacement is 4.8 (up to 8 m), and the maximum along-fault, up-dip displacement of the Domino unit is 25 m. The fault plane was modeled from eleven Highland drill holes in total. Orvana drill hole CW-09-82 and Highland drill hole CW-17-186 are only two drill holes that intersected a repetition of the LCBS in the Deposit.

A basin-wide basal gouge exists near the bottom of the Domino and the contact of the Copper Harbor Formation. It usually occurs within the Domino a few centimeters from the bottom contact with the Copper Harbor Siltstone. It is comprised of a weaker, deformed shale/siltstone and its contacts are sharp and parallel to laminae. The basal gouge was identified in 177 drill holes within the Deposit and has a median thickness of 5.1 cm and an average thickness of 7.1 cm, as shown in Figure 7.7. The stiffness of the gouge is variably soft, moist (clay-like) to hard, dry (striated) and is sometimes healed.

Figure 7.7: Thrust Fault and Basal Gouge Thickness

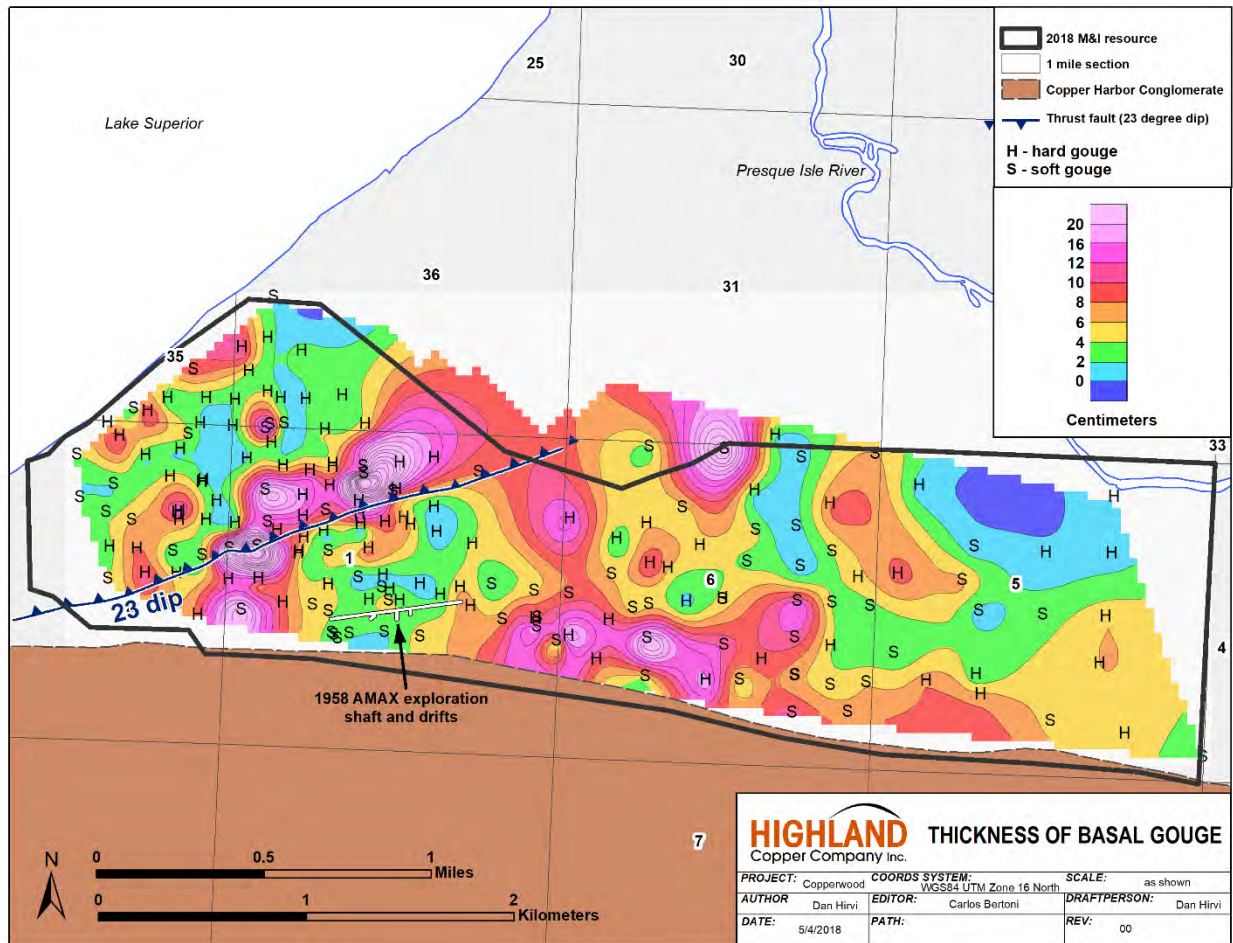


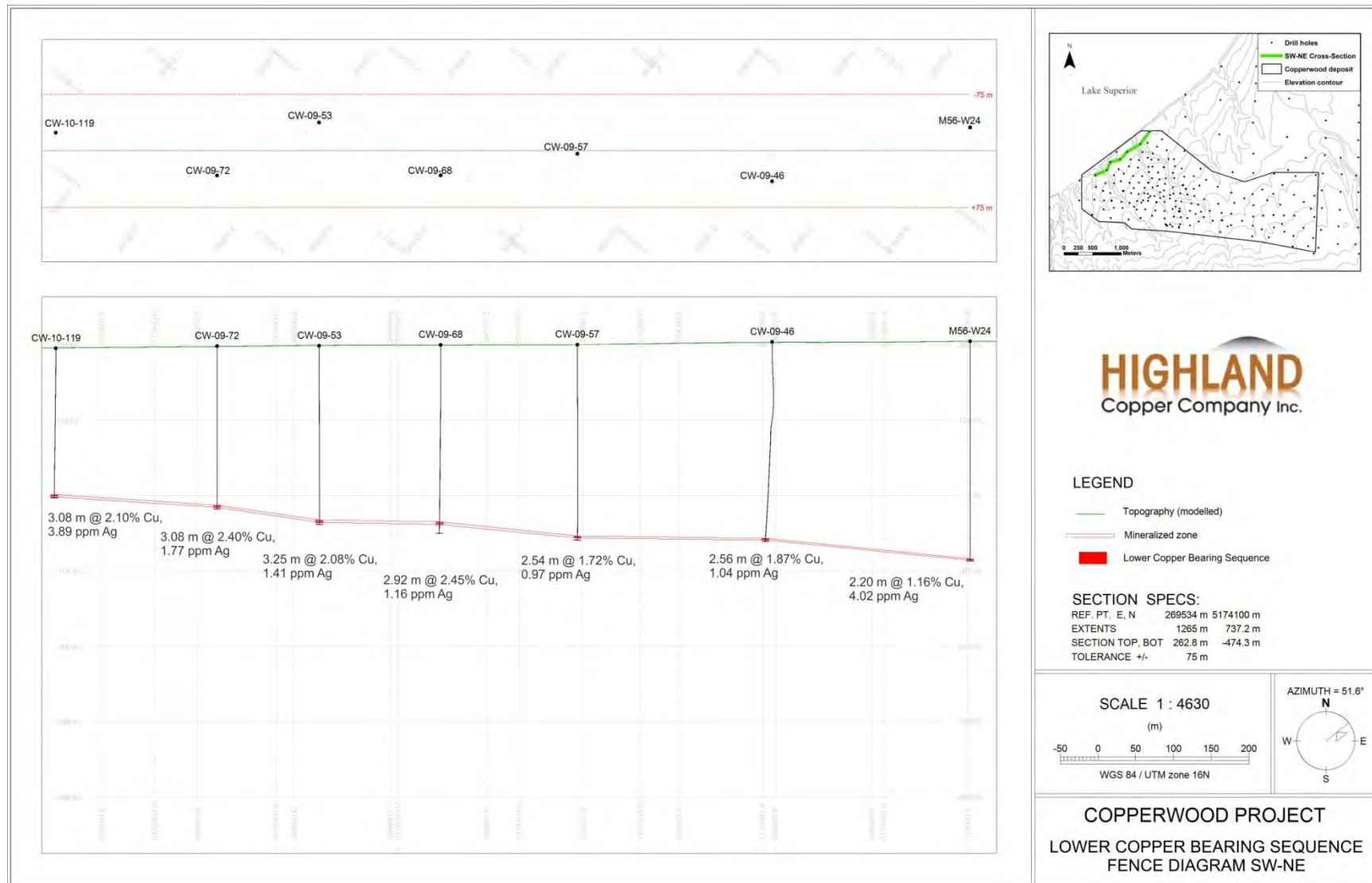
Figure 7.8: Cross Section Showing the LCBS – South West-North East Fence Diagram – Western Copperwood Deposit

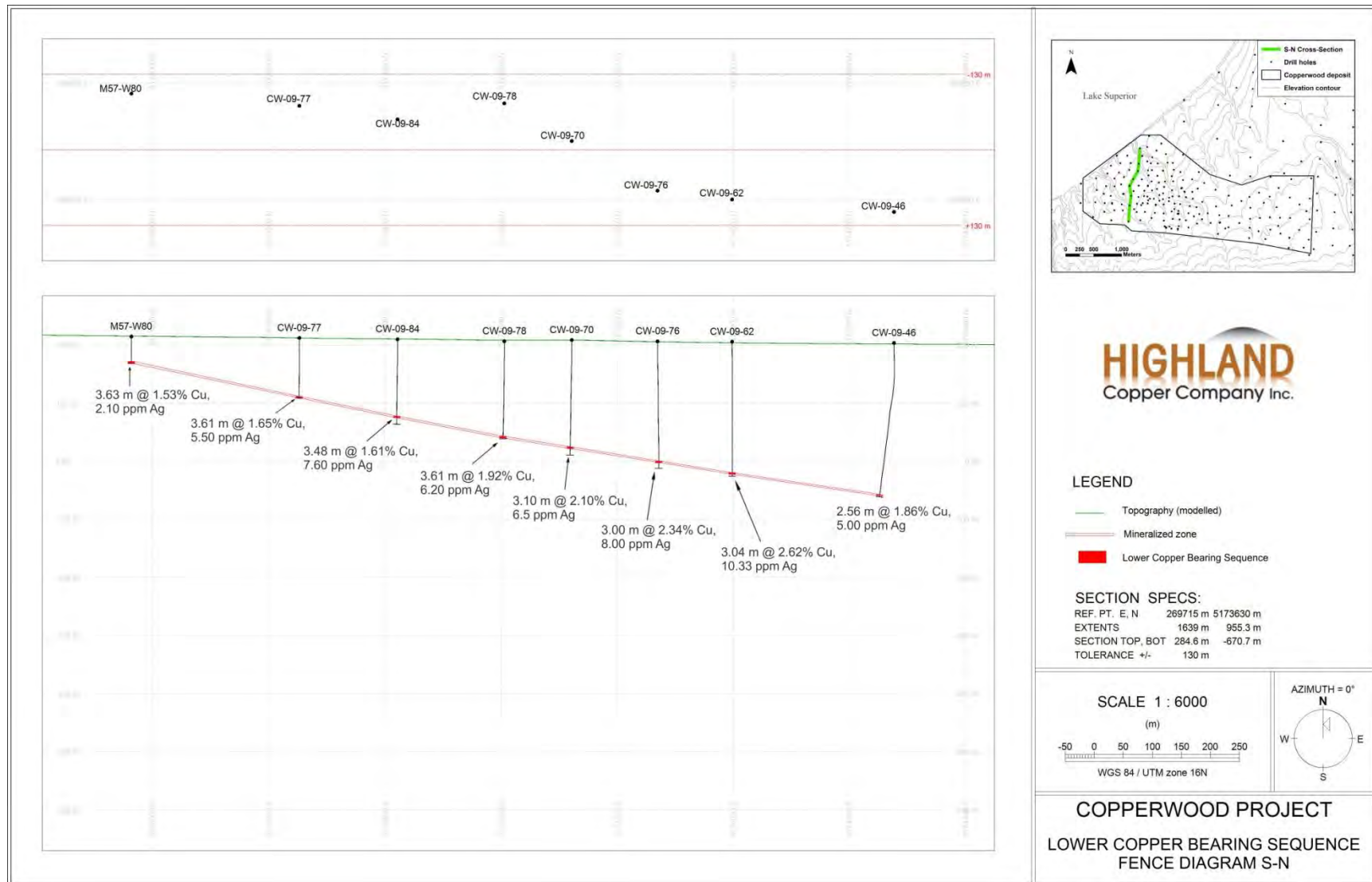
Figure 7.9: Cross Section Showing the LCBS – South-North Fence Diagram – Western Copperwood Deposit

Figure 7.10: Cross Section Showing the LCBS – South-North Fence Diagram – Central Copperwood Deposit

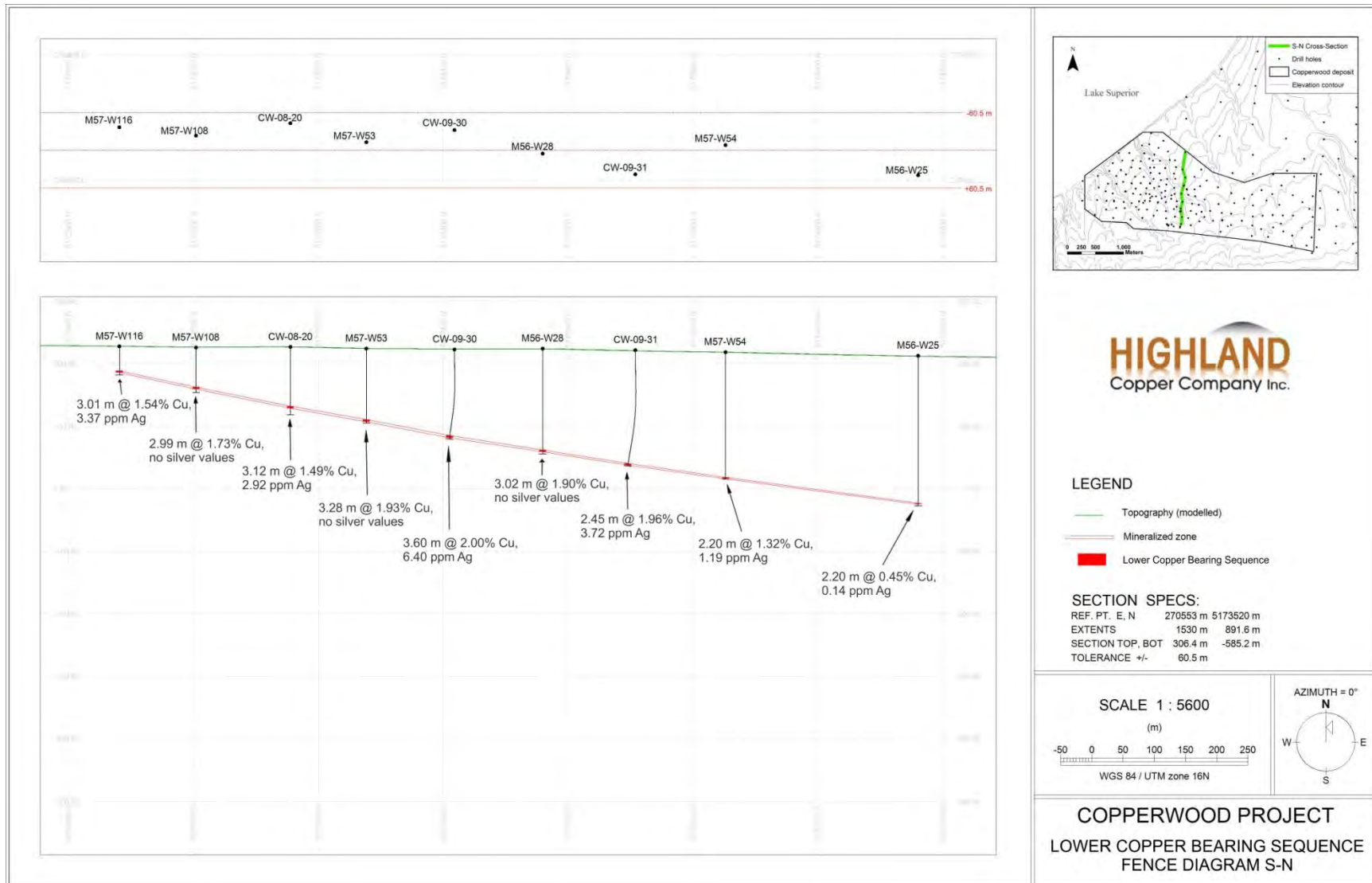


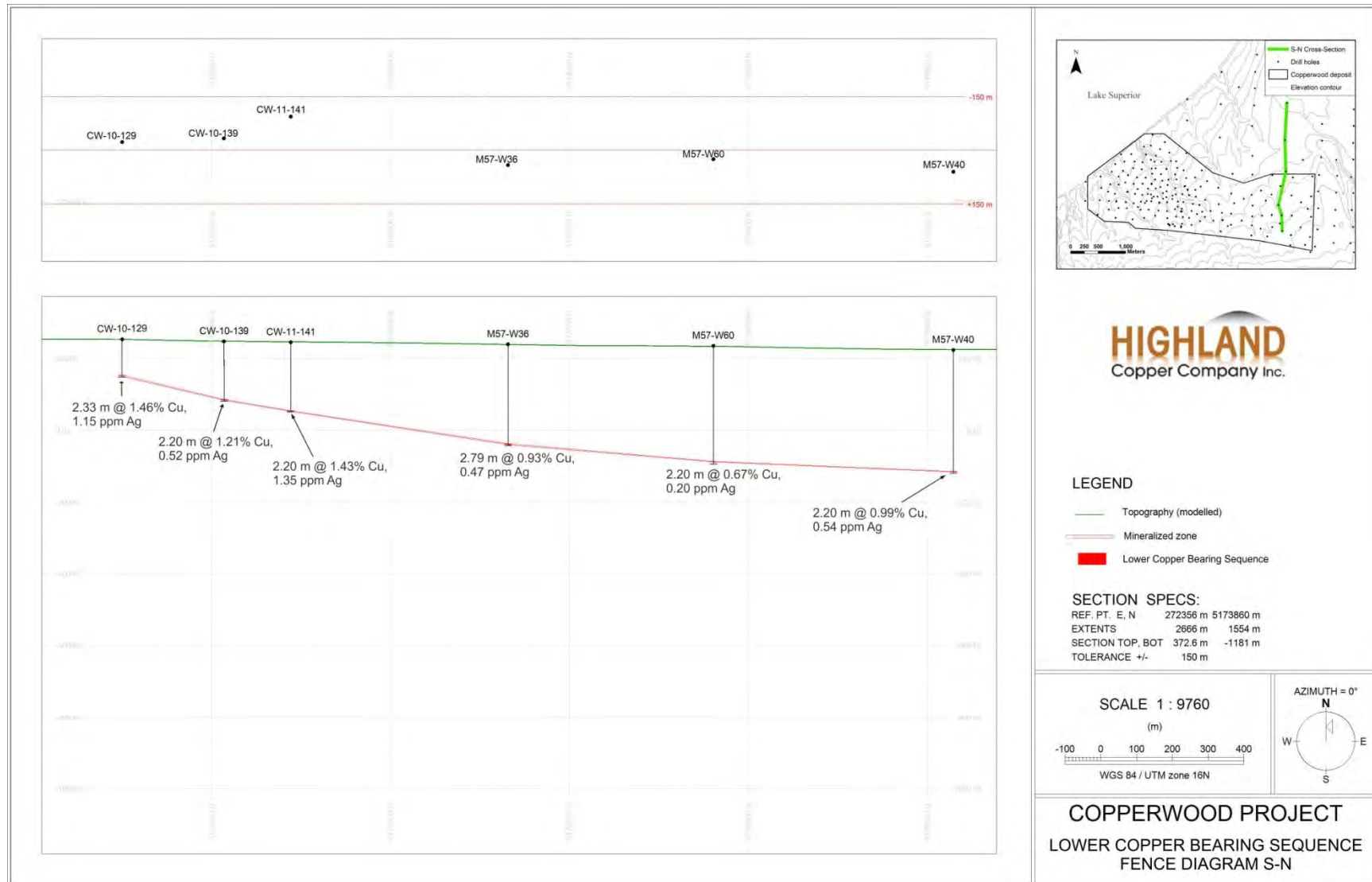
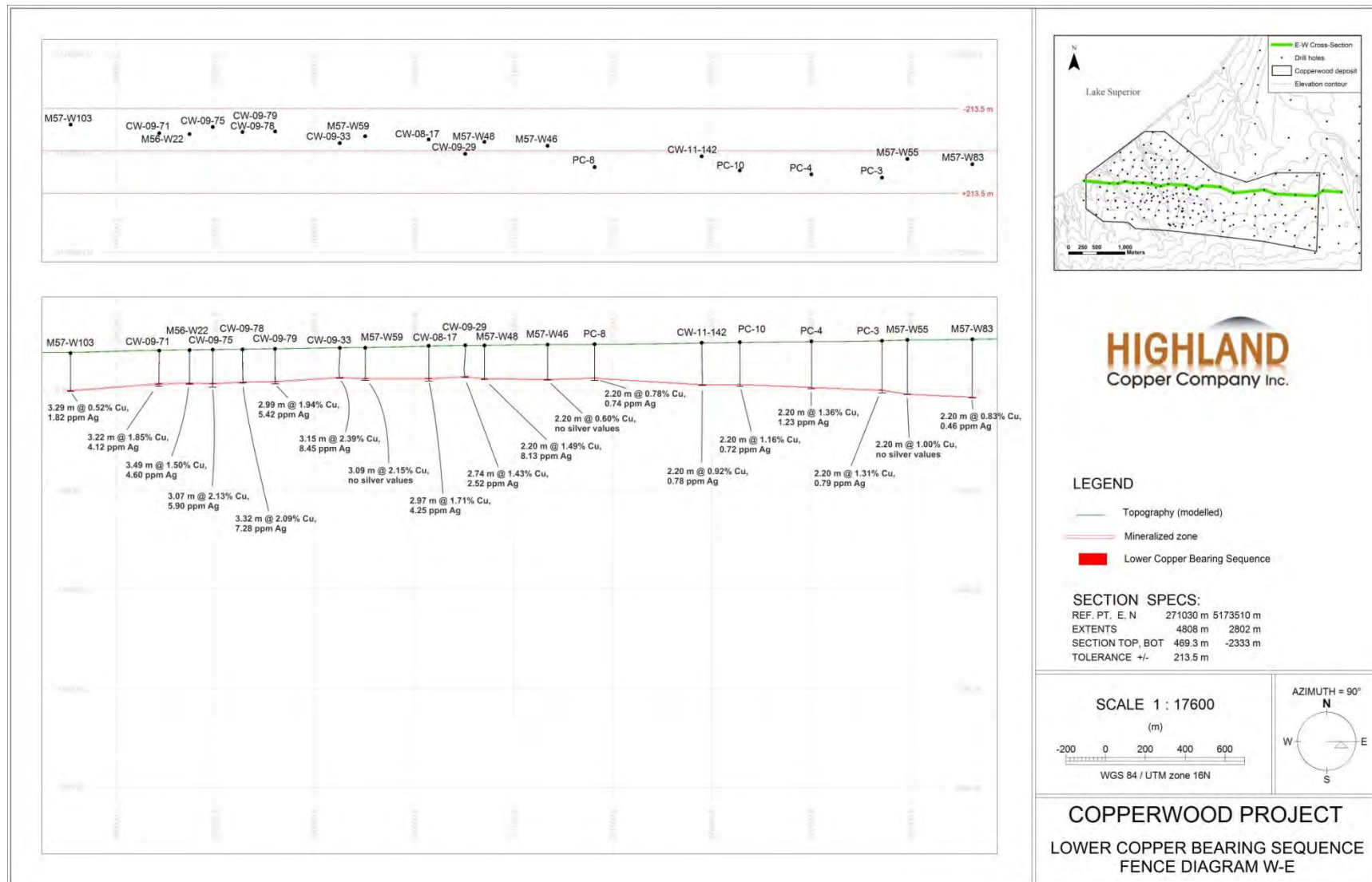
Figure 7.11: Cross Section Showing the LCBS – South-North Fence Diagram – East Copperwood and Satellite Deposits

Figure 7.12: Longitudinal Section Showing the LCBS – West-East Fence Diagram – Copperwood and Satellite Deposits

7.3 Mineralization

The Copperwood and Satellite Deposits are situated on the limbs of the Presque Isle Syncline within the Nonesuch Formation. The Nonesuch Formation contains two mineralized sequences, one located at the base and called LCBS, and a stratigraphically higher one called UCBS, separated by poorly mineralized sediments from 0.5 to 6.0 m thick.

The Domino is the main mineralized subunit, averaging 1.6 m in thickness, but thinning to about 0.5 m on the eastern edge of the Copperwood Deposit. Copper assays at Copperwood are remarkably consistent within individual units with mean copper grades of 2.58 wt.%, 0.39 wt.%, and 1.32 wt.% for the Domino, Red Massive and Gray Laminated subunits, respectively. The Red Laminated demonstrates a localized 1% increase in copper grades occurring at the base of the unit adjacent to the Gray Laminated. Silver is also present, with mean grades of 5.5 g/t.

Chalcocite is the only observed copper sulfide-bearing mineral at Copperwood, occurring principally as disseminations within shale and siltstone. Individual disseminated grains of chalcocite are most commonly very fine-grained, approximately 5 to 50 microns (μ :) in diameter. Chalcocite occurs as free grains and as complex grains where it appears to have replaced pyrite grains, as evidenced by remnant patchy domains of an iron oxide mineral (probably hematite). In the highest-grade samples, located in the top 0.3 m of Domino subunit, chalcocite occurs as layers that are parallel to laminations in the rock. These layers are usually less than 2 mm thick. Occasionally, ovoids of chalcocite occur that are up to 3 mm in their long axis. They possibly result from the replacement of organic carbon.

There is an overall negative correlation with the degree of oxidation of the host rock within the LCBS and the abundance of chalcocite within the LCBS. The dark-gray to gray colored Domino subunit has the highest copper grades; the medium to light-gray-colored Gray Laminated has medium copper grades; and, the red-brown colored Red Massive has distinctly the lowest copper grades.

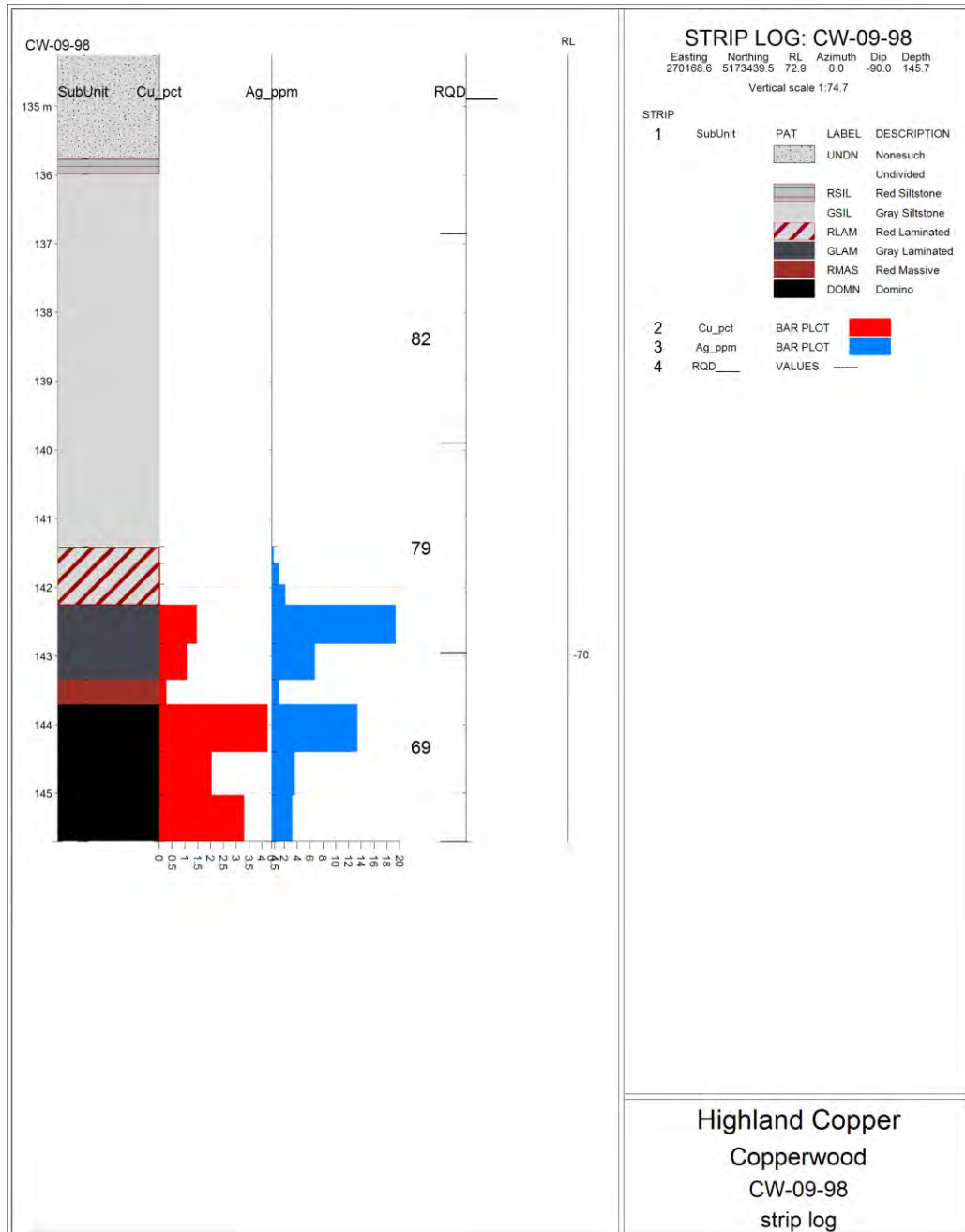
Grade profiles for each of the LCBS units show that there is a natural break in the grade profile, at approximately 1 wt.% copper. The 1 wt.% copper grade is a natural cut-off and is extensively used in Zambian and other African sediment-hosted copper deposits, where most intercepts grade a few tenths of a percent copper above or below the mineralized interval and well over 1 wt.% copper inside the mineralized interval.

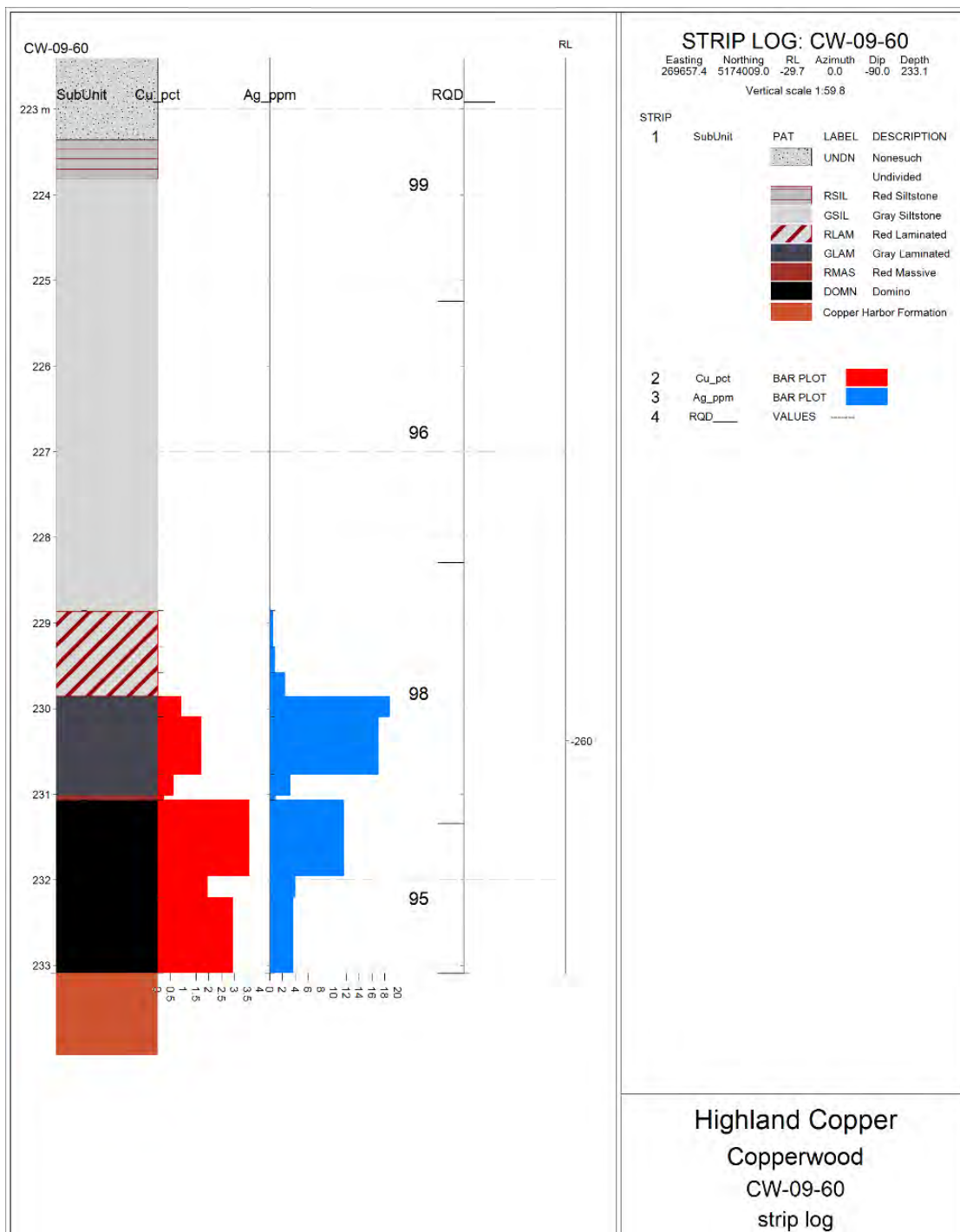
The UCBS hosts the same style of chalcocite mineralization as the LCBS, but contains trace to no chalcocite mineralization the western, thicker part of the Deposit. The copper grade gradually increases towards the center of the Western Syncline and Section 6 contains an UCBS grade of 0.5 to 0.8 wt.%

copper. The UCBS becomes more mineralized in Section 5 and has a copper grade greater than 1.0 wt.% in the eastern half of the section where the thickness of the UCBS ranges from 2.5 to 3.2 m. Here the copper grades are greater than 1.5 wt.%, 3.0 wt.%, 0.3 wt.%, and 0.9 wt.% for the Upper Transition, Thinly, Brown Massive, and Upper Zone of Values subunits, respectively. The Upper Transition and Thinly units are of economic interest and were the focus of the resource estimate.

Although the average grades of silver in the Domino and Grey Laminated are of low economic importance (4-6 g Ag/t), the spatial distribution of silver grades are highly variable. A sub-population of higher-grade silver assays (up to 108 g Ag/t) are present in the Domino to the north of the Copperwood Deposit, located within the keel of the syncline. The vertical distribution of copper and silver grades within the LCBS are shown in Figure 7.13.

Figure 7.13: Strip Log Showing Typical Distribution of Copper (red) and Silver (blue) in the LCBS





7.4 Comparison to White Pine Deposit

The White Pine deposit is located about 30 km northeast of the Copperwood Project. The White Pine mine operated from 1952 to 1995, producing over two million metric tonnes of copper. The White Pine and Copperwood deposits are both considered stratiform copper deposits hosted by shale and siltstone.

Geologically, the sites encompass the same overall stratigraphic position at the base of the Nonesuch Formation. The chalcocite mineralization is interpreted to have the same origin and the two deposits mirror each other on either side of the Porcupine Mountains volcanic structure.

The similarities and differences between White Pine and Copperwood are described and commented below. A comparison of the stratigraphy of the base of the Nonesuch Formation at the Copperwood and the White Pine North (the area to the north and northeast of the mined-out part of the deposit) areas is depicted in Figure 7.14. The White Pine North stratigraphy was developed by Highland based on its 2014 drilling of the deposit.

The LCBS at Copperwood is the partial equivalent of the Parting Shale sequence at White Pine. The term “Parting Shale” describes a mining configuration, not a stratigraphic sequence and includes three non-mineralized subunits. While the LCBS is typically twice as thick at Copperwood, the thickness of the mineralized horizons is about the same, 2.5 m thick at both sites. The most significant difference is that the Domino subunit at Copperwood is much thicker, averaging 1.6 m, compared to 0.6 m at White Pine. Since the Domino is the highest-grade subunit, the average copper grade at Copperwood is higher than White Pine.

Another difference between the two sites is the potential mining configurations. Both sites have two mineralized sequences: the Parting Shale and Upper Shale at White Pine, and the LCBS and the UCBS at Copperwood. Much of the mining at White Pine included a configuration called the Full Column, which included the complete Parting Shale, the Upper Sandstone and the basal two subunits of the Upper Shale. The Upper Sandstone contains little or no mineralization, but at White Pine the dilution from this zone is compensated for by the very high-grade mineralization of the overlying Upper Transition and Thinly subunits. At Copperwood, the thickness of non-copper-bearing units between the two mineralized sequences is much greater and the use of a Full Column-equivalent configuration needs to be investigated.

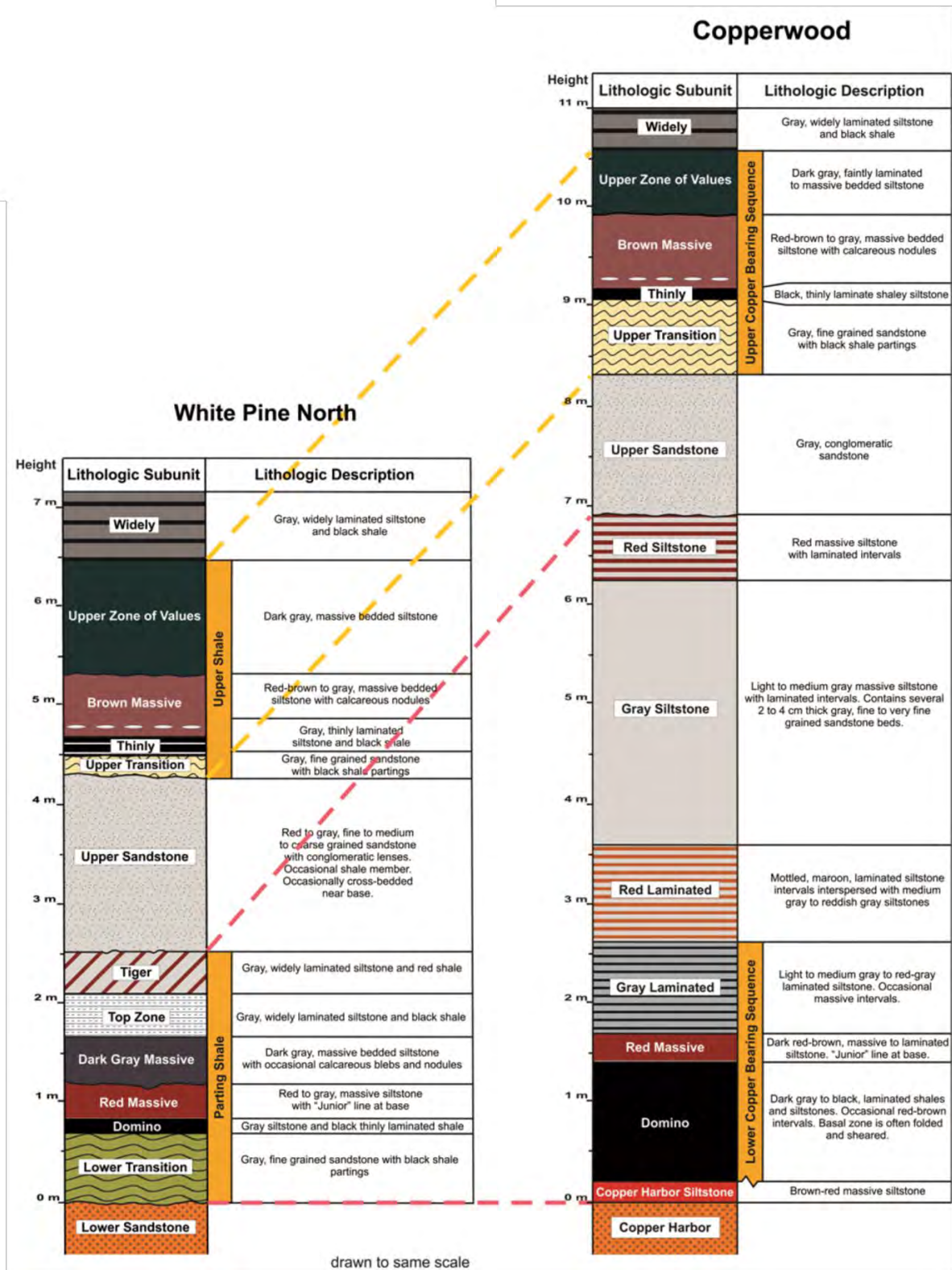
Structurally, there are significant differences between Copperwood and White Pine. The White Pine deposit straddles an anticline and a right-lateral strike-slip fault. Both the southwest and northwest domains of the White Pine deposit contain strike-slip and thrust faults. These faults are interpreted as being generated during the regional late rift compressional event. In contrast, the Copperwood deposit is structurally located on a simple dipping plane, appears to be less faulted. Only one significant thrust fault has been identified at Copperwood so far.

The mineralization type differs slightly between Copperwood and White Pine. The copper-bearing mineral at Copperwood is essentially fine-grained chalcocite. In contrast, the White Pine deposit has two distinct types of mineralization; about 80% to 85% of the copper occurs as chalcocite and the rest as native copper.

At White Pine, most of the native copper occurs as disseminations and coatings along fractures. Some of the native copper occurs as sheets and veinlets along fault zones. There does not appear to be a similar style of mineralization at Copperwood.

The copper grades are very consistent within individual units averaging 2.58 wt.%, 0.39 wt.% and 1.32 wt.% for the Domino, Red Massive, and Gray Laminated, respectively, in the Copperwood Deposit. A similar pattern of relatively consistent grades occurs at White Pine with the stratigraphic equivalent subunits, the Domino, Red Massive and Dark Gray Massive.

Figure 7.14: Comparison of Copperwood and White Pine North Stratigraphy



8. DEPOSIT TYPES

The following descriptions and conclusions related to sediment-hosted copper deposits have taken in considerations the work by several authors, including Gustafson and Williams, 1981; Kirkham, 1989; Lindsey *et al.*, 1995; Cox *et al.*, 2003; and Hitzman *et al.*, 2005.

The Copperwood Project consists of sediment-hosted stratiform copper deposits. Such deposits consist of copper and copper-iron sulfide minerals hosted by siliciclastic rocks in which a relatively thin (typically less than 3 m thick) copper-bearing zone is mostly conformable with stratification of the host sedimentary rocks. Copper occurs as disseminations and veins.

Sediment-hosted deposits have been grouped on the basis of the reductant into three subtypes: reduced facies, red-bed copper and Revett Copper. They can also be classified based on basinal setting into two subtypes: Kupferschiefer and red-bed. The reduced facies and Kupferschiefer subtypes are similar. Examples of the reduced facies or Kupferschiefer subtypes include most of the deposits within the Central African Copperbelt (such as Nkana, Nchanga, Mufulira, Tenke–Fungurume and Kolwezi), the Kupferschiefer (Germany/Poland), Redstone (Canada) and White Pine (USA).

The following are common features of the reduced facies or Kupferschiefer subtype sediment hosted copper deposits as summarized by Cox *et al.*, 2003 and Hitzman *et al.*, 2005.

Geological setting: Intracratonic rift with coarse-grained sub-aerial sediments overlain by fine-grained sediments or restricted marine setting/basin margin followed by widespread euxinic marine deposits; near paleo-equator; partly evaporitic on the flanks of basement highs; footwall sediments highly permeable; and, host ranging in age from early Proterozoic to late Tertiary, but predominate in late Mesoproterozoic to late Neoproterozoic.

Host Rocks: Marine or lacustrine; thin-bedded to finely-laminated green, black or gray shale, thinly laminated tidal/sabkha facies or reefoid carbonate rocks, and dolomitic shales; common organic carbon and finely disseminated pyrite; tend to have large lateral extent; and, during transgression over oxidized sequences of hematite-bearing sandstones, siltstones, and conglomerates (red-beds).

Mineralization: Chalcocite and other Cu_2S -CuS minerals + bornite are diagnostic; typical minerals hematite–chalcocite–bornite–chalcopyrite–pyrite; may be zoned with chalcocite–bornite central, chalcopyrite–pyrite medial, galena–sphalerite peripheral; finely disseminated; copper sulfides replace framboidal or colloform pyrite; and, carbon-rich materials in favorable host rocks but usually consumed by redox reactions during copper mineralization processes.

Alteration: Diagenetic alteration minerals in host rocks and underlying red-beds (albite, potassic feldspar, chlorite, quartz, carbonate minerals, dolomitization, etc.); and, bleaching of red sediments to greenish gray or light gray where in contact with reducing fluids.

Timing of mineralization: Textures and fabrics indicate that all were precipitated after host-rock deposition; exact timing variable; and, may take place early to very late in the diagenetic history or in the post-diagenetic history.

Mineralization controls: Basin-scale fluid flow system in highly permeable footwall red-bed sediments; giant deposits form from multiple stages or long-term progressive fluid flow; copper is mobilized from footwall red-beds by oxidizing low-temperature brines and metal carried as chloride complexes; mineralizing fluid focusing by marginal basin faults, stratigraphic pinch-outs or anticlinal traps; copper mineralization in lowermost reduced beds overlying red-beds; and, pyritic black shale/siltstone and algal mats, perhaps hydrocarbon fluids, provide source of biogenic sulfur and reducing environment for precipitation of copper.

Global-scale grade-tonnage model: Median reduced facies deposit has 33 Mt and 2.33 wt.% Cu.

The Copperwood Project deposits are interpreted as being classic examples of a reduced-facies sediment-hosted copper type, formed during early diagenesis. Syn-sedimentary faults may have provided important conduits for cupriferous brines flowing from underlying red beds of the Copper Harbor conglomerate into the reduced silt and shale of the Nonesuch Formation, where main-stage copper sulfides and native copper were precipitated.

9. EXPLORATION

9.1 Exploration History

All pre-2014 exploration activities undertaken on the Copperwood Project were performed by various owners, namely Orvana, AMAX and United States Mineral Refining Company (“USMR”), and Highland.

A summary of historical exploration activities conducted on the Copperwood Project is presented in Section 6 of this Report. The following sections focus primarily on the exploration programs implemented by Orvana between 2008 and 2013 and Highland.

9.2 Orvana Exploration Programs

Beginning in 2008, Orvana conducted a series of exploration drilling programs at Copperwood (2008, 2009, 2010, 2011 and 2013). Additionally, Orvana commissioned several independent technical reports for the Copperwood and Satellite Deposits in 2010 and 2011.

Orvana completed a major resampling and surveying program for Section 6 and the Satellite Deposits. During late 2010, Orvana drilled an additional 23 diamond drill holes in the Project area and 15 new holes in Section 6. The resampling program involved the collection of archived core, rejects and pulps from 87 historic drill holes, which included all but one of the legacy drill holes in Section 6 (drill hole PC-13).

Orvana contracted Coleman Engineering Co. of Ironwood, Michigan, to survey historical drill collars in the Satellite Deposits area. They were able to locate and survey 111 drill hole collars, and coordinates were estimated for an additional 56 drill holes based on the presence of sumps or other evidence was observed, but no monuments were found.

9.3 Highland Exploration Program

In 2017, Highland carried out a drilling program comprising of 35 HQ diameter, five PQ-diameter drill holes and an additional 13 wedges for a total of 7,666 m of core. The drilling provided 526 samples for copper and silver assaying and 607 kg taken for metallurgical testing. The 2017 drill program was designed to upgrade the Inferred Mineral Resources at the eastern section of the deposit – including Section 5, obtain metallurgical samples and carry out geotechnical studies to refine the mining plan. Nineteen holes were acoustic televised by DGI Geoscience (www.dgigeoscience.com) for an improved understanding of the

rock's in situ geotechnical characteristics. An additional hole was optically televised by DGI Geoscience in December 2017 for geotechnical studies requested by Golder.

In early 2018, Highland completed a drilling program comprising of eight diamond drill holes and one wedge. The aim of this drilling program was to upgrade the remaining portions of Inferred Resources in the eastern portion of the deposit for inclusion into the Feasibility Study.

9.4 Airborne Geophysical Studies

There are no known surface geophysical exploration programs for the Copperwood Project. Delineation of mineralization has primarily been completed through drilling from surface and limited underground channel sampling.

9.5 Geochemical Surveys

There are no known surface geochemical exploration programs for the Copperwood Project. Delineation of mineralization primarily has been completed through drilling from surface and limited underground channel sampling.

10. DRILLING

10.1 Drilling History

Before 2017, all drilling activities undertaken on the Copperwood Project were performed by previous owners, namely Orvana, AMAX and United States Metal Refining Company (“USMR”).

The Historical drilling on the Copperwood Project property and surrounding leases was completed in two different phases. USMR and BCM drilled 184 core holes in 1956 and 1958. BCM drilled 23 holes in Section 6 in 1959. USMR drilled an additional 119 drill holes in the Satellite Deposits between 1956 and 1958. The core diameter for these holes was between 3.01 cm (AX size core) and 4.20 cm (BX size core). The longest hole reached a depth of 354 m. The second phase of drilling at Copperwood commenced in 2008, with Orvana drilling five holes for environmental purposes. These drill holes intersected significant copper mineralization. Orvana subsequently completed 82 drill holes in 2009. Orvana commissioned an NI 43-101 compliant resource estimate from AMEC and followed up on this during 2010 with 24 additional core holes for 2,801 m in order to firm up the resource, to collect metallurgical and geotechnical data and to investigate a suspected fault. Another 15 holes, totaling 1,250 m, were cored in Section 6 during 2010 to verify copper mineralization in area. In 2013, Orvana drilled 21 drill holes for collecting metallurgical and geotechnical studies; of which 13 holes were drilled primarily for metallurgical purposes and seven holes were drilled primarily for geotechnical purposes with one hole drilled for both metallurgical and geotechnical purposes.

The 2017 drilling program began in February 2017 and finished in August 2017. An additional program began in November and ended in December 2017 in order to address specific geotechnical and metallurgical questions. The 2017 drilling program in total contained 40 diamond drill holes and 13 wedges located at the “Main”, Section 5 and Section 6 zones. Only 17 drill holes were assayed for copper, silver and multi-elements. The remainder of the holes were used for metallurgical and geotechnical test work. In January 2018, Highland began another drill program of infill drilling in Section 5 to upgrade Inferred Mineral Resources to Indicated category. This drill program consisted of eight holes, one wedge, and the completion of CW-17-184 for a total of 2,925 m which was completed in March 2018.

Table 10.1 summarizes the completed drill holes.

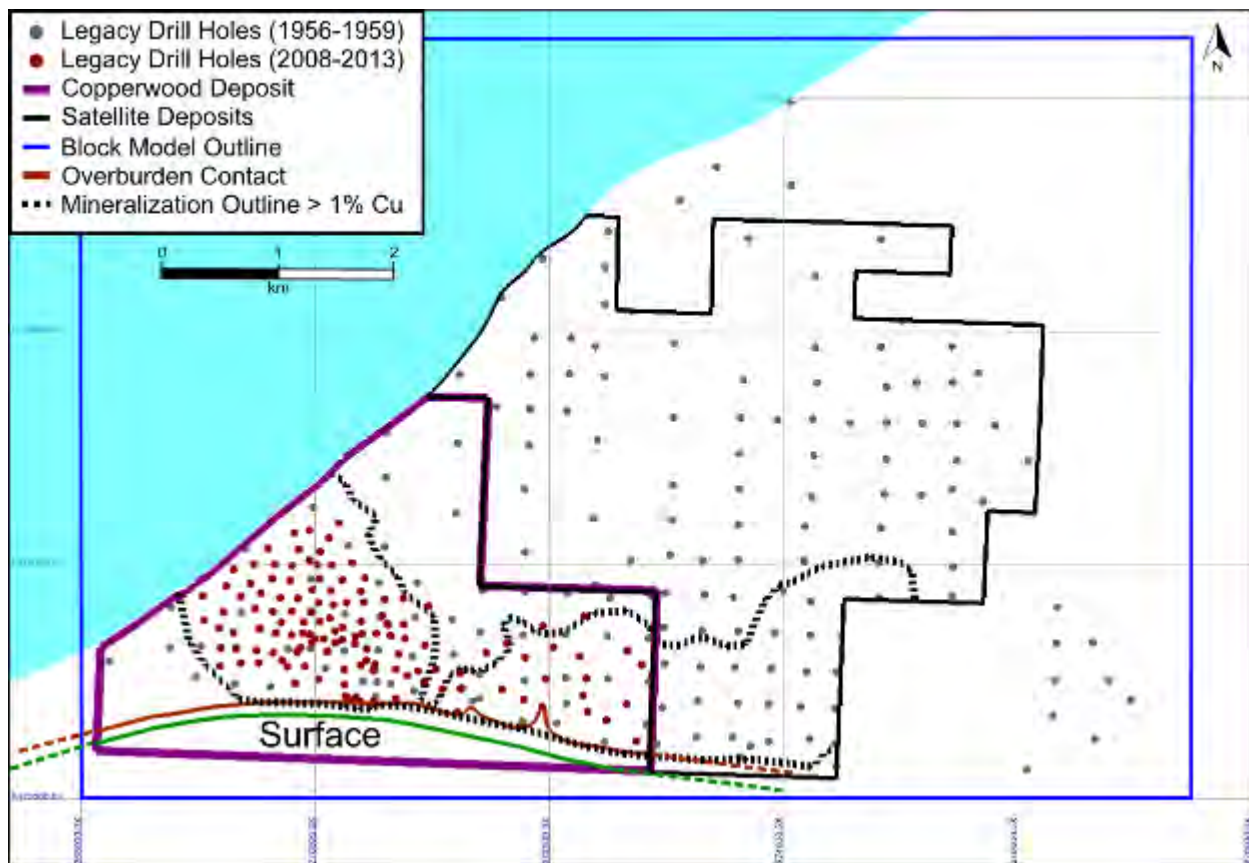
Table 10.1: Drilling Statistics by Company and Exploration Campaign

Company	Period	Core Size	Drill Hole Count	Length (m)	% of Total Drilling
USMR	1956 to 1958	BX & AX	161	34,050	49%
BMC	1959	BX & AX	23	3,998	6%
Orvana	2008	NQ	6	744	1%
Orvana	2009	NQ	82	12,858	18%
Orvana	2010	NQ	33	4,274	6%
Orvana	2011	NQ	4	776	1%
Orvana	2013	HQ	21	2,814	4%
Highland	2017	HQ & PQ	40*	7,666	11%
Highland	2018	NQ	8**	2,925	4%
All Programs	1956 to 2018	BX, AX NQ & HQ	378	70,105	100%

*40 drill holes and an additional 13 wedges, **8 drill holes and one additional wedge

Most of the drilling was undertaken on the southwestern limb of the Presque Isle Syncline, where the LCBS dips to the north at 10° to 15°. Most of the drilling has been vertical; therefore, intercepts are slightly greater than true widths.

Figure 10.1 shows the location of the legacy pre-2017 drill holes.

Figure 10.1: Plan View of the Historical Drilling (2017/18 Highland drilling excluded)

10.2 Drilling Procedures

The 2017 and 2018 drillings were performed by IDEA Drilling (www.ideadrilling.com), a company based in Virginia, Minnesota, which used Atlas Copco CS 14C, Longyear LF90 and Hagby track-mounted rigs. In addition, a truck-mounted Atlas Copco CT 14 was used with metric HQ rods and all the usual ancillary drilling equipment (Figure 10.2). For the NQ and PQ-diameter holes, rods were in Imperial units (10 ft / 3 m). Geologists converted the drill blocks to meters at the core logging facility. All drill holes were cased to bedrock to limit and prevent contact with groundwater and were cemented from bottom to top, as per State of Michigan NREPA Part 625, with the exception of holes which were left open for televising and will be cemented at the next opportunity when frozen conditions permit. All equipment and vehicles were cleaned to limit the potential for introduction of exotic and invasive plants. All drill cuttings and sump water from Section 5 were disposed off-site within sumps dug on the company property in Sections 1, 2 and 6.

Figure 10.2: Winter Drilling at the Section 5 Area



10.2.1 Collar Surveys

Coleman Engineering Company from Ironwood, MI, using a combination of conventional survey, RTK GPS and static GPS methods, surveyed the collar coordinates. The static GPS field data was submitted to OPUS for determining coordinates and elevations and used a Trimble S7 robotic total station or a Sokkia GRX2 GPS unit. The RTK GPS survey used a Topcon Hyper V GPS unit. All data was reduced to WGS 84 UTM Zone 16 coordinates in meters. The elevations were also converted to meters in NAVD 88, Geoid 12A. Ronald K. Jacobson, professional surveyor P.S. # 46671, signed the survey work.

10.2.2 Down-Hole Surveys

The downhole surveys were measured by IDEA Drilling with a DeviShot magnetic downhole survey tool. A reading was taken at the pull of every three metres or 10 ft drill rod. The geologists on site analysed the surveys and made sure that the data downloaded correctly and indicated which surveys to reject due to casing interference.

10.2.3 Core Logging

A Highland geologist was on site to field log and preserve the mineralized zones within approximately 15 m from the bottom of the Low Copper Bearing Sequence (“LCBS”). While on site, the geologist marked natural fractures with a blue lumber crayon and made sure that the driller helper was marking mechanical breaks with a yellow lumber crayon while boxing the core. Core recovery and the boxing of the drill core were supervised before every hole was abandoned.

Detailed geotechnical and lithologic logging of the entire drill core was completed from the glacial overburden to the end of coring in the Copper Harbor Sandstone by geologists Daniel Hirvi, Eric Shepeck and Stacy Saari. Logging was completed in a secure building in White Pine, Michigan on Microsoft Excel spreadsheets using laptops (Figure 10.4). Spreadsheet templates were designed with pull-down menus to ensure that data entry was error free.

Logging was performed with a precision of 5 mm after depths were marked every meter by the geotechnical logger. Geotechnical logging was completed before lithologic logging and sampling to ensure that driller depths were correct throughout the entire core length. Geotechnical logging was completed in intervals between drill runs, between the contacts of the UCBS and the LCBS, and never exceeded three meters. Each interval was logged for depth, total core recovery, solid core recovery, RQD, fracture count, mechanical break count, vein count, vein type, vein thickness, weathering, joint set number, and weathering. Following each geotechnical interval, every discontinuity was logged for depth, discontinuity

type, alpha angle (angle to core axis), mating, planarity, roughness, weathering, infill character, infill thickness, and infill hardness.

Lithologic logging recorded bedding type, dominant grain size, percent black shale, bedding angle to core axis, and a lithologic description for each unit. Metallic mineralization style and quantity were also estimated for the UCBS and LCBS using a hand lens and handheld XRF device (Olympus Innov-X Delta Professional, model "DS-4000").

Each drill hole was photographed entirely, one box at a time after logging and samples were marked. Boxes containing remaining core cut from assay sampling and wrapped core for metallurgy were rephotographed for sample documentation (Figure 10.5).

Highland performed routine point load testing on the entire length of core (Figure 10.6), with a greater emphasis on the bottom 19 units, for a total of 5,430 tests. The Itasca Consulting Group from Minneapolis, MN, prescribed the point load and other geotechnical testing methodology. If possible, ten tests were performed in both the axial and diametral directions per subunit below the "Dark Grey Laminated Siltstone" unit. A Bemek Rock tester portable field unit with a 12.4 kip capacity was borrowed from Michigan Technological University under the supervision of Dr. Stanley Vitton.

10.2.4 Core Storage

Core from the Orvana 2008 to 2013 and Highland's 2017-18 drilling programs is stored in covered core boxes organized on core racks inside a locked facility, the former mall in White Pine, Michigan.

10.3 Sampling Method and Approach

Quarter core from HQ size or half-core from NQ size core was sent for assay. Half core was kept for metallurgical testing, and the remaining quarter core was kept for reference. Sample intervals were picked between lithologic contacts and never exceeded 0.5 m in the LCBS or the UCBS, but samples up to 1.0 m were taken in the Upper Sandstone, Red Siltstone, Grey Siltstone, and Copper Harbor Sandstone units. Typically, samples 0.25 m long were taken as a first sample outside of both the UCBS and LCBS contacts. Assay intervals were marked with a red crayon and were separated by plastic chocks after cutting. The beginning of each sample interval was marked with unique sample ID from a hand-written sample tag booklet that was later entered into a Microsoft Excel spreadsheet. Core was then sawed in half and then cut into quarters (Figure 10.3). For sampling consistency, the core cutter/sampler always took the core remaining in the left hand after cutting and placed it into the sample bag and the remaining quarter core

was returned to the box for reference. A geologist supervised the cutting and re-boxed half core for metallurgy in separate boxes labelled with the sample intervals.

Whole core metallurgical drill holes were logged, shrink wrapped, and photographed for documentation (Figure 10.8). All core including and in between the UCBS and LCBS were shrink wrapped to at least 0.5 m from the contacts.

A representative sample from each subunit conforming to an assay interval was chosen for density determination (Figure 10.7). The general location within each subunit was noted, e.g., upper, middle, lower, or entire to ensure a good distribution of measurements. If a sample contained more than one piece, then each piece was numbered starting with the top sample as “1”.

Figure 10.3: Core Saw Station at White Pine Site



Figure 10.4: Core Logging at White Pine Site



Figure 10.5: Core Photography Setup at White Pine Site



Figure 10.6: Point Load Testing (Bemek Rock tester)



Figure 10.7: Specific Gravity Station



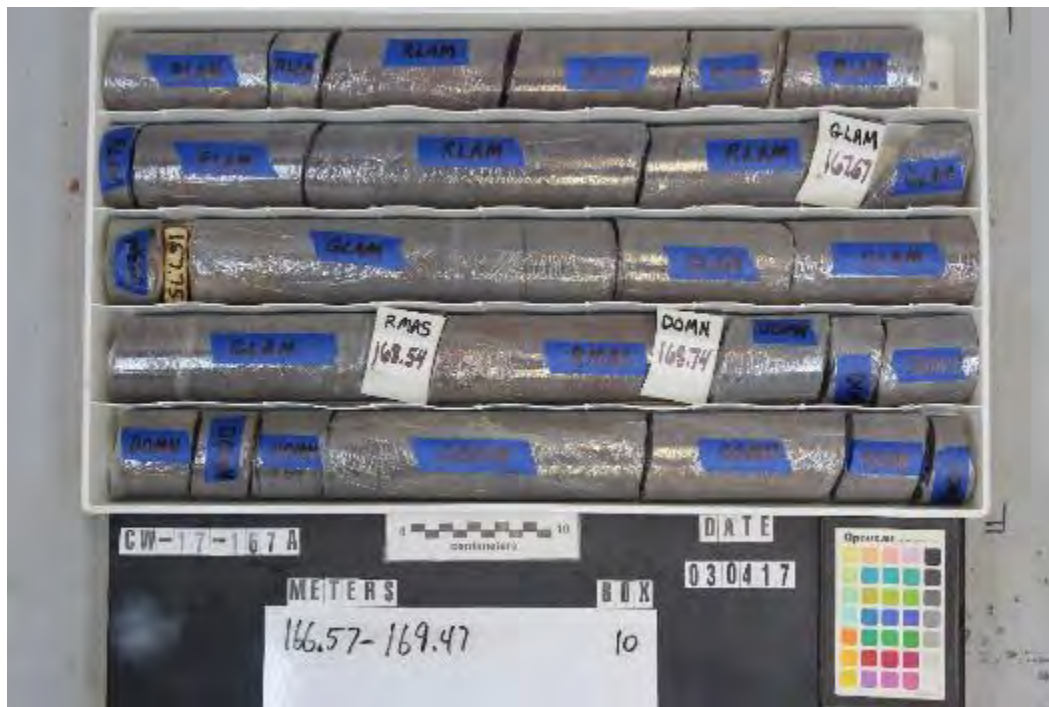
Figure 10.8: Wrapped Metallurgical Core Samples from Wedge**Figure 10.9: Top of LCBS Showing Marked Intervals for Assay Sampling**

Figure 10.10: Bottom of UCBS Showing Marked Intervals for Assay Sampling

Core recovery and the boxing of the drill core was supervised by a geologist before every hole was abandoned. An overall average recovery from the 2017-18 drilling was 98% including the LCBS.

In addition to the existing Orvana specific gravity measurements, Highland collected 57 specific gravity measurements of which 49 were completed in-house using the water immersion method and eight were performed at the Actlabs laboratory in Thunder Bay, Ontario. Summaries of this data for the LCBS and UCBS are shown in Table 10.2 and Table 10.3 respectively.

Table 10.2: Specific Gravity Summary for the LCBS

Statistical Element	Domino	Red Massive	Grey Laminated	Red Laminated
Mean	2.7	2.7	2.72	2.72
Standard Deviation	0.04	0.02	0.02	0.02
Minimum	2.63	2.65	2.68	2.68
Maximum	2.79	2.75	2.76	2.75
Coefficient of Variation	0.013	0.007	0.007	0.006
Count	76	37	91	25

Table 10.3 Specific Gravity Summary for the UCBS

Statistical Element	Upper Transition	Thinly	Brown Massive	Upper Zone of Values
Mean	2.73	2.71	2.69	2.7
Standard Deviation	0.02	0.05	0.01	0.04
Minimum	2.7	2.68	2.67	2.68
Maximum	2.76	2.79	2.7	2.79
Coefficient of Variation	0.008	0.017	0.005	0.016
Count	6	5	5	6

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

The drill hole sample data was recorded by the site geologists on standard logging templates using standard codes. The sample data was emailed directly by the geologists to the Highland independent database manager, GDAT Solutions (www.gdatsolutions.com). The analytical results and certificates were emailed directly by the analytical laboratory to GDAT Solutions. The sample and analytical data is stored in the SQL based relational database management system acQuire designed for exploration and mining data. An in-house QA/QC on import analysis was carried out for each set of analytical results in order to spot and stop potential QA/QC issues in a timely manner.

11.1 Sample Preparation and Reduction

11.1.1 Analysis

The mass of each sample was recorded prior to crushing. The entire sample was crushed to 80% passing 2 mm, with the jaw crusher cleaned and inspected before use and after each sample. For samples below 2 kg, the entire sample was then pulverized to 95% passing 150 mesh. For samples above 2 kg a split of 1 to 2 kg is pulverized. After each sample, the equipment is cleaned with pulverizing sand and visually inspected for discoloration. All remaining pulps were saved and returned to Highland for storage. Lab equipment used was a TM or Boyd Crusher, TM or LM Pulverizer, Jones Riffle Splitter, and an Agilent 735 ICP optical emission spectrometer.

All 2017 and 2018 drilling program samples submitted by Highland Copper Company Inc. (Highland) were analyzed at the Actlabs analytical laboratory in Thunder Bay, Ontario. With the exception of the 2018 assays, the samples were analysed for Ag and Cu with 4-acid ICP-OES (method code 8) and for 36 elements (Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Te, Ti, Tl, U, V, W, Y, Zn, & Zr) including Ag and Cu with ICP total digestion (method code 1F2). The 4-acid ICP-OES analysis is the higher-ranked analysis for silver and copper and to be used for silver and copper. The lower detection limits for the 4-acid ICP-OES are 0.001% for copper and 3 g/t for silver.

Due to the relatively high lower-detection limit of the ICP-OES 4-acid digest method for silver (3 g/t) and poor resolution (1 g/t), the total digest assays (with a lower detection rate of 0.3 g Ag/t) for silver were used in the resource estimation. G Mining Services Inc. (GMSI) found that the total digest silver analyses were on average 17% lower than the 4-acid silver analyses. Therefore, the resource estimate will use the more conservative method (total digest) for silver, which is of low economic importance anyway.

11.1.2 Quality Control

Highland implemented a QA/QC program for its 2017 and 2018 analytical sampling including core sampling duplicates, OREAS certified standards (CRM) of sedimentary deposits and coarse blanks collected and inserted according to the company sampling and assay quality procedures. In addition, the laboratory routinely inserts crushing stage duplicates, analytical stage pulp split duplicates and internal laboratory standards and blanks. The company and internal laboratory QA/QC samples included in the 2017 and 2018 drilling programs are outlined in Table 11.1.

Table 11.1: Overview of QA/QC Sampling

QA/QC Sample Type	No of Samples	Sampling %
Certified Coarse Blank	82	15.3
CRM - OREAS 162 (certified value = 0.761 wt.% Cu)	17	3.2
CRM - OREAS 97 (certified value = 6.31 wt.% Cu)	23	4.3
CRM - OREAS 930 (certified value = 2.52 wt.% Cu)	14	2.6
CRM Total	54	10.1
Sampling Stage Core Duplicate	26	4.8
Crushing Stage Duplicate	12	2.2
Laboratory Internal Standard - Cu ICP-OES (%)	178	33.1
Laboratory Internal Standard - Ag ICP-OES (g/t)	105	17.8
Laboratory Internal Blank - Cu ICP-OES (%) and Ag ICP-OES (g/t)	30	5.6
Laboratory Pulp Split Duplicate - Cu ICP-OES (%) and Ag ICP-OES (g/t)	50	9.3

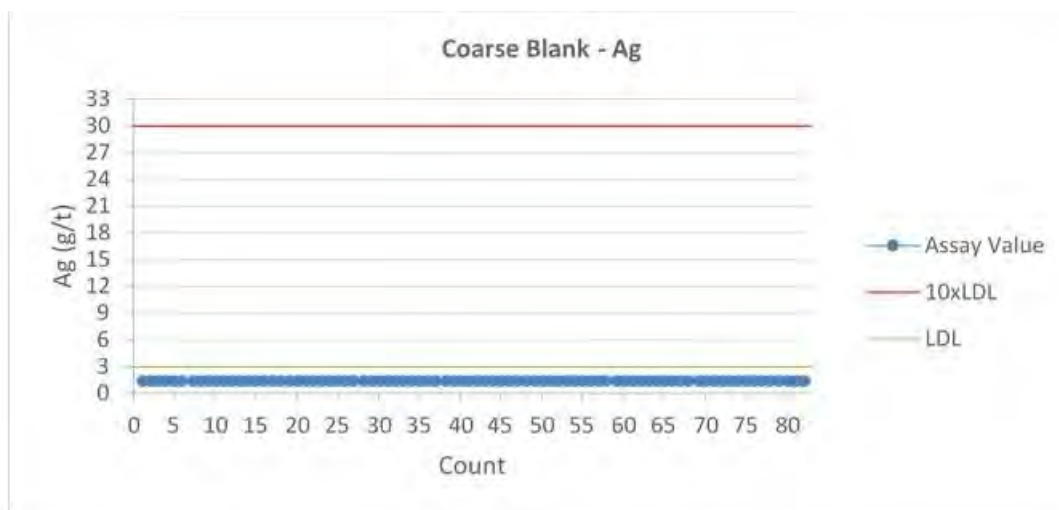
A geologist regularly inserted two standard CRM's, three coarse blanks, and one core duplicate for each drill hole. CRMs with a high Cu wt.%, medium Cu wt.%, and low Cu wt.% were inserted in a high grade, medium grade, and low-grade interval, respectively. Coarse blanks were inserted between high-grade intervals. A quarter core from the same assay interval was taken for a coarse duplicate.

11.1.3 Blanks and Assessment of Contamination

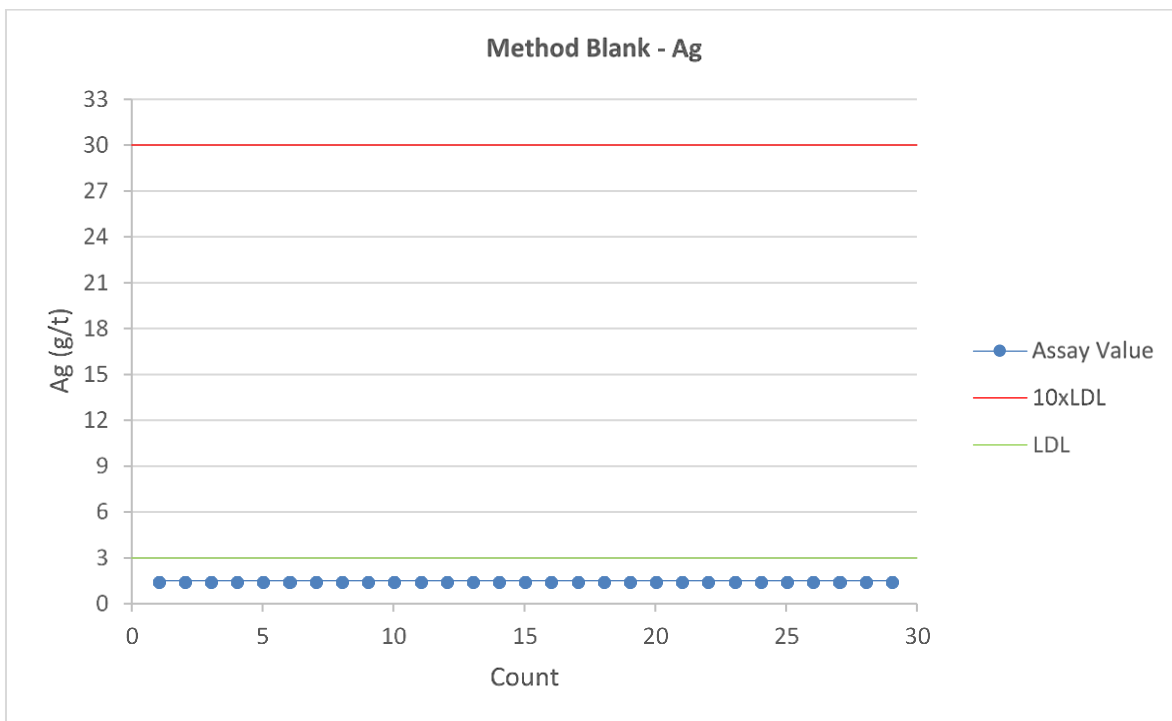
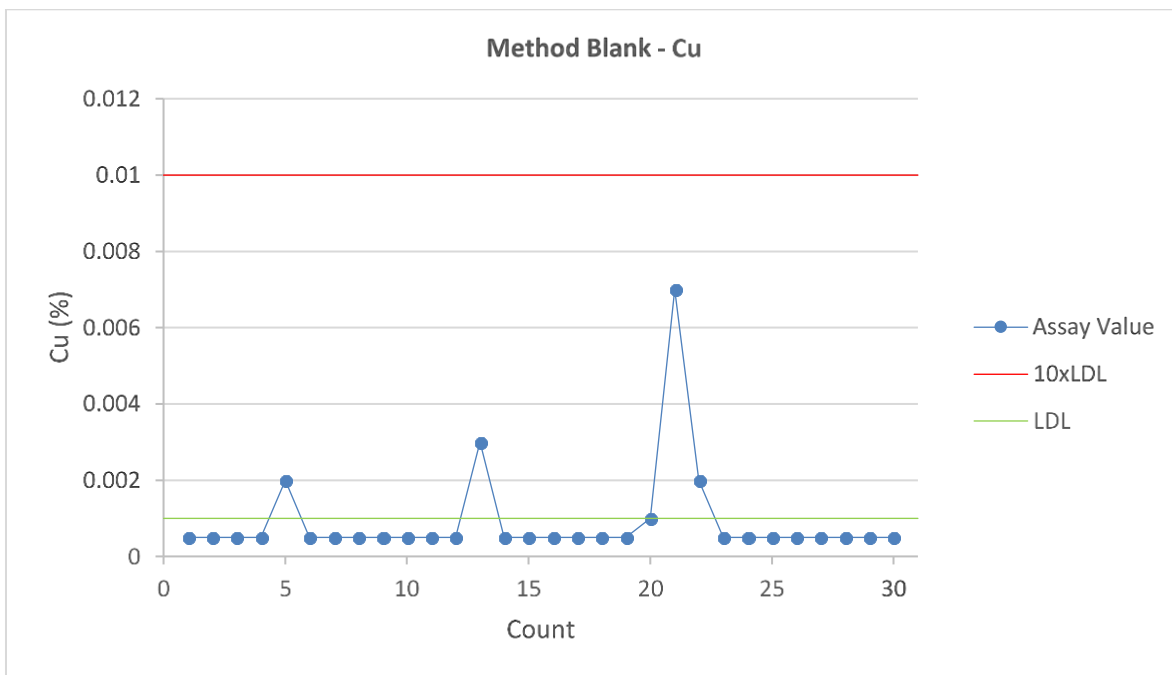
Highland inserted the certified coarse blank 1/2" mesh silica blank by ASL Analytical Solutions into the sample stream as part of the 2017 drilling program QA/QC at a 15.3% rate. A total of 82 coarse blanks were used during 2017-18 analytical assaying.

Less than 4% (3 samples) of the coarse blanks show greater values than 0.01% Cu (10 x lower detection limit). All three blanks fall after a previous sample with high grade Cu (>1% Cu). Two blanks failing the QA/QC and the surrounding primary samples were re-analysed. The results for both the failing blanks and the surrounding primary samples are very similar to original analysis. The original failed blank result is 0.027% Cu (Figure 11.1) and the reanalysis result is 0.029% Cu. 100% of the coarse blank silver assay values were under the detection limit 3 ppm Ag. With the exception of the one-time Cu contamination the coarse blanks show no contamination for copper and silver. Recent results (2018 assays) for coarse blanks included one analysis of greater than 0.01% Cu but was considered within acceptable limits.

Figure 11.1: Highland-inserted Blank Material Analytical Results (coarse CRM) for Cu (top) and Ag (bottom)



The internal laboratory blank "Method Blank" was inserted by Actlabs at a 5.6% rate. The internal laboratory blanks performance is good with all 30 blanks both for copper and 29 for silver ICP-OES having values less than 10 x lower detection limit.

Figure 11.2: Internal Laboratory Blank Material Analytical Results for Copper and Silver

11.1.4 Duplicate Sample Performance

The duplicate samples included in the 2017-18 drilling program consist of sampling stage core duplicates, crushing stage duplicates and analytical stage pulp split duplicates. The core duplicates were sampled and inserted by the geologists on site. The crushing stage duplicates were collected in the preparation laboratory after jaw crushing and the analytical stage duplicates are split in the analytical laboratory. Core duplicates were inserted at a 4.8% rate, crush duplicates at a 2.2% rate and split duplicates at a 9.3% rate.

The core duplicates performance is considered to be acceptable reflecting good overall precision and negligible sampling and analytical error (field and laboratory). Two copper core duplicates out of 26 core duplicates have a mean pair relative difference greater than 20% and possibly highlight variability characteristics of the ore deposit. Three silver core duplicates also have a mean pair relative difference greater than 20% and one of the silver duplicates coincident with one of the two deviating copper core duplicates. All the crush duplicate silver values for the primary sample or the check sample or both are under 10 x lower detection limit. For copper 6 core duplicates have values less than 10 x lower detection limit.

The crush duplicates performance is considered acceptable reflecting good overall laboratory precision and negligible preparation and analytical error. All 12 copper crush duplicates have a mean pair relative difference less than 10% while one silver crush duplicate is marginally over 20%. Again, all the crush duplicate silver values for the primary sample or the check sample or both are under 10 x lower detection limit. For copper crush duplicates all values are above 10 x lower detection limit.

The analytical pulp split duplicates performance is considered to be acceptable reflecting good analytical precision exclusive of dominant sampling errors. All 50 copper analytical pulp split duplicates have a mean pair relative difference less than 10% and two silver analytical pulp split duplicates are over 20%. Again, all the crush duplicate silver values for the primary sample or the check sample or both are under 10 x lower detection limit. For copper analytical pulp split duplicates, all except five have values above 10 x lower detection limit.

Duplicate performance graphs are shown in Figure 11.3 to Figure 11.5.

Figure 11.3: Core Duplicate Performance for Cu (top) and Ag (bottom)

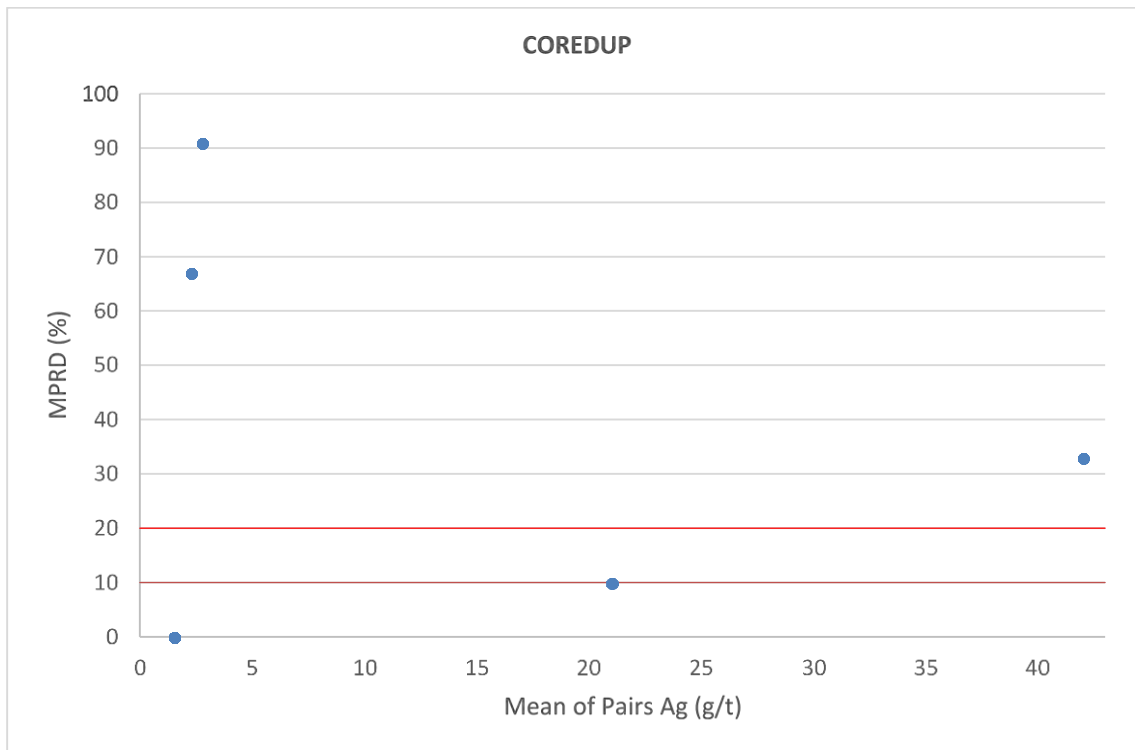
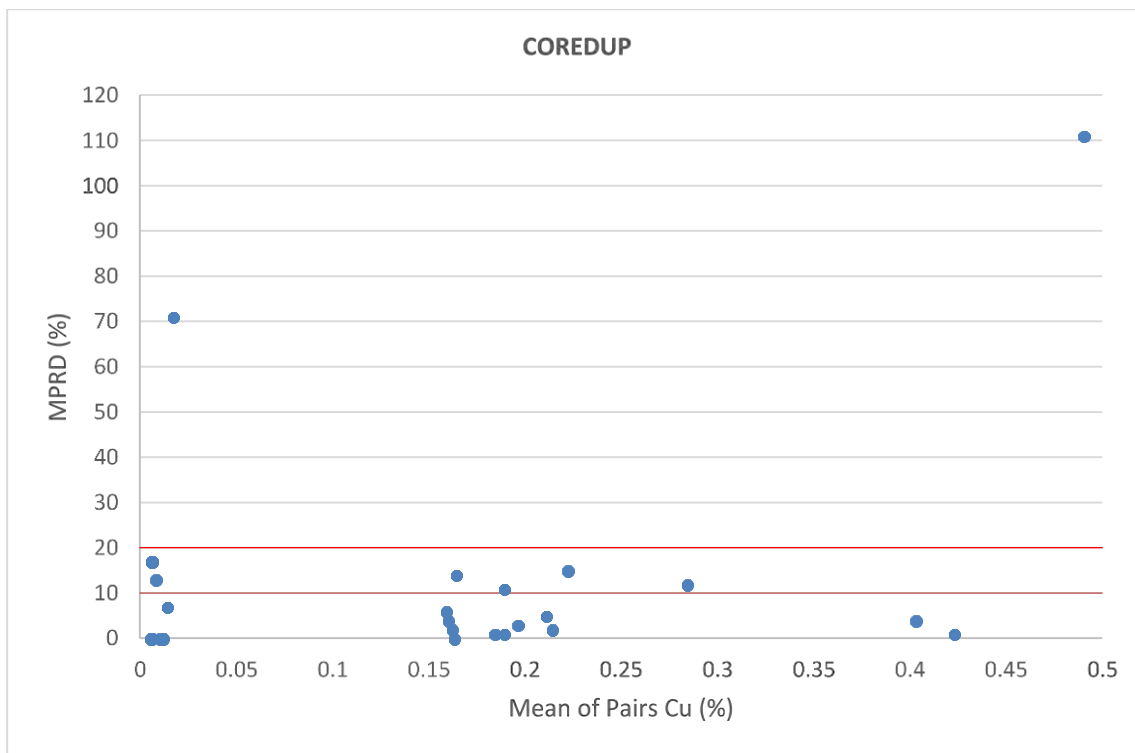


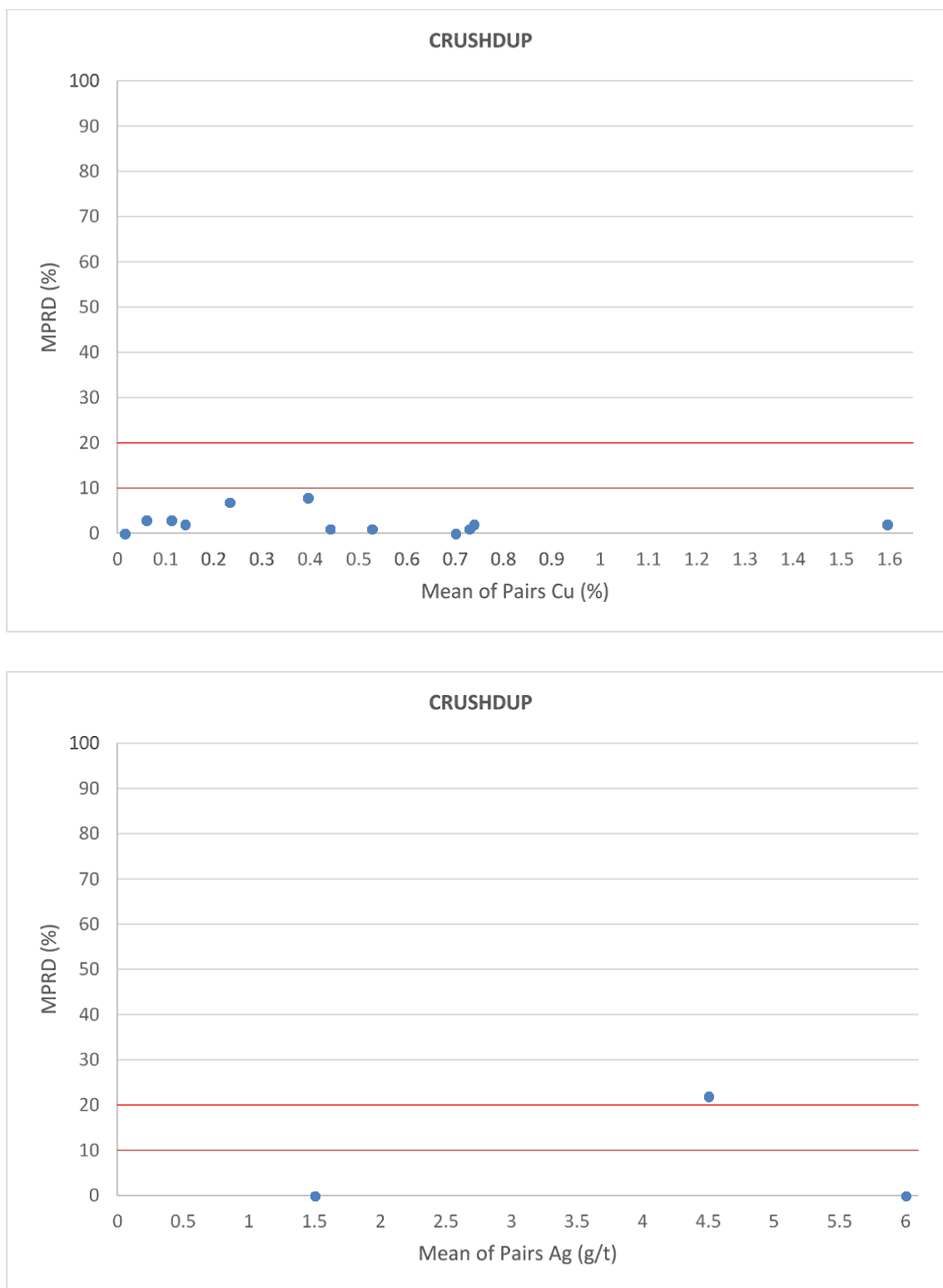
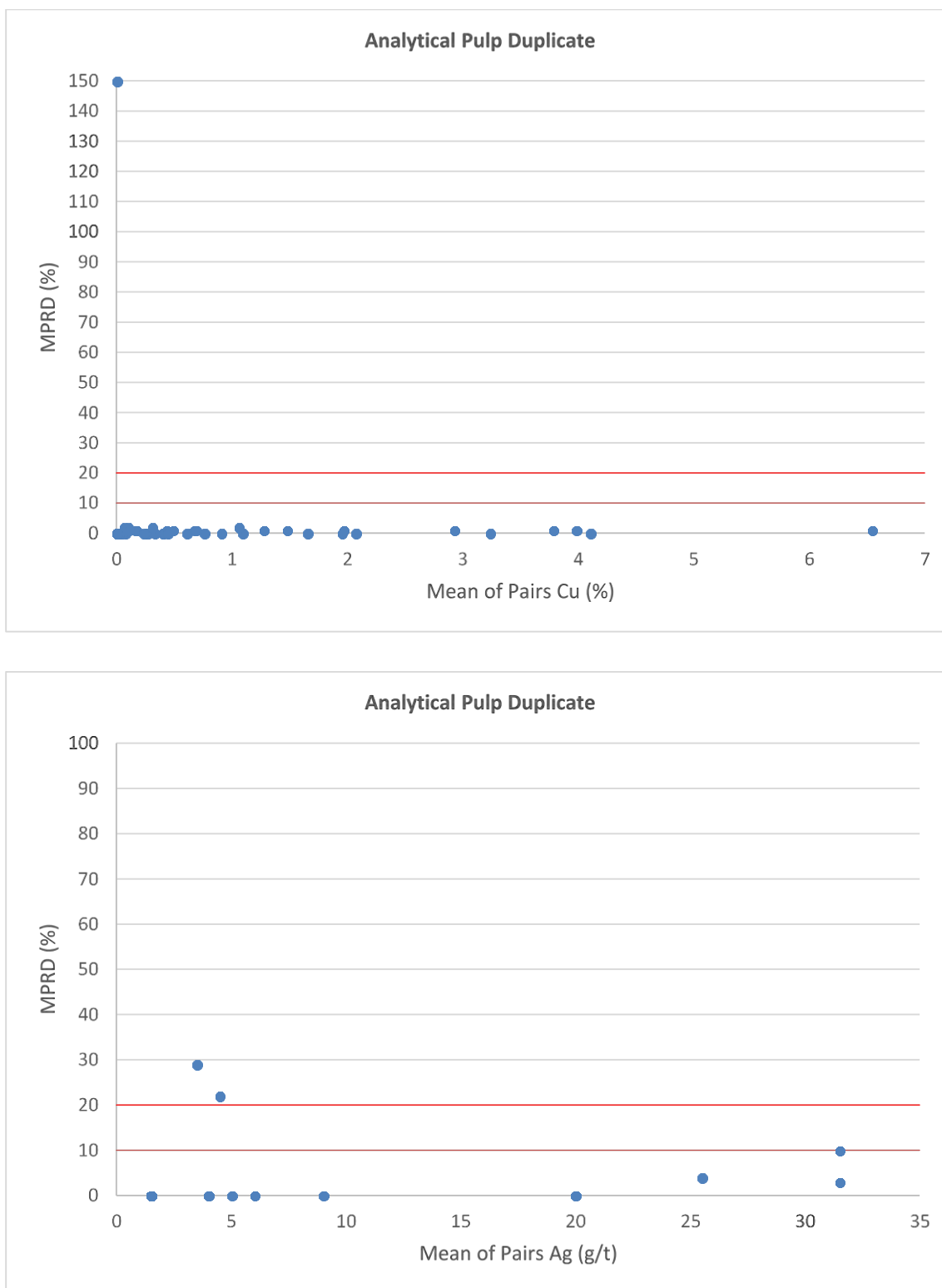
Figure 11.4: Crush Duplicate Performance for Cu (top) and Ag (bottom)

Figure 11.5: Analytical Pulp Performance for Cu (top) and Ag (bottom)

11.2 Performance of Standards

Throughout the analysis of 2017-18 drilling program standards were inserted at an 10.1% rate. A total of 54 standards were used during the 2017-18 analytical assaying. Three different standards OREAS 162, OREAS 97 and OREAS 930 were used with principle certified values of 0.772% Cu, 6.31% Cu, 2.52% Cu and 3.5 g Ag/t, 19.6 g Ag/t and 9 g Ag/t respectively. The standards are from Ore Research and Exploration Pty Ltd. (OREAS), an independent provider of commercial analytical standards from Australia.

The overall standard performance is acceptable. Five standards out of 54 have analytical values greater than ± 2 standard deviations from the certified value for copper and two of these have an analytical value greater than ± 2 standard deviations from the certified value for silver. Three of the copper standards fail only marginally with analytical values of 0.718, 0.714 and 0.711% Cu. The lower acceptance limit for the standard is 0.720% Cu and the standards were considered to pass the QA/QC test.

The five standards with analytical values greater than ± 2 standard deviations from certified values along with the surrounding primary samples were re-analysed. The standard consisting of the certified reference material OREAS 162 fails for copper while the standard consisting of the certified reference material OREAS 97 fails for both copper and silver. Again, the original and reanalysis results both for the failing standards and the surrounding primary samples are very similar and the original analysis was accepted. The original analytical value for the standard OREAS 162 is 0.695% Cu and the reanalysis result is 0.729% Cu. The original analytical value for the standard OREAS 97 is 3.98% Cu and 14 g Ag/t and the reanalysis result is 3.97% Cu and 13 g Ag/t respectively.

Standard performance graphs are shown in Figure 11.6 to Figure 11.12.

Figure 11.6: Performance of Control Reference Material OREAS 162 for Cu (top) and Ag (bottom)

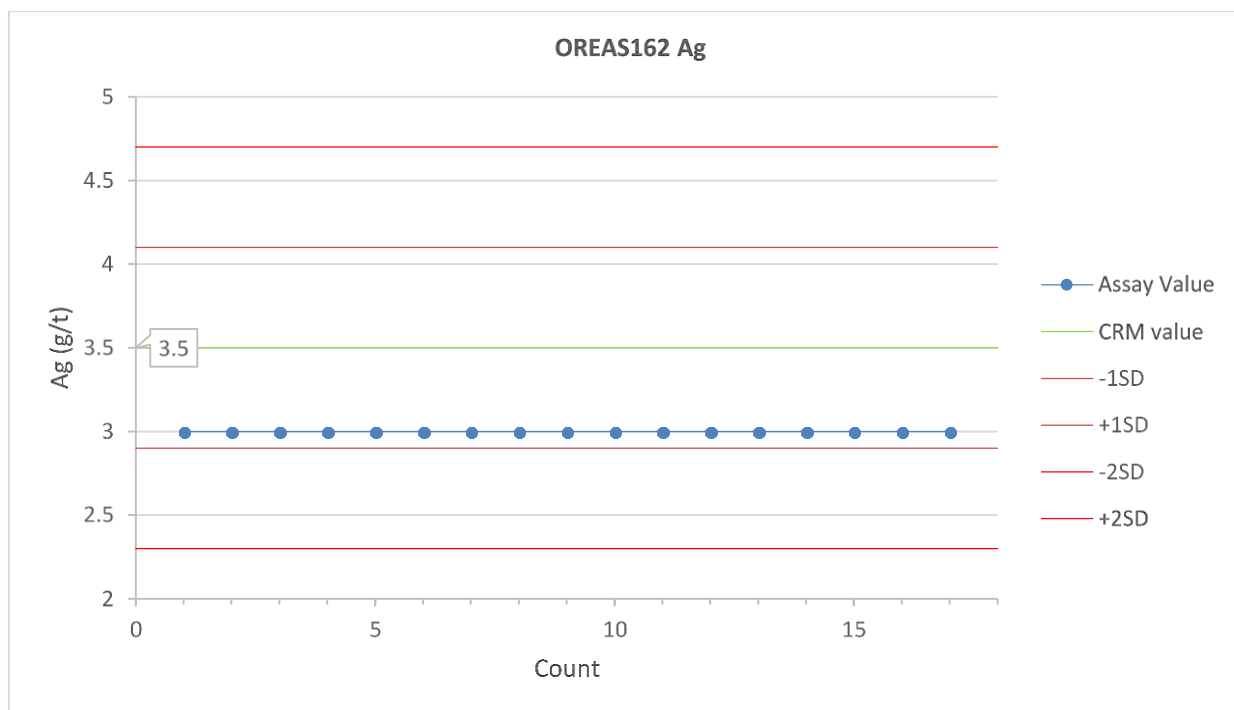
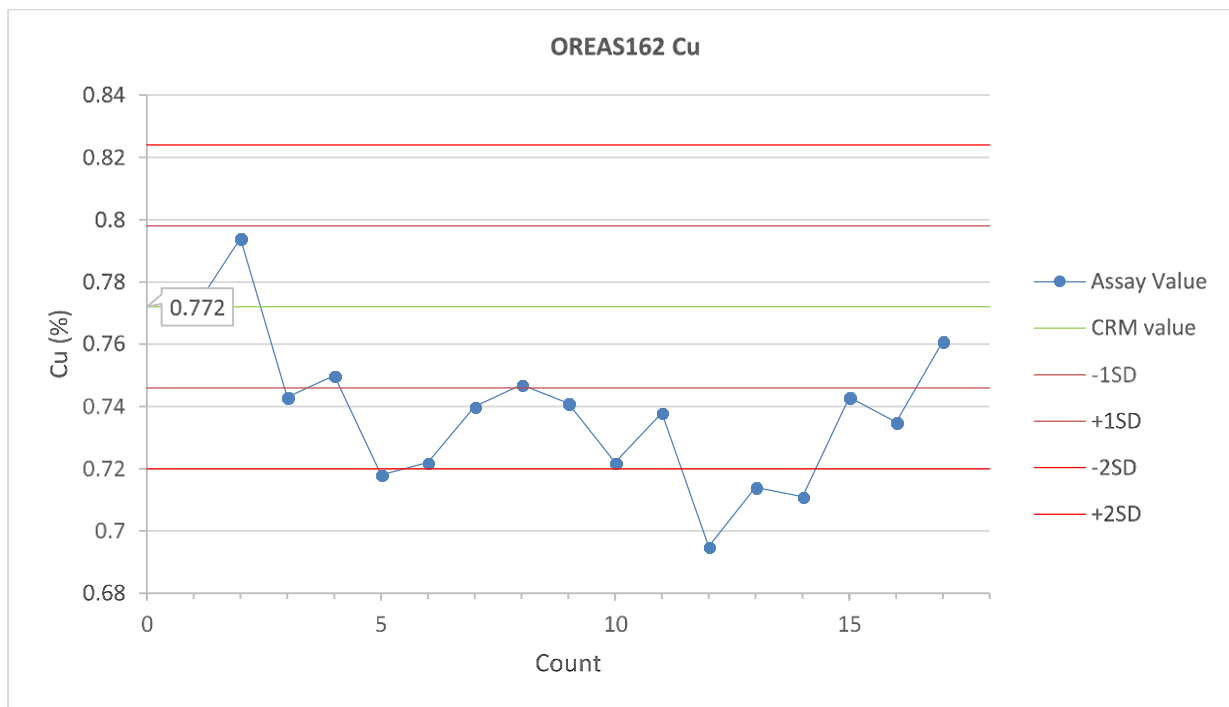


Figure 11.7: Performance of Control Reference Material OREAS 97 for Cu (top) and Ag (bottom)

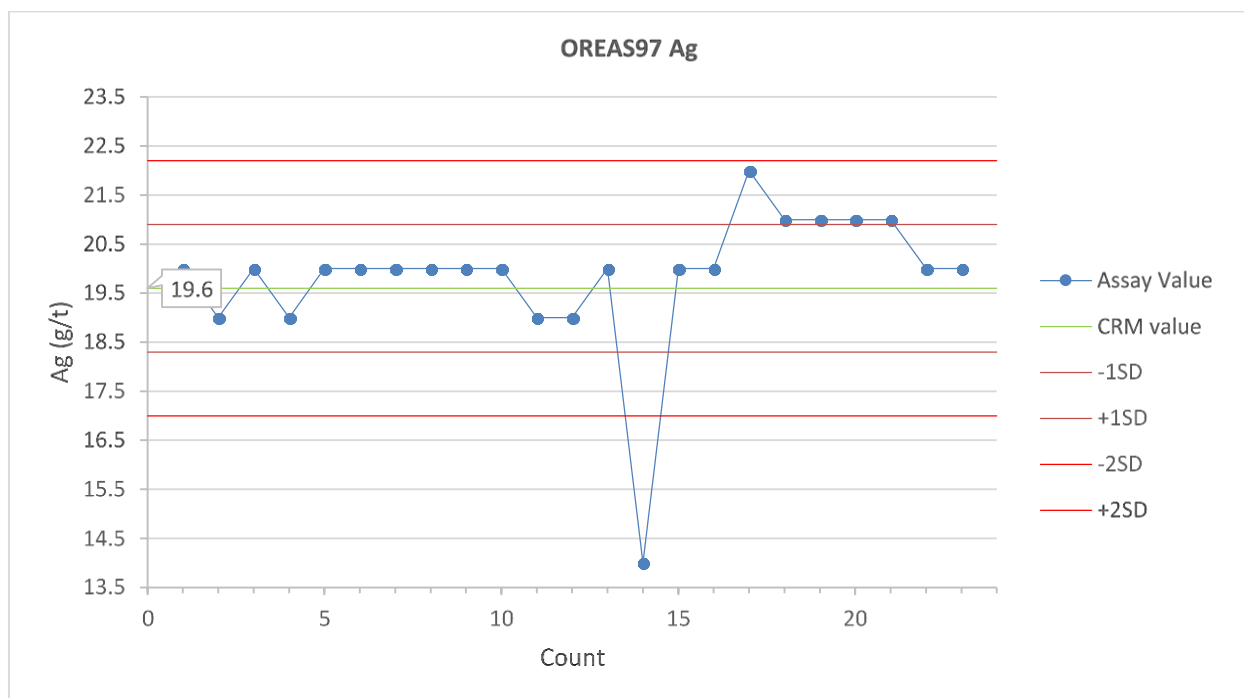
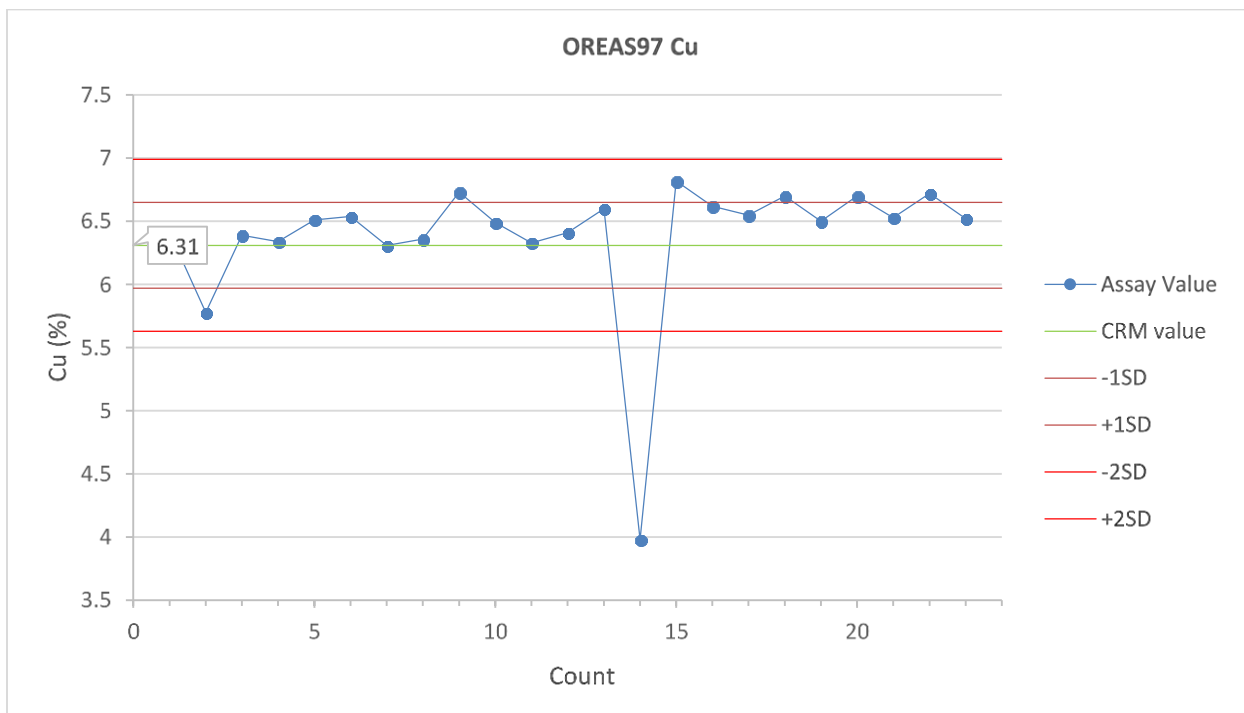
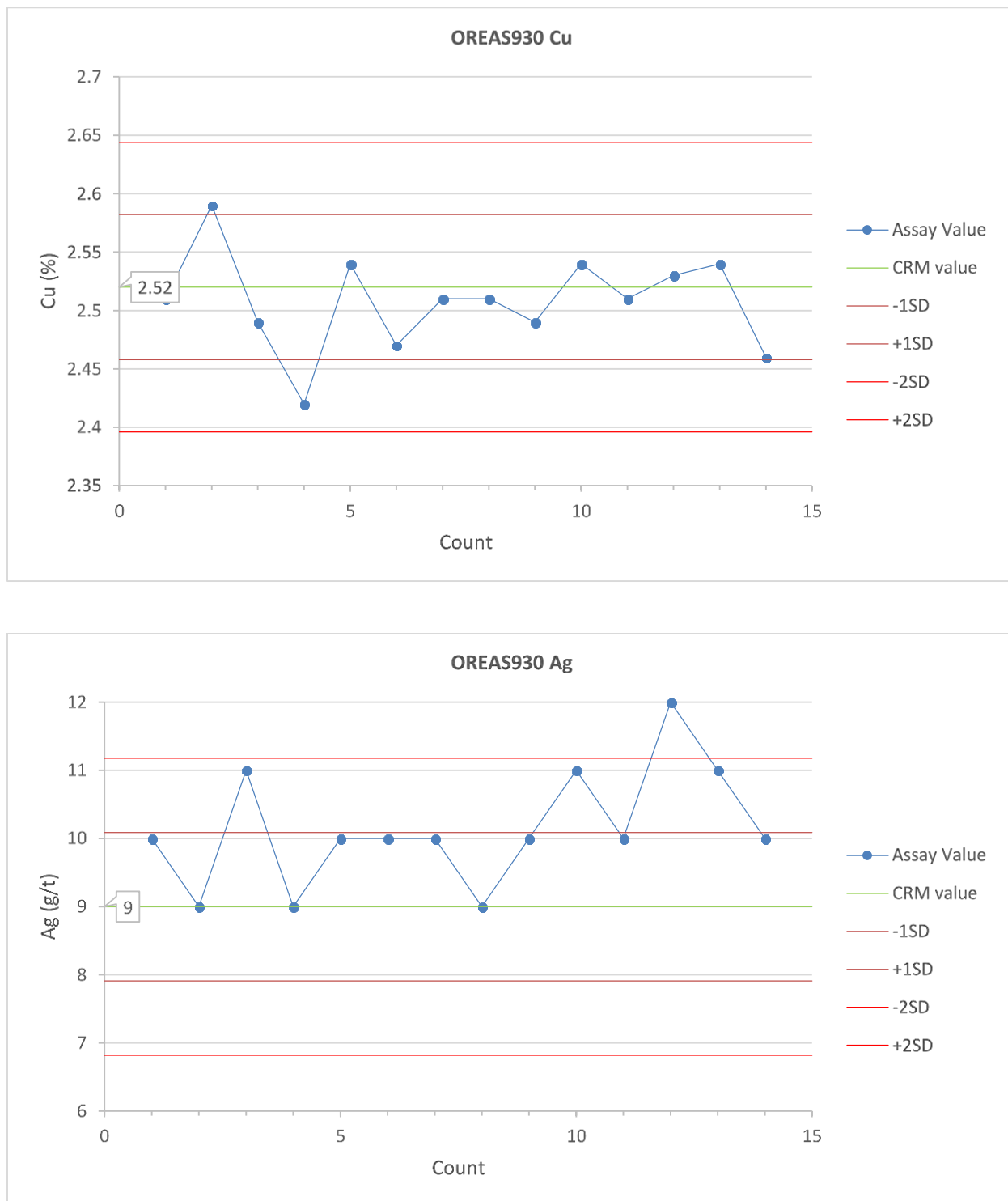


Figure 11.8: Performance of Control Reference Material OREAS 930 for Cu (top) and Ag (bottom)



Four different internal laboratory standards were inserted by the Actlabs at a 33.1% rate for Cu ICP-OES and at a 19.6% rate for Ag ICP-OES. The certified standards include CCU-1d, CZN-4 and MP-1b from Natural Resources Canada and OREAS 14P from Ore Research and Exploration Pty Ltd. All four standards were analysed for copper and three of the standards excluding OREAS 14P were analysed for silver. The certified expected values for the standards are: CCU-1d 23.93% Cu and 120.7 g Ag/t, CZN-4 0.403% Cu and 51.4 g Ag/t, MP-1b 3.069% Cu and 47 g Ag/t, OREAS 14P 0.997% Cu.

The internal laboratory standards performance is good, all the copper standard except five having values within ± 2 standard deviations from the certified value. Initially, two copper standards failed significantly for the standard CZN-4 and the laboratory was questioned. The laboratory stated a reporting error and a new certificate was issued excluding the two failing standards. The silver internal laboratory standards are within ± 2 standard deviations from the certified value with the exception of four standards. The four silver standards are, however, within the laboratory's own acceptance limits.

Figure 11.9: Performance of Control Reference Material CCU-1D for Cu (top) and Ag (bottom)

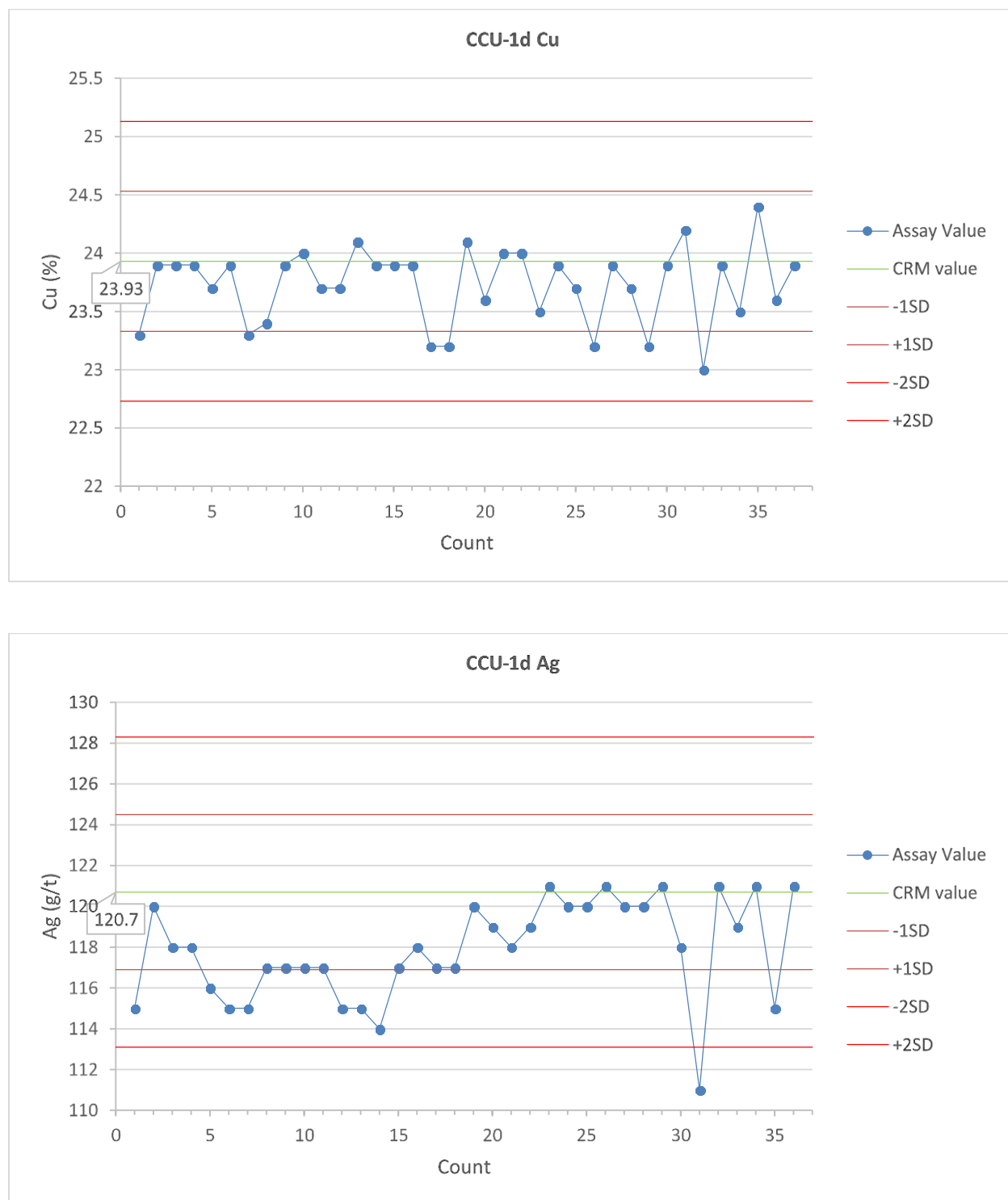


Figure 11.10: Performance of Control Reference Material CZN-4 for Cu (top) and Ag (bottom)

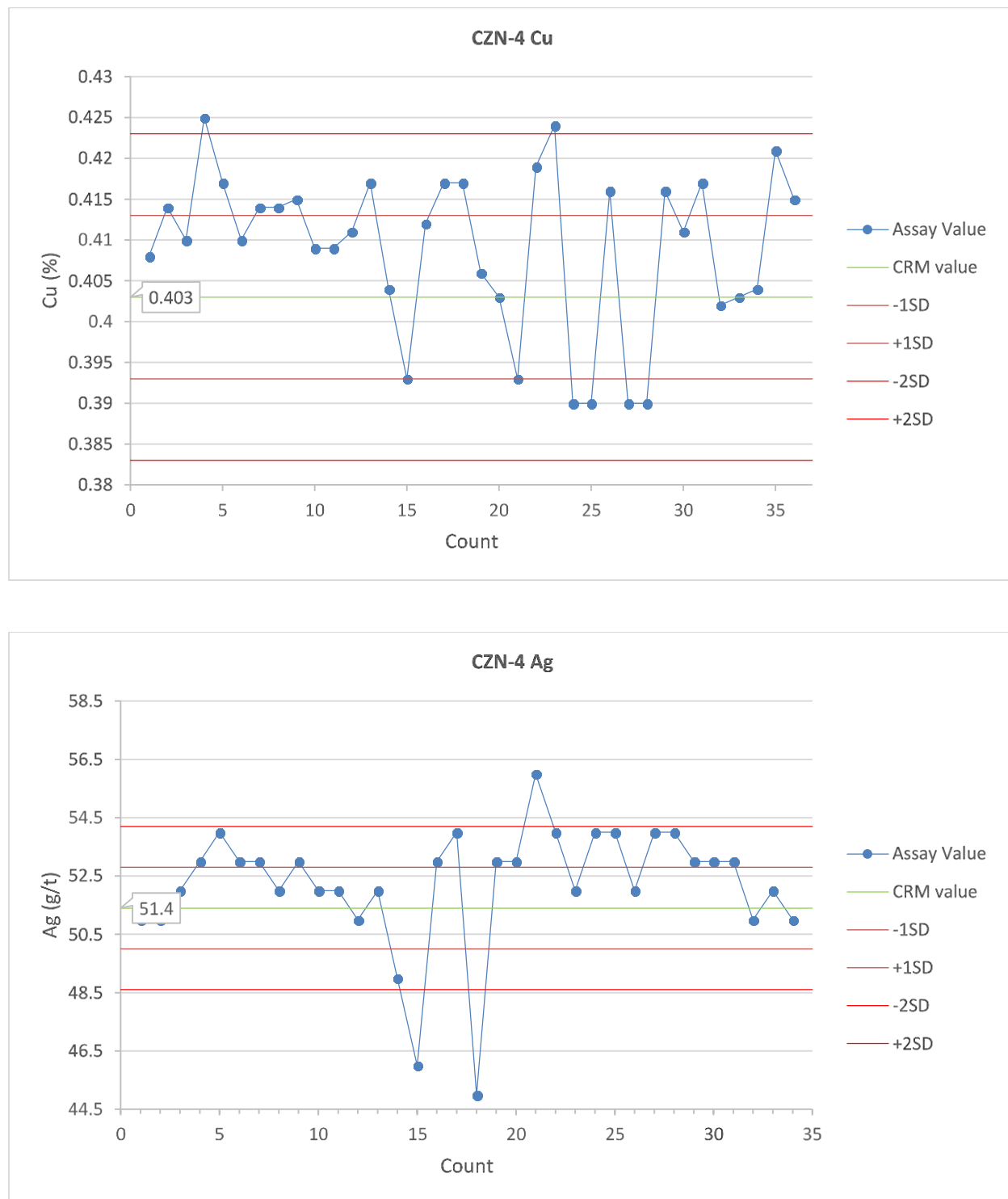


Figure 11.11: Performance of Control Reference Material MP-1b for Cu (top) and Ag (bottom)

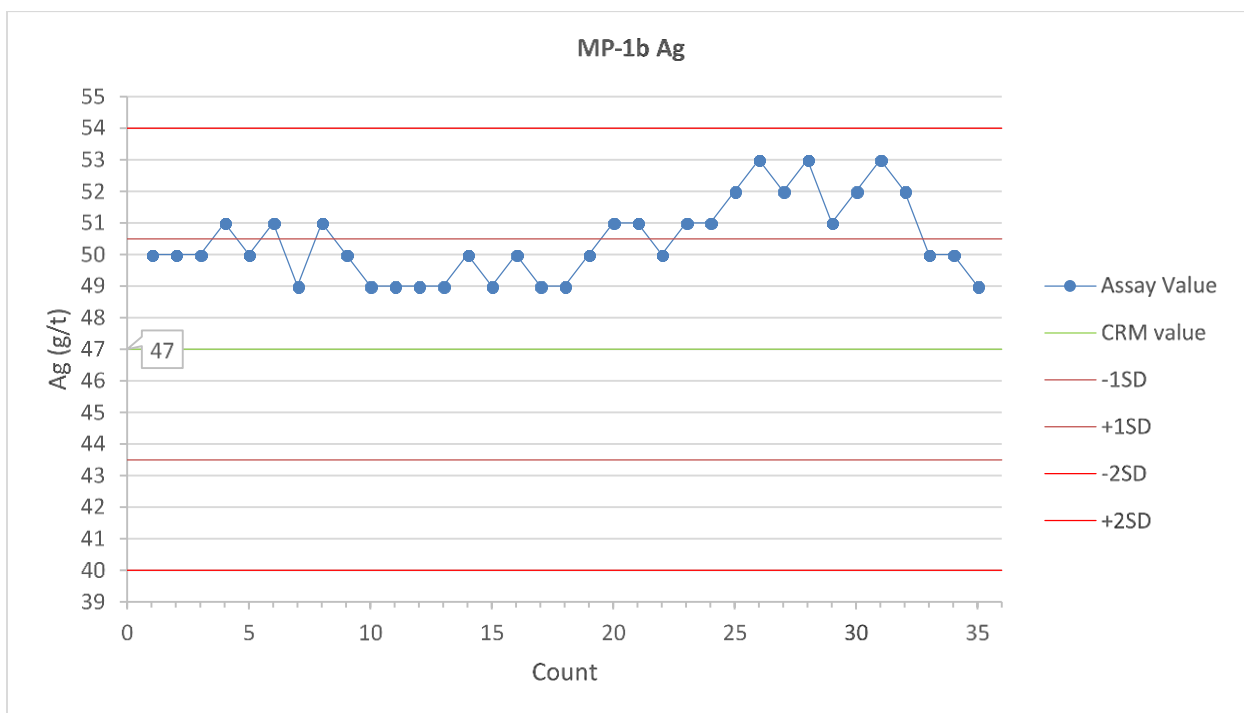
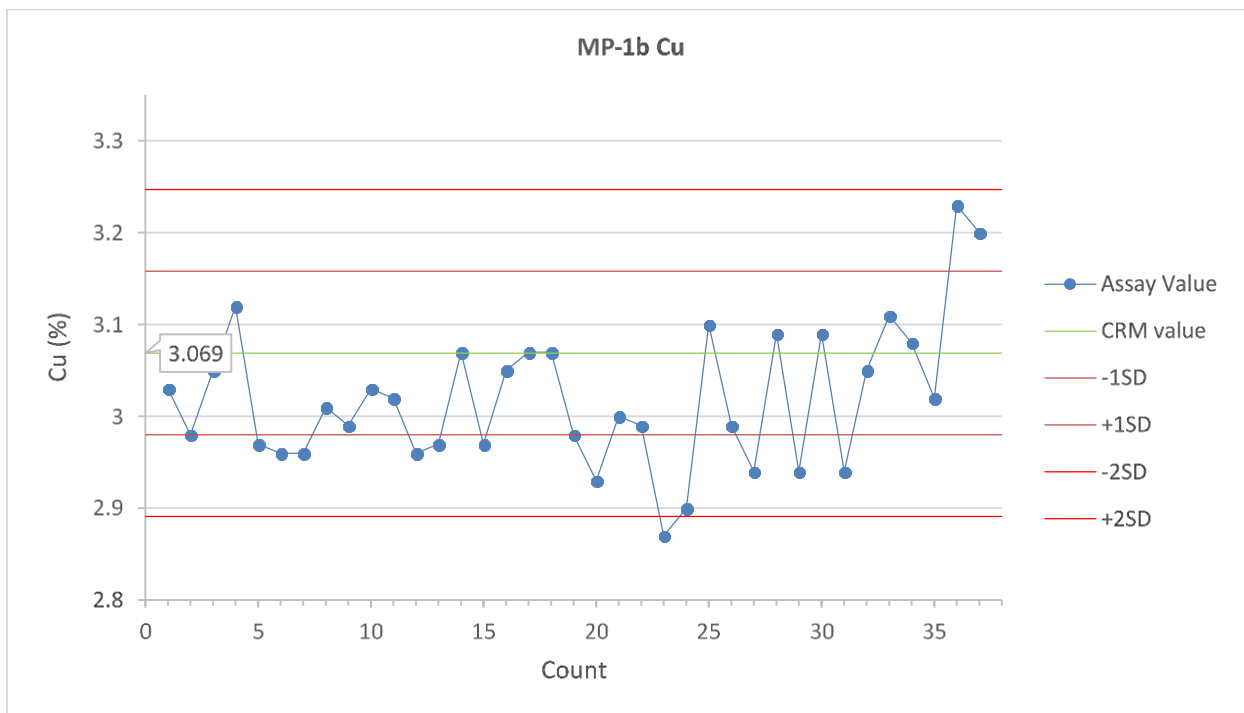
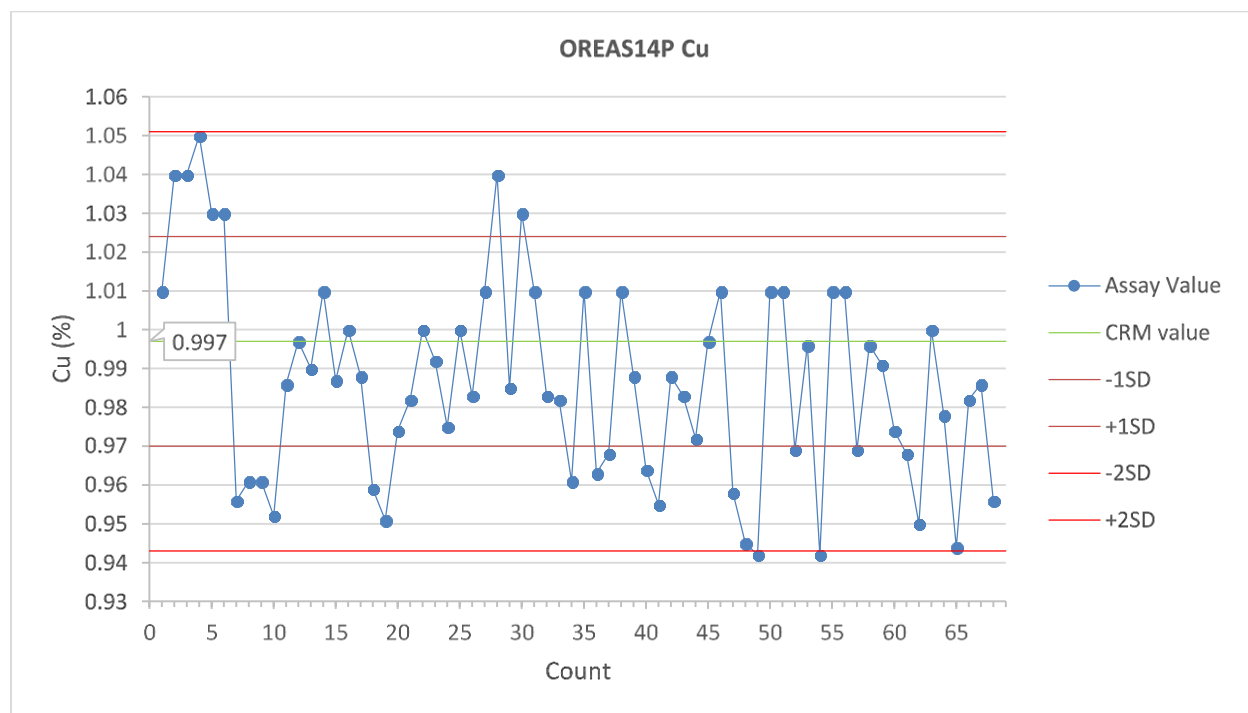


Figure 11.12: Performance of Control Reference Material 14P for Cu

Density Procedures

In-house bulk density was determined per lithologic unit by measuring specific gravity by the water immersion method on whole core. Quarter core was sent to Actlabs for bulk density determination using the wax immersion method following the American Society for Testing and Materials (ASTM) Designation C914-09. In-house samples were dried in a drying oven at 110° C for 12 to 24 hours and measured on Ohaus Scout Pro SP6001 scale with a 0.1 gram precision. The scale was checked so that it was completely level and calibrated with a 5 kg and 1 kg weight before measurements were taken. The specific gravity of the drill core had to be multiplied by the density of water to yield density. The water temperature was recorded for each measurement and a water temperature/density correction was programmed for each sample. Each measured mass was at least four significant digits and the final bulk density was reported to 0.01 gm/cc.

11.3 Security

Highland maintained sample chain of custody protocols on every step of sample handling, from the drilling site to the delivery of assay results to the independent database manager, who did a direct database handout to the qualified person doing the resource estimate.

11.4 Conclusions

The quality control and quality assurance procedures meet or exceed industry standards for the 2017-18 drilling program. The performance of inserted blanks and standards indicate that the sample preparation and the lab accuracy have been of good quality. Sample duplicate results were reasonable for copper values indicating a reasonable level of accuracy and precision from the contracted laboratory.

In the 2015 NI 43-101 report on the Copperwood Deposit, GMSI concluded that the QA/QC and security protocols established by Orvana and the quality of the results support resource and future reserve estimation. For further details on historical sampling practices, refer to the NI 43-101 report released by GMSI in 2015.

12. DATA VERIFICATION

12.1 Database

Drill hole information for the 2018 drilling program at the Copperwood Project was provided to GMSI by gDat Solutions, the independent database manager in the form of Microsoft Excel spreadsheets in CSV format. Data was provided as a single tranche on April 12, 2018. GMSI imported the files into the original MS Access database used in the October 2017 resource estimate, using the Geovia® GEMS software. The following drill hole information was imported in the GEMS database:

- Collar information: Hole ID, X, Y and Z coordinates of collar (UTM), length;
- Down-hole survey: Hole ID, downhole depth, dip, azimuth;
- Assay: Hole ID, depth from and to, Cu values in %, Ag values in ppm;
- Geology: Hole ID, depth from and to, lithology unit.

A total of 314 diamond drill holes with assay information were available for grade estimation, and a further 72 drill holes contained lithology information which was used to build the geological model). The database was reviewed and corrected if necessary prior to final formatting for resource evaluation. The following activities were performed during database validation:

- Validate total hole lengths and final sample depth data;
- Verify for overlapping and missing intervals;
- Check drill hole survey data for out of range or suspect downhole deviations;
- Visual check of spatial distribution of drill holes and trenches;
- Validate lithology codes.

Table 12.1: Drill Holes Available in the Database for Resource Estimation

BC-10-113	CW-09-52	CW-09-94	CW-13-148	CW-17-180	M56-W13	M57-W120	M57-W32	M57-W74	PC-3
BC-10-117	CW-09-53	CW-09-95	CW-13-149	CW-17-180A	M56-W14	M57-W121	M57-W33	M57-W75	PC-4
BC-10-118	CW-09-54	CW-09-96	CW-13-150	CW-17-181	M56-W16	M57-W123	M57-W34	M57-W76	PC-5
CW-08-09	CW-09-55	CW-09-97	CW-13-151	CW-17-181A	M56-W17	M57-W124	M57-W35	M57-W77	PC-6
CW-08-11	CW-09-56	CW-09-98	CW-13-152	CW-17-182	M56-W18	M57-W125	M57-W36	M57-W78	PC-7
CW-08-13	CW-09-57	CW-09-99	CW-13-153	CW-17-183	M56-W19	M57-W126	M57-W37	M57-W79	PC-8
CW-08-16	CW-09-58	CW-10-103	CW-13-154	CW-17-184	M56-W2	M57-W127	M57-W38	M57-W80	PC-9
CW-08-17	CW-09-59	CW-10-104	CW-13-155	CW-17-185	M56-W20	M57-W128	M57-W39	M57-W81	
CW-08-20	CW-09-60	CW-10-105	CW-13-156	CW-17-186	M56-W21	M57-W130	M57-W40	M57-W82	
CW-09-100	CW-09-61	CW-10-106	CW-13-157	CW-17-187	M56-W22	M57-W131	M57-W41	M57-W83	
CW-09-101	CW-09-62	CW-10-107	CW-13-158A	CW-17-188	M56-W23	M57-W132	M57-W42	M57-W84	
CW-09-102	CW-09-63	CW-10-108	CW-13-159	CW-17-189	M56-W24	M57-W133	M57-W43	M57-W85	
CW-09-21	CW-09-64	CW-10-109	CW-13-160	CW-17-189A	M56-W25	M57-W134	M57-W44	M57-W86	
CW-09-22	CW-09-65	CW-10-110	CW-13-161	CW-17-190	M56-W26	M57-W135	M57-W45	M57-W87	
CW-09-23	CW-09-66	CW-10-111	CW-13-BC-01	CW-17-190A	M56-W28	M57-W136	M57-W46	M57-W88	
CW-09-24	CW-09-67	CW-10-112	CW-13-BC-02	CW-17-191	M56-W2A	M57-W137	M57-W47	M57-W89	
CW-09-25	CW-09-68	CW-10-114	CW-13-BC-03	CW-17-191A	M56-W3	M57-W138	M57-W48	M57-W90	
CW-09-26	CW-09-69	CW-10-115	CW-13-BC-04	CW-17-192	M56-W4A	M57-W139	M57-W49	M57-W91	
CW-09-27	CW-09-70	CW-10-116	CW-17-162	CW-17-192A	M56-W5	M57-W140	M57-W50	M57-W92	
CW-09-28	CW-09-71	CW-10-119	CW-17-163	CW-17-193	M56-W6	M57-W141	M57-W51	M57-W93	
CW-09-29	CW-09-72	CW-10-121	CW-17-164	CW-17-194	M56-W7	M57-W142	M57-W52	M57-W94	
CW-09-30	CW-09-73	CW-10-122	CW-17-165	CW-17-194A	M56-W8	M57-W143	M57-W53	M57-W95	
CW-09-31	CW-09-74	CW-10-123	CW-17-165A	CW-17-195	M57-W100	M57-W144	M57-W54	M57-W96	
CW-09-32	CW-09-75	CW-10-125	CW-17-166	CW-17-196	M57-W101	M57-W145	M57-W55	M57-W97	
CW-09-33	CW-09-76	CW-10-126	CW-17-167	CW-17-197	M57-W102	M57-W146	M57-W56	M57-W98	
CW-09-34	CW-09-77	CW-10-127	CW-17-167A	CW-17-198	M57-W103	M57-W147	M57-W57	M57-W99	
CW-09-35A	CW-09-78	CW-10-128	CW-17-168	CW-17-199	M57-W104	M57-W148	M57-W58	PC-1	
CW-09-36	CW-09-79	CW-10-129	CW-17-169	CW-17-200	M57-W105	M57-W149	M57-W59	PC-10	
CW-09-37	CW-09-80	CW-10-130	CW-17-170	CW-17-201	M57-W106	M57-W150	M57-W60	PC-11	
CW-09-38	CW-09-81	CW-10-131	CW-17-171	CW-18-202	M57-W107	M57-W151	M57-W61	PC-12	
CW-09-39	CW-09-82	CW-10-132	CW-17-171A	CW-18-203	M57-W108	M57-W152	M57-W62	PC-13	
CW-09-41	CW-09-83	CW-10-133	CW-17-172	CW-18-204	M57-W109	M57-W153	M57-W63	PC-14	
CW-09-42	CW-09-84	CW-10-136	CW-17-172A	CW-18-205	M57-W110	M57-W154	M57-W64	PC-15	
CW-09-43	CW-09-85	CW-10-137	CW-17-173	CW-18-206	M57-W111	M57-W155	M57-W65	PC-16	
CW-09-44	CW-09-86	CW-10-138	CW-17-174	CW-18-207	M57-W112	M57-W156	M57-W66	PC-17	
CW-09-45	CW-09-87	CW-10-139	CW-17-175	CW-18-208	M57-W113	M57-W157	M57-W67	PC-18	
CW-09-46	CW-09-88	CW-11-140	CW-17-176	CW-18-209	M57-W114	M57-W158	M57-W68	PC-19	
CW-09-47	CW-09-89	CW-11-141	CW-17-177	M56-W09	M57-W115	M57-W159	M57-W69	PC-2	
CW-09-48	CW-09-90	CW-11-142	CW-17-178	M56-W1	M57-W116	M57-W27	M57-W70	PC-20	
CW-09-49	CW-09-91	CW-11-143	CW-17-179	M56-W10	M57-W117	M57-W29	M57-W71	PC-21	
CW-09-50	CW-09-92	CW-13-146	CW-17-179A	M56-W11	M57-W118	M57-W30	M57-W72	PC-22	
CW-09-51	CW-09-93	CW-13-147	CW-17-179B	M56-W12A	M57-W119	M57-W31	M57-W73	PC-23	

12.2 GMSI Data Verification

Most of the content in this section is sourced from the NI 43-101 technical report prepared by GMSI on the Copperwood Project in June 2015, which outlines the data verification procedures undertaken on historical data. Regarding the data collected in 2017, drill hole locations were visited, and drill core was viewed during the site visit between the 6th and 9th November 2017. Drill hole data received in 2018 (14 drill holes) was not verified during a site visit. However, 50% of the assay certificates were checked against the database export to ensure that the drilling database is truthful and representative.

GMSI performed data verification checks of the drill logs, assay certificates, downhole surveys, and additional information sources on site at Highland's office located in White Pine, Michigan, in April 2015.

The following validation checks were made for the copper and silver assays in 2015:

- Approximately 50% of the assay database (2,671 assays) was checked against the original laboratory certificates for possible typographical errors, wrong sample numbers or duplicates. Minor errors were found in less than 0.5% of the database investigated and were corrected accordingly;
- Five random laboratory certificates were also directly sent to GMSI from Actlabs to compare with Highland's certificates. No error was found;
- GMSI has high confidence in the assay database.

The following validation checks were made for the lithology information in 2015:

- Approximately 20% of the drill holes were randomly selected to compare the database with the original paper logs. Some 76 drill holes were selected this way with good overall representation of the Copperwood Project (Table 12.2);
- Lithological information of beds and From / To intervals was validated;
- No error was found; GMSI has high confidence in the lithological information.

These other validation checks were made:

- Validation of the downhole survey of 40 drill holes randomly selected. Comparison between the original survey files and the survey database showed only minor errors, for less than 1% of the database;

- Validation of the drill hole collar survey: check of the survey certificate from U.P. Engineers & Architects, Inc. The certificates details on the conversion process from Copper Range Company local coordinates system (in feet) to UTM Zone 16T (in meters) and on the surveyed drill holes;
- Validation of QA/QC, density, metallurgical and logging procedures with Highland's professional staff. All information pertaining to the aforementioned procedures are rigorously recorded in procedure manuals easily accessible to Highland's personnel.

Table 12.2: Drill Holes Randomly Selected from the Database for Lithology Validation

CW-09-101	CW-09-62	CW-10-105	M56-W19	M57-W117	M57-W151	M57-W65	PC-19
CW-09-24	CW-09-63	CW-10-108	M56-W2	M57-W120	M57-W153	M57-W66	PC-21
CW-09-25	CW-09-71	CW-10-110	M56-W20	M57-W124	M57-W155	M57-W74	PC-23
CW-09-37	CW-09-77	CW-10-121	M56-W25	M57-W126	M57-W158	M57-W82	PC-3
CW-09-41	CW-09-81	CW-10-138	M56-W26	M57-W128	M57-W159	M57-W87	PC-5
CW-09-46	CW-09-82	CW-13-148	M56-W6	M57-W130	M57-W27	M57-W89	PC-7
CW-09-49	CW-09-85	CW-13-149	M57-W100	M57-W131	M57-W36	M57-W93	
CW-09-53	CW-09-89	CW-13-151	M57-W107	M57-W133	M57-W43	M57-W96	
CW-09-54	CW-09-92	CW-13-BC-04	M57-W113	M57-W135	M57-W49	PC-1	
CW-09-60	CW-09-95	M56-W12A	M57-W116	M57-W150	M57-W54	PC-12	

12.3 Drill Hole Collar Location

GMSI personnel visited numerous drill collars from the 2017 drilling campaign during the site visit between the 6th and 9th of November 2017. Drill collars were randomly chosen.

In Section 6, drill collars were identified by a concrete base with the name of the drill hole engraved onto it. Due to stringent rehabilitation requirements on Section 5, drill collars were characterized by a single stake with the name of the drill hole. All drill hole locations visited were easily identifiable. Examples are shown in Figure 12.1 and Figure 12.2.

Figure 12.1: Drill Hole Collar Example in Section 6 - CW17-195



Figure 12.2: Drill Hole Collar Example from Section 5 - CW-17-184

12.4 QA/QC Validation

GMSI reviewed the results of the QA/QC from the 2017 and 2018 drilling campaigns (as discussed in Section 11) and found them to be within acceptable limits.

12.5 Conclusions

Overall, GMSI is comfortable that the data, analyses, QA/QC and geological interpretation presented in the previous historical reports was performed in a professional manner using industry best practices. GMSI believes that all data is reliable for use in the statement of Mineral Resources presented in this Report.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Early Metallurgical Testing (before 2012)

Metallurgical testwork for the Copperwood Project has been completed by Kappes, Cassiday & Associates (“KCA”) located in Reno, Nevada, Mountain State Research and Development Inc. (“MSRDI”) of Vail, Arizona and METCON Research (“METCON”) located in Tucson, Arizona. Results from these testwork programs were presented and detailed in the Copperwood Project Feasibility Study (“FS 2012”) with a file date of March 21, 2012, by KD Engineering Company.

The main conclusion drawn from the previous work was that the composites were readily amenable to conventional sulphide flotation methods. The major process design criteria developed from these testwork programs are as follows:

- Main copper mineral is chalcocite which is finely disseminated;
- Overall copper recovery of 82 to 87% producing a concentrate of 23 to 26% Cu;
- Silver recovery varies from 50-55%;
- Primary grind size P80 of 63 microns;
- Regrind size P80 of 25 microns;
- No processing factors or deleterious elements identified to have negative impact on copper grade/recovery.

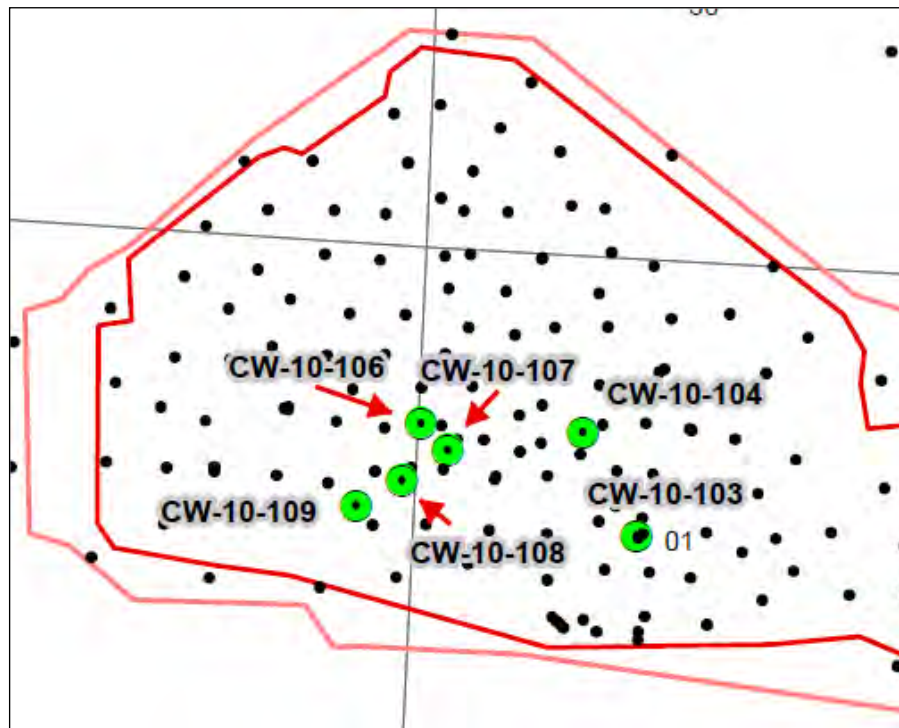
The FS 2012 was predominately developed based on METCON testwork, thus the METCON results are further discussed below. Table 13.1 shows the chemical analysis of the composites used. The composites and location of samples are shown in Table 13.2 and Figure 13.1, respectively.

Table 13.1: Composites No.4 and No.5 Assays

Sample ID	Assays (%)			Sequential Copper Analysis (%)			
	Cu (%)	Fe (%)	Ag (g/t)	As Cu	CNs Cu	Residual Cu	Calculated Cu
Composite No.4	1.40	5.7	4.0	0.146	1.25	0.022	1.42
Composite No.5	1.49	5.9	3.0	0.156	1.24	0.034	1.43

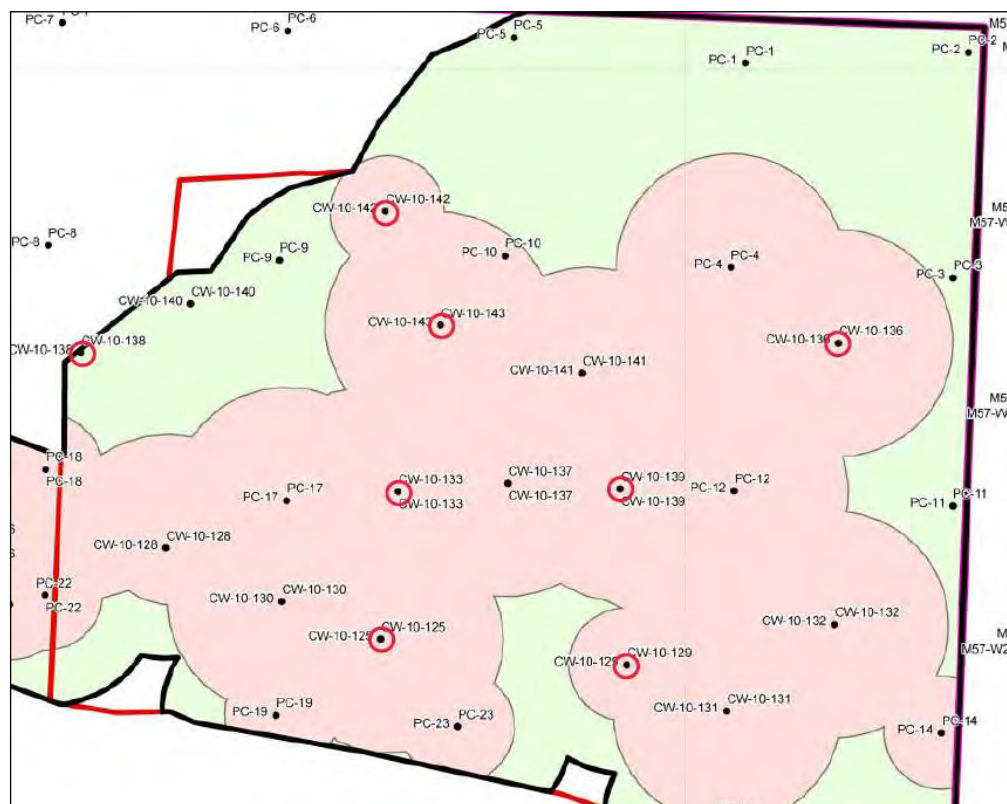
Table 13.2: Details of Composites No. 4 and Composite No.5

Sample ID (2011)	Sample	Zone
CBS4	CW-10-103	Main Zone
	CW-10-104	Main Zone
	CW-10-106	Main Zone
	CW-10-107	Main Zone
	CW-10-108	Main Zone
	CW-10-109	Main Zone
CBS5	CW-10-125	Zone 6
	CW-10-129	Zone 6
	CW-10-133	Zone 6
	CW-10-136	Zone 6
	CW-10-138	Zone 6
	CW-10-139	Zone 6
	CW-10-142	Zone 6
	CW-10-143	Zone 6

Figure 13.1: Main Zone Sample Location

The CBS4 samples were mainly provided from a specific area located in the center of the Eastern part of the deposits as presented in Figure 13.2.

The samples from Zone 6 were more distributed over the Copperwood deposit.

Figure 13.2: Zone 6 Sample Locations

The Locked Cycle Test (“LCT”) results of Composite No. 4 and No. 5 are illustrated in Table 13.3 and Table 13.4.

Table 13.3: LCT Composite No.4 Results

Table 2.2.1 Locked Cycle Flotation Testing On Composite No. 4 For Metallurgical Mass Balance Summary of Results								
Cycle	Grade (%)				Recovery (%)			
	Cu	Ag (g/t)	Fe	Insol.	Cu	Ag	Fe	Insol.
1	24.50	52	9.00	41.00	89.58	61.90	6.81	2.67
2	25.00	46	9.60	36.60	85.46	52.90	6.86	2.20
3	21.90	42	9.70	40.90	85.24	52.56	7.46	2.66
4	24.30	48	9.50	39.50	83.16	52.87	6.65	2.45
5	23.50	45	10.00	41.05	85.07	56.77	8.36	3.01
6	24.30	46	10.40	34.05	87.46	60.56	8.42	2.70
Average	24.03	47	9.97	38.20	85.23	56.26	7.81	2.72

Table 13.4: LCT Composite No.5 Results

Cycle	Mass Recovery (%)	Grade (%)					Recovery (%)				
		Cu	Ag (g/t)	Fe	S _T	Insol	Cu	Ag	Fe	S _T	Insol
A	4.50	31.30	74	9.19	9.58	33.92	84.71	55.05	6.49	70.76	2.14
B	4.59	28.10	69	9.31	8.51	40.18	81.04	56.31	7.48	71.10	2.58
C	4.86	30.20	68	9.04	9.16	35.34	82.94	54.33	7.38	71.97	2.41
D	5.84	24.00	58	8.90	7.31	41.44	83.99	57.05	9.35	72.51	3.43
E	4.95	29.50	75	9.52	8.93	36.88	82.11	59.10	8.24	70.71	2.59
F	4.80	29.80	72	9.50	9.00	35.92	81.70	52.42	8.12	61.45	2.43
G	5.33	25.50	65	9.01	7.89	39.76	80.31	55.85	8.10	61.57	3.00
Average	4.98	28.34	69	9.21	8.62	37.63	82.40	55.73	7.88	68.58	2.65

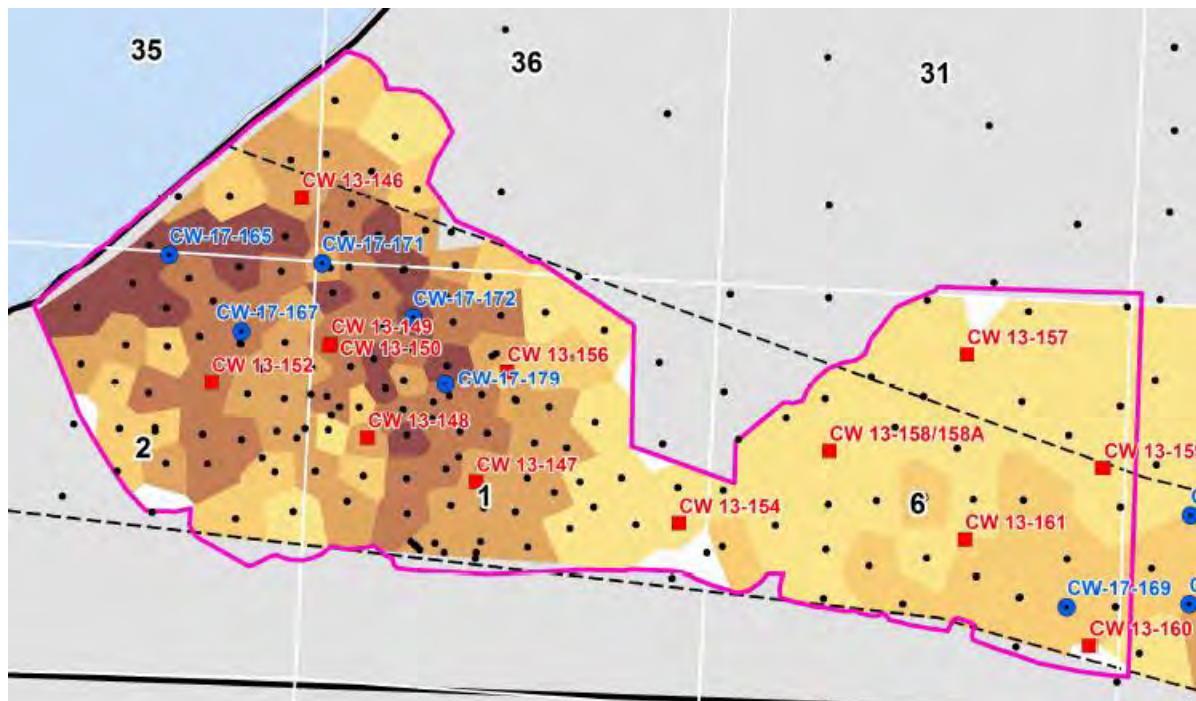
The results from the locked cycle tests suggested an average Cu recovery of 85.5% with a copper concentrate of 23.9% (Main Zone) and 16.6% Cu (Zone 6) for the last two cycles (5 & 6). A copper concentrate grade of 24% with 86% Cu recovery has been used for the FS 2012. Additional testwork was recommended in the FS 2012 due to limited sampling areas.

13.2 2013 Locked Cycle Flotation Tests

Following the 2012 FS recommendations, additional testwork was carried out on new drill cores from the Main Zone and Zone 6 at SGS Lakefield (CBS composite sample). The main purpose of this testwork program was to validate the proposed flowsheet in the FS 2012 and to evaluate ore variability. Alternative flowsheets and reagent schemes did not improve the results. On November 15, 2013, SGS Tucson received a sample identified as CBS2 Composite (20 test charges of 1 kg each). These samples were homogenized and test charges of 1.2 kg were split for head assays, grind calibration, NaHS dosage series and locked cycle flotation testing. Additional samples (CBS3 composite) were provided later and were composed of samples from the Main Zone compared to CBS2, which were comprised of both the Main Zone and Zone 6.

Figure 13.3 shows the locations of CBS samples (CBS, CBS2, CBS3 composite) collected for the 2013 testwork.

Figure 13.3: CBS 2 Sample Locations



The testwork results are summarized below. For detailed testwork procedures and results, refer to document "RRC-078-13" prepared by SGS North America Inc, dated December 2, 2013.

13.2.1 Head Assay

One test charge was selected at random, pulverized and submitted for total copper (1.84%), total iron (5.55%), total sulphur (0.50%), insoluble (69.82%) and silver assays (5.0 g/t).

13.2.2 NaHS Dosage Series

A NaHS dosage series was conducted under rougher flotation kinetics to determine the optimum dosage required to increase copper recovery. A summary of the results is summarized in Table 13.5.

Table 13.5: NaHS Dosage Series Rougher Flotation Test Results Summary

NaHS Dosage (g/t)	Cumulative Time (Minute)	Mass Recovery (%)	Cumulative Grade (%)					Cumulative Recovery (%)				
			Cu	Ag (g/t)	Fe	St	Insol	Cu	Ag	Fe	St	Insol
667	5	6.38	15.00	37	7.02	4.25	55.90	54.58	39.22	7.82	51.98	5.02
	10	10.32	10.78	28	6.88	3.08	59.49	63.48	47.34	12.39	60.98	8.64
	15	14.61	8.32	22	6.77	2.40	61.67	69.31	53.33	17.28	67.16	12.67
	20	18.40	7.03	19	6.69	2.03	62.91	73.74	58.93	21.48	71.55	16.28
	25	22.11	6.14	17	6.62	1.77	63.68	77.44	63.12	25.54	75.16	19.80
	30	25.11	5.56	16	6.55	1.61	64.41	79.65	65.96	28.70	77.46	22.75
	35	29.52	4.99	14	6.48	1.44	65.18	84.00	70.73	33.38	81.75	27.06
833	5	6.61	14.10	34	6.96	4.20	55.18	53.44	38.66	8.20	51.62	5.10
	10	11.53	9.45	24	6.73	2.85	60.68	62.45	48.38	13.83	60.91	9.78
	15	15.56	7.65	20	6.65	2.30	62.80	68.21	54.13	18.45	66.53	13.67
	20	20.07	6.37	17	6.51	1.93	64.16	73.32	60.18	23.28	71.90	18.01
	25	23.23	5.76	16	6.45	1.74	64.71	76.72	63.54	26.70	75.04	21.02
	30	26.97	5.18	15	6.42	1.56	65.25	80.07	67.41	30.88	78.19	24.62
	35	31.68	4.68	13	6.37	1.42	65.94	84.89	72.99	35.95	83.58	29.21
1,167	5	8.60	12.30	30	6.65	3.64	58.90	59.58	43.71	10.09	58.08	7.11
	10	13.79	8.78	23	6.53	2.61	61.97	68.18	53.73	15.89	66.83	12.00
	15	18.35	7.14	19	6.40	2.14	63.80	73.79	60.37	20.71	72.78	16.44
	20	22.00	6.27	18	6.34	1.89	64.55	77.64	65.32	24.60	77.16	19.94
	25	25.83	5.57	16	6.26	1.68	65.28	81.02	69.14	28.53	80.46	23.67
	30	29.72	5.03	14	6.21	1.52	65.86	84.18	72.63	32.56	83.59	27.48
	35	34.23	4.54	13	6.16	1.37	66.38	87.48	76.60	37.21	87.06	31.90
1,667	5	8.82	11.70		6.91	3.84	54.94	61.79		10.74	57.48	7.15
	10	17.31	7.23		6.50	2.40	60.29	75.02		19.87	70.57	15.40
	15	24.09	5.64		6.34	1.88	62.44	81.39		26.93	77.06	22.19
	20	30.66	4.64		6.22	1.56	63.41	85.17		33.62	81.37	28.68
	25	36.40	4.10		6.16	1.39	63.92	89.40		39.53	85.90	34.32
	30	41.08	3.71		6.12	1.27	64.28	91.20		44.39	88.24	38.96
	35	45.40	3.42		6.12	1.17	64.44	93.03		49.05	90.40	43.16

The results indicated that a NaHS dosage of 1,667 g/t is required to obtain a mass recovery of 45.4% and a total copper recovery of 93.0% with a rougher flotation time of 35 minutes.

13.2.3 Locked Cycle Flotation

Seven cycles were conducted using the CBS2 composite samples. Locked cycle flotation testing was conducted using the simplified flowsheet as shown in Figure 13.4. Results from the seven tests are summarized in Table 13.6.

Figure 13.4: Locked Cycle Flowsheet

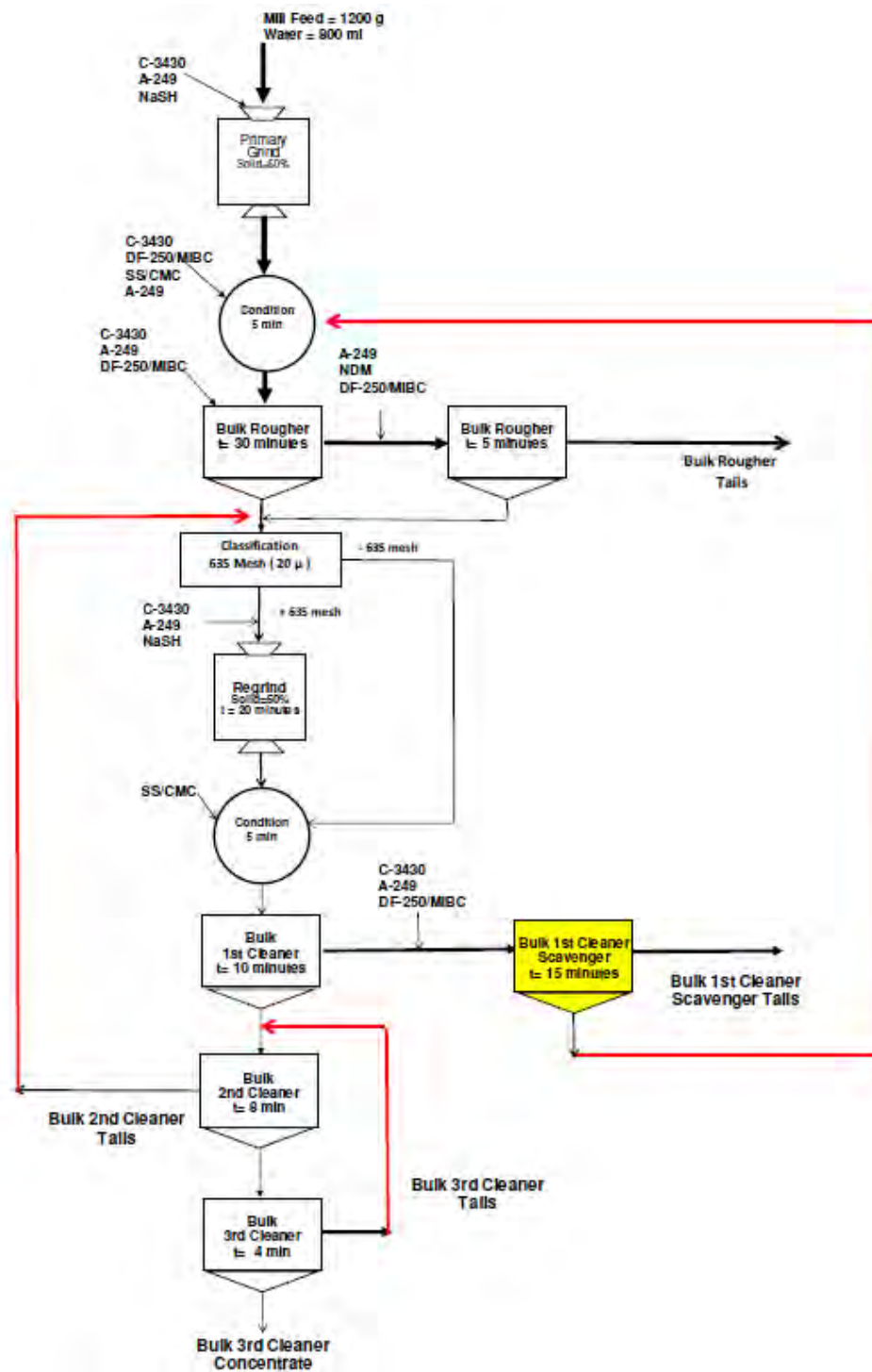


Table 13.6: CBS2 Locked Cycle Flotation Test Results Summary

Cycle	Mass Recovery (%)	Grade (%)					Recovery (%)				
		Cu	Ag (g/t)	Fe	S _T	Insol	Cu	Ag	Fe	S _T	Insol
A	4.50	31.30	74	9.19	9.58	33.92	84.71	55.05	6.49	70.76	2.14
B	4.59	28.10	69	9.31	8.51	40.18	81.04	56.31	7.48	71.10	2.58
C	4.86	30.20	68	9.04	9.16	35.34	82.94	54.33	7.38	71.97	2.41
D	5.84	24.00	58	8.90	7.31	41.44	83.99	57.05	9.35	72.51	3.43
E	4.95	29.50	75	9.52	8.93	36.88	82.11	59.10	8.24	70.71	2.59
F	4.80	29.80	72	9.50	9.00	35.92	81.70	52.42	8.12	61.45	2.43
G	5.33	25.50	65	9.01	7.89	39.76	80.31	55.85	8.10	61.57	3.00
Average	4.98	28.34	69	9.21	8.62	37.63	82.40	55.73	7.88	68.58	2.65

The following observations are made from the locked cycle tests:

- Average total copper recovery of 82.4% and concentrate copper grade of 28.3% for the CBS2 (27.7% Cu and 81% Cu recovery based on Cycles 5 and 6);
- Average total copper recovery of 81.7% and concentrate copper grade of 24.9% for the CBS3 (25.5% Cu and 79.2% Cu recovery based on Cycles 5 and 6);
- Chalcocite floated at a very slow kinetic rate in the first cleaner scavenger stage;
- Rougher and cleaner flotation time increased compared to the 2012 flowsheet;
- Liberated chalcocite was observed during the first cleaner scavenger stage;
- Copper recovery in the first cleaner scavenger flotation stage could be improved by optimizing collector type and dosage;
- Overall flotation performances were lower than the FS 2012;
- Variability locked cycle tests on all 12 individual samples were planned but did not proceed.

13.2.4 2017 Grindability Tests – Main Zone

Various main zone samples from the Copperwood deposit were submitted for a series of comminution tests which included the JK drop-weight and SMC tests, the Bond rod mill and Bond ball mill grindability tests, and the Bond abrasion test. One composite sample, made from three PQ holes, was submitted for all the tests, while the rest of the samples were submitted for selected tests, based on weight availability.

The testwork results are summarized below. For detailed testwork procedures and results, refer to document “*An Investigation into the Grindability Characteristics of Samples from the Copperwood Project*” prepared by SGS Canada Inc, dated August 24, 2017.

The grindability test results are summarized in Table 13.7 and the grindability test statistics are presented in Table 13.8.

The samples were generally characterized as moderately soft to moderately hard when tested at the coarsest sizes (DWT, SMC, and RWI), except for one sample, labelled 'Grey Laminated + Red Massive', which was significantly harder than the other samples. The samples were softer at a finer size (BWI), with the hardness ranging from soft to medium. All the samples submitted for Bond abrasion testing were classified as very mild to mild in terms of their degree of abrasiveness.

Overall, the sample named 'Grey Laminated + Red Massive' was the hardest sample tested, while the sample named 'Domino' was among the softest samples. The PQ composite was the softest sample among the ten samples tested.

Table 13.7: Grindability Test Summary

Sample Name	Relative Density	JK Parameters				RWI (kWh/t)	BWI (kWh/t)	AI (g)
		A x b ¹	A x b ²	t _a	SCSE			
CW-17-185/186/187 PQ Comp	2.70	54.7	48.9	0.63	8.6	14.2	10.3	0.009
CW-17-165	2.73	-	39.5	0.37	10.0	-	12.1	-
CW-17-167	2.76	-	42.1	0.39	9.8	-	13.0	-
CW-17-170	2.73	-	39.1	0.37	10.0	-	12.1	0.031
CW-17-173	2.74	-	42.2	0.40	9.7	15.4	11.3	-
CW-17-174	-	-	-	-	-	15.6	11.6	0.013
CW-17-176	2.72	-	41.0	0.39	9.8	-	13.5	0.017
Grey Laminated	-	-	-	-	-	-	14.2	-
Grey Laminated + Red Massive	2.73	-	33.1	0.31	10.9	17.0	13.6	0.011
Domino	2.74	-	44.5	0.42	9.5	15.0	10.9	0.001

¹ A x b from DWT

² A x b from SMC

Table 13.8: Grindability Test Statistics

Statistics	Relative Density	JK Parameters		RWI (kWh/t)	BWI (kWh/t)	AI (g)
		A x b	SCSE			
Average	2.73	42.6	9.7	15.4	12.3	0.014
Std. Dev.	0.02	6.5	0.7	1.0	1.3	0.010
Rel. Std. Dev.	1	15	7	7	10	73
Minimum	2.70	54.7	8.6	14.2	10.3	0.001
10th Percentile	2.71	50.6	8.9	14.5	10.8	0.005
25th Percentile	2.73	43.9	9.5	15.0	11.4	0.010
Median	2.73	41.5	9.8	15.4	12.1	0.012
75th Percentile	2.74	39.4	10.0	15.6	13.4	0.016
90th Percentile	2.75	37.3	10.3	16.4	13.7	0.024
Maximum	2.76	33.1	10.9	17.0	14.2	0.031

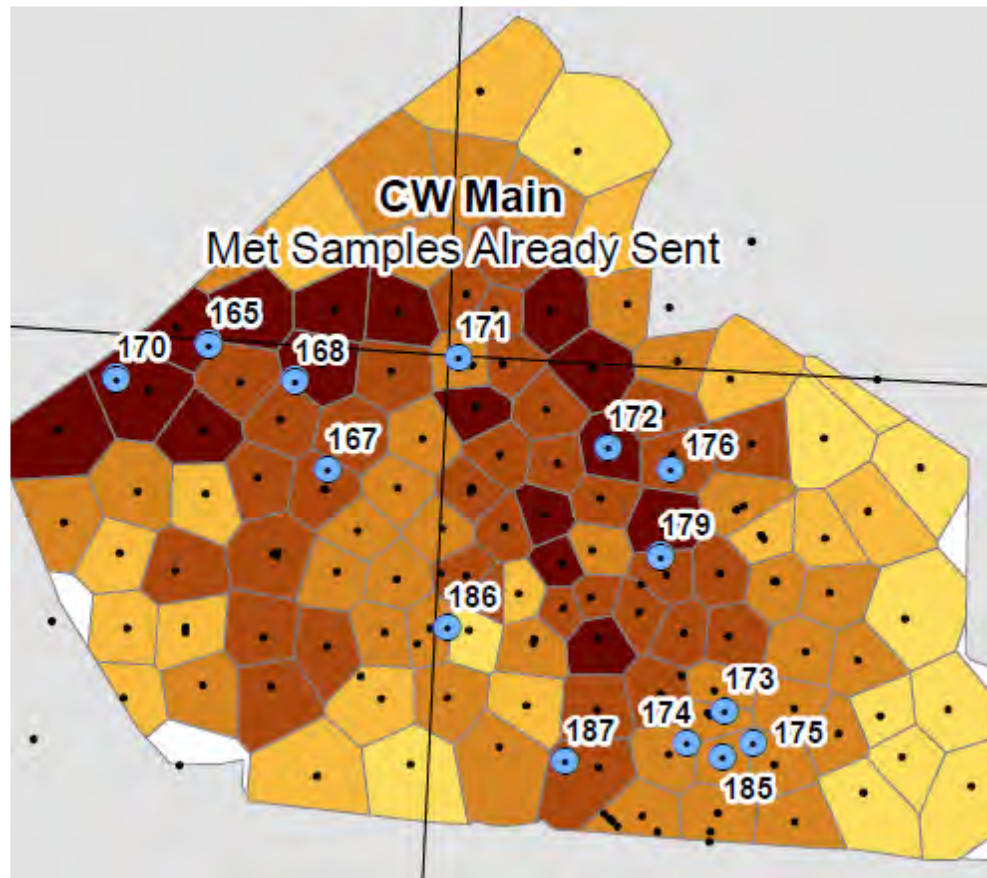
13.2.5 Sample Preparation and Testing Matrix

A total of 26 core boxes from 11 drill holes were received at SGS Lakefield on June 22, 2017. These samples were used to generate 9 comminution samples. Of these samples, six consisted of drill hole composites representing a blend of the three ore types, while the three other samples were ore type composites. One ore type composite represented the Grey Laminated ore type, made from three holes, while material from three drill holes was combined to make the Grey Laminated + Red Massive composite and the Domino composite. The information for the 9 comminution samples is summarized in Table 13.9 and the drill hole locations are shown in Figure 13.5.

Table 13.9: Sample Preparation Information**Comminution Samples - Shipment 1**

Sample Name	Drill Hole ID	From (m)	To (m)	Length (m)	Ore Type
CW-17-165	CW-17-165	228.84	229.85	1.01	Grey Laminated
	CW-17-165	229.85	229.91	0.06	Red Massive
	CW-17-165	229.91	231.80	1.89	Domino
CW-17-167	CW-17-167	167.61	168.50	0.89	Grey Laminated
	CW-17-167	168.50	168.72	0.22	Red Massive
	CW-17-167	168.72	170.49	1.77	Domino
CW-17-170	CW-17-170	299.15	300.30	1.15	Grey Laminated
	CW-17-170	300.30	300.54	0.24	Red Massive
	CW-17-170	300.54	302.57	2.03	Domino
Grey Laminated	CW-17-171	203.11	203.98	0.88	Grey Laminated
	CW-17-172	175.47	176.37	0.90	Grey Laminated
CW-17-173	CW-17-173	112.15	113.38	1.23	Grey Laminated
	CW-17-173	113.38	113.61	0.23	Red Massive
	CW-17-173	113.61	116.95	3.34	Domino
CW-17-174	CW-17-174	101.75	102.88	1.13	Grey Laminated
	CW-17-174	102.88	103.62	0.74	Red Massive
	CW-17-174	103.62	106.10	2.47	Domino
CW-17-176	CW-17-176	235.17	236.78	1.61	Grey Laminated
	CW-17-176	236.78	237.18	0.40	Red Massive
	CW-17-176	237.18	239.63	2.45	Domino
Grey Laminated + Red Massive	CW-17-168	309.13	310.70	1.57	Grey Laminated
	CW-17-168	310.70	311.37	0.67	Red Massive
	CW-17-175	99.39	100.58	1.19	Grey Laminated
	CW-17-175	100.58	100.78	0.20	Red Massive
	CW-17-179B	145.83	146.43	0.60	Grey Laminated
	CW-17-179B	146.43	147.14	0.70	Red Massive
Domino	CW-17-168	311.37	314.75	3.38	Domino
	CW-17-175	100.78	103.66	2.88	Domino
	CW-17-179B	147.14	149.03	1.90	Domino

Figure 13.5: Main Zone Samples Location Map



On July 12, 2017, a second shipment of one crate with 12 bags of whole PQ core was received. The 12 bags represented material from four drill holes, and each bag contained one ore type sample. A single PQ composite was made by combining the material in nine of these bags (3 holes) from this shipment. The weights of the composite samples, as well as the testing matrix, are presented in Table 13.10.

Table 13.10: Sample Received Weights and Testing Matrix

Sample Name	Weight (kg)	Test				
		DWT	SMC	RWI	BWI	AI
CW-17-185/186/187 PQ Comp	136.8*	X	X	X	X	X
CW-17-165	24.7	-	X	-	X	-
CW-17-167	21.5	-	X	-	X	-
CW-17-170	29.7	-	X	-	X	X
CW-17-173	12.7	-	X	X	X	-
CW-17-174	36.0	-	-	X	X	X
CW-17-176	36.7	-	X	-	X	X
Grey Laminated	28.3	-	-	-	X	-
Grey Laminated + Red Massive	45.0	-	X	X	X	X
Domino	61.8	-	X	X	X	X
Total	433	1	8	5	10	6

13.2.6 JK Drop-Weight and SMC Tests

The JK drop-weight test ("DWT") was performed on the composite labelled 'CW-17-185/186/187 PQ Comp'. The SMC test is an abbreviated version of the standard JK drop-weight test performed on 100 rocks from a single size fraction (-22.4/+19.0 mm in this case). The SMC test was performed on a total of eight samples, including the sample on which the DWT test was performed. The SMC test results are preferably calibrated against reference samples submitted for the standard DWT to consider the natural 'gradient of hardness' by size, which can widely vary from one ore to another.

The test results are summarized in Table 13.11.

Table 13.11: JK Drop Weight and SMC Test Results Summary

Sample Name	A	b	A x b	Hardness Percentile	t ₃ ¹	Hardness Percentile	DWI (kWh/m ³)	M _{1a} (kWh/t)	M _{1b} (kWh/t)	M _{1c} (kWh/t)	SCSE (kWh/t)	Relative Density
CW-17-185/186/187 PQ Comp	51.1	1.07	54.7	37	0.63	30	-	-	-	-	8.6	2.70
CW-17-185/186/187 PQ Comp	58.2	0.84	48.9	45	0.47	-	5.8	17.4	12.5	6.5	9.0	2.70
CW-17-165	77.5	0.51	39.5	62	0.37	-	7.2	20.5	15.4	7.9	10.0	2.73
CW-17-167	69.0	0.61	42.1	57	0.39	-	6.9	19.6	14.6	7.5	9.8	2.76
CW-17-170	71.0	0.55	39.1	63	0.37	-	7.3	20.6	15.5	8.0	10.0	2.73
CW-17-173	72.8	0.58	42.2	57	0.40	-	6.9	19.5	14.5	7.5	9.7	2.74
CW-17-176	71.9	0.57	41.0	59	0.39	-	7.0	19.9	14.8	7.7	9.8	2.72
Grey Laminated + Red Massive	78.9	0.42	33.1	77	0.31	-	8.7	23.7	18.4	9.5	10.9	2.73
Domino	68.5	0.65	44.5	52	0.42	-	6.4	18.6	13.6	7.0	9.5	2.74

SMC results are presented in italics

¹The t₃ value reported as part of the SMC procedure is an estimate

The DWT sample was characterized as moderately soft with respect to resistance to both impact (A x b) and abrasion (t₃) breakages. The SMC test done on the same sample was slightly harder and was

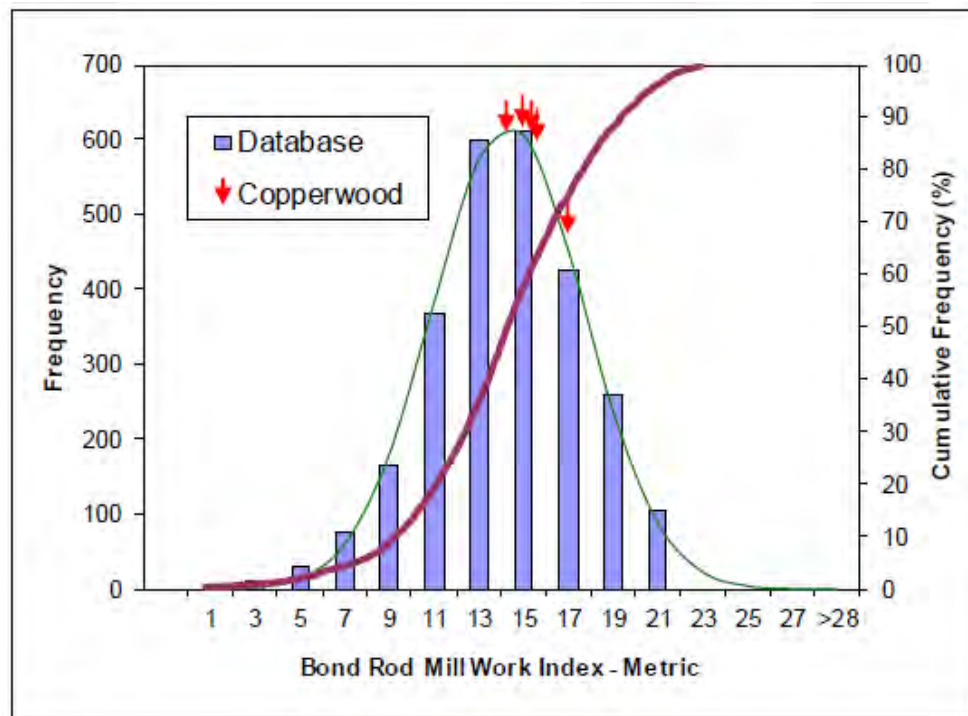
categorized as medium in terms of A x b. The rest of the samples fell in the medium to moderately hard range of JKTech's database in terms of A x b, with the exception of the sample labelled 'Grey Laminated + Red Massive', which was categorized as hard. The measured rock relative density varied from 2.70 to 2.76. The PQ composite was the softest sample among the eight samples tested.

13.2.7 Bond Rod Mill Grindability Test

Five samples were submitted for the Bond rod mill grindability test at 14-mesh of grind (1,180 microns). The test results are summarized in Table 13.12 and the Bond Rod Mill Work Indices ("RWI") are compared to the SGS database in Figure 13.6.

Table 13.12: Bond Rod Mill Grindability Test Results Summary

Sample Name	Mesh of Grind	F ₈₀ (μm)	P ₈₀ (μm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
CW-17-185/186/187 PQ Comp	14	11,206	972	9.85	14.2	50
CW-17-173	14	10,498	953	8.66	15.4	63
CW-17-174	14	10,352	972	8.72	15.6	66
Grey Laminated + Red Massive	14	10,238	967	7.59	17.0	79
Domino	14	10,456	976	9.25	15.0	59

Figure 13.6: Bond Ball Mill Work Index Database

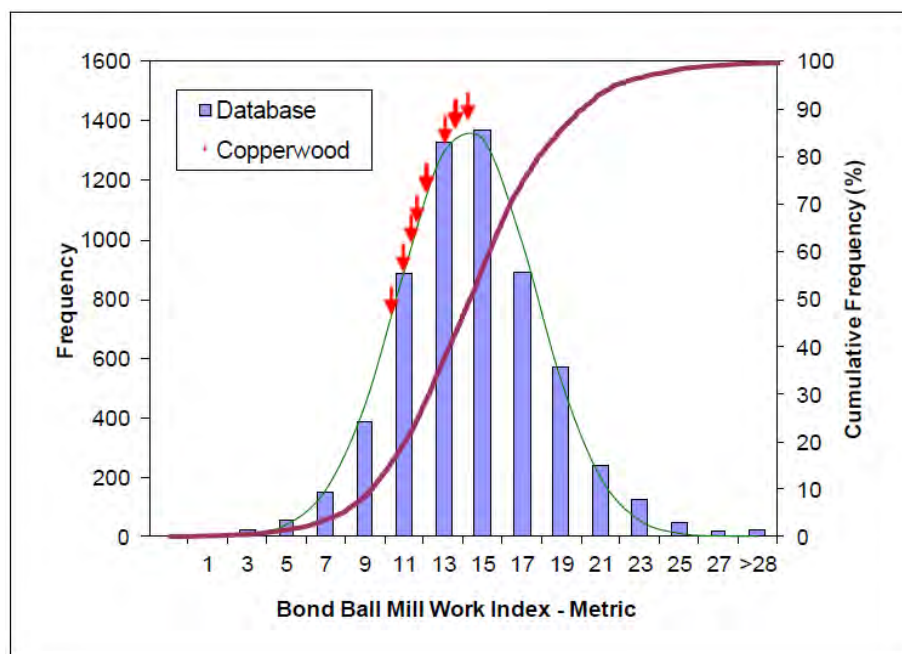
The RWI's varied from 14.2 to 17.0 kWh/t. Most of the samples fell in the medium to moderately hard range of hardness of the SGS database, with one sample ('Grey Laminated + Red Massive') being categorized as hard. The PQ composite was the softest sample among the five samples tested.

13.2.8 Bond Ball Mill Grindability Test

Ten samples were submitted for the Bond ball mill grindability test which was performed at 230-mesh of grind (63 microns). The test results are summarized in Table 13.13 and the Bond Ball Mill Work Indices ("BWI") are compared to the SGS database in Figure 13.7.

Table 13.13: Bond Ball Mill Grindability Test Results Summary

Sample Name	Mesh of Grind	F ₈₀ (μm)	P ₈₀ (μm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
CW-17-185/186/187 PQ Comp	230	2,456	43	1.48	10.3	12
CW-17-165	230	2,359	42	1.22	12.1	25
CW-17-167	230	2,553	42	1.10	13.0	36
CW-17-170	230	2,449	43	1.23	12.1	25
CW-17-173	230	2,416	43	1.34	11.3	19
CW-17-174	230	2,326	43	1.30	11.6	21
CW-17-176	230	2,452	45	1.10	13.5	41
Grey Laminated	230	2,347	46	1.06	14.2	48
Grey Laminated + Red Massive	230	2,416	43	1.07	13.6	42
Domino	230	2,433	42	1.36	10.9	16

Figure 13.7: Bond Mill Work Index Comparison

The BWI's varied from 10.3 to 14.2 kWh/t. Six out of ten samples fell in the soft range of hardness of the SGS database, while four samples ('CW-17-167', 'CW-17-176', 'Grey Laminated', and 'Grey Laminated + Red Massive') were categorized as moderately soft to medium. The attained P80 values varied from 42 to 46 microns. The PQ composite was the softest sample among the ten samples tested.

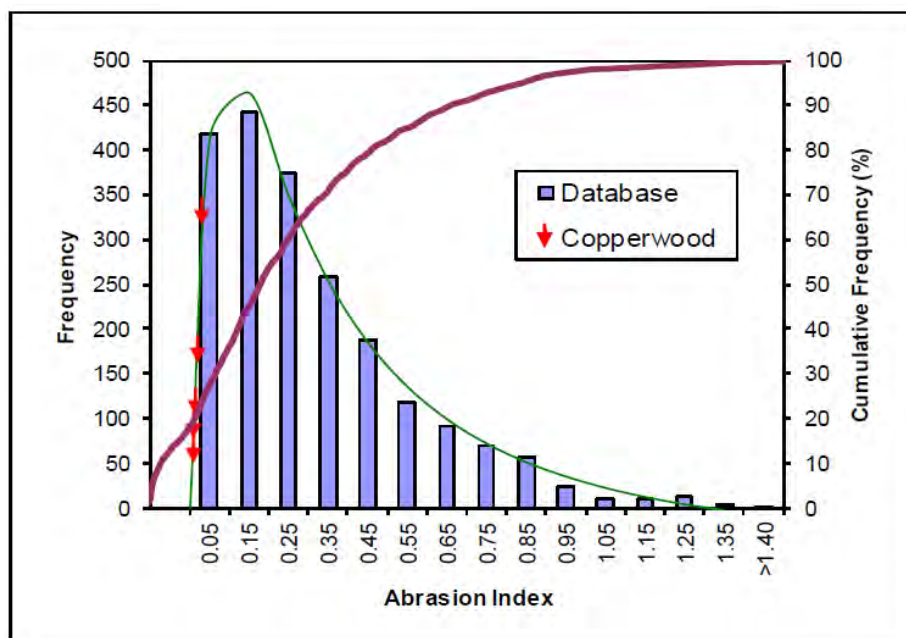
13.2.9 Bond Abrasion Test

Six samples were submitted for the Bond abrasion test. The test results are summarized in Table 13.14 and the Bond Abrasion Indices ("Ai") are compared to the SGS database in Figure 13.8.

Table 13.14: Bond Abrasion Test Results Summary

Sample Name	Ai (g)	Percentile of Abrasivity
CW-17-185/186/187 PQ Comp	0.009	7
CW-17-170	0.031	11
CW-17-174	0.013	8
CW-17-176	0.017	9
Grey Laminated + Red Massive	0.011	7
Domino	0.001	3

Figure 13.8: Bond Abrasion Index Comparison



The Ai values ranged from 0.001 to 0.031 g, which placed all samples in the very mild to mild range of abrasiveness in the SGS database.

13.3 2017 Grindability Tests – Zones 5 and 6

Additional samples from Zone 5 and Zone 6 of the Copperwood deposit were submitted for a series of comminution tests which included SMC tests, Bond ball mill grindability tests, and the Bond abrasion test. Most of the samples were half core and provided with limited weight; therefore, different holes were combined to make different composites according to their location.

The testwork results are summarized below. For detailed testwork procedures and results, refer to document “16256-002 Copperwood Grinding” prepared by SGS Minerals Services, dated December 2017.

The grindability test results are summarized in Table 13.15. Results from the grinding tests are inline or similar to the results from the Main Zone samples.

Table 13.15: Grindability Results Summary for Zones 5 and 6

Sample Name	Relative Density	JK Parameters			BWI (kWh/t)	AI (g)
		A x b	t_a ¹	SCSE		
CW-17-162	2.70	47.5	0.46	9.1	12.9	0.065
CW-17-178	2.71	47.4	0.45	9.2	12.6	0.010
CW-17-182	2.73	55.9	0.53	8.6	14.6	0.069
Comp-17-163-166	-	-	-	-	12.6	-
Comp-17-169-196	-	-	-	-	12.6	0.034
Comp-17-181-183	-	-	-	-	13.1	0.093
Comp-17-188-195	-	-	-	-	12.6	0.072
Comp-17-189-194	-	-	-	-	12.8	-
Comp-17-190-192	-	-	-	-	14.2	0.087
Comp-17-191-197	-	-	-	-	12.5	-

¹The t_a value reported as part of the SMC procedure is an estimate

13.3.1 Sample Preparation

A total of 36 boxes from 11 drill holes were received at SGS Lakefield. These samples are used to generate 13 comminution samples. From these samples, six are from Zone 5 and seven are from Zone 6. The information for the thirteen comminution samples is summarized in Table 13.16 and the drill hole locations are shown in Figure 13.9.

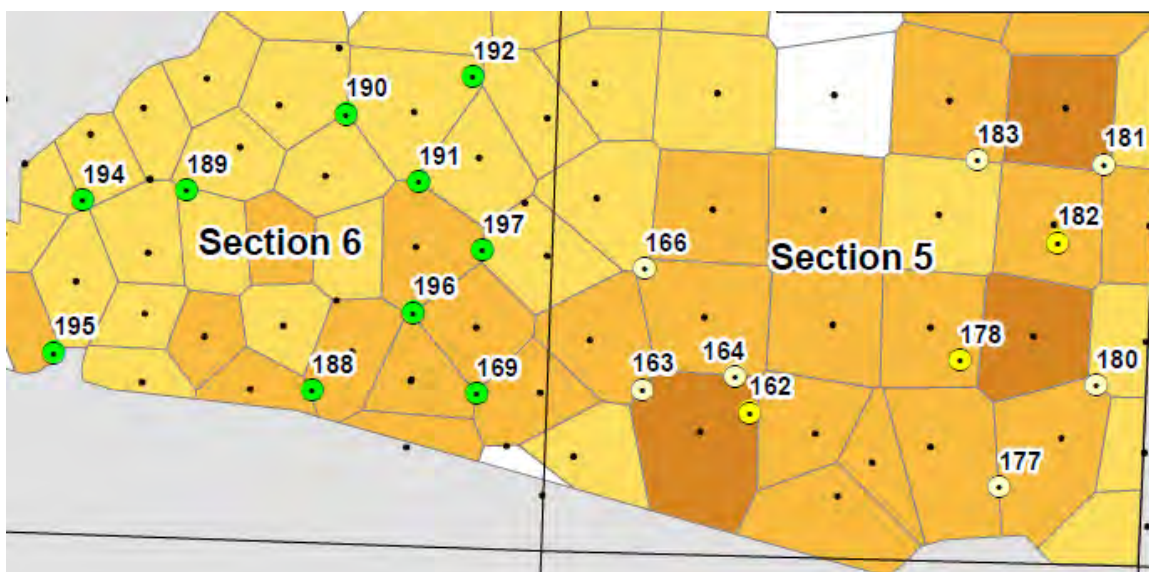
Table 13.16: Zones 5 and 6 Test Matrix

Sample	Sample ID	SMC	BWI	Head Characterization	Bond Abrasion	Flotation
Sample 1	CW-17-162	x	x	x		x
Sample 2	CW-17-178	x	x	x		x
Sample 3	CW-17-182	x	x	x		x
Sample 4	CW-17-164			x		x
Sample 5	CW-17-177			x		x
Sample 6	CW-17-180			x		x
Composite 1	CW-17-169/196		x	x	x	x
Composite 2	CW-17-1188/195		x	x	x	x
Composite 3	CW-17-190-192		x	x	x	x
Composite 4	CW-17-181-183		x	x	x	x
Composite 5	CW-17-189/194		x	x		x
Composite 6	CW-17-181/194		x	x		x
Composite 7	CW-17-163		x	x		

Table 13.17: Sample Inventory and Preparation Summary

Sample/Composite	Box No.	Core No.	Core Location	Core Type	Rock Type
Sample 1	1	CW-17-162	Section 5	Whole Core	DOMN to CHSA
	2	CW-17-162	Section 5	Whole Core	RLAM
Sample 2	3	CW-17-178	Section 5	Whole Core	DOMN to CHSA
	4	CW-17-178	Section 5	Whole Core	RLAM to DOMN
Sample 3	5	CW-17-182	Section 5	Whole Core	LTRA to CHSA
	6	CW-17-182	Section 5	Whole Core	UPSA to LTRA
Sample 4	7	CW-17-164	Section 5	1/2 Core	RLAM to DOMN
	8	CW-17-164	Section 5	1/2 Core	CHSA to CHSA
Sample 5	9	CW-17-177	Section 5	1/2 Core	DOMN to CHSA
	10	CW-17-177	Section 5	1/2 Core	UPSA to DOMN
Sample 6	11	CW-17-180	Section 5	1/2 Core	CHSI to CHSH
	12	CW-17-180	Section 5	1/2 Core	RSIL to DOMN
Composite 1	13	CW-17-169	Section 6	1/2 Core	DOMN to CHSA
	14	CW-17-169	Section 6	1/2 Core	RLAM to DOMN
	15	CW-17-196	Section 6	1/2 Core	DOMN to CHSA
	16	CW-17-196	Section 6	1/2 Core	RLAM to DOMN
Composite 2	17	CW-17-188	Section 6	1/2 Core	DOMN to CHSA
	18	CW-17-188	Section 6	1/2 Core	RLAM to DOMN
	19	CW-17-195	Section 6	1/2 Core	RLAM to DOMN
	20	CW-17-195	Section 6	1/2 Core	DOMN to CHSA
Composite 3	21	CW-17-190	Section 6	1/2 Core	DOMN to CHSA
	22	CW-17-190	Section 6	1/2 Core	RLAM to DOMN
	23	CW-17-192	Section 6	1/2 Core	CHSA to CHSA
	24	CW-17-192	Section 6	1/2 Core	RLAM to CHSA
Composite 4	25	CW-17-181	Section 5	1/2 Core	DOMN to CHSA
	26	CW-17-181	Section 5	1/2 Core	RSIL to DOMN
	27	CW-17-183	Section 5	1/2 Core	UPSA to CHSI
Composite 5	28	CW-17-189	Section 6	1/2 Core	RLAM to DOMN
	29	CW-17-189	Section 6	1/2 Core	CHSA to CHSA
	30	CW-17-194	Section 6	1/2 Core	DOMN to CHSA
	31	CW-17-194	Section 6	1/2 Core	RLAM to DOMN
Composite 6	32	CW-17-191	Section 6	1/2 Core	DOMN to CHSA
	33	CW-17-191	Section 6	1/2 Core	RLAM to DOMN
	34	CW-17-197	Section 6	1/2 Core	RLAM to CHSA
Composite 7	35	CW-17-163	Section 5	1/2 Core	RLAM to CHSA
	36	CW-17-166	Section 5	1/2 Core	RLAM to CHSA

Figure 13.9: Zones 5 and 6 Samples Location Map



13.3.2 SMC Tests

SMC tests were done on three samples. The test results are summarized in Table 13.18.

Table 13.18: SMC Test Results Summary

Sample Name	A	b	A x b	Hardness Percentile	t_a^1	DWI (kWh/m ³)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	SCSE (kWh/t)	Relative Density
CW-17-162	72.0	0.66	47.5	47	0.46	5.66	17.0	12.1	6.3	9.14	2.70
CW-17-178	71.8	0.66	47.4	47	0.45	5.68	17.0	12.2	6.3	9.16	2.71
CW-17-182	73.5	0.76	55.9	35	0.53	4.89	15.0	10.4	5.4	8.56	2.73

¹The t_a value reported as part of the SMC procedure is an estimate

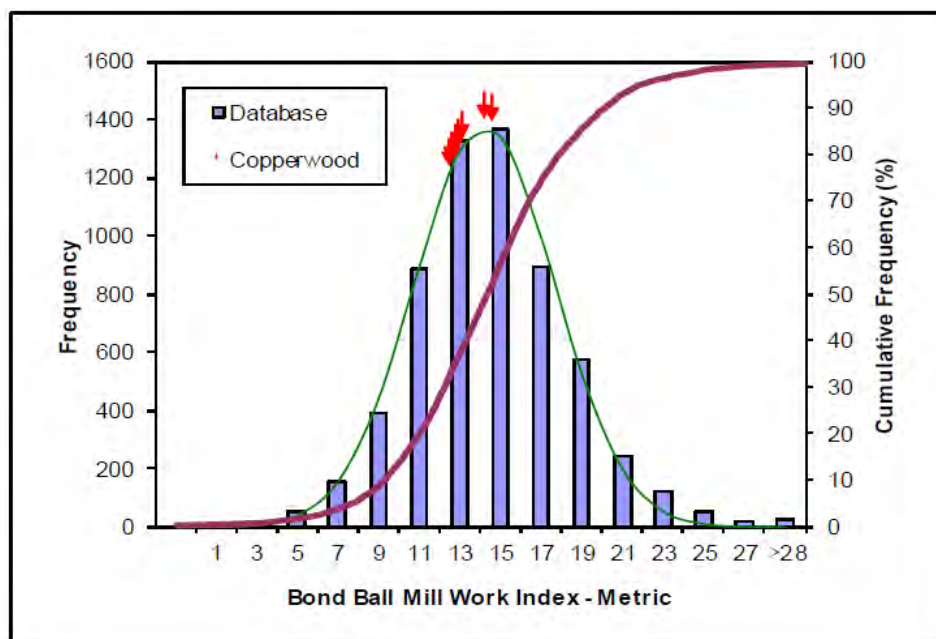
Zone 5 and Zone 6 samples SMC results are in line with the Main Zone results. The A x b fell in the medium to moderately hard range of JK Tech's database. The measured rock relative density varied from 2.70 to 2.73, which also fell in the range of the Main Zone results.

13.3.3 Bond Ball Mill Grindability Test

Ten samples were submitted for the Bond ball mill grindability test which was performed at 230 mesh of grind (63 microns). The test results are summarized in Table 13.19 and the Bond ball mill work indices ("BWI") are compared to the SGS database in Figure 13.10.

Table 13.19: Bond Ball Mill Grindability Test Results Summary

Sample Name	Mesh of Grind	F ₈₀ (µm)	P ₈₀ (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
CW-17-162	230	2,394	44	1.16	12.9	35
CW-17-178	230	2,422	44	1.19	12.6	31
CW-17-182	230	2,458	45	1.01	14.6	52
Comp-17-163-166	230	2,367	45	1.21	12.6	31
Comp-17-169-196	230	2,400	45	1.20	12.6	31
Comp-17-181-183	230	2,474	45	1.14	13.1	37
Comp-17-188-195	230	2,467	43	1.17	12.6	31
Comp-17-189-194	230	2,306	44	1.17	12.8	34
Comp-17-190-192	230	2,484	46	1.06	14.2	48
Comp-17-191-197	230	2,467	44	1.19	12.5	30

Figure 13.10: Bond Ball Mill Work Index Comparison

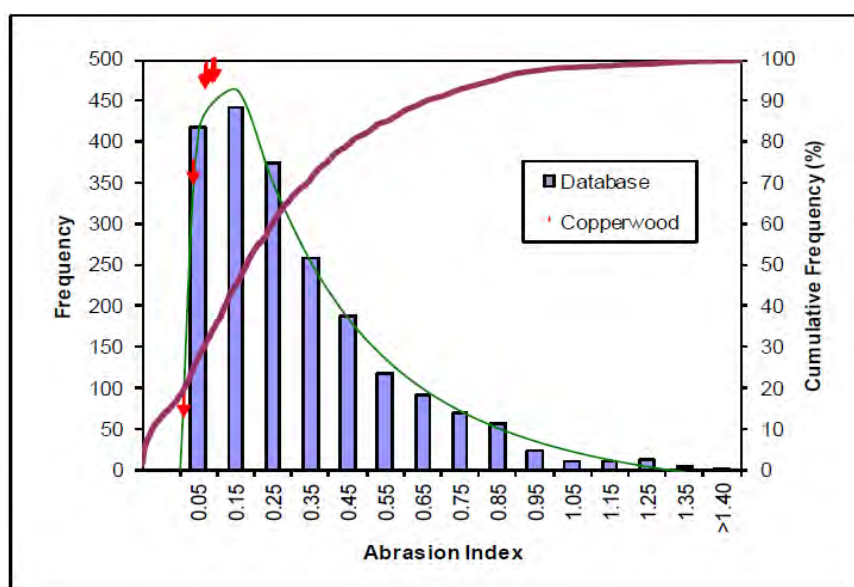
The BWI's varied from 12.5 to 14.6 kWh/t. All 10 samples fell in the moderately soft to medium range of hardness. The attained P80 values varied from 43 to 46 microns. All of these results are comparable to the Main Zone results.

13.3.4 Bond Abrasion Test

Seven samples were submitted for the Bond abrasion test. The test results are summarized in Table 13.20 and the Ai are compared to the SGS database in Figure 13.11.

Table 13.20: Bond Abrasion Test Results Summary

Sample Name	AI (g)	Percentile of Abrasivity
CW-17-162	0.065	15
CW-17-178	0.010	7
CW-17-182	0.069	15
Comp-17-169-196	0.034	12
Comp-17-181-183	0.093	19
Comp-17-188-195	0.072	16
Comp-17-190-192	0.087	18

Figure 13.11: Bond Abrasion Index Comparison

The Ai values ranged from 0.01 to 0.093 g, which placed all samples in the mild range of abrasiveness in the SGS database. Samples from Zone 5 and Zone 6 are slightly more abrasive than samples from Main Zone.

13.3.5 Special Jar Mill Grindability Test and SMD Lab Test

One sample of copper flotation concentrate was received at the Metso York test plant in February 2018. The Special Jar Mill Grindability test was performed to determine the specific energy required to grind the as-received material to eighty percent passing 15.0 μm using US standard test sieve and laser size analysis ("LSA") methods.

A Stirred Media Detritor Test (“SMD”) was also performed to determine the specific energy required to grind the as-received material to eighty percent passing 15 µm using LSA. Table 13.21 summarizes the results of the Jar Mill and the SMD tests. The specific energy reported includes a ten percent safety factor. An additional efficiency factor can be applied to the Jar Mill specific energy to generate a Vertimill specific energy. This Vertimill efficiency factor is based on the specific operating parameters of the Vertimill application.

The SMD specific energy is a direct scale from test operation to equipment sizing. The difference in particle size distribution of the Jar Mill test products to SMD test feed is attributed to the sizing methodologies used. Typically, the LSA sizing methodology reflects a coarser d80 than the sieve analysis. The jar mill 1 specific energy result should be considered an estimate due to the first estimates of the d80 on Run 2 and Run 3.

Table 13.21: Metso Grinding Testwork Results

Test	F ₈₀ (µm)	P ₈₀ (µm)	Jar Mill Specific Energy (kWh/mt)	SMD Specific Energy (kWh/mt)
Jar Mill 1	47.8	15.0	11.24	N/A
Jar Mill 2	50.6	15.0	27.14	N/A
SMD	50.6	15.0	N/A	4.69

13.3.6 High Intensity Grindability Test

Outotec received a sample of copper rougher concentrate from SGS Lakefield Inc. to be tested with HIG5 (7.5 kW, 8-liter HIGmill™) test unit in ORC, Finland. The sample was Highland Copper’s Copperwood flotation concentrate. Target product fineness was P80 = 15 µm. The sample received had a F80 = 41 µm and solids SG = 2.96 g/cm³. The sample weight received was 6.5 kg, allowing ORC to perform a standard small sample HIGmill™ test. The coefficient of determination from the testwork data, denoted as R2 (‘R squared’) = 0.9948, indicating good accuracy of the results. The grindability signature plot curve has the equation:

$$SGE = 79395 \times P80^{-3.434}$$

The range of Specific Grinding Energy (SGE) = 8.7 to 51.8 kWh/t, corresponding to product particle sizes, P80 = 14 to 9 µm. To the target grind P80 = 15 microns, the grind was relatively easy with the Specific Grinding Energy (SGE) = 7.3 kWh/t, below 10 to 11 microns, it is likely that natural mineral grain boundaries have been met resulting in relatively higher required SGE compared to the corresponding particle reduction size.

13.4 2017 / 2018 Flotation Optimization Tests

Approximately 120 kg of Grey Laminate/Red Massive and Domino ores was shipped to SGS Canada Inc. to confirm historical results from previous flowsheets and to further optimize the process by targeting maximum copper recoveries and copper concentrate grade.

The testwork results are summarized below. For detailed testwork procedures and results, refer to document “*An Investigation Into Optimization Flotation Testwork on Material From Copperwood Deposit*”, Project No. 16256-002”, prepared by SGS Canada Inc, dated May, 2018.

13.4.1 Test Program Summary

The test program is summarized below:

- Receipt and preparation of samples for the main flotation program;
- Head mineralogy and assay characterization;
- Bench scale batch rougher and cleaner flotation optimization testing;
- Flash flotation;
- Locked cycle flotation optimization testing;
- Flotation product mineralogy.

13.4.2 Sample Location and Composite Definition

Samples from the Main Zone were collected from five different drill holes. Two composites were collected from each drill holes giving a total of ten composites for testing. A portion of each composite is collected and stored for LCT. The remaining material is mixed and prepared to create the master composite for developmental work. A summary of the sample preparation for the Main Zone material is shown in Table 13.22.

Table 13.22: Main Zone Sample Preparation Summary

Drill Hole Number	Sample ID	Ore Type	Material Mass for LCT (kg)	Residue Material Mass for Master Comp. (kg)
LCT-CW-17-165A	Composite 1	Grey Laminate + Red Massive	3.7	5.3
	Composite 2	Domino	6.0	8.3
LCT-CW-17-167A	Composite 3	Grey Laminate + Red Massive	3.5	5.7
	Composite 4	Domino	6.0	9.6
LCT-CW-17-171A	Composite 5	Grey Laminate + Red Massive	3.2	4.7
	Composite 6	Domino	6.0	9.3
LCT-CW-17-172A	Composite 7	Grey Laminate + Red Massive	3.2	8.0
	Composite 8	Domino	6.0	11.0
LCT-CW-17-179A	Composite 9	Grey Laminate + Red Massive	3.9	6.0
	Composite 10	Domino	6.0	9.1
Total				77

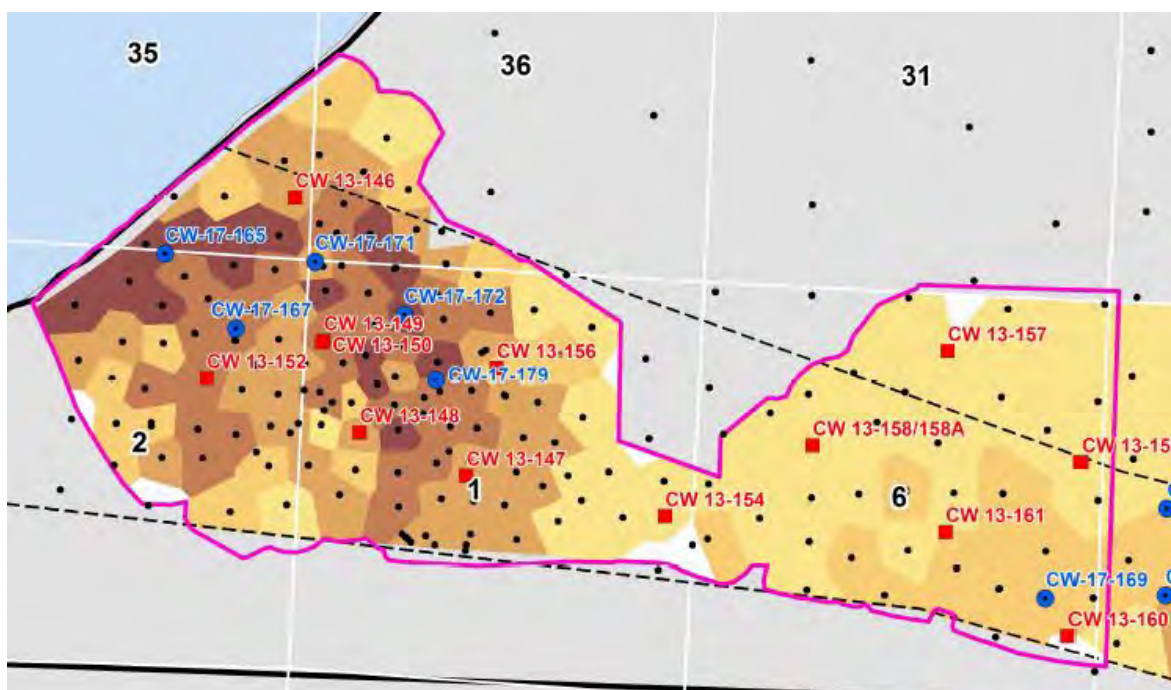
Figure 13.12: Sample Location Main Zone (in blue)

Table 13.23: Flotation Sample Grinding

	Approx. wt. (kg)	
Grey Laminated + Red Massive	17.0	SMC/AI Rej. Drum 55486
Domino	15.5	SMC/AI Rej. Drum 55486
Domino	17.8	Unused material, Drum 55486
CW-17-165	16.5	SMC/AI Rej. Drum 55486
CW-17-167	13.0	SMC/AI Rej. Drum 55486
CW-17-170	16.5	SMC/AI Rej. Drum 55486
CW-17-173	12.5	SMC/AI Rej. Drum 55486
CW-17-176	16.0	SMC Rej. Drum 55486
Various SMC Prod.	9.5	Drum 55486
CW-17-185/186/187 PQ Comp	14.5	DWT Rej. +1/2" 1 of 2 Drum 55486
CW-17-185/186/187 PQ Comp	14.0	DWT Rej. +1/2" 2 of 2 Drum 55486
CW-17-185/186/187 PQ Comp	26.5	DWT Rej. -1/2" Drum 55486
CW-17-185/186/187 PQ Comp	15.0	DWT Prod 1 of 2 Box 56314
CW-17-185/186/187 PQ Comp	10.0	DWT Prod 2 of 2 Box 56315
	214.3	

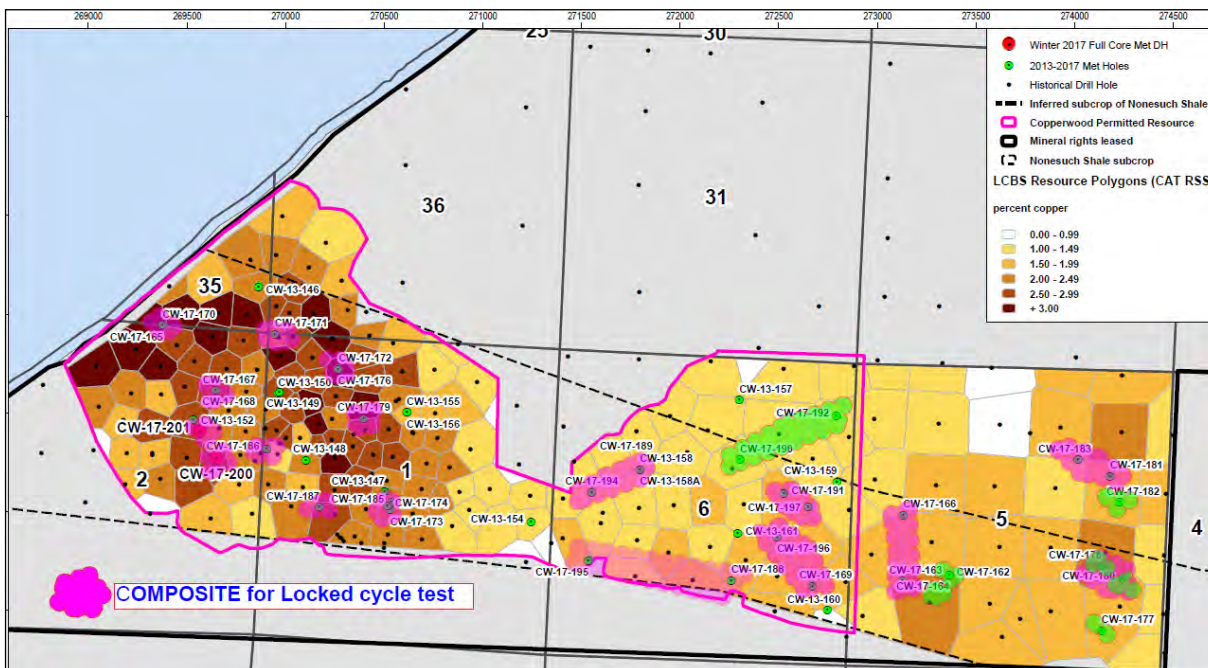
**Refer to for sample location.*

Overall 77 batch flotation tests were performed. The first 55 batch flotation and the 1st locked cycle tests were carried out on the master composite. Flotation tests F56-F59 were done on a composite composed of CW-17-185, 186 and 187. The grindability testwork remaining samples were used for flotation tests F60 to F73 and LCT4 to LCT8 (Table 13.23). Table 13.24 and Figure 13.13 show the list of drill holes and location respectively. For the variability testwork, a total of 17 samples were selected over the deposit.

Table 13.24: Overall Drill Holes

Drill Hole	Location	Drill Hole	Location
CW-17-201	Main Zone	CW-17-169-196	Zone 6
CW-17-200	Main Zone	CW-17-163-166	Zone 5
CW-17-179A	Main Zone	CW-17-165A	Main Zone
CW-17-185	Main Zone	CW-17-187	Main Zone
CW-17-171A	Main Zone	CW-17-186	Main Zone
CW-17-189-194	Zone 6	CW-17-167A	Main Zone
CW-17-172A	Main Zone	CW-17-191-197	Zone 6
CW-17-188-195	Zone 6	CW-17-181-183	Zone 5
CW-17-178-180	Zone 5	-	-

Figure 13.13: LCT Samples Drill Holes Map



13.4.3 Head Assays

The head assays for the ten composites from the Main Zone are summarized in Table 13.25.

Table 13.25: Heady Assay for Main Zone Composites

Description	Unit	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8	Comp 9	Comp 10	Master Comp
Cu	%	1.24	3.15	1.42	3.14	1.64	3.44	1.70	3.13	1.24	2.90	2.13
Cu Acetic	%	0.097	0.35	0.097	0.21	0.091	0.25	0.080	0.20	0.073	0.16	0.13
Cu H2SO4	%	0.15	0.44	0.17	0.35	0.16	0.34	0.14	0.34	0.15	0.30	0.20
Cu NaCN	%	1.19	2.98	1.39	2.84	1.30	2.95	1.32	3.25	1.19	2.76	2.10
Ag	g/t	< 10	< 10	13	< 10	14.1	< 10	14.9	< 10	< 10	< 10	8
S	%	0.31	0.77	0.32	0.66	0.34	0.64	0.32	0.71	0.28	0.64	0.54
S=	%	0.27	0.67	0.30	0.66	0.32	0.61	0.31	0.62	0.28	0.62	0.52
Al	g/t	73,700	81,300	77,500	82,700	71,100	82,100	75,800	80,600	78,300	84,000	79,941
As	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30
Ba	g/t	858	3,260	416	672	542	413	347	413	401	397	714
Be	g/t	1.66	2.56	1.86	2.62	1.82	2.46	1.86	2.54	1.94	2.65	2.28
Bi	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Ca	g/t	20,200	6,630	11,800	10,500	22,800	5,720	9,530	5,810	13,600	5,730	10,007
Cd	g/t	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2
Co	g/t	34	39	38	38	35	38	36	39	34	40	39
Fe	g/t	69,300	68,000	73,400	68,100	65,500	68,300	68,600	67,200	68,800	69,300	66,918
K	g/t	21,600	28,600	23,600	30,000	22,100	31,800	23,600	31,500	26,300	32,200	28,640
Li	g/t	38	44	40	42	38	45	44	48	42	50	43
Mg	g/t	27,600	30,000	28,500	28,100	26,800	28,700	28,900	28,900	28,000	29,600	28,641
Mn	g/t	1,410	1,150	1,310	1,140	1,590	1,140	1,320	1,230	1,430	1,240	1,316
Mo	g/t	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 5
Na	g/t	12,800	9,840	13,700	11,300	12,700	10,400	12,700	10,100	13,000	10,900	10,758
Ni	g/t	49	57	52	55	48	54	51	53	47	55	52
P	g/t	805	946	838	985	804	915	829	957	784	1,010	989
Pb	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 10
Sb	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Se	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	<30
Sn	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
Sr	g/t	93.6	153	95.5	96.6	95.1	82.0	86.3	83.3	89.9	87.4	96.7
Ti	g/t	6,350	6,560	6,580	6,670	7,310	8,560	6,190	7,200	6,510	7,370	6,954
Tl	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	< 30	<30
U	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20	<20
V	g/t	130	156	139	140	133	136	134	138	132	142	134
Y	g/t	30.0	36.8	32.0	36.2	31.2	35.4	31.2	35.2	32.0	36.9	34.4
Zn	g/t	121	150	130	157	120	151	125	165	125	167	143
Si	%	25.9	24.4	27.0	25.0	25.9	25.7	27.6	25.8	27.7	25.8	27.2

Bench Scale Flotation Tests

Bench scale flotation work commenced with rougher kinetics tests that examine the effect of primary grind size, pH and reagent scheme on the differential flotation rates of minerals. These were followed by cleaner tests, which determines the effect of regrind size, reagents and cleaner configuration. The metallurgical performance will be confirmed through locked cycle testing.

A total of 73 rougher and cleaner tests were conducted to investigate the effect of flowsheet, reagent scheme, grind size on final copper concentrate grade and recovery.

13.4.4 Bench Scale Rougher Flotation Tests

Approximately thirty-six rougher tests were conducted to investigate the effect of primary grind size, pH, residence time and reagent scheme.

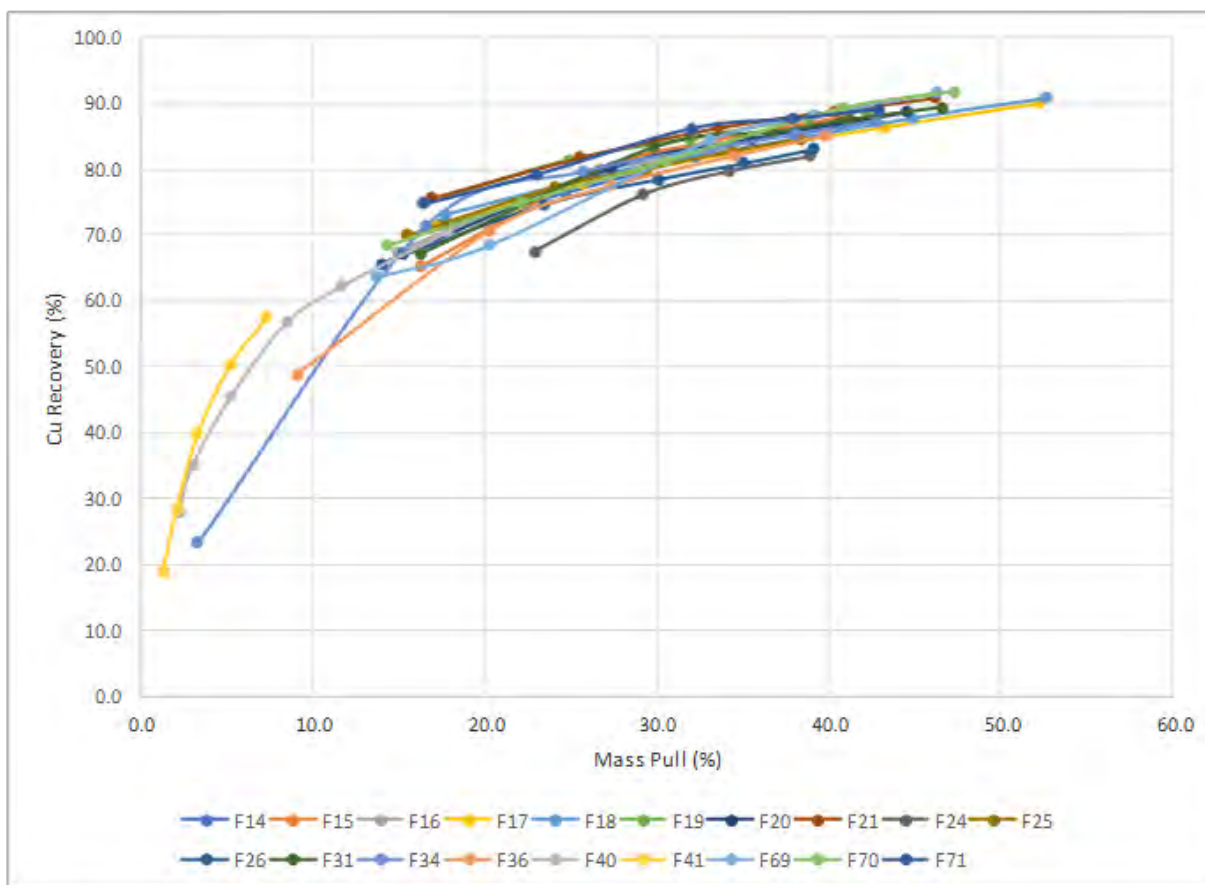
The first set of tests (Tests F1, F3, F5, F6, F7, F9 and F12) was conducted to reproduce and improve the testwork carried out by KCA. Rougher concentrate mass pull ranged from 12.3% to 32.3% and copper recovery ranged from 58.8% to 81.7%.

The second set of tests was conducted to reproduce and improve on the 2013 METCON rougher tests. Test F1 duplicated testwork carried out by KCA. Test F2 was conducted under the 2013 METCON conditions as a baseline for this round of testing. Mass pull and copper recovery for Test F2 is 22.7% and 73.0% Cu respectively. The 2013 METCON rougher tests produced higher average mass pull and copper recovery at 35.2% and 87.4% respectively. A testing matrix summarizing the objective of this set of tests is shown in Table 13.26. The mass pulls and copper recoveries for these tests are plotted in Figure 13.14 at a mass pull of 35%, the copper recovery ranged from 81% to 87%. These tests produced slightly lower copper recovery than the 2013 METCON results for the same mass pull. The first locked cycle test used the F55 flowsheet and reagents but failed to deliver copper targeted copper recovery. Similar to the 2013 METCOM tests, higher mass pull (>40%) was required to achieve the targeted copper recovery.

Modifications were made to the flotation time and reagents dosages, which provided higher mass pull and copper recovery (i.e. a mass pull from 40 to 46 % mass pull and a copper recovery of 91-92%. See results from tests F69 to F71). These tests formed the basis for the LCT 8 which was considered the reference for the lock cycle test campaign.

Table 13.26: Batch Rougher Flotation Test Matrix

Test	Objective	Standard Cell		Primary Grind			pH		Collector g/t		Cell Size at End		HIC		Low	NaHS	
		8L	4L	53	45	36	Nat	9/10	Standard	High	Same	1L	Yes	No		Standard	High
F2	Baseline - Metcon	X		X			X		X		X			X		X	
F10	As F2, finer grind, higher pH	X			X			X	X		X			X		X	
F14	As F10, smaller cell		X		X			X	X		X			X		X	
F15	As F14, natural pH		X		X		X		X		X			X		X	
F16	As F14, higher collector		X		X			X		X	X			X		X	
F17	As F14, small cell last increment		X		X			X	X			X		X		X	
F18	As F17, with HIC		X		X			X	X			X	X			X	
F19	As F10, natural pH	X			X		X		X		X			X		X	
F20	As F10, higher NaHS	X			X		X		X		X			X			X
F21	As F10, longer ret time	X			X			X	X		X			X		X	
F22	As F10, natural pH (same as F19)		X		X			X	X		X			X		X	
F24	As F19, finer grind	X				X	X		X		X			X		X	
F25	As F19, higher collector dosage	X			X		X			X	X			X		X	
F26	As F19, lower NaHS	X			X		X		X		X			X	X		
F29	As F28, pH 10 with soda ash	X			X			X	X		X			X		X	
F30	As F28, pH 10 with lime	X			X			X	X		X			X		X	
F31	As F19, more retention time	X			X		X		X		X			X		X	
F34	As F19, with flash flotation	X				X	X		X		X			X		X	
F36	As F19, different reagents	X			X		X		X		X			X		X	
F40	As F19, shorter increments	X			X			X	X					X		X	
F41	As F34, with flash flotation	X				X	X		X		X			X		X	
F69	A19, lower pH	X			X		X		X					X			X
F70	With copper sulphate	X			X			X	X					X			X
F71	Staged NaHS	X			X			X						X			X

Figure 13.14: Batch Rougher Flotation Tests – Mass Pull vs. Recovery

13.4.5 Bench Scale Cleaner Flotation Tests

Approximately 50 batch cleaner tests were conducted to investigate the effect of regrind size, pH, retention time and reagent scheme.

The tests were conducted to reproduce and improve on the 2013 METCON rougher tests. Test F10 was conducted under the 2013 METCON conditions as a baseline for this round of testing. Third cleaner concentrate grade and copper recovery for Test F10 is 28.2% Cu and 70.2% Cu respectively. The 2013 METCON cleaner tests produced a similar third cleaner concentrate grade of 28.3% Cu, but at a higher copper recovery of 82.4%. A testing matrix summarizing the objective of selected tests of this set of tests is shown in Table 13.27. A list of all tests carried out can be found in the 2018 SGS report. The copper grade recovery for these tests are plotted in Figure 13.15.

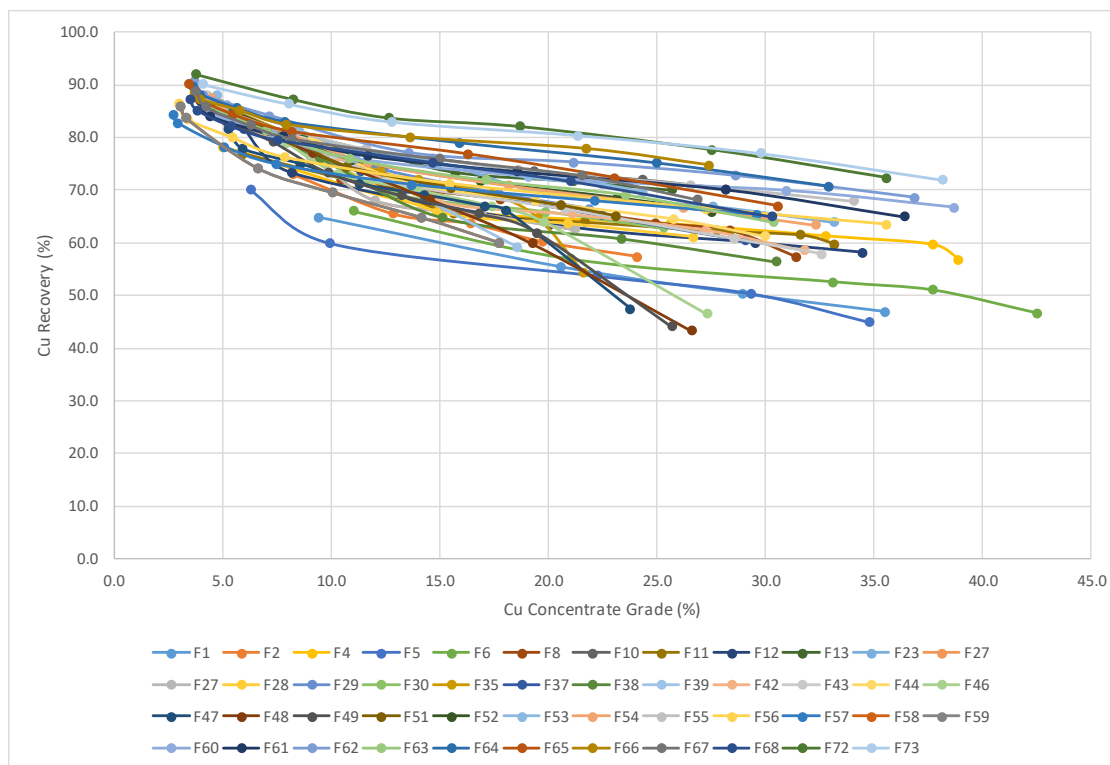
This set of cleaner tests were not able to replicate the 2013 METCON results. For a 28% Cu concentrate grade, the copper recovery ranged from 58% to 70% at cleaner and 88% at rougher.

The initial locked cycle tests (LCT-1 to 3) were carried out using the F55 test flowsheet and conditions. The results were promising. Additional cleaner tests were performed using flowsheet and reagents modifications to improve the first cleaner recovery. Tests F56 to F68 improved the cleaner copper recovery. Locked cycle test LCT-4 to LCT-7 were performed using tests F62 and F64 conditions with some variations. Finally, additional cleaner tests F72 and 73 provided the best recovery results. LCT-8 was developed from rougher kinetic test F71 and F73 cleaner's conditions with some adjustment in cleaner's flotation time.

Figure 13.15: Batch Cleaner Flotation Test Matrix

Test	Purpose	Primary Grind Size (P80) approx. (µm)				Secondary Grind Size (µm)	5100	3418-A	407	CuSO4	PAX	SIBX	W-31/MIBC	A-249	NDM	Fuel Oil	pH		Recovery (%) at 25% Grade
		53	45	36	17												Nat	9/10	
F2	To duplicate testwork carried out by SGS Tucson (formerly Metcon).	x				20						x	x	x	x	x	x		NA
F4	To duplicate testwork carried out on the White Pine deposit.		x			17	x	x					x					x	63.7
F6	Repeat F5, but finer primary grind.			x					x		x							x	55.5
F7	Repeat F3, but higher pH and collector g/t, and add a rougher scavenger.		x			24		x			x						x		46.1
F8	Repeat of F4, but higher reagent dosage and pH.		x			17	x	x					x					x	63.8
F10	Repeat F2 (Metcon), but finer primary grind and higher pH in all stages.		x			17		x				x	x	x	x	x		x	71.7
F11	Same as F8, but higher impeller speed and a Ro Scav 4 stage added.		x			18	x	x					x					x	63.0
F12	Same as F6, but smaller rougher cell and higher impeller speed, additional rougher time and scavenger time, finer grind.				x				x				x					x	61.6
F13	Screen Ro and Ro Scav. Conc. at 500 mesh, regrind oversize.	x				20						x	x	x	x	x	x		70.4
F22	Repeat F10, but finer regrind natural pH at rougher and longer Ro Scav, higher rpm.		x			17						x	x	x	x	x		x	64.5
F23	Repeat F22, but finer primary grind.			x		17						x	x	x	x	x		x	67.8
F27	F19 rougher conditions targeting 40% mass pull, F10 cleaner conditions targeting approx. 20 microns regrind P80.		x			23						x	x	x	x	x		x	NA
F28	F19 rougher conditions targeting 40% mass pull, F10 cleaner conditions targeting approx. 15 microns regrind P80.		x			19						x	x	x	x	x		x	61.8
F29	Same as F28, but pH 10.0 using soda ash in rougher.		x			18						x	x	x	x	x		x	NA
F30	Same as F29, but pH 10.0 using lime in rougher.		x			18						x	x	x	x	x		x	63.0
F32	F19 rougher conditions, with split flowsheet.		x			16						x	x	x	x	x		x	NA
F33	As F32, but finer regrind.		x			16						x	x	x	x	x		x	NA
F35	Repeat F32, lower residence time for Ro Con 1, longer residence time for Ro Con 2-5.	x				16						x	x	x	x	x	x		NA
F37	Repeat F28 with increase in 1st cleaner reagent and residence time to reduce Cu in 1 st cleaner tailing.	x				12						x	x	x	x	x	x		63.4
F38	Repeat F28 with MIBC as frother in cleaning stages and no fuel oil.	x				15						x	x	x	x		x		59.8
F39	As per F28 to generate 1st cleaner kinetic concentrates for potential mineralogical analysis.	x				13						x	x	x	x		x		NA
F42	Repeat F22 and add more 1st cleaner retention time to reduce cleaner tails losses. Note the use of a 4 litre cell in the roughers.		x			18						x	x	x	x			x	63.6
F43	1 x 1 kg charge of minus 10 mesh Master Composite.			x		19						x	x	x	x	x		x	63.6
F44	Repeat F43, larger cell size at rougher.			x		19						x	x	x	x	x		x	64.9
F45	Flash flotation Kinetics using F34 conditions.			x								x	x	x	x		x		NA
F46	Repeat F35, lower residence time for Ro Con 1.			x		23						x	x	x	x	x	x		51.9
F47	Repeat F42, flash float for 20s, RO Con #1 for 3 mins.		x			23						x	x	x	x	x	x		32.4
F48	As F39R, increased SS/CMC in 1 st Cleaner.		x			20						x	x	x	x	x	x		46.9

Test	Purpose	Primary Grind Size (P80) approx. (µm)				Secondary Grind Size (µm)	5100	3418-A	407	CuSO4	PAX	SIBX	W-31/MIBC	A-249	NDM	Fuel Oil	pH		Recovery (%) at 25% Grade
		53	45	36	17												Nat	9/10	
F49	Repeat F47, stage grind for flash float for 20s, Ro Con #1 for 3 mins, standard regrind <20 microns.		x			21						x	x	x	x	x	x		46.1
F50	Repeat F49, but finer regrind.		x			14						x	x	x	x	x	x		46.8
F51	Repeat F42 but using 8 l cell in roughers and higher CMC/SS dosages throughout.		x			21						x	x	x	x	x		x	NA
F52	Repeat F51, but finer regrind.		x			18						x	x	x	x	x	x		67.4
F53	Split F/S with short Ro1 (1 min with no Cleaner), using F52 conditions in cleaners and with finer regrind.		x			11						x	x	x	x	x	x		NA
F54	As F53, but no Ro 1.		x			12						x	x	x	x	x	x		67.4
F55	As F54, but natural pH throughout.		x			14						x	x	x	x	x	x		71.6
F56	New Composite (MC-2), repeat F55.			x		11						x	x	x	x	x	x		67.4
F57	Repeat F56, but longer rougher times and soda ash to cleaners to reach pH 10.5.			x		11						x	x	x	x	x		x	66.9
F58	F57 Conditions, F12 F/S and Primary Grind.				x							x	x	x	x	x		x	NA
F59	As F58, but excessive soda ash additions (similar to KCA).				x							x	x	x	x	x		x	NA
F60	As F55, but new composite.			x		11						x	x	x	x	x	x		71.2
F61	Increased rougher time and using Metcon CSB2 NaHS dosages and no fuel oil.			x		11						x	x	x	x		x		71.3
F62	As F61, but longer cleaner retention times.			x		x						x	x	x	x		x		74.0
F63	As F60, but different regrind mill and media.			x		13						x	x	x	x	x	x		68.0
F64	As F62, but longer rougher, cleaner retention times.		x			11						x	x	x	x		x		75.1
F65	As F64, but no NDM (and replace with higher SIBX and 249).		x			13						x	x	x			x		70.9
F66	As F64, but higher NaHS.			x		12						x	x	x	x		x		76.1
F67	As F64, but coarser regrind.		x			20 to 25						x	x	x	x		x		69.8
F68	Repeat F64 Without SS/CMC in Rougher stages.		x			15						x	x	x	x		x		68.8
F72	Batch Cleaner test using F71 rougher conditions and CuSO4 as well.		x			25				x		x	x	x	x		x		78.9
F73	Batch Cleaner test using staged NaHS additions at the roughers.		x			26						x	x	x	x		x		78.9

Table 13.27: Batch Cleaner Flotation Tests – Copper Recovery vs. Concentrate Grade

13.4.6 Flash Flotation Tests

Several flotation tests were carried out to investigate the potential of adding a flash flotation circuit to the flowsheet to increase copper recovery. The current flash flotation testwork shows 28-30% recovery at 1-minute residence time. It is difficult to simulate the flash flotation in laboratory scale in terms of particle size (cyclone underflow) and pulp densities, it is therefore proposed that flash flotation be further evaluated in the next phase of the Project through pilot plant scale testing.

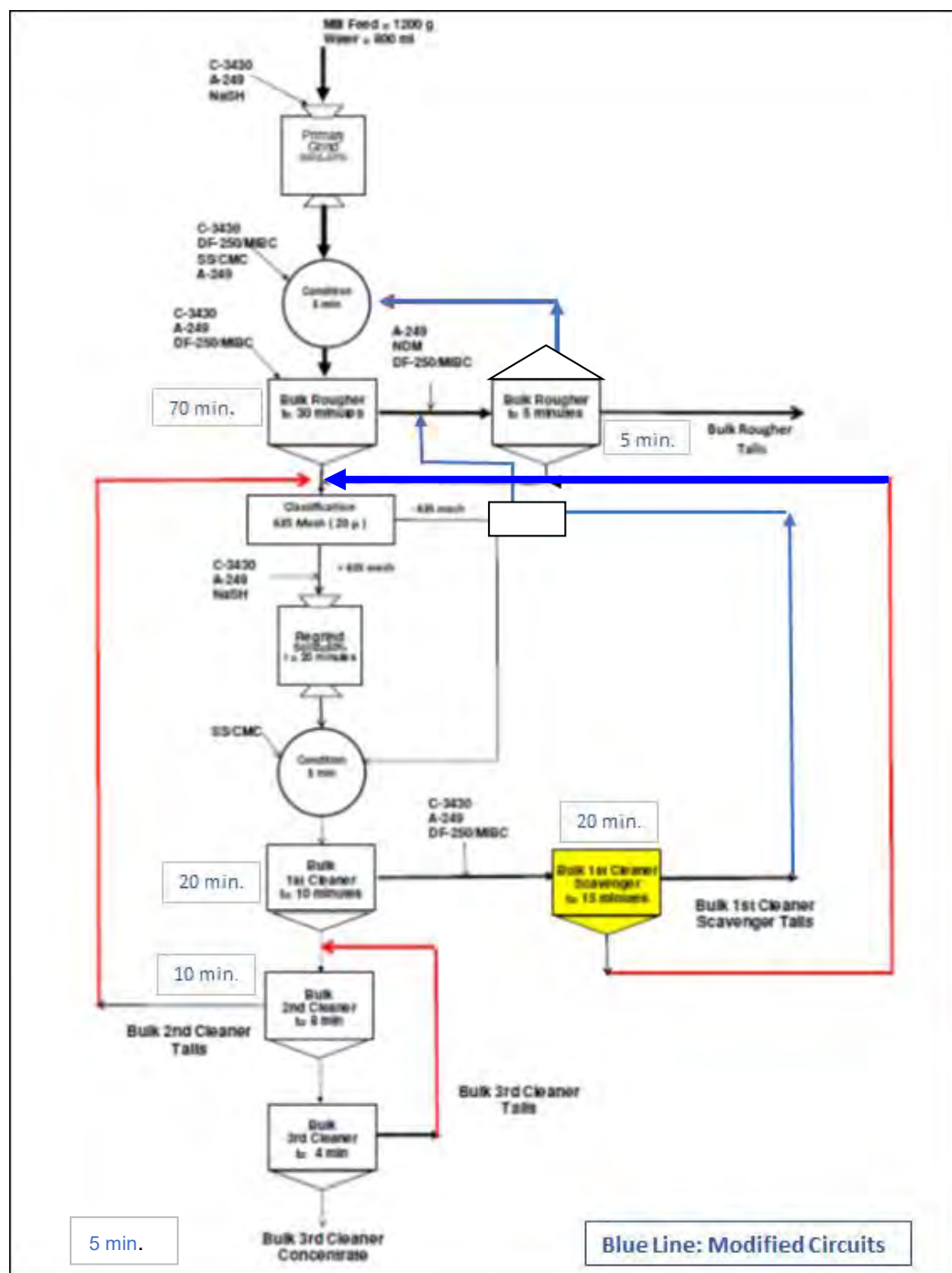
For additional detailed information, refer to document “*Memorandum: Flash Flotation for Copperwood Project, USCW-A-LYC-PR-600-MEM-0001*” prepared by Lycopodium Minerals Canada Inc., dated November 22, 2017.

13.4.7 Locked Cycle Flotation Tests

Locked cycle flotation tests which were in progress at the time of the finalization of the process design for the Study to be reviewed and compared with the process design criteria used as summarized in Table 13.33. The LCT results will be used as confirmation of the Study's process design.

Eight locked cycle tests combined with optimization batch tests were carried out to develop the selected flowsheet for the variability testwork program. The final proposed flowsheet is different from the design flowsheet which might require adjustment of the process plant configuration at the next phase of the Project. The main differences are in the flotation time. Flotation kinetic is much slower than the test performed in 2012 (Table 13.32). A rougher scavenger stage has been introduced for first cleaner tailings recirculation. This change closes the circuit with only one combined final tailings. This configuration combined with the actual process plant design will provide both options, closed or open first cleaner circuit with minimum impact. Another major change concerns the cleaner scavenger concentrate, which now recirculates to regrind and not rougher feed. The reagents type remains the same, but consumptions change. LCT-8 reagents addition has been used for the processing cost evaluation.

Figure 13.16 shows the modified block flow diagram used for locked cycle test No.8 (LCT8).

Figure 13.16: Modified Locked Cycle Testwork No.8 Block Flow Diagram

13.4.8 Variability Testwork

Seventeen variability tests were performed to support the copper recovery and grade for the Study using the LCT-8 test flowsheet and conditions to simultaneously understand the ore variability across the

deposit. Ten composites were used for the variability; 10 from the Main Zone which represents the major part of the Copperwood resource, four from Zone 6 and three from Zone 5. Table 13.28 shows the overall locked cycle testwork results.

Table 13.28: Overall Locked Cycle Testwork Results

LCT No.	Hole No.	Zone	Head Grade		Concentrate		Cu (%)	Ag (gt)
			Cu (%)	Ag (g/t)	Cu (%)	Ag (g/t)		
8	Grind Composite	Main Zone	2.14	6.4	23.5	68.0	83.9	81.1
9	CW-17-201	Main Zone	1.79	5.10	21.3	59.4	89.1	86.5
10	CW-17-201	Main Zone	1.90	5.40	25.2	69.0	89.7	86.2
11	CW-17-179A	Main Zone	2.29	6.40	22.7	60.0	89.5	85.7
12	CW-17-185	Main Zone	1.88	4.40	23.2	51.1	88.9	83.5
13	CW-17-171A	Main Zone	2.38	6.33	27.4	64.1	85.3	75.1
14	CW-17-189-194	Zone 6	1.00	1.30	22.5	22.0	86.9	64.1
15	CW-17-169-196	Zone 6	1.24	1.40	22.0	18.0	88.1	63.2
16	CW-17-163-166	Zone 5	1.29	3.10	23.8	42.4	84.5	62.0
17	CW-17-165A	Main Zone	2.26	4.20	29.2	47.8	77.1	68.3
18	CW-17-187	Main Zone	1.24	3.90	19.6	48.3	85.5	66.4
19	CW-17-186	Main Zone	1.93	1.80	27.8	22.3	79.6	68.5
20	CW-17-167A	Main Zone	2.45	8.80	26.2	90.6	85.7	82.7
21	CW-17-200	Main Zone	2.46	6.90	24.8	65.8	89.4	84.2
22	CW-17-172A	Main Zone	2.54	9.00	27.6	97.4	87.2	86.7
23	CW-17-188-195	Zone 6	1.17	2.20	24.9	37.0	88.1	68.1
24	CW-17-191-197	Zone 6	1.15	1.10	21.9	15.0	86.3	58.4
25	CW-17-181-183	Zone 5	1.36	1.70	27.0	29.0	83.4	71.6
26	CW-17-178-180	Zone 5	.14	1.80	23.1	31.0	87.4	73.5
Average			1.74	4.09	24.5	47.4	86.0	73.4

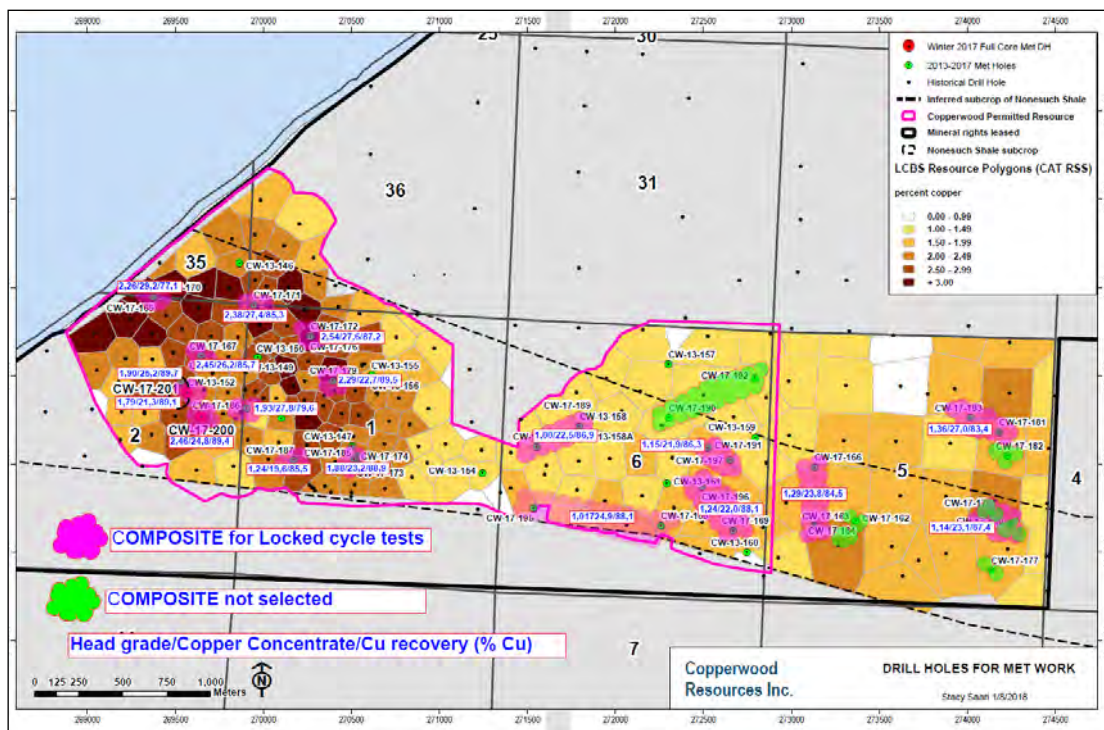
Table 13.29 illustrates the locked cycle testwork results by zone.

Table 13.29: Locked Cycle Testwork Results by Zone

LCT No.	Hole No.	Zone	Head Grade % g/t		Concentrate %		% Recovery	
			Cu	Ag	Cu	Ag	Cu	Ag
8	Grind composite	Main Zone	2.14	6.4	23.5	68	83.9	81.1
9	CW-17-201	Main Zone	1.79	5.10	21.3	59.4	89.1	86.5
10	CW-17-201	Main Zone	1.90	5.40	25.2	69.0	89.7	86.2
11	CW-17-179A	Main Zone	2.29	6.40	22.7	60.0	89.5	85.7
12	CW-17-185	Main Zone	1.88	4.40	23.2	51.1	88.9	83.5
13	CW-17-171A	Main Zone	2.38	6.33	27.4	64.1	85.3	75.1
17	CW-17-165A	Main Zone	2.26	4.20	29.2	47.8	77.1	68.3
18	CW-17-187	Main Zone	1.24	3.9	19.6	48.3	85.5	66.4
19	CW-17-186	Main Zone	1.93	1.8	27.8	22.3	79.6	68.5
20	CW-17-167A	Main Zone	2.45	8.8	24.8	90.6	85.7	82.7
21	CW-17-200	Main Zone	2.46	6.9	27.6	65.8	89.4	84.2
22	CW-17-172A	Main Zone	2.54	9	25	97.4	87.2	86.7
Average	Zone	Main Zone	2.13	5.70	25.06	61.16	85.76	78.75
14	CW-17-189-194	6	1.00	1.30	22.5	22.0	86.9	64.1
15	CW-17-169-196	6	1.24	1.40	22	18.0	88.1	63.2
23	CW-17-188-195	6	1.17	2.20	24.9	37.0	88.1	68.1
24	CW-17-191-197	6	1.15	1.10	21.9	15.0	86.3	58.4
Average	Zone	6	1.14	1.50	22.83	23.00	87.35	63.45
16	CW-17-163-166	5	1.29	3.10	23.8	42.4	84.5	62.0
25	CW-17-181-183	5	1.36	1.70	27.0	29.0	83.4	71.6
26	CW-17-178-180	5	1.14	1.80	23.1	31.0	87.4	73.5
Average	Zone	5	1.26	2.20	24.63	34.13	85.10	69.03

The details of the locked cycle testwork results are illustrated on the Copperwood drill hole map in Figure 13.17.

Figure 13.17: Locked Cycle Testwork Results over the Copperwood Deposit



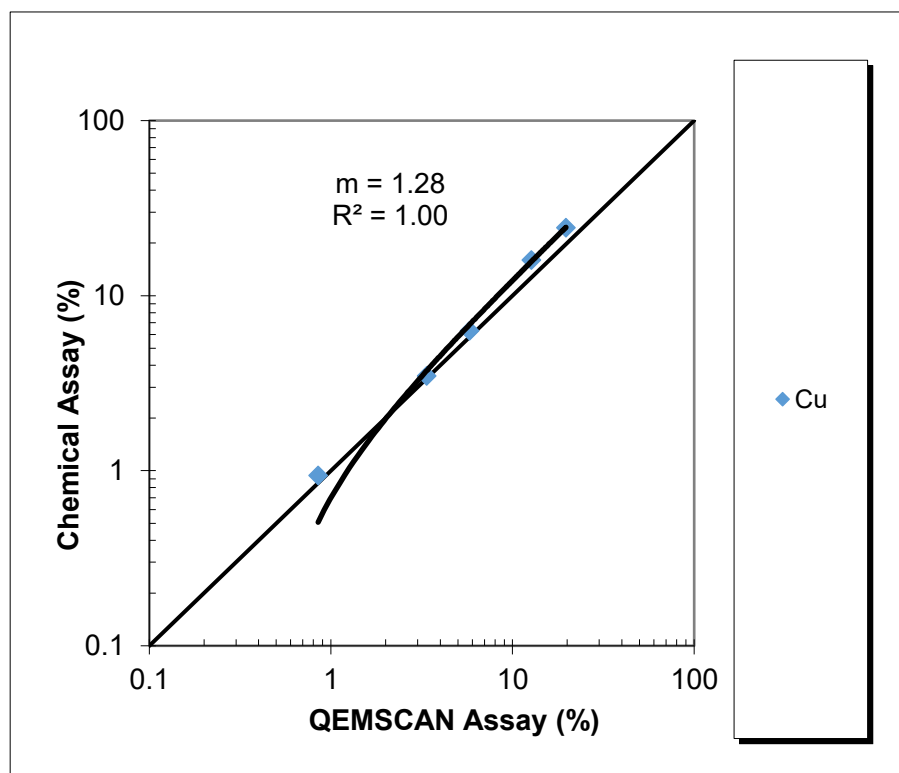
13.4.9 MLA Analysis

Concentrate and tails produced in Test F39R were sent to undergo mineral liberation analysis (MLA). Copper, molybdenite and overall mineralogy were determined from scanning electron microscope / electron dispersive spectroscopy (“SEM/EDX”) data and MLA.

13.4.10 QEMSCAN Assay Reconciliation

The QEMSCAN mineralogical assays were regressed with the chemical assays and are shown in Figure 13.18.

The QEMSCAN calculated assays present good correlation with chemical assays with overall correlation, as measured by the R-squared criteria of 1.0.

Figure 13.18: QEMSCAN Assay Reconciliation**13.4.11 Modal Mineralogy**

Cleaner concentrates 1. 2. 3., cleaner scavenger concentrate and cleaner scavenger tails from test F39R were sent for modal mineralogy. The modal mineral concentrations for each stream are shown in Table 13.30.

Fine copper sulphide/silicate was the primary mineral and main copper mineral in the cleaner concentrate samples. Other copper-bearing minerals include chalcocite, bornite, covellite, chalcopyrite, tetrahedrite and chrysocolla. The main gangue minerals were chlorite and quartz in the samples.

The mean grain size of each mineral in the samples are shown in Table 13.31.

Table 13.30: Model Mineral Concentrations for Test F39R

Sample		F39R CI Con 1	F39R CI Con 2	F39R CI Con 3	F39R CI Scav Con	F39R CI Scav Tails
Mineral Mass (%)	Chalcocite	12.3	6.32	2.15	1.05	0.20
	Fine Cu-Sulph/Sil	44.4	36.1	20.4	12.8	3.53
	Bornite	3.58	2.48	1.23	0.74	0.19
	Covellite	0.01	0.02	0.01	0.01	0.00
	Chalcopyrite	0.36	0.25	0.11	0.04	0.01
	Tetrahedrite	0.09	0.06	0.02	0.01	0.01
	Chrysocolla	0.18	0.20	0.09	0.06	0.01
	Other Sulphides	0.34	0.17	0.09	0.02	0.02
	Fe-Oxides	3.70	3.30	2.55	1.65	0.96
	Fe-Ox/CC	0.65	0.54	0.25	0.16	0.04
	Titanite/sphene	0.70	0.88	1.00	0.91	1.11
	Other Oxides	0.29	0.44	0.45	0.40	0.44
	Calcite	0.91	5.29	8.22	9.49	0.96
	Other Carbonates	0.01	0.00	0.01	0.01	0.00
	Quartz	12.2	14.9	17.0	16.6	23.9
	K-Feldspar	1.93	2.36	2.92	3.31	5.01
	Plagioclase	3.34	3.67	4.65	4.63	6.11
	Micas	1.16	1.36	1.71	1.68	3.07
	Clays	1.27	1.56	2.04	2.18	2.83
	Chlorite	11.9	18.7	32.6	41.4	50.1
	Amphibole / Pyroxene	0.31	0.98	1.82	2.20	0.69
	Other Silicates	0.05	0.04	0.05	0.05	0.04
	Apatite	0.32	0.38	0.60	0.56	0.69
	Other	0.01	0.02	0.01	0.02	0.03
	Total	100.0	100.0	100.0	100.0	100.0

Table 13.31: Mean Grain Size by Frequency (micron) for Test F39R

Sample		F39R CI Con 1	F39R CI Con 2	F39R CI Con 3	F39R CI Scav Con	F39R CI Scav Tails
Calculated ESD Particle Size		17	17	20	21	18
Mean Grain Size by Frequency (µm)	Chalcocite	15	14	12	11	9
	Fine Cu-Sulph / Sil	13	13	13	13	13
	Bornite	11	10	10	10	10
	Covellite	11	11	13	6	0
	Chalcopyrite	8	7	8	7	7
	Tetrahedrite	10	9	8	9	18
	Chrysocolla	7	7	7	7	8
	Other Sulphides	10	7	7	6	7
	Fe-Oxides	11	11	11	10	11
	Fe-Ox/CC	7	7	7	7	7
	Titanite/sphene	9	9	10	9	10
	Other Oxides	8	8	7	7	7
	Calcite	12	12	13	12	11
	Other Carbonates	6	6	6	6	6
	Quartz	13	12	12	12	12
	K-Feldspar	11	11	10	11	11
	Plagioclase	14	12	13	13	13
	Micas	8	8	8	8	8
	Clays	9	9	9	9	10
	Chlorite	11	12	13	15	15
	Amphibole/Pyroxene	8	9	9	9	9
	Other Silicates	7	6	6	6	7
	Apatite	9	8	9	9	9
	Other	7	7	7	7	6

13.4.12 Copper Sulphides and Silicate Association

The copper sulphides and silicates association for the samples are shown in Figure 13.19 and Figure 13.20, respectively.

In the final cleaner concentrate (cleaner concentrate 3). Copper is predominately carried out in the silicates and complex minerals while the copper is predominately associated with the silicates minerals in the tails.

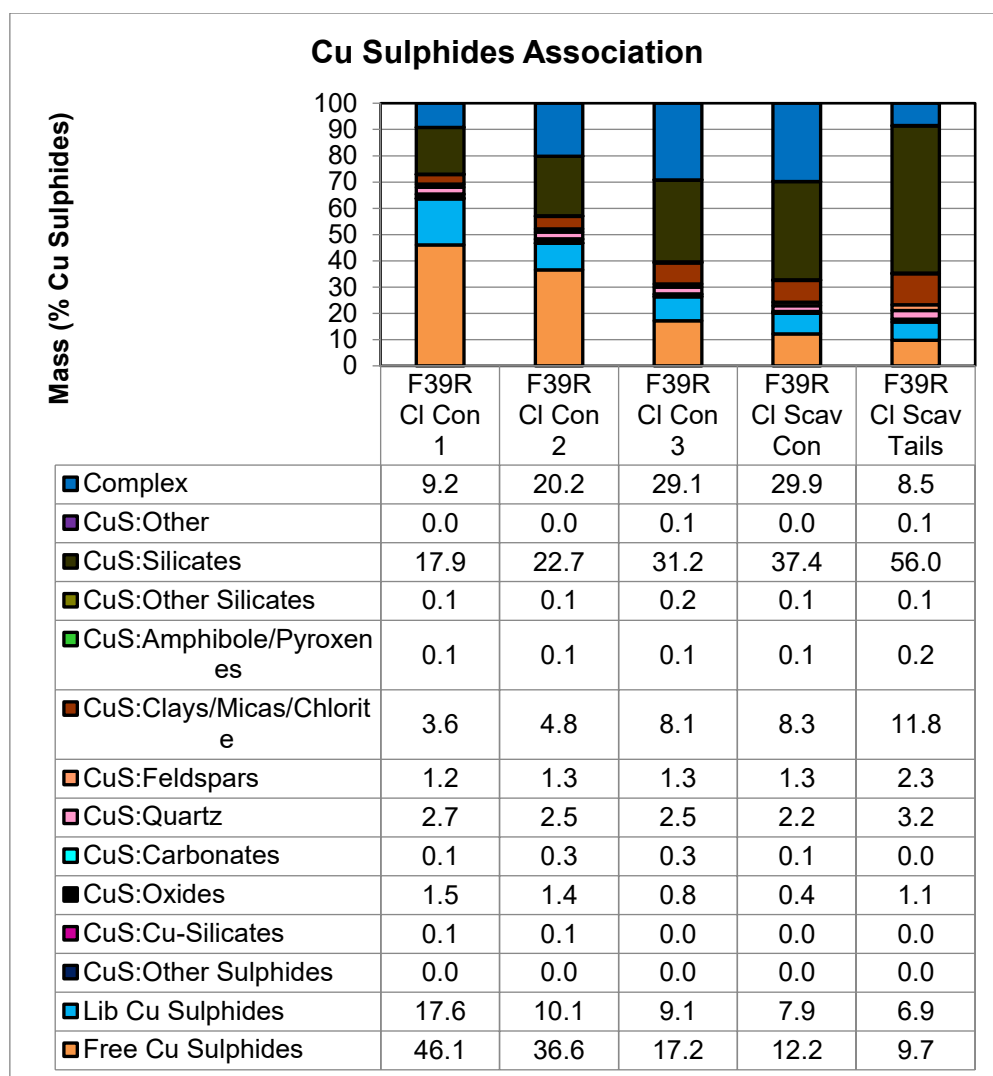
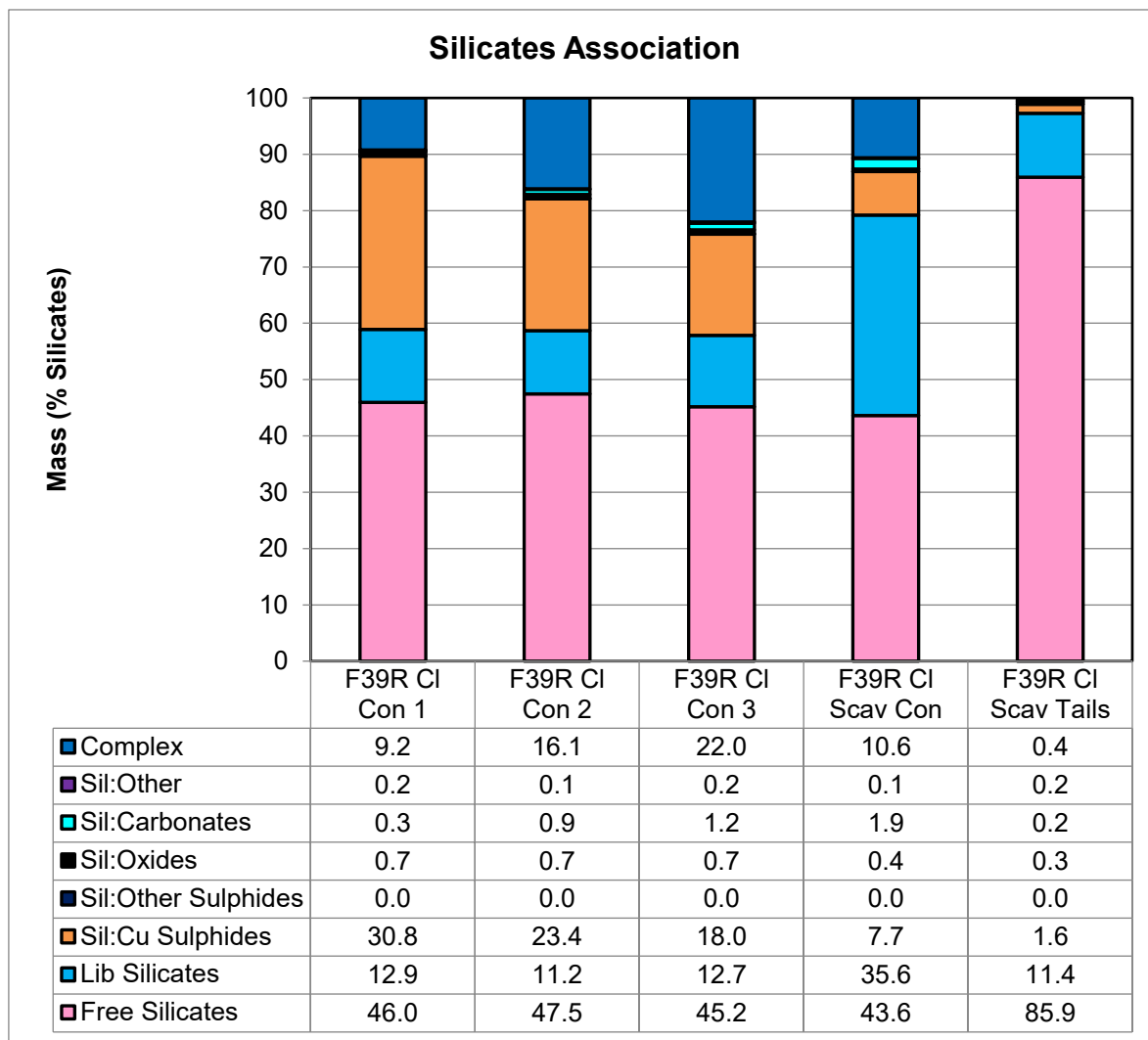
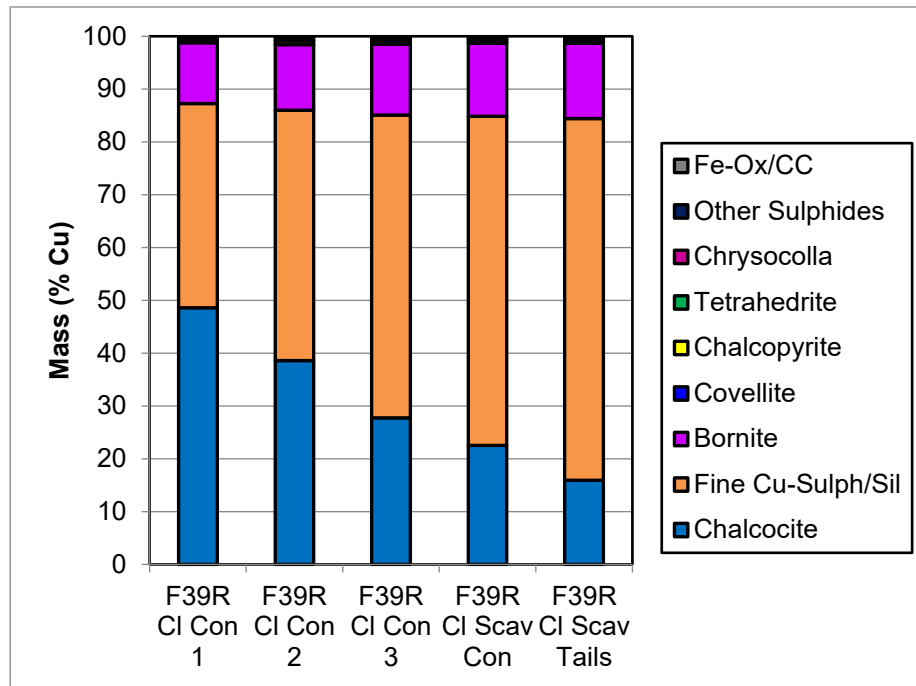
Figure 13.19: Copper Sulphides Association

Figure 13.20: Silicates Associations**13.4.13 Copper Deportment**

The copper deportment data for the five samples are illustrated in Figure 13.21 and tabulated in Table 13.32. Copper is primarily carried in the chalcocite and fine Cu-Sulphides/Silicates mineral for the concentrates. Copper is predominately carried in the fine Cu-Sulphides/Silicates mineral for the tail sample.

Figure 13.21: Elemental Department of Copper**Table 13.32: Elemental Department of Copper**

Mineral Name	F39R CI Con 1	F39R CI Con 2	F39R CI Con 3	F39R CI Scav Con	F39R CI Scav Tails
Chalcocite	48.6	38.6	27.7	22.5	15.9
Fine Cu-Sulph/Sil	38.7	47.4	57.3	62.3	68.5
Bornite	11.5	12.4	13.5	13.8	14.3
Covellite	0.03	0.10	0.11	0.11	0.00
Chalcopyrite	0.63	0.68	0.67	0.45	0.37
Tetrahedrite	0.16	0.16	0.10	0.13	0.24
Chrysocolla	0.31	0.54	0.54	0.57	0.56
Other Sulphides	0.05	0.11	0.05	0.03	0.11
Fe-Ox/CC	0.02	0.02	0.02	0.02	0.03
Total	100.0	100.0	100.0	100.0	100.0

Further mineralogical analysis of 1st cleaner concentrate (F39R) showed that fine chalcocite particles (-10 microns) cannot be floated efficiently compared to coarse particles. The recovery of coarse particles is approximately 3 times greater than the recovery of fine particles (Figure 13.22).

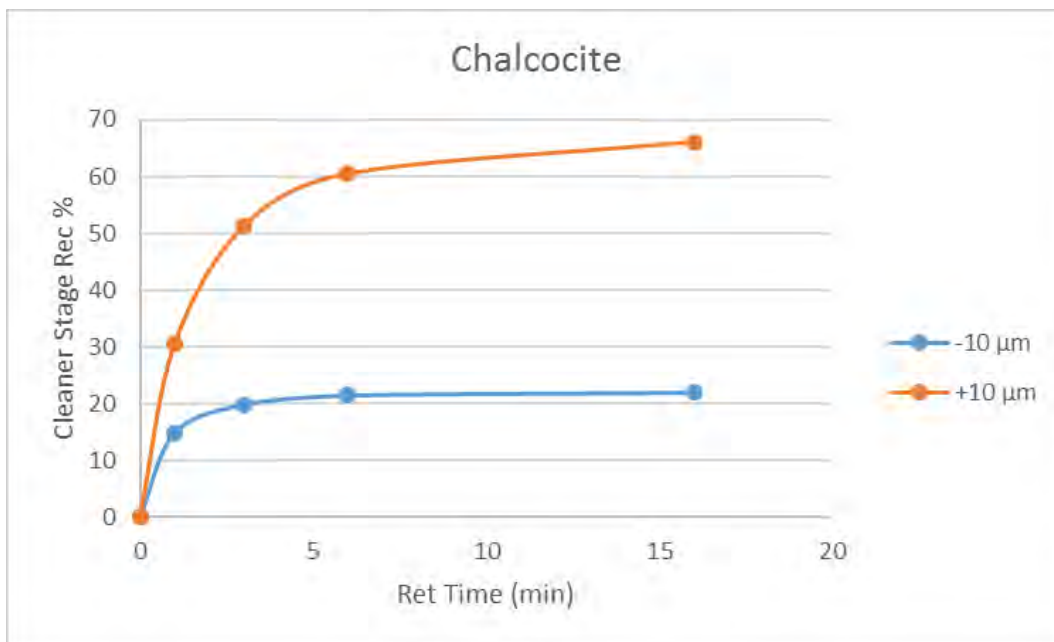
Figure 13.22: Chalcocite Recover vs. Particle Size**13.4.14 Locked Cycle Concentrate Specifications**

Table 13.33 shows full chemical analyses of the locked cycle testwork.

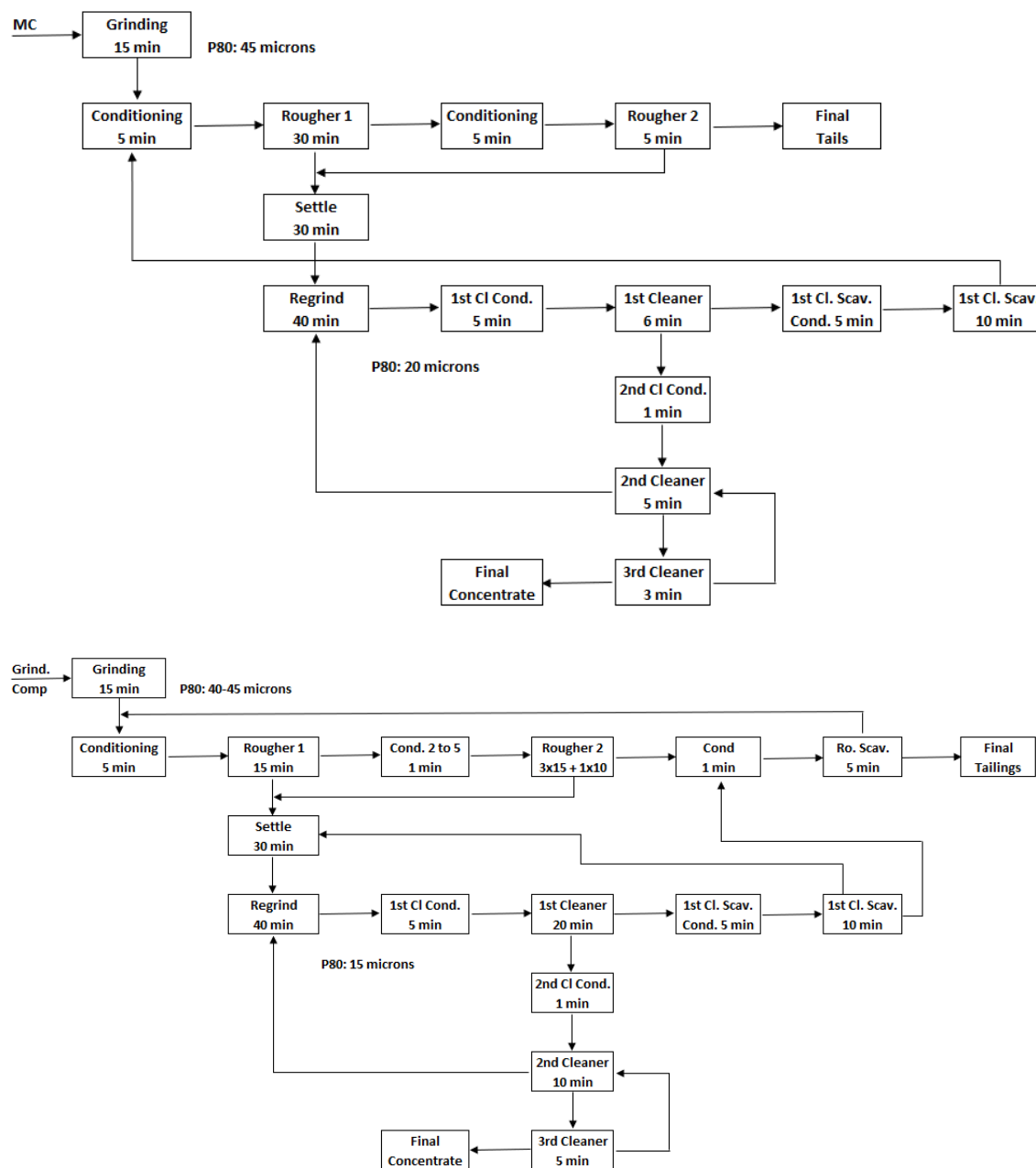
Table 13.33: Copper Concentrate Specification

Locked Cycle Test Zone	LCT-10 3 rd Cleaner Conc F	LCT-9 3 rd Cleaner Conc F	LCT-12 3 rd CI Maine Zone	LCT-13 3 rd CI Main Zone	LCT-23-24 Comb 3 rd CI Zone 6	LCT-25-26 Com 3 rd CI Conc F Zone 5
Cu %	24.7	19.7	22.8	28.1	22	24.5
Fe %	10.2	9.79	9.22	9.93	9.89	7.87
As g/t	-	-	< 0.001	0.001	-	-
C(t) %	0.78	0.81	0.71	0.65	1.04	0.87
S %	9.99	5.45	6.4	7.71	6.68	7.35
S= %	6.46	5.22	6.09	7.32	6.21	6.91
Au g/t	0.35	0.14	0.13	0.22	0.11	0.16
Pt g/t	0.02	0.02	0.09	0.1	0.06	0.14
Pd g/t	0.03	0.02	0.24	0.07	0.05	0.04
Ag g/t	67.4	53.4	44.7	66.5	27.3	29.3
Hg g/t	0.3	< 0.3	0.4	0.4	0.8	0.5
Cl g/t	90	90	60	300	-	-
F %	0.042	0.046	0.038	0.04	0.043	0.04
SiO ₂ %	34.8	40.2	38.6	32.6	36.2	35.9
Al ₂ O ₃ %	8.30	9.34	8.81	7.93	9.07	8.63
Fe ₂ O ₃ %	14.4	13.7	13.2	14.2	14.2	11.3
MgO %	2.85	3.11	3.06	2.76	3.51	3.37
CaO %	0.59	0.70	0.85	0.68	0.63	0.6
K ₂ O %	1.83	2.11	2.01	1.75	2.16	1.83
TiO ₂ %	0.88	0.95	0.95	0.92	0.99	1.04
MnO %	0.11	0.11	0.12	0.11	0.14	0.15
Cr ₂ O ₃ %	0.043	0.069	0.1	0.082	0.14	0.18
V ₂ O ₅ %	0.025	0.023	0.021	0.021	0.024	0.022
As g/t	< 30	< 30	-	-	< 30	< 30
Ba g/t	174	201	207	172	211	190
Be g/t	1.46	1.62	1.46	1.38	1.73	1.57
Bi g/t	55	< 20	< 30	< 30	< 30	< 30
Cd g/t	< 2	< 2	< 2	< 2	< 2	< 2
Co g/t	25	26	29	27	30	33
Li g/t	24	28	25	21	38	43
Mo g/t	< 20	< 20	< 30	< 30	< 30	< 30
Na g/t	6530	7690	7420	5770	6370	7170
Ni g/t	51	77	143	114	160	224
P g/t	666	728	558	628	640	647
Pb g/t	< 20	< 20	< 30	< 30	< 30	< 30
Sb g/t	< 30	< 30	< 10	< 10	< 10	< 10
Se g/t	< 30	< 30	< 30	< 30	< 30	< 30
Sn g/t	< 20	< 20	< 20	< 20	< 20	< 20
Sr g/t	46.2	53.1	49.2	41.9	50.2	49
Tl g/t	< 30	< 30	< 30	< 30	< 30	< 30
U g/t	< 20	< 20	< 20	< 20	< 20	< 20
Y g/t	23.3	24.8	23.7	23.9	24.9	24.5
Te g/t	<4	< 4	< 4	< 4	< 4	< 4
Zn g/t	2940	2330	99	102	110	144
Total (%)	96.88	97.22	98.23	97.91	97.09	96.17

13.4.15 Testwork Discussion and Recommendations

Comprehensive testwork programs have been carried out on Copperwood ores over the years with variable results. During the last testwork program in 2017 and 2018, the main objective was to evaluate the process performance selected in the FS 2012 to improve the performance and verify the variability of the ore over the deposit. Alternative reagents were examined but finally the reagents used in the METCON testwork appeared to deliver better performance for the samples processed. However, modification to the process flowsheet. grind size target combined with modified reagents additions and dosage delivered better performance. The major modifications consisted of finer primary grind (40 microns), finer regrind (15 microns), re-circulation of the first cleaner scavenger concentrate to regrind and recirculation of the first cleaner tailings to rougher scavenger. The flotation time for most circuits increased which will require further investigation in a next testwork program campaign. Closing the first cleaner circuit with recirculation of the first cleaner scavenger concentrate to regrind with the same conditions appeared to increase the copper recovery by 3%. Figure 13.23 illustrates the testwork block flow diagram used for this Study and LCT8 block flow diagram for comparison.

Figure 13.23: Flowsheet Comparison between Design and Final Located Cycle Test



The primary observation of variability testwork showed that the copper recovery varies from 77% up to ~90% with a concentrate grade from 20% up to 29% Cu. The overall average Cu recovery was at 86% with an average Cu concentrate grade of 24.5%. However, long flotation time combined with fine grind required some particular procedures during the locked cycle flotation test to complete the test in the same day (critical in chalcocite flotation). The settling process required prior to the regrind stage created fine particles

(slimes). Approximately 5 to 10% of the material removed with an average of 1 to 2% of the Cu content. This material was not recirculated and was not put into account. Depending on where this copper will report, it might affect the overall recovery in proportion, positively or negatively.

The copper recovery might further be optimized by concentrate grade and reagents optimization. Review of the past testwork revealed that there is a correlation between the location of samples and the metallurgical results.

Additional characterization might be done specifically in the area where the metallurgical results were lower than most of the other drill holes (i.e. CW-17-165 and to some extent CW-17-186).

In the next set of testwork it will be appropriate to verify the impact of the desliming on the copper concentrate grade and recovery. In case of negative impacts of slimes, it might be worthwhile to introduce a desliming stage in the process plant design.

Considering the challenge of the processing of Copperwood ore and the fact that ore will be available a long time before plant start-up, it might be a real advantage to proceed with pilot plant campaign to validate and optimize the process flowsheet, retention time, reagents type and addition points.

13.5 Key Process Design Criteria

The key process design criteria listed in Table 13.34 form the basis of the detailed process design criteria and mechanical equipment list. The design criteria selected based on the best information available at the time of completion of the Study and will have to be adjusted at detailed engineering based on the final testwork results. Confirmatory metallurgical testwork may result in minimizing the impact of latest locked cycle testing; otherwise, the flotation circuit will need to be modified accordingly.

Table 13.34: Key Process Design Criteria

Parameter	Units	Value	Value LCT-8	Source
Plant Throughput	mtpd	6,600	-	Highland
Head Grade - LoM	% Cu	1.35		Highland
	g/t Ag	3.41		Highland
Plant Availability	%	91.3		Lycopodium
Bond Crusher Work Index (CWi)	kWh/t	20.3		Consultant
Bond Ball Mill Work Index (BWi)	kWh/t	16.2		Testwork
SMC Axb ¹		34.5		Consultant
Bond Abrasion Index (Ai)	g	0.014		Testwork
Grind Size (P ₈₀)	µm	45	40-45	Testwork
Rougher Residence Time – Lab	min	50	75	Testwork
Cleaner 1 Residence Time – Lab	min	6	20	Testwork
Cleaner 1st Scavenger Residence Time – Lab	min	10	20	Testwork
Cleaner 2 Residence Time – Lab	min	5	10	Testwork
Cleaner 3 Residence Time – Lab	min	3	5	Testwork
Regrind Mill Product Size (P ₈₀)	µm	20	15	Testwork
Concentrate Production Rate	t/h	15.1		Calc
Concentrate Thickener Solids Loading	t/m ² .h	0.20		Lycopodium
Filter Solids Loading	kg/m ² .h	160		Lycopodium

2. Design A x b value derived from the 85th percentile ranking of specific energies determined for each individual ore type.

14. MINERAL RESOURCE ESTIMATE

GMSI prepared a Mineral Resource estimate for the Copperwood Project based on data provided up to and including April 12th, 2018. Resource estimation methodologies, results and validations are presented in this Section 14 of this Report.

The resource estimate was prepared in accordance with CIM Standards on Mineral Resources and Reserves (adopted May 10, 2014) and is reported in accordance with NI 43-101. Classification, or assigning a level of confidence to Mineral Resources, has been undertaken with strict adherence to CIM Standards on Mineral Resources and Reserves. In the opinion of GMSI, the resource evaluation reported herein is a reasonable representation of the global Mineral Resources found in the Copperwood Project at the current level and spacing of sampling.

The mineral estimate was prepared under the supervision of Mr. Réjean Sirois, Eng. GMSI, Vice President Geology and Resources, an independent “Qualified Person” as defined in NI 43-101. Geovia GEMS™ and Leapfrog Geo™ software was used to facilitate the resource estimation process.

The Mineral Resource estimate includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these Inferred Mineral Resources will be converted to the Indicated and Measured categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

14.1 Data

Raw data incorporated into this Report consist of all diamond drilling data obtained from the Copperwood Project between 1956 and April 12th, 2018. This includes the database used for the October 2017 Mineral Resources, and all additional diamond drilling data collected in 2018 (14 drill holes, of which 9 contain assays). The nine new drill holes with assays from 2018 do not contain silver analyses due to an inconsistent laboratory method being applied at the time. Holes included in the database comprise those from the following series: M56, M57, PC and CW-08 to CW-18. GMSI has reviewed the database to verify the historical resources initially published by Highland and is satisfied that the integrity of the drilling database is of a high standard and can be used for resource estimation.

14.1.1 Drill Hole Spacing

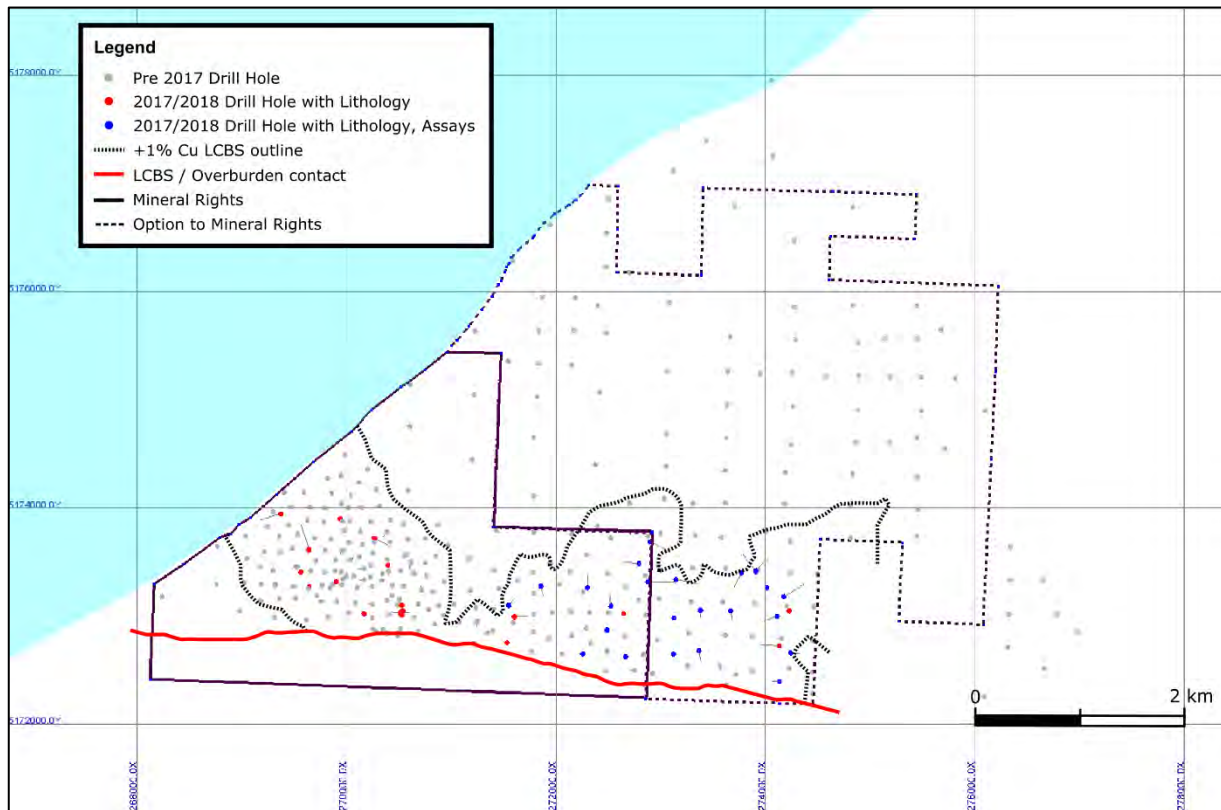
The legacy drill holes from the Copperwood Project were drilled between 1956 and 1959, and between 2008 and 2013 by three different companies. These drill holes are summarised in Table 14.1, and were produced using the drill hole database collar table. The drill hole spacing of the Copperwood Deposit is variable between 100 m to 150 m for the western area and Section 6, and from 150 to 300 m in Section 5. Drilling density in the Satellite Deposits is also irregular, from 300 m to 700 m. The large majority of drill holes are vertical or near-vertical, and increasing length heading northwards depending on the mineralized horizon depth. Figure 14.1 illustrates the grid spacing for the Copperwood Project.

The final drill spacing is judged adequate to develop a reasonable model of the mineralization distribution, and to quantify its volume and quality with a high level of confidence.

Table 14.1: Legacy Drill Holes by Company

Company	Years of Drilling	Drill Hole Series	# Holes	Length (m)
US Metal Refining	1956-1957	M56, M57	161	34,050
Bear Creek Mining	1959	PC	23	3,998
Orvana	2008-2010, 2013	CW-08, CW-09, CW-10, CW-13, BC	146	21,466
Highland	2017, 2018	CW-17, CW-18	48*	10,594
Total			366	70,105

*48 drill holes with an additional 14 wedges

Figure 14.1: Drill Status Plan as of April 12th, 2018

14.1.2 Data Conditioning

GMSI made some adjustments to the database to facilitate surface generation in Leapfrog Geo™ software, where the consistency of logging of the stratigraphic column is integral to produce an accurate geological model.

It was noted that there was often a single sample directly above the LCBS (logged as Red Laminated unit) containing grades greater than 1% Cu. These samples would be excluded from the LCBS in the current state (the samples are around 30 cm in length and are present in 39 historical drill holes). These sample likely reflect a change in logging procedure, as they mostly pertain to drill holes with a prefix CW-09. In addition, the boundary between the Grey Laminated and Red Laminated is transitional, and it not easily distinguished.

GMSI subsequently recoded these samples into the Grey Laminated unit to ensure they were captured in the resource estimate.

In addition, it was noted that the Domino and Red Massive were grouped for laboratory analysis for 42 of the drill holes in the database (yet logged separately in the lithology table). GMSI will include these samples in the compositing process described in Section 14.3.3.

Lastly, minor changes were made to the top of the LCBS in nine drill holes to account for grouped logging codes in historical logging. The new logging code "LTRA" (found at the base of the Domino in the 2017/2018 logging data) was recoded to the Domino (23), as it represents a thin mineralised transition zone between the Domino and the underlying Copper Harbour siltstone/sandstone.

14.2 Modelling Approach

Numerous 2D and 3D modelling elements such as topography, structure and lithology surfaces and/or solids were generated for this resource estimate. The surfaces were created using the 3D geological modelling software Leapfrog Geo™ and then imported into Geovia GEMS™ (version 6.7.4).

GMSI applied the following approach for building the geological block model:

- Model the thrust fault identified in July 2017 to produce two fault blocks within the model;
- Model the individual LCBS units using the lithology codes provided by Highland (Domino, Red Massive and Grey Laminated units);
- Model hanging wall and footwall dilution zones using a 0.3 m "skin" above and below the LCBS, to ensure accurate representation of dilution grades;
- Model the remaining portion of the Red Laminated above the hanging wall dilution zone for use in geotechnical studies;
- Model the UCBS using a 1% Cu cut-off to define a continuous unit, whilst applying a minimum thickness of 2.0 m (considered the minimum mining height at the time of writing). The UCBS is defined geologically as the Upper Transition Shale and the Thinly units which present grades greater than 1% Cu in general.

As the lithology units within the LCBS have a strong control on copper grade, no additional lower grade cut-off was applied during modelling of the LCBS. The constraints applied by modelling each unit are considered sufficient to accurately represent mineralisation boundaries.

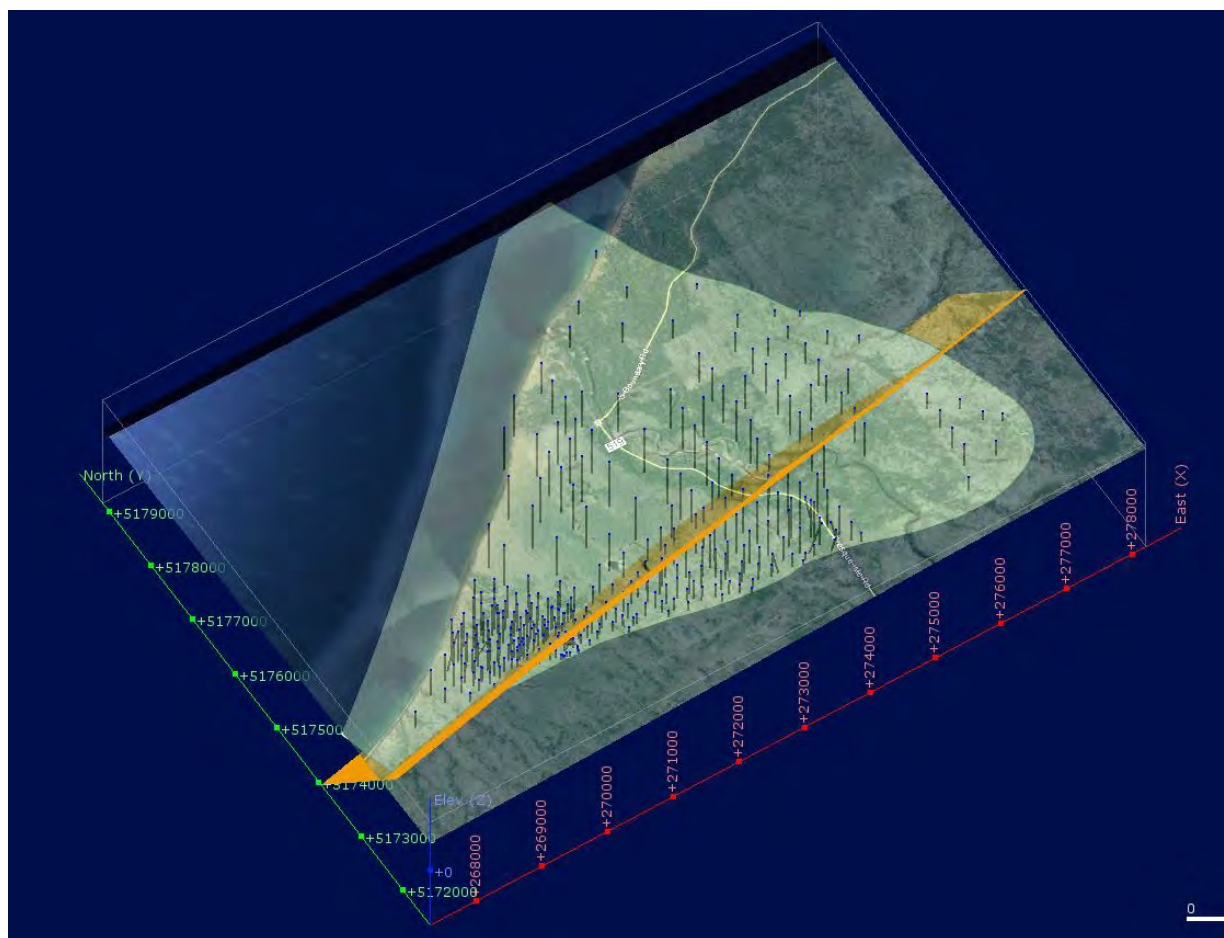
The UCBS is not consistently logged as individual stratigraphic units in the historical data (often logged as "undefined") in the lithology table, so it was not possible to apply the same approach as the LCBS.

Alternatively, GMSI applied the mining lower cut-off considered at the time of modelling (1% Cu) to define a coherent unit of mineralisation. A minimum thickness of 2.0 m was applied during the interpretation to ensure a diluted grade was represented in the block model.

14.2.1 Structural Model

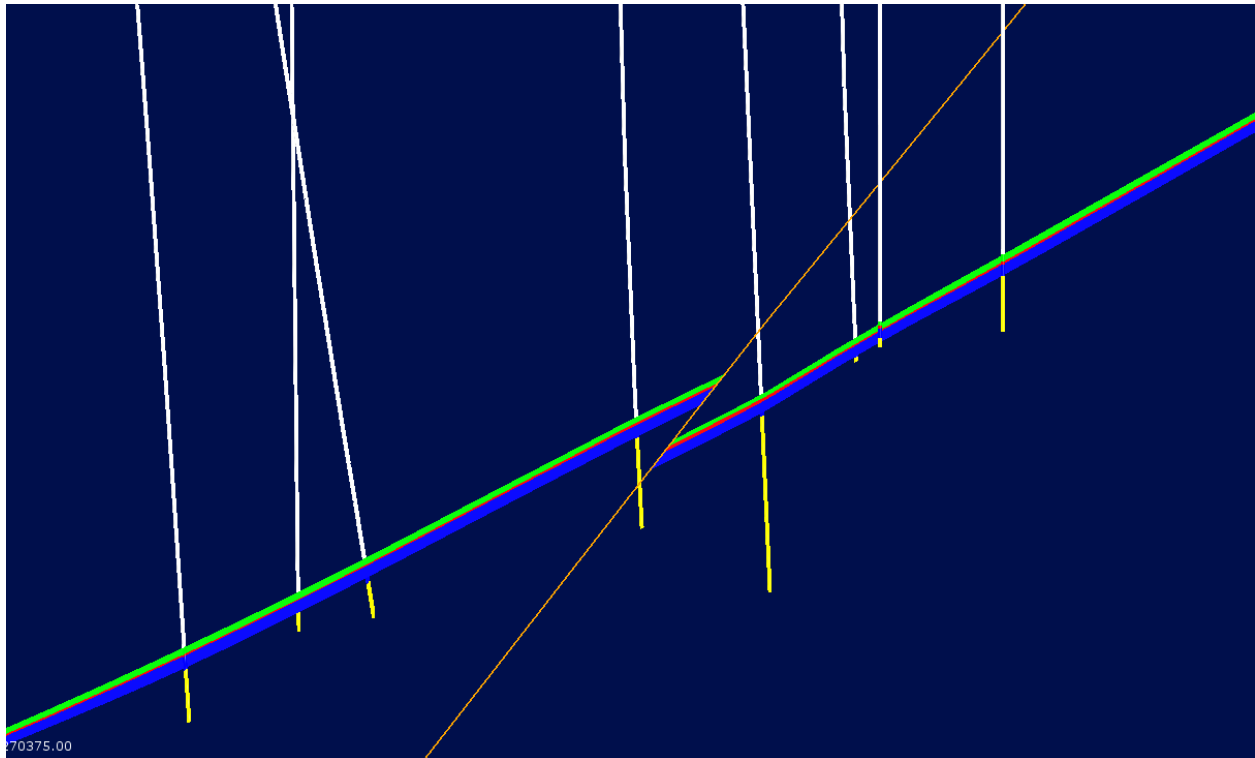
During the 2017 drilling program, a repetition of the LCBS was intersected in CW-17-186, which prompted a review of structural data with the Main Zone of the Copperwood deposit. The review delineated a thrust fault within the extents of 269,500 mE – 271,000 mE and was based off drill core observations from 11 drill holes. The thrust fault strikes around 80° azimuth, with a dip of 20° – 25° to the NNW. GMSI was provided with pierce points of the thrust fault identified within drill core, which were used to construct a 3D plane in Leapfrog Geo™ (Figure 14.2).

Figure 14.2: Orthogonal View (looking NE) Showing the Thrust Fault in Yellow



Although the thrust fault is shown to the extents of the block model, displacement of lithological units is only permitted between 269,500 mE and 271,000 mE. Vertical displacement of lithological units is usually less than 5 m, however, is up to 8 m in places (Figure 14.3).

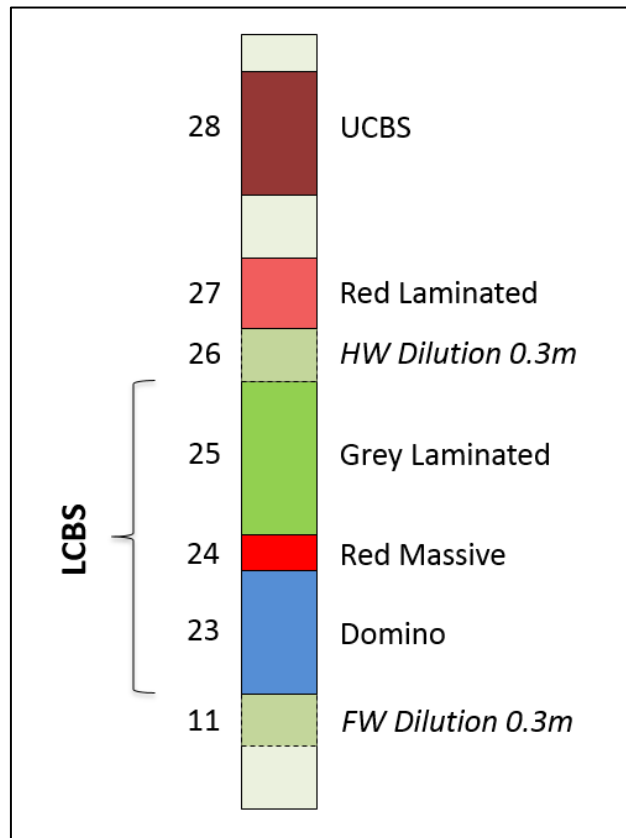
Figure 14.3: Section 270375 mE showing displacement of the UCBS (vertical exaggeration x 3)



14.2.2 Lithology Model - LCBS

Three lithology subunits were coded into the LCBS model: Domino (23), Red Massive (24) and Grey Laminated (25), as shown in Figure 14.4. The overall average of the combined sequence was 2.66 m as stated in the Table 14.2. As mentioned in Section 14.2, the Upper Copper Bearing Sequence (“UCBS”) was modelled with a minimum thickness of 2.0 m applied, which is rarely exceeded as the UCBS is usually between 0.75 m and 1.5 m thick.

The small separation distance (often < 5 m) between the metallurgical wedge drill holes and their respective parent drill holes caused issues during wireframe construction. This was mainly due to suspected small inaccuracies of the distance of the wedge downhole, which caused unrealistically steep dips of the geological contacts over short distances. As the metallurgical wedge drill holes provided little additional information from a Mineral Resource perspective, lithology information from these holes were ignored (the parent drill hole information was retained).

Figure 14.4: Modelling of the Stratigraphy and Associated Rock Codes**Table 14.2: Average Vertical Thicknesses of the LCBS Units**

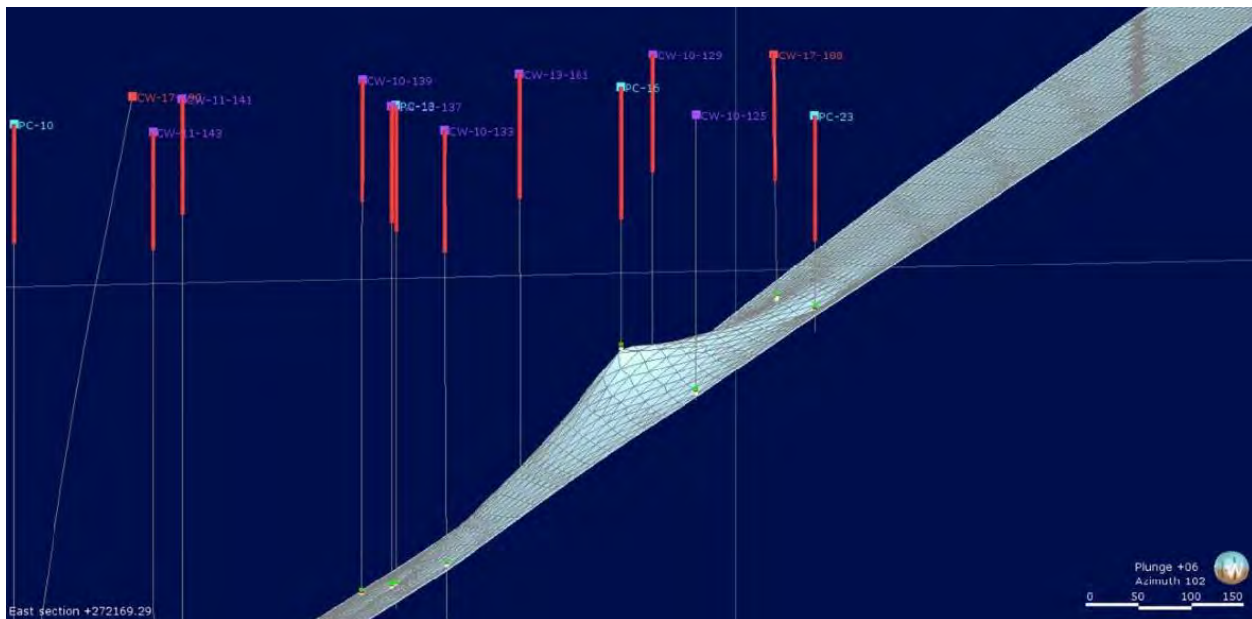
Lithology (Code)	Average Thickness (m)
Gray Laminated (25)	1.21
Red Massive (24)	0.36
Domino (23)	1.09
LCBS (2345)	2.66

Two 0.3 m thick zones of dilution were also coded as the hanging wall (26) and the footwall (11) of the LCBS to ensure accurate representation of dilution grades within the block model. In addition, the remaining portion of the Red Laminated unit (27) was modelled above the hanging wall dilution for geotechnical purposes.

No minimum thickness was applied during modelling of the LCBS, as GMSI will apply a post-processing dilution algorithm to the block model to account for areas where the LCBS is less than the minimum mining thickness (2.0 m)

Lastly, a single historical drill hole (PC-16) was noted to be inconsistent with the LCBS interpretation, causing a geologically unrealistic “cone” effect in the lithology wireframes (Figure 14.5). The intersection in PC-16 is 10-12 m higher than anticipated. Follow-up drilling in 2017 (CW-17-188) near this drill hole confirmed the depth of the LCBS in line with the surrounding drilling. Representatives of Highland revisited the original logs, downhole logging and down hole survey data. However, no error was found. Despite this, it is the opinion of GMSI that PC-16 requires further confirmation, so for this Study the drill hole collar was adjusted to bring PC-16 in line with the geological interpretation.

Figure 14.5: Drill Holes PC-16 and Subsequent Diversion of the LCBS Interpretation



14.2.3 Weathering Wireframes

No oxidation or weathering of the Copperwood orebody is observed in drill core due to erosion and deposition of glacial sediments. Glacial sediments have an average thickness of 29 m and lie unconformably above fresh rock.

The base of overburden surface was modelled using the overburden code “OVb” in the database to produce an upper limit to the interpretation of the LCBS and UCBS.

14.2.4 Topography Surface

A triangulated surface was created from a combination of drill collars and topographic contours derived from LiDAR and was coded into the block model as a topography.

14.3 Statistical Analysis**14.3.1 Statistics of the Raw Assays**

Length-weighted group-wise statistics of the copper and silver raw assays were computed using the geostatistical software R for the entire drilling database. The statistics were studied by lithology groups: Domino (23), Red Massive (24), Gray Laminated (25) and the UCBS (28). Table 14.3 and Table 14.4 respectively present the results of the Study for the copper and silver raw assay grades.

The Domino unit hosts the highest copper and silver grades with averages of 2.19% Cu and 5.26 g Ag/t. The coefficient of variation in this unit is relatively low. The Red Massive is the thinnest unit with an average thickness of 0.36 m and presents the highest coefficient of variation (1.01) of all three separate units due to higher grade variability. The Grey Laminated is lower grade than the Domino and shows a low coefficient of variation indicating grade is very continuous in nature.

The statistics of the UCBS are impacted by the 2.0 m minimum thickness which includes many low-grade samples into the unit, and presents an average grade of 0.73% Cu. Without applying a minimum thickness of 2.0 m, at a 1% Cu cut-off the UCBS is thinner (between 0.75 m and 1.5 m), and grades between 1.5 and 2% Cu.

Table 14.3: Length-weighted Statistics of the Copper Raw-Assays

Lithology (Code)	No. of Assays	Copper Raw Assays (% Cu)					CoV
		Min	Max	Average	Median	Standard Deviation	
UCBS (28)	759	0.004	5.17	0.73	0.45	0.89	1.07
Grey Laminated (25)	921	0.014	6.36	1.13	1.08	0.68	0.60
Red Massive (24)	315	0.004	2.13	0.29	0.20	0.29	1.01
Domino (23)	672	0.003	7.30	2.19	2.06	1.28	0.60

Table 14.4: Length-weighted Statistics of the Silver Raw-Assays

Lithology (Code)	No. of Assays	Silver Raw Assays (g Ag/t)					CoV
		Min	Max	Average	Median	Standard Deviation	
UCBS (28)	510	0.1	240.0	4.44	1.70	13.40	2.57
Grey Laminated (25)	680	0.1	42.0	4.38	2.10	6.06	1.34
Red Massive (24)	238	0.1	12.3	1.29	0.90	1.61	1.21
Domino (23)	542	0.1	108.3	5.26	2.90	11.75	2.03

Cumulative probability plots presented in Figure 14.6 and Figure 14.7 were generated for raw assays of copper and silver for the individual units of the LCBS, and the UCBS. GMSI considers there to be no outliers present in the populations of assays regarding Cu %. The Domino unit shows a natural break in the data at around 1% Cu, which likely represents the natural cut-off of mineralisation.

There appears to be several outliers present in the raw assays for silver (Figure 14.7). These will be investigated further after compositing.

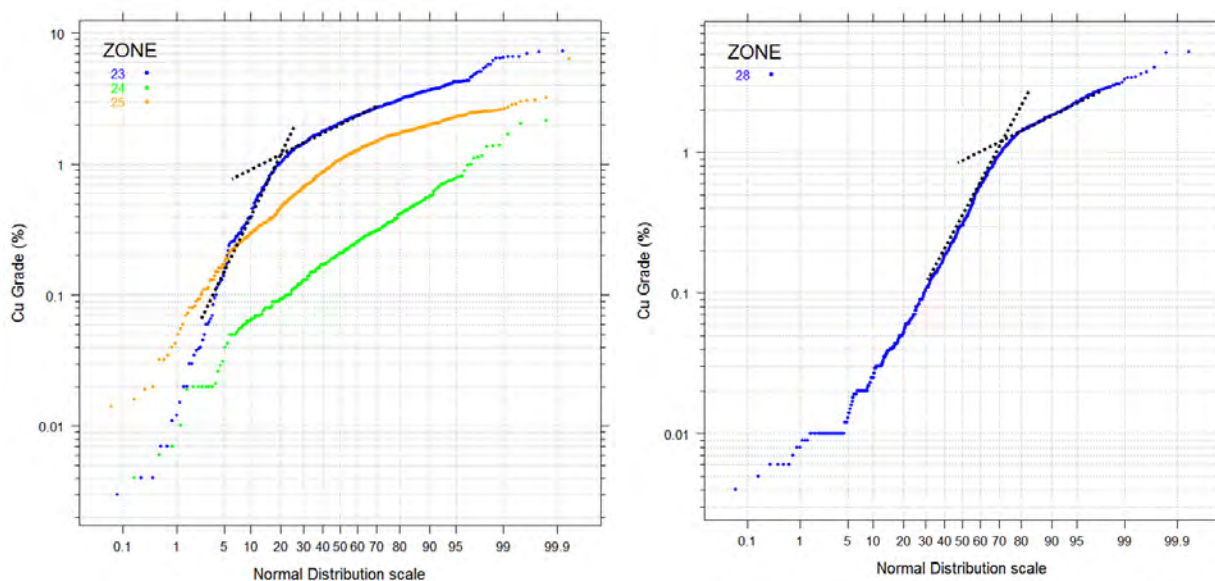
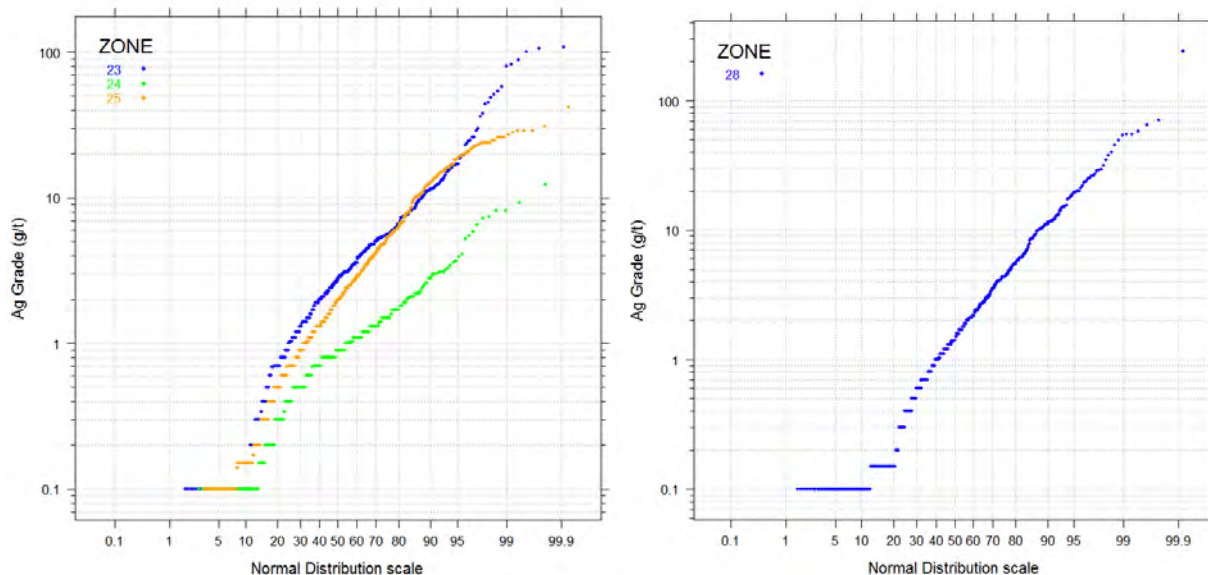
Figure 14.6: Overlaid Cumulative Probability Graphs of Cu % Raw Assays for units of the LCBS (left) and the UCBS (right)

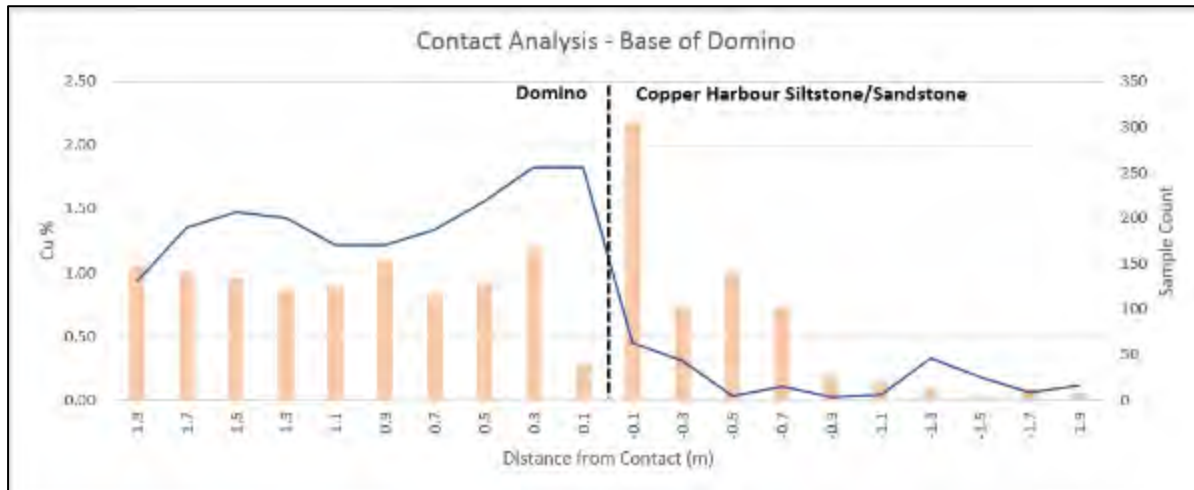
Figure 14.7: Overlaid Cumulative Probability Graphs of Ag g/t Raw Assays for units of the LCBS (left) and the UCBS (right)



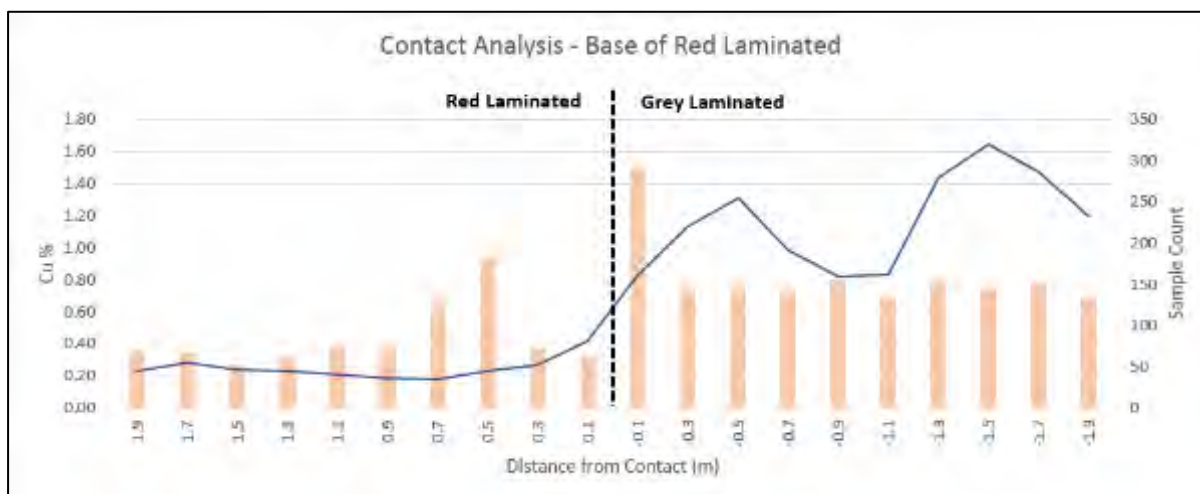
14.3.2 Contact Analysis

To assist in choosing an appropriate estimation methodology, it can be advantageous to determine the nature of the contacts between the individual sub-units of the LCBS (to determine if contacts are sharp or transitional, and to what extent). To quantify this, average grades were calculated as a function of distance from the basal contact of a given subunit (average grades calculated at 20 cm increments away from the boundary). These slopes of these grades can then be examined to see how they behave moving away from a given contact. The key results are presented in Figure 14.8. Positive distances reflect upward distances above the contact, and negative distances reflect downwards distances beneath the contact. The orange bar reflects the number of samples used to calculate the averages, and the blue line represents the average grade.

Figure 14.8: Contact Analysis Plots of the Basal Contact of the Domino (upper image) and Basal Contact of the Red Laminated (lower image)



Blue line represents the average Cu% grade; orange bar reflects the number of samples



The contact between the Domino unit and the Copper Harbour Siltstone/Sandstone (footwall unit) is sharp and reflects a significant drop in grade (from > 1.5% Cu to < 0.5% Cu over a short distance). This implies that a hard boundary must be applied, where composites cannot be shared during estimation between these units. Conversely, the upper boundary of the LCBS (the base of Red Laminated) is a transitional boundary, where over a distance of 0.5 m the grade gradually reduces from 1.2% Cu to 0.2% Cu. The geological boundary between the Red Laminated and Grey Laminated units is not visually sharp in drill core, and grade distributions imply that mineralisation occasionally continues into the basal portion of the Red Laminated unit. For this reason, the hanging wall dilution domain (26) will be estimated in the model to accurately represent the grade of mining dilution.

14.3.3 Compositing

Drill holes intervals were flagged in Leapfrog GEO™, using the constructed wireframes for the LCBS and UCBS. Visual checks were made to ensure that all drill holes were flagged accurately. These intervals were subsequently imported into GEMS as a downhole interval table (LF_APR18) to use during the compositing process.

The uncapped raw assays were composited downhole inside each of the LCBS units (rock codes 23, 24, and 25), the UCBS (rock code 28), the hanging wall / foot wall dilution units (rock codes 11 and 26), and the remainder of the Red Laminated unit (27). For each drill hole, a single length-weighted composite was calculated within each rock code (i.e. composites are limited by geological boundaries).

Statistical checks were undertaken to ensure that the composites were an accurate representation of the raw assays (i.e. length-weighted statistics should be more or less equal for each unit).

14.3.4 Statistics of the Composites

Length-weighted group-wise statistical analysis was undertaken to describe the characteristics of the composites within the zone of mineralization. Table 14.5 and Table 14.16 present the statistics calculated from the copper and silver composites.

Table 14.5: Statistics of the Copper Composites

Lithology (Code)	No. of Composites	Copper Composites (% Cu)					
		Min	Max	Average	Median	Standard Deviation	CoV
UCBS (28)	171	0.002	1.74	0.80	0.78	0.37	0.48
Gray Laminated (25)	314	0.060	2.49	1.13	1.20	0.40	0.34
Red Massive (24)	314	0.004	2.13	0.35	0.24	0.32	0.92
Domino (23)	313	0.004	3.88	2.19	2.20	0.81	0.40

Table 14.6: Statistics of the Silver Composites

Lithology (Code)	No. of Composites	Silver Composites (g Ag/t)					
		Min	Max	Average	Median	Standard Deviation	CoV
UCBS (28)	111	0.37	64.75	4.90	3.73	6.82	1.41
Gray Laminated (25)	242	0.1	20.94	4.34	4.60	2.40	1.04
Red Massive (24)	243	0.1	12.30	1.32	1.36	1.00	1.10
Domino (23)	241	0.1	108.34	5.27	5.76	3.13	2.12

Cumulative probability plots presented in Figure 14.9 and Figure 14.10 were generated for raw assays of copper and silver for the individual units of the LCBS, and the UCBS. GMSI considers there to be no outliers present in the populations of assays regarding Cu %.

Figure 14.9: Overlaid Cumulative Probability Graphs of Cu % Composites for Units of the LCBS (left) and the UCBS (right)

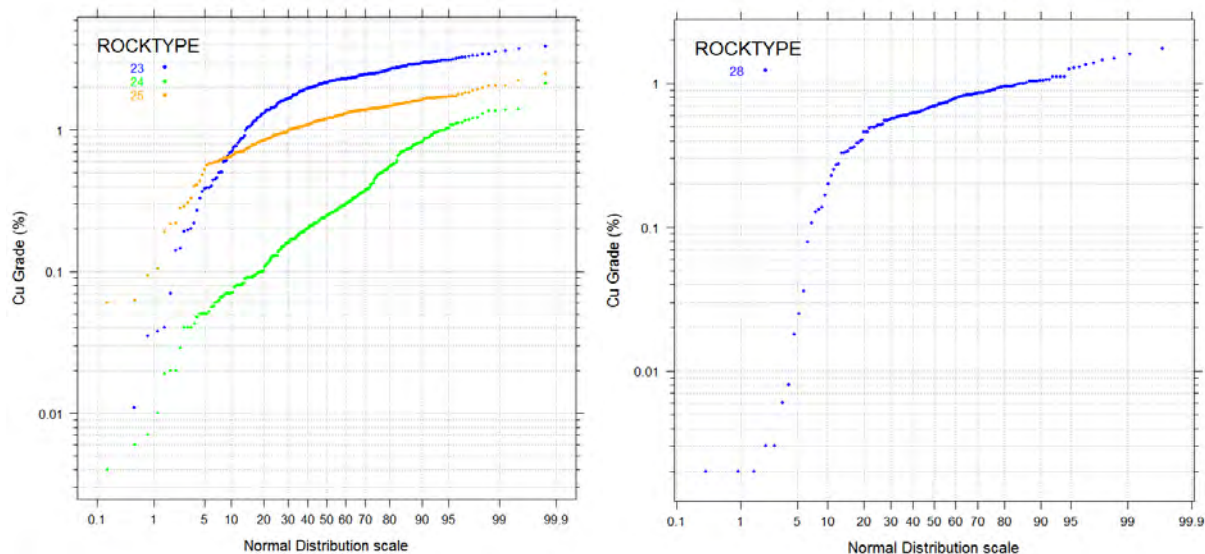
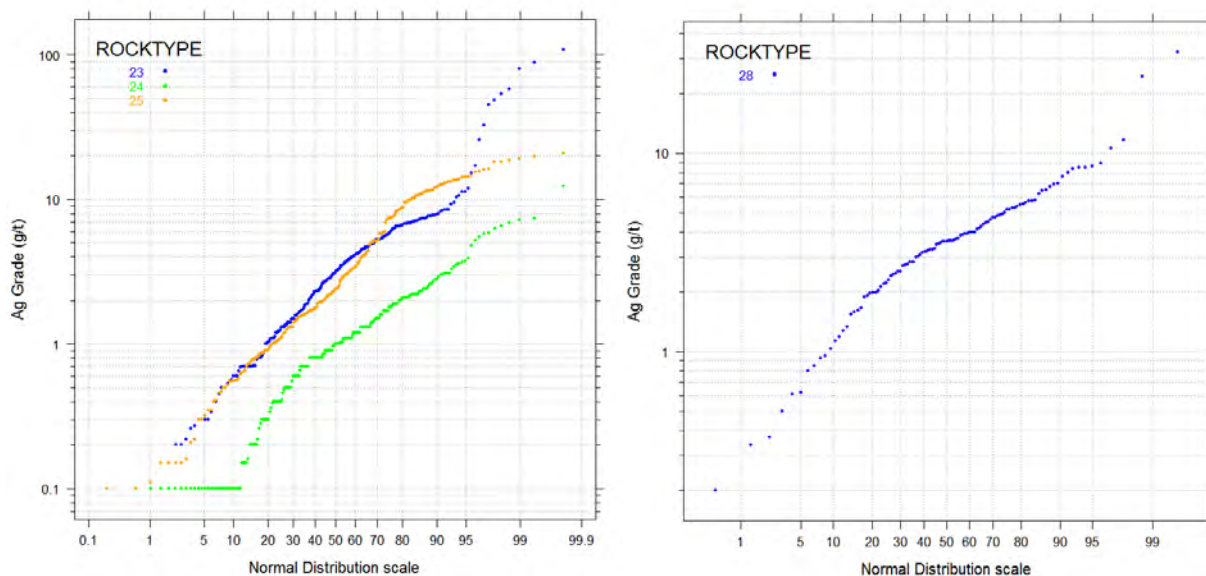


Figure 14.10 Overlaid Cumulative Probability Graphs of Ag g/t Composites for Units of the LCBS (left) and the UCBS (right)



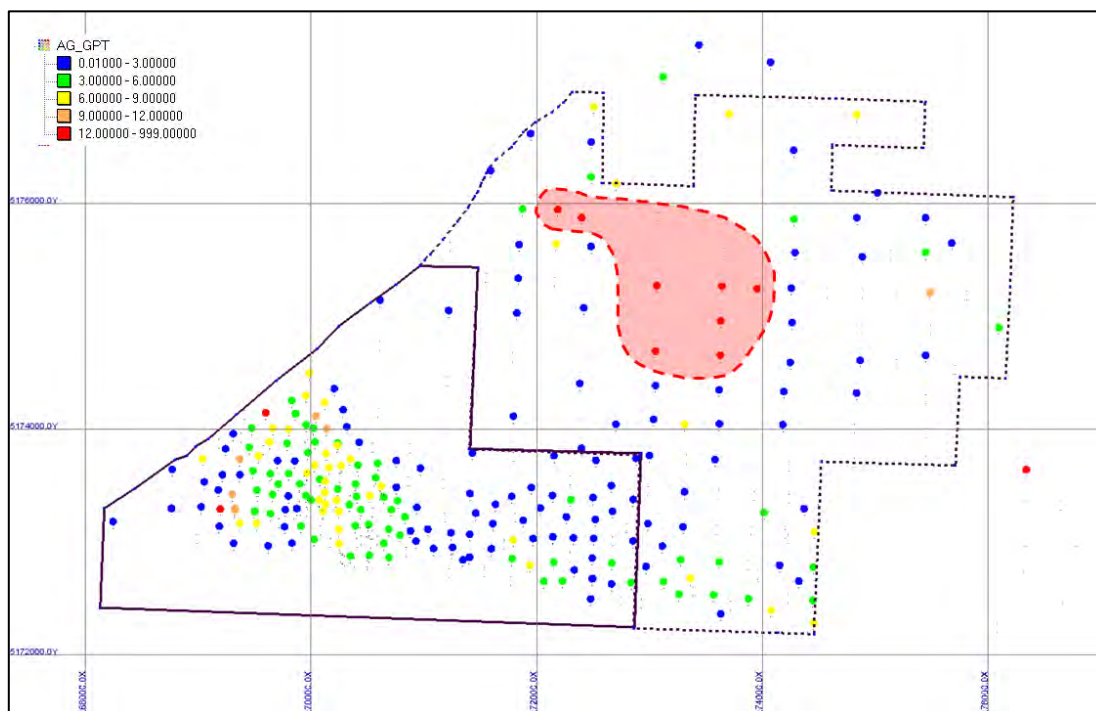
The silver outliers (> 10 g Ag/t) of the Domino unit were further examined to investigate their spatial distribution, and their potential impact on the estimation of the Copperwood deposit. Figure 14.11 shows that the outliers are spatially limited to a zone in the northern extents of the sparsely drilled satellite deposits and appear as a continuous zone of high-grade silver mineralisation. As they represent a natural sub-

population within the data confined to a limited aerial extent, GMSI has not applied any grade capping of silver composites within the Domino.

No significant silver outliers were identified in the Red Massive (24) or Grey Laminated (25) units, and two potential outliers in the UCBS are located on the extremities of the lease boundaries, where extrapolation will be limited.

As a result of this review, no grade capping was applied to either copper or silver composites for this resource estimate.

Figure 14.11: Composites from the Domino Unit Colored by Ag with Leasing Outlines. Note the Sub-population in the Northern Area (within the sparse drilling)



14.4 Bulk Density Data

The database includes 316 samples of specific gravity measurement taken in the drill holes throughout the Copperwood Deposit. Table 14.7 and Table 14.18 present the statistics of the measurements by years of sample collection for the LCBS, and by subunit within the LCBS and UCBS. The average density observed was 2.71 g/cm³ for the LCBS. The range of the density data is minimal, where the minimum and maximum values were respectively 2.62 g/cm³ and 2.79 g/cm³. Due to the low variability observed in the density data,

no study was undertaken to quantify the relationship between density and Cu %. Table 14.9 summarizes the values of density utilized in the resource estimation.

Table 14.7: Statistics of the Specific Gravity Measurements Presented by Years of Collection for the LCBS

Year	No. of Measurements	Specific Gravity Measurement (g/cm ³)				
		Min	Max	Average	Median	Standard Deviation
1956-1957	25	2.70	2.74	2.72	2.73	0.014
2009-2011	171	2.62	2.79	2.71	2.70	0.029
2017	16	2.62	2.75	2.69	2.70	0.033
All Years	212	2.62	2.79	2.71	2.71	0.028

Table 14.8: Statistics of the Specific Gravity Measurements Presented by Lithology

Lithology	No. of Measurements	Specific Gravity Measurement (g/cm ³)				
		Min	Max	Average	Median	Standard Deviation
Domino	76	2.63	2.79	2.70	2.70	0.036
Red Massive	37	2.65	2.75	2.70	2.70	0.019
Grey Laminated	99	2.62	2.76	2.72	2.72	0.021
UCBS*	47	2.56	2.79	2.69	2.70	0.051

*Determined from all density samples within the UCBS solid wireframe

Table 14.9: Specific Gravity Averages Used in the Resource Estimation

Lithology	Specific Gravity (g/cm ³)
Overburden	2.20
Domino	2.70
Red Massive	2.70
Grey Laminated	2.72
UCBS	2.69

14.5 Variography

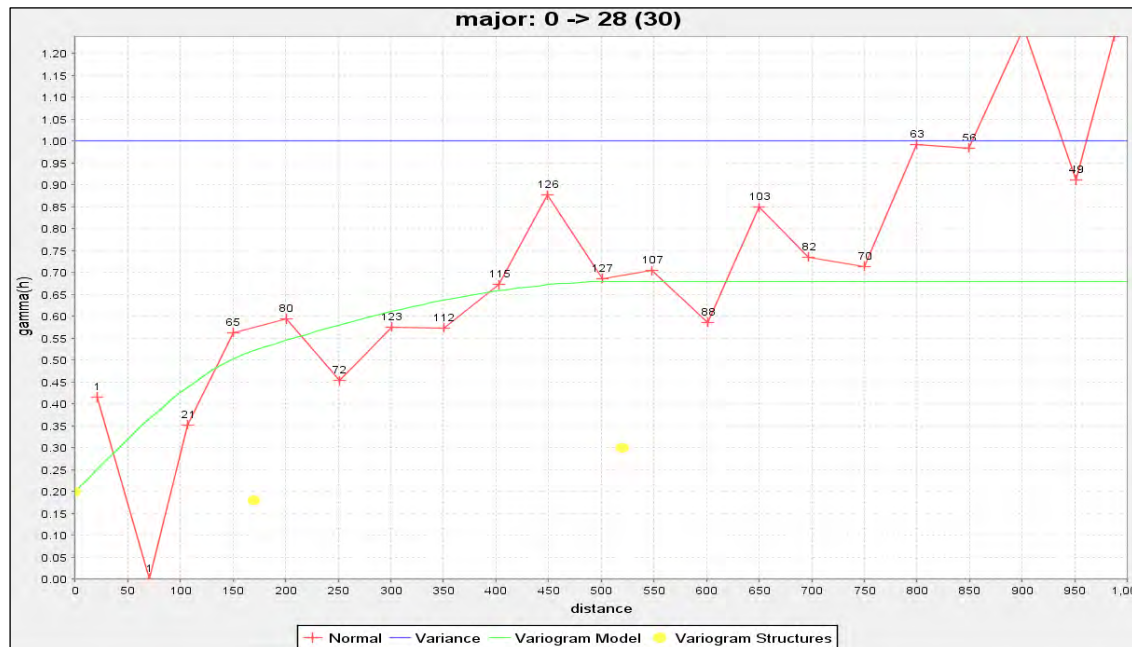
Grade variography was generated in preparation for the estimation of copper and silver grades using the Ordinary Kriging interpolation method. The variography was undertaken on the composites for each unit of the LCBS and the UCBS. Geovia GEMS™ was used to perform the variographic analysis.

A series of variograms was generated from the composites of each unit every 5 degrees azimuth and 5 degrees dip increments. The spread angle was set to 30 degrees, with a bandwidth of 250 m. A lag distance of 50 m was applied. Only composites selected between 268000 mE and 275000 mE, and 5172000 mN and 5174500 mN were selected to produce the variograms (Main Zone, Section 5 and Section 6). The manually-fitted variogram models included a nugget effect and two spherical structures. The variography study highlighted a near horizontally isotropic distribution of copper and a low nugget effect on copper and silver grades. The results of the models for copper and silver are tabulated in Table 14.10.

Table 14.10: Variogram Models for the Copper and Silver Composites of Zone

Element	Rock Codes	Nugget	Ranges of Influence (m)								Rotation		
			1 st Structure				2 nd Structure				Azi	Dip	Azi Int.
			X	Y	Z	Sill	X	Y	Z	Sill			
Cu	23	0.026	350	268	60	0.028	600	459	100	0.200	150	5	240
	24	0.024	175	132	60	0.031	500	378	100	0.027	118	0	208
	25	0.032	170	104	60	0.029	520	318	100	0.048	28	-5	118
	28	0.036	250	204	60	0.025	575	470	100	0.036	118	0	208
Ag	23	1.01	260	210	60	1.70	630	500	100	4.19	150	5	240
	24	0.36	250	150	60	0.36	600	340	100	0.6	140	5	230
	25	3.25	550	363	60	1.30	740	489	100	10.85	150	5	240
	28	3.11	400	314	60	2.24	550	432	100	5.59	118	0	208

Figure 14.12 shows an example of a relative semi-variogram for Cu % for the principal direction (X), with the spherical model overlain in yellow. The range of 500 m corresponds to the maximum distance of grade continuity between pairs of composites for this subunit.

Figure 14.12: Variogram Model Cu% for the Grey Laminated subunit of the LCBS

14.6 Block Modelling

A single block model was constructed for the Copperwood Project, including both the Copperwood Deposit and the Satellite Deposits. The block model covers an area large enough to manage underground developments. The block model was set in the Geovia GEMS™ 6.7.4 database environment.

The drilling pattern, the anticipated “room and pillar” mining scenario and minimum mining height considerations guided the choice of block dimension and orientation. The block model parameters for the Copperwood Project are summarized in Table 14.11.

Table 14.11: Block Model Parameters - Copperwood Project

Block Model Name	Orientation	Origin	Number of Columns, Rows, Levels	Block Size (m)	Rotation ¹
ENG_APR18	East	268,000	480	20	0°
	North	5,172,000	330	20	
	Elevation	320	270	2.5	

Note: For a positive value, the direction of rotation is counterclockwise around the elevation axis

The rock type model, or domain coding, relied on the wireframe constraints presented in Section 14.2.2. A “percentage” type block model was adopted, where a single block can contain numerous rock codes, with their proportions expressed as percentages of the block. This methodology was adopted due to the thin nature of the subunits of the LCBS, and the large spatial extent of the deposit (10 km x 6 km), which minimizes the size of the block model whilst retaining a high level of precision. Sub-blocking was not applied for the Mineral Resource; however the block model was converted to a sub-blocked model for mine planning purposes at a later date.

Table 14.12 describes the coding and the associated domain used in the mapping of the Lower Copper Bearing Sequence (LCBS: Gray Laminated, Red Massive and Domino beds) in the block model. All densities associated to hard rock are set to a uniform 2.7 g/cm³. Overburden blocks were assigned a density of 2.2 g/cm³.

Table 14.12: Rock Codes Used in the Rock Type Model

Rock Code	Description	Specific Gravity
9	Overburden	2.20
0	Host Rock	2.69
11	Foot Wall Dilution	2.63
23	Domino Subunit	2.70
24	Red Massive Subunit	2.70
25	Grey Laminated Subunit	2.72
26	Hanging Wall Dilution	2.71
27	Red Laminated Subunit	2.71
28	UCBS	2.69

Additionally, a series of attributes needed during the block modelling development were incorporated into the block model project. Table 14.13 presents the list of attributes found in the block model project ENG_APR18.

Table 14.13: List of Attributes Found in the Block Model

Folder Name	Model Name	Description
ENG_APR18	Rock_##	Individual Rock Coding (11, 23, 24, 25, 26, 27, 28, 28t)
	Density_WA	Specific Gravity
	Perc_##	Percent Attributes (11, 23, 24, 25, 26, 27, 28, 28T)
	Cu_##	OK Cu % (11, 23, 24, 25, 26, 27, 28, 28T)
	Ag_##	OK Ag ppm (11, 23, 24, 25, 26, 27, 28, 28T)
	CATEG_Apr18	Resource Category
	Rock_LCBS	LCBS Rock Code 232425 (blocks pertaining to 23, 24, or 25)
	Perc_LCBS	LCBS Percentage (blocks pertaining to 23, 24, or 25)
	Cu_LCBS	LCBS Weighted Average Cu % (undiluted)
	Ag_LCBS	LCBS Weighted Average Ag ppm (undiluted)
	Thick_Calc	LCBS Thickness (undiluted)
	Thick_LCBS_Dil	LCBS Thickness (diluted to 2m)
	Cu_Dil	LCBS Diluted Cu % to 2m thickness
	Ag_Dil	LCBS Diluted Ag ppm to 2 m Thickness
	Perc_Dil	LCBS Diluted Percentage
	RC_All	Rock code 1 for all Modelled Units
	Cu_RLAM	Red Laminated (26, 27) Weighted Average Cu %
	Ag_Col	LCBS Weighted Average Ag ppm (undiluted) for entire column
	Ag_Col_Dil	LCBS Weighted Average Ag ppm (diluted to 2m) for entire column. Used for accurate reporting of Ag ppm in Resource Statement.
	Cu_Col	LCBS Weighted Average Cu % (undiluted) for entire column.
	Cu_Col_Dil	LCBS Weighted Average Cu % (diluted to 2 m) for entire column. Used for accurate reporting of Cu % in Resource Statement.
	Cu_RLAM_Col	Red Laminated Weighted Average Cu % for entire column

14.7 Grade Estimation Methodology

The final interpolation technique selected for the Copperwood Project is the Ordinary Kriging (“OK”) method.

Grade estimates were generated using the drill hole composites (one per drill hole, per rock code). The boundaries of each domain were considered as hard boundaries through each interpolation step. Only composites pertaining to a given domain were used to estimate that domain. Geovia® GEMS 6.7.4 software was used for the estimate.

The sample search approach used to estimate copper and silver for all units of the LCBS (23, 24, 25) and the UCBS (28) for the Copperwood Project is summarized below:

- **First Pass:** A minimum of 2 and a maximum of 10 composites within the Pass 1 search ellipse ranges.
- **Second Pass:** A minimum of 2 and a maximum of 10 composites within the Pass 2 search ellipse ranges. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Pass:** A minimum of 1 and a maximum of 10 composites within the Pass 3 search ellipse ranges. Only blocks which were not estimated during the first and second pass could be estimated during the third pass.

For the foot wall (11), hanging wall (26) dilution domains and the red laminated subunit (27), Inverse Distance Square ("ID²") interpolation method was used (applying the same passes and search ellipses for the estimation of Cu and Ag).

It was judged unnecessary to apply restriction on search ellipse ranges for high grade composites, based on the high-grade sub-populations identified in Section 14.3.4. The various profiles for interpolation and search ellipses utilized in the estimation of the resource are tabulated in Table 14.14 and Table 14.15.

Table 14.14: Interpolation Profile Settings for Resource Estimation - Copperwood Project

Profile Name	Element Estimated	Pass	Sample			Ellipses Name	Semi-Variogram Name
			Min	Max	Max per Hole		
CU_11_1	Cu	1	2	10	1	CU_175	-
CU_11_2	Cu	2	2	10	1	CU_250	-
CU_11_3	Cu	3	1	10	1	CU_350	-
CU_23_1	Cu	1	2	10	1	CU_175	CU_23
CU_23_2	Cu	2	2	10	1	CU_250	CU_23
CU_23_3	Cu	3	1	10	1	CU_350	CU_23
CU_24_1	Cu	1	2	10	1	CU_175	CU_24
CU_24_2	Cu	2	2	10	1	CU_250	CU_24
CU_24_3	Cu	3	1	10	1	CU_350	CU_24
CU_25_1	Cu	1	2	10	1	CU_175	CU_25
CU_25_2	Cu	2	2	10	1	CU_250	CU_25
CU_25_3	Cu	3	1	10	1	CU_350	CU_25
CU_26_1	Cu	1	2	10	1	CU_175	-
CU_26_2	Cu	2	2	10	1	CU_250	-
CU_26_3	Cu	3	1	10	1	CU_350	-
CU_27_1	Cu	1	2	10	1	CU_175	-
CU_27_2	Cu	2	2	10	1	CU_250	-
CU_27_3	Cu	3	1	10	1	CU_350	-
CU_28_1	Cu	1	2	10	1	CU_175	CU_28
CU_28_2	Cu	2	2	10	1	CU_250	CU_28
CU_28_3	Cu	3	1	10	1	CU_350	CU_28
AG_11_1	Ag	1	2	10	1	AG_175	-
AG_11_2	Ag	2	2	10	1	AG_250	-
AG_11_3	Ag	3	1	10	1	AG_350	-
AG_23_1	Ag	1	2	10	1	AG_175	AG_23
AG_23_2	Ag	2	2	10	1	AG_250	AG_23
AG_23_3	Ag	3	1	10	1	AG_350	AG_23
AG_24_1	Ag	1	2	10	1	AG_175	AG_24

Profile Name	Element Estimated	Pass	Sample			Ellipses Name	Semi-Variogram Name
			Min	Max	Max per Hole		
AG_24_2	Ag	2	2	10	1	AG_250	AG_24
AG_24_3	Ag	3	1	10	1	AG_350	AG_24
AG_25_1	Ag	1	2	10	1	AG_175	AG_25
AG_25_2	Ag	2	2	10	1	AG_250	AG_25
AG_25_3	Ag	3	1	10	1	AG_350	AG_25
AG_26_1	Ag	1	2	10	1	AG_175	-
AG_26_2	Ag	2	2	10	1	AG_250	-
AG_26_3	Ag	3	1	10	1	AG_350	-
AG_27_1	Ag	1	2	10	1	AG_175	-
AG_27_2	Ag	2	2	10	1	AG_250	-
AG_27_3	Ag	3	1	10	1	AG_350	-
AG_28_1	Ag	1	2	10	1	AG_175	AG_28
AG_28_2	Ag	2	2	10	1	AG_250	AG_28
AG_28_3	Ag	3	1	10	1	AG_350	AG_28

Table 14.15: Sample Search Ellipsoid Settings for Resource Estimation - Copperwood Project

Rock Code	Element	Pass	Ellipse Profile Name	Anisotropy Range (m)			Rotation		
				X	Y	Z	Z	X	Z
2345	Cu	1	CU_175	175	175	75	0	-10	0
		2	CU_250	250	250	100			
		3	CU_350	350	350	100			
	Ag	1	AG_175	175	175	75			
		2	AG_250	250	250	100			
		3	AG_350	350	350	150			

14.8 Classification and Resource Reporting

The CIM Definition Standards on Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Resource Definition and adopted by the CIM council on May 10, 2014, provide standards for the classification of Mineral Resources and Mineral Reserves estimates into various categories. The

category to which a resource or reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit, the quality and quantity of data available, the level of detail of the technical and economic information which has been generated about the deposit and the interpretation of that data and information. Under CIM Definition Standards:

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 or 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.

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 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.

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.

In addition, the classification of interpolated blocks is undertaken by considering the following criteria:

- Quality and reliability of drilling and sampling data;
- Distance between sample points (drilling density);
- Confidence in the geological interpretation;
- Continuity of the geologic structures and the continuity of the grade within these structures;
- Variogram models and their related ranges (first and second structures);
- Statistics of the data population;

- Quality of assay data.

The resources were classified according to the above-mentioned criteria which also directed the choice of the search parameters for each interpolation pass during the block estimation.

While strongly based on interpolation passes described above, resource categories were not defined solely on this basis. To delineate Measured, Indicated and Inferred Mineral Resources, GMSI outlined groups of globally similar interpolation passes. Figure 14.13 shows how the resource categories are outlined around interpolation passes for the Copperwood Deposit.

Measured Mineral Resources are limited to the blocks located inside the “Measured Outline”. Measured Mineral Resources include blocks generally interpolated in the first pass. No Measured Resources are estimated in the Satellite Deposits.

Indicated Mineral Resources are limited to the blocks located at the periphery of the Measured category blocks and inside of the “Indicated Outline”. Indicated Mineral Resources are generally interpolated in the second pass. No Indicated Resources are estimated in the Satellite Deposits.

Inferred Mineral Resources are all the blocks not included in the Measured or Indicated Mineral Resources but included inside the “Inferred Outline”. All interpolated blocks inside the Satellite Deposits outline are categorized as Inferred.

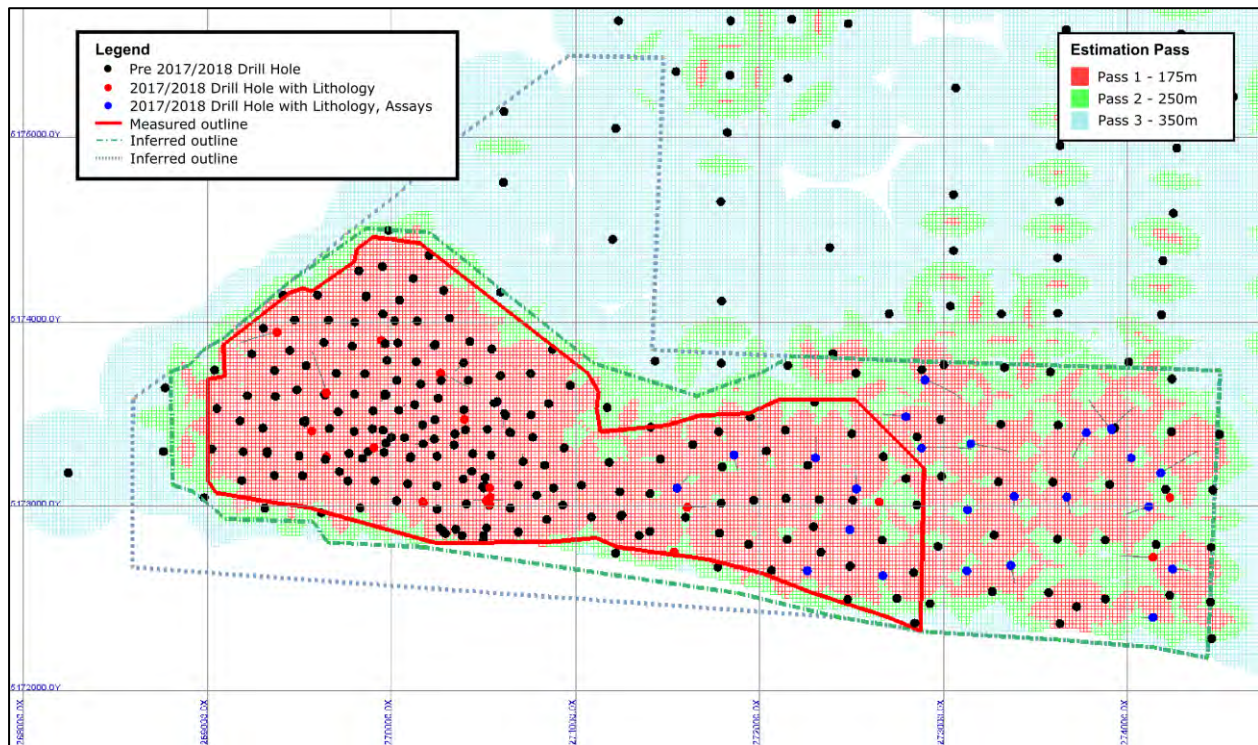
Figure 14.13: Interpolation Passes with April 2018 Mineral Resource Categories – Copperwood Deposit

Figure 14.14 shows the previous resource categories applied by GMSI for the April 2018 Mineral Resource estimate, compared to Figure 14.15 which shows the resource categories from the October 2017 Mineral Resource estimate. Measured Resources constitute essentially the bulk of the Mineral Resources in the Copperwood Deposit, where the drilling density is the highest. Indicated Resources surround the latter category and are mostly present in the eastern half of the Copperwood Deposit (Sections 5 and 6) where the drill spacing is sparser. Inferred Resources constitute 100% of the Mineral Resources found in the Satellite Deposits. Most of the Inferred Mineral Resources of the Copperwood Deposit are of copper grading between 0.5 and 1.0% Cu.

Figure 14.14: Resource Categories - Copperwood Project – April 2018

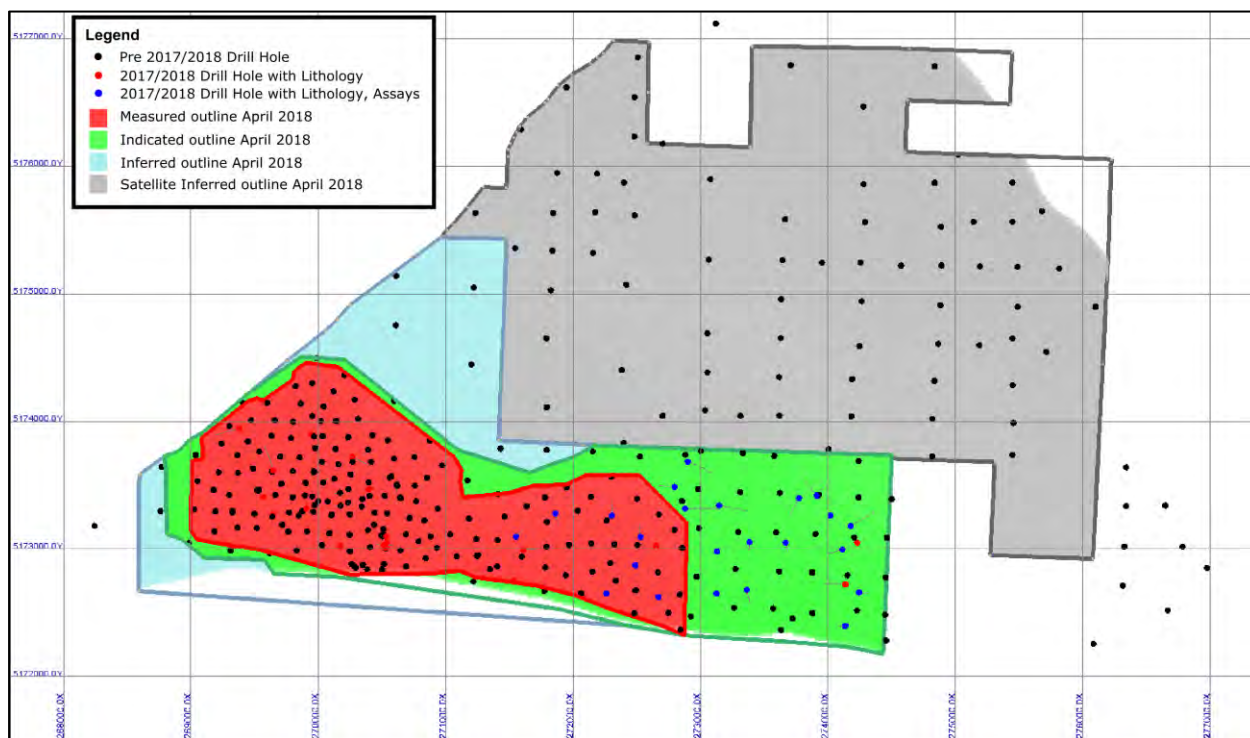
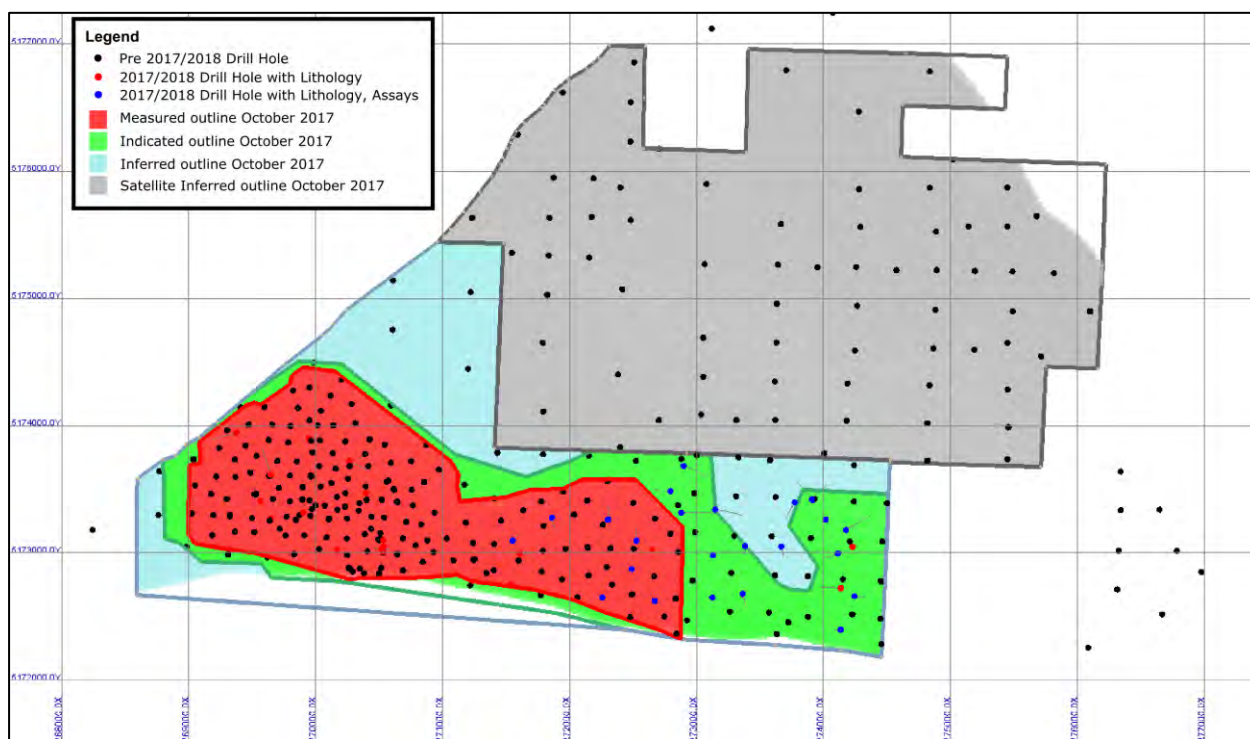


Figure 14.15: Resource Categories - Copperwood Project – October 2017



14.9 Grade Estimation Validation

Validation was completed on the Copperwood Project block model. The validation process included visual checks, statistical validation of the model, local validation by swath plots and an assessment of grade smoothing (conditional bias).

14.9.1 Visual Validation

The visual checks consisted of 2D plan views of the block model (for each rock code), the relevant lithology wireframes, and the drill hole composites. In addition, the slicing was performed vertically on 100 m intervals orientated North-South. Various attributes (rock type, percent attribute, density, Cu and Ag grades) throughout the strike length of the deposit were reviewed. The LCBS and associated percent attribute are well represented in their proper attribute model. The Ordinary Kriging based copper and silver resource estimate was found to be a good visual representation of the drill hole composites.

14.9.2 Statistical Validation

A statistical comparison between composites used in the interpolation and block grades was performed to evaluate if samples used in the estimation are well represented in the block model. Statistics were calculated for the key zones of mineralisation (Main Zone, Sections 5 and 6), defined by blocks (Pass 1 and 2 only) and composites between 268000 mE – 275000 mE, and 5172000 mN – 5174500 mN. Declustering of composites is necessary due to the variable sample spacing, therefore weightings were calculated for each composite and applied during the compilation of descriptive statistics.

Table 14.16 and Table 14.17 present the comparison between the composite grades and block grades for copper and silver.

Table 14.16: Comparative Statistics for Cu (%) Between Composites and Blocks Grouped by Rock Code

Domain	No. of Composites	Composites (Cu %)		Variance of Composites	Number of Blocks	Blocks (Cu %)		Variance of Blocks	Reduction in Variance	No. of Blocks for Each Composite
		Mean	Median			Mean	Median			
23	241	2.36	2.31	0.48	60,603	2.31	2.29	0.33	31%	251
24	241	0.35	0.27	0.11	57,257	0.34	0.30	0.06	49%	238
25	241	1.19	1.27	0.16	62,879	1.18	1.24	0.11	32%	261
28	98	0.91	0.88	0.14	43,442	0.87	0.87	0.10	28%	443

Table 14.17: Comparative Statistics for Ag (g/t) Between Composites and Blocks Grouped by Rock Code

Domain	No. of Composites	Composites (Ag g/t)		Variance of Composites	Number of Blocks	Blocks (Ag g/t)		Variance of Blocks	Reduction in Variance	No. of Blocks for Each Composite
		Mean	Median			Mean	Median			
23	195	4.55	3.20	11.7	60,146	4.21	3.21	8.8	25%	308
24	197	1.37	1.10	1.6	56,810	1.54	1.18	1.2	22%	288
25	196	4.59	3.11	25.2	62,338	3.52	2.03	12.8	49%	318
28	65	3.43	3.47	2.6	42,857	3.32	3.22	1.3	51%	659

In general, the reconciliation of grade between the composites and blocks is good (less than 10% difference in mean grades). Silver grade reconciliation for rock code 25 (Grey Laminated) are adversely affected by a localised area of higher composite grades, hence the blocks appears under-estimated in the comparative statistics.

14.9.3 Quantile: Quantile Plots

In addition to descriptive statistics, Q:Q plots were generated to assess the distribution of copper and silver grades of composites against blocks on a domain by domain basis. These plots are useful in assessing the degree of smoothing (conditional bias) observed during the grade estimation process and can identify any significant over/under estimation of grades.

Regarding copper grades, the Q:Q plots show minimal smoothing of copper grade, which is also supported by the small reduction in variance observed between the composite and block statistics shown in Table 14.16. For silver, an under-estimation was observed in the Grey Laminated (as highlighted by the comparative statistics), however, due to the economic value silver in the Copperwood deposit, this was not investigated further.

Figure 14.16: Quantile: Quantile Plots of Cu % distributions for the Domino (23) and Grey Laminated (25) Subunits of the LCBS.

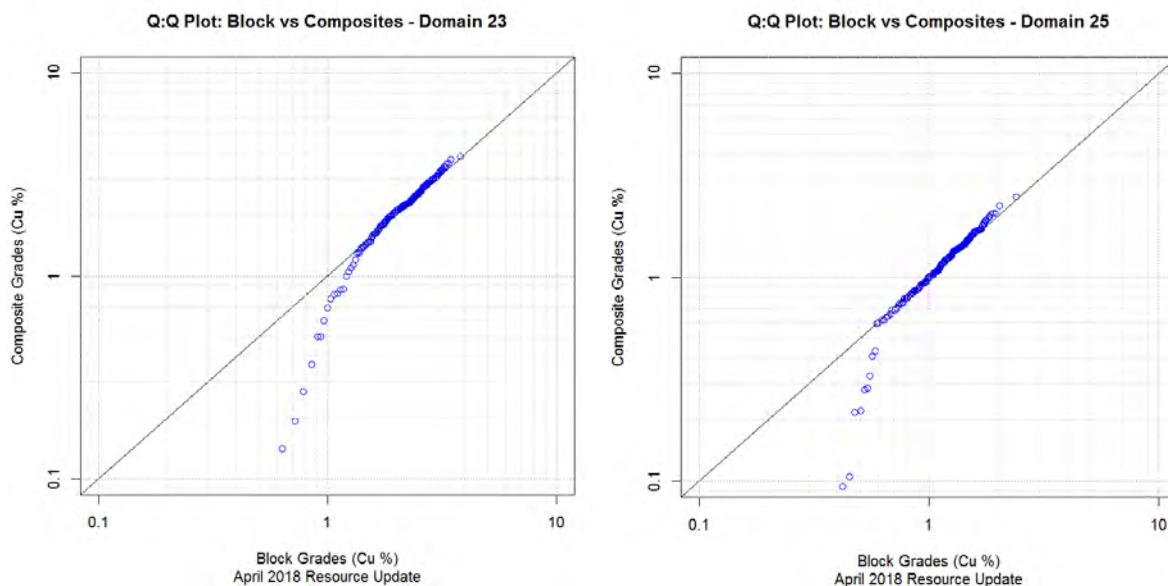
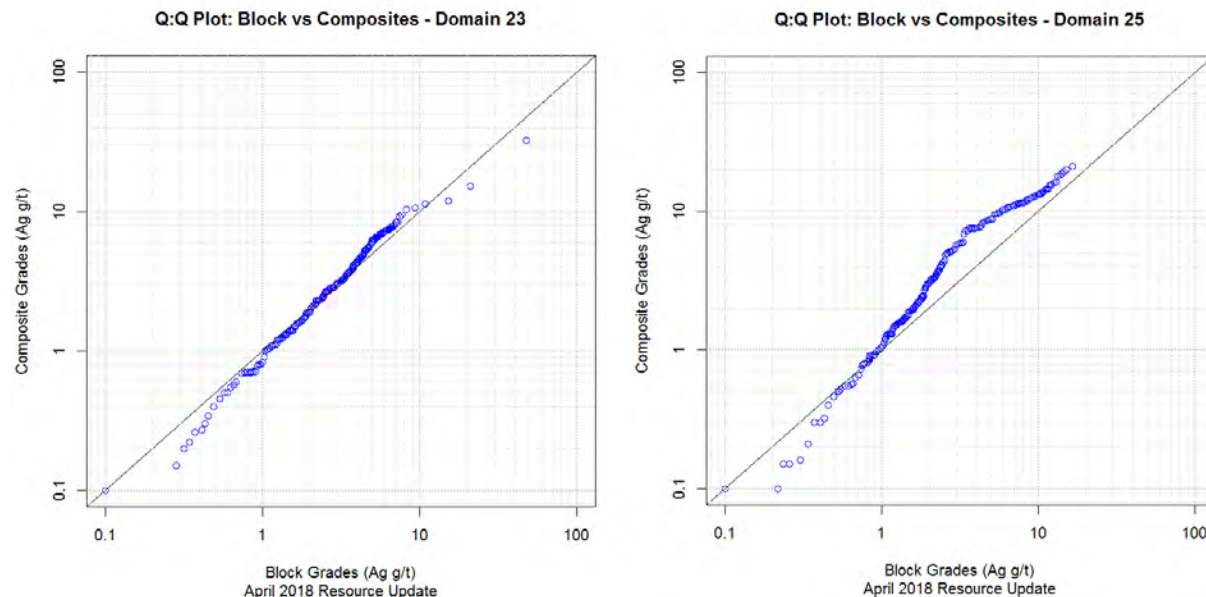


Figure 14.17: Quantile:Quantile plots of Ag g/t Distributions for the Domino (23) and Grey Laminated (25) Subunits of the LCBS

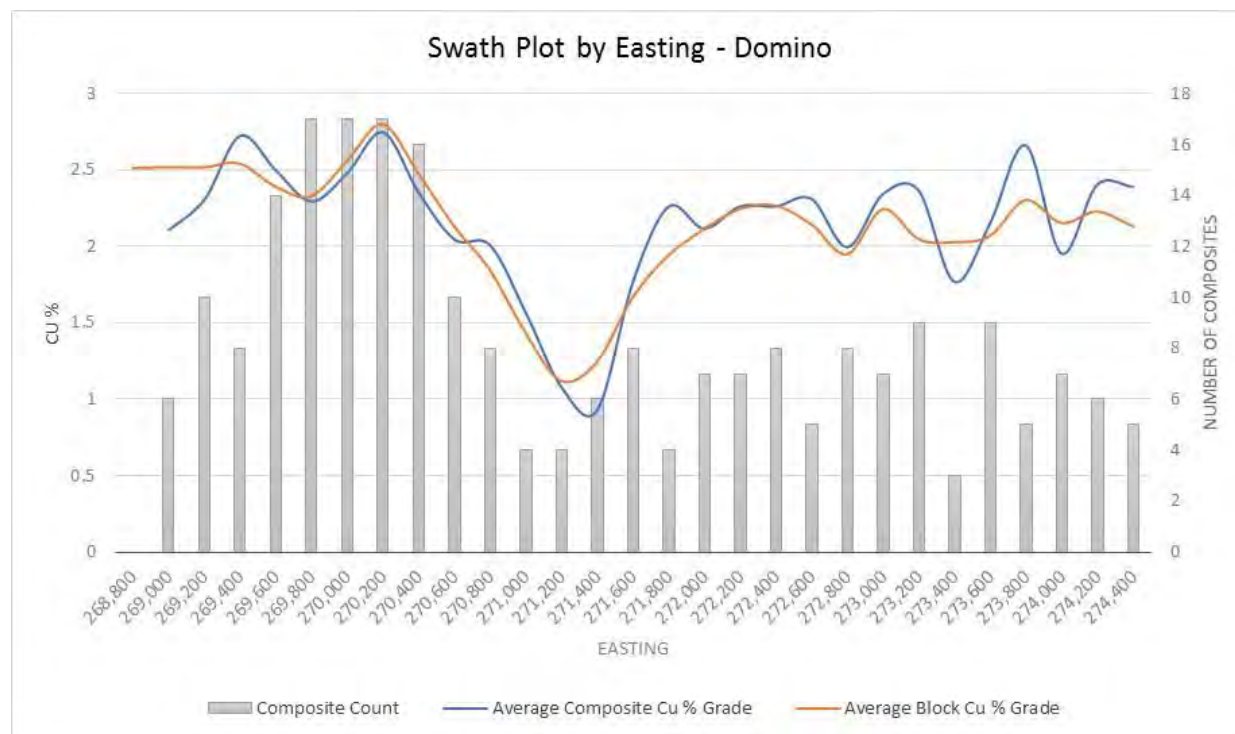


14.9.4 Local Statistical Validation - Swath Plots

The swath plot method is considered a local validation, which works as a visual means to compare estimated block grades against composite grades within a 3D moving window. It is used to identify possible bias in the interpolation (i.e. over/under estimation of grades).

Swath plots were generated for all subunits of the LCBS and the UCBS at increments of 200 m (Easting) for both Cu % and g Ag/t. Peaks and lows in estimated grades should generally follow peaks and lows in composite (or point) grades in well-informed areas of the block model, whereas less informed areas can occasionally show some discrepancies between the grades.

Figure 14.18 illustrates an example swath plot for the Domino subunit of the LCBS by Easting. Peaks and lows in copper content match peaks and lows in composite grades; no bias was found in the resource estimate in this regard. For all other rock codes, no significant bias was observed.

Figure 14.18: Swath Plot of Cu % for the Domino (23) by Easting

14.9.5 Discussion on Block Model Validation

Overall, the Copperwood block model is a good representation of composite copper and silver grades used in the estimation. Global statistical validations show the degree of smoothing is minimal, and no significant over/under-estimation of copper grades has occurred. Local statistical validations show good local correlation of block and composite gold grades, and no excessive extrapolation of grades was observed.

14.10 Global Resources

For the purposes of Mineral Resource Reporting, weighted-average copper and silver grades were calculated for the LCBS, using the grades and percentages estimated individually in each subunit (Domino, Red Massive and Grey Laminated).

14.10.1 Grade Dilution

The minimum mining height for the purposes of reporting a Mineral Resource has been set at 2 m for this Study. Therefore, to ensure that Mineral Resources are reported in line with RPEE ("Reasonable Prospects for Economic Extraction") as stipulated by the CIM guidelines for Mineral Resource Reporting, GMSI applied the following procedure for grade dilution within the LCBS using a minimum mining height of 2 m.

True thickness of the LCBS (Domino, Red Massive and Grey Laminated combined) was calculated and coded into each block within the LCBS unit. For blocks where the true LCBS thickness was less than 2 m, the block grades for Cu and Ag were diluted using the grades estimated in the hanging wall (26), and the block percentages were adjusted accordingly.

The copper grade distribution within the LCBS and the UCBS are presented in Figure 14.19 and Figure 14.20 respectively. The higher-grade copper resources are located in the western Measured Resource, with grades ranging from 1.5% to 2.5% Cu, and the eastern Indicated and Inferred Resource (Section 5) where grades are generally 1.5% to 2.0% Cu.

Figure 14.19: Copper Grade Distribution (diluted to 2 m) in the LCBS with Mineral Resource Classification

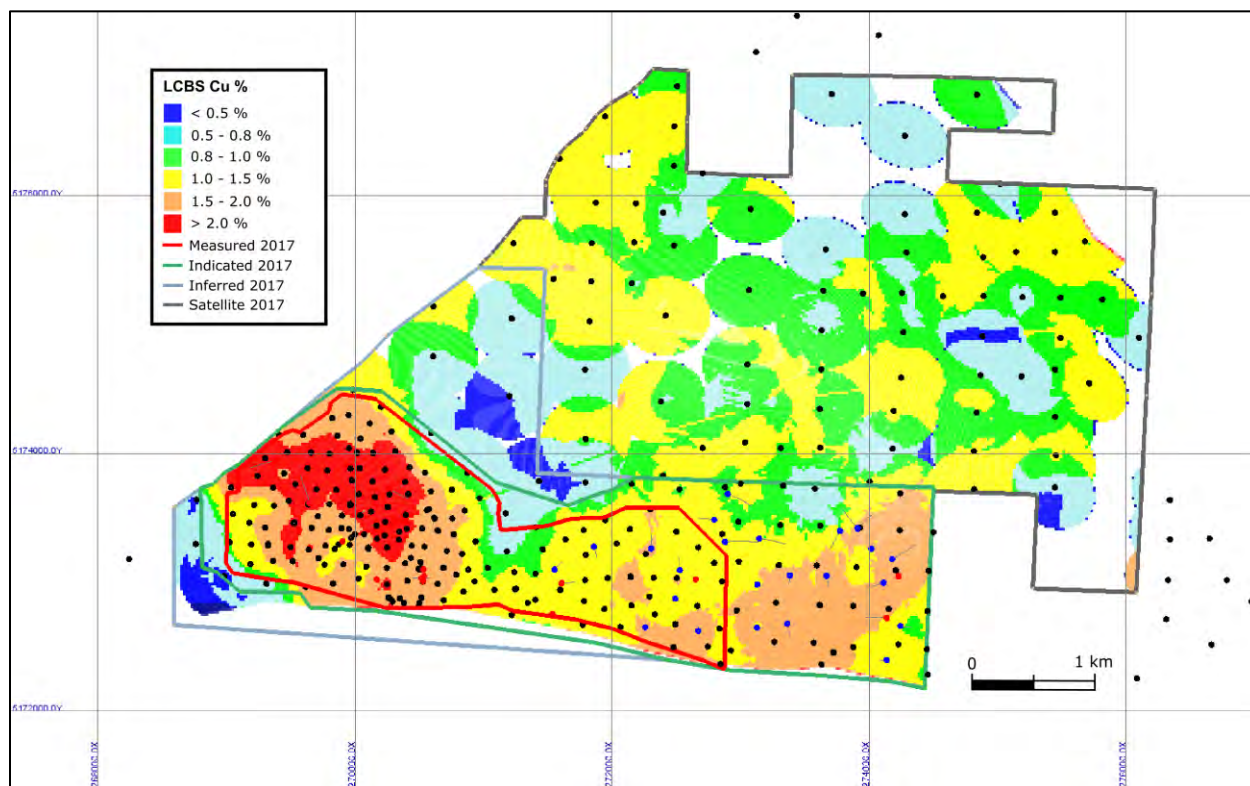
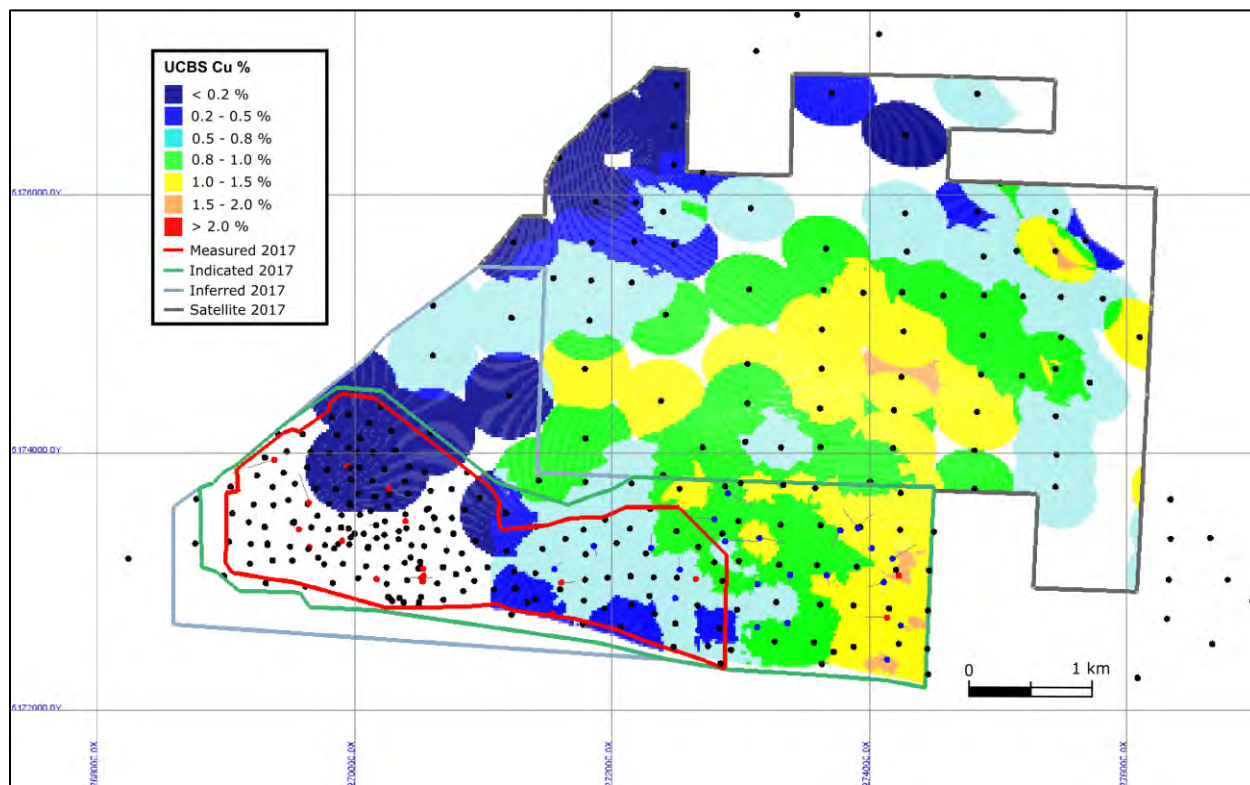


Figure 14.20: Copper Grade Distribution in the UCBS (2 m minimum thickness) with Mineral Resource Classification



Due to the minimum width of 2.0 m applied to the interpretation of the UCBS, only the far-eastern portion of Indicated Resources is above a grade of 1% Cu. The UCBS is not sampled or logged above the Main Zone of the Copperwood deposit.

14.10.2 Constrained Underground Mineral Resources Sensitivity - LCBS

Table 14.18, Table 14.19 and Table 14.20 summarize the sensitivity of the constrained underground Mineral Resources of the LCBS for the Copperwood and Satellite Deposits for a series of selected cut-offs. The sensitivity analysis uses cut-off grades between 0.8% and 2.0% Cu. For the Copperwood deposit, minimal tonnage (3.3 Mt) is gained when using a cut-off grade of 0.8% instead of 1.0% Cu for Measured and Indicated Resources. On the contrary, in the satellite deposits, a significant proportion (28.3 Mt) for the LCBS grades between 0.8% and 1.0% Cu.

Figure 14.21 and Figure 14.22 illustrate grade-tonnage curves for the Measured and Indicated Resources and Inferred for the LCBS of the Copperwood Deposit. Figure 14.23 illustrates grade-tonnage curves for the Inferred Resources for the LCBS of the Satellite Deposits.

Table 14.18: LCBS Constrained Mineral Resource Sensitivity – Measured and Indicated

Cut-off Grade (% Cu)	Copperwood Deposit - Measured & Indicated				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	6.9	2.15	327	7.17	1.6
1.5%	25.1	1.83	1,008	4.99	4.0
1.0%	42.2	1.60	1,488	3.84	5.2
0.8%	45.5	1.55	1,554	3.66	5.4

Table 14.19: LCBS Constrained Mineral Resource Sensitivity - Inferred

Cut-off Grade (% Cu)	Copperwood Deposit - Inferred				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	-	-	-	-	-
1.5%	0.1	1.66	2	7.74	-
1.0%	1.6	1.18	43	1.55	0.1
0.8%	3.2	1.04	74	0.94	0.1

Table 14.20: LCBS Constrained Mineral Resource Sensitivity – Satellite Inferred

Cut-off Grade (% Cu)	Satellite Deposit - Inferred				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	-	-	-	-	-
1.5%	0.3	1.56	9	0.39	-
1.0%	34.4	1.17	888	2.29	2.5
0.8%	62.7	1.05	1,456	2.76	5.6

Figure 14.21: Grade-Tonnage Curve of Measured + Indicated Resources for the LCBS at the Copperwood Deposit

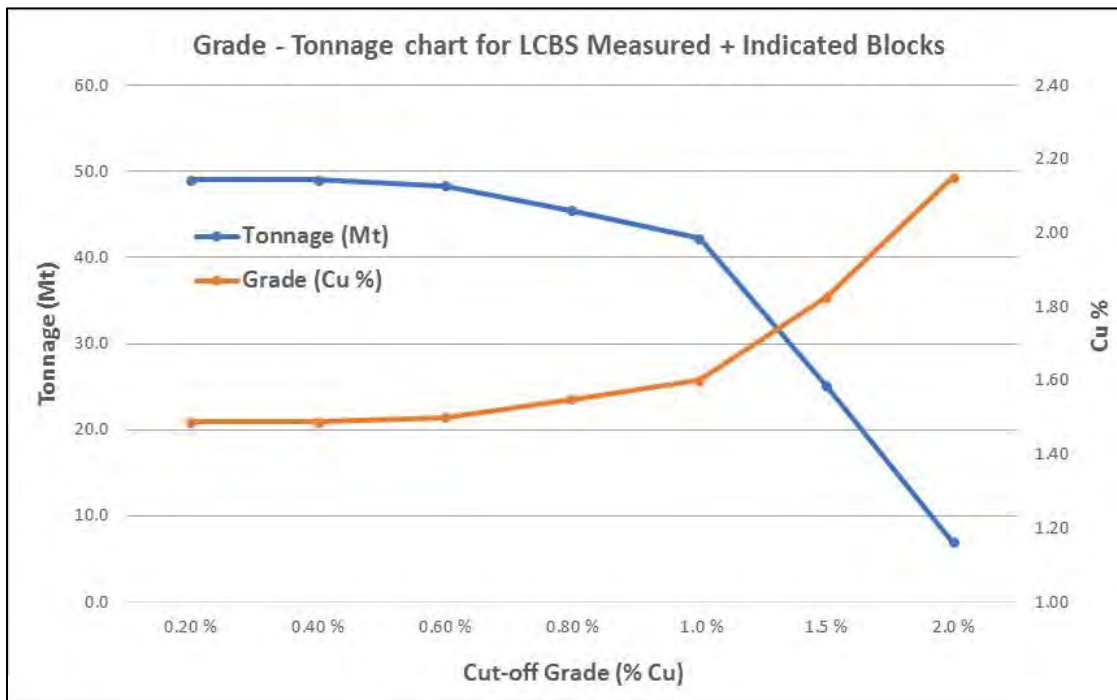


Figure 14.22: Grade-Tonnage Curve of Inferred Resources for the LCBS at the Copperwood Deposit

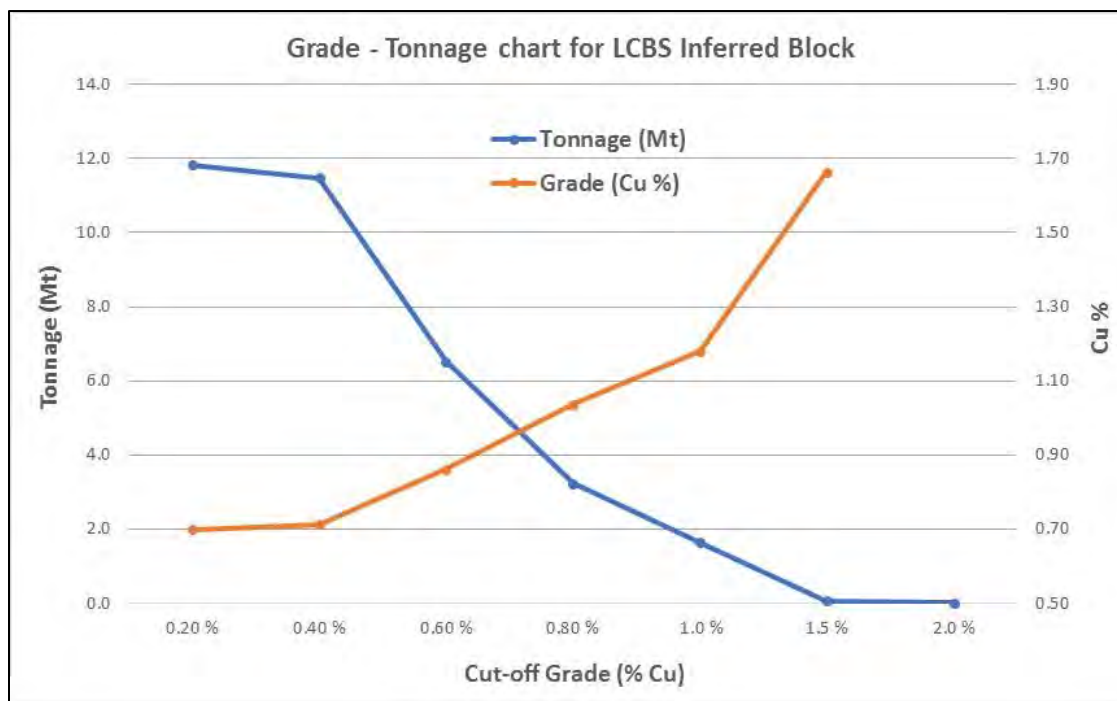
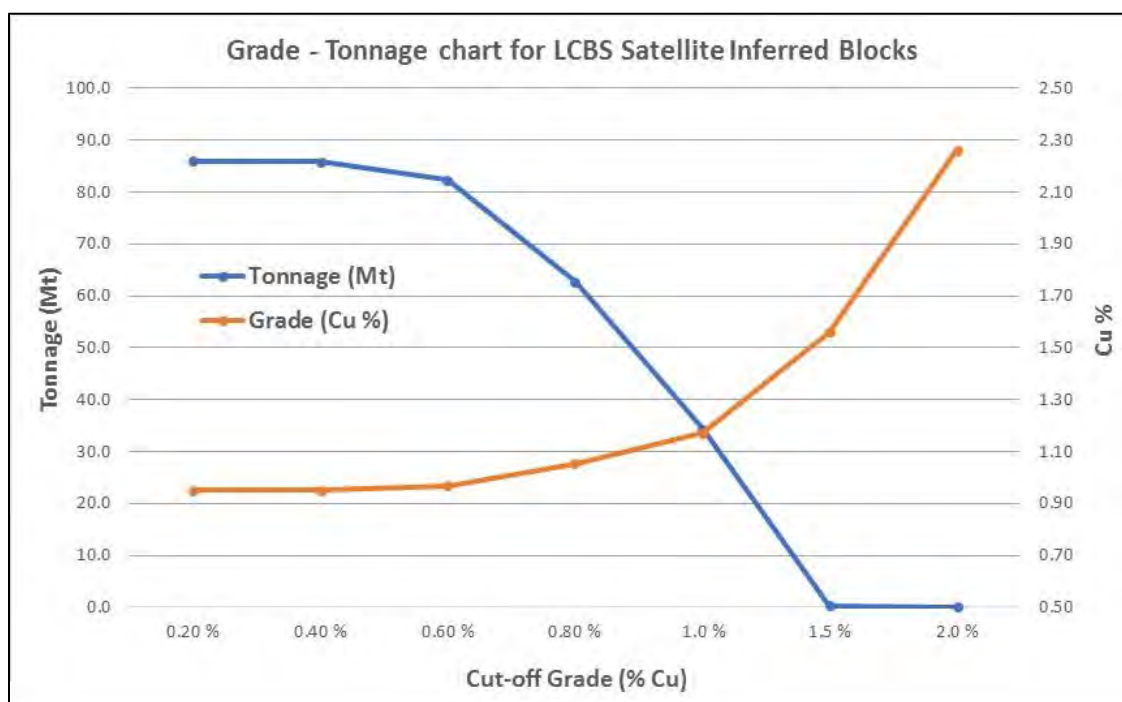


Figure 14.23: Grade-Tonnage Curve of Inferred Resources for the LCBS at the Satellite Deposits

14.10.3 Constrained Underground Mineral Resources Sensitivity – UCBS

Table 14.21, Table 14.22 and Table 14.23 summarize the sensitivity of the constrained underground Mineral Resources of the LCBS for the Copperwood and Satellite Deposits for a series of selected cut-offs. The sensitivity analysis is using cut-off grades between 0.8% and 2.0% Cu. As seen in the satellite deposits, a significant proportion (18 Mt) for the UCBS grades between 0.8% and 1.0% Cu.

Figure 14.24 and Figure 14.25 illustrate grade-tonnage curves for the Measured and Indicated Resources and Inferred for the LCBS of the Copperwood Deposit. Figure 14.26 illustrates grade-tonnage curves for the Inferred Resources for the LCBS of the Satellite Deposits

Table 14.21: UCBS Constrained Mineral Resource Sensitivity – Measured and Indicated

Cut-off Grade (% Cu)	Copperwood Deposit - Measured & Indicated				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	-	-	-	-	-
1.5%	0.4	1.54	14	4.04	0.4
1.0%	7.1	1.21	189	3.26	0.7
0.8%	13.7	1.06	319	3.03	1.3

Table 14.22: UCBS Constrained Mineral Resource Sensitivity - Inferred

Cut-off Grade (% Cu)	Copperwood Deposit - Inferred				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	-	-	-	-	-
1.5%	-	-	-	-	-
1.0%	-	-	-	-	-
0.8%	0.5	0.87	10	2.59	-

Table 14.23: UCBS Constrained Mineral Resource Sensitivity – Satellite Inferred

Cut-off Grade (% Cu)	Satellite Deposit - Inferred				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
2.0%	-	-	-	-	-
1.5%	0.4	1.58	15	3.71	0.1
1.0%	15.5	1.12	384	5.92	3.0
0.8%	33.5	1.01	749	5.23	5.6

Figure 14.24: Grade-Tonnage Curve of Measured + Indicated Resources for the UCBS at the Copperwood Deposit

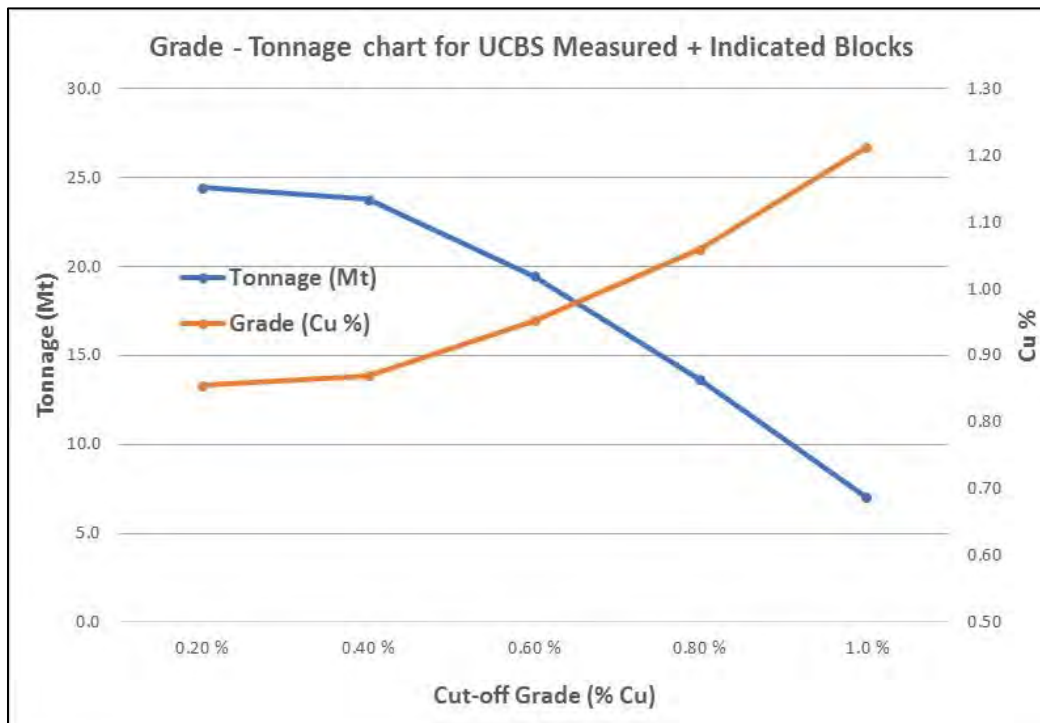


Figure 14.25: Grade-Tonnage Curve of Inferred Resources for the UCBS at the Copperwood Deposit

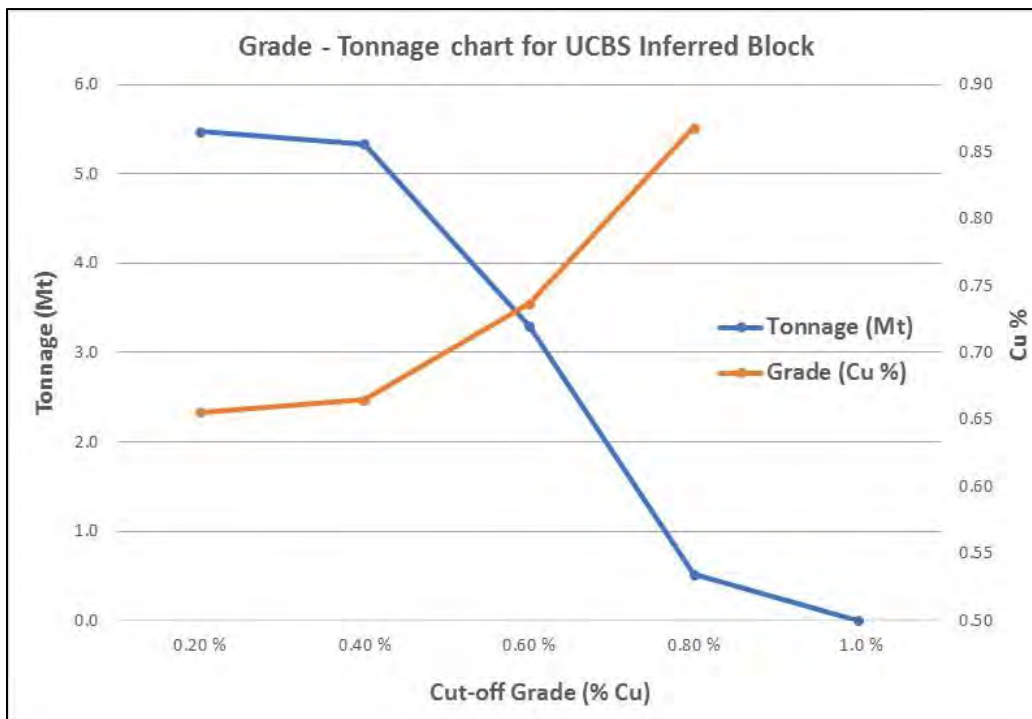
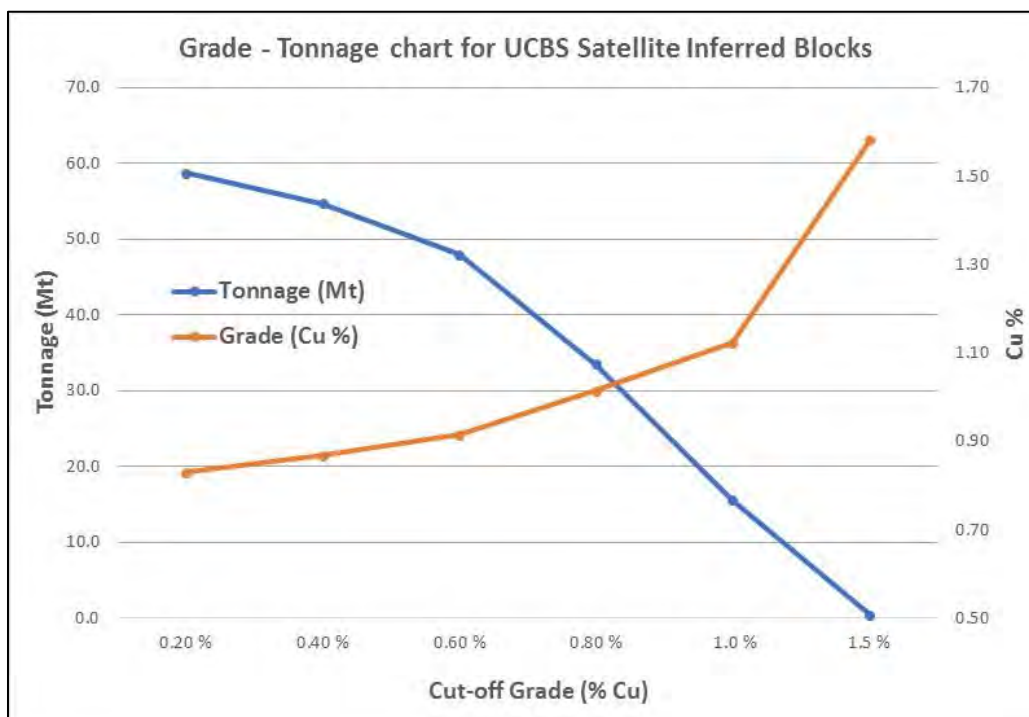


Figure 14.26: Grade-Tonnage Curve of Inferred Resources for the UCBS at the Satellite Deposits

14.11 Underground Constrained Resources

To establish a Mineral Resource estimate, an underground Room and Pillar (R&P) mining scenario is judged to be the most adapted to the geometry and dip of the LCBS, as well as to the tonnage of the deposits. To assess reasonable prospects of economic extraction by underground mining, GMSI considered several parameters such as concentrate prices, process recoveries, operating costs and mining costs to evaluate a copper cut-off grade. All blocks below this cut-off grade were removed from the constrained Mineral Resources. As mentioned, a minimum mining height of 2.0 m was used to dilute the resource grades.

14.11.1 Underground Optimization Parameters

The following conceptual mining parameters were considered:

- An NSR sliding scale royalty is applicable and equivalent to 3% at US\$3.00/lb;
- No mining loss and no mining dilution was considered at this stage for the Mineral Resources;
- Mineral Resources are reported using a copper price of US\$3.00/lb and a silver price of US\$18/oz;
- Metallurgical recovery of 86% for copper and 73.4% for silver;

- A payable rate of 96.5% for copper and 90% for silver was assumed;
- A cut-off grade of 1.0% Cu was used to report the Mineral Resources;
- Operating costs are based on a processing plant located at the Copperwood site.

14.11.2 Underground Mineral Resource Estimate

Copperwood Deposit total underground R&P Measured and Indicated Mineral Resources are reported at 49.3 Mt grading an average 1.54% Cu and 3.76 g/t Ag containing 1.68 Blbs Cu and 5.9 Moz Ag using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined. Inferred Mineral Resources are reported at 1.6 Mt grading an average 1.18% Cu and 1.55 g/t Ag containing 43 Mlbs Cu and 0.1 Mozs Ag using a cut-off grade of 1.0% Cu.

The Satellite Deposits total underground R&P Inferred Mineral Resources are reported at 49.9 Mt grading 1.15% Cu and 3.42 g/t Ag containing 1.27 billion pounds of copper and 5.5 million ounces of silver using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined.

Table 14.24 reports Mineral Resources for an underground R&P mining scenario for the Copperwood and Satellite Deposits by resource categories. All parameters used in the calculations are presented in the table's notes.

Table 14.24: Mineral Resource Estimate - Copperwood Project
1.0% Cu Cut-off Grade – April 30th, 2018

Deposits	Resource Category	Tonnage (Mt)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs)	Silver Contained (M oz)
LCBS	Measured	27.3	1.68	4.58	1,009	4.0
	Indicated	14.9	1.46	2.47	479	1.2
	M + I	42.2	1.60	3.84	1,488	5.2
	Inferred	1.6	1.18	1.55	43	0.1
UCBS	Measured	-	-	-	-	-
	Indicated	7.1	1.21	3.26	189	0.7
	M + I	7.1	1.21	3.26	189	0.7
	Inferred	-	-	-	-	-
Satellite LCBS	Inferred	34.4	1.17	2.29	888	2.5
Satellite UCBS	Inferred	15.5	1.12	5.92	384	3.0

Notes on Mineral Resources:

- 1) Mineral Resources are reported using a copper price of US\$3.00/lb and a silver price of US\$18/oz.
- 2) A payable rate of 96.5% for copper and 90% for silver was assumed.
- 3) The Copperwood Feasibility Study reported metallurgical testing with recovery of 86% for copper and 73.5% for silver.
- 4) Cut-off grade of 1.0% copper was used, based on an underground "room and pillar" mining scenario.
- 5) Operating costs are based on a processing plant located at the Copperwood site.
- 6) Assuming a US\$3.00/lb Cu price, a sliding scale 3.0% NSR royalty on the Copperwood Project is payable to leaseholders. Assuming closing of the acquisition of the White Pine Project, a 3% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd is reduced to a 1.5% NSR royalty.
- 7) Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.
- 8) No mining dilution and mining loss were considered for the Mineral Resources.
- 9) Rock bulk densities are based on rock types.
- 10) Classification of Mineral Resources conforms to CIM definitions.
- 11) The qualified person for the estimate is Mr. Réjean Sirois, P.Eng., Vice President Geology and Resources for GMSI. The estimate has an effective date of 30th April 2018.
- 12) Mineral Resources that are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- 13) LCBS: Lower Copper Bearing Sequence.
- 14) UCBS: Upper Copper Bearing Sequence.
- 15) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.

15. MINERAL RESERVE ESTIMATES

The Mineral Reserves for the Copperwood Project are estimated at 25.4 Mt, at an average grade of 1.43% Cu and 3.83 g/t Ag, as summarized in Table 15.1. The Mineral Reserve estimate was prepared by GMSI. The resource block model was also generated by GMSI.

The mine design and Mineral Reserve estimate were completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste.

Table 15.1: Mineral Reserve Estimate - Copperwood Project

Reserve by Category	Tonnes (Mt)	Cu Grade (%)	Ag Grade (g/t)	Cu contained (M lbs)	Ag contained (M oz)
Proven	17.5	1.50	4.43	579.6	2.5
Probable	7.9	1.28	2.5	222.2	0.6
Proven & Probable	25.4	1.43	3.83	801.8	3.1

Notes:

- 1) The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.
- 2) Mineral Reserves are estimated at a cut-off grade of 1% Cu. The cut-off will vary depending on the economic context and the operating parameters.
- 3) Mineral Reserves are estimated using a long-term copper price of US\$3.00/lb and a silver price of US\$16.00/oz.
- 4) Assuming a US\$3.00/lb Cu price, a sliding scale 3.0% NSR royalty on the Copperwood Project is payable to leaseholders. Assuming closing of the acquisition of the White Pine Project, a 3% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd is reduced to a 1.5% NSR royalty.
- 5) Mineral Reserves are estimated using an ore loss of 3%, a dilution of 0.1 m for the floor and a 0.25 m for the back of the stope and the development.
- 6) The economic viability of the mineral reserve has been demonstrated.
- 7) A minimum mining height of 2.1 m was used.
- 8) The copper recovery was estimated at 86%.
- 9) The qualified person for the estimate is Mr. Carl Michaud, Eng., Underground Engineering Manager for GMSI. The estimate has an effective date of May 25, 2018
- 10) The number of metric tonnes was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.

15.1 Estimation Procedures

The resource block model ENG_APR18 described in this Report was used for the mineral reserve conversion process. The “percentage” type block model performed with the Geovia GEMS™ software (version 6.7.4) was converted to a Datamine™ Sub-block model type. To do this conversion the Deswick.Cad™ (version 2017.2) software was used. A new sub-blocked model created for each rock type relied on the wireframe presented in Section 14.2.2. The original block model was sub-blocked by 20 to a minimum size of 1 m East x 1 m North x 0.125 m to have maximum precision according to the wireframe resolution. This division by 20 is the Deswick.Cad™ software’s maximum possible division. All sub-blocks are subsequently merged together to create a unique block model. Table 15.2 compares the two model blocks and the percentage block conversion to the sub-blocked model.

Table 15.2: Resource Model versus Mining Model

Rock Code	Description	ENG_APR18			Sub-block Model			Variation		
		Tonnage	Cu	Ag	Tonnage	Cu	Ag	Tonnage	CU	AG
		(kt)	% Cu	(g/t)	(kt)	% Cu	(g/t)	%	%	%
11	FW Dilution	8,951.63	0.10	1.95	8,940	0.10	1.95	0.13	0.83	0.09
23	Domino	22,770.93	2.25	4.24	22,770	2.26	4.29	0.00	0.37	1.11
24	Red Massive	11,160.27	0.31	1.42	11,160	0.31	1.42	0.01	-0.89	0.26
25	Grey Laminated	35,403.54	1.14	2.88	35,403	1.14	2.88	0.00	0.10	0.01
26	HW dilution	9,529.43	0.37	1.25	9,528	0.37	1.25	0.01	0.08	-0.18
27	Red Laminated	30,803.45	0.17	0.34	30,803	0.17	0.34	0.00	0.68	-0.27
28	UCBS	19,926.04	1.49	5.69	19,926	1.49	5.69	0.00	-0.20	-0.01

Once the model block is produced, the mine design is created according to the process described in Figure 15.1. The entire UCBS unit was removed from the reserve calculation for this Study. Currently, it is more cost-effective to only mine the LCBS unit rather than the UCBS unit alone or in combination with the LCBS unit. An economic outline (Table 15.2) is determined, considering the cut-off grade in Section 15.2, the minimum mining height and the mine dilution. All tonnage outside of this outline is removed since it does not meet the economic criteria for room and pillar stoping. The mineral reserve is net of all pillars including those in the mine panels, the Lake Superior 30 m offset, a crown pillar providing for 25 m vertical of rock above openings and a 15 m barrier pillar around the historical test mine openings.

A height of 0.3 m is also removed from the Mining Reserves to meet Golder's geotechnical recommendations. This 0.3 m of laminated gray remains in place to allow for better control over the red

laminated unit. Dilution is then added to the floor to ensure a minimum height of 2.1 m. This dilution is added to the floor to keep the 0.3 m of gray laminated to the back of the stope. Once all these manipulations are carried out, the design of the stope and the drift is completed, considering the efficiency, the limit of the equipment and the geotechnics. The size of the pillars to be maintained is described in Section 16 of this Report. Once the economic design has been completed, the pillar tonnages are removed from the reserve calculation and an unplanned dilution is added as described in Section 15.3. Finally, a mining recovery factor of 97% is applied to reach the mining reserve.

Figure 15.1: Conversion from Resources M+1 to Mining Reserves Proven and Probable

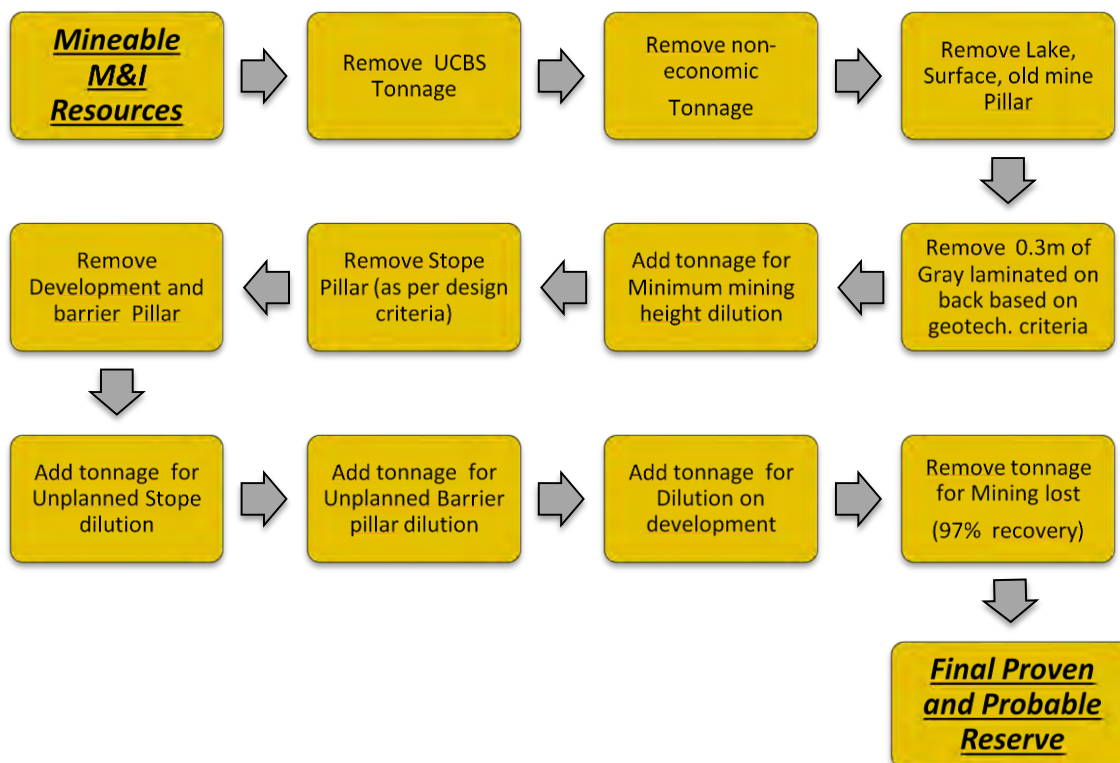
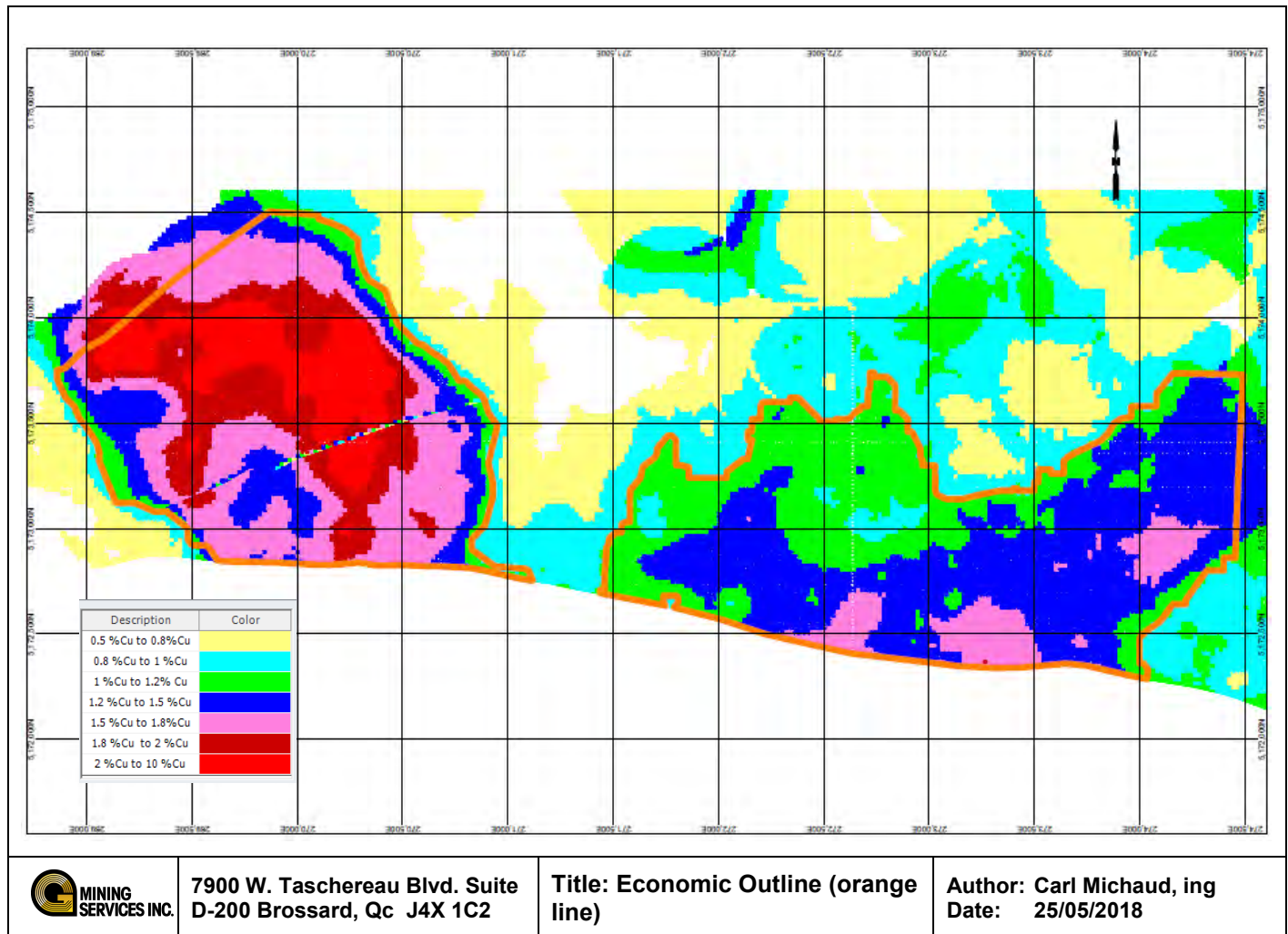


Figure 15.2: Economic Outline (orange line)

15.2 Cut-Off Grade

To calculate the portion of exploitable reserves of the measured and indicated resource, the economic part of the resource needs to be identified. To achieve this a cut-off grade including the dilution is calculated. The cut-off grade must include mining dilution and mining recovery. These factors take into consideration the mining method and the deposit's characteristics.

The following economic parameters for a production rate of 6,600 mtpd are estimated to determine the copper equivalent cut-off grade with regards to the Copperwood Project:

Table 15.3: Underground Room and Pillar Mining Method

Description	Unit of Measure	Price (US\$)
Metal Prices		
Copper	\$/lb	3.00
Silver	\$/oz	16.00
Process Recovery		
Copper	%	86.0
Silver	%	73.4
Effective Payable Rate		
Copper		96%
Silver		54%
Silver Credit Net of Refining	\$/t ore	0.70
Operating Costs		
Operating Costs		
Processing	\$/t ore	12.57
G&A	\$/t ore	3.10
Sustaining CAPEX	\$/t ore	8.17 *
UG Mining Costs	\$/t ore	21.60
Royalties	\$/t ore	2.18
Total	\$/t ore	47.62
Cut-Off Grade	%Cu	1.0

Note: *Includes Mine Sustaining

15.3 Ore Recovery and Mine Dilution

Dilution is defined as the ratio of waste to mineralized material. There are two types of dilutions anticipated in the Copperwood Project:

- Internal dilution, also known as planned dilution;
- External dilution, also known as unplanned dilution.

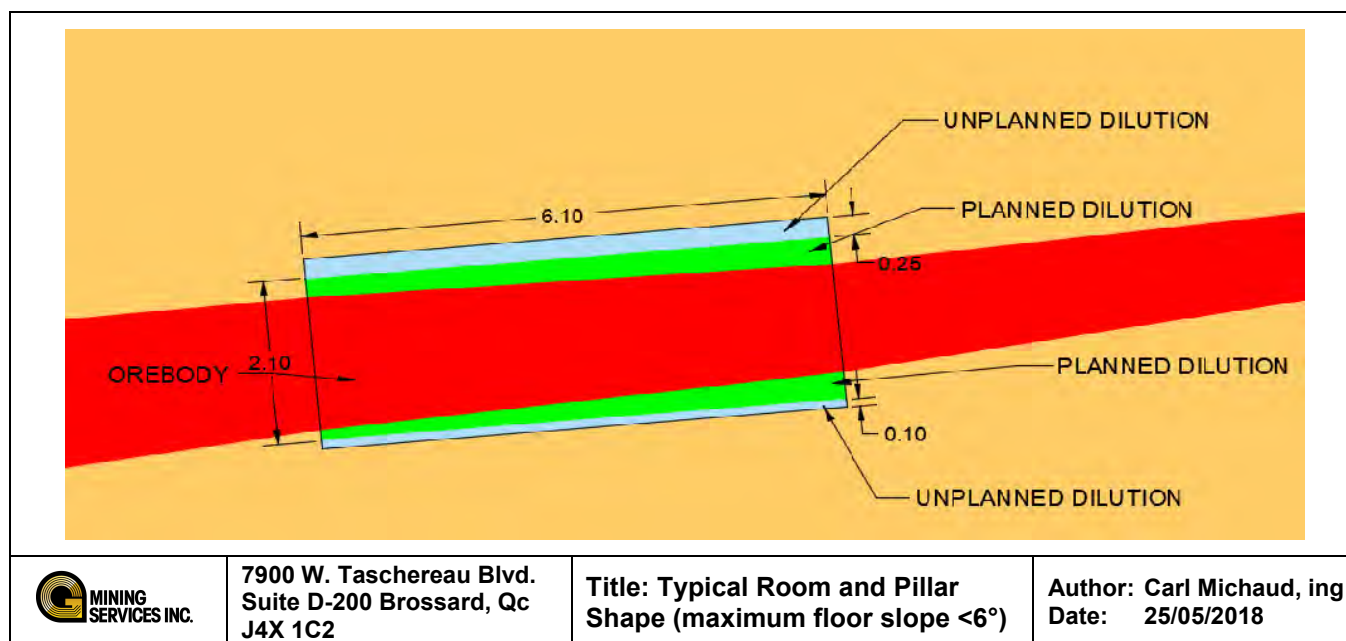
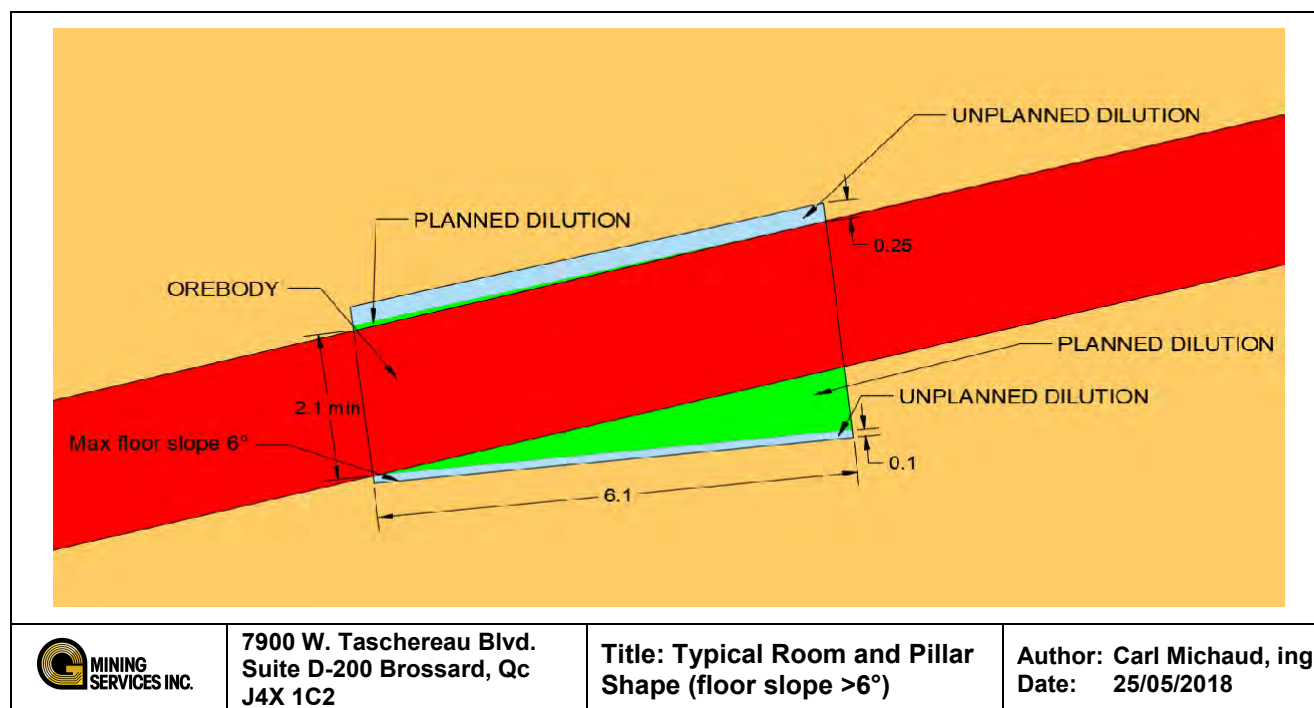
The dilution grade assigned depends whether the material is inside the block model or not. If the material is inside the block model, the grade value of the block model is given. Should the material be outside of the block model, a grade of 0.15 % Cu and 1.5 g/t Ag is specified. This represents the average grade of the block model around the orebody. This approach may appear conservative; however, it is the QP's opinion, based on the information available and the knowledge of the rock types, that it is appropriate at this stage of the Project.

Internal Dilution (planned dilution)

Planned dilution is the part of the dilution included in the stope design. Two scenarios can create dilution in the mining rooms with regards to the Copperwood Project. In the first scenario, this dilution is added to reach a minimum mining height of 2.1 m. In the case of development, the minimum mining height is 3 m. In the second scenario, the dilution is added to the floor so as to not exceed 6° of side dip.

External Dilution (unplanned dilution)

Unplanned dilution is the part of the dilution that is outside of the mining room design. This dilution is the over break of the excavation. It can be caused by several factors; bad blasting practice, unfavorable geological structure, etc.

Figure 15.3: Typical Room and Pillar Shape (maximum floor slope <6°)**Figure 15.2: Typical Room and Pillar Shape (floor slope >6°)**

No dilution has been added to the excavations' walls, given that these are still located in the ore. A 3% ore loss was applied to estimate the final reserve. The ore loss factor is to provide for ore lost from stopes due to geotechnical issues and for tonnage left in place around the pillars.

15.4 Minimum Mining Height.

The minimum mining height is primarily a function of the orebody's geometry. The selection of a room and pillar mining methods and the choice of mining equipment will allow mining to reach a minimum of 2.1 m mining height before dilution. This minimum height allows the production equipment to move easily and the drilling equipment to have enough space for the operation.

Figure 15.4: Minimum Mining Height

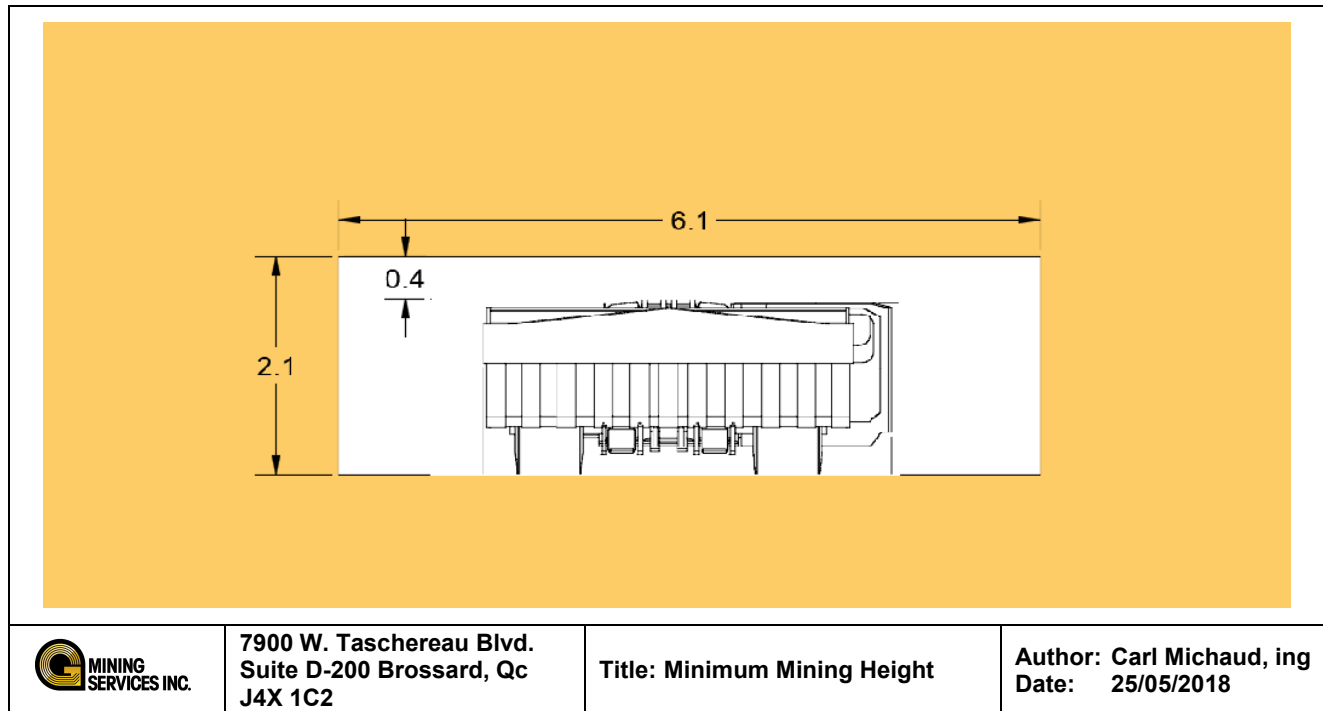


Table 15.4 presents a summary of the overall mining dilution and mining recovery factors included in the mineral reserves.

Table 15.4: Mining Recovery and Dilution Summary

Mining Reserve Dilution and Recovery factors	Reserve (Mt)	Orezone Height (m)	Mining Recovery	Dilution		
				Total	Planned	Unplanned
West Zone	10.20	2.80	69.8%	14.3%	1.8%	12.6%
East Zone	11.50	1.76	74.0%	46.4%	30.5%	15.9%
Barrier Pillar	1.13	2.49	65.0%	20.3%	8.3%	12.0%
Total Room-and-Pillar	22.84	2.26	71.7%	30.8%	16.6%	14.2%
Development	2.55	2.49	65.0%	70.4%	58.7%	11.7%
Total Ore	25.39	2.28	71.0%	34.8%	20.8%	14.0%

15.5 Factors Possibly Affecting Mineral Reserves

Areas of uncertainty that may materially impact the mineral reserve estimates include the following:

- Commodity prices, market conditions and foreign exchange rate assumptions;
- Costs assumptions, particularly cost escalation;
- Geological complexity and continuity;
- Dilution and recovery factors;
- Geotechnical assumptions concerning rock mass stability;
- Hydrogeological assumptions concerning water seepage;
- Licenses with third parties;
- Cut-off NSR estimations;
- Capital and operating cost assumptions;
- Geological complexity and resource block modelling;
- Slope stability, dilution and mining recovery factors;
- Metallurgical recoveries and contaminants;
- Rock mechanics (geotechnical) constraints and the ability to maintain constant underground access to all working areas;
- In situ stress in the rock. Currently no in situ stress measurements were performed in the Copperwood area. These measurements should be made as soon as the development is sufficiently

advanced for the tests to be representative. Changes in extraction sequence and pillar size may be required if higher in situ constraints are indicated in this program;

15.6 Comments

As of the effective date of this Report, the QP is unaware of any risks, legal, political or environmental factors that would materially affect the potential development of the Mineral Reserves.

16. MINING METHODS

16.1 Introduction

The proposed mining method for the Copperwood Project is conventional drill and blast room-and-pillar given the relatively sub-horizontal orebody that varies in thickness from 1.6 m to 3.7 m.

The method consists of the extraction of a series of entries and cross-cuts in the ore leaving pillars in place to support the back. The entries cross cuts and pillars are sized using a geotechnical analysis of the rock, and experience from other mines sharing similar ground conditions.

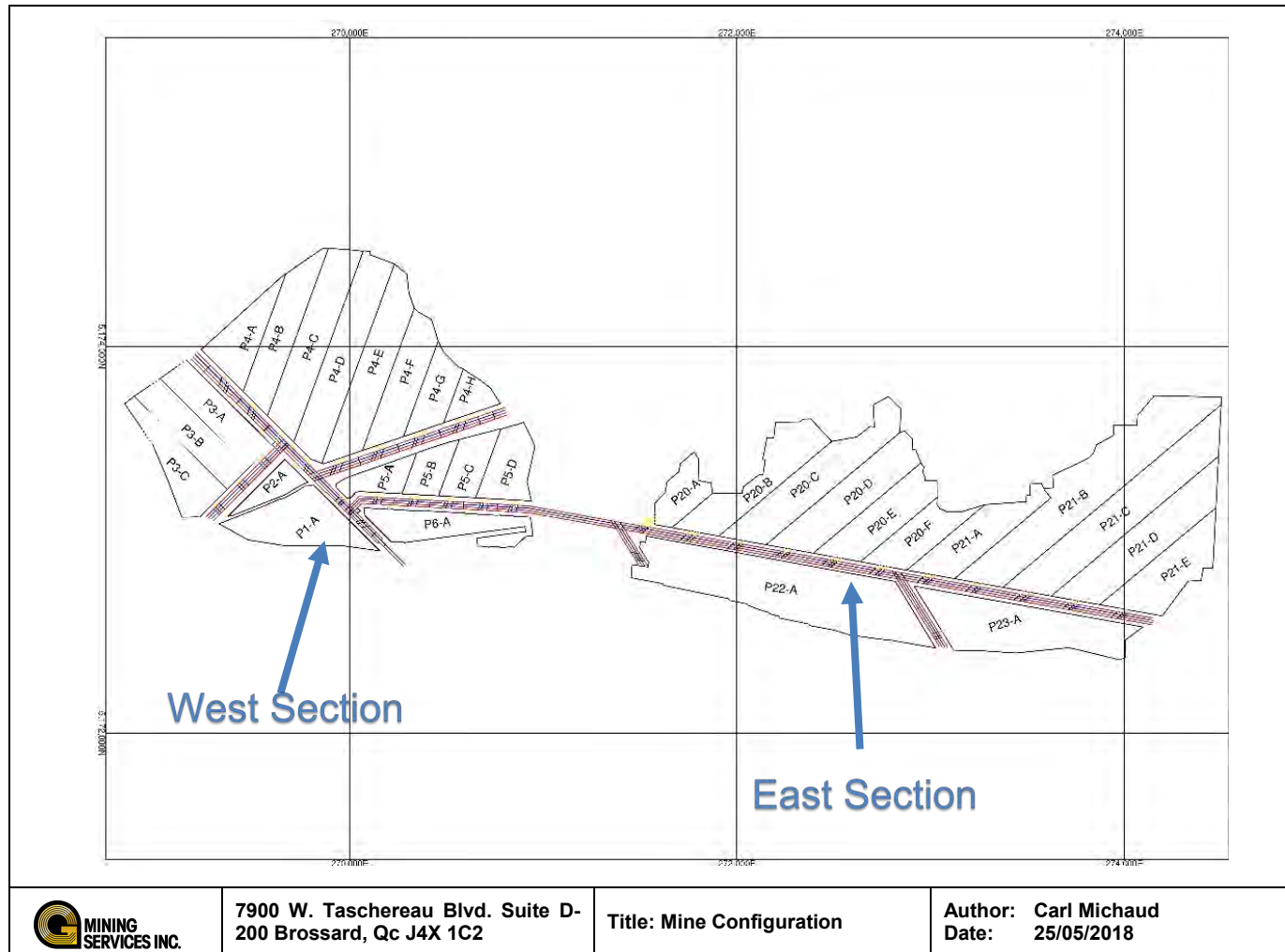
The Project's mining equipment consists of a low-profile two-boom electric-hydraulic jumbo for drilling. A one-boom electric-hydraulic low-profile bolter is considered for the installation of ground support. A load haul dump ("LHD") unit with a 10 t (6 yd³) capacity is planned for ore removal from the face and transport of the broken ore to a rock breaker-loading point. A rock breaker will reduce the size of larger particles in the blasted ore, which will be placed on a belt conveyor and transported to the surface crushed ore storage bins from which the mill is fed.

Main accesses and haulage of ore from certain distant working areas are developed using 30 t underground mining trucks to transport the ore to the rock breaker or to the surface stockpile. A mix of ANFO and emulsion explosives are used for blasting to reduce the excavation overbreak. The rooms are mined with a single pass approach, such that the pillars will immediately have their final dimensions. This approach is recommended for better control and better productivity.

The mine is comprised of two sectors; the Eastern part and the Western part. The Western part contains higher grades and a thicker mineralized zone. For these reasons, mining will begin in the western part which is subdivided into 6 extraction panels as detailed in Figure 16.1. The East part is subdivided into 4 extraction panels; panels 20 to 23. The mining direction will generally follow the dip of the orebody, but in some areas the dip is too steep to follow. In the areas where the dip is too steep, the mining will be done at an angle to the dip direction.

Mining the UCBS as a full column with the LCBS was evaluated but was not retained. This option was possible at the east end of Section 5. It was deemed less economical than the option of mining only the LCBS unit. This Feasibility Study focuses on the LCBS unit to generate better operating margins.

Figure 16.1: Mine Configuration



16.2 Geotechnical Considerations

A detailed geotechnical evaluation of the Copperwood deposit was completed by Golder in 2018 and established many of the mine design criteria, in particular the pillar design which affects the mine recovery factor. The following subsections summarize the Golder geotechnical assessment.

16.2.1 Geotechnical Background

Historically, mining took place in the region at the White Pine mine. The White Pine mine was in operation from 1955 to 1995 as a room and pillar operation. Conditions in the mine are reported as variable, depending on the proximity to major structures and the syncline axis. For the most part, back conditions were observed to be good where the back was formed in sandstone. In general, back stability issues were a problem in an area of faulting that was exacerbated by high horizontal stresses. Previous studies and literature about the Copperwood deposit make many assumptions about the expected performance of the proposed Copperwood mine based on experience at White Pine. However, it is important to note that there are many key differences between White Pine and Copperwood.

These include the following:

- The geology in the back of the mining horizon was very different at White Pine than what is proposed at Copperwood;
- There is a much thicker sequence of parting shale at Copperwood. Consequently, the back will be formed in thinly laminated siltstone and shale (Red Laminated) at Copperwood, whereas the back was generally formed in high quality sandstone at the White Pine Mine;
- There is no confirmation that the same high horizontal stress field present at White Pine is present at Copperwood;
- A zone of shearing and associated gouge has been identified at the lower contact of the Copper Bearing Sequence (“CBS”) with the underlying sandstone at Copperwood. There was no Basic Shear Zone (“BSZ”) at the base of the White Pine deposit.

A test mine was developed at the Copperwood deposit in the 1950's. The test mine consisted of a 230 ft (70 m) deep shaft with approximately 2,000 ft (610 m) of lateral drift development and two test stopes. The test mine is located at the western part of the orebody. Significant stability issues were experienced in the test mine.

Reports at the time covered observations and experiences in the test mine. A summary of the documented observations is as follows:

- The back conditions were described as *“thin, weakly bonded shale has very little inherent strength and is cut up by jointing, faults, slumpage structures, and numerous incipient fractures which intersect the bedding at various angles”* (Lambly 1958);
- The development was bolted to the face however *“...after a few rounds the roof started to break up and scale off in large patches. The shale crumbled around the roof bolts rendering them ineffective”*. In areas where a portion of the ore was left in the back (i.e. probably the back was formed in the grey laminated (“GLAM”), more suitable conditions were reported: *“...a very stable back and required very little re-scaling of the roof”* (USMR, 1958);
- Test holes in the back indicated differential movement across the bedding planes in the shale (Lambly 1958);
- Joints were often observed to be dripping with water (USMR, 1958);
- A dominant set of sub-vertical joints striking N80W spaced at up to 8 ft (2,4 m) and a second less dominant set striking at N20W (USMR, 1958)
- Vertical fault zones were observed to generally strike N-S at a spacing of 100 to 400 ft (30.5 to 122 m) (USMR, 1958).

In one of the test stopes, the face of each round formed on a prominent E-W striking structure and a vertical N-S striking feature ran down the centerline of the drift (USMR, 1958). The following observations suggest relatively close spacing of sub-vertical structures:

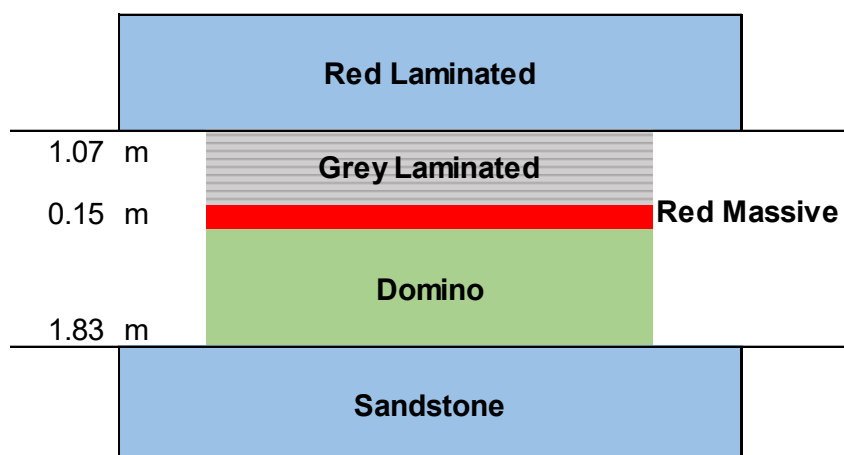
- One of the more notable observations was that significant noise occurred in the 4 to 8 hours after each round was taken (Lambly, 1958). *“It is reported that after each round in the stopes there is considerable noise which appears to be created by cracking or parting of the shales in the back. The noise, at times, becomes so pronounced that the miners will leave the area until the noise has subsided, which generally does not happen until four to eight hours after a round has been blasted”*.
- A soft gouge present along the bottom of the drift ribs that was observed to squeeze out into the drift (Lambly 1958). This gouge was observed along the length of the development at thicknesses of up to 6 in. (USMR, 1958).

In addition to the unique observation at Copperwood relating to the presence of the soft gouge material at the base of the deposit, it is very interesting that at such low depth and low stress state, extensive and dramatic back failures in the Red Laminated unit were experienced.

There are several reports and published papers from the time that suggest that these failures may be the result of poor quality bolting, high horizontal stresses, and the use of water during drilling of bolts that may have led to weakening of the RLAM. It is not clear, based on current knowledge, whether the observations could be attributed to these potential causes. Water was already observed leaking from joints (and the test mine would have been below the water table given the proximity to the lake) so the introduction of drilling water should not have been impactful. Bolting may not have been effective given the methods at the time. However, even if the back was not bolted at all, the reported back failures would not have been anticipated based on the shallow depth of the test mine. At 220 ft (67 m) depth, the maximum principal stress should be no greater than ~750 psi (if a high 3 to 1 horizontal to vertical stress ratio is assumed) and the strength of the RLAM rock forming the back is approximately 8,500 psi. This mechanism of failure needs to be more clearly understood to develop a defensible design for the proposed Copperwood mine. Significant effort has been put forth to reconcile these observations with the available data for the Copperwood deposit to allow for a detailed geotechnical characterization of the rock and appropriate considerations for the geotechnical design of the proposed mine.

16.2.2 Geotechnical Characterization

The strength of the pillars and the overall design of the proposed room-and-pillar mine will be governed by the strength and behaviour of the geological units in the pillars and in the immediate roof. The conceptualized stratigraphy in the ore and surrounding rock mass is shown below in Figure 16.2. Characterization focused on the units in the pillars (Domino, Red Massive, Grey Laminated) and the back (Red Laminated).

Figure 16.2: Idealized Pillar Stratigraphy

16.2.3 Available Geotechnical Data

The following drilling campaigns incorporated geotechnical data collection:

- In 2008, a delineation and infill drilling program collected RQD data, UCS testing data, PLT data and Young's Modulus results;
- During 2009 to 2011 geotechnical and metallurgical drilling investigations collected geotechnical data such as RQD, UCS, Young's Modulus, and structural data from televiewer logging in select drill holes;
- In 2013, Golder conducted a geotechnical drilling investigation which consisted of vertical and inclined drill holes to collect structural data (alpha and beta orientations) as well as total core recovery, fracture frequency, RQD, field strength estimates, joint roughness, joint conditions, UCS testing, Young's Modulus, Brazilian tensile strength testing, direct shear strength testing, and PLT testing;
- In 2017 geotechnical data such as RQD and PLT data was collected as part of a field investigation program. Samples were also collected and tested primarily in UCS by Advanced Terra Testing Inc (ATT). Specific laboratory strength tests such as direct shear on the Basal Shear Zone (BSZ), and triaxial testing and UCS testing on the DOMN unit.

16.2.4 Intact Rock Strength

A few laboratory testing campaigns were conducted on core samples from the Copperwood deposit. A summary of the UCS strength testing considered representative of the subunits of interest is presented in Table 16.1. Trends in the data suggest the following:

- Somewhat higher strength materials are present in the East orebody as compared to the same units in the West orebody;
- There is a subtle pattern in the data that suggests that the strength of the Domino decreases with proximity to the lower contact (i.e. it is stronger higher in the ore column). However, this trend is not evident in the available point load tests.

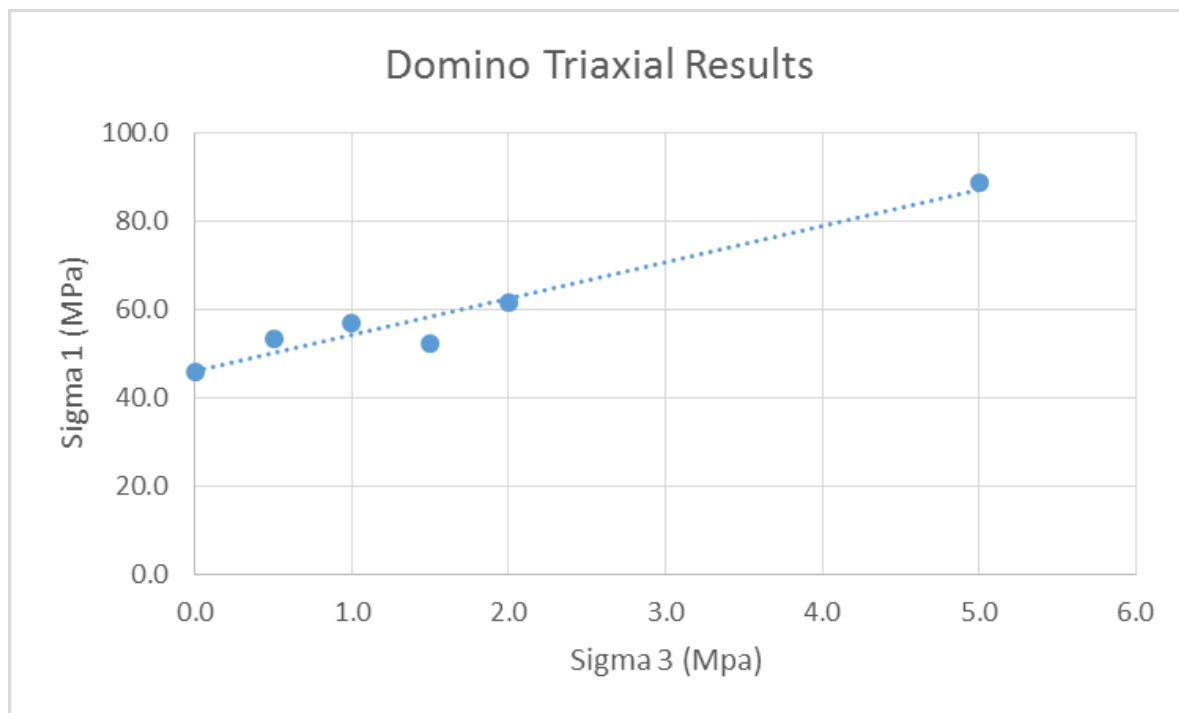
Table 16.1: UCS Testing Result Summary

Subunit	West Orebody		East Orebody	
	Number of Tests	Average UCS (psi)	Number of Tests	Average UCS (psi)
RLAM	17	8,550	16	10,600
GLAM	16	8,550	19	12,750
RMAS	11	10,600	5	12,900
DOMN	16	6,700	7	7,800

No triaxial testing data were available from previous work to estimate the influence of confinement on rock strength.

The Confinement Strength Factor (“CSF”) is very critical to pillar strength estimation and previous studies were lacking measurements. With a focus on the Domino unit (the weakest unit in the ore column), additional samples were obtained during the 2017 field program to allow for 5 triaxial compressive strength tests to be carried out over the stress range of interest for pillar design. The strength envelope for the Domino unit, based on these current laboratory results, is shown on Figure 16.3. The resulting friction CSF was estimated to be approximately 8 (equivalent to a friction angle of 51 degrees in a Mohr-Coulomb envelope). This same CSF was assumed for the other units of interest (Red Massive, Grey Laminated, Red Laminated).

It is important to note that the in-situ strength of the RLAM unit will be significantly influenced by the presence of weak laminations in the unit. UCS tests have typically been undertaken on dry samples with bedding oriented perpendicular to the loading axis and therefore these strength values should be considered upper limits.

Figure 16.3: Domino 2018 Laboratory Strength Testing Results

16.2.5 Rock Structure

Data on structural orientation was available from:

- Televiewer data collected in several holes from the 2009 and 2017 geotechnical drilling campaigns;
- Oriented core from the 2013 drilling campaign;
- Mapping of the Presque-Isle River;
- Observations in the test mine.

The structural information in the immediate ore zone is reasonably consistent between the different sources of data.

- The rock mass fabric is dominated by bedding;
- Bedding spacing is variable in the different units;
- Units have been named 'massive' or 'laminated' to distinguish between widely and thinly laminated rock, respectively;
- Laminated rock, particularly RLAM was observed to break easily along clay laminations when exposed to water. This is discussed in more detail later in this section.

A dominant set of sub-vertical joints striking N80W spaced at up to 2.4 m (8 ft) and a second less dominant set striking at N20W were observed in the test mine. These are consistent with structural data obtained from the drilling investigations. Two main thrust faults have been identified on the property. The first is a shallow thrust fault along the base of the Domino that has resulted in a variable thickness of sheared material and gouge along the base of the Domino. The second thrust fault, cuts across the deposit, striking ENE and dipping at approximately 30 degrees. This fault was originally identified by USMR and Orvana based on a repeated thickness of approximately 6.1 m (20 ft) of the strata in drill hole M57-W159. The condition of the fault in drill holes is not particularly adverse however the design of the mine will need to consider the changing dip of the orebody in proximity to the thrust fault.

There is no evidence to date of additional thrust faults on the property. However, identification of these shallow angle faults in drill holes is difficult. Additional drilling and/or mine development may identify one or more additional faults that will need to be considered in design.

16.2.6 Rock Quality

The rock generally has high values of rock quality designation ("RQD"), indicating relatively massive conditions. The average RQD reported as part of the 2017 geotechnical drilling campaign is approximately 88%. There were no extensive zones of lower RQD noted in the orebody or surrounding rock mass that would suggest poorer quality zones that would require a separate design.

16.2.7 Rock Mass Strength

The rock mass strength has been estimated at 25% of the intact strength. This is broadly equivalent to the rock mass strength estimated using a Hoek-Brown approach assuming a disturbance factor of zero (appropriate for underground excavations with good blasting practices).

Table 16.2: Rock Mass Strength Parameters

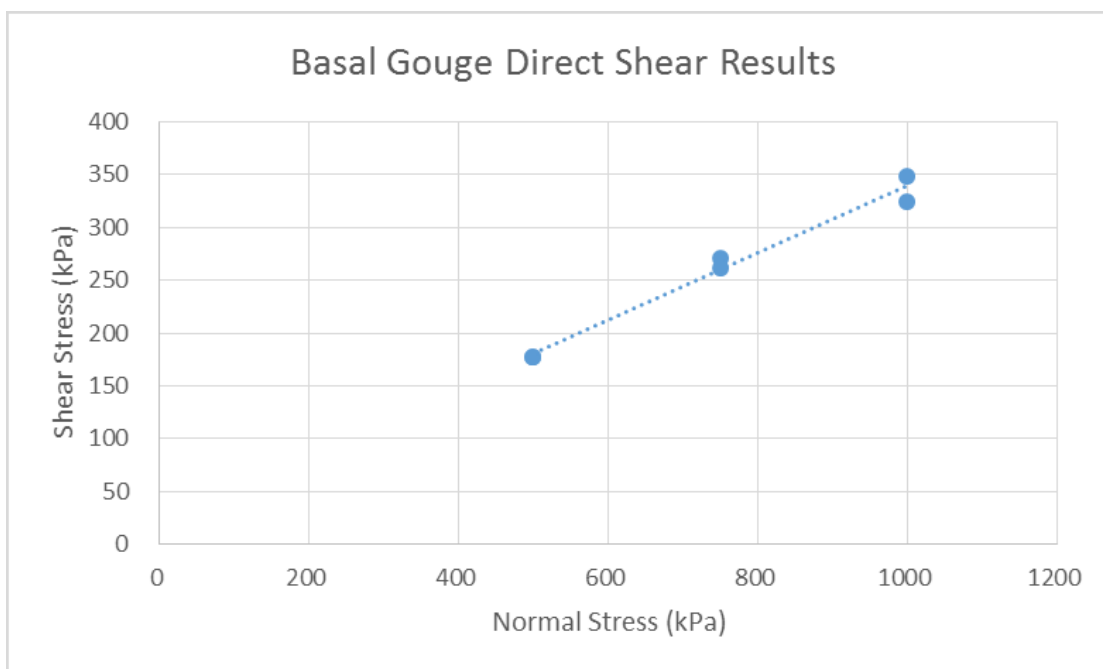
Subunit	West Orebody					East Orebody				
	UCS (psi)	Friction Angle	CSF	Cohesion (psi)	Tensile Strength (psi)	UCS (psi)	Friction Angle	CSF	Cohesion (psi)	Tensile Strength (psi)
RLAM	8,550	51	8	1510	151	10,600	51	8	1880	188
GLAM	8,550	51	8	1510	151	12,750	51	8	2260	226
RMAS	10,600	51	8	1880	188	12,900	51	8	2280	228
DOMN	6,700	51	8	1190	119	7,800	51	8	1380	138

16.2.8 Basal Gouge

The basal gouge zone (“BSZ”) at the contact between the Domino and the underlying sandstone will affect the ability of the pillars to maintain confinement and will therefore result in a loss of pillar strength (as compared to a pillar with no basal gouge). Preliminary modelling results indicated that the basal gouge is a key factor in controlling pillar strength.

The characteristics of the gouge were inspected in core from several historic drill holes. The core indicates that the gouge is variable in nature. In some areas it is a very soft and plastic clay while in other areas it is a harder material observed. Highland geologists have characterized the material as either ‘soft’ or ‘hard’ in the most recent drilling program. It is clear from the distribution of soft gouge across the deposit that the pillar designs need to consider its presence.

Two samples of gouge were tested in direct with slickensides fabric aligned with the contact shear at confining stresses between 500 and 1000 psi. Both samples were characterized as soft gouge. The harder material was not possible to test since it was very highly sheared and fractured. These results indicated a friction angle of 17 degrees as shown on Figure 16.4.

Figure 16.4: Basal Gouge 2018 Laboratory Strength Testing Results

16.2.9 Other Design Considerations

The characterization work based solely on laboratory and drilling data suggests that the ore and overlying strata are high quality rock with a reasonable strength. There are no obvious areas of concern in the mine where the ore column or overlying strata are weak and would require a special design. These conclusions from the data analyses for the RLAM unit are inconsistent with the observations in the test mine where significant back instabilities were experienced in the RLAM unit despite the high rock quality and the high strength (relative to the low-stress environment in the test mine). Design studies could not be undertaken with confidence until the field data and test mine observations were reconciled.

A detailed review of the RLAM condition (as observed in drill core) near the test mine was undertaken to determine whether the RLAM was of poorer quality in the local area of the test mine as compared to the average conditions observed across the deposit. The photographs for holes near the test mine indicate that the RLAM rock quality local to the test mine is consistent with the quality elsewhere in the deposit. The inconsistencies in observed conditions underground vs. those observed in drill core are therefore not attributable to local rock quality variations. Observations consistently indicated that fine-grained laminations in the 'laminated rock' tended to absorb water when wetted and lose considerable strength. Very competent laminated rock would fall apart along laminations after being exposed to water. This observation was also reported during sample preparation in the laboratory, the samples would break apart during grinding if water was used in the process. Interestingly, when dry core was wetted and then a section of core was squeezed

axially, the laminations were observed to expel water. Observations consistently indicated that strength loss occurs along these fine-grained laminations that have a propensity to absorb water. Wetting and squeezing of the core (by hand) was found to be a reliable method of identifying the problematic laminations in the core. Sections of core from the RLAM, GLAM, and Domino were wetted and squeezed to identify problematic laminations (by observations of which laminations expelled water). The RLAM was found to consistently have the closest spacing of these features. After investigation the spacing of the laminations in the GLAM was found to be almost twice of the RLAM. The fact that the laminated rock has very thinly spaced laminations that lose strength when wet provides an explanation for why the back became unstable in such a low-stress environment in the test mine. It is likely that the beds began to shear and separate upon excavation and because of the very thin spacing of the beds in the RLAM in particular, the rock successively failed in platy slabs (consistent with reported observations). It is important to note that the back conditions were much more favourable in portions of the drift where some GLAM was left in the back. This is consistent with the observation that the beds are more widely spaced in the GLAM as compared to the RLAM. The presence of these laminations, which would not be identified during the geotechnical logging process, provides an explanation for the observations of back instability in otherwise strong high-quality rock. The design of the mine will need to consider the propensity for the back to unravel if not properly supported. In consideration of the propensity for delamination in the RLAM, controlled blasting practices and pattern bolting will be required. Where possible, stability is expected to be enhanced if a 1 ft (30 cm) “beam” of GLAM is left remaining in the back.

16.2.10 Pillar Design

Conventional design methods for room and pillar mines generally rely on empirical methods. These methods involve comparing proposed pillar dimensions and rock strength to a design curve constructed based on a database of historical pillars – both stable and unstable. There are several different empirical design charts presented in the literature, each based on different historical datasets for stable and unstable pillars. Most of these pillar databases are for hard rock mines. None of these datasets, to Golder’s knowledge, include pillars that have a zone of soft gouge along the base of the pillars. In fact, Golder is unaware of analogous conditions at other mining operations where there is a consistent layer of gouge along the contact at the bottom of pillars.

Since the strength of a pillar depends on the degree of confinement in the pillar, the BSZ will have a significant impact on pillar strengths at Copperwood. The presence of the very weak and often soft gouge infilled BSZ will result in a reduction of confinement in the pillar. The BSZ will reduce the friction at the base of the pillar and allow the pillar to expand into the opening once the room is excavated. This expansion will effectively relieve some of the pillar confinement thereby reducing pillar strength. The design of the pillars for Copperwood has therefore required the development and use of 3D numerical models that can fully

capture the impact of the basal gouge on pillar confinement and hence, the load-carrying capacity of the pillars. Note that all analysis and recommendations for pillar design assume a uniform 6.1 m (20 ft) wide room. Given the variability of the expected conditions between the eastern orebody and western orebody, such as the varying room height, stratigraphy, dip and depth of the ore body, different stratigraphic cases were constructed to represent the governing geological and geometrical conditions. The following nine cases were developed to represent the most expected conditions in the western and eastern orebodies as shown in Table 16.3.

Table 16.3: Ground Parameters

Subunit	Panels 1 to 6			Panel 20		Panel 21		Panel 22	Panel 23
	Upper	Mid	Lower	Upper	Lower	Upper	Lower	N/A	N/A
Depth (ft)	0-300	300-600	600-900	300-600	600-900	300-600	600-900	0-400	0-400
GLAM	3.5			3.0	2.0	3.5	3.5	3.0	2.5
RMA5	0.5			2.0	1.5	1.5	1.0	1.5	2.0
DOMIN	6.0			2.5	3.0	2.0	2.0	3.0	3.0
GSIL (Floor)	-			-	1.0	0.5	1.0	-	1.0
Dip (°)				12	7	14	7	14	20

Based on these cases and given the stable pillar criteria provided previously, the following pillar dimensions are recommended (Table 16.4).

Table 16.4: Pillar Size Recommendations

Orebody	Panel	Depth (m)	Assumed Pillar Height (m)	Recommended Pillar Dimensions (m)	Theoretical Recovery (%)
East	20	183	2.3	5.8 x 5.8	78
		274	2.9	7.6 x 7.6	70
	21	183	2.3	6.1 x 6.1	63
		274	2.3	7.6 x 7.6	76
	22	122	3.0	4.9 x 4.9	69
	23	122	2.9	5.2 x 5.2	75
West	1 to 6	91	3.0	5.5 x 5.5	69
		183		7.3 x 7.3	80
		274		9.4 x 9.4	79

16.2.11 Regional Pillars

Golder recommends a minimum rock crown pillar thickness of 80 ft (approximately 25 m) for the Copperwood Project. The possibility exists of locally reducing crown pillars after reviewing the rock carrying grade's local rock conditions. However, for this Report, 80 ft (approximately 25 m) is used.

16.2.12 Superior Lake Protection

Based on Golder's study, a minimum setback distance of 100 ft (30 m) is recommended between Lake Superior's shoreline and the mine excavation. This setback distance is more related to permitting as mining beneath the lake is possible once it is demonstrated that excavations remain stable with the proposed ground support.

16.2.13 Ground Support

Since overstressing is expected to develop on the pillar ribs, bolting is recommended. Golder recommends the use of 6 ft (1.8 m) long bolts on a 5 ft x 5 ft (1.5 m x 1.5 m) pattern with mesh. Initial review of geotechnical data suggested that RLAM material forming the back of the excavations is a high quality, medium strong rock and should not pose any stability issues at the proposed depths and 20 ft (6 m) room width planned for Copperwood. However, significant observational data indicated otherwise.

Considering the propensity for delamination in the RLAM, controlled blasting practices and pattern bolting will be required. 6 ft (1.8 m) long bolts on a 4 ft (1.2 m) pattern are recommended in the rooms, 8 ft (2.4 m) bolts on a 4 ft (1.2 m) pattern are recommended in intersections. Bolts used in the back should be either

resin-grouted rebar or inflatable. Where possible, stability is expected to be enhanced if a 1 ft (30 cm) GLAM “beam” is left remaining in the back.

16.2.14 Subsidence

The 3D geotechnical design models were interrogated to estimate the potential surface subsidence. The greatest vertical displacement is predicted above the deepest panels where the pillars are under the greatest load. The model predicts a maximum pillar compression of approximately 0.1 ft (3 cm). If we assume that all this deformation is experienced as subsidence on surface, a maximum surface subsidence of approximately 0.1 ft (3 cm) would be experienced. In practice, mines routinely find that only a portion of the underground deformation transfers to surface. Thus, it is expected that the small magnitudes of subsidence would be difficult to detect without precision surveys and would have minimal impact.

16.2.15 Hydrogeological Considerations.

AECOM conducted a study of the groundwater seepage to underground mine workings. In this study, the groundwater modelling was revised, and the groundwater inflow were found to be similar to the previously reported rate.

- Approximately 400 USGPM (25 l/sec) at full build-out (actual mine plan);
- Inflow increases as mining advances towards Lake Superior;
- Massive uniform matrix-supported diamicton;
- Consistent over several square miles;
- Silty clay with trace to some sand and gravel;
- Minimal seasonal variation in potentiometric surface in the overburden of bedrock,
- Some variation in shallow sections of overburden;
- Limited (or very slow) migration between units.

The amount of water flowing into the mine and the water management system will be discussed in more detail in Section 16 of this Report. According to previous studies, there are no aquifers that are affected by subsidence, and subsidence will not facilitate the inflow of water into the mine. The water pumped from the mine will probably be rich in Totalled Dissolved Solids (“TDS”), and water flowing to the mine main access from the glacial overburden will be minimal.

16.3 Selection of Mining Method

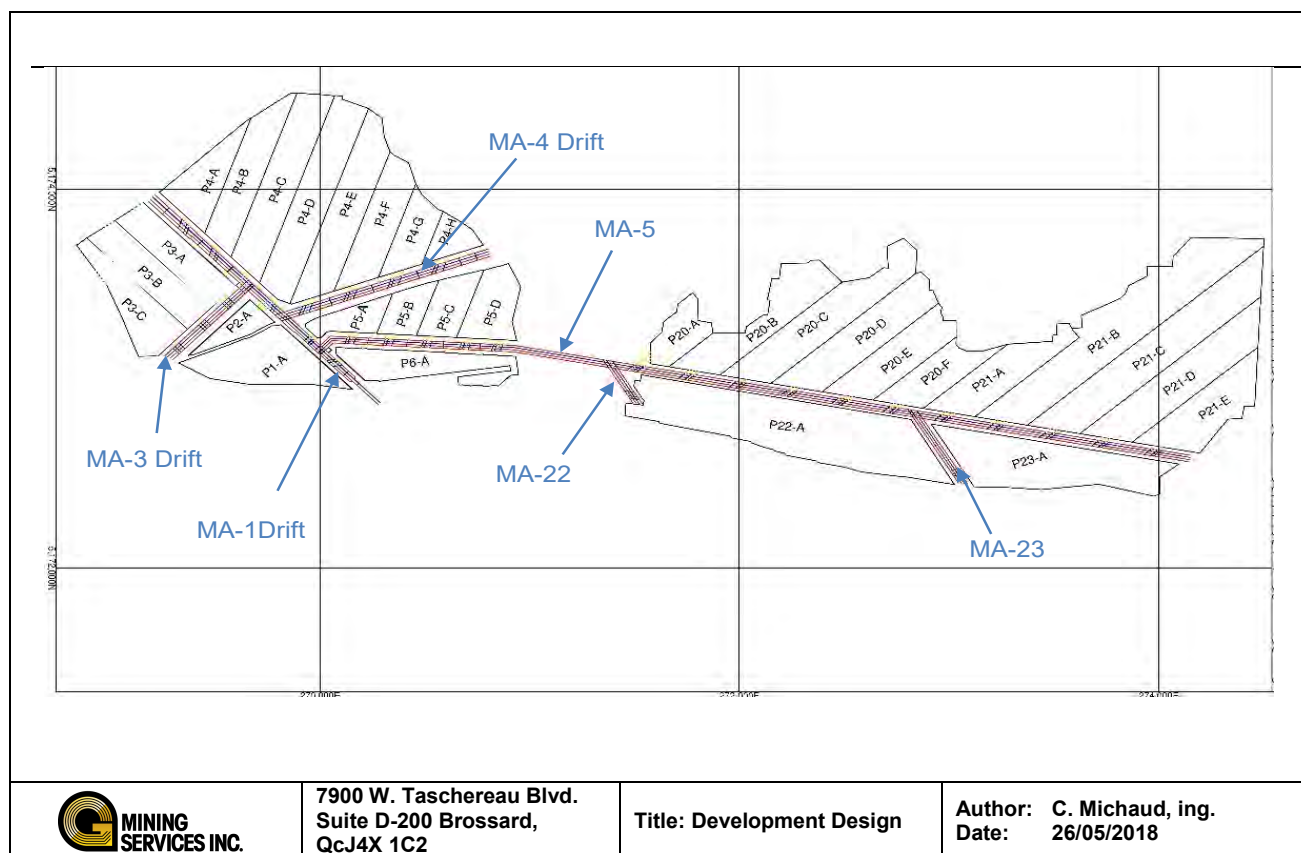
Based on geotechnical information and mineralization geometry, an underground room-and-pillar method is selected for the Copperwood deposit. This mining method allows for both a good ore selectivity and productivity. However, a series of pillars are left in place to provide roof stability. The mining design was based on a mining rate of approximately 2.4 Mt/yr. The underground access and infrastructure development were designed to support the mining method and size based on mining equipment and production rate requirements.

16.3.1 Development Design

16.3.1.1 Main Access Drift

The mine will be accessed via an open box-cut to establish a portal at the mine entrance from surface. Only two drifts are excavated from the surface portal, which develop into four drifts at a depth of 35 m. The mine consists of two mining sectors: East and West. The mine development is designed with four parallel drifts per main access including: a fresh air intake drift, an ore conveyor drift, a hauling drift and a return air drift.

The main access drifts will be excavated in the ore from the box-cut. If waste is encountered during the development, it will be stored in a closed underground excavation. All drifts are set at a width of 6.1 m, and their height varies from a minimum of 3 m to a maximum of 6 m. The drift back will follow the deposit's geology to allow for a better resource recovery. The floor however will be flat for equipment purposes. The height in the intersections of the two conveyor drifts is set at 6 m to allow the installation of a transfer point between the two conveyors. If a drift intersects a conveyor drift, the height of this section of conveyor drift will also be 6 m to allow for the installation of a steel overpass system. A series of barrier pillars between the main access drift and the stope will remain in place until mining has ended in this mining area. These barrier pillars are designed to be recovered but will respect Golder's recommendations for pillar size in this area of the mine. The first access (MA-1 drift), goes through the portal and heads to the western sector and divides in two. The position of this drift allows the working faces in the western section to multiply. Two secondary drifts are excavated from the main western drift. One heading north-east (MA-4), which is parallel to the thrust fault, the second heading south-west (MA-3). From the MA-1, the MA-5 drift is excavated to develop the eastern part of the mine and panel 5 of the western part of the mine. Some connection drifts will be excavated between the main access drift to facilitate ventilation and the transportation of equipment and personnel.

Figure 16.5: Development Design

16.3.2 Stope Entry

To access the mining production panel, four stope entry drifts will be excavated. The first stope entry drift will be used for fresh air intake, the second one for hauling and traveling, the third for the stope conveyor and the last one as an exhaust drift. The width of the drift will be 6.1 m and its height will be the same as the production panel.

16.3.2.1 Intake Ventilation Raise

In addition to the drift, three raises will be excavated to allow efficient ventilation of the mine. A fresh air raise with an emergency egress will be excavated in the center of the western section of the mine and will be raise bored 5 m in diameter and 148 m long. The raise will provide fresh air for the production period and allow a second emergency exit for the mine.

16.3.3 Exhaust Ventilation Raise

Two exhaust air raises will be required to ventilate the eastern and western sections of the mine. The first exhaust air raise will be located in the southern part of the western part of the mine and will be raise bored 4 m in diameter and 85 m long. The second exhaust raise will be located in the middle of the eastern part of the mine and will be raise bored 5 m diameter and 160 m long.

16.3.4 Stope Design

The orebody was divided into 10 main panels. The western part of the mine includes panels 1 through 6 and the eastern part includes panels 20 to 23. The thrust fault located in the western horizon splits panel 2 from panel 1 and panel 5 from panel 4. Due to the orebody dip in different areas, the access point had to be designed to mine in the best direction to reduce slope on mining equipment; therefore panels 2 and 3 are separated but are mined from the same drift. Panel 6 includes historical mining where stopes collapsed. In the eastern horizon, panels 20 and 2 are accessed from the west-east main access. Panels 22 and 23 are accessed from the south-west towards the north-east from secondary drifts. A 10 m horizontal pillar with the old mine was maintained. This last pillar could be revaluated in the future to be reduced or mined. Panels 3 to 23 have been split into smaller panels to allow for more production faces and consequently increase mine productivity. Figure 16.6 presents the panel division for the western section of the mine and Figure 16.7 presents the eastern section of the mine.

Figure 16.6: Panel Division – Western Section

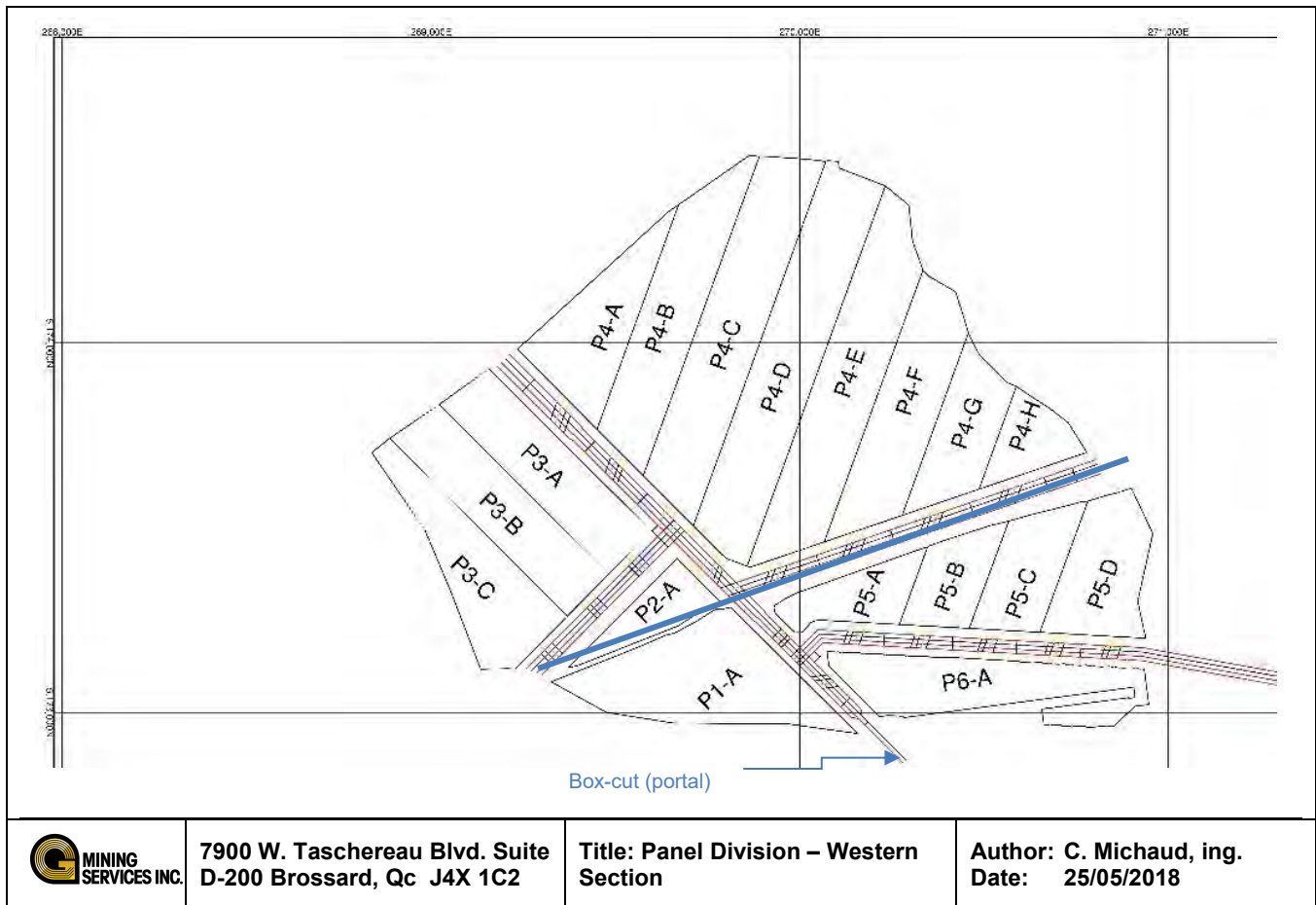


Figure 16.7: Panel Division – Eastern Section

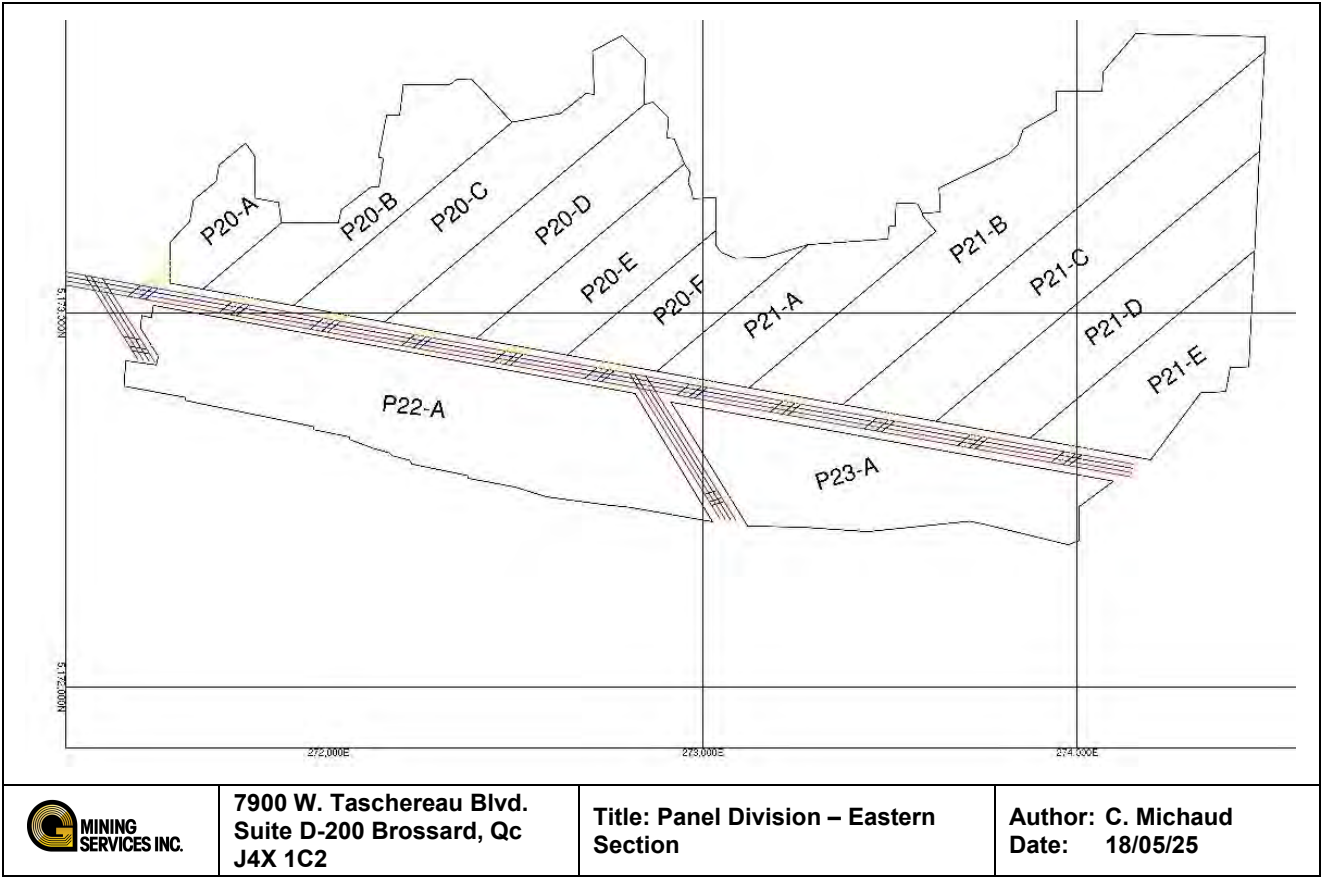


Table 16.5: Mine Design Summary

LoM Physicals	
Ore Tonnes Development (t)	2,552,134
Cu Grade %	1.07%
Ag Grade (g/t)	3.21
Stope Production (t)	22,837,101
Cu Grade %	1.47%
Ag Grade (g/t)	3.90
Total Underground Production (t)	25,389,236
Cu Grade %	1.43%
Ag Grade (g/t)	3.83
Waste Tonnes (t)	151,668
Development	Metres
Main Drift	23,765
Conveyor Drift	7,946
Connection Drift	4,477
Stope Entry	4,539
Vent Access	137
Conveyor Drift	1,178
Ventilation Raise 5 m	308
Ventilation Raise 4 m	85
Rock Breaker Excavation (waste - tonnes)	132,418

Table 16.6: Stope Pillar Size and Mining Recovery

Sector	Panel	Depth (m)	Pillar Dimensions (m)	R&P Mining Recovery
West	1 to 6	91	5.5	78%
		183	7.3	70%
		274	9.4	63%
	20	183	5.8	76%
East	20	274	7.6	69%
		183	6.1	75%
	21	274	7.6	69%
		183	4.9	80%
	22	122	5.2	79%

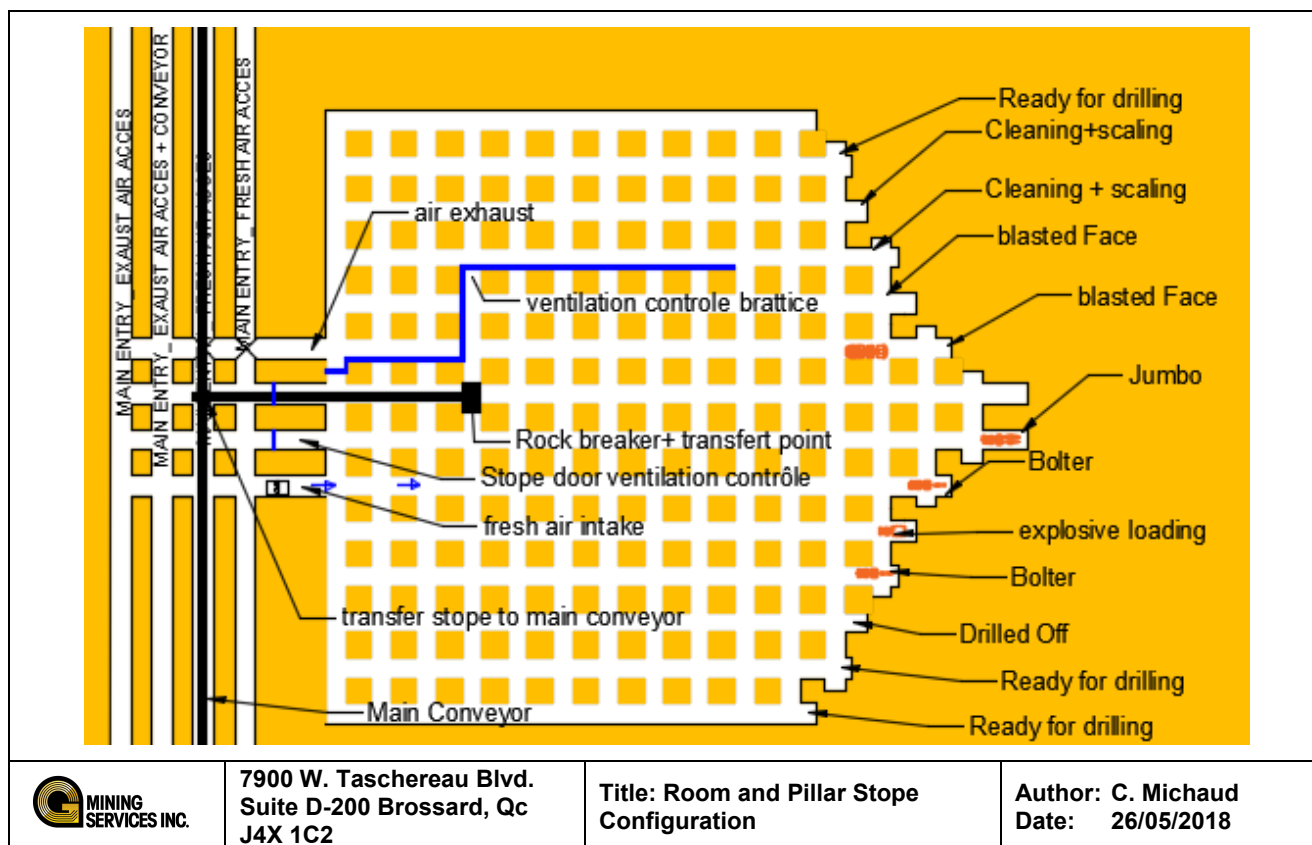
16.4 Mine Operations

16.4.1 Stopping

To access the stope four access drifts are excavated at the entrance of the stope. One of these drifts is used for fresh air ventilation and the second for exhaust air ventilation. One drift will be used for the stope conveyor and the last one for circulation. From these accesses, the panel operation begins with the drilling and blast method. To achieve and maintain an adequate level of production, the panel must contain at least 12 rooms (headings) in operation simultaneously. If the panel contains less rooms, the mining cycle may be delayed, and productivity will decrease. The mining cycle includes drilling, blasting, ore mucking, ore transportation to a rock breaker and the stope conveyor, scaling and finally ground support.

Figure 16.8 shows the configuration of the production panel.

Figure 16.8: Room and Pillar Stope Configuration



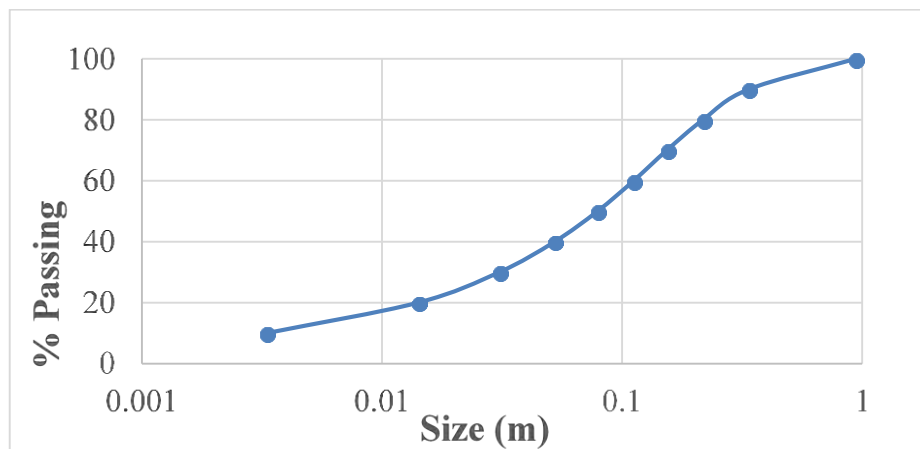
In the room-and-pillar mining method the mining cycle begins with the drilling of the working face. To perform face drilling, a low-profile hydraulic-electric jumbo with 2 booms is planned. The drilling technique

will use a burn cut to allow drilling a length of 4.25 m with an effective break length of 4.0 m. The drilling diameter is 51 mm; however this dimension can be adjusted according to blasting results. The drilling penetration rate is evaluated at 1.85 m/s and the average drilling time per round is evaluated at 3.3 h/round. The rock at Copperwood has very low abrasion as confirmed by metallurgical testing done for this Study.

Blasting crews will load the rounds with explosives and initiate blasts at the end of each shift. Explosives will consist of a mixture of ANFO and emulsion where there is presence of water. A decoupled explosive charge is recommended to presplit the back of the room. Control of drilling and blasting is very important for the Copperwood Project. The perimeter control of the drilling should allow to reduce the dilution to a minimum but also to keep a 0.3 m beam of Gray Laminated rock on the back.

A fragmentation study by an explosives provider was carried out for the Copperwood Project. Several rock types are present during blasting operations which produce a different particle size for each rock type. The Red Massive geological unit produces the largest fragments during blasting. Figure 16.9 presents the results for Red Massive with ANFO as explosives.

Figure 16.9: Particle Size Distribution Red Massive with ANFO Explosives



The blasting of the loaded round will be performed at the end of every shift. A period of 2 hours is planned between shifts to vent blasting fumes from the mine. The main access and ventilation raises will be monitored with gas detectors.

The third mining activity is to muck the blasted ore from the face and to transport it with a low-profile 10t LHD. The performance of the LHD is a function of the dip of the stope and the distance between the face heading and the rock breaker. The LHD performance will vary from 3.9 km/h at 17% (loaded) to 8.9 km/h at -17% (unloaded). To reduce the haulage distance, the unloading point will be moved regularly to be normally less than 250 m from the working face. however, a case-by-case evaluation was made for

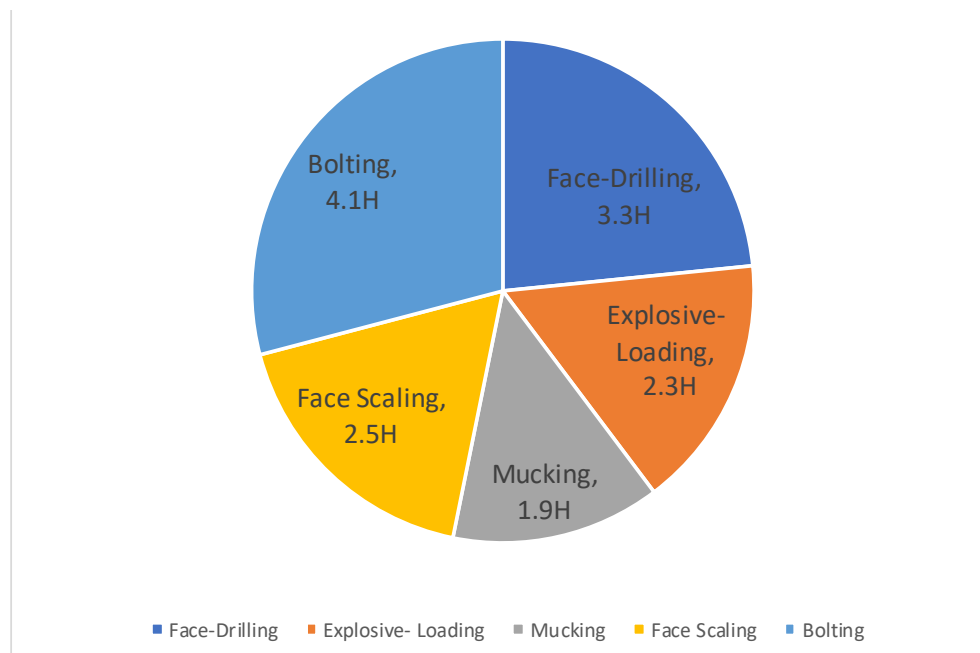
each of the planned rock breaker moves (more than 67), to justify economically this displacement. For the economic evaluation of the Project, the average hauling distance was calculated for each of the planned rock breaker positions. For operating cost calculations, a capacity of 9.12 t per bucket is used which considers the fill factor and the loading equipment.

The next step in the mining cycle is to scale the back and wall of the excavation. To do this a smaller low-profile LHD equipped with a scaling arm is used. the LHD's arm repeatedly rubs the roof and wall of the drift to remove the loose rock. This method was used at the White Pine mine and is very effective in sedimentary (stratified) rock.

A low-profile rock bolter is used to install the roof and wall support. There is a lot of ground support to do for each working face. In the room excavation 1.8 m rebar bolts are required according to a 1.2 m x 1.2 m pattern. Friction bolts of 1.8 m according to a pattern of 1.5 m x 1.5 m are also installed on the wall pillars. In this Study, friction bolts are currently included in the primary rock bolting cycle. However, bolts could be added in a second step behind the rock bolter. Wire mesh should be added to the roof and wall of the excavation. 2.4 m rebar bolts must also be added at the intersection of the rooms. Since these bolts are too long for lower height rooms (under 2.4 m), the connectable bolts planned for these excavations. As bolting demands are high, the bolter-jumbo ratio is 1.5 on average for the production period. The drilling performance of the bolter is estimated at 2 m/s.

The total round cycle time is estimated at an average of 14.1 hrs/round.

Figure 16.10: Round Cycle Time



Once the area mining is completed, the maximum amount of ore will be recovered from the barrier and drift pillars as illustrated in Figure 16.11 and Figure 16.12.

Figure 16.11: Drift, Room and Pillar before Recovery

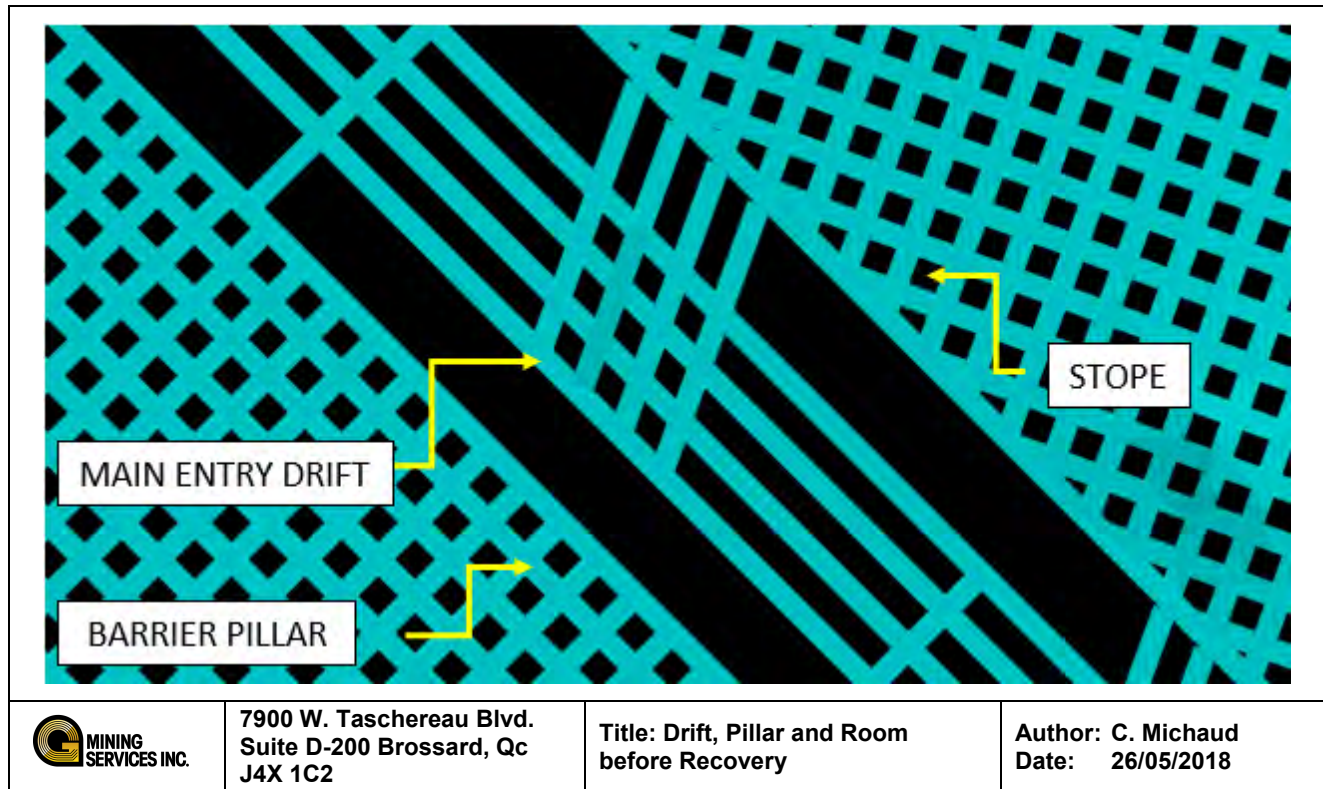
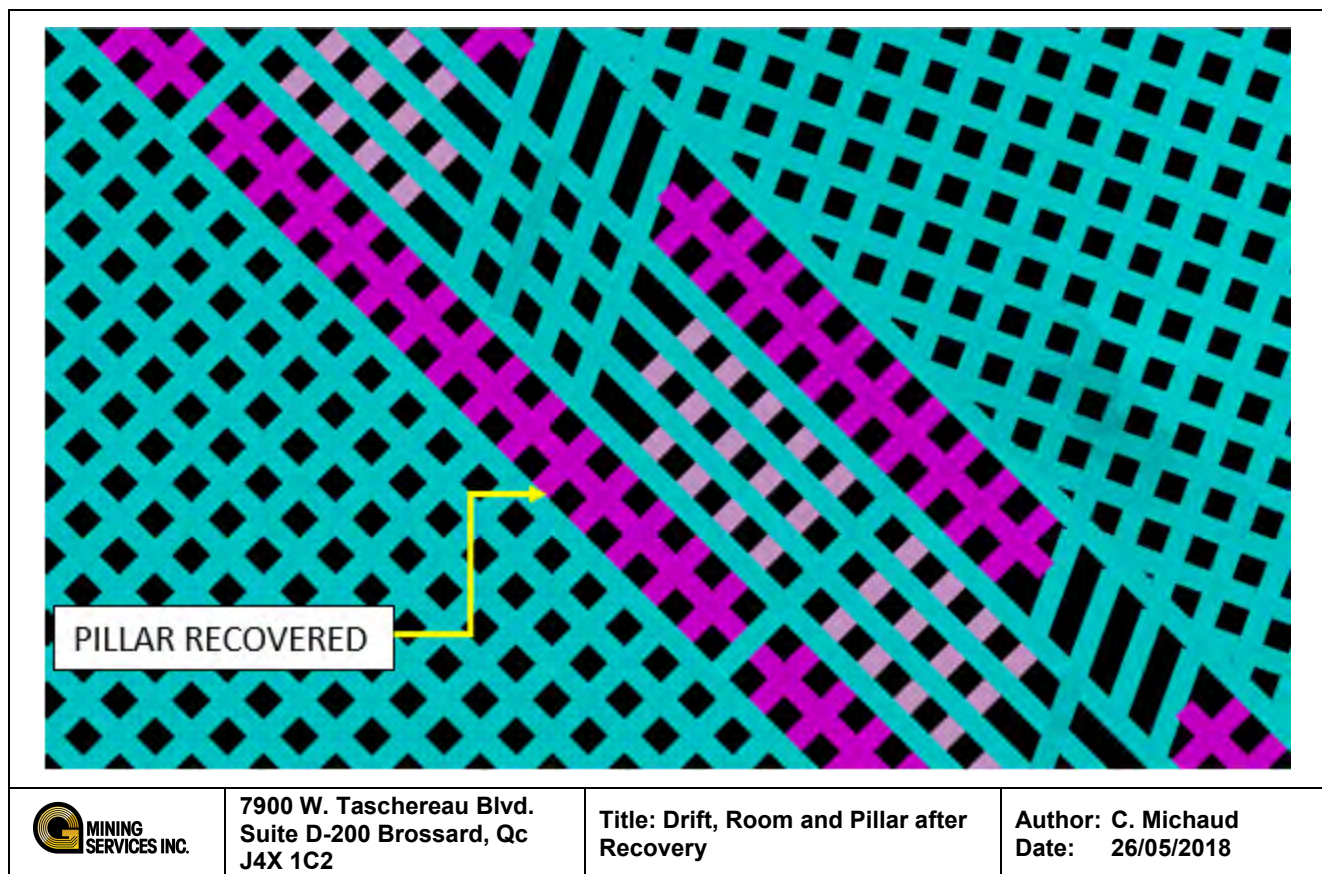


Figure 16.12: Drift, Room and Pillar after Recovery

16.4.1.1 Mining Parameters

The basic operational assumptions are summarized as follows:

- Minimum mining height 2.1 m (limited by the equipment);
- Maximum mining height 5.8 m;
- Average mining height 2.5 m;
- Average mining height western sector 2.81 m;
- Average mining height eastern sector 2.21 m;
- Cut-off grade 1% Cu;
- Annual production – 2.4 Mt;
- Entry drift (main access) and room and pillar width 6.1 m;
- Lake Superior horizontal protection 30 m;

- Surface pillar 25 m;
- Old test mine pillar 10m;
- Fresh air raise 5 m;
- East exhaust air raise 5 m;
- West exhaust air raise 4 m;
- Conveyor maximum optimal distance to the face heading 250 m;
- Minimum of 12 rooms per operating panel.

16.4.2 Ore Handling System

The broken ore from the development headings will be mucked by a 10 t low-profile LHD to temporary remuck bays located up to 200 m from the face, and then hauled by 30 t low-profile trucks to the surface or to a rock breaker loading point. The broken ore from the stope will be mucked by a low-profile LHD to a stope dumping point. The stope dumping point is a system composed of a grizzly, a rock-breaker and loading points to the conveyor system. This system will be installed in every production panel and can be moved when the faces are too far apart. The parallel bar grizzly with 200 mm openings prevent oversize material from entering the conveyor system. The hydraulic rock breaker will be used to break oversized material on the grizzlies. The hydraulic rock breaker will be remotely controlled from the surface by an operator. The present Study presumes that one operator can operate 4 rock breakers from surface. The ore will be transferred on the stope conveyor. The 42 in wide belt stope conveyor, comprised of a 500 HP motor can be extended depending on the progress of the stope. It is currently planned to advance these conveyors every 250 m according to the progression of the stope. the broken ore is then transferred to the principal conveyor located in the main drift conveyor. The maximum length of one main conveyor is 1,200 m. After this distance, a second is installed and interrelated between them. In the west sector 2 main underground conveyors are required. In the East section, four main conveyors are required. Each of these conveyors is equipped with 500 HP electric motor. The main conveyor transports the ore to the surface.

16.4.3 Mining Equipment:

Table 16.7 shows the equipment requirements to support the planned 6,600 mtpd nominal production rate.

Table 16.7: Mine Equipment Requirements

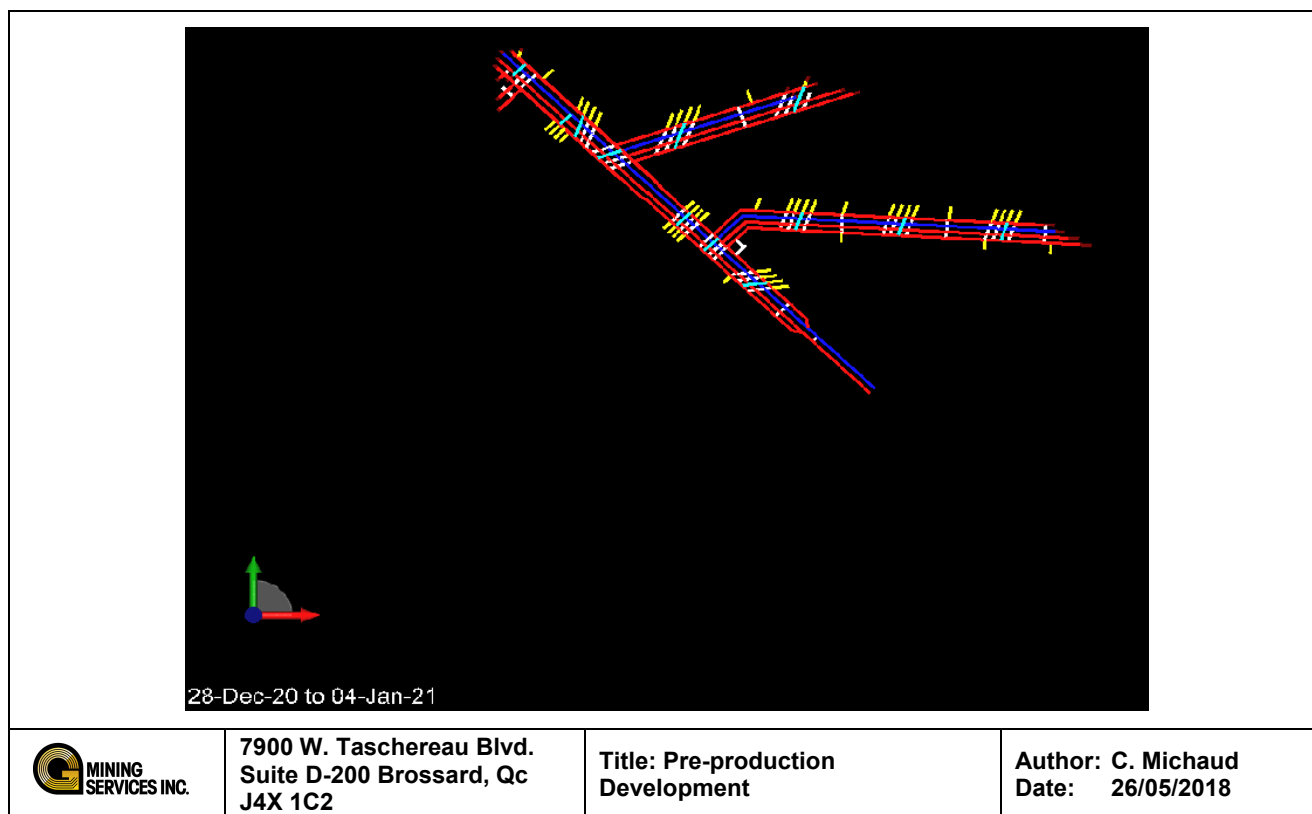
Mobile Equipment	
Low-Profile 2 Booms Jumbo Drill	10
Low Profile 1 Boom Electric-Hydraulic Bolter	18
Low Profile LHD 10 Mt	10
Low Profile LHD 8 Mt	2
Scaler	5
Development Truck	4
Lube Trucks	3
Flat Bed Trucks	3
Scissor Lift	10
Grader	1
Tractor -Underground	24
ATV -Underground	24
Cable Bolt Drill Stope Mate Drill	1
Ore Handling System	
Loading Point+ Rock Breaker	12
Main Conveyor 1200m- 500 HP	8
Stope Conveyor 500m – 500 HP	11
Dewatering	
Electric-Sumps-Pumps	8
Orca Series Station	2
Mini Orca Series Station	4
3" Versa-Matic Pump	7
Ventilation	
Production Panel Auxiliary Fan	9
15 MBTU Pre-Production Propane-Heater	1
Preproduction Fan	2
Main Ventilation Fan 1250hp	2
50 MBTU Propane-Heater	1
Other	
Shotcrete Machine	2
Communication System	1
Surveying Equipment (Lot)	1
Jackleg Drill C/W Air Leg	30
Stoper Drill	30
Ictus Grout Pump	2
Mobile Mine Refuge Chamber	2
Head Lamp	350
Head Lamp-48 Units Charger	8
Blast hole Charger 360	12

16.5 Development Schedule

Development will be divided into two periods: a pre-production development period (from the beginning to the 20th month) and a production period (from the 20th month to the end).

16.5.1 Pre-production Objectives

- Achieve early production from higher-grade areas of the west part of the mine;
- Provide access for equipment;
- Provide ventilation and emergency egress;
- Establish ore handling systems;
- Install first mining services (power distribution, IT communications system, dewatering system, compressed air and water supply);
- Develop sufficient production panels to support the mine production rate.

Figure 16.13: Pre-production Development Period**16.5.2 Production**

It was assumed that pre-production and production drift development will be excavated by the Owner's mining department. The owner approach is preferred to reduce development costs and mining contractors typically do not have low profile equipment. Once the portal is built, development of the main access drifts will begin. The production of the 2 main access drifts from the portal will be 5 m/d. Once the main access

drifts divide into 4 drifts, production will increase to 10 m/d. As soon as a new heading is available, a new team will be added to reach a maximum of 3 teams. From February 2022 the number of development teams will be reduced to two and subsequently only one will be remaining in June 2022. Drift development will be completed in 2026.

Elaboration of the vertical and inclined ventilation raises will be performed by the contractor's raise boring crew. Raise development was used in the elaboration of the mine schedule. It was assumed that a raise boring crew can drive the raise at an advance rate of 90 m/mo. It was estimated that all pre-production development will be completed in 20 months. Development sequences were performed and optimized with the Deswik.Sched™ software.

16.6 Production Schedule

The production schedule is based on mining a fixed target of 2.4 M t/yr. To achieve this annual production, seven to nine production panels must be in production simultaneously. The number of required panels depends on the tonnage from the development as well as the height of the rooms of each panel.

On March 2021, the first stope will begin, to reach a production rate of 6,690 t/d in January 2022. Before January 2022 the difference between the daily underground production and the daily mill production, will come from the surface stockpile accumulated during the pre-production period. In the pre-production period, the priority is to start the production from the western part of the mine as this zone has better grade and higher rooms allowing for higher productivity.

Table 16.8: Ramp-up Summary

Tonnage	Pre-Production								Production Ramp-up				Production			
	Year 1				Year 2				Year 3				Year 4			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Development	-	-	17,439	53,160	72,968	106,557	157,825	163,687	166,040	166,423	164,758	162,058	126,842	97,502	58,122	49,602
Stoping	-	-	-	-	-	-	-	-	8,584	168,171	315,553	387,530	484,607	93,217	567,251	556,916
Total	-	-	17,439	53,160	72,968	106,557	157,825	163,687	174,624	334,594	480,312	549,587	611,449	590,719	625,373	606,519
Tonnage/days	-	-	190	578	802	1,171	1,715	1,779	1,940	3,677	5,221	5,974	6,794	6,491	6,798	6,593

The stoping productivity varies for each stope depending on the mining height. For the minimum stope height of 2.1 m the production rate is estimated at 951 mtpd and can reach up to 1,285 mtpd for a 3.9 m high stope. The limit of stope production is the productivity of the jumbo drill.

Table 16.9: Productivity per Mining Panel

Panel Height	Panel Productivity mtpd
2.1	951
2.3	992
2.5	1,030
2.7	1,078
2.9	1,111
3.1	1,154
3.3	1,183
3.5	1,223
3.7	1,248
3.9	1,285

The western portion of the orebody has a higher average copper grade than the eastern part. Therefore, western portion is mined at the beginning (shown in blue shades) and slowly introduces tonnage from the eastern portion of the orebody in 2024 (shown in green shades). The copper grade drops in value in 2024 and levels around 1.30%; whereas it averages 1.69% from 2019 to 2028. In 2026-2027 and from 2030, pillar recovery occurs (shown in purple) where main drifts are no longer useful.

Table 16.10: Mine Production Schedule Summary

Mine Production		Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Development Mining															
Tonnage	kt	2,552	71	501	659	332	217	223	339	210	-	-	-	-	-
Cu Head Grade	% Cu	1.07	1.34	1.33	1.39	0.86	0.72	0.75	0.81	0.75	-	-	-	-	-
Ag Head Grade	g/t	3.21	4.66	4.40	4.47	2.60	1.91	1.45	1.94	2.11	-	-	-	-	-
Cu Contained Metal	kt	27	0.9	6.7	9.2	2.9	1.6	1.7	2.8	1.6	-	-	-	-	-
Ag Contained Metal	k oz	263	10.6	70.9	94.8	27.8	13.4	10.4	21.2	14.3	-	-	-	-	-
Production Mining															
Tonnage	Kt	22,837	-	-	880	2,102	2,174	2,191	2,115	2,196	2,359	2,430	2,411	2,456	1,524
Cu Head Grade	% Cu	1.47	-	-	1.90	1.97	1.78	1.73	1.46	1.33	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.90	-	-	6.31	6.82	6.23	6.06	4.35	3.32	2.90	2.17	2.10	2.19	2.21
Cu Contained Metal	kt	337	-	-	17	41	39	38	31	29	32	30	30	31	20
Ag Contained Metal	k oz	2,866	-	-	178	461	435	427	296	235	220	169	163	173	108
Total Mining															
Tonnage	kt	25,389	71	501	1,539	2,434	2,391	2,414	2,454	2,406	2,359	2,430	2,411	2,456	1,524
Cu Head Grade	% Cu	1.43	1.34	1.33	1.68	1.82	1.68	1.64	1.37	1.28	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.83	4.66	4.40	5.52	6.24	5.84	5.63	4.02	3.22	2.90	2.17	2.10	2.19	2.21
Cu Contained Metal	kt	364	1	7	26	44	40	40	34	31	32	30	30	31	20
Ag Contained Metal	k oz	3,129	11	71	273	489	449	437	317	249	220	169	163	173	108

Figure 16.14: Panel Sequence

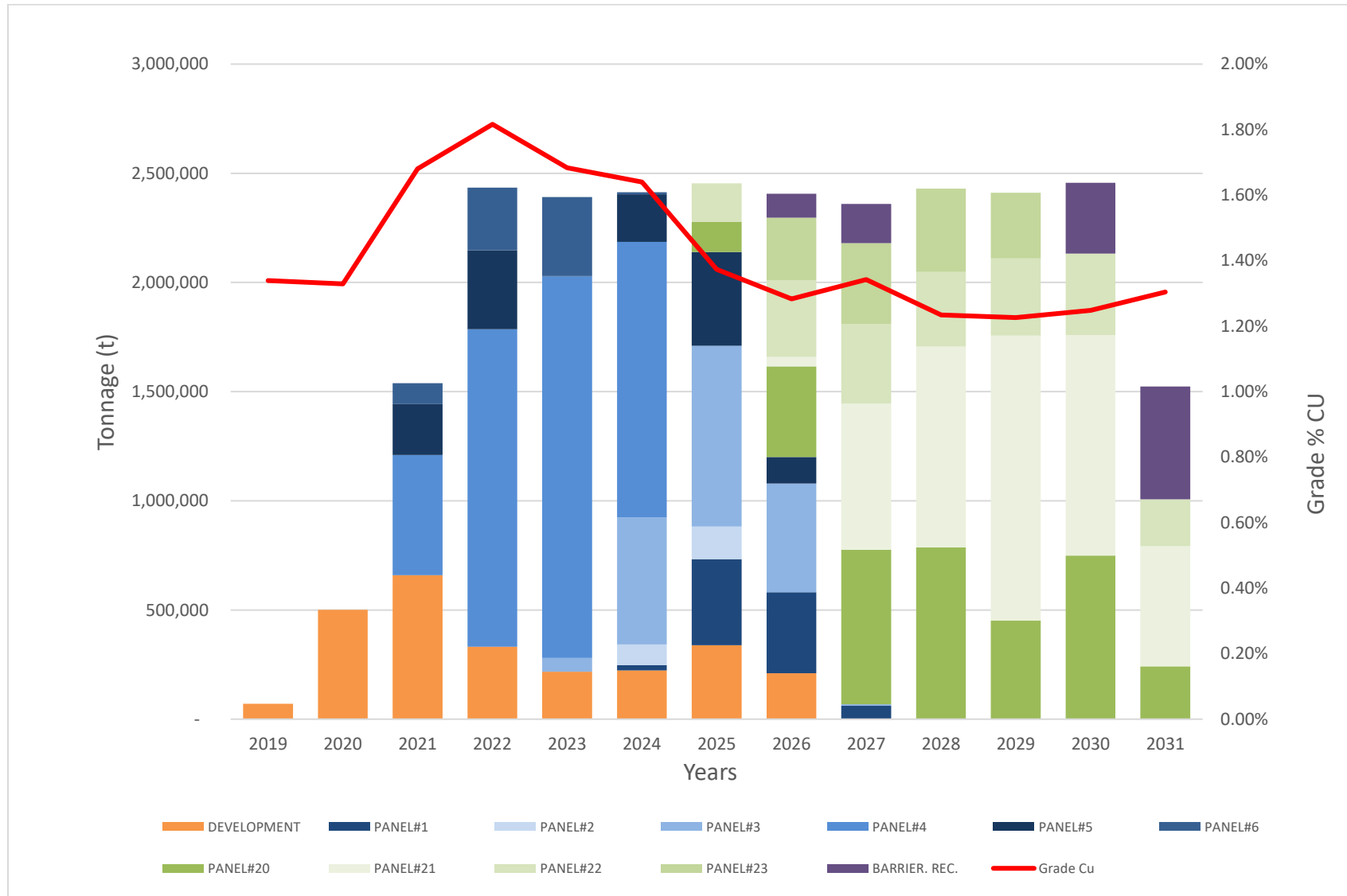


Table 16.11: Operating Shift Assumptions

Operating Parameters	
Days in period	365
Shifts per Day	2
Hours per shift	10
Total hours/year	7,300
Total days lost/ year	5
Total days operated/year	360
Scheduled Hours/year	7,200
Equivalent scheduled shifts	720
Shift Composition (minutes)	
Travelling to work place	30
Workplace inspection	15
Equipment inspection/set-up	15
Lunch (+ travel to and back)	45
Supervision	15
Operation Delays	30
Travelling to surface	30
Change	0
Total time loss (minutes/shift)	180
Total time loss (hours/year)	2,160
Jumbos availability	85%
Jumbos available hour	6,120
Utilization %	65%
JUMBO operating hour	3,960

16.7 Manpower and Working Schedule

Labor levels are estimated based on the production schedule and equipment requirements to reach a production level of 2.4 Mt/yr. To achieve the level of productivities used in this Study, the workforce must be a mix of skilled labor with an experienced management team. The mine work schedule is based on

working two shifts per day, seven days per week, 360 days per year. A rotation schedule of 7 days in and 7 days out has been selected for mine operation requirements, with rotation days and nights.

Several mine services will however be on a 5-2 schedule of 5 or 7 days in and 7 days out on day shifts only. Table 16.12 represents the different schedules for the underground mining operation

Table 16.12: Production Working Schedule

Grade	Job Title	Rotation Schedule	Worked Hours
	Mine Supervision		
Staff	Mine Manager	5-2	2,080
Staff	Mine Ops. Superintendent	5-2	2,080
Staff	Mine Secretary	5-2	2,080
Staff	Mine Captain	5-2	2,080
Staff	Mine Ops. Foreman	7-7 on 2 shifts	1,825
Staff	Mine Ops. Trainer	7-7 on 1 shift	1,825
	Mine Operation		
Hourly Class 1	Jumbo Operator + Bolter + LHD operator + Truck Operator	7-7 on 2 shifts	1,825
Hourly Class 1	Grader Operator	5-2	2,080
Hourly Class 1	Feeder Breaker Operator	7-7 on 2 shifts	1,825
Hourly Class 1	U/G Construction Maintenance + Material Handling + Ventilation Crew	7-7 on 1 shift	1,825
hourly	Material Handling	7-7 on 1 shift	1,825
hourly	Conveyor Serviceman	7-7 on 1 shift	1,825
Hourly	Labour , Dryman, Drill Burs Carpenter	7-7 on 1 shift	1,825
	Technical Services		
Staff	Chief Mine Engineer + Chief Geologist	5-2	2,080
Staff	Long-Term +Short-Term Planning+ Geotech + Project + Geologist	5-2	2,080
Staff	Mine Technician	5-2	2,080
Staff	Senior Surveyor	5-2	2,080
Staff	Surveyor + Geology Technician + Geotech Technician	7-7 on 1 shift	1,825
	Mechanical Services		
Staff	Mechanical Superintendent	5-2	2,080
Staff	General Foreman +Maintenance Planner + Mechanical Engineer	5-2	2,080
Staff	Supervisors	7-7 on 2 shifts	1,825
Hourly Class 1	Mechanics - Mobile + Fixed Equipment	7-7 on 2 shifts	1,825
	Electrical Services		
Staff	Electrical Superintendent	5-2	2,080
Staff	Supervisors	7-7 on 2 shifts	1,825
Staff	Electrical Engineers	5-2	2,080
Hourly Class 1	Technicians	7-7 on 1 shift	1,825
Hourly Class 1	Electricians + Electricial Technicians	7-7 on 2 shifts	1,825

No allowance has been made for absenteeism, sickness, snow days, or dumped shifts. Holidays and vacation expenses are covered in the fringe benefit allowance.

Table 16.13: Mine Manpower Requirements

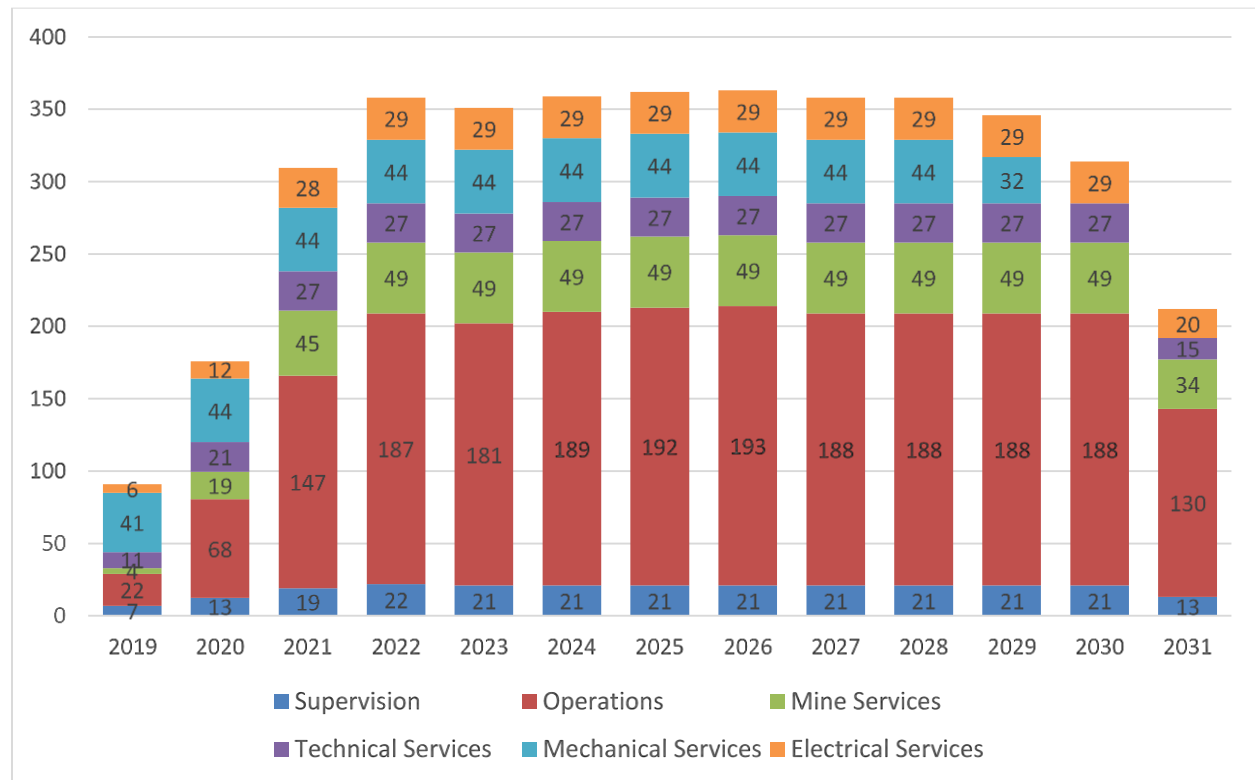


Table 16.14: Mine LoM Manpower Requirements

Mine Manpower by Year	1	2	3	4	5	6	7	8	9	10	11	12	13
Mine Supervision													
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Secretary	0	1	1	1	1	1	1	1	1	1	1	1	1
Mine Captain	0	1	2	2	2	2	2	2	2	2	2	2	2
Mine Ops. Foreman	4	8	13	16	16	16	16	16	16	16	16	16	13
Mine Ops. Trainer	2	2	2	2	1	1	1	1	1	1	1	1	0
Mine Operation													
Jumbo Operator	4	10	24	32	31	32	32	32	32	32	32	32	32
Blaster + Miner	8	20	48	64	62	64	64	64	64	64	64	64	64
Bolter Operator	2	12	35	52	53	56	56	58	60	60	60	60	51
LHD Operator	6	13	26	33	31	32	32	32	32	32	32	32	27
Truck Operator	2	13	14	6	4	5	8	7	0	0	0	0	0
Mine Services													
Grader Operator	0	1	1	1	1	1	1	1	1	1	1	1	1
Feeder Breaker Operator	0	0	4	8	8	8	8	8	8	8	8	8	8
U/G Constructions Maintenance	2	4	8	8	8	8	8	8	8	8	8	8	7
Material Handling	0	6	12	12	12	12	12	12	12	12	12	12	12
Ventilation Crew	0	2	4	4	4	4	4	4	4	4	4	4	4
Conveyor Serviceman	0	0	8	8	8	8	8	8	8	8	8	8	7
Labour - Lunchroom, Tool Crib ,etc	0	2	4	4	4	4	4	4	4	4	4	4	3
Lamps-Dry	2	2	2	2	2	2	2	2	2	2	2	2	2
Drill Bits Sharpener, Tool Crib, etc	0	2	2	2	2	2	2	2	2	2	2	2	2
Technical Services													
Chief Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Long-Term Planning Engineer	0	1	1	1	1	1	1	1	1	1	1	1	0
Short-Term Planning Engineer	0	1	1	1	1	1	1	1	1	1	1	1	1
Project Engineer	0	1	1	1	1	1	1	1	1	1	1	1	0
Senior Geotechnical Engineer	0	1	1	1	1	1	1	1	1	1	1	1	0
Mine Technician	1	4	4	4	4	4	4	4	4	4	4	4	3
Geotech. Technician	2	2	2	2	2	2	2	2	2	2	2	2	1
Senior Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor	2	3	4	4	4	4	4	4	4	4	4	4	3
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Geologist	0	1	1	1	1	1	1	1	1	1	1	1	1
Geology Technician	2	4	8	8	8	8	8	8	8	8	8	8	7
Mechanical Services													
Mechanical Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
General Foreman	0	0	1	1	1	1	1	1	1	1	1	1	1
Supervisors	1	2	4	4	4	4	4	4	4	4	4	4	4
Maintenance Planner	1	1	2	2	2	2	2	2	2	2	2	2	1
Mechanical Engineers	0	1	2	2	2	2	2	2	2	2	2	2	1
Mechanics - Mobile Equipment	6	13	28	30	30	30	30	30	30	30	30	30	30
Mechanics - Fixed Equipment	2	2	4	4	4	4	4	4	4	4	4	4	4
Electrical Services													
Electrical Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant Superintendent	0	0	1	1	1	1	1	1	1	1	1	1	1
Supervisors	0	1	4	4	4	4	4	4	4	4	4	4	4
Electrical Engineers	0		1	1	1	1	1	1	1	1	1	1	1
Technicians	0		2	2	2	2	2	2	2	2	2	2	1
Electricians	3	5	9	10	10	10	10	10	10	10	10	10	9
Electrical Technicians	3	5	9	10	10	10	10	10	10	10	10	10	9
Total	60	153	307	358	351	359	362	363	358	358	358	358	325

16.8 Mine Services

16.8.1 Ventilation

During the pre-production period, air requirement will be supplied through two 300 HP 1.4m diameter parallel Van axial surface fans. The two fans will be installed on a metallic stand then connected with vent tubes directed to the portal. The two fans in parallel will generate approximately 55 m³/s each at 2.5 kPa of water gauge. These two fans will be used until the main fan intake is commissioned. The fresh air will circulate in two of the main drifts, and the exhaust air will be returned to the surface in the two other drifts.

The ventilation system will consist of a push system whereby two 1250 HP 2.60 m diameter parallel main fans will be installed at surface providing approximately 200 m³/s each at 3.34 kPa. The two main fans will be installed and provide heated air through a 5 m ventilation raise and air will be distributed throughout the mine using ventilation regulators, auxiliary fans, doors and bulkheads. Also included is a 4 m diameter exhaust ventilation raise located at western side of the mine, and a 5 m diameter exhaust raise in the eastern side of the mine. Emergency egress will be installed in the fresh air raise. A 125 cfm/hp factor was used to estimate ventilation requirements if the equipment was not MSHA approved.

Figure 16.15: Ventilation Layout during Production Period (western side)

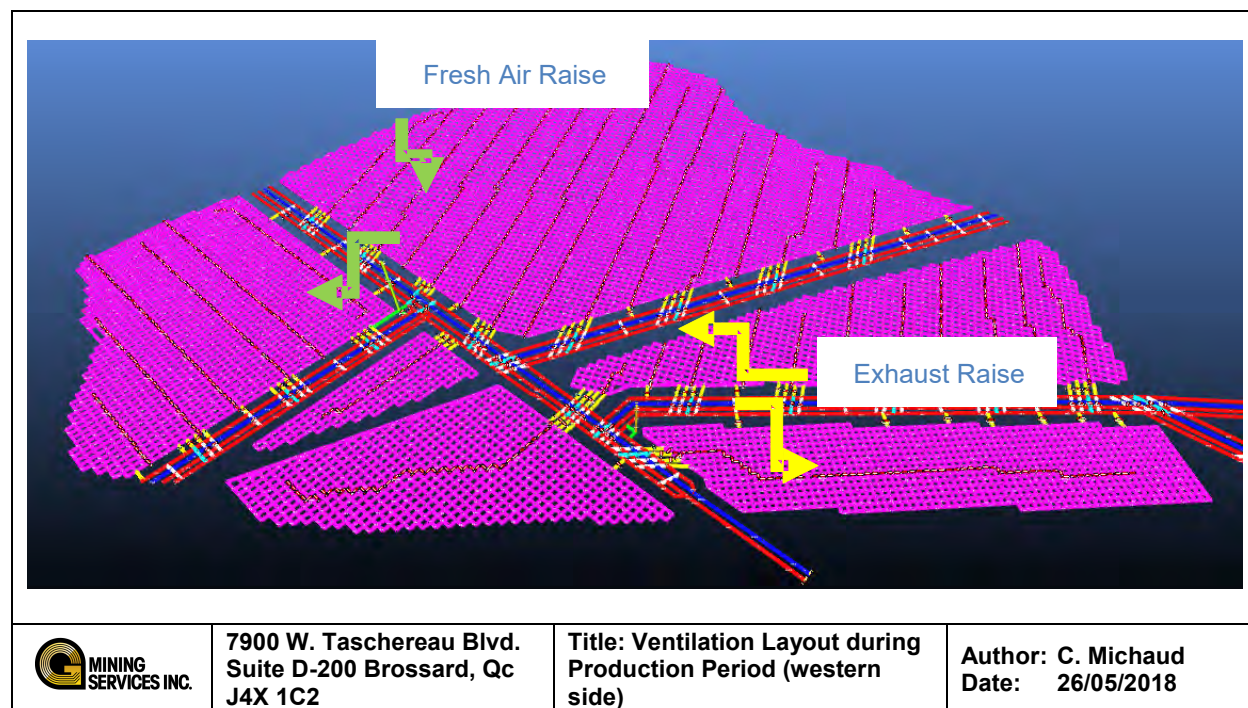
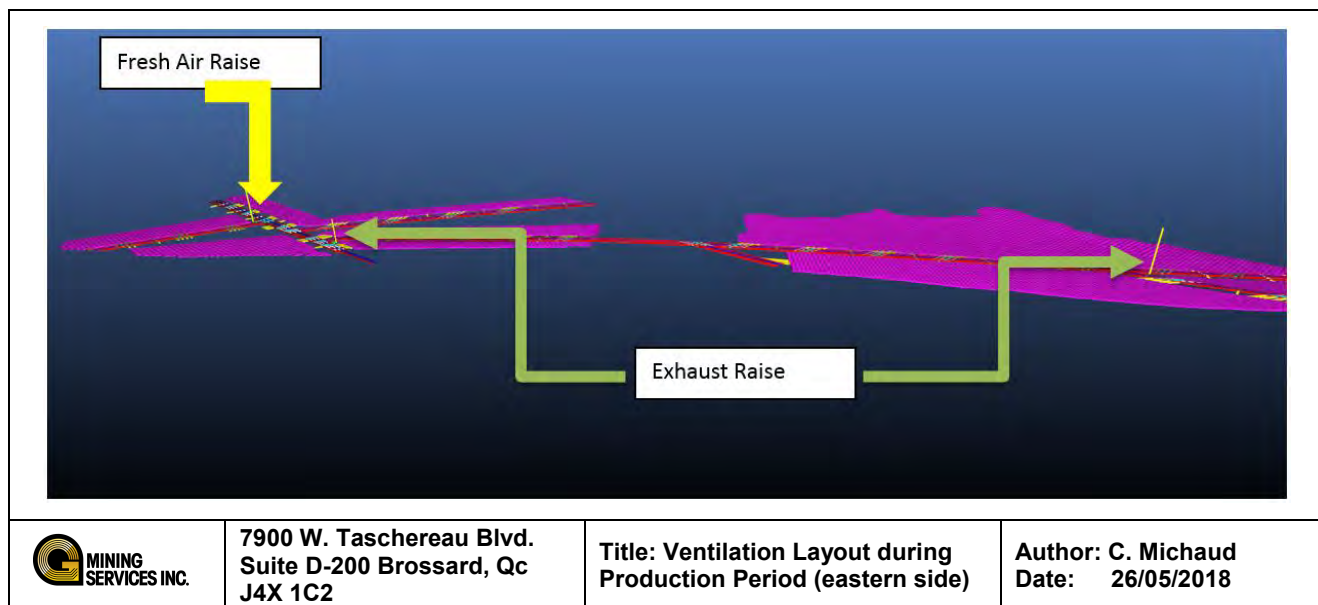


Figure 16.16: Ventilation Layout during Production Period (eastern side)**16.8.2 Water Supply**

Water is required underground for drilling, dust control and fire protection. Water will be distributed underground by a 4 in (10.2 cm) steel pipe schedule 40 in (1 m) the main access drift and 2 in (5 cm) light wall steel pipe in the stopes. This pipe size will provide adequate quantity and pressure to meet the needs of dust control and fire protection.

Table 16.15: Equipment Water Consumption

Underground Water	Consumption (l/min)	Use (eff. time)
Washing Working Faces	15	5%
Jumbo Drilling	40	65%
Bolters	45	65%
Cable Bolters	45	15%
Shotcrete Machines	45	35%
Diamond Drilling	60	0%
Raise Boring Machines	65	25%
Feeder-Breakers	0	85%
Wetting Muck Piles	5	85%
Dust Suppression	25	50%

16.8.3 Power

Major electrical power consumption in the mine will be required for the following equipment:

- Main and auxiliary ventilation fans;
- Main conveyor system;
- Stope conveyor system and rock breaker-loading points;
- Jumbo and bolter equipment;
- Mine dewatering pumps.

A high voltage cable (13.8 kV) will be installed in the conveyor drift access. This high voltage cable will connect to a substation in each production panel which will drop the voltage to 600 V for the electrical needs of the operation.

16.8.4 Dewatering

Water in the mine will emanate from the underground water inflow and mining operations (total of 2,220 l/min). The dewatering system will pump commonly called “dirty water”. This water will be cleaned and sent to sedimentation ponds at the surface preventing mining operations from cleaning sumps underground. Pumping stations have been designed to operate 50% of the time, allowing at least double the maximum required capacity. The two main pumping stations, P1 and P2, have 12.0 m³ and 9.0 m³ water tanks, equipped with agitators to prevent mud from settling at the bottom. The four other pumping stations will have a 3.5 m³ tank without agitators.

The Copperwood dewatering system consists of six permanent pumping stations (Figure 16.17). The main pumping station is P1, pumping all underground water towards the surface; it receives water from P2, P3, and P6 as well as mining panels 1 and 6. Pump P2 receives water from mining panels 2, 3 and 4. Pump P3 receives water from mining panel 4. Pump P 6 is the second main pump, which pumps all the water from the eastern part of the mine and sends it to pump P1. Pump P6 receives water from pump P5 and panels 5, 20 and 22. Pump P5 receives water from pump P4 and panels 20, 21 and 23. Pump 4, the smallest of the pumps; P4, pumps water from half of panel 21.

Figure 16.17: Dewatering Circuit

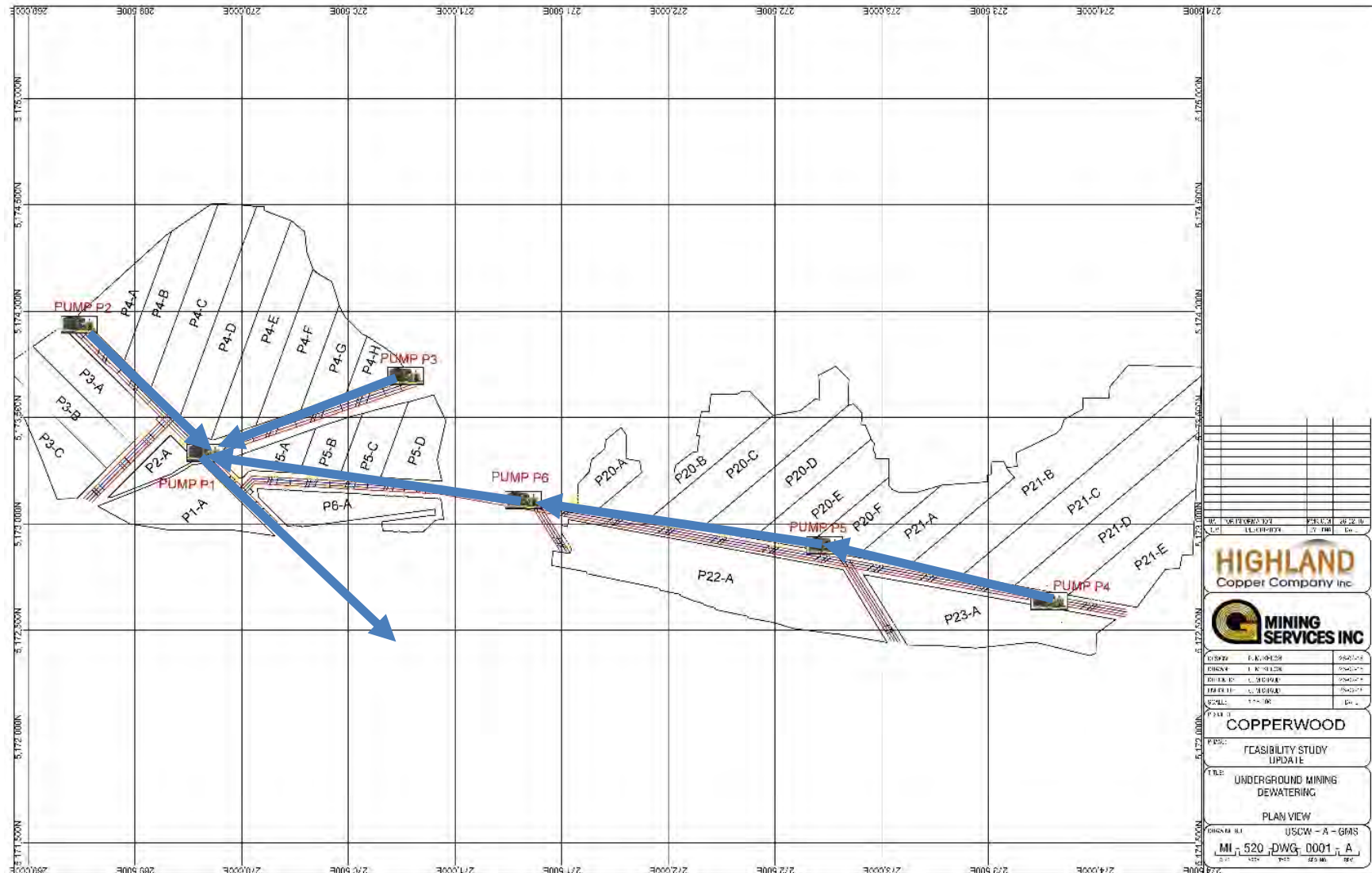
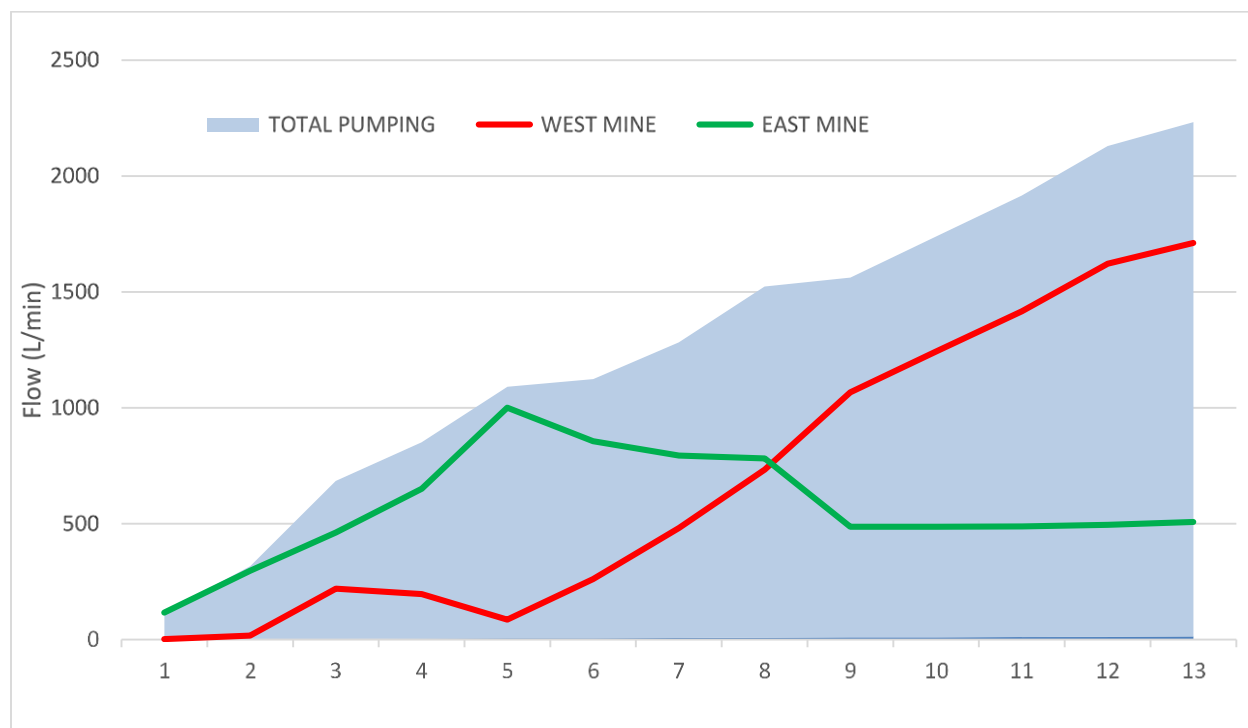


Figure 16.18: Pumping Requirement over LoM in L/min Operating @ 100%

Auxiliary pumps will also be required to redirect water towards the main pumping system. The auxiliary pumps will be resistant in abrasive slurries and have a capacity of 1,000 L/min with a 20 HP motor. Eight pumps will be required when mining operations reach desired production in Year 4.

16.8.5 Compressed Air

Compressed air supply will be provided by electric compressors installed temporarily for the pre-production period. For the production period compressed air supply will be provided by 1,200 cfm electric compressors. The compressed air piping network will be installed along the main access consisting of an 8 in diameter steel pipe. A smaller 4 in line will be installed in the production panel in the main room. Compressed air will provide power to a small pump for dewatering development work, handheld drills will also provide an emergency supply of air to the refuge station.

16.8.6 Fuel Storage and Distribution

The haulage trucks and all auxiliary vehicles will be fuelled at surface fuel stations. Two fuel/lube cassette truck will be used to distribute the fuel underground to the LHD, Jumbo, Bolter and Scissor lift equipment.

16.8.7 Communications

The mine's communication system will consist of an LTE communication system. Telephones will be located at key infrastructure locations such as the refuge and lunchrooms. Key personnel (mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (LHD, truck, grader and utility vehicle operators) will be supplied with an underground radio connected to the LTE network. This system also makes it possible to transmit the necessary data for the teleoperation of certain equipment.

16.8.8 Explosives Storage and Handling

During the pre-production and first years of production, the explosives will be stored at the surface in permanent magazines. The accessories (detonators) will be stored in a separate magazine at the surface. Once panel rooms become available, an underground explosive and detonator magazine will be prepared. The Study provides for two underground explosives. One at the western part of the mine and the other to the east. Explosives will be transported from the surface magazine to the underground magazine by flat bed service trucks. ANFO will be used as the major explosive for mine development and production. Packaged emulsion will be used as a primer, lifter holes and pre-split blasting.

16.8.9 Personnel and Underground Material Transportation

Supplies and personnel will access the underground via the main access drift. A series of farm tractors modified for the underground will be used to shuttle men from surface to the underground. Supervisors, engineers, geologists will use diesel-powered ATV's for transportation underground. Mechanical and electricians will use maintenance farm tractors. A flat bed with a service boom will be used to move supplies from the surface to the underground active panel. Two service LHD's with forks will be used for material transportation.

16.8.10 Underground Construction and Mine Maintenance

Several crews will be assigned to mine construction and maintenance. Teams will be assigned to maintain ventilation fans, mine brattice and other installations to allow for a good ventilation of the work areas. Another team will be assigned to the maintenance and installation of the conveyors. This team will install the main conveyor, stope conveyor, extend the stope conveyor, move them as needed and provide for their maintenance. Another team would be used to do the remaining underground construction, which includes the shotcrete wall construction and any other construction work. Another team will be used to transport underground material with flatbed trucks and fuel with fuel-lube truck.

16.8.11 Equipment Maintenance

All major mechanical maintenance will be performed at the surface at the workshop. Only minor maintenance and emergency work will be performed underground by mobile maintenance crews. The surface workshop has sufficient warehouse storage for operational requirements.

16.9 Safety Measures**16.9.1 Industrial Hygiene**

All employees will perform health tests (audiogram, breath, etc.) to allow the company to follow their conditions during their tenure at the mine and apply adequate accident prevention programs.

16.9.2 Emergency Exits

Emergency exits underground will consist of the portal ramp, fresh air ventilation raises and manways. The underground alarm system will have a radio alert signal to all the workforce simultaneously when Mercaptan stench gas is introduced in the ventilation system to alert employees, that they need to reach for safety. Pursuant to *Regulation 57.4363*, underground workers need to be drilled every 12 months on emergency exit underground requirements. Pursuant to *Regulation 57,4361*, mine evacuation drills shall be held every 6 months for each shift. All exercises and instruction records will be kept at least one year.

16.9.3 Refuge Stations

Refuge stations are positioned in a way that an employee will need 30 min or less to access the refuge from the moment he leaves his workplace. At Copperwood, both moving and permanent refuge stations will be installed to be airtight and fire resistant. Two permanent and two moving refuges are planned for the Copperwood life of mine. Each refuge station will be equipped with the following:

- Telephone or radio to surface, independent of mine power supply;
- Compressed air, water lines and water supply;
- Emergency lightning;
- Hand tools and sealing material;
- Plan of underground work showing all exits and the ventilation plans.

16.9.4 Fire Protection

Underground mobile vehicles and conveyor belts will be equipped with automatic fire suppression systems in accordance with regulations.

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the electrical installations, pump stations, conveyors, service garages and wherever a fire hazard exists. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

A mine stench gas warning system will be installed at the ventilation and compressed air system to alert underground workers in the event of an emergency.

16.9.5 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with regulations. A mine rescue room will be provided in the administration building. Mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies. An Emergency Response Plan will be developed, kept up to date, and followed in the event of an emergency.

16.9.6 Emergency Stench System

A mine stench gas warning system will be installed at the ventilation (temporary and permanent system) and compressed air system to alert underground workers in the event of an emergency.

16.9.7 Dust Control

Broken ore will be wetted down after blasting and mucking.

17. RECOVERY METHODS

17.1 Process Design

The process plant design for the Copperwood Project is based on a metallurgical flowsheet designed to produce copper concentrate. The flowsheet is based on well proven unit operations in the industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 6,600 mtpd sulphide ore;
- Process plant availability of 91.3% through the use of standby equipment in critical areas and reliable grid power supply;
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

17.1.1 Selected Process Flowsheet

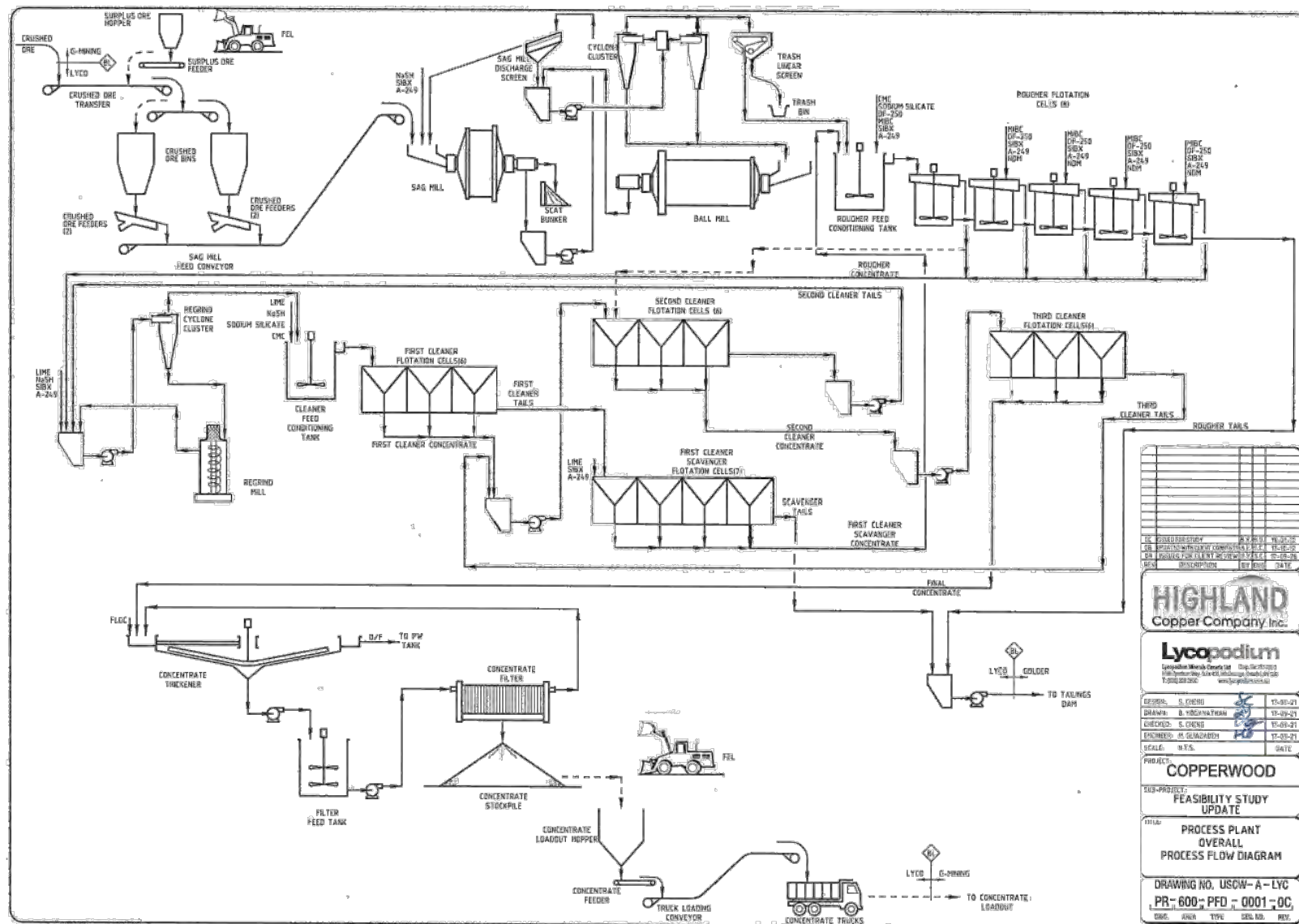
Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork.

The process plant has been designed for a throughput of 6,600 mtpd (dry). The overall flowsheet includes the following steps:

- Grinding and classification;
- Rougher flotation;
- Rougher concentrate regrinding;
- Cleaner flotation, using three stages of cleaning;
- Concentrate thickening and filtration;
- Tailings pumping and disposal in the common Tailings Disposal Facility ("TDF");

Figure 17.1 presents an overall flow diagram depicting the major unit operations incorporated on the selected process flowsheet

Figure 17.1: Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17.1 form the basis of the detailed process design criteria and mechanical equipment list. Process parameters selected are based on preliminary metallurgical testwork carried out at SGS Lakefield (“SGS”) in 2017 and earlier work conducted at Metcon 2011 with consideration of the design head grade. Ongoing optimization metallurgical testwork confirmed process selection and number of flotation stages; however, flotation residence time, flowsheet configuration and reagents may need adjustments according to the final results. It is worthwhile to mention that the high head grade selected for design compared to LoM grade may offset increased residence time and minimize the adjustments required to meet the optimization requirements.

Table 17.1: Key Process Design Criteria

Parameter	Units	Value	Source
Plant Throughput	mtpd	6,600	Highland
Head Grade - LoM	% Cu	1.35	Highland
Head Grade - Design	% Cu	2.2	Highland
	g/t Ag	3.41	Highland
Plant Availability	%	91.3	Lycopodium
Bond Crusher Work Index (CWi)	kWh/t	20.3	Consultant
Plant Operating Time	hr	8000	Lycopodium
Bond Ball Mill Work Index (BWi)	kWh/t	16.2	Testwork
SMC Axb ¹		34.5	Consultant
Bond Abrasion Index (Ai)	g	0.014	Testwork
Grind Size (P ₈₀)	µm	45	Testwork
Rougher Residence Time - Laboratory	min	50	Testwork
Cleaner 1 Residence Time - Laboratory	min	6	Testwork
Cleaner 1 Scavenger Residence Time - Laboratory	min	10	Testwork
Cleaner 2 Residence Time - Laboratory	min	5	Testwork
Cleaner 3 Residence Time - Laboratory	min	3	Testwork
Regrind Mill Product Size (P ₈₀)	µm	20	Testwork
Concentrate Production Rate	t/h	15.1	Calc
Target Concentrate Grade	% Cu	24	Highland
Target Overall Recovery	%	86	Highland
Concentrate Thickener Solids Loading	t/m ² .h	0.20	Lycopodium
Filter Solids Loading	kg/m ² .h	160	Lycopodium

Note: Design A x b value derived from the 85th percentile ranking of specific energies determined for each individual ore type.

17.1.2.1 Comminution

Design parameters for the comminution circuit were sourced from testwork conducted at various laboratories from 2010 and 2017. Orway Mineral Consultants carried out ore characterization and comminution modelling based on this testwork.

Major observations and conclusions from the ore characterization were as follows:

- The comminution testwork has focused on the Copperwood Main Zone which represents 75% of the Mineral Resources. The other 25% of the resources lies to the east of the Main Zone, these are the Bridge zone, Section 6 and, since 2017 resource estimate, Section 5. The comminution testwork for these zones has been analyzed separately but the grinding characteristics has revealed small difference from the CW zone parameters.
- The grinding characteristics of the three ore types that make up the CBS in the Main Zone were analyzed. The variance analysis indicates that the blend of the three ore types can be considered as a single ore with respect to grinding characteristics. As a note of interest, one test each of the Domino and Grey Laminated ore types indicate that Domino may be the softer ore in the CBS blend.
- The results for Sector 5 and Sector 6 show that the material is slightly less competent than the Main Zone. The design of the circuit and equipment sizing will be based primarily on the CW zone due to the small difference in grinding characteristics and the percentage of these zones in the orebody.
- The DFS comminution design criteria will be based on the 85th percentile values of 13.9 kWh/t for the BWi and 34.5 Axb for the impact breakage SMC test. The selected 85th percentile values indicate that Main Zone ore has a high resistance to grinding both in terms of impact and abrasion energy requirement. The available results range from high to moderately high resistance to grinding.
- The A_{is} 0.039 and it can be considered as soft ore.

17.1.2.2 Flotation Circuit

The flotation circuit configuration, residence times, reagent addition rates and concentrate mass recoveries have been selected based on the metallurgical testwork conducted at SGS in 2017 and earlier work conducted at Metcon in 2011 with consideration of the design head grades.

17.2 General Process Description

The process plant has been designed for a throughput of 6,600 mtpd (dry). The overall flowsheet includes the following steps:

- Crushed ore reclaim;
- Grinding and classification;
- Rougher flotation;
- Rougher concentrate regrind;
- Cleaner flotation, using three stages of cleaning;
- Concentrate thickening and filtration;
- Tailings disposal.

17.3 Crushed Ore Reclaim

Crushed ore from the underground mine will be conveyed to a crushed ore transfer conveyor equipped with a weightometer. This conveyor will discharge onto a bidirectional / reversible conveyor which in turn feeds the crushed ore bins. The two crushed ore bins will be equipped with two pan feeders, each to reclaim material to feed the SAG mill feed conveyor. The conveyor will be equipped with a weightometer for measuring and controlling the SAG mill feed rate.

A surplus ore feeding system, comprised of a hopper and a feeder, will allow ore material to be fed to the crushed ore bin via a front-end loader from the ore stockpile if required.

17.4 Grinding and Classification Circuit

The grinding circuit will receive ore at a nominal top size of 203 mm with an 80% passing size of 150 mm. The circuit will consist of a SAG mill in closed circuit with a screen and a ball mill in closed circuit with a cyclone cluster.

The SAG mill will be a 7.92 m diameter x 4.21 m EGL mill with a 5,500 kW motor. The SAG mill will operate with 12% to 15% ball charge. Ore will be fed to the SAG mill at a controlled rate, nominally 274 dry mtph, and water added to the feed chute to achieve the desired milling feed density. Flotation reagents including sodium hydrosulphide (NaSH), alkylaryl dithiophosphate (A-249) and sodium isobutyl Xanthate (SIBX), will also be added to the mill feed. Product from the SAG mill will discharge over a grate with the oversize

reporting to the scats bunker where it will be periodically removed by the skid-steer loader. Grate undersize will be pumped to the SAG mill discharge screen. The screen will be a single-deck inclined screen with a width of 2.4 m and length of 3.7 m. The screen deck will have an aperture of 2.0 mm. The screen oversize will be recycled back to the SAG mill and the undersize will gravitate to the cyclone feed pump box where it will be further diluted to achieve the required cyclone feed density.

The cyclone feed pumps will deliver slurry to the cyclone cluster. Cyclone underflow will gravitate to the ball mill, while cyclone overflow will gravitate to the trash screen. The ball mill will be a 5.80 m diameter x 9.86 m EGL overflow mill, with a 5,500 kW fixed speed motor. The mill will operate with between 30% to 35% ball charge. Product from the ball mill will discharge over a trommel, with oversize reporting to the rejects bin. Trommel undersize will gravitate back to the cyclone feed hopper to be classified again.

Two vertical spindle sump pumps will service the grinding and classification area. The concrete floor under the mill area will slope to the sumps to facilitate cleanup. Grinding media for the mills will be introduced by use of a dedicated kibble.

A separate layout model was developed to accommodate a flash flotation. Space has been identified in the building for future installation of a flash flotation circuit if required.

17.5 Rougher Flotation

Cyclone overflow will gravitate to the trash screen, which will be a linear screen designed to remove foreign material prior to flotation. Trash will report to the trash bin which will be periodically removed for emptying. Screen undersize will gravitate to the rougher conditioner tank. A sampler will be installed on the screen underflow line to take a sample to the On-stream Analyzer ("OSA") for metallurgical, process control and particle size measurement purposes.

SIBX, A-249, frother, and a sodium silicate-carboxymethyl cellulose sodium mixture ("SS/CMC") will be added into the rougher conditioner tank. Process water can be added if required to dilute the feed to the appropriate slurry density.

The rougher flotation cells will consist of eight 130 m³ forced air tank cells in series. Rougher concentrate will gravitate into the regrind cyclone feed hopper. A sampler will be installed on the rougher concentrate discharge line to take a sample to the OSA for process control purposes. The first rougher flotation cell is installed such that the concentrate from the first tank can be directed to the second cleaner flotation circuit and bypassing the regrind circuit during operations if required.

The rougher tailings will gravitate to the flotation tails pump box and a sampler will be installed to take a sample to the OSA for metallurgical and process control purposes.

The facility to dose SIBX, frother, A-249 and n-Dodecyl Mercaptan (“NDM”) along the rougher flotation cells train will be provided so that stage collector and frother additions can be used if required.

The flotation building gantry crane will be used for all maintenance lifting functions within the flotation area.

A vertical spindle sump pump will service this area for spillage cleanup.

Space has been allocated in the area to allow the rougher concentrate from the last three rougher flotation cells be collected in a pump box and pumped to the second cell of the rougher flotation circuit in the future if required. Also, space have been considered for future expansion or circuit reconfiguration (e.g. recycling cleaning concentrate or tailings) following the optimization flotation testwork results.

17.6 Regrind

Rougher concentrate and second cleaner tailings will report to the regrind cyclone feed pump box. The slurry will be pumped to the regrind cyclone cluster by the regrind cyclone feed pumps. The cyclone underflow will gravitate to the regrind mill where water and lime (if required) will be added to achieve the desired milling density and desired operating pH respectively. The regrind mill will be a vertical mill and grinding will be achieved via attrition and abrasion of the particles in contact with steel media.

Mill discharge will gravitate back to the regrind cyclone feed hopper for classification in the regrind cyclones.

Regrind cyclone overflow will gravitate to the cleaner conditioner tank. A sampler will be installed on the cyclone overflow line to take a sample to the OSA for process control and particle size measurement purposes.

Media will be introduced via the regrind media hopper. The media hoist will be installed to allow filling of the regrind media hopper from bulk bags.

A vertical spindle sump pump will service this area for spillage cleanup.

17.7 Cleaner Flotation

Cleaner flotation will consist of three stages of closed circuit cleaning. Final arrangement regarding recirculation of cleaning streams will be made according to ongoing optimization testwork program. The final arrangement includes recirculation of the first cleaner scavenger concentrate and tailings to the regrinding/first cleaner circuit and rougher last cells (scavenger) respectively. The number of cleaning stages and regrinding arrangement will remain unchanged.

Regrind cyclone overflow will gravitate to the cleaner conditioner tank. NaSH, lime and SS/CMC will be added to this tank. The facility to add process water to dilute the slurry to the desired density will also be provided.

The first cleaner flotation cells will consist of six 18 m³ trough cells in series. First cleaner concentrate will gravitate to the first cleaner concentrate, while the first cleaner tailings will gravitate to the first cleaner scavenger flotation cells.

The first cleaner concentrate will be pumped to the second cleaner flotation cells. A sampler will be installed on the discharge line of the pump to take a sample to the OSA for process control purposes.

The first cleaner scavenger flotation cells will consist of seven 18 m³ trough cells in series. Lime, A-249 and SIBX will be added to the first cleaner scavenger flotation feed box where they will mix with the first cleaner flotation tail. First cleaner scavenger concentrate will be collected in a pump box and will be pumped back to the rougher flotation circuit. First cleaner scavenger tailings will gravitate to a pump box from where the material is pumped to the flotation tailings pump box. A sampler will be installed on this stream to take a sample to the OSA for metallurgical and process control purposes.

The second cleaner flotation cells will consist of six 8 m³ trough cells in series. Lime and SIBX will be added to the second cleaner flotation feed box where they will mix with the first cleaner concentrate. Second cleaner concentrate will be collected in a pump box and will be pumped to the third cleaner flotation circuit. Second cleaner tailings will be collected in a pump box and will be pumped to the regrind cyclone feed pump box.

The third cleaner flotation cells will consist of six 2 m³ trough cells in series. Third cleaner concentrate will be collected in a pump box and will be pumped to the concentrate thickener. A sampler will be installed on the pump discharge line to take a sample to the OSA for metallurgical and process control purposes. Third cleaner tailings will gravitate to the first cleaner concentrate pump box.

Two vertical spindle sump pumps will service the cleaner flotation area for spillage clean-up.

17.8 Concentrate Thickening and Filtration

Final concentrate at 15.1 mtpd solid will be pumped to the 16 m diameter high rate concentrate thickener, along with filtrate return from the filtration area. Flocculant stock solution will be further diluted to 0.25% w/w with process water in an in-line mixer prior to addition to the concentrate thickener. Thickener overflow at a flow rate of 41.6 m³/h will gravitate to the process water tank for re-use.

Concentrate thickener underflow, at approximately 60% solids w/w, will be pumped to the agitated concentrate filter feed tank by the one operating, with one standby, 3 x 2 concentrate thickener underflow pump. This tank will provide 12 hours of surge capacity between the thickener and filter. Concentrate will be pumped to the concentrate filter by the filter feed pumps.

Thickened concentrate will be pumped batch wise to the concentrate filter press using the one operating, and one standby, GIW LCC-H 150-500 filter feed pumps. The filter 1,500 mm x 1,500 mm x 40 mm will remove water from the concentrate to meet the target moisture of approximately 9% w/w using a series of pressing and air blowing steps. After the desired filtration time of approximately 12 minutes, the filter press will open and discharge concentrate directly to the floor of the concentrate shed. Following discharge of concentrate, the filter cloth will be washed prior to the next cycle using raw water and Grundfos CR32-9 pump. Some 9.9 m³/h filtrate from the concentrate filter will be returned to the concentrate thickener by gravity. Filter cloth wash will be drained into the filter area sump pump.

A front-end loader ("FEL") will be used to remove the concentrate from beneath the filter press and transfer it to the adjacent 542 t concentrate storage areas. Concentrates will be loaded into the loadout hopper by the FEL when required. Concentrate from the load-out hopper will be transferred to the concentrate trucks via a 900 mm wide concentrate feeder and 750 mm wide truck loading conveyor. The truck loading conveyor will be equipped with a weightometer.

Two vertical spindle sump pumps will be provided in the thickener and filtration area to return spillage to the concentrate thickener.

17.9 Tailings Handling

Rougher and first cleaner scavenger tailings will be combined in a mixing box from where a final flotation sampler will take a sample to the OSA for metallurgical and process control purposes. The mixing box

discharge will combine with a number of intermittent reagent sump pump streams in the flotation tails pump box. Flotation tailings will be pumped to the Tailings Dam Facility ("TDF").

A vertical spindle sump pump will be provided to return spillage to the flotation tails pump box.

17.10 Raw Water, Potable Water and Process Water

Raw water make-up will be supplied to the raw water tank.

Raw water will be used for the following duties:

- Filter cloth wash via the raw water pumps;
- Reagent make-up via the raw water pumps;
- Cooling water, via the raw water pumps.

The decant water will be filtered and used for:

- Low pressure gland water, using the low-pressure gland water pumps;
- OSA.

The quality of filtered water used for GSW and OSA needs to be confirmed by suppliers during detail engineering.

Potable water will be supplied to the potable water tank where a ring main system will be installed to provide potable water to the safety showers and drinking fountains around the plant.

Concentrate thickener overflow and TDF decant water will be sent to the process water tank for re-use in the process plant. Raw water will be used as make-up as required. Anti-scalant will be added to the process water tank as required.

Process water will be used for the following duties:

- Filter manifold wash via the manifold wash water pumps;
- General process uses in the grinding, flotation and thickener areas via the process water pump.

17.11 Reagents**17.11.1 Frother (MIBC/D-250)**

MIBC and D-250 will be delivered in bulk boxes and stored in the reagent shed until required. A permanent bulk box for each reagent will be installed to provide storage capacity local to the flotation area. MIBC and D-250 will be dosed neat, without dilution in a 1:1 weight ratio. MIBC and D-250 will be mixed in a tank and then transferred to a storage tank. Multiple diaphragm style dosing pumps will deliver the reagent to the required locations within the flotation circuit. Top up of the permanent bulk boxes will be carried out manually as required.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.2 Sodium Isobutyl Xanthate (SIBX)

SIBX will be delivered in pellet form in bulk bags within boxes and stored in the reagent shed. Raw water will be added to the agitated SIBX mixing tank. Bags will be lifted into the SIBX bag breaker, located on top of the tank, using the SIBX lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required dosing concentration. SIBX solution will be transferred to the SIBX storage tank using the SIBX transfer pump. Both the mixing and storage tanks will be ventilated using the SIBX tank fan to remove carbon disulphide gas.

SIBX will be delivered to the flotation circuit using the SIBX circulating pump and a ring main system. Actuated control valves will provide the required SIBX flowrates at a number of locations around the flotation circuit.

The SIBX mixing area will be ventilated using the SIBX area roof fan.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.3 Sodium Silicate/Carboxymethyl Cellulose Sodium (SS/CMC)

SS/CMC is a mixture of SS and CMC with a 3:1 weight ratio respectively.

SS will be delivered in bulk boxes and stored in the reagent shed. CMC will be delivered in pellet form in bulk bags and stored in the reagent shed. Bags will be lifted into the CMC bag breaker, located on top of the mixing tank. The solid reagent will fall into the tank and be dissolved in SS and raw water to achieve

the required dosing concentration. SS/CMC solution will be transferred to the SS/CMC storage tank using the SS/CMC transfer pump. Both the mixing and storage tanks will be ventilated using the SS/CMC tank fan.

Multiple diaphragm style dosing pumps will deliver the solution to the required locations within the flotation circuit.

The SS/CMC mixing area will be ventilated using the SS/CMC area roof fan.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.4 N-Dodecyl Mercaptan (NDM)

NDM will be delivered in bulk boxes and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. NDM will be dosed neat, without dilution. A diaphragm style dosing pump will deliver the reagent to the rougher flotation circuit. Top up of the permanent bulk boxes will be carried out manually as required.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.5 Flocculant

Powdered flocculant will be delivered to site in 25 kg bags and stored in the reagent shed. A vendor supplied mixing and dosing system will be installed, which will include flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant will be loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with water. Flocculant solution, at 0.25% w/v will be agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant will be transferred to the flocculant storage tank using the flocculant transfer pump.

Flocculant will be dosed to the concentrate thickener using variable speed helical rotor style pumps. Flocculant will be further diluted to approximately 0.025% w/v just prior to the addition point.

A dedicated vertical spindle sump pump will be provided in this area.

17.11.6 Sodium Hydrosulphide (NaSH)

NaSH will be delivered in bulk boxes and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. NaSH will be dosed neat, without dilution. Multiple diaphragm style dosing pumps will deliver the reagent to the SAG mill and flotation circuit. Top up of the permanent bulk boxes will be carried out manually as required.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.7 Aeroflot 249 (A-249)

A-249 will be delivered in bulk boxes and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. A-249 will be dosed neat, without dilution. Multiple diaphragm style dosing pumps will deliver the reagent to the SAG mill and flotation circuit. Top up of the permanent bulk boxes will be carried out manually as required.

Spillages in the A-249 area will be directed to the frother sump pump.

17.11.8 Hydrated Lime

Hydrated lime will be delivered to site in a tanker and will be pneumatically conveyed from the tanker to the lime storage silo. The hydrated lime will be extracted from the lime storage silo via a rotary valve and screw feeder and discharged into the lime slurry storage tank. Raw water will also be added to the slurry storage tank to achieve the desired lime density.

The lime slurry from the lime storage tank will be distributed throughout the process plant by the lime slurry circulation pump and a ring main, with take-offs distributing lime to the process as required.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.11.9 Anti-scalant

Anti-scalant will be delivered in bulk boxes and stored in the reagent shed until required. Permanent bulk boxes will be installed to provide storage capacity local to each dosing point. Anti-scalant will be dosed neat, without dilution. Positive displacement style dosing pumps will deliver the anti-scalant to the process water tank. Top up of the permanent bulk boxes will be carried out manually as required.

17.12 Services and Utilities

17.12.1 On-stream Analysis System

The performance of the flotation circuit will be monitored by a dedicated OSA system, to allow the operator to make air, level or reagent changes based on real time assays. Analysis will include percent solids, copper, iron, and silver assays.

Cumulative shift samples for laboratory analysis will also be collected via the OSA sampling system. The system will have a stand-alone control, calibration and reporting system but will have the capacity to provide assay data to the plant control system if required.

Process streams that will be analyzed are listed as follows:

- Flotation feed;
- First rougher concentrate;
- Rougher concentrate;
- Regrind cyclone overflow;
- First cleaner concentrate;
- Cleaner scavenger tailings;
- Third cleaner concentrate;
- Rougher tailings;
- Flotation tailings.

Samples will be collected using a combination of sample pumps, pressure pipe samplers and linear samplers as required. Samples will be logically combined after analysis and returned back to the process using vertical spindle style pumps.

17.12.2 High and Low-pressure Air

High pressure air at 700 kPa (g) will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high-pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via the main plant air receiver, with an additional receiver in the grinding area.

Rougher flotation air will be supplied by two low pressure blowers. Cleaner flotation air will be supplied by two low pressure blowers.

18. PROJECT INFRASTRUCTURE

18.1 General

This section discusses the required infrastructure to support the mining and processing operations and includes the following areas:

- Public access road upgrade (County Road 519);
- Site access roads;
- Parking lot;
- High voltage power line and main substation;
- Site electrical distribution;
- Gatehouse;
- Communications network;
- Lake Superior water intake;
- Potable water treatment plant;
- Sewage treatment;
- Covered box-cut for mine access;
- Ore stockpile pad;
- Truck shop, wash bay, warehouse and offices;
- Explosives storage;
- Fuel storage;
- Mill offices and metallurgical laboratory;
- Administration office and assay laboratory;
- Concentrate transload facility;
- Tailings disposal facility in three stages;
- Stream relocations;
- Effluent treatment plant.

Figure 18.1 presents the Copperwood Project site general arrangement and Figure 18.2 presents a close - up view of the general arrangement of the plant area.

Figure 18.1: Copperwood Project Site General Arrangement

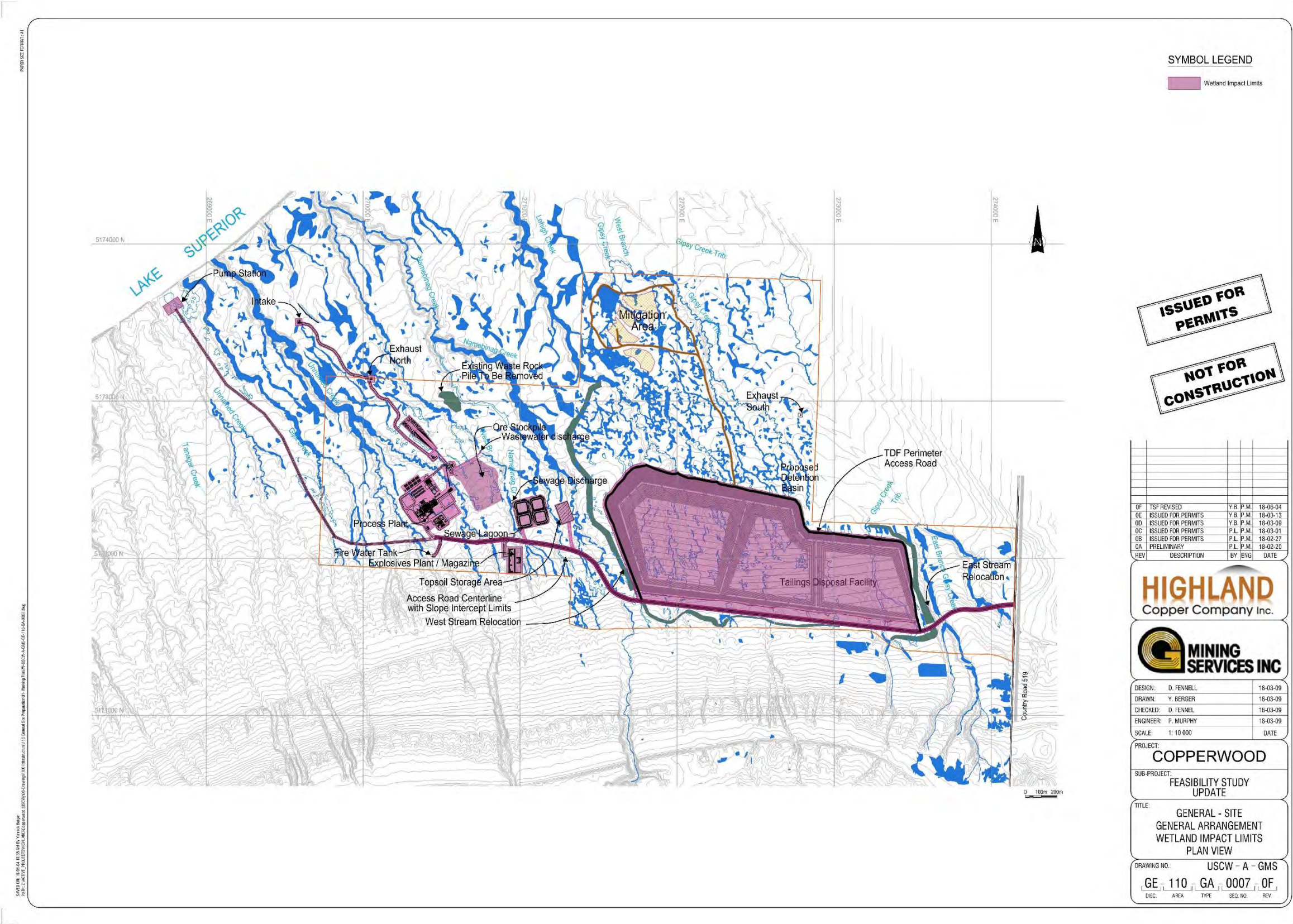
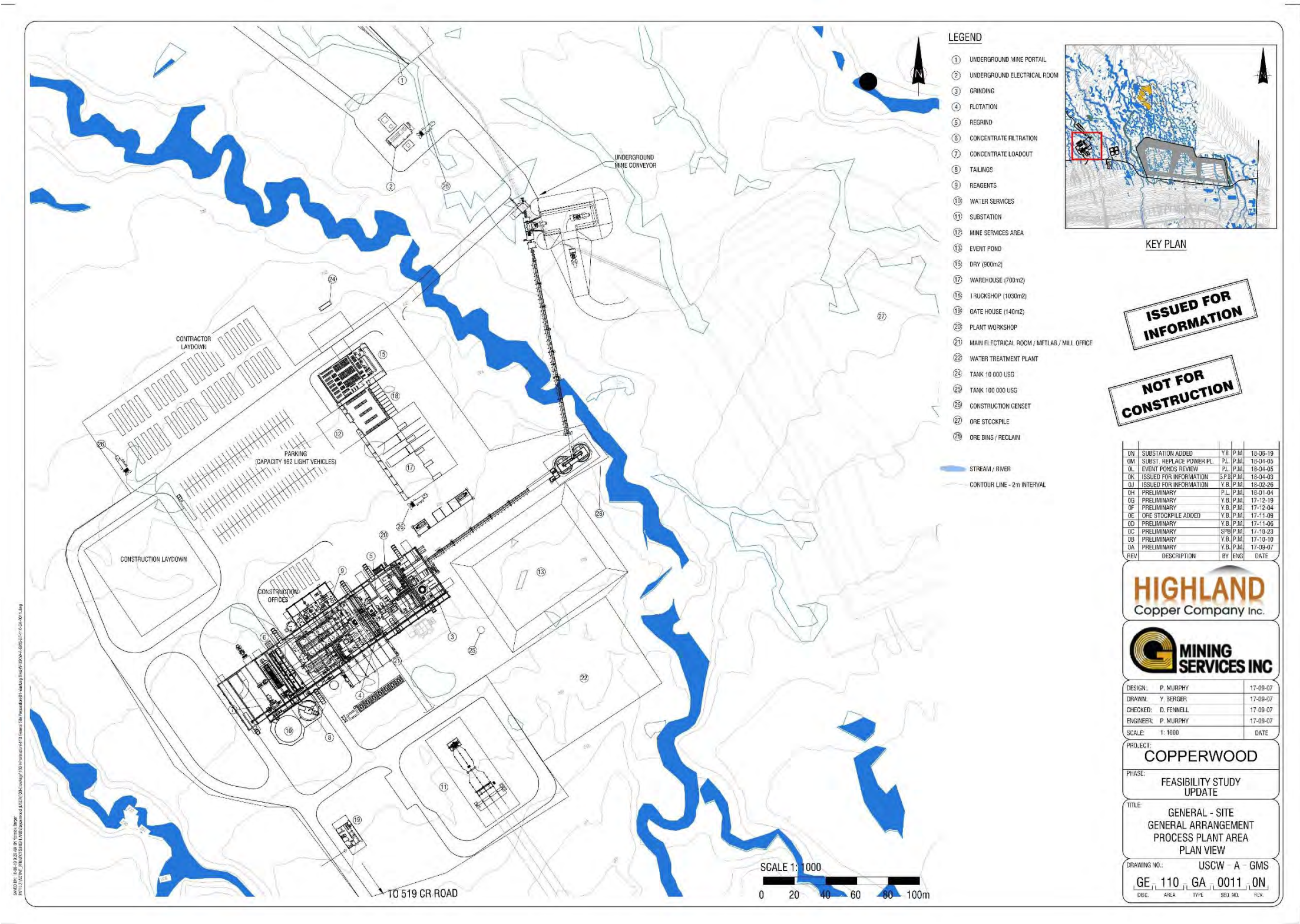


Figure 18.2: Copperwood Project Plant Site Area General Arrangement



18.2 Public Access Road

The Project is accessed via the existing County Road 519 ("CR 519") located on the East boundary of the site. CR 519 connects the site entrance to major roads in the area and will handle all traffic to the site. The site entrance is located approximately 22 km from the Highland Copper Office in Wakefield, MI. Owned and maintained by Gogebic County Road Commission, the road has seasonal limits on truck weight during spring thaw conditions (around the end of April). CR 519 will undergo a major improvement to better handle the increased traffic associated with the development of Copperwood. The improvement will allow the road to be designated as a Class 1 Highway and accept higher vehicle weights without seasonal restrictions. A portion of the road improvement cost is expected to be funded by the Michigan Department of Transportation ("MDOT"). However, a large portion is included in the Project capital expenditures. MDOT is responsible for the design and execution of the road improvement project.

18.3 Communications

A 10 Gb fiber optic cable will run to the Copperwood site. A "backbone" point-to-point ("P2P") radio wave connection using proprietary dishes at emitting and receiving towers will also be put in place and will be connected to a second local fiber provider in order to ensure full redundancy of signal to the mine site in case of interruption of service due to defective equipment or damage to the fiber optic cable.

The proprietary or lease tower will be built at the mine site to install the P2P receiving dish and the LTE's antennas to cover the area of the property. LTE antennas placed on the tower will be part of a surface/underground "Private LTE Network" ("PLTEN") to insure communication between workers (within as well as outside of the mine site). PLTEN will also be used to maximize any potential use of the "IoT" (Internet of Things) by connecting mobile and fixed equipment, computers and telemetries to help in performing live monitoring and data capture.

A traditional Gigabit Wi-Fi connection connected to a "LAN" (Local Area Network) will also be installed in the offices, mill, maintenance shop and other specific locations; in order to upgrade to the LTE/5G network once all the personnel and routing equipment capable of handling the increased network capacity are in place.

Cloud based software applications, including ERP, are preferable in limiting CAPEX expenses as well as maintenance/support costs related to the equipment's "On Premise" software licenses.

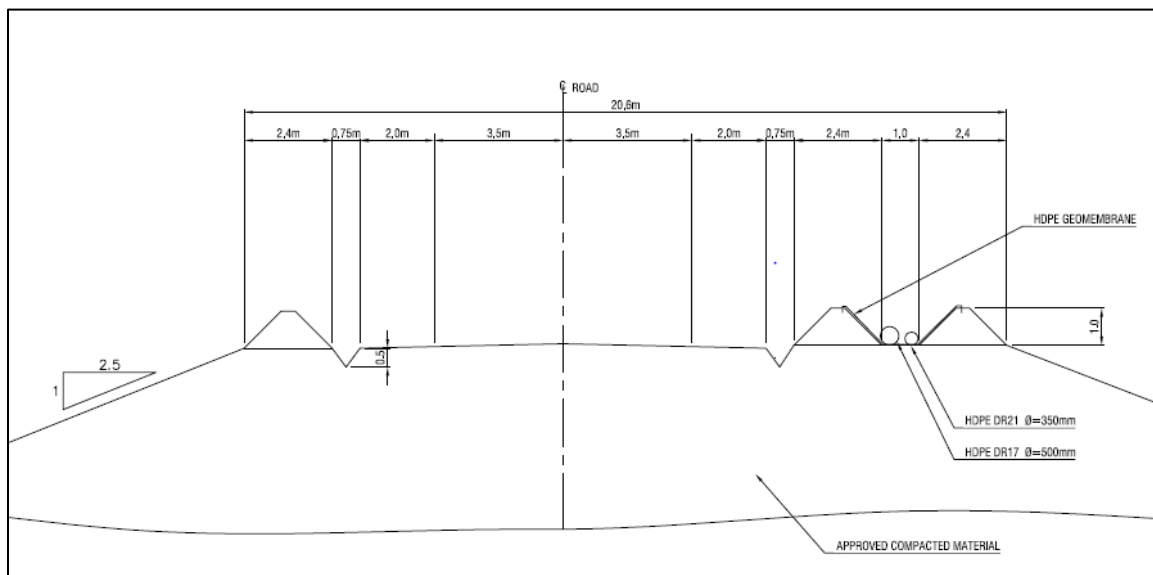
18.4 Site Roads

18.4.1 Main Access Road

The site is largely undeveloped except for a network of trails that have been maintained or improved to allow for access for various site exploration activities including drilling and environmental monitoring. In general, roads will use existing trails as much as possible, including the main access road.

The main access road connects the mill area to the public road, CR 519. All traffic coming to and leaving the site will use the main access road as it is the only road connecting to CR 519. The road runs in a primarily East-West direction across the site on the south side of the Tailings Dam Facility (“TDF”). The distance between the site entrance and the mill site gate house is approximately 4.1 km. The reclaim system pipelines run along the north side of the road between the decant barge location and the mill area. The geometry of the road is designed based on a speed of 40 kmph (25 mph) with consideration given to maximum and minimum grades required for heavy trucks travelling on this road. Steel culverts will be used for the road’s stream crossings. The full length of the road will use an aggregate surface course placed directly on compacted subgrade. The road has one 3.5 m wide lane and one 2.0 m shoulder in each direction with containment ditches and safety berms outside of both shoulders. The north side of the road has an additional safety berm to contain the reclaim pipeline. The two-berm containment system is lined with a high-density polyethylene (“HDPE”) membrane for potential spill containment. Safety berms have a height of 1.0 m. The resulting total road width is 20.6 m.

Figure 18.3: Main Road Cross-section

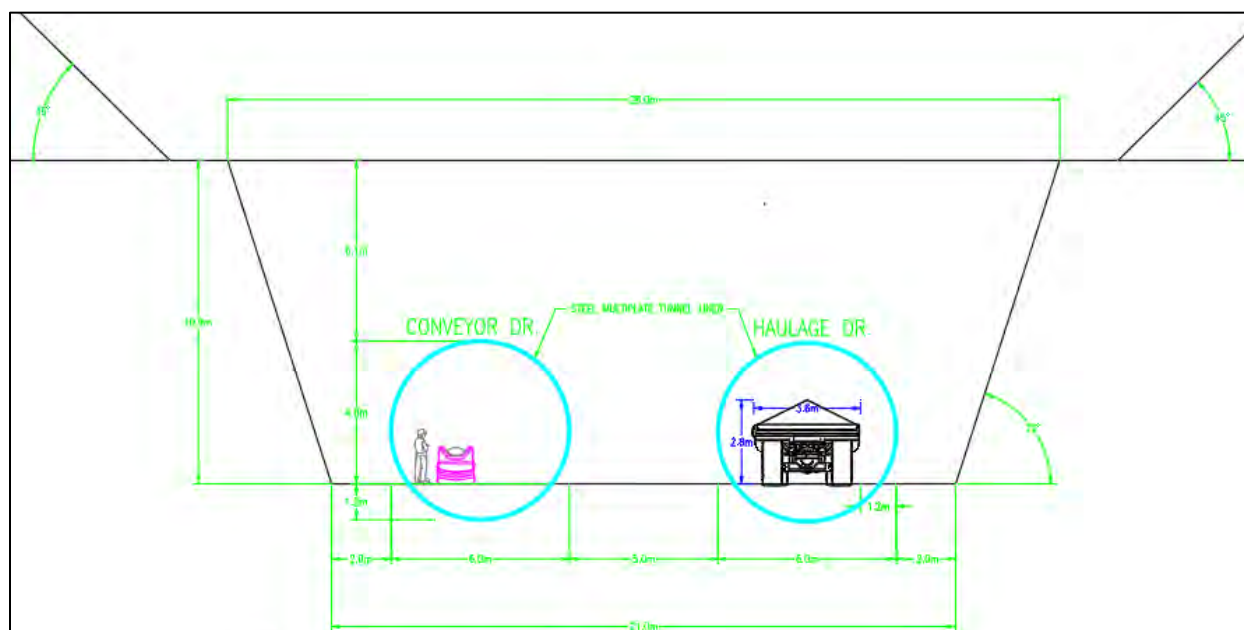


18.4.2 Site Roads

Site roads will connect the infrastructure located outside of the mill area to the actual mill area and the main access road. The site roads will use existing trails as much as possible to reduce clearing and negatively impacting the wetlands. The existing trail will provide access to the topsoil storage area, sewage lagoons, ventilation raises, mitigation area and water intake. The total length of the upgraded trails is approximately 6.5 km. Secondary access roads that will require entirely new construction include; a small portion of the lake intake road, explosive magazine access road, fire water tank access road, and the road between the box cut and the site. The total length of newly constructed site roads is approximately 1.0 km. Both newly constructed site roads and upgraded trails will have the same cross section with a 6 m wide aggregate driving surface placed directly on the subgrade along with ditches on both sides. The typical ROW for site roads is approximately 11 m.

18.5 Box-Cut and Ore Stockpile**18.5.1 Box-Cut**

The box-cut entrance is located approximately 150 m northeast of the mill area and provides access to the mine dry, maintenance shop and warehouse from the South, and the ore stockpile to the West. The box cut design will have approximately a 250 m long ramp with a 15% elevation grade that grants access to the mine portal and underground mine. The box-cut will be excavated at a minimum of 15 m into the fresh rock, where tunnel multi-plate liners will be placed, and then backfilled for water management. The box-cut uses two separate 6 m diameter fully round steel tunnels. The culvert of the steel tunnel is backfilled to create a driving surface for the mine equipment.

Figure 18.4: Box-cut Design

18.5.2 Ore Stockpile Pad

The ore stockpile pad is located 200 m South East of the top of the box cut ramp. The ore stockpile is designed with a capacity of 600,000 mt at a maximum height of 15 m. Over the pre-production period, the ore will be hauled with mining trucks to the stockpile pad. After the end of the initial construction period a stacker will be used to manage the stockpile. Ore will be transferred from the ore stockpile to the mill feed conveyors using a front-end loader and a feeding chute. The stockpile will reach its maximum capacity around the same time as the commissioning period of the process plant.

The pad is approximately 65,000 m² in area and will consist of at least 300 mm of low permeability fill placed on top of the existing ground. The fill will be covered by an HDPE geomembrane. Water that contacts ore on the pad is considered contact water and must be directed to the TDF. The stockpile has a cross-slope that directs all runoff water into lined ditches. The water will eventually drain to a collection point on the NW corner of the stockpile where it will be pumped to the event pond and ultimately to the TDF or the water treatment plant at a later date in the life of the mine.

18.6 Water Management

18.6.1 Sewage Treatment

Sewage treatment will be handled using stabilization ponds. The stabilization pond will be twice as big as the design used in Orvana's Feasibility Study. The ponds will release the treated water into the West branch Namebinag Creek via a pipeline to a discharge point approved by MDEQ.

18.6.2 Water Filtration

Gland water and OSA water will come from filtered reclaim water from the TDF. The water filtration unit will process all the water that is to be used as gland or OSA water.

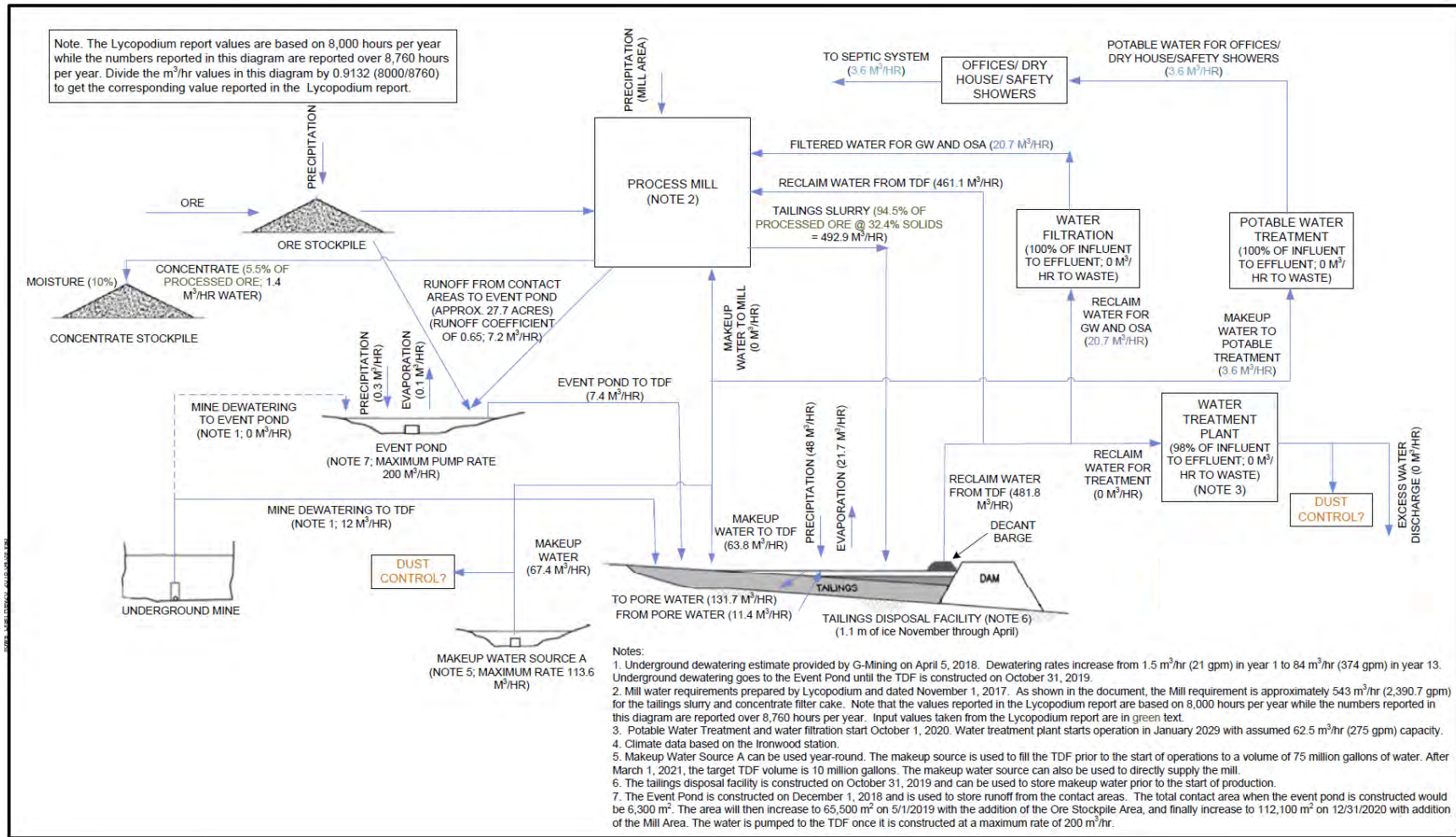
18.6.3 Water Treatment Plant

The water balance model was developed to reclaim and re-use as much water as possible during operations. The general operating strategy assumes that the makeup water source will be used to supply water to the TDF prior to the start of operations so that there is sufficient water available for the mill. The target volume of unfrozen water prior to the start of operations through March 1, 2021, is 283,906 m³.

The total volume of water is obtained from direct precipitation, run-on from the TDF liner area, water pumped from the event pond at the mill site, and makeup water (at a maximum rate of 500 gpm). After the start of mill operations, the makeup water source will continue to be used as needed to maintain a total TDF pool volume (frozen and unfrozen water) of 94,635 m³. After March 1, 2021, the makeup water source will continue to be used as needed to maintain a total TDF pool volume (frozen and unfrozen water) of 37,854 m³. The makeup water source will not be used to store additional water in the TDF once the WTP is operational. This strategy will help to maintain discharge and makeup water requirements during operations. The model assumes the WTP will start treating and discharging at the beginning of year 5.

Figure 18.5 shows the water balance schematic.

Figure 18.5: Water Balance Schematic



The water balance model inputs include the following:

- Production schedule;
- Climate data;
- Underground mine flow;
- TDF operation approach;
- WTP start date and capacity;
- Potable water treatment;
- Event pond and contact area runoff to the TDF.

The Water Treatment Plant (“WTP”) influent water quality by source and the blended influent water quality. The blended IDB water quality is based on a mass balance using the predicted quantity of each source water being blended. The TDF contaminant concentration can be managed throughout active mining operations. The TDF water quality in Year 10 is projected to contain the maximum influent contaminant concentration during WTP operations.

The WTP design is discussed in the TDF section. Its design capacity is estimated at 275 USGPM and discharges in the Namebinag Creek at a permitted point north of the mine portal.

18.6.4 Water Treatment Plant Design

A feasibility design of the WTP was completed by Golder in 2012. As a part of the current Feasibility Study and in relation with the updated water balance evaluation, Golder reviewed their design to verify its compliance.

18.6.5 Influent Design Basis

The Influent Design Basis (“IDB”) model made by Golder includes, water quality, water quantity and the treatment requirements for the TDF water. For the design, Golder considered the following three sources:

- Water in tailings slurry;
- Underground mine water;
- Precipitation and contact water surface run-off.

With the actual water balance data, it is estimated that the WTP would be required in year 5 of mining operations. The WTP operation would be required for a period of 12 years after the end of the tailings discharge when the supernatant pond would be drained.

Golder acquired data on the water chemistry from the tailings slurry, underground mine water and precipitation from different sources. The 2012 model was reviewed and updated.

For the current design, it is assumed that the geochemical models developed for the 2012 study were still valid due to the lack of new information.

The treatment goals from the 2012 Study were reviewed and based on the following:

- The Michigan Department of Environmental Quality Rule 57 Water Quality Values (October 21, 2016);
- The National Pollutant Discharge Elimination System (NPDES) permit expiring October 1, 2019, for the Kennecott Humboldt Mine.

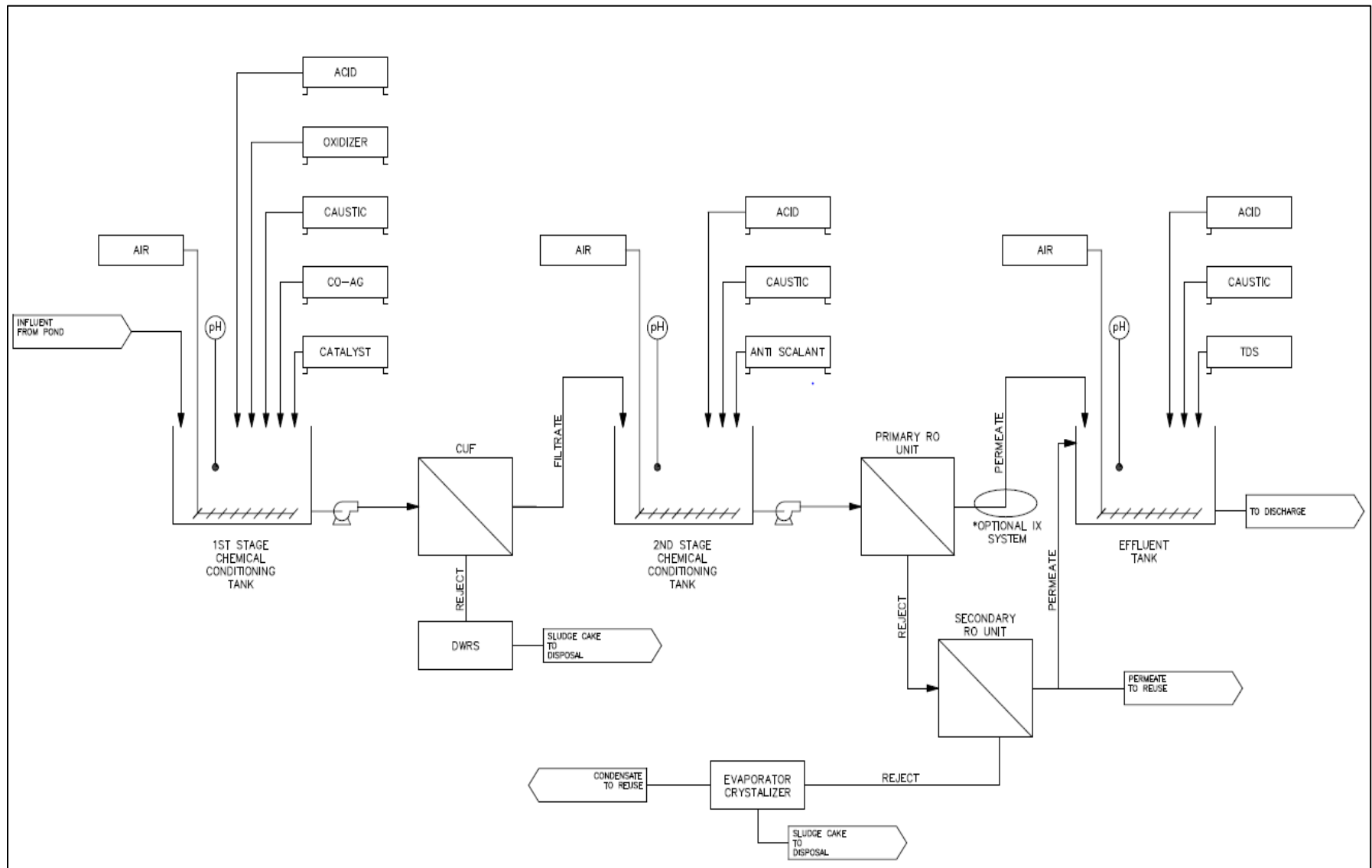
18.6.6 Water Treatment Plant Process

Based on the IDB and requirements for the Chemicals of Potential Concerns (COPC's), the treatment system will include:

- Influent equalization;
- Chemical treatment and microfiltration;
- Granular activated carbon;
- Reverse osmosis;
- Ion exchange;
- Effluent equalization and PH adjust;
- Evaporator;
- Chemical storage and feed;
- Compressed Air.

Figure 18.6 shows the process flow diagram of the WTP.

Figure 18.6: Process Flow Diagram of Water Treatment Plant



18.7 Potable Water – Lake Superior In-Take

The site water balance shows that fresh make-up water is required to facilitate the process as well as to provide easier treatment for potable water. Therefore, the process to secure a permanent water source from Lake Superior has started in 2018. The water in-take construction, plus other related infrastructure, such as power supply, access road and pipeline are also included in this Study. The nominal capacity of 500 USGPM (115 m³/hr) is taken into account and will provide make-up fresh water to the plant and potable water for the offices, showers at the dryhouse and safety showers in all process and laboratory areas.

18.8 Diesel Fuel Storage

A fuel storage, strictly for mining and support mine equipment, will be built. The dike tank for diesel is designed to have a capacity of 10,000 l with pumps and concrete pads, which are located south of the mine entrance.

18.9 Power Supply and Distribution

The feasibility study investigated the construction of a natural gas power plant or the building a power line to the Norrie substation. For the purpose of this Study, the power line option was retained.

The cost estimates for a 40 km long 115 kV line includes the tie-ins to the Norrie substation, the Copperwood site main substation and easements to connect to the Norrie substation in Ironwood. Conversations with the utility company indicate that the capital costs would be factored into the power rates and planned consumption over the life of mine of the Copperwood Project. These rates are reflected in the mine OPEX. Based on the information from the utility company, completion of the power line and other related infrastructure will be completed by Q4 2020 or early Q1 2021, which falls in line with the current Project schedule.

From the mine substation, a site distribution network of 13.8 kV will be built to provide power to all areas. Overall, approximately 22 MW will be required to adequately service the Project.

18.10 Fire Protection

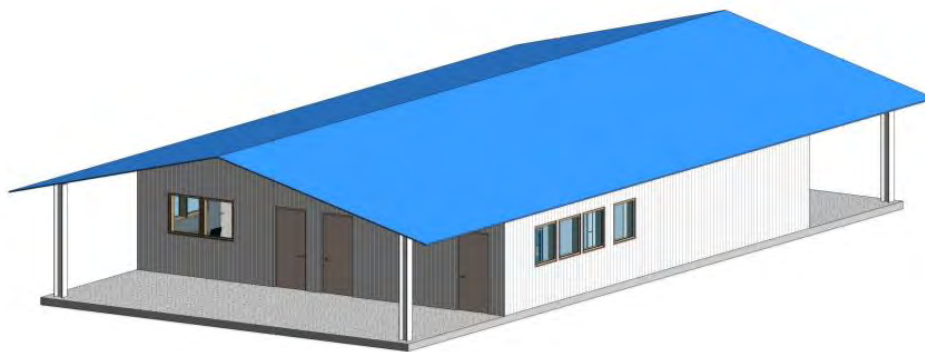
Water for emergency fire extinguishing will be stored in an underground tank south of the Gate House. The tank will be located south of the main access road to isolate it from other infrastructure and increase its elevation compared to the process plant. Two tanks will have a 50,000 USG capacity each for a total of

100,000 USG. Fire pumps will provide the proper water flow and pressure as stipulated in the North American codes.

18.11 Security

The site access will be secured by the gatehouse located adjacent to the main access road in the southern portion of the process area. All traffic coming to or leaving the process area will pass through the gate house. The rooms in the gate house will include; visitor registration, security office, induction room, vehicle control room, and bathrooms. The interior area of the gate house will be 150 m with two covered 50 m² concrete aprons. Vehicle access will be controlled by a boom gate.

Figure 18.7: Gatehouse



18.12 Explosive Magazine

The explosive magazine. South East of the process area. will be located on the South side of the main access road. The dimensions of the explosive magazine are 85 m x 155 m design with protective earth berms that will ease the traffic in and out of the storage facilities. The explosive material will be stored in containers designed to satisfy safety requirements.

18.13 On-site Buildings

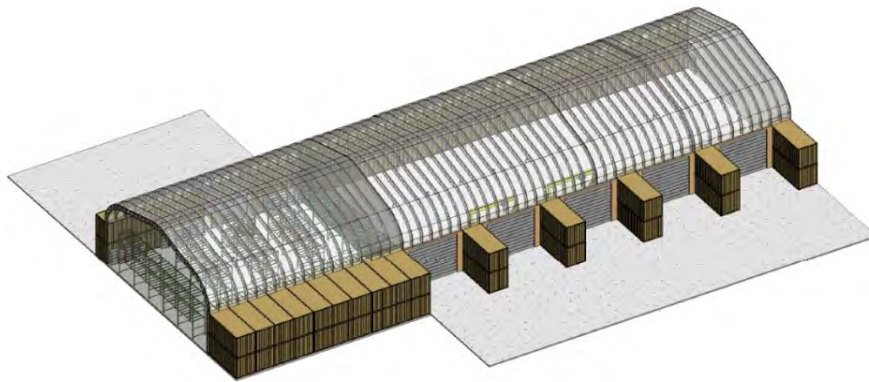
18.13.1 Truckshop, Warehouse and Related Offices

The truck shop and warehouse are connected to and located in the northwest part of the site. The truck shop and warehouse will share a single insulated tension fabric roof set on top of the containers but are divided with a fabric wall. The truck shop will be used primarily for heavy-duty vehicle maintenance. The truck shop will have 5 separate bays which will each be equipped with a 6 m wide by 5 m high roll-up door. Two bays will have their own railed gantry crane with a 15 t capacity which can be moved inside or outside

of the building. One bay will be used for washing purposes and the remaining bays may be used for welding or general maintenance. Some of the containers supporting the dome will be insulated and converted into offices. Water used to clean vehicles in the truck shop is to be considered as contact water and will be collected and sent to the event pond.

The warehouse will have racking to store spare parts and consumables. The warehouse's interior dimensions will be 20 m x 25 m. The warehouse and truck shop will both have concrete aprons to better handle heavy vehicle traffic.

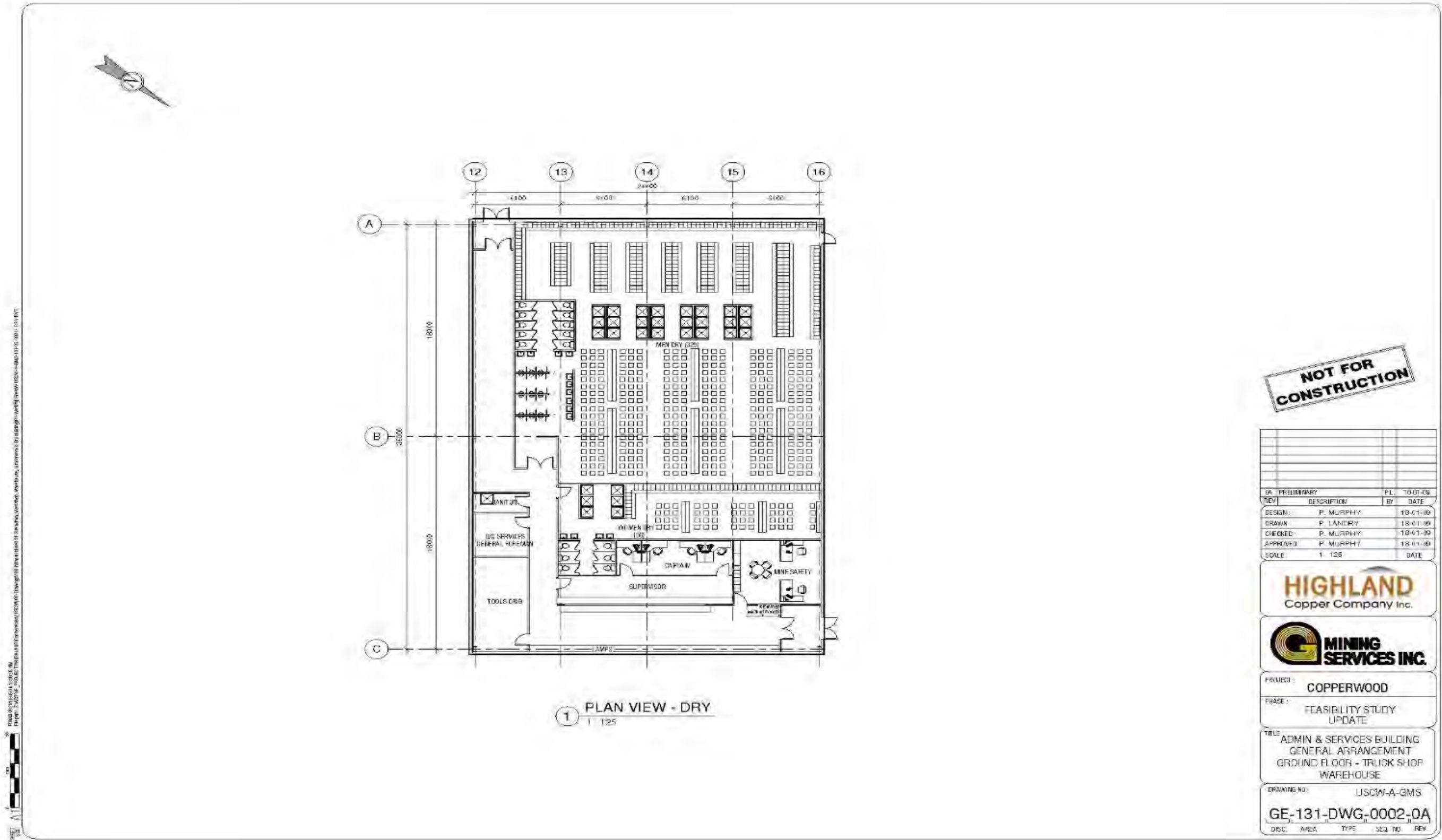
Figure 18.8: Maintenance Shop and Warehouse



18.13.2 Mine Dry

The mine dry will be adjacent to the truck shop and warehouse. The dry will serve as the locker room for the mine workers between shifts and contain the mine rescue equipment, medical offices and a few offices for management personnel. The dry has enough locker space for a total of 375 workers. The men's portion accommodates 325 workers and includes showers, toilets, urinals, lockers and baskets. The women's portion accommodates 50 workers and includes showers, toilets, lockers, and baskets. The dry is a pre-engineered steel-clad building.

Figure 18.9: Plan View – Dry



18.13.3 Construction Offices

The construction offices are containerized and located just north of the process plant. Six trailers with built-in-place corridors are planned and provide enough space for the Owner's construction management. The construction offices will serve as the office space during the construction phase and for general office space over the life of the mine.

18.13.4 Met Lab and Mill Offices

The met lab and mill offices are located on the south side of the mill area. This building will provide a metallurgical testing area and office space in the process building. The building can be accessed from inside the mill area or from the outside.

Contractor Laydown

The laydown area will be North of the parking lot. It provides space for contractor container units and access to utilities. Eight to ten contractors should be active at the same time on the Project and sufficient space is available for offices and shops. A common laydown under the supervision of the owner's team will control accesses and items location.

18.13.5 Off-Site Buildings

The following areas are considered Project infrastructure for mining operations, but are located off the Copperwood site:

- Administration building and assay laboratory;
- Transload facility.

18.13.6 Administration Building and Assay Laboratory

The administration building and assay laboratory will be located in Wakefield using already built spaces. The actual plan takes into consideration lease spaces in the vicinity of Wakefield. Included in the Project costs are major upgrades for plumbing and HVAC as well as architecture renovations and furniture. For the assay lab, all technical equipment is to be purchased by the Project.

18.13.7 Transload Facility

The transload facility will be located at a rail siding in Park Falls, Wisconsin 140 km from site. The location has been chosen due to the costs and mainly because it provides access of the Canadian National Railway networks, for easy shipment to known economical smelters. The facility accepts concentrate shipments from site via side-dump haul trucks. Haul trucks enter the building, dump the concentrate, and exit the building. Concentrate is loaded into rail cars using a front-end loader. The building is fully enclosed to ensure the control of air quality with sufficient air changes, as per the usual codes. Entrances and exits will have roll-up style doors to regulate airflow through the building. Each haul truck carries a concentrate payload of around 18 metric tons, a weight that is limited by the Wisconsin DOT to a maximum gross vehicle weight of 80,000 lb.

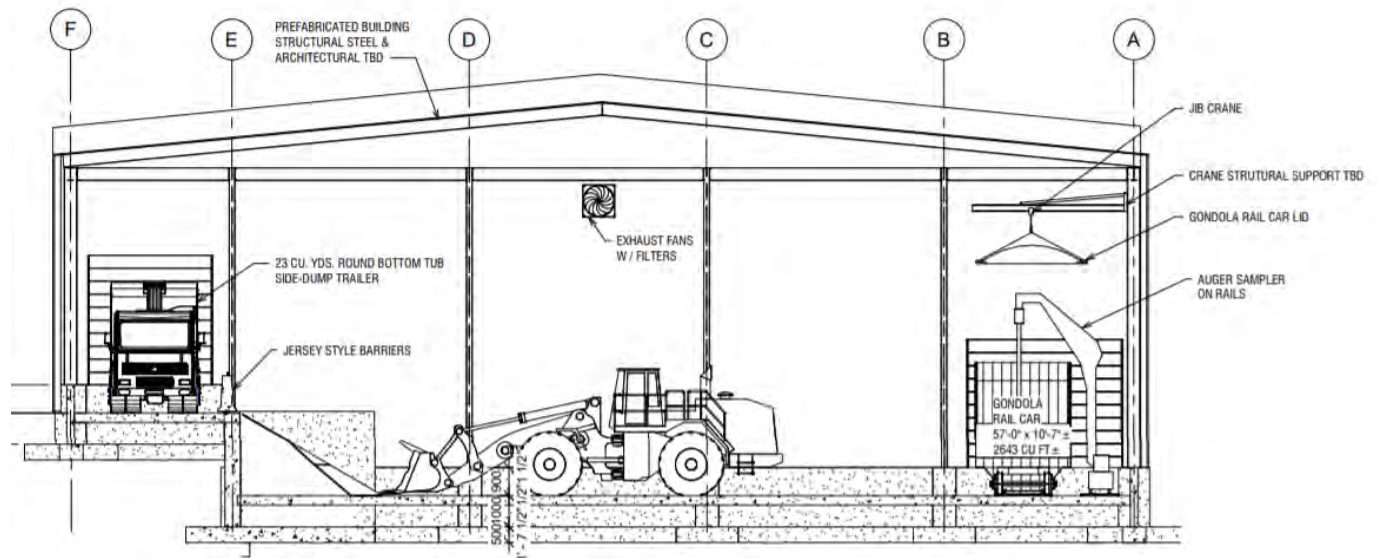
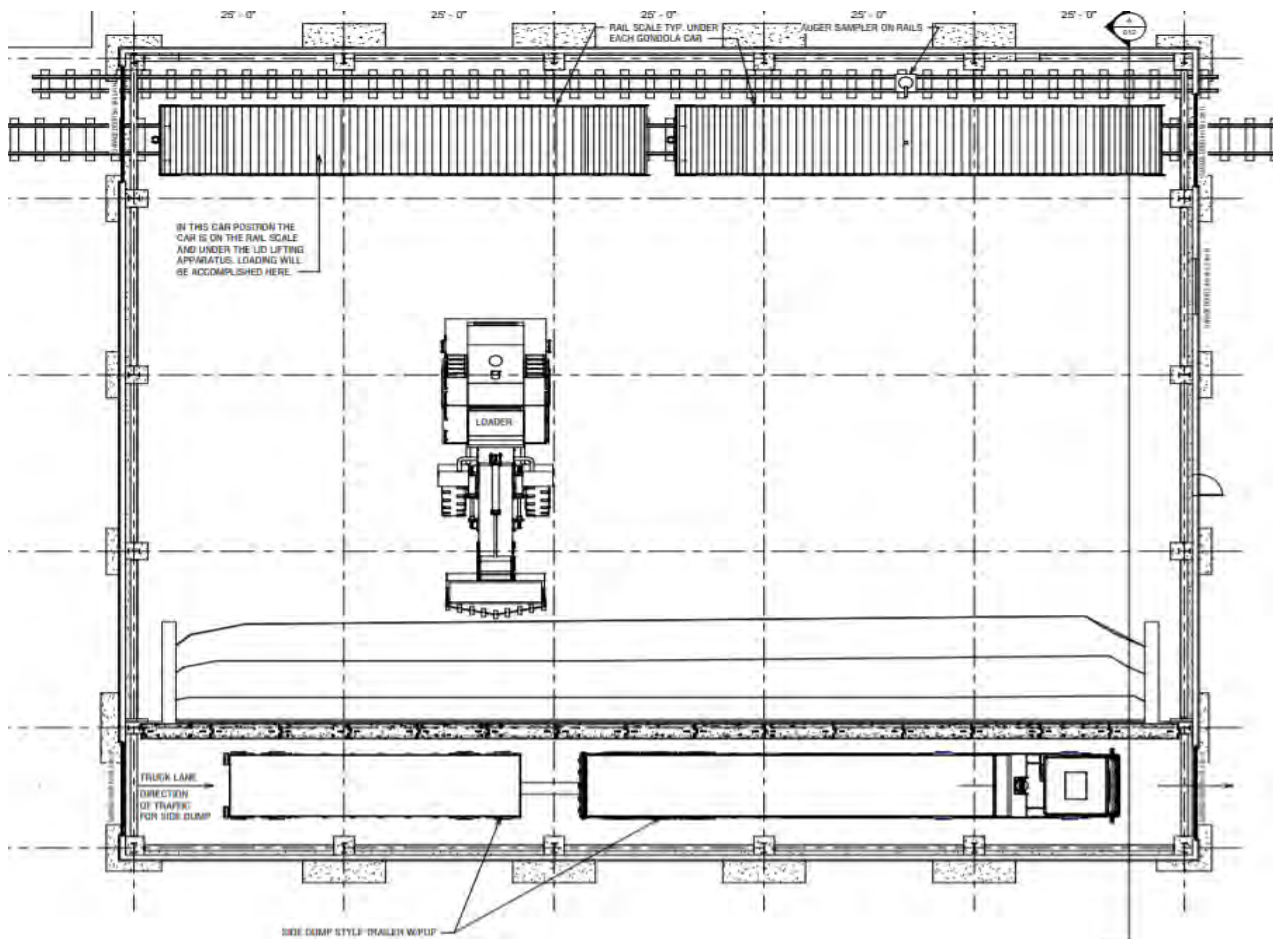
Figure 18.10: Transload Building Cross-section

Figure 18.11: Transload Building Plan View

18.14 Site Vehicles and Mobile Equipment

Light vehicles and pick-up trucks are planned for the construction managers and services at the warehouse.

A front-end loader will be utilized partly at the ore stockpile, partly as a fork lift at the warehouse as well as for other purposes.

18.15 Tailings Storage Facility

18.15.1 General Arrangement and Development

The TDF has been designed to account for the subsurface conditions, the anticipated embankment fill materials, the water and tailings storage requirements, and the physical characteristics of the tailings. The principles for optimizing the TDF to the proposed design were:

- Mostly balanced cut and filled within the TDF's Footprint;
- Construction of the facility by stages;
- Storage capacity for the planned mine production and estimated TDF water balance.

For conservatism, the TDF was designed to sustain the entire ore to be processed. The proposed facility footprint will cover approximately 320 acres (2,000 m x 780 m). The staged construction of the facility will extend 2,000 m from East-to-West and 780 m in the North-South direction.

Table 18.1: TDF Capacity

Stage	Stage 1	Stage 2	Stage 3
Embankment Crest Elevation (m)	289	289	289
Total Cut (m ³)	1,743,487	3,668,544	5,552,327
Total Fill (m ³)	1,075,167	2,924,060	6,165,041
Footprint Area (m ²)	332,938	716,767	1,193,217
Total Cumulative Storage (Mm ³)	4.01	9.96	19.21
Assumed Average Tailings Density (t/m ³)	1.077	1.183	1.270
Approximate Tailings Storage (Mt)	4.32	11.78	24.40

The embankment will be constructed sequentially using downstream methods, meaning that the upstream toe will remain fixed while the downstream toe will progressively advance downstream as the embankment height increases. The crest elevation for each stage was estimated using the current mine production schedule and the storage capacity curves developed for the TDF basin presented in Figure 18.1. The development stages are presented in Table 18.1.

The construction stages are presented in Figure 18.12, Figure 18.13, and Figure 18.14.

18.15.2 Embankment Disposition

The embankment was designed as basin-fill and as a water containment dam. It will be raised in stages using the conventional downstream method of construction. The embankment will be defined by the following layers:

- Seal Zone (Zone 1) – Moisture conditioned and well compacted glacial till creating a low-permeability zone to minimize seepage through the embankment;

- Chimney Drain (Zone 2) – Free-draining materials acting as a filter and drain between Zone 1 and the Embankment Fill (Zone 3). If any seepage, it would be collected routed out of the dam to prevent a phreatic surface from developing across the dam.
- Embankment Fill (Zone 3) – Compacted glacial till material. This zone provides the structural stability to the embankment.
- Embankment Foundation Drains – Free-draining materials. The drains will cover two-thirds of the embankment footprint and will be connected to the chimney drain to prevent a phreatic event within the embankment.

The configuration and dimensions of the embankment are shown on Figure 18.15 and Figure 18.16.

Figure 18.12: Tailings Disposal Facility – Stage 1

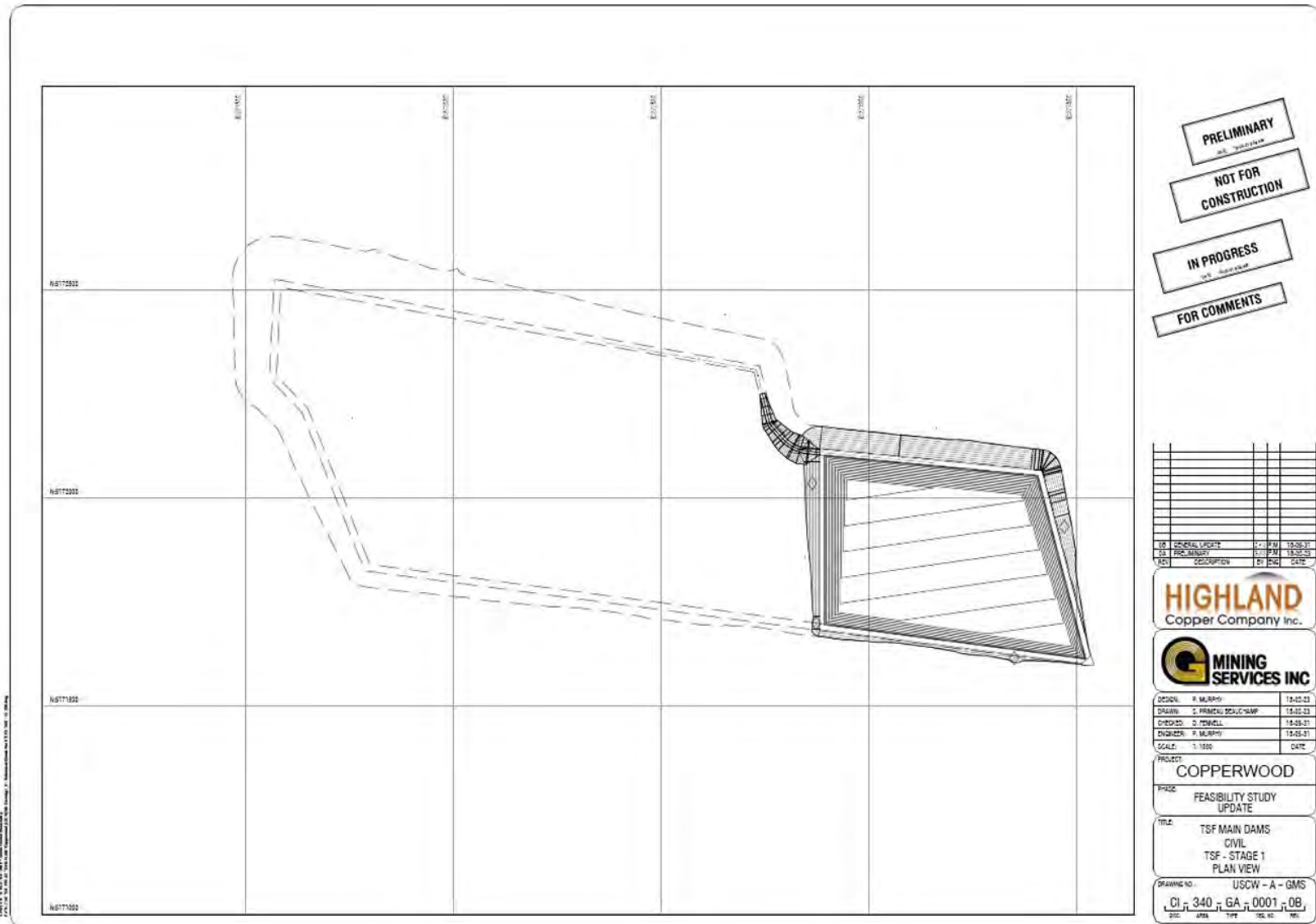


Figure 18.13: Tailings Disposal Facility – Stage 2

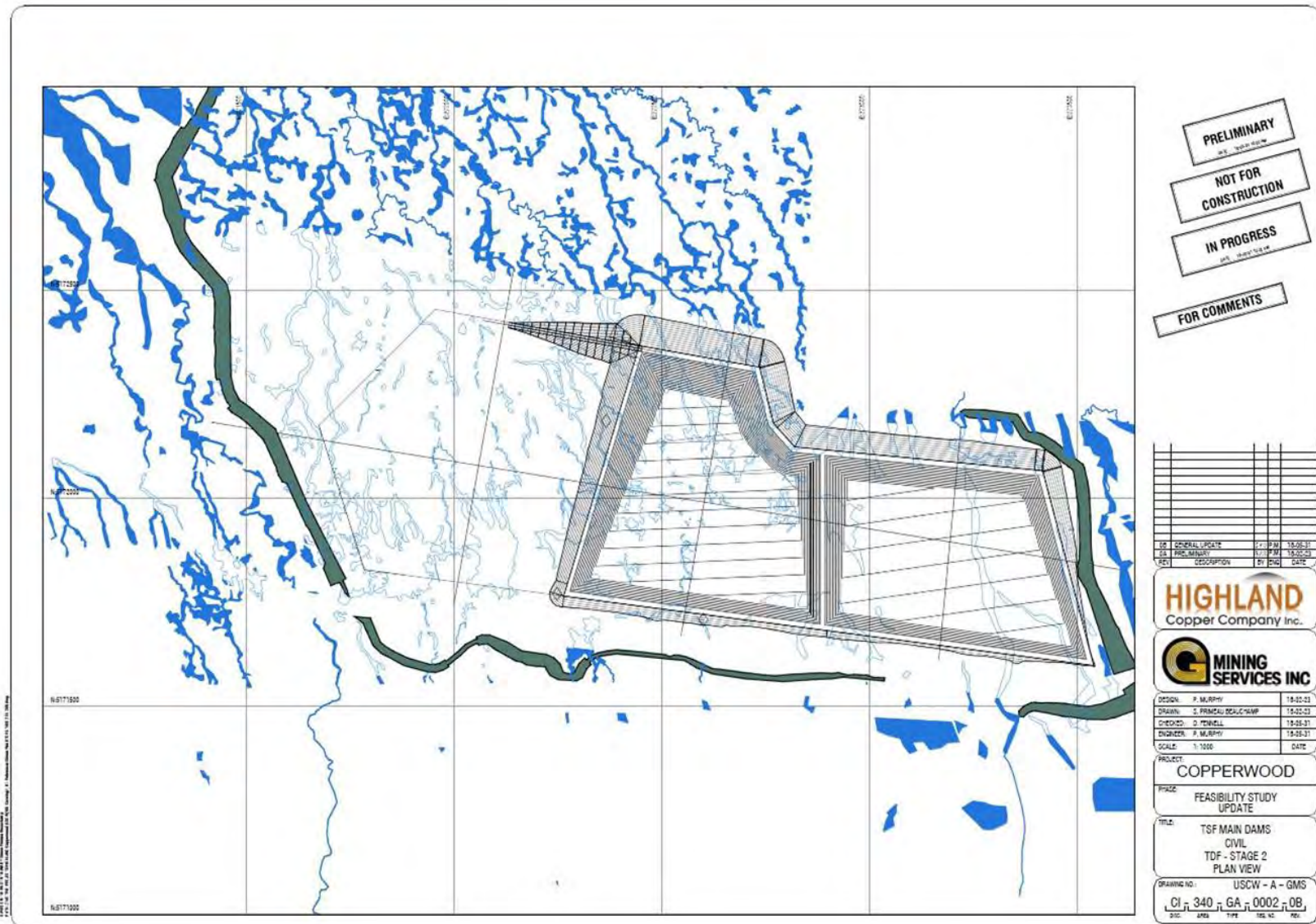


Figure 18.14: Tailings Disposal Facility – Stage 3

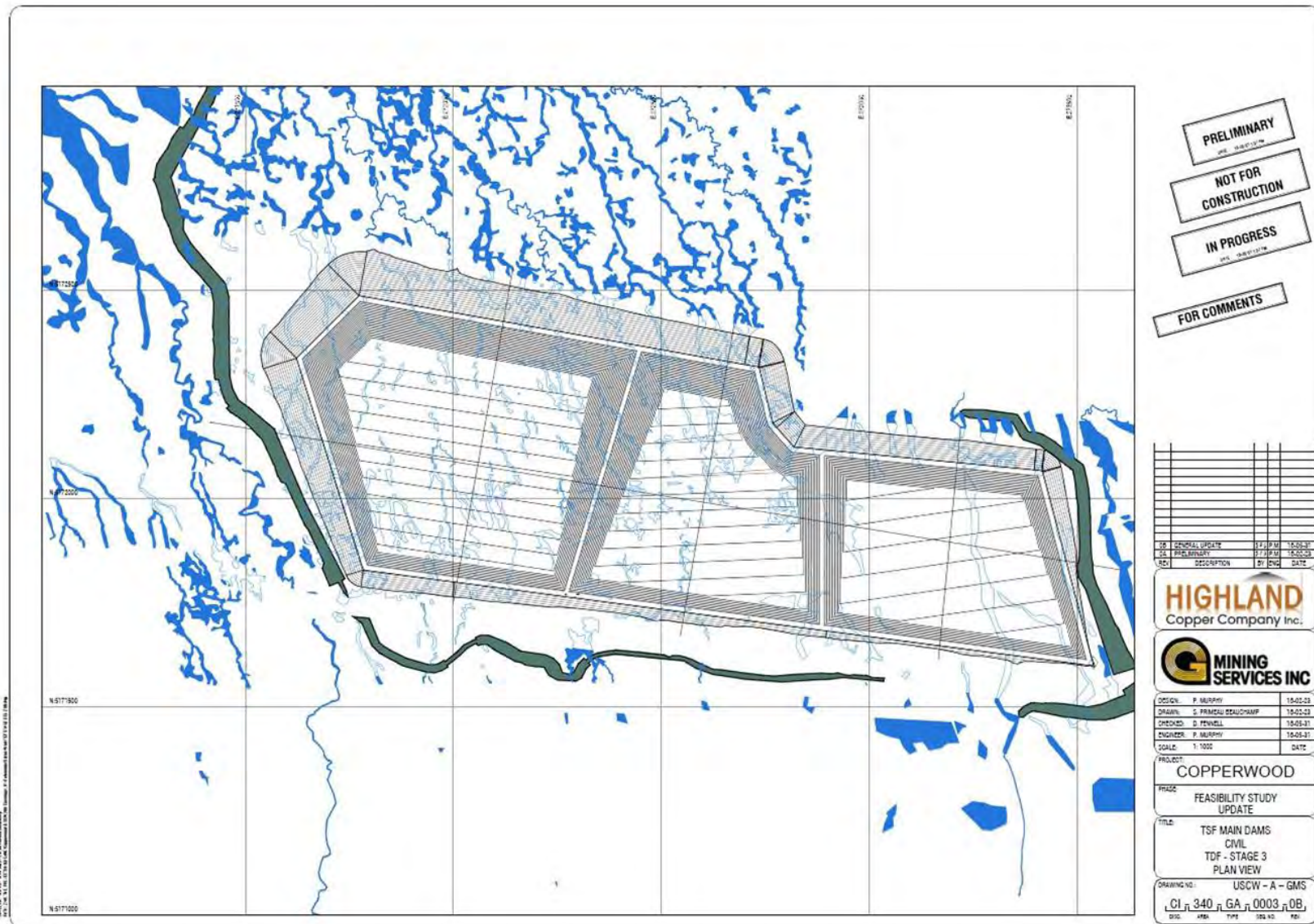


Figure 18.15: Embankment and Basin Details (1 of 2)

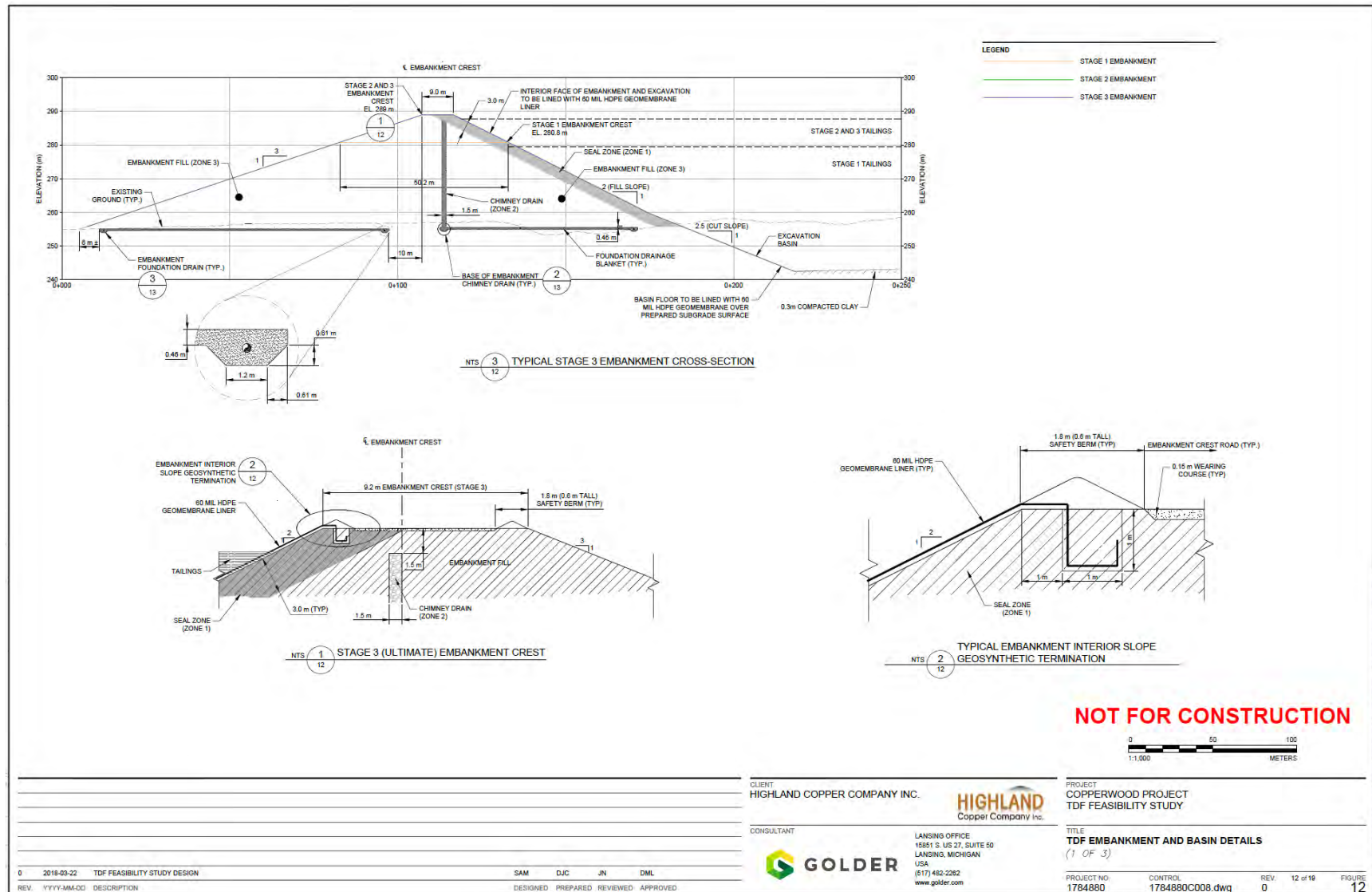
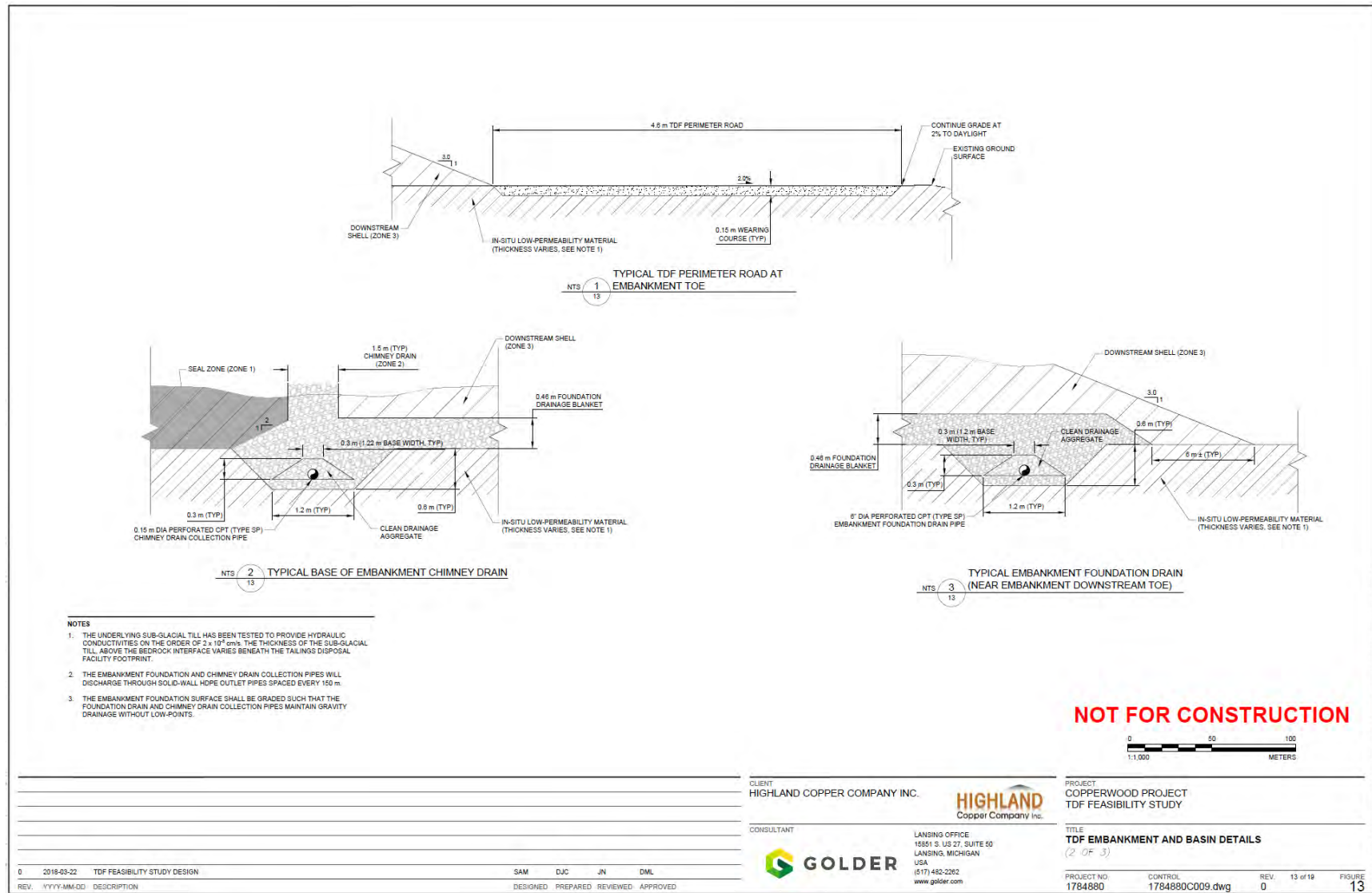


Figure 18.16: Embankment and Basin Details (2 of 2)



In order to protect the Seal Zone (Zone 1) from erosion, an HDPE geomembrane will be installed over the upstream face of the dam and shed any direct precipitation and wave action from the pond. In addition, the tailings deposition forming a beach against the geomembrane will protect the slopes. The embankment foundation drain (made of free-draining material at the base of Zone 3) will help dissipate excess pressure created in the embankment fill foundation during construction. The drains are presented in Figure 18.15 and Figure 18.16. The embankment will be constructed straight over the glacial till. The preparation of the foundation includes; topsoil stripping and stockpiling, removal of unsuitable material within the top layer and rough grading.

The Glacial till within the basin will be used to build the upstream Seal Zone (Zone 1) and the Embankment Fill (Zone 3). The glacial till is relatively fined grained and clayed, with a native moisture content greater than optimum. To avoid pore pressure buildup, these materials will need to be conditioned to reduce their moisture content. Within the Seal Zone, the moisture content will be allowed to remain slightly wet of optimum and near optimum or less for the balance of the embankment. The materials will be placed and compacted using lifts not exceeding 30 cm (1 ft.). Test pads should be carried out prior to construction in order to establish compaction specifications for material type.

The excavation within the basin for fill material will vary from 0 to 20 m. In general, it increases from South to North, following the depth of the bedrock. Once the basin excavation is completed, an average of 4 to 23 m of the till will remain above the bedrock surface. Figure 18.15 shows the completed excavation of basin. For subgrade preparation and stability reasons, the slopes of the excavation will be cut at 2.5H: 1V. The design was put together using information from boreholes, wells and piezometers. With field observations and monitoring wells logs, the glacial till will provide an impermeable unit over the bedrock.

18.15.3 Decant System and Tailings Management

The slurry from the mill will be carried to the TDF through pipes and discharged within the basin. A discharge model was developed to manage the supernatant pond and optimize the use of the basin capacity. First, the tails will be deposited from numerous points along the North, then along the East and West sides of the TDF.

The decant water will be returned to the mill and the treatment plant through a barge-mounted system. The barge will be relocated sporadically during deposition to create a tailings beach near the embankments. The initial barge location should be where the basin elevation is the lowest (North-West) and as the tailings are discharged move towards the South.

19. MARKET STUDIES AND CONTRACTS

19.1 Metal Prices

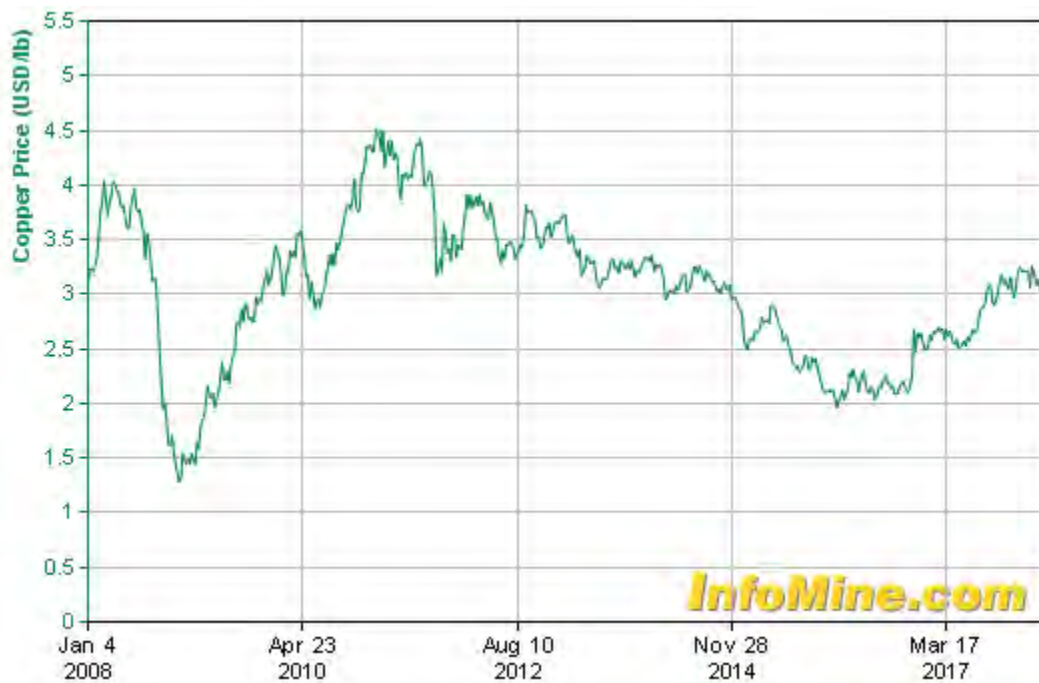
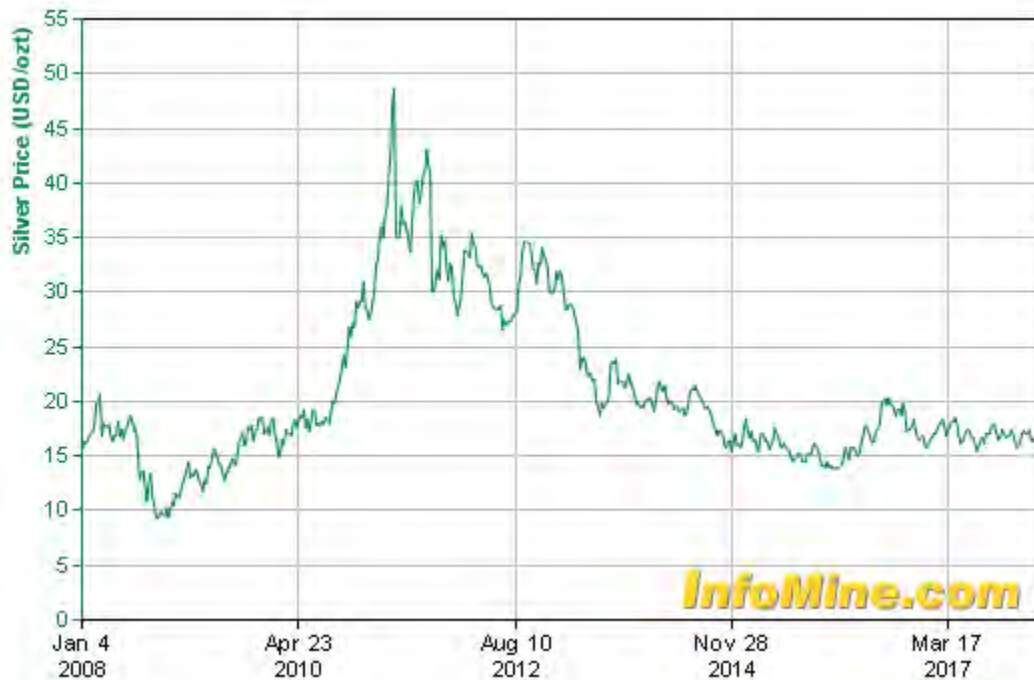
The metal prices selected for the economic evaluation in this Report are presented in Table 19.1. Higher near-term copper prices are assumed reflecting commodity price forecasts from analysts and reverting to a lower long-term price of US\$3.10/lb. The silver price has been assumed constant at US\$16.00/oz over the Project life.

Table 19.1: Metal Price Assumptions

Metal Price Scenario	Yr 1 (2021)	Yr 2 (2022)	Yr 3 (2023)	Yr 4+ (2024+)
Copper (US\$/lb)	3.40	3.25	3.15	3.10
Silver (US\$/oz)	16.00	16.00	16.00	16.00

There is no guarantee that copper and silver prices used in this Study will be realized at the time of production and will be subject to normal market price volatility and global market forces of supply and demand. Prices could vary significantly higher or lower with a corresponding impact on Project economics.

The 10-year historical price for copper as presented in Figure 19.1 highlights the variable nature of metal prices with a high of approximately US\$4.50/lb seen in 2011 and a low of US\$1.30/lb in mid-2008. The 10-year historical price for silver is similarly presented in Figure 19.2.

Figure 19.1: 10-year Historical Copper Prices**Figure 19.2: 10-year Historical Silver Prices**

19.2 Market Studies

19.2.1 Copper Concentrate

The copper concentrate produced from Copperwood will require downstream smelting and refining to produce marketable copper and silver metal. Several smelters could receive concentrate with the nearby candidates being the Horne smelter located in Noranda, Quebec or the copper smelter in Sudbury, Ontario. Other alternatives include seaborne export to Asia or Europe. Concentrate transportation charges will be a function of the final destination and will be a combination of trucking, rail and possibly shipping.

The concentrate treatment and refining charges (TC/RC) vary depending on the state of the economy and the supply and demand dynamics for copper concentrates available for smelting.

Copper payment is based on copper content of the concentrate. For a concentrate less than 32% but above 22% the payable rate is typically 96.5%, subject to a minimum deduction of 1%. Payment of precious metals in copper concentrates varies by region and customer but typically pays 90% if greater than 30 g/dmt with a 30 g minimum deduction. A summary of the copper concentrate marketing assumptions is summarized in Table 19.2.

Table 19.2: Concentrate Marketing Assumptions

Copper Concentrate Marketing Assumptions	
Copper Payable Rate	96.5% payment of Cu in concentrate >22%Cu and <32%Cu subject to a 1% minimum deduction
Silver Payable Rate	90% payment of Ag subject to 30g/dmt minimum deduction
Copper Treatment & Refining Charge (TC/RC)	TC = US\$70/dmt of concentrate, RC = \$0.070/lb of Cu
Silver Refining Charge	RC = US\$0.50/oz of Ag

Penalties may be applied to copper concentrates that have excessive amounts of deleterious elements such as lead, zinc, arsenic, antimony, bismuth, nickel, alumina, fluorine, chlorine, magnesium oxide, and mercury. The Copperwood concentrate can be classified as a clean concentrate and no penalties for deleterious elements are foreseen based on the analysis of concentrate produced from six locked cycle tests which cover all sections of the mine. The concentrate specifications with minimum and maximum values are presented in Table 19.3.

Table 19.3: Concentrate Specifications

Concentrate Analysis	Minimum	Maximum	Expected (Average)
Cu %	19.7	28.1	24.7
Fe %	7.87	10.2	9.5
As g/t	< 0.001	0.001	0.0
C(t) %	0.65	1.04	0.8
S %	5.45	9.99	7.3
S= %	5.22	7.32	6.4
Au g/t	0.11	0.35	0.2
Pt g/t	0.02	0.14	0.1
Pd g/t	0.02	0.24	0.1
Ag g/t	27.3	67.4	48.1
Hg g/t	< 0.3	0.8	0.5
Cl g/t	0	300	135.0
F %	0.038	0.046	0.042
SiO ₂ %	32.6	40.2	36.4
Al ₂ O ₃ %	7.93	9.34	8.7
Fe ₂ O ₃ %	11.3	14.4	13.5
MgO %	2.76	3.51	3.1
CaO %	0.59	0.85	0.7
K ₂ O %	1.75	2.16	1.9
TiO ₂ %	0.88	1.04	1.0
MnO %	0.11	0.15	0.12
Cr ₂ O ₃ %	0.043	0.180	0.102
V ₂ O ₅ %	0.021	0.025	0.023
As g/t	< 30	< 30	< 30
Ba g/t	172	211	192.5
Be g/t	1.38	1.73	1.5
Bi g/t	55	55	55.0
Cd g/t	< 2	< 2	< 2
Co g/t	25	33	28.3
Li g/t	21	43	29.8
Mo g/t	< 20	< 20	< 20
Na g/t	5770	7690	6825.0
Ni g/t	51	224	128.2
P g/t	558	728	644.5
Pb g/t	< 30	< 30	< 30
Sb g/t	< 10	< 10	< 10
Se g/t	< 30	< 30	< 30
Sn g/t	< 20	< 20	< 20
Sr g/t	41.9	53.1	48.3
Tl g/t	< 30	< 30	< 30
U g/t	< 20	< 20	< 20
Y g/t	23.3	24.9	24.2
Te g/t	< 4	< 4	< 4
Zn g/t	99	2940	954.2

19.3 Realization Costs

19.3.1 Concentrate Transportation

The transportation of concentrate was evaluated by Concept Consulting LLC with the current assumption that concentrate would be destined for the Horne smelter in Noranda. However, no contracts are in place at this time and other smelters can be considered.

The concentrate from Copperwood will be loaded into heavy-duty dump trailers with a cover and transported to a truck to rail transload facility located in Park Falls, Wisconsin. The truck configuration consists of 5 axles and will transport approximately 20 t per shipment. Park Falls is a preferred transload location as it is currently served by the CN railroad and is approximately 130.3 km (80 mi) from the mine site. The CN is a Class 1 rail road and its network spans three coasts with over 33,800 km (21,000 mi) of track and access to 75% of the North American continent and currently has operating lines in Michigan and Wisconsin. The pertinent operating line is the Ashland Sub Line in Wisconsin (Figure 19.3) which has a maximum rail line load rating of 121,560 t (268,000 lbs) and only operates north up to Park Falls. The CN line from Marengo Junction to White Pine is inactive.

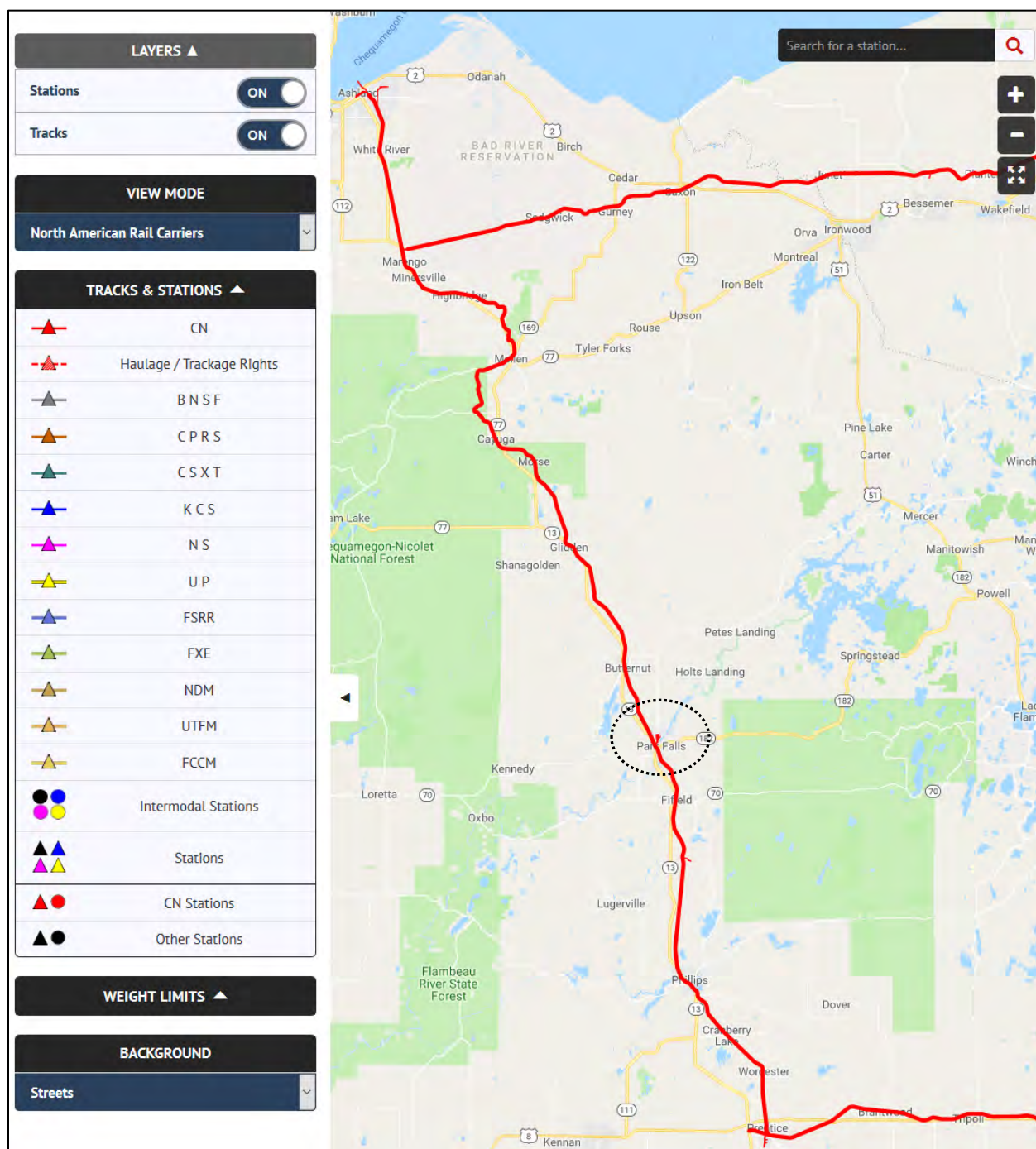
The transload facility is described in the Section 18.

The concentrate transportation costs are estimated at US\$67.16/t of concentrate which includes trucking, transload operations, CN rail transportation and gondola lease costs as summarized in Table 19.4.

Table 19.4: Concentrate Transportation Cost (Mine to Horne Smelter)

Concentrate Transportation	Cost (US\$/t)
Truck Transportation	18.00
Transload Operations	3.00
CN Rail Transportation	38.00
Gondola Lease Costs	8.16
Total Transport Cost	67.16

Figure 19.3: CN Rail – Ashland Sub Line



19.3.2 Insurance

An insurance rate of 0.10% was applied to the provisional value of the concentrate to cover transport from the mine site to the smelter.

19.3.3 Losses

Concentrate losses are estimated at 0.2% during shipment from the mine to the smelter.

19.4 Contracts

There are no mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts, or arrangements for the Project. This situation is typical for a development stage project still several years away from production.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides an overview of the environmental studies and consultation efforts that have been completed to support the state and federal permit approval requirements for the Copperwood Project. Information on the environmental studies and preliminary environmental effects can be found in the currently permitted Environmental Impact Assessment (“EIA”) found in the issued Part 632 Michigan Non-Ferrous Metallic Mining Permit. Any known environmental issues that could impact the Copperwood design through operations and closure are also discussed.

Environmental baseline studies were initiated for the Copperwood Project in late 2008 through the spring of 2011. These studies were used to identify potential siting of infrastructures based on an environmental management and permit approvals perspective. This approach is known as finding the most feasible and prudent alternative, this forms the basis for predicting environmental effects associated with the Project.

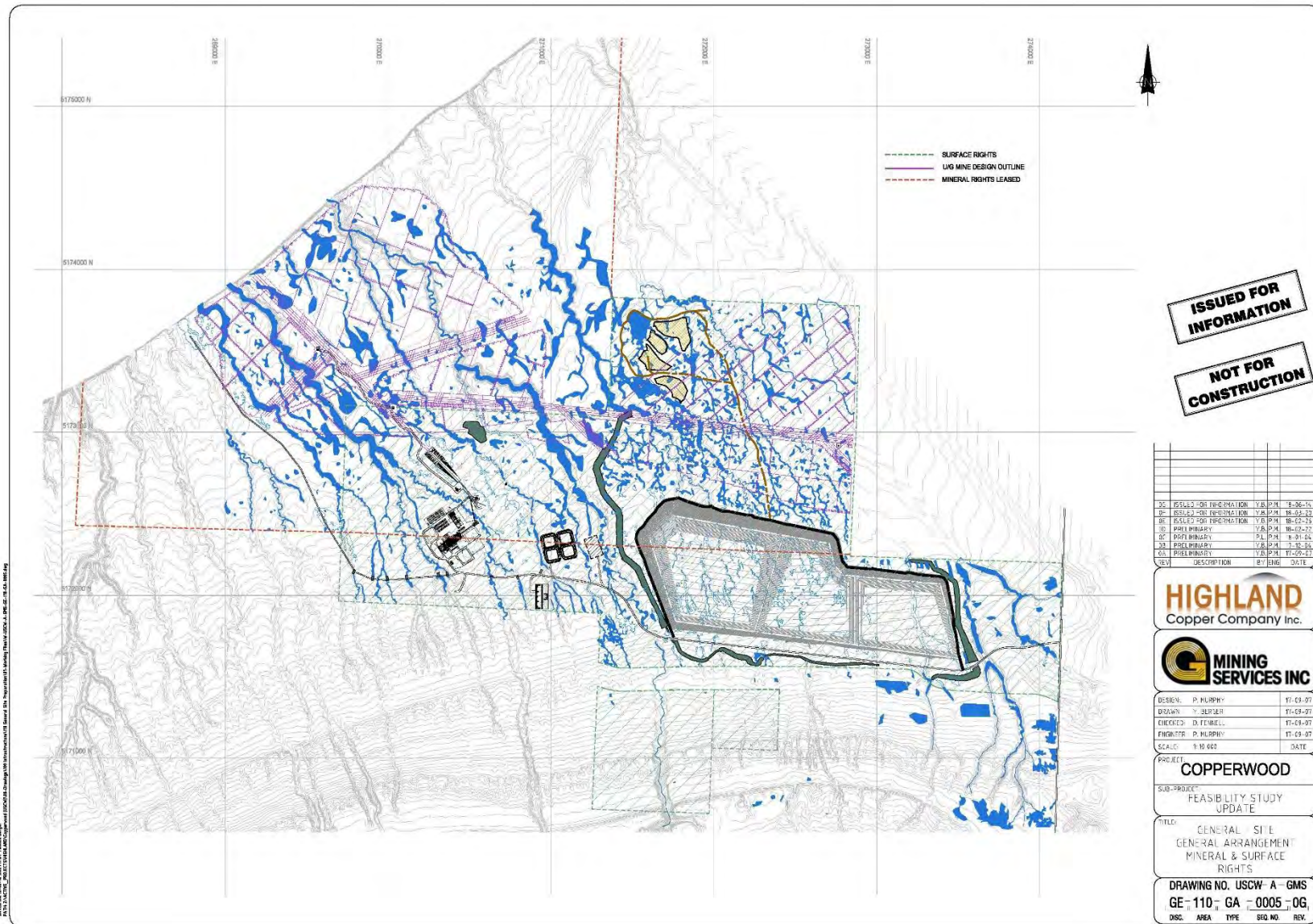
20.2 Environmental Studies

An EIA was prepared to comply with the State of Michigan requirements of Rule 425.202 of Part 632 of Act No. 451 of Public Acts of 1994 as amended. This document outlines the baseline monitoring and studies conducted for the Copperwood Project. This includes characterization of the natural, social, economic, cultural, and historical aspects of the environment that may be potentially impacted by the Copperwood Project design.

A proposed mining area chosen for the Study and shown below in Figure 20.1, encompasses the anticipated area of physical disturbance associated with the construction and operation of the Copperwood Project.

An Environmental Assessment (“EA”) was completed with the Part 301, Part 303, and Part 325 (Inland Lakes and Streams, Wetlands, and Great Lakes Bottomlands) permit applications to serve as a more comprehensive document outlining the initial site characteristics and impacts associated with the Copperwood Project construction and operation.

Figure 20.1: Proposed Mining Area Boundary



20.3 Description of Site Features

20.3.1 Topography and Drainage

The land surface at the Project site slopes toward the Lake Superior Shoreline. The ground surface elevation along the southern boundary of the Project area is approximately 365.8 mamsl (1,200 ft). The elevation of the top of the bluff at the Lake Superior shoreline is at an approximate elevation of 198.1 mamsl (650 ft). The topographic contours across the area are generally parallel to the Lake Superior shoreline. The ground surface slopes at a rate of approximately 100 ft per mile to the northwest. The elevation of the surface water level in Lake Superior is approximately 183.5 mamsl (602 ft). The current shoreline of Lake Superior is dominated by a bluff which rises as much as 15.2 m (50 ft) above the lake surface. This bluff is composed of silt/clay-rich till and is experiencing significant erosion at a range of approximately 0.2 to .49 m/y (0.6 to 1.6 ft/y). Figure 20.1 shows the contour lines of the Project site.

The surficial drainage system across the Project site is part of the Lake Drainage watershed. This watershed is located between the Black River watershed to the west and south and the Presque Isle watershed to the east and southeast. The section of the Lake Drainage watershed surrounding the project site encompasses 14 square miles ("mi²") and is composed entirely of small incised stream valleys flowing northwest directly into Lake Superior. These streams include (from west to east) Gijik Creek, Unnamed Creek, West Branch of Nambinag Creek, Namebinag Creek, Lehigh Creek, and Gypsy Creek. The flow within these streams could be described as ephemeral in nature, controlled by precipitation runoff as no groundwater contribution has been observed. This is apparent by the observations of no flow during the drier portions of the year.

The valleys are generally flat, vary in width between 50 and 200 ft (15.25 and 61 m), and can be as deep as 40 ft (12.2 m). The valleys widen as they near Lake Superior and are narrow upstream. Beaver dams are found throughout the site and create tiered meadows in the valleys.

20.3.2 Soil Conditions

There is a distinct transition of soil mapping units from higher land surface elevations of the southeast downslope to the northwest area of the Project site. The southern and southeastern portion of the Project area is predominantly mapped as Amasa cobbly fine sandy loam and Flintsteel silt loam, while the northwestern portion is principally the Big Iron-Flintsteel complex and Flintsteel silt loam mapping units. A narrow, northeast-southwest trending band of the Flintsteel loam and Flintsteel silt loam crosses through the central to northern area of the Big Iron-Flintsteel complex mapped soils in the northwestern portion of the Project site. Several isolated pockets of Big Iron – Belding complex soils located within an area of

predominantly Flintsteel silt loam occur in the central area of the Project site as well as in the upper elevations of a large hill occurring on the southern boundary of the Project site.

The surficial soils in the Copperwood Project area were formed primarily in loamy glacial till on ground moraines, colluviums from stream valley slope failures and alluvial deposits within stream floodplains. Drainage characteristics in the Project area range from well-drained to poorly-drained. Areas of poorly-drained soils may support wetland vegetation.

20.3.2.1 Soil and Sediment Chemistry

Samples of soil and sediment were collected to determine background concentrations of metals. This was done predominantly to differentiate between natural and any potential mining-related impacts.

20.3.3 Acid-Rock Drainage Potential of Site Geology

The geochemical characteristics of the Copperwood overburden and bedrock were characterized using industry accepted practices and carried out by several laboratories. The mineralogy of the Copperwood overburden, rocks and ore have limited variation within a particular lithologic unit. Overall, even including the Copper Bearing Sequence (“CBS”), the mineralogy of the different bedrock subunits and layers are similar to one another with a lack of significant mineralogic differences. Mineralogically, the overburden and bedrock generally lack acid-generating sulfide minerals while containing acid-neutralizing calcite.

The overburden at Copperwood is similar to overburden found over large areas of Gogebic and Ontonagon Counties and other areas in the Lake Superior region. The geochemical testing program for overburden consists of determination of the bulk chemical composition to ensure that it lacks anomalous concentrations of potentially environmentally sensitive components.

20.3.4 Hydrogeology

The unconsolidated overburden sequence consists primarily of fine-grained deposits of a massive, matrix-supported diamicton. The soil sequence overlies bedrock that includes the Nonesuch Shale and Copper Harbor Conglomerate. The Nonesuch Shale is a sequence of siltstones, shales, and sandstones. The Copper Harbor Conglomerate is a sequence of well-cemented sandstones and conglomerates. Properties of these soil and bedrock layers influence the hydrogeology. As such, the characterization of the hydrogeologic conditions depend on these layers and builds upon the geologic characterization.

A field investigation was conducted that included borings to establish the nature and extent of the stratigraphic units found on-site. Groundwater monitoring wells were installed in these boreholes to obtain hydrogeologic information necessary to evaluate the direction and flow of groundwater beneath the Project site. Groundwater level measurements were collected manually using an electronic water level meter and automatically using programmable pressure transducers to define the water table. Hydraulic conductivities were measured using several methods, they included packer tests, slug tests, and single-well pump tests. The vertical and horizontal hydraulic conductivities were determined from these.

Upon these tests, it was concluded that the hydrostratigraphic units underlying the Project site do not meet the definition of an aquifer. Recharge rates are also very slow due to the fine-grained nature of the Upper Till layer. Therefore, most precipitation ends up as surface runoff or is stored in the surficial soils and lost mostly to evapotranspiration compared to evaporation.

The complex nature of the groundwater flow system through the overburden and bedrock lithology in the vicinity of the proposed underground mining operation has necessitated the use of computerized groundwater flow models to simulate the baseline and possible future conditions of the project area. The finite difference modeling program MODFLOW was selected to simulate groundwater flow assuming that the fractured bedrock behaves as an equivalent porous media ("EPM"). The model was set up to simulate existing groundwater flow conditions based on the information collected during the EIA data collection period and to predict future groundwater patterns based on information on the build-out of the mining operation.

20.3.5 Streams, Wetlands, Ponds, and other Surface Water Bodies

Surface water at the site consists of a variety of features, from Lake Superior to intermittent streams. Data was collected to characterize each feature during this Study. The site is dominated by the Lake Superior shoreline on the northern edge and contains a variety of ephemeral streams flowing from southeast to northwest into Lake Superior. Small ponds are present along the streams, the result of impoundment by beaver dams. No other lakes are present within the Project area. No springs, seeps, or other sources of discharge of groundwater to the surface water system have been identified in the Project area. The Project area contains abundant wetlands as a result of the silt/clay-rich glacial till soil that hinders the infiltration of precipitation.

20.3.5.1 Predicted Seasonal and Long-Term Variations

It is expected that seasonal fluctuations of surface water levels decrease into summer, and then increase into the fall, will continue into the future. The major influences affecting these trends are the spring

melt/runoff period, followed by increased temperatures and evaporation rates in summer, and then increased precipitation in the fall combined with decreased temperatures and evaporation rates. The silty and clayey surface soils of the site are relatively resistant to infiltration, making surface water loss to the subsurface essentially negligible. Precipitation rates and temperatures are the primary influences relative to seasonal surface water levels.

Precipitation and temperature will continue to be the most relevant in the future with regard to long-term variation. In this case however, macro-scale changes such as extended periods of drought, intense or continuous rains, or consistently high or low winter snowfall amounts will influence the overall amount of surface water present at the site throughout the year. These types of changes would likely result in consistently higher or lower water levels overall, with evidence of typical seasonal fluctuations remaining.

20.3.6 Surface Water (Stream) Flow

Manual monitoring of stream flow at twenty-one monitoring locations across the Project site were conducted. Similar to the lake, pond, and wetland monitoring, these measurements were completed on a quarterly basis from January 2009 to April 2011 and are planned to continue during site development and use. The purpose of this monitoring is to determine typical flow rates of various streams on site and to characterize their long-term and storm-event flow patterns. The chosen frequencies and locations of monitoring were designed to capture the largest variety of geomorphic and temporal settings possible, while at the same time measuring flow upgradient of, downgradient of, and within the proposed mining area.

As described in Section 20.3.1, the on-site streams are ephemeral in nature and are at their highest during the spring snowmelt and following rain events. The overall period of storm response is relatively short, and streams on-site have been determined to be “flashy”.

20.3.7 Water Balance

An existing water budget was evaluated for the site of the Copperwood Project. A water budget is an accounting of all the water that flows into and out of the Project area, including precipitation (“P”), groundwater flow (“GF”), evaporation (“E”), evapotranspiration (“ET”), and surface water runoff (“SRO”). The water budget is expressed as an equation relating these components:

$$\Delta S = P - E - ET \pm SRO \pm GF \text{ (LWMD, 2010)}$$

where ΔS is the change in storage. This baseline budget may be used to evaluate impacts and possible mitigation actions associated with the mine. Under natural conditions, the water budget should be expected to be zero over the long term, i.e., water into the site equals water out of the site.

The water-budget analysis encompassed approximately 8,960 acres of wooded land in the west portion of Gogebic County between the Presque Isle and Black Rivers, and about one mile southeast of Lake Superior. A weather station was installed and monitored by AECOM from December 2008 through late 2014. The weather station collected daily climatic data and stream-flow data was collected from twenty-one points within the proposed Project limits.

20.3.8 Floodplains

Federal Emergency Management Agency ("FEMA") floodplain mapping carried out in 1977 for the Project area and surrounding vicinity indicated that a Zone A floodplain occurs along the immediate shoreline of Lake Superior and extending south along the Black River in Ironwood Township, Gogebic County, Michigan. Wakefield Township, the adjacent township to the east, has no recorded FEMA mapping. The streams that occur on the Project site are not shown to have mapped floodplains.

20.3.9 Great Lakes Shoreline

The northern boundary of the Project area borders approximately 609.6 m (2,000 ft) of undeveloped Lake Superior shoreline. The shoreline at this location consists of level and gently sloping rock/cobble and sand beach fronting along steep, eroded bluffs. The development of the mining surface facility will be set back from the Lake Superior shoreline by a distance of at least 1,524 m (5,000 ft).

20.3.10 Wetlands Near the Mine Site

Wetlands on-site were delineated by several consultants and also evaluated using Michigan Rapid Assessment Method ("MiRAM") to determine the quantity and quality of on-site wetlands.

20.3.11 Natural Rivers and Wild & Scenic Rivers

There are no rivers listed as State Natural Rivers within Gogebic County or the western Upper Peninsula.

The Federal Wild and Scenic Rivers Act of 1968, Public Law 90-542, declared a national policy that selected rivers, or sections of rivers, that possess outstanding scenic, recreational, geologic, fish and wildlife, historic, cultural or other similar values be preserved in a free-flowing condition.

The Presque Isle River, located approximately two miles east of the Project area, became a designated Wild and Scenic River area in March 1992. The designated reach of the Presque Isle includes the main stem from the confluence of the East and West Branches to Minnewawa Falls, and the East, South and West Branches of the Presque Isle within the Ottawa National Forest. There are 19 mi of this River that were included because of scenic value and 38 mi included for recreational value, for a total of 57 mi of Federal designation.

The Black River is located approximately two miles west of the Project area. The designated reach of this River begins at the Ottawa Forest Boundary and extends to Lake Superior, for a total length of approximately 14 mi, being designated for its scenic values.

20.3.12 Air Quality

Monitoring for ambient pre-mining air quality was conducted. The parameters were recorded using a weather station testing for particulate matter, NO_x, SO₂, Ozone, NH₃, and environmental conditions (temperature, precipitation, etc.). There are no major sources of air pollution near the Project site, the closest air pollution emitter is Great Lakes Transmission Station #7 located 24.14 km (15 mi) south-southeast of the Project.

20.3.13 Existing Infrastructures and Utilities

No dwellings, buildings, or other structures currently exist on the project site. Several seasonal camps are located throughout the area. Two of these seasonal camps are located approximately 3.2 km (2 mi) to the southeast and 3.2 km (2 mi) northeast of the center of the site. No public buildings are in the project area. The nearest public building would be in the Porcupine Mountains Wilderness State Park ("PMWSP") campground, located at the north end of CR 519, more than 4.0 km (2.5 mi) from the Project site.

Several concrete foundations are present near the site of the historic test shaft and rock pile.

The nearest public road is CR 519 located approximately 4.3 km (2.7 mi) to the east of the center of the Project area. No other public roads are present within the Project area.

No utilities are currently present at the site. The nearest electrical service terminates along CR 519 several miles to the south of the Project site. No pipelines or other utilities are present along CR 519.

20.3.14 Historical and Current Land Use

During the past 100 years, the use of the Project site was mostly for commercial forest industries, this included a rail service in the late 1800's as evidenced by the small gauge rails and grades found in recent surveys. The current access road is the Camp 7 Grade from these times. In 1949, a forest fire dubbed "the big burn" heavily burnt some of the site and surrounding forests and the existing red pines show the reforestation effort from this event.

In 1956, the United States Metal Refining Company ("USMR") began mineral exploration of the area, and in 1958, USMR developed a test mine, sinking a shaft and creating a drift. The remnants of the fenced in shaft, waste piles and foundations remain today.

In the late 1980's the site was clear cut and an extensive skid and road system was developed, the site clearly shows the signs of the extensive logging taken place over the last 100 years. Currently, CRI owns the surface properties and have been engaging in silvicultural management and core drilling on site, in addition to the environmental baseline studies.

20.3.15 Flora and Fauna**20.3.15.1 Aquatic Flora and Fauna**

Aquatic resources within the Project site were studied and documented in 2009 and 2010. The surface waters on site were found to be good to excellent habitats, but due to the ephemeral nature of the on-site streams, the potential for fish and macroinvertebrates to inhabit the site are severely limited.

The aquatic fauna found on site included the Creek Chub, Mottled Sculpin, Rainbow Trout, and Redside Dace. Aquatic flora found on site included the water-plantain, water shield, water-starwort, spike-rush, water horsetail, wild blue flag, common bur-reed, broad-leaved cat-tail, and bulrush.

20.3.15.2 Terrestrial Flora Fauna

Terrestrial flora and fauna resources within the Project site were studied and documented in 2009 and 2011. The flora on site consists of a total of 15 communities/timber stand types and three additional natural communities. Forest species consist of red maple, sugar maple, black ash, green ash, white ash, large-tooth aspen, trembling aspen, eastern hemlock, ironwood, yellow birch, basswood, red oak, and white birch. Understory species consist of service berry, hazelnuts, and honeysuckle among assorted fern and forb

species. Non-native species represent a minor component of the floral diversity and are small, scattered populations in recently disturbed areas.

The fauna on site consists of mammals, amphibians and reptiles, and birds. Medium and large mammal species encountered were as expected with a total of 16 species detected. White-tailed deer and black bear were the most abundant large mammals observed. Other species included coyote, muskrat, beaver, bobcat, pine marten, fisher, porcupine, raccoon, gray (timber) wolf, gray fox, badger, river otter, striped skunk, and snowshoe hare. Small mammals included a total of 12 species of shrews, mice, voles, moles, and squirrels. A total of five species of bat were positively identified feeding above the site. Several amphibious species and five reptile species were found within the Project site. The bird survey included owls, red-shouldered hawks, spring migratory birds, breeding birds, northern goshawks, nightjars, fall migratory birds, and incidental observations. No federally-listed bird species were detected during any of the surveys, but two state-listed species were documented. As many as 82 species were detected during the 2009 surveys and 53 species during 2011 surveys.

20.3.16 Plant and Animal Species of Special Concern

A query of the Michigan Natural Features Inventory (“MNFI”) database indicated the occurrences of five different plant and animal species of significant concern within the investigation area:

- Bald Eagle (*Haliaeetus leucocephalus*) - Listed “Special Concern” in State of Michigan, no federal designation. Identified in 2002 in Section 5, T49N, R45W, and in Sections 31 and 32, T50N, R45W.
- Extra-Striped Snaketail (*Ophiogomphus anomalus*) – Listed “Special Concern” in State of Michigan, no federal designation. Identified on June 16, 1997, in Section 31, T50N, R45W.
- Spoonhead Sculpin (*Cottus ricei*) – Listed “Special Concern” in State of Michigan, no federal designation. Identified on June 28, 1998, in Lake Superior within the area covered by the Black River Harbor USGS Quadrangle. This area includes the majority of the lakeshore on the north side of the Project site.
- Cisco or Lake Herring (*Coregonus artedii*) – Listed as “Threatened” in State of Michigan, no federal designation. Identified in 2001 in Lake Superior within the area covered by the Black River Harbor USGS Quadrangle. This area includes the majority of the lakeshore on the north side of the Project site.
- Large Toothwart (*Dentaria maxima*) – Listed as “Threatened” in State of Michigan, no federal designation. Identified on May 24, 1995, in Section 26, T49N, R46W (Michigan Natural Features Inventory, 2008).

- A review of database information completed by the Ottawa National Forest yielded the following occurrence data regarding plant species of concern:
- Large Toothwort (*Dentaria maxima*) was identified between 1994 and 2000 in the SW ¼ of the NW ¼ of Section 26, T49N, R46W. This species is currently listed as “Threatened” in the State of Michigan.
- Other non-designated rare/watch species of interest that were identified during this survey included: Northern wild comfrey (*Cynoglossum boreale*) found in Section 23, T49N, R46W, trail plant (*Adenocaulon bicolor*), Braun’s holly fern (*Polystichum braunii*), Canada yew (*Taxus canadensis*), and squawroot (*Conopholis americana*) all found in the SW ¼ of the NW ¼ of Section 26, T49N, R46W (S. Trull, Personal Communication, December 12, 2008).
- The proposed Project area is located within the home territory of a gray wolf pack (*Canis lupus*). The pack on or near the Study site at the time of initial information inquiries of Michigan Department of Natural Resources (“MDNR”) wolf biologists in 2009 may have been associated with the Foley Creek Wolf Pack. This pack’s activity center is located about 16.1 km (10 mi) to the west, across the Black River, but still within Ironwood Township.

20.3.17 Threatened or Endangered Species

Gray wolf (*Canis lupus*) is currently listed as endangered by the U.S. Fish and Wildlife Service (“USFWS”) and as a species of special concern by the MDNR. The USFWS has attempted to delist wolves in the recent past but have been successfully challenged in the Federal court system by animal rights activists. Camera traps, sign, and aural evidence confirmed their presence throughout the Project area. MDNR wolf biologists indicate there are two separate packs which inhabit the Project area.

Canada lynx (*Lynx canadensis*) is listed as a threatened species in the contiguous United States by the USFWS and Endangered by the MDNR. Although no recent occurrences of Canada lynx have been documented in the Western Upper Peninsula, it is understood that Canada lynx could potentially disperse to this area if suitable habitat and food source is present.

No reptile and amphibian species identified during surveys are included on federal or state threatened or endangered species listings.

During the surveys, no federally-listed endangered, threatened or candidate bird species were detected on any of the bird surveys conducted at the Study area. The common loon, a State-threatened bird species, was identified flying over the Project site during bird surveys occurring in the breeding season and fall migration of 2009. The preferred habitat of the common loon is moderate to large freshwater lakes with fish

populations. Although beaver ponds of various sizes occur on the Project site, the ponds are not large enough to allow access and departure flights and would also provide limited sources of food for the bird. Therefore, loons may utilize the area off-shore of the Project site for breeding and migration, as well as inland lakes beyond the Project area, but are unlikely to frequent the Study area itself due to lack of suitable habitat.

The second state-listed bird was a Peregrine Falcon, which was observed along the Lake Superior shoreline in May 2009. This bird was likely a late spring migrant or a foraging breeder with a nest site north of the Study area. Peregrine Falcons usually nest on cliffs or rock outcrops, neither of which are present within the Study area.

A Michigan endangered fish species, the Redside Dace (*Clinostomus elongates*), was identified within the boundaries of the Study area. Several of these fish were identified in the lower sections of Namebinag Creek and the Unnamed Creek.

One plant, the showy orchid (*Galearis spectabilis*), was identified in the expanded Project area in the spring of 2011. This plant is a Michigan State Threatened species. No other plants listed as Endangered, Threatened or Special Concern in Michigan were observed by Stantec during the survey of the site. Suitable habitat was observed for other threatened or endangered species, but no protected plants were observed.

20.3.18 Species of Special Concern

No mammal species having a status of Special Concern were identified during Project surveys.

The wood turtle (*Glyptemys insculpta*) is listed as a species of Special Concern by the MDNR. Two wood turtles were observed within the Study focus area during seasonal field work. However, no suitable nesting habitat was found during these surveys. Suitable non-nesting habitat areas can be found in stream corridors, but it is only used seasonally by this highly mobile turtle species.

State special concern species noted during bird surveys included the Northern Harrier (*Circus cyaneus*). A single Northern Harrier was inventoried during spring mitigation. This bird was not observed during breeding season or during fall migration. Special concern species are not afforded legal protection under the Endangered Species Act of the State of Michigan, but because they have declining or relict populations in the state, they are being monitored to determine whether they should be recommended for Threatened or Endangered status.

Several species inventoried within the study focus area appear on the Michigan “Species of Greatest Conservation Need” (“SGCN”) list. The SGCN list is maintained by Michigan as a species watch list and used primarily for management purposes. Bird species identified in the field surveys that appear on the SGCN list included:

- Coopers hawk (*Accipiter cooperii*);
- American woodcock (*Scolopax minor*);
- Black-billed Cuckoo (*Coccyzus erythrophthalmus*);
- Northern flicker (*Colaptes auratus*);
- Least flycatcher (*Empidonax minimus*);
- Ruby crowned kinglet (*Regulus calendula*);
- Brown thrasher (*Toxostoma rufum*);
- Golden-winged warbler (*Vermivora chrysoptera*);
- Northern parula (*Parula americana*);
- Black-throated blue warbler (*Dendroica caerulescens*);
- Blackburnian Warbler (*Dendroica fusca*);
- Palm warbler (*Dendroica palmarum*);
- Evening grosbeak (*Coccothraustes vespertinus*).

No plant species listed as Special Concern in Michigan were observed during the survey of the site. Suitable habitat was observed, but no protected plants were observed.

20.3.19 Non-Native or Invasive Species

Reviews of biological inventories for the site have revealed the presence of Michigan-listed invasive species. The species identified on the site are plants that are commonly found in disturbed areas, especially those where the canopy has been removed. Should the canopy return, these species would likely be phased out through community succession. The numbers of species and infestation levels are very low within the focus study area and surrounding vicinity. Invasive plant species identified within the study focus area included:

- Quack grass (*Agropyron repens*);

- Redtop (*Agrostis gigantea*);
- Common burdock (*Arctium minus*);
- Yellow rocket (*Barbarea vulgaris*);
- Mouse-ear chickweed (*Cerastium fontanum*);
- Ox-eye daisy (*Chrysanthemum leucanthemum*);
- Canada thistle (*Cirsium arvense*);
- Bull-thistle (*Cirsium vulgare*);
- Field bindweed (*Convolvulus arvensis*);
- Orchard grass (*Dactylis glomerata*);
- Queen Anne's lace (*Daucus carota*);
- Helleborine (*Epipactis helleborine*);
- Common hemp nettle (*Galeopsis tetrahit*);
- Orange hawkweed (*Heiracium aurantiacum*);
- Common St. John's Wort (*Hypericum perforatum*);
- Nipplewort (*Lapsana communis*);
- Birdsfoot trefoil (*Lotus conculata*);
- Field scorpion-grass (*Myosotis arvensis*);
- Woodland forget-me-not (*Myosotis sylvatica*);
- Timothy (*Phluem pratense*);
- English Plantain (*Plantago lanceolata*);
- Common plantain (*Plantago major*);
- Kentucky bluegrass (*Poa pratensis*);
- Tall or Common buttercup (*Ranunculus acris*);
- Glossy buckthorn (*Rhamnus frangula*);
- Sheep sorrel (*Rumex acetosella*);
- Curly dock (*Rumex crispus*);
- Bittersweet nightshade (*Solanum dulcamara*);

- Garden tansy (*Tanacetum vulgare*);
- Common dandelion (*Taraxacum officinale*);
- Clover species (*Trifolium pretense*, *Trifolium repens*);
- Narrow-leaved cat-tail (*Typha angustifolia*);
- Common mullein (*Verbascum thapsus*);
- Common speedwell (*Veronica officinalis*).

Non-native species represent only a minor component (15%) of the floral diversity at the Project site and were often noted as small, scattered populations in recently disturbed areas. Large populations of non-native species were not observed to be present.

No non-native or invasive species of mammals, birds, amphibians, reptiles, or fish were observed.

20.3.20 Identified Ecological Systems

Eco-regions are ecosystems of regional extent. Bailey's eco-regions (McNab, et al., 2005) distinguish areas that share common climatic and vegetation characteristics. A four-level hierarchy is used to differentiate the eco-regions with the broadest classification being the domain. Domains are groups of related climates and are differentiated based on precipitation and temperature. There are four domains used for worldwide eco-region classification and all four appear in the United States: 1) the polar domain, 2) the humid temperate domain, 3) the dry domain, and 4) the humid tropical domain. Divisions represent the climates within domains and are differentiated based on precipitation levels and patterns as well as temperature. Divisions are subdivided into provinces, which are differentiated based on vegetation or other natural land covers. Mountainous areas that exhibit different ecological zones based on elevation are identified at the province level. The finest level of detail is described by sub-regions, called sections, which are subdivisions of provinces based on terrain features.

The purpose of ecological land classification is to provide information for both the development of resources and the conservation of the environment. Government and private land managers use this information to estimate ecosystem productivity, to determine probable responses to land management practices, and to address environmental issues over large areas, such as air pollution, forest disease, or threats to biodiversity. This map layer was compiled by the USDA Forest Service.

Based on this classification, the Copperwood Project site is located within the Warm Continental division, Laurentian Mixed Forest province, Southwest Lake Superior Clay Plain section, and the Superior-Ashland Clay Plain subsection.

Omernik's Level III Eco-regions, (USEPA, Western Ecology Region, website) are defined by a wide variety of characteristics, including vegetation, animal life, geology, soils, water, climate, and human land use, as well as other living and non-living ecosystem components. Under the Omernik system the Copperwood Project falls within the "Northern Lakes and Forests" of Level III mapping.

20.3.21 Archaeological and Historical Resources

Historical and archeological studies were conducted on a majority of the field site, still pending a spring 2018 field delineation. According to records at the Office of the State Archaeologist, there are no known archaeological sites from any cultural period within miles of the Project area. AVD Archeological Services Inc. was contracted for a Phase 1 Archaeological Survey and after extensive analysis of the anticipated area for mine development, no archaeological sites or artifacts were found in these studies. Only modern-day items, such as old railroad grades from logging and an accumulation of rusted tin cans were identified from past use. Therefore, it appears unlikely that archaeological sites will be disturbed by the development of the Project.

Historically, the area of the Project site was used for commercial timber production, as evidenced by the old railroads grades and forest communities. A test mine from the 1950's is present on site and has no historical significance. Therefore, the development of the Project will not impact any historical sites.

20.4 Potential Impacts to Site Features and Mitigations

20.4.1 Topography and Drainage

During operation the topography will change relative to locations of facility development, but post-closure the topography and drainages will be brought back to near pre-mining conditions. Albeit, the Tailings Disposal Facility ("TDF") will have permanent impacts to the topography and drainages post-closure.

20.4.2 Surface Water Flow

During development, operation, and closure of the Copperwood Project, several changes to surface water (stream) flow will occur. These include:

- Storm water diversions and storm water runoff management;
- Diversion of water around TDF;
- Discharge of treated water to surface water.

Storm water will be classified as both contact and non-contact water. Contact water is storm water that encounters mining operations, dust, ore or concentrate. This water will be directed to the TDF for treatment prior to discharge and will not have a direct impact on stream flow. Non-contact storm water will be from roofs, paved parking, roadways and other sources where water does not encounter potential contaminants. This water will be diverted as necessary to detention ponds, allowed to settle to aid in removal of suspended material, and then allowed to enter the streams. This form of storm water discharge will increase the flow of water in streams slightly, but the site streams are currently flashy and dominated by surface runoff. The on-site storm water management plans provide that storm water will generally be directed to the nearest stream, and therefore stay within the original, existing watershed.

To construct the TDF, dikes will be built and streams diverted to prevent water from entering the TDF. The TDF is in the upper headwaters of Lehigh Creek and Gipsy Creek. Some of the upstream portions of Lehigh Creek and several branches of Gipsy Creek are proposed to be filled and their flows will be diverted around the TDF. Surface water to the south of the TDF will also be diverted.

Treated water from the proposed wastewater treatment plant ("WWTP") and the sewage lagoons will be discharged according to a NPDES discharge permit. Three alternatives were considered for this discharge, drain field infiltration, Lake Superior discharge, and a discharge to a stream on the site. A stream discharge was determined to be the most feasible option.

To the extent possible, the mining operation has been designed to minimize impacts to streams on the site. Relocation of the headwater areas of streams around the TDF will occur, into approximately 13,700 linear feet of new natural stream design diversion channels.

During its active life, the exterior embankments of the TDF will be graded to capture discharge into on-site streams. Post-Closure, the TDF will be graded to discharge surface runoff into the stream diversions.

20.4.3 Water Balance

The proposed mining operation is anticipated to have minimal impacts on the existing site water balance. Precipitation will remain unchanged. Minimal impacts to the water balance will be created by the mine dewatering, development of the surface facilities and consequent capture of contact are runoff to the TDF

during operation. The primary factors in the water balance equation that are anticipated to be affected by the proposed mining operation are groundwater extraction, surface water flow, evaporation and evapotranspiration.

20.4.4 Surface Water Quality

Mining operations will have several potential impacts to surface water quality. These include potential impacts due to:

- Sediment input;
- Storm water runoff;
- Contact water;
- Discharge of treated water;
- Septic systems;
- Storage of chemicals and fuel;
- Milling and TDF operations.

These impacts will be reduced by the implementation of various permits and regulated plans corresponding to these topics and discussed later in this section of the Feasibility Study.

20.4.5 Wetlands

Development of a mining operation on the Project site will directly impact wetlands within the footprint of facility components by grading and filling. The wetlands impacted on the development site consist primarily of those classified as saturated palustrine deciduous forested (PFO1B). The wetlands can be further described as occurring in isolated depressions or shallow drainage corridors that meander across level to gently sloping plateaus between eroded stream valleys. Groundwater and surface water studies conducted on the Project site have identified only very limited direct hydrologic inputs to wetlands or streams on the site from groundwater. Therefore, the wetlands occurring on the site are supported predominantly by direct precipitation and surface water runoff. Being supported by surface water, the wetlands have standing water or saturated soils for an extended time after spring runoff. Storm events contribute water to the wetlands throughout the growing season, although during most years the runoff volume is not sufficient to maintain either inundated or saturated soil conditions within the wetlands. This is evidenced by a lack of sustained aquatic communities without the presence of beaver activities. Operation of a subsurface development that

may influence groundwater levels would likely have little direct effect on the surface water supported wetlands.

Development activities will likely result in changes to some of the current surface water drainage patterns that support wetlands occurring in depressions on low permeability clay soils. Wetlands located within the footprint of the mine facility occur in shallow depressions that are isolated or meandering drainage corridors that lead to the larger drainage valleys of the various on-site streams. Isolated wetlands will be filled and graded with little to no resulting impacts to the water supply of other down gradient wetlands. The contiguous wetlands within the facility footprint occurring in shallow drainage corridors are situated in the upper reaches of the drainage system. Although portions of the contiguous wetlands will be directly impacted by fill and grading, the remaining watershed on the low permeability clay soils should adequately maintain the remaining down-gradient portions of those wetlands. Wetlands located within the footprint of the TDF occur within isolated depressions on low permeability clay and silt soils, in shallow, meandering drainage corridors leading to stream valleys, and in large depressions on an old lake beach front at the toe of a large hill that is located on the south boundary of the Project site. Development of the TDF will directly impact isolated wetlands within the footprint by filling and grading. The wetlands within the meandering, shallow drainage corridors will also be impacted by filling and grading. However, like wetlands in the mine facility area, the contiguous wetlands occur in the upper reaches of the drainage system between stream corridors. Stream channels currently located within the TDF footprint will be relocated to maintain water supply to the down gradient surface water features. Ventilation shafts and the associated road were planned to avoid wetlands as feasibly possibly but will affect a small area of isolated wetlands unavoidably as the underground mine configuration dictates the locations of the shafts.

Direct impacts to wetlands by grading or filling have been minimized to the extent that is feasible and prudent by situating facility components on the Project site to avoid these resources, as documented in the Part 301/Part 303 (Inland lakes and streams/Wetland) permit application. Unavoidable wetland impacts are proposed to be mitigated through the preservation of high-quality off-site wetlands through the establishment of conservation easements as well as the on-site creation of 18.3 acres of forested and emergent wetlands, as also documented in the above-referenced permit application documents.

20.4.6 Great Lakes Shorelines

A water intake structure will be constructed below the bottom of Lake Superior to deliver water to the mining operation. This structure will also include a submerged pipeline below the lake bottom, that will also be buried beneath the ground in the upland areas all the way to the water-delivery facility. While some impacts may be observed during construction activities they will only be a temporary disruption to the aquatic resources.

Surface mining and other affiliated operations will be situated more than a mile from the Lake Superior shoreline and are not expected to have any direct impact to Lake Superior or its shoreline. The view shed of the shoreline from offshore boats may potentially be affected later in the mining operation as the tailings disposal area grows and eventually exceeds the height of vegetation cover.

20.4.7 Flora and Fauna

20.4.7.1 Aquatic Flora and Fauna

The presence of aquatic communities within the Project area is limited to relatively seasonal intermittent water accumulations or flows and beaver impoundments. The beaver impoundments, and small quantities of pooled water retained in pockets within depressions in steep gradient stream corridors in the lower reaches near Lake Superior, are the only areas on the Project site that have currently have some sustained water capable of supporting aquatic communities. Wetlands on the Project site are primarily supported by surface water runoff and most have no standing water during summer months.

Aquatic communities with sustained water levels currently do not occur within the TDF footprint due to steep gradient streambeds and the lack of beaver activity/habitat in that area. Wetlands within the TDF footprint are supported by runoff, and therefore are dry during the summer months limiting the potential for development of aquatic communities needing inundated or saturated conditions year-round. Beaver activity in streams near the mine facility area creates potential for the development of aquatic communities within impoundments, however, the impoundments are often temporary in nature with poor water quality. Given the poor quality of the aquatic macroinvertebrate community, very limited fish habitat and seasonal nature of stream flow, the Project is anticipated to have limited impact on aquatic plants and animals. Currently there are no beaver impoundments within the facility development footprint.

Development of the mine entrance, ore stockpile, and processing facility is anticipated to impact wetlands near the west branch of the Namebinag Creek and the Unnamed Creek, which will alter runoff patterns to stream corridors. Development of the TDF will also eliminate wetlands and stream corridors which will alter runoff patterns to the upper reaches of Namebinag, Lehigh and Gipsy Creeks. During construction of the Lake Superior water intake, higher than baseline turbidities will occur in close vicinity to the pipeline and mitigations will be in place to minimize that impact.

20.4.7.2 Terrestrial Flora and Fauna

The terrestrial mixed forest communities, including deciduous hardwoods and wetlands occurring within the project site and in the surrounding vicinity, are typical of the mixed forest landscapes in the western Upper

Peninsula. Except for the forests in the adjacent PMWSP, the majority of the woodlots in the area are routinely harvested for various commercial uses.

The development of the mine facilities will permanently remove approximately 410 acres of historically well-harvested mixed forest community from the landscape, with approximately 50 to 60 acres in the area being for the process plant, ore stockpile, box cut and the remainder of approximately 350 acres in the footprint of the TDF. After closure, the mine operating areas of the Project site will likely reforest unless a second beneficial re-use of the facility is pursued. The TDF will be closed with a grass cover, which will result in a change of community type from forest to grassland. Although the grassed portion of the site remaining after mine closure will be approximately 350 acres, on a relative scale it only represents a very small change in community type from the current large and contiguous thousands of acres of wooded tracts common to the region.

The existing mixed forest vegetation communities within the footprint of the mine facilities will be removed and terrestrial flora and fauna habitats altered during mining facility development and operation. In the context of the approximate 8.0 km squared (5.25 mi²) Copperwood property, this is a relatively small area and there will be very little in the way of wildlife displacement due to habitat loss. In addition, the area surrounding the Copperwood property is generally similar in land use composition. Equipment noise, vehicle traffic along the access roads and other human-induced disturbance associated with the mining operation will likely deter some species such as gray wolf, bobcat and fisher from using the immediate mining area. Other species, such as some song birds or other species commonly present within more urban settings, will adapt to this activity or relocate to nearby suitable habitat. Large mammals present in the Project area such as white-tailed deer, black bear and raccoon, are highly adaptable to a shifting landscape mosaic and moderate human disturbance.

20.4.7.3 Plant and Animal Species of Special Concern

The Redside Dace occurs in the lower reaches of streams crossing the Project site. These fish are currently located in downstream areas below the proposed mine and TDF site footprints, and therefore will not be directly impacted by grading or filling. However, development of the mine components will likely change existing surface water flow patterns and quantities to the upper reaches of the streams. Treated wastewater will also need to be discharged into surface water drainage streams on the site. The treated wastewater may have fewer constituents like sediment and dissolved minerals than natural surface water discharge which may have the potential to alter the current water quality in the streams. Discharge of the treated water is planned to be into the west branch of Namebinag Creek near the mill facility. This water will have over one mile of stream valley and beaver ponds to obtain sediment, dissolved minerals, and dissolved oxygen

before entering the main branch of Namebinag Creek where the Redside Dace have been found. The treated water will have “natural” characteristics by the time it enters the main branch of Namebinag Creek.

Wolves are currently present on and in the vicinity of the Project site. No impact to wolf populations of the area are expected, although local individuals will likely be affected by increased human use of the site and its associated equipment noise, vehicle traffic along access roads, and maintenance of equipment and buildings and will likely leave the immediate area of the disturbance. The 2015 MDNR Wolf Management Plan Update reports an estimated population of 636 wolves (2014 biennial survey) in the Upper Peninsula. The updated plan also notes that Upper Peninsula wolf populations have not changed significantly since 2011. According to the MDNR, the wolf population in the Upper Peninsula increased annually between 1989 and 2009 and surpassed the recovery goal of 200 animals in 2000. Therefore, any displacement caused by the mining facility and operations will not likely jeopardize the recovery or well-being of the overall Michigan wolf population.

Canada lynx have been very rarely documented in the Upper Peninsula. They are dependent upon snowshoe hares as their predominant prey and typically disperse due to fluctuations in snowshoe hare populations. Canada lynx do not inhabit areas devoid of snowshoe hares because they typically consume a snowshoe hare every two or three days. Snowshoe hare populations rise and fall on about a 10-year cycle, therefore Canada lynx may temporarily inhabit an area until such time as snowshoe hare populations decline, and then the Canada lynx disperse. During the 2009 Baseline Wildlife Inventory on the Copperwood Project, only three snowshoe hares were observed (with trail cameras). The 2011 Baseline Wildlife Inventory did not record any snowshoe hare observations. This extremely low density of snowshoe hares reduces the likelihood of Canada lynx being present at the Copperwood Project site.

Although it is possible that a dispersing Canada lynx could travel through the Copperwood Project area, it is highly unlikely that this species would remain in this area unless snowshoe hare populations increase substantially. Therefore, it is unlikely that the Copperwood Project will impact the Canada lynx population in any way.

The showy orchid has been observed in several locations of the Project site. Some occurrences are random, with others having several individuals at one location. Randomly occurring orchid plants within the mine facility and TDF will be lost during facility development work. Larger populations are proposed to be removed intact prior to construction and transplanted to suitable locations in other non-disturbed portions of the site having suitable habitat. Two populations of 23 orchids were transplanted from within the proposed TDF footprint under Part 365 Endangered Species Permit #2004 in October of 2012 with additional monitoring of these two areas and a third community not defined in 2012 continuing since that time. If TDF construction on the proposed site is approved in a new permit, a second transplant permit will

be applied for to move known orchid plants to similar habitat (acidic soils in ephemeral stream drainage areas) that is fairly common on the Copperwood Project site.

Two wood turtles (*Glyptemys insculpta*) were observed within the Study focus area during the 2009 and 2011 Baseline Wildlife Surveys. However, no past nesting areas were found during these surveys and no currently suitable nesting habitat was found during these surveys. Suitable non-nesting habitat areas are found within most stream corridors, but they are mostly used seasonally as the turtles are highly mobile.

20.4.8 Invasive Species

Removal of tree canopy for development of mining site components will expose a greater area of the site to invasive species for colonization. The increase in occurrence of invasive species in fringe areas around developments could increase the overall spread of the species into areas of the site that currently provide specific habitat with diverse numbers of native species. Invasive species have the tendency to colonize areas in dense stands, which can change the overall characteristics of woodlands and wetlands, which in turn affects habitat quality for terrestrial mammal, bird, reptile and amphibian species. There is a monitoring and response plan for dealing with the invasive species.

20.4.9 Air Quality Impacts

Impacts are expected during facility construction, operations, and reclamation. These include but are not limited to; particulate matter from vehicle travel, grading of soils for facility and mine development and materials handling (conveyors and ore stockpile). Combustion source emissions from the power plant on site, underground blasts, and mobile equipment.

All fugitive dust emissions will be controlled using a fugitive dust control plan outlined in the Michigan Air Use Permit – Permit to Install application. This permit application also outlines estimated potential emissions and dispersion modeling for full buildout of the operations phase and must demonstrate compliance with the Clean Air Act's national ambient air quality standards. The models' validity will be monitored on site through rigorous inspections of air quality to show strict adherence to the aforementioned standards.

20.4.10 State Park Impacts

The nearest public park to the Project site is PMWSP. The campground at the mouth of the Presque Isle River is located approximately 3.2 km (2 mi) northeast of the Project site. A campground and day-use scenic picnic areas are located on the west end of the park along the shoreline of Lake Superior. Because

the mining project area is set back from the public roadways in heavily forested land and over a mile from the park, it is unlikely that development and operation of the mine will result in impacts to the view sheds from view points in the use areas of the park. Long-term operation of the mine with subsequent increase in height of tailings disposal piles will eventually change the view shed in the immediate area of the mining operation. However, the view shed within the forested campground and picnic areas of the park will likely remain unchanged.

Portions of the PMWSP, Section 5 of T49N, R45W, and portions of Section 31, of T50N, R45W are adjacent to property owned by Copperwood. Those areas of the Park contain no campgrounds, trails, or other development, and the mine portal and milling facility will be located more than 2.4 km (1.5 mi) to the west of the Park. The TDF, however, is proposed to be constructed adjacent to the Park, within portions of Sections 6, 7, and 8 of T49N, R45W. The completed dike of the TDF will be adjacent to the corner of Section 5 and will be visible from this parcel of the Park.

Indirect impacts of mine development and operation will result in increased traffic on CR 519 which will become the main access route to the mine site. CR 519 is currently the only western access road from the south to the Park and during certain time periods the sound of the mining operation may travel to the Park due to its location to the east and down-gradient of prevailing westerly winds.

20.4.11 Aesthetic Impacts

Portions of the operation will be visible from the segment of the North Country National Scenic Trail ("NCNST"), which is near the mine site. Hikers on the NCNST may be able to see traffic, berms for the TDF, mine-related structures, fencing, and lights, and hear noise from the operation. These impacts may be visible through a cover of trees and may be more visible during periods when leaves are not on the trees. These limited views will not adversely impact the aesthetics or use of the hiking trail. Nonetheless, relocation of the NCNST about 0.76 km (2,500 ft) to the south is currently being negotiated and would minimize any perceived adverse impacts. Some lights may also be visible from the lake. However, due to the remote location of the Project, no potentially sensitive receptors have been identified near the site, and very few people will be present within the immediate vicinity of the operation. Therefore, impacts to aesthetic resources will be minimal.

20.4.12 Acoustic Impacts

During construction activities, noise from heavy equipment will be associated with the Project site. During mining operations, noise from vehicles, material handling, crushing, ventilation and other mine-related operations will be evident in the area immediately surrounding the mining operations. This noise, however,

will be attenuated by the surrounding forest cover and should not degrade the aesthetic resources of the surrounding area. Noise may travel slightly more during the periods of the year when leaves do not cover the trees. Slight impacts may be encountered along the NCNST, but only along that portion of the trail which is closest to the Project site.

During routine operations traffic, material handling, ore handling and crushing, the mine ventilation system, and the power generators will generate noise. Crushing and separation activities will be contained within a building, thereby reducing this source of noise to the outside environment. The surrounding forest cover will mitigate operational noise levels. The mine ventilation fans will be located at the bottom of the box cut entrance thereby reducing the ambient surface noise.

No flora or fauna have been identified that would be potentially affected by the anticipated facility noise conditions.

20.4.13 Seismic Impacts

Due to the remoteness of the Project site, no potential receptors will be impacted by seismicity related to the Project. No structures, other than mine-related infrastructure, will be within the area impacted by seismic disturbances related to the mining activity. No flora or fauna have been identified which will be potentially affected by the anticipated seismicity related to the facility. No seismic monitoring plan will be initiated at the site.

20.5 Tailings Management

Impacts associated with the TDF are discussed throughout this section and the design parameters are included in Section 18.

20.6 Cumulative and Additive Impacts

Cumulative impacts are environmental impacts resulting from the proposed mining activities when added to the other past, present, and reasonably foreseeable future activities. As such, the potential cumulative impacts resulting from the mining activities are minimal to non-existent. The site is located in a remote, forested area where the only current activities are managed forest activities and recreation. No other active mining operations are currently present within Gogebic or Ontonagon Counties. Highland has an interest in restarting mining at the former White Pine, MI mine site in Ontonagon County but at a distance of 33.8 km (21 mi) from the Copperwood site. There has also been recent mineral exploration activity in Iron County, WI at an iron ore prospect located 51.5 km (32 miles) from the Copperwood site.

Discharges of water will have negligible impact to Lake Superior and no other sources of regulated discharges are near the mine site to combine for a cumulative effect. Air emissions from the proposed mine will be minimal and will not combine with other sources to produce regionally degraded air quality. Removal of groundwater from the mine workings and impacts to groundwater will not have significant impact beyond the immediate area of the proposed mine. Mining operations will not impact public or private water supply sources. Due to the remote location of the proposed mine, the low population density of the western Upper Peninsula, and the large distances between the scattered towns in the surrounding area, impacts from any one individual source is not expected to combine with other sources, which are separated by distances of many miles, to create a cumulative impact.

Minimal impacts may include the aesthetic impacts resulting from the proposed mining operation to patrons of the nearby state and federal recreational land. Increased traffic will be present on CR 519 but will not be significantly different than logging trucks which currently utilize the road.

Additive impacts are those which combine together to create a more significant impact together than they would have individually. Impacts to topography and drainage, surface water flow, groundwater and surface water quality, wetlands, air quality and air deposition, and aesthetic resources may be impacted by an additive effect from a variety of site activities.

Potential impacts to the site features due to the proposed mining activities range from negligible to potentially significant. Expected potential impacts are discussed in the previous sections.

Overall, expected impacts to various features are generally minor. At the completion of mining operations, the site will be reclaimed to a self-sustaining forest habitat. The TDF will be capped and remain in place, and groundwater in the immediate vicinity of the TDF may be impacted by seepage from the TDF. Groundwater flow is very slow, as was documented during the baseline survey work in the Copperwood Project area, and therefore impacts will be minor. In sum total, impacts will be more significant during site development, less significant during the operational period, and are expected to be minor after reclamation is complete.

20.7 Environmental Management Plan

During construction, operation, and post-closure of the mine facilities, permit required monitoring and mitigation plans will be executed to minimize environmental impacts. The monitoring is done on varying timelines but includes:

- Surface Water Quality;

- Surface Water Flow;
- Groundwater Quality;
- Wildlife and Vegetation Surveys;
- Leachate Collection and Leak Detection Systems;
- Liners, Pipelines, Berms, and Embankments.

Plans associated with each phase of mine operation include:

- Soil Erosion and Sediment Control Plan;
- Spill Prevention Control and Countermeasures Plan;
- Pollution Incident Prevention Plan;
- Storm Water Pollution Prevention Plan;
- Invasive Species Management Plan;
- Wetland Mitigation Plan;
- Stream Mitigation Plan;
- Sampling Analysis Plan;
- Contact Water Management Plan;
- Quality Assurance and Quality Control Plan for Site Monitoring;
- Fugitive Dust Control Plan;
- Reclamation and Closure Plan;
- Treatment and Containment Plan;
- Integrated Contingency Plan;
- Monitoring Plan for Reactive Material;
- Threatened and Endangered Species Plan.

20.8 Reclamation Plan and Financial Assurance

Upon the start of mine construction, under R 425.301 of Part 632, a financial assurance instrument must be put in place to cover the cost to administer and to hire a third party to implement the reclamation,

remediation, and post closure monitoring plan for the mine site. Release of the financial assurance instrument will be made based on the requirements listed below:

- Reclamation of the mine box-cut and impacted area;
- Demolition and/or removal of the process plant and the related features;
- Demolition and/or removal of ancillary structures;
- Demolition and/or removal of utilities infrastructure;
- Reclamation of the TDF.
- The reclamation of the TDF is expected to take six years. Release of the financial assurance instrument should be proportional to the work completed; to be determined and agreed upon by MDEQ and Copperwood.
- Full release of the financial assurance instrument upon documentation of:
 - ✓ Successful reclamation of TDF;
 - ✓ Successful reclamation of mine site;
 - ✓ Successful reclamation of processing facility;
 - ✓ Successful completion of the 20-year post closure monitoring plan.

The reclamation plan serves to bring the Project site back to a self-sustaining ecosystem that is close to pre-existing conditions. This includes:

- Reclaimed topography and land use;
- Surface features remaining after reclamation;
- Roads and dikes;
- TDF reclamation;
- Plant site reclamation;
- Disposal of waste materials;
- Closure of the underground mine access;
- Site revegetation;
- Groundwater and surface water quality monitoring.

20.9 Legal Framework

Michigan's Public Act 451 of 1994 as Amended, commonly known as the NREPA, sets the framework for environmental regulations in Michigan. This Act is subdivided into many Parts, encompassing all major natural resource and environmental protection topics.

20.10 Permitting Process

The permitting process begins by conducting a baseline study of the conditions of the Project area. In Michigan's Part 632, this means a 2-year survey of topics relating to water, soil, air, vegetation, wildlife, social, cultural, and historical resources. The Project plans and related discussions of those plans must be completed to fully grasp and discuss the impacts associated with the Project. Then the application for a permit can be created and submitted to the responsible regulatory agency. The regulatory agency reviews the documents and requests corrections, clarifications, or amplifications, if needed, from the permittee until they deem the application administratively and technically complete. Once this is done, the application goes out for public comment and a public hearing may be held on the application. Any outstanding questions during this period will need to be answered and, once this process is completed the regulatory agency can either grant or deny the permit. The permit application must demonstrate compliance with all of the regulatory requirements in the governing Part of Michigan's NREPA.

20.11 Permits to Obtain

To start construction and begin operation of this project a plethora of permits must be obtained and agreed upon between Highland and the Regulators, on both the state and federal levels. The major environmental permits required include:

- Part 632 Non-Ferrous Metallic Mining Permit;
- Part 31 National Pollutant Discharge Elimination System Permit;
- Part 55 Air Permit to Install;
- Part 301 Inland Lakes and Streams Permit;
- Part 303 Wetland Permit;
- Part 315 Dam Safety Permit;
- Part 325 Bottomlands Permit;
- Section 10 US Army Corps of Engineers Water Intake Permit.

Other minor and local permits are also required to start construction and mine operation that include:

- Local building and zoning permits;
- Explosives handling permit from the US Bureau of Alcohol, Tobacco, and Firearms;
- Storage tank permits;
- Mine Safety and Health Administration registration.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

The capital cost estimate is a detailed, bottoms-up, built-up effort by major facility and discipline. Each discipline executed a detailed cost build up by cost type, labor, material, equipment, consumables, construction materials and services costs.

This capital cost is estimated at US \$275 M and has an accuracy within a range of -10% / +15%. A summary of the capital expenditures is presented in Table 21.1.

Labor and equipment costs for the Project were built up in a separate analysis to be included in each individual estimate. Material take offs were also performed to generate the baseline quantities for the Project. Each discipline estimate cost, in complete cost type details and quantities and consistent with the Project's work break-down schedule ("WBS"), was then accumulated in a master estimate summary.

Most of the critical materials and components will be sourced in North America and more specifically in the USA.

The estimate was developed by major group areas, which are then further subdivided in distinct areas, disciplines and activities and are included in each estimate line item per GMSI's standard WBS.

The approach allows for an efficient conversion of the estimate data, which is identical in WBS format to a control budget for project execution.

According to standards established at the outset of the Project, pricing of equipment, material and labor were estimated according to the following guidelines:

- Equipment proposals received specifically for the Project;
- Equipment prices derived from recent project or from databases;
- Material prices based on quotations received from suppliers;
- Labor rates based on quotations received from contractors, labor suppliers and wage surveys in the Upper Peninsula of Michigan.

Table 21.1: Capital Expenditures Summary

Capital Expenditures	USD
000 - General	1,149,855
100 - Infrastructure	36,649,739
200 - Power and Electrical	5,156,000
300 - Water	22,874,761
400 - Mobile Equipment	27,240,392
500 - Resettlement	53,528,725
600 - Process Plant	45,771,089
700 - Construction Indirects	27,609,372
800 - General Services	22,250,925
900 - Pre-production, Start-up, Commissioning	9,837,990
990 – Contingency	22,889,282
Grand Total	274,968,132

Locally available material was used when possible for estimation purposes and prices were sourced from regional suppliers.

No escalation was built into the capital cost estimates. The estimates are as of Q1-2018.

The estimates include earthworks, construction material, equipment, and labor. Earthworks will be performed by regional contractors when possible.

21.1.1 Infrastructures

A CAPEX summary for infrastructures is presented in Table 21.2. The detailed description of infrastructures and roads are presented in Section 18.

Table 21.2: Infrastructures CAPEX

Capital Expenditures	US\$
110 – Roads	6,515,298
111 - Main Access Road	1,409,900
112 – Site Roads	1,373,100
114 - Fencing	5,000
117 – Employee Parking Lot	543,750
119 - Road 519 improvements	3,183,548
120 - Workshops / Storage	1,860,558
123 - Plant Workshop & Stores	496,715
124 - Reagents Storage Building	888,747
125 - Explosives Plant / Magazine	475,096
130 - Support Buildings	12,490,359
131 - Workshop, Warehouse, Lunchrooms & Dry Building	6,453,710
133 – Mill Office (Construction Office) / Met Lab / Control Room	1,985,806
135 - Main Gatehouse	383,655
138 - Off-Site Facilities - Transload Building & facilities	3,667,187
150 - Process Plant Buildings system	13,601,423
151 - Process Plant Main Building	13,601,423
160 - Laboratories	1,986,100
161 - Assay, Environmental Laboratory	1,986,100
170 - Fuel Systems	196,000
173 - Diesel Fuel Storage	196,000
Grand Total	36,649,739

21.1.2 Power Supply and Communications

A summary of the CAPEX for electrical and communications is presented in Table 21.3. They include all equipment and installations for power supply and distribution. The power line and main site substation costs are negotiated with the power rates with the utility company and therefore are not shown in this table. The electrical infrastructures are detailed in Section 18 of this Report.

Table 21.3: Power Supply and Electrical Capital Expenditures

Area	US\$
210 - Main Power Generation	484,000
217 - Emergency Power Generation (Surface)	484,000
220 - Process Plant Electrical Rooms	2,355,000
221 - Process Plant E-Room	1,560,000
225 – Tailings E-Room	245,000
225 – Other E-Room	550,000
240 - Site Power Distribution	599,000
241 - Site Powerlines	599,000
270 - U/G Communications Network	468,000
271 - U/G Communications Network	468,000
Grand Total	5,156,000

21.1.3 Water and Tailings Disposal Management

Details and description of Tailings and Water Disposal Management (“TDM”) installation and systems are provided in Section 18. The Tailings Disposal Facility (“TDF”) is built in three phases in which the phase 1 costs are included in the initial CAPEX. The two other phases are planned for construction and delivery for 2023 and 2026 and therefore are included in sustaining expenditures. Capital costs include earthworks, concrete, structure steel, mechanical, electrical and instrumentation equipment and labor.

The surface water management system is constructed to gather all contact water generated on site. It includes the lined ditches, pumping station and pipelines from pumping stations to the event pond. From the event pond, the plan is to ultimately pump the water to the TDF.

The Lake Superior water in-take works include the directional boring/pipe and the pumping station.

The fire water estimate includes the fire pumps, the distribution network within the processing and mine plant.

A CAPEX summary for water is presented in Table 21.4.

Table 21.4: Tailings & Water Capital Expenditures

Area	US\$
310 - Raw Water Supply & Potable Water	4,668,970
311 - Process Water	597,200
312 - Potable Water Treatment & Filtration	453,020
316 - Lake Superior In-Take	3,618,750
320 - Reclaim Water	2,452,110
321 - Reclaim Water System	864,782
322 - Reclaim Pipeline	1,395,600
322 - Gland Water	191,728
330 - U/G Water Management	494,375
331 - Water Management Surface	494,375
340 - Tailings Disposal Facility	13,326,081
341 - TDF Roads	42,625
342 - TDF Main Dams	11,346,396
346 - TDF Pipeline	1,937,060
360 - Effluent Water Management	200,000
361 - Final Effluent Pipeline and Diffuser	200,000
370 - Fire Water	755,500
371 - Fire Water Distribution	755,500
380 - Domestic Sewage	977,725
381 - Sewage Treatment System	977,725
Grand Total	22,874,761

21.1.4 Mobile Equipment

Mine Equipment includes all capital expenditures related to the acquisition of primary mining and support equipment. Equipment CAPEX include the purchasing cost, assembly cost and all safety and optional installs on the equipment.

Construction mobile equipment includes purchasing costs for a front-end loader to be used to lift equipment. All other equipment is either included in construction contracts or rented. Rental costs for light vehicles required for the construction commissioning period.

A summary for the capital expenditures for mobile equipment is presented in Table 21.5.

Table 21.5: Mobile Equipment Capital Expenditures

Area	US\$
410 - U/G Mining Equipment & Maintenance	26,518,771
412 – U/G Mining Equipment	19,728,842
414 - U/G Support Equipment	6,682,882
419 - Mining Equipment Capital Spares	107,046
420 - Construction Vehicles and Equipment	321,621
422 - Light Vehicles and Other Equipment	321,621
430 - Surface Mobile Equipment	400,000
431 - Surface Mobile Equipment	400,000
Grand Total	27,240,392

21.1.5 Mine Infrastructure

Mine infrastructure CAPEX include the portal excavation, installation of multi-plate culverts, and backfill. Hauling starts outside of the ramp to the ore stockpile. Mine development includes labor, consumables to complete the drifts to reach mining panels.

Other costs are all related to safety, utilities work and infrastructure such as refuge, lunchrooms, ventilation raises, in-take and exhaust and pumping systems.

Mine infrastructure also includes the feeders and underground main conveyor to be installed over the pre-production period.

A summary of the CAPEX for mine infrastructure is presented in Table 21.6.

Table 21.6: Mine Infrastructure Capital Expenditures

Area	US\$
510 - Surface Mine Infrastructure	2,444,710
512 – Haul Road	479,610
515 - Ore Handling / Reclaim	1,020,100
517 - Ore Stockpile Pad	945,000
520 - U/G Mine Infrastructure	39,798,329
522 - Portal (Box-cut)	5,455,026
526 - Level Development	34,046,930
529 - U/G Mine Refuge / Lunch Room	296,373
530 - Ventilation raise & Escapeways	7,056,120
531 - Collar & Excavation	6,810,878
533 - Power Supply / HVAC	245,241
550 - U/G Mine Dewatering System	359,294
551 - U/G Mine Dewatering System	359,294
570 - U/G Explosives Storage	17,500
571 - U/G Explosive Storage Facility	17,500
580 - U/G conveying/crushing system	3,852,770
581 - Feeder breakers and Primary conveyors	3,852,770
Grand Total	53,528,725

21.1.6 Process Plant and Related Infrastructures

The initial capital cost estimate for the processing facility is provided in Table 21.7. The estimate includes earthworks, concrete, structural steel, mechanical, piping, electrical / instrumentation and architecture equipment and labor.

Quantities for the earthwork, concrete, structure, piping, electrical, instrumentation and architecture material take-offs were estimated by Lycopodium. The unit rates for material were estimated by GMSI. The list of mechanical equipment was derived from PFDs and P & IDs.

The estimate covers all costs and construction works related to the processing plant. The process plant

building, and other secondary structural steel are included in Area 150. Scope includes the haul ramps to access the feed hopper and finishes at the tailings pumps located after cyanide destruction. All related plant auxiliary services and reagents are also included.

The capital costs estimate for the processing areas is presented in Table 21.7.

Table 21.7: Processing Capital Expenditures

Processing Capital Costs	US\$
610 - Crushing and Ore Handling	4,728,043
611 - Conveyor/Stacker	1,411,000.0
615 - Reclaim Circuit	3,317,043
620 – Grinding	21,390,500
621 - Grinding & Cyclopak	21,376,300
622 - Media Storage	14,200
630 – Flotation/Regrind Circuit	13,285,925
631 - Conditioning Tank	2,018,794
632 - Rougher Cells	2,714,380
633 - Scavenger/1st Cleaner Cells	2,148,847
634 - 2nd Cleaner Cells	697,928
635 - 3rd Cleaner Cells	535,172
636 - Cyclone & Regrind	5,170,804
640 – Tailings	803,600
642 - Flotation Tailings	803,600
650 – Copper Concentrate Filtration; Thickening & Handling	3,458,180
651 - Cu Concentrate Thickening	1,274,779
652 - Cu Concentrate Filtration	1,861,113
670 – Reagents	1,565,735
671 - Lime Circuit	757,443
672 – MIBC	160,914
673 – PAX	254,740
674 – NaHS	116,375
675 - Na ₂ SiO ₃	122,482
676 – Flocculant	153,779
680 – Plant Services	539,104
681 - Compressed Air	117,368
682 - Low Pressure Compressed Air	421,736
Grand Total	45,771,089

21.1.7 Construction Indirect Costs

Construction indirect costs include all the engineering activities as well as site construction management. A full suite of temporary facilities is also included as well as tools and operating and maintenance costs for construction equipment.

Construction Indirect Costs are presented in Table 21.8.

Table 21.8: Construction Indirect Capitals

Construction Indirects	US\$
710 - Engineering, CM, PM	25,029,372
711 - Site CM staff and consultants	7,487,274
713 - Montreal CM staff and consultants	7,361,190
715 - External Engineering	6,230,907
716 – Surveying	1,000,000
717 - QA/QC	2,200,000
718 - Commissioning and Vendor's Rep	250,000
719 - Induction / Travel / Visas / Working Permits	500,000
720 - Construction Facilities & Services	2,580,000
722 - Construction Temporary Services	1,080,000
727 - Construction Tools / Consumables	500,000
729 - Construction Equipment Rentals	1,000,000
Grand Total	27,609,372

21.1.8 General Services

General Services include all the support departments, generally directly hired by Highland, that will be staffed and organized to assist during the development stage of the Project and will continue their functions during the operating phase; it includes the following:

- General Administration (GM);
- Supply Chain Local;
- HR & Training;
- Health and Safety;

- ESR;
- Security;
- IT;
- Accounting and Finance.

All freight is estimated from quotations or from similar recent projects. Corporate costs are not charged to the Project. Temporary power costs include fuel and maintenance for power consumption the construction and plant needs. Cost estimates are presented in Table 21.9.

Table 21.9: General Services Expenditures

General Service's Owner's Costs	US\$
810 – Departments	12,279,664
811 - General Administration	1,230,935
812 - Supply Chain	1,295,050
813 - HR & Training	1,272,117
814 – ESR	719,593
815 – Health & Safety	418,027
816 – Security	874,950
818 – IT	5,668,060
819 - Accounting & Financing	800,932
820 - Logistics / Taxes / Insurance	7,406,000
821 – Freight	7,406,000
840 - Other Costs	2,565,261
841 - Mitigation Area Works	1,575,401
842 - Stream Relocation	989,860
Grand Total	22,250,925

21.1.9 Pre-production and Commissioning Expenditures

The pre-production costs are those of the process plant as mining pre-production costs are covered in Area 526 and Owner's costs are captured in Areas 811 to 819.

The process plant pre-production includes initial fills as well as salaries and reagents and fuel during the commissioning and ramp-up period to commercial production. Staffing and training of mill personnel is planned progressively in the 12-month period before commissioning.

Pre-production and commissioning expenditures are presented in Table 21.10.

Table 21.10: Pre-Production and Commissioning Expenditures

Area	US\$
950 - Process Plant Pre-Prod. & Commissioning	9,837,990
955 - Process Plant Mgmt and Training	7,090,843
956 - Process Plant Commissioning	1,425,000
958 - First Fill	871,147
959 - Commissioning Spares	451,000
Grand Total	9,837,990

A 9.1% contingency on all costs was included for a total of US\$22.9 M.

21.2 Sustaining Capital

Sustaining capital of US\$156.5 M is required over the life-of-mine for the following main items:

- TDF expansion;
- WTP;
- Mine equipment purchases;
- Mine development expenditures.

Sustaining capital is required for the TDF expansion for Stage 2 and Stage 3. Stage 2 is constructed in 2022 and Stage 3 is constructed in 2025.

The effluent water treatment plant is constructed in 2025 to be operational when Stage 3 of the TDF is used for tailings disposal in 2026 as required by the water balance estimates.

A summary of sustaining capital is presented in Table 21.11 and on an annual basis in Table 21.12.

Table 21.11: Summary of Sustaining Capital Costs

Sustaining CAPEX	LoM (\$M)	\$/t ore	\$/lb Cu Payable
Tailings Disposal Facility Expansion	28.45	1.14	0.04
Water Treatment Plant	6.13	0.25	0.01
Mine Equipment Purchases	43.69	1.75	0.07
Mine Development Expenditures	78.21	3.13	0.12
Total Sustaining CAPEX	156.47	6.26	0.24

Table 21.12: Sustaining Capital Costs

Sustaining CAPEX (US\$M)	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Mine Equipment												
Jumbo Drills	3.88	2.33	0.78	-	-	-	0.78	-	-	-	-	-
Bolters	8.90	5.19	1.48	1.48	-	-	0.74	-	-	-	-	-
LHD 10t	3.07	2.30	-	-	0.77	-	-	-	-	-	-	-
Scaler	0.61	0.61	-	-	-	-	-	-	-	-	-	-
Loading Point & Rock Breaker	7.78	2.83	2.12	1.42	-	-	1.42	-	-	-	-	-
Scissor Lift	1.76	1.76	-	-	-	-	-	-	-	-	-	-
Tractor & ATV	0.62	0.26	0.13	-	-	0.03	0.03	-	-	-	0.16	-
Equipment Major Components	18.72	0.51	1.20	2.65	2.02	3.19	1.88	1.13	2.05	2.27	1.64	0.18
Sub-Total Mine Equipment	45.35	15.80	5.72	5.55	2.79	3.22	4.85	1.13	2.05	2.27	1.80	0.18
Material Handling												
Conveyor Purchases	16.84	2.40	2.87	3.13	7.16	0.40	0.87	-	-	-	-	-
Conveyor & Rock Breaker Moves	12.46	2.04	3.77	3.25	0.33	0.10	0.91	0.09	0.55	1.17	0.20	0.05
Sub-Total Materials Handling	29.31	4.44	6.65	6.38	7.49	0.50	1.78	0.09	0.55	1.17	0.20	0.05
Other Mining Equipment												
Ventilation Raises & Fans	2.11	0.10	0.08	-	-	1.93	-	-	-	-	-	-
Other mining equip. & Small Tools	1.44	0.24	0.43	0.10	0.06	0.06	0.19	0.21	0.04	0.04	0.04	0.02
Sub-Total Other Mining	3.55	0.34	0.51	0.10	0.06	1.99	0.19	0.21	0.04	0.04	0.04	0.02
Sustaining Mining Capital	78.21	20.59	12.87	12.04	10.34	5.72	6.81	1.43	2.64	3.48	2.04	0.25
Mining Development	43.69	13.30	7.56	4.99	5.12	7.76	4.96	-	-	-	-	-
Tailings Facility Expansion	28.45	-	13.98	-	-	14.47	-	-	-	-	-	-
Water Treatment Plant	6.13	-	-	-	-	3.06	3.06	-	-	-	-	-
Total Sustaining CAPEX	156.47	33.89	34.41	17.02	15.46	31.01	14.84	1.43	2.64	3.48	2.04	0.25

21.3 Closure Costs and Salvage Value

The closure costs are estimated to US\$28.24 M net of US\$10.75 M of salvage value from plant major equipment.

Closure costs would cover the following activities:

- Tailings reclamation;
- Site closure, dismantling and reclamation;
- Salvaging of plant major equipment;
- Post closure monitoring;
- MDEQ oversight.

The closure cost estimate is presented in Table 21.13 with these costs incurred over a two-year period after commercial operations (i.e. during 2032 and 2033).

Table 21.13: Closure Cost & Salvage Value

Closure Cost Estimate	Unit	Unit Price	Qty	Cost (\$k)
TDF Reclamation				
TDF Disposal Area Reclamation Stage 1	sq.m	28.41	205,000	5,823
TDF Disposal Area Reclamation Stage 2	sq.m	28.41	295,000	8,380
TDF Disposal Area Reclamation Stage 3	sq.m	28.41	400,000	11,363
Sub-Total			900,000	25,566
Site Closure & Reclamation				
Place and Compact Soil Cover	cu.m	2	200,000	450
Place and Hydroseed Topsoil	sq.m	2	2,330,000	4,544
Structural Steel Demolition	tonnes	600	2,500	1,500
Concrete Demolition	tonnes	8	35,000	280
Concrete Disposal	tonnes	2	35,000	70
Modular Building Removal	sq.m	50	200	10
Mechanical Pipelines	lot	500,000	1.00	500
Electrical Distribution	lot	500,000	1.00	500
Tank Removal and Disposal	lot	10,000	1.00	10
Admin Support	%	0	1.00	1,180
Sub-Total				9,043
General / Reclamation	lot	56,275	1.00	56
Salvage Value				
120-Workshops/Storage	lot	18,750	(1)	(19)
130-Support Facilities	lot	375,198	(1)	(375)
160-Laboratory	lot	540,600	(1)	(541)
210-Main Power Generation	lot	-	(1)	-
220-Process Plant Electrical Rooms	lot	824,250	(1)	(824)
310-Raw Water Supply & Potable Water	lot	786,449	(1)	(786)
400 - Mobile Equipment	lot	4,192,459	(1)	(4,192)
430-Surface Mobile Equipment	lot	80,000	(1)	(80)
610-Ore Handling	lot	411,000	(1)	(411)
620-Grinding	lot	1,609,500	(1)	(1,610)
630-Flotation/Regrind Circuit	lot	1,399,485	(1)	(1,399)
640-Tailings	lot	24,750	(1)	(25)
650-Copper Con. Filtration, Thickening & Handling	lot	314,775	(1)	(315)
670- Reagents	lot	101,204	(1)	(101)
680- Plant Services	lot	72,750	(1)	(73)
Sub-Total				(10,751)
Post Closure Monitoring (DCF 5%)	lot	2,900,392	1.00	2,900
MDEQ Admin Oversight	%	5.0%	1.00	1,423
Total Cost				28,238

21.4 Operating Costs

OPEX are summarized in Table 21.14. The operating costs include mining, processing, G&A and royalties. The costs for concentrate transportation to smelters and smelting and refining charges are not considered site operating costs and are therefore excluded from the OPEX estimate.

The transportation costs and smelter conversion charges ("TC/RC") are deducted from gross smelter revenues to estimate the NSR. These costs are detailed in Section 19 on Market Studies and Contracts.

The LoM operating cost summary is presented in Table 21.14 and that for the first 5-years in Table 21.15. The OPEX by year is presented in Table 21.16. The LoM unit operating cost is estimated at US\$1.53/lb of payable copper and lower at US\$1.25/lb for the first 5-years due to the higher grades processed initially.

Table 21.14: LoM Operating Cost Summary

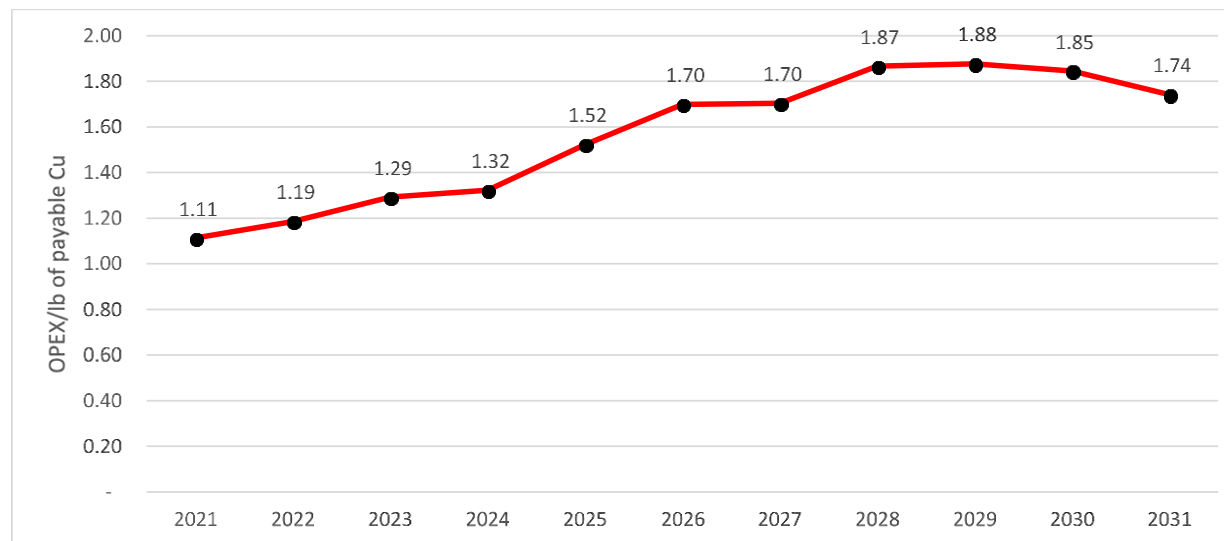
LoM OPEX by Area	Total Cost (\$M)	Unit Cost (\$/tonne milled)	Unit Cost (\$/payable lb)	%
Royalties	85	3.39	0.13	8.5%
Mining	531	21.26	0.82	53.4%
Processing	308	12.31	0.47	30.9%
General & Administration	72	2.88	0.11	7.2%
Total Site Costs (incl. Royalties)	996	39.84	1.53	100%

Table 21.15: First 5-year Operating Cost Summary

First 5-Year OPEX	Total Cost (\$M)	Unit Cost (\$/tonne milled)	Unit Cost (\$/payable lb)	%
Royalties	45	3.95	0.13	10.4%
Mining	219	19.30	0.65	50.6%
Processing	140	12.34	0.42	32.4%
General & Administration	29	2.52	0.09	6.6%
Total Site Costs (incl. Royalties)	432	38.11	1.29	100%

Table 21.16: Annual Operating Costs

OPEX Summary (\$M)	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Royalties	84.71	7.52	10.93	9.61	9.11	7.63	7.11	7.43	6.85	6.79	6.90	4.81
Mining	531.36	22.69	47.36	49.97	50.36	48.42	51.27	55.12	56.66	56.41	56.61	36.50
Processing	307.74	21.24	29.64	29.64	29.70	29.64	29.64	29.64	29.70	29.64	29.46	19.77
G&A	72.01	4.51	6.01	6.01	6.01	6.01	7.36	7.81	7.81	7.81	7.59	5.07
Total	995.82	55.96	93.94	95.23	95.18	91.70	95.39	100.00	101.03	100.65	100.57	66.16
Unit Cost (\$/t milled)	39.84	33.05	39.00	39.53	39.40	38.07	39.60	41.51	41.82	41.78	41.75	41.18
Unit Cost (\$/pay. lb Cu)	1.53	1.11	1.19	1.29	1.32	1.52	1.70	1.70	1.87	1.88	1.85	1.74

Figure 21.1: Operating Cost per lb of Payable Copper

21.4.1 Mining Costs

The operating mining costs were evaluated based on the LoM, supplier quotations, a detailed wage scale and standard industry practice.

The mining costs are divided into ten categories that represent the major mining activities. Table 21.18 presents the annual mining costs over the LoM.

Table 21.17: Mining Operating Cost Summary

Mine OPEX Summary	LoM Cost (\$M)	\$/t ore milled	\$/lb Payable	%
Mine Supervision	21.39	0.86	0.03	4.0%
Production Drilling	53.81	2.15	0.08	10.1%
Blasting	86.65	3.47	0.13	16.3%
Stope Piping, Scaling & Serv.	43.56	1.74	0.07	8.2%
Ground Support	103.73	4.15	0.16	19.5%
Hauling	38.24	1.53	0.06	7.2%
Mine Services and Const.	66.71	2.67	0.10	12.6%
Mechanical Maintenance	40.74	1.63	0.06	7.7%
Electrical Maintenance	51.59	2.06	0.08	9.7%
Technical Services	24.95	1.00	0.04	4.7%
Total Mining Cost	531.36	21.26	0.82	100%

The four main costs for mining is labor, equipment maintenance, explosives and ground support. These four main costs represent 75% of all mining costs. Manpower alone accounts for 39% of the mining costs (US\$8.11/t).

Table 21.18: Annual LoM Mining OPEX

Mining Costs (\$M)	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Mine Supervision	21.39	1.03	2.00	2.03	2.03	1.93	2.03	2.23	2.24	2.23	2.23	1.40
Production Drilling	53.81	2.18	4.94	5.09	5.15	4.98	5.18	5.55	5.66	5.63	5.70	3.76
Blasting	86.65	3.41	7.88	8.19	8.24	7.96	8.33	8.99	9.21	9.16	9.29	5.99
Stope Piping, Scaling & Serv.	43.56	2.23	4.25	4.20	4.28	4.27	4.26	4.23	4.30	4.28	4.32	2.94
Ground Support	103.73	3.71	8.52	9.43	9.37	8.98	9.71	11.15	12.09	11.97	12.03	6.78
Hauling	38.24	2.30	3.75	3.78	3.97	4.01	3.96	3.47	3.58	3.55	3.63	2.25
Mine Services and Const.	66.71	2.70	5.96	6.39	6.45	6.01	6.53	7.02	6.97	7.01	6.97	4.70
Mechanical Maintenance	40.74	1.80	3.52	3.78	3.79	3.56	3.87	4.28	4.29	4.31	4.28	3.28
Electrical Maintenance	51.59	2.10	4.27	4.69	4.69	4.45	5.01	5.58	5.68	5.65	5.54	3.94
Technical Services	24.95	1.23	2.27	2.39	2.39	2.27	2.40	2.63	2.64	2.63	2.63	1.47
Total Mining Cost	531.36	22.70	47.36	49.97	50.36	48.42	51.27	55.12	56.66	56.41	56.61	36.50
Unit Cost (\$/t milled)	21.26	13.40	19.66	20.74	20.85	20.10	21.28	22.88	23.46	23.41	23.50	22.72
Unit Cost (\$/payable lb Cu)	0.82	0.45	0.60	0.68	0.70	0.80	0.91	0.94	1.05	1.05	1.04	0.96

Note: Excludes costs during pre-production which are included in the initial CAPEX

21.4.2 Processing Costs

The process plant operating costs were evaluated based on results of metallurgical testwork, supplier quotations, a detailed wage scale and standard industry practice. The process costs are divided into seven categories: labour, reagents, grinding media, liners, maintenance supplies and electrical power. The costs include tailings and water pumping but exclude water treatment costs which are included in the G&A environmental costs.

Total process operating cost summary is presented in Table 21.19 and the annual expenditures over the LoM in Table 21.20.

Reagents are the principal cost item in the mill OPEX represent 42% of cost or US\$5.26/t of ore. The reagent consumption rates, reagent prices and resulting unit costs is presented in Table 21.21. Among the reagents required, sodium hydrosulphide (NaHS) is the high consumer and cost item.

The process plant manpower comprises 75 people, including the laboratory staffing of 7 people.

The power consumption is estimated from a detailed load list by plant area by the Lycopodium process engineer. The process plant power includes power for the mill only as power for G&A and mining are provisioned for in each respective budget. The power supply is planned from the utility company grid with an indicative price of US\$0.0619/kWh for interruptible service with the main substation provided. The power consumption at 6,600 mtpd is estimated at 49.9k Wh/t milled (Table 21.23).

Table 21.19: Process Operating Cost Summary

Mill OPEX	LoM Cost (\$M)	Avg. Cost (\$M/yr)	\$/t ore	\$/lb	%
Mill Labour	58.25	5.44	2.33	0.090	18.9%
Reagents	131.47	12.27	5.26	0.202	42.7%
Grinding Media	12.33	1.15	0.49	0.019	4.0%
Liners	3.39	0.32	0.14	0.005	1.1%
Maintenance Supplies	11.30	1.05	0.45	0.017	3.7%
Operating Supplies	13.75	1.28	0.55	0.021	4.5%
Power	77.25	7.21	3.09	0.119	25.1%
Total Mill OPEX	307.74	28.73	12.31	0.473	100.0%

Table 21.20: Annual LoM Processing OPEX

Mill OPEX (\$M)	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Mill Labour	58.25	4.22	5.62	5.62	5.62	5.62	5.62	5.62	5.62	5.62	5.44	3.62
Reagents	131.47	8.91	12.67	12.67	12.71	12.67	12.67	12.67	12.71	12.67	12.67	8.45
Grinding Media	12.33	0.83	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	0.79
Liners	3.39	0.24	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.23
Maintenance Supplies	11.30	0.81	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08	0.77
Operating Supplies	13.75	0.94	1.32	1.32	1.33	1.32	1.32	1.32	1.33	1.32	1.32	0.89
Power	77.25	5.29	7.43	7.43	7.45	7.43	7.43	7.43	7.45	7.43	7.43	5.02
Total Mill OPEX	307.74	21.24	29.64	29.64	29.70	29.64	29.64	29.64	29.70	29.64	29.46	19.77
Unit Cost (\$/t milled)	12.12	10.17	12.31	12.31	12.30	12.31	12.31	12.31	12.30	12.31	12.23	12.31
Unit Cost (\$/pay. lb Cu)	0.47	0.35	0.37	0.40	0.41	0.49	0.53	0.51	0.55	0.55	0.54	0.52

Note: Excludes costs during pre-production which are included in the initial CAPEX

Table 21.21: Process Plant Reagent Consumption

Reagents	Dosage	Reagent Pricing	Reagent Consumption	Unit Cost (US\$/t)
Sodium Hydrosulphide (NaHS)	2,650 g/t	990 US\$/t	2,624 t/yr	2.62
Sodium Isobutyl Xantate (C-3430)	365 g/t	2,090 US\$/t	763 t/yr	0.76
Methyl Isobutyl Carbinol (MIBC)	45 g/t	1,760 US\$/t	79 t/yr	0.08
Dowfroth 250 (D-250)	45 g/t	4,000 US\$/t	180 t/yr	0.18
Alkylaryl Dithiophosphate (A-249)	215 g/t	3,750 US\$/t	806 t/yr	0.81
n-Dodecyl Mercaptan (NDM)	55 g/t	3,900 US\$/t	215 t/yr	0.21
Sodium Silicates	431.3 g/t	550 US\$/t	237 t/yr	0.24
Carboxymethyl Cellulose Sodium	143.8 g/t	2,310 US\$/t	332 t/yr	0.33
Hydrated Lime	- g/t	125 US\$/t	- t/yr	-
Flocculant	0.3 g/t	3,970 US\$/t	1.1 t/yr	0.00
Anti-Scalant	5.5 L/h	4,250 US\$/m ³	23.4 m ³ /yr	0.02
Total				5.26

Table 21.22: Grinding Media and Liner Consumption

Grinding Media & Liners	Dosage	Consumable Pricing	Media & Liner Consumption	Unit Cost (US\$/t)
SAG Mill Grinding Media	164 g/t	1,093 US\$/t	179 t/yr	0.18
Ball Mill Grinding Media	248 g/t	1,230 US\$/t	305 t/yr	0.31
Regrind Mill Grinding Media	5 g/t	1,766 US\$/t	9 t/yr	0.01
SAG Mill Liner	31 g/t	173,800 US\$/set	0.43 set/yr	0.03
Ball Mill Liner	38 g/t	474,500 US\$/set	0.53 set/yr	0.11
Total				0.63

Table 21.23: Mill Power Consumption by Area

Mill Power by Area	6,000 mtpd Power (kWh/t)	6,600 mtpd Power (kWh/t)
Crushed Ore Conveying, Storage & Reclaim	0.31	0.28
Grinding Circuit	32.86	32.81
Rougher Flotation	2.85	2.59
Regrind Circuit	4.73	4.71
Cleaner Flotation	2.35	2.15
Concentrate Dewatering	1.73	1.65
Tailings	1.42	1.29
On Stream Analyser	1.03	1.02
Reagents Storage and Handling	0.09	0.09
Plant Services	1.96	1.91
Buildings and Power	1.50	1.37
Total	50.82	49.85

21.4.3 General and Administration

G&A includes general management, finance and accounting, supply chain, IT, human resources, health, safety and environment, surface support and corporate and insurance costs.

In most cases, these services represent fixed costs for the site as a whole. The G&A costs exclude certain costs such as transport of concentrates and environmental rehabilitation costs. Water treatment costs are included in environment which represents US\$1.80 M/yr starting in Q2-2026 to the end of mine life.

The G&A labor includes 40 people whose total labor cost represents 55% of the G&A OPEX.

A summary of G&A costs is presented in Table 21.24 and the annual expenditures over the LoM in Table 21.25. The average annual G&A budget is US\$6.72 M or US\$2.88/t of ore.

Table 21.24: General Management and Administration Cost Summary

G&A OPEX by Department	LoM Cost (\$M)	Avg. Cost (\$M/yr)	\$/t ore	\$/lb	%
General Management	4,991	466	0.20	0.008	6.9%
Finance & Accounting	6,076	567	0.24	0.009	8.4%
Supply Chain	8,258	771	0.33	0.013	11.5%
Information Technology	7,481	698	0.30	0.011	10.4%
Human Resources	9,152	854	0.37	0.014	12.7%
Health, Safety & Environment	25,083	2,342	1.00	0.039	34.8%
Surface Support	4,985	465	0.20	0.008	6.9%
Insurance	5,984	559	0.24	0.009	8.3%
Total G&A Costs	72,011	6,723	2.88	0.111	100.0%

Table 21.25: Annual LoM G&A OPEX

G&A Cost (\$M)	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
General Management	4.99	0.36	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.34
Finance & Accounting	6.08	0.44	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.35
Supply Chain	8.26	0.59	0.79	0.79	0.79	0.79	0.79	0.79	0.79	0.79	0.79	0.56
Information Technology	7.48	0.54	0.73	0.73	0.73	0.73	0.73	0.73	0.73	0.73	0.73	0.40
Human Resources	9.15	0.67	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.78	0.56
Health, Safety & Environment	25.08	1.12	1.49	1.49	1.49	1.49	2.84	3.29	3.29	3.29	3.18	2.13
Surface Support	4.99	0.36	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.34
Insurance	5.98	0.43	0.57	0.57	0.57	0.57	0.57	0.57	0.57	0.57	0.57	0.41
Total G&A Costs	72.01	4.51	6.01	6.01	6.01	6.01	7.36	7.81	7.81	7.81	7.59	5.07
Unit Cost (\$/t milled)	2.84	2.16	2.49	2.49	2.49	2.49	3.06	3.24	3.23	3.24	3.15	3.16
Unit Cost (\$/payable lb Cu)	0.11	0.07	0.08	0.08	0.08	0.10	0.13	0.13	0.14	0.15	0.14	0.13

Note: Excludes costs during pre-production which are included in the initial CAPEX

22. ECONOMIC ANALYSIS

The economic analysis presented in this Report uses an economic model that estimates cash flows on a quarterly basis for the life of the Project at the level appropriate to the feasibility level of engineering and design. However, annual amounts are presented for presentation purposes in this Report.

Cash flow projections are estimated over the LoM based on the sales revenue, OPEX, CAPEX and other cost estimates. CAPEX is estimated in four categories, initial, sustaining, closure and reclamation, and working capital. OPEX estimates include labour, reagents, maintenance, supplies, services, fuel and electrical power. Other costs such as royalties, depreciation and taxes are estimated in accordance with the present stage of the Project.

The financial model results are presented in terms of Net Present Value ("NPV"), payback period, and internal rate of return ("IRR") for the Project. The economic analysis is carried out in real terms (i.e. without inflation factors) in Q1 2019 US Dollars without any project or equipment financing assumptions. The economic results are calculated as of the start of initial capital expenditures with all prior costs treated as sunk costs but considered for purposes of taxation calculations.

22.1 Assumptions

The key assumptions influencing the economics of the Project include:

- Metal prices of copper in US\$/lb and silver price in US\$/oz;
- Off-highway diesel fuel price in US\$/L;
- Exchange rates, the US\$/C and US\$/Euro.

22.1.1 Metal Prices

Metal prices and price scenarios are presented in Section 19.1. The base case copper price for economic evaluation follows a declining price profile (2021=US\$3.40/lb, 2022 = US\$3.25/lb, 2023 = US\$3.15/lb) with a long-term price of US\$3.10/lb (2024+). The silver price is kept constant at US\$16.00/oz.

22.1.2 Fuel

The reference diesel fuel price used for estimating operating costs is US\$0.66/L. The diesel fuel price is for off-road or off-highway use by the mine equipment that will not be operated on public roadways. The off-road diesel fuel is not subject to state and federal excise taxes that are applied to retail sales of diesel fuel

or for use in vehicles operated on public roadways (Table 22.1). The off-road diesel fuel is dyed red to make it distinguishable. Under the Nonferrous Metallic Minerals Extraction Severance Tax Act, the operation would be exempt of sales tax once in operation.

Table 22.1: Off-Highway Diesel Fuel Price Assumption

Fuel Price	Pre-Production		Operations	
	US\$/gal.	US\$/L	US\$/gal.	US\$/L
Retail Diesel Fuel Price	3.000	0.793	3.000	0.793
Less: Federal Excise Tax	(0.243)	(0.064)	(0.243)	(0.064)
Less: State Tax	(0.263)	(0.069)	(0.263)	(0.069)
Less: Prepaid Sales Tax	-	-	(0.151)	(0.040)
Less: Petroleum Transfer Fee	-	-	-	-
Off-Highway Diesel Fuel Price	2.494	0.659	2.343	0.619

22.1.3 Exchange Rates

Exchange rates are used to convert certain capital cost and operating cost items in US dollars. The exchange rate assumptions are summarized in Table 22.2.

Table 22.2: Exchange Rate Assumptions

Exchange Rate	Base Value
US\$/C\$	0.78
US\$/Euro	1.22

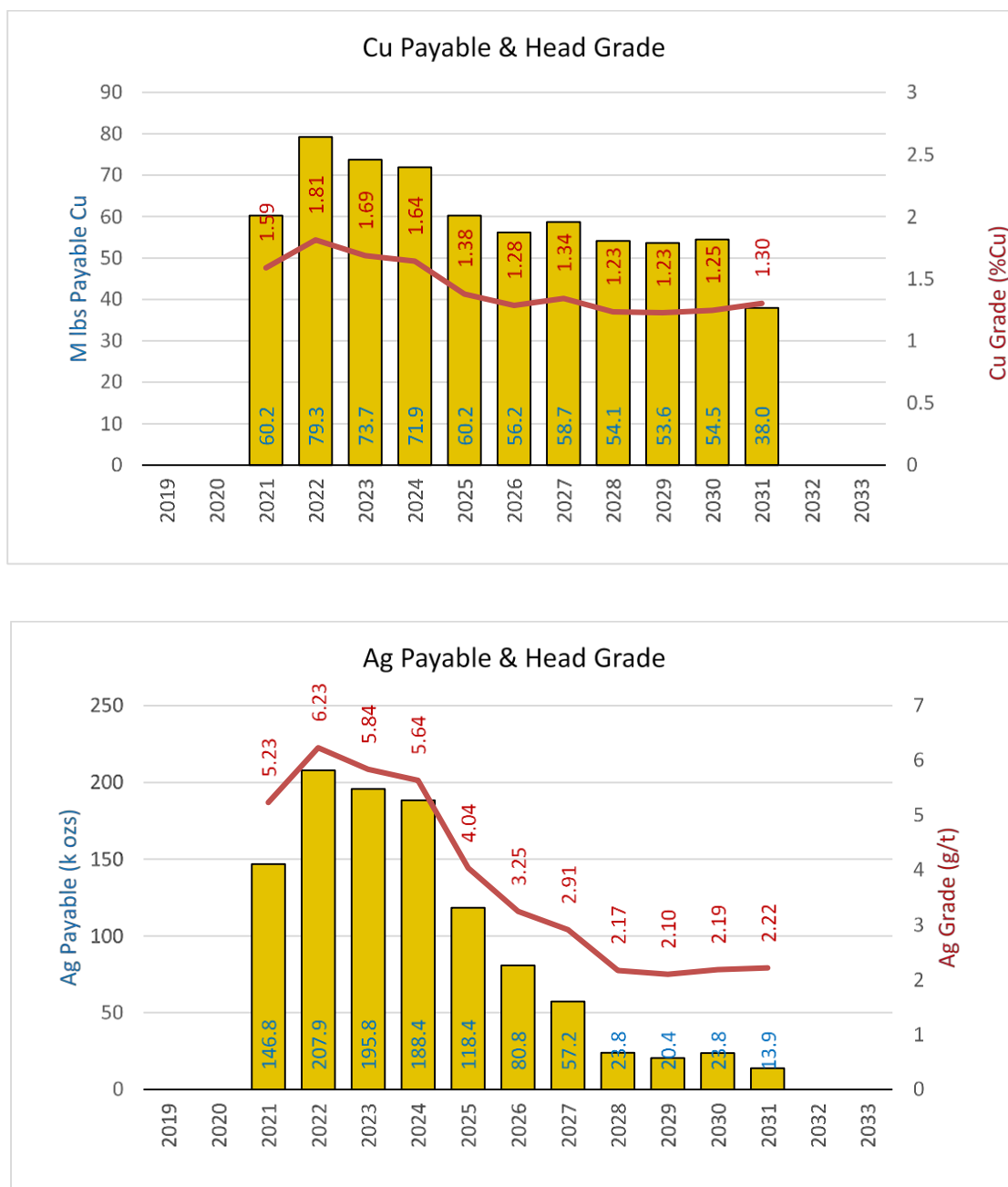
22.2 Metal Production and Revenue

Payable copper produced over the Project life is 300 kt (660 M lb) with an annual average of 28 kt (61.7 M lb) over the 10.7-year life. The average payable copper rate is 95.8% which includes the 0.2% concentrate loss. Payable silver production over the LoM is 1.08 M oz with an annual average of 100 k oz with an average payable rate of 46.9% which is affected by low payable rates in the second half of the mine life when the silver concentrate grade often falls below the minimum payable of 30 g/dmt. The metal production is presented on an annual basis in Table 22.3.

Table 22.3: Metal Production

Production Physicals		Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Tonnage Processed	kt	25,389	2,089	2,409	2,409	2,416	2,409	2,409	2,409	2,416	2,409	2,409	1,606
Cu Head Grade	% Cu	1.43	1.59	1.81	1.69	1.64	1.38	1.28	1.34	1.23	1.23	1.25	1.30
Ag Head Grade	g/t	3.83	5.23	6.23	5.84	5.64	4.04	3.25	2.91	2.17	2.10	2.19	2.22
Concentrate (dry)	dmt	1,264	115.3	151.7	141.1	137.7	115.3	107.5	112.3	103.6	102.7	104.3	72.7
Concentrate (wet)	wmt	1,389	126.7	166.7	155.0	151.3	126.7	118.1	123.4	113.9	112.8	114.6	79.9
Cu Contained Metal	kt	364	33	44	41	40	33	31	32	30	30	30	21
Cu Contained Metal	M lbs	802	73.13	96	90	87	73	68	71	66	65	66	46
Ag Contained Metal	k ozs	3,129	352	483	452	438	313	251	225	169	163	169	114
Cu Recovery	%	86.00	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0
Ag Recovery	%	74.20	74.2	74.2	74.2	74.2	74.2	74.2	74.2	74.2	74.2	74.2	74.2
Cu Metal Production	kt	313	28.5	37.5	34.9	34.1	28.5	26.6	27.8	25.6	25.4	25.8	18.0
Cu Metal Production	M lbs	690	62.9	82.8	77.0	75.1	62.9	58.7	61.3	56.5	56.0	56.9	39.7
Ag Metal Production	k ozs	2,321	261	358	336	325	232	187	167	125	121	126	85
Cu Payable Rate	%	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76	95.76
Ag Payable Rate	%	47.48	57.37	59.14	59.44	59.11	52.09	44.40	35.25	20.10	17.97	19.97	17.42
Cu Payable Metal	kt	300	27.3	35.9	33.4	32.6	27.3	25.5	26.6	24.6	24.3	24.7	17.2
Cu Payable Metal	M lbs	660	60.2	79.3	73.7	71.9	60.2	56.2	58.7	54.1	53.6	54.5	38.0
Ag Payable Metal	k ozs	1,102	149.6	211.7	199.5	191.9	120.9	82.8	59.0	25.1	21.7	25.1	14.8
Operating periods	yrs	10.7	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.7

Note: Concentrate production and payable metal reflects transportation losses, Q1-2021 is pre-production and commissioning

Figure 22.1: LoM Payable Metal Profile


The commissioning and ramp-up schedule is presented in Table 22.4. The commissioning during pre-production is planned over a period of three months where the tonnage ramps-up from 3,000 to 5,400 mtpd in the first quarter of 2021. From Q2-2021 commercial operations are declared with an average milling rate of 6,000 mtpd for six months and then a final increase to steady state throughput of 6,600 mtpd in the last quarter of 2021. The operations period last 10.5 years (when excluding the pre-production period) based on the currently defined mineral reserves being depleted in Q3-2031.

Table 22.4: Mill Commissioning and Ramp-up

Mill Commissioning and Ramp-up		Days	Tonnage (t/month)	Mill Rate (t/d)	% Nameplate
Pre-Prod Month 1	Jan 2021	31	93,000	3,000	45.5%
Pre-Prod Month 2	Feb 2021	28	135,000	4,821	73.1%
Pre-Prod Month 3	Mar 2021	31	167,400	5,400	81.8%
Total Pre-Prod.	Q1-2021	90	395,400	4,393	66.6%
Operations Month 1	Apr 2021	30	180,000	6,000	90.9%
Operations Month 2	May 2021	31	180,000	5,806	88.0%
Operations Month 3	Jun 2021	30	180,000	6,000	90.9%
Operations Month 4	Jul 2021	31	182,000	5,871	89.0%
Operations Month 5	Aug 2021	31	182,000	5,871	89.0%
Operations Month 6	Sep 2021	30	182,000	6,067	91.9%
Operations Month 7	Oct 2021	31	202,400	6,600	100.0%
Operations Month 8	Nov 2021	30	202,400	6,600	100.0%
Operations Month 9	Dec 2021	31	202,400	6,600	100.0%
Total Operations Yr 1	Q2 to Q4 2021	275	1,693,200	6,157	93.3%
Total Yr 1 (Pp + Ops)	2021	365	2,088,600	5,722	86.7%
Total Operations Yr 2	2022	365	2,409,000	6,600	100.0%

Note: Represents 100% of capacity which is 6,000 t/d vs. full nameplate capacity at 6,600 t/d

22.3 Capital Expenditures

The capital expenditures include initial CAPEX as well as sustaining capital to be spent after commencement of commercial operations.

22.3.1 Initial Capital Expenditures

The CAPEX for Project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be US\$275 M. The total initial Project capital includes a contingency of US\$22.9 M which is 9.1% of the total CAPEX before contingency excluding pre-production revenue of US\$30.36 M. Net of pre-production revenue, the initial CAPEX is estimated at US\$244.6 M as presented in Table 22.5. The initial Project CAPEX is spent over a period of 27 months starting in January 2019 and ending in March 2021.

Table 22.5: Initial Capital Expenditure Summary

Initial CAPEX	US\$ k
000 - General	1,150
100 - Infrastructure	36,650
200 - Power & Electrical	5,156
300 - Water & TSF Mgmt.	22,875
400 - Mobile Equipment	27,240
500 - Mine Infrastructure	53,529
600 - Process Plant	45,771
700 - Construction Indirects	27,609
800 - General Services & Owner's Costs	22,251
900 - Pre-Production, Commissioning	9,838
Sub-Total Before Contingency	252,069
Contingency 9.1%	22,899
Total Incl. Contingency	274,968
Less: Pre-Production Revenue	(30,364)
Total Incl. Contingency & Pre-Prod. Revenue	244,604

22.3.2 Sustaining Capital Expenditures

Sustaining capital expenditures during operations are required for additional mine equipment purchases, mine development work, tailings storage expansion for stages 2 and 3, and the water treatment plant. The total LoM sustaining CAPEX is estimated at \$156.5 M with the breakdown presented in Table 1.11.

Table 22.6: Sustaining Capital Expenditure Summary

Sustaining CAPEX	LoM (\$M)	\$/t ore	\$/lb Cu Payable
Tailings Disposal Facility Expansions	28.4	1.14	0.04
Water Treatment Plant	6.1	0.25	0.01
Mine Equipment Purchases	43.7	1.75	0.07
Mine Development Expenditures	78.2	3.13	0.12
Total Sustaining CAPEX	156.5	6.26	0.24

Note: Ore tonnage and payable copper unit costs during operations period only

22.3.3 Closure and Reclamation

The reclamation and closure cost estimate include the following scope:

- Demolition of infrastructures;
- Salvaging of major equipment;
- Site reclamation, principally for the TDF;
- Post closure monitoring.

The closure and reclamation activities are planned over a two-year period from 2032 to 2033 with an overall estimate of US\$28.2 M net of salvage value.

Table 22.7: Closure and Reclamation Cost Estimate by Stage

Closure Cost Estimate	Cost (\$k)
TDF Reclamation	25,566
Site Closure & Reclamation	9,043
General Reclamation	56
Salvage Value	(10,751)
Post Closure Monitoring	2,900
MDEQ Admin Oversight	1,423
Total Cost	28,238

22.3.4 Working Capital

Working capital ("WC") is required to finance supplies in inventory. Given the accessibility of the site, the working capital requirements are considered low compared to remote operations. For concentrate sales an estimate based on 45 days of production was included as receivables which could be longer for overseas export. The WC estimate includes US\$8 M of parts and consumable inventory built-up during the pre-production period.

22.4 Operating Cost Summary

OPEX include mining, processing, G&A services, concentrate transportation and concentrate treatment and refining charges. The concentrate transportation, treatment charges and refining are deducted from gross revenues to calculate the NSR. The NSR for the Project during operations is estimated at US\$1,821 M excluding US\$30.35 M of NSR generating during pre-production and treated as a reduction

of initial capital expenditures. The average NSR over the LoM is US\$2.80/lb of payable copper net of silver credits. Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. The operating costs are detailed in Section 21 of this Report. The average OPEX over the LoM is US\$39.84/t of ore or US\$1.53/lb of payable copper with mining representing 53.4% of the total OPEX, or US\$21.26/t of ore. A summary of operating cash flow and operating costs is presented in Table 1.12.

Table 22.8: Operating Cost & Summary

Operating Cash Flow	LoM (US\$ M)	US\$/t ore	US\$/lb Cu Payable
Cu Revenue	2,047	81.92	3.15
Ag Credits	17	0.69	0.03
Revenue	2,065	82.61	3.17
Concentrate Transportation Costs	94	3.75	0.14
Treatment & Refining Charges	149	5.98	0.23
Net Smelter Return	1,821	72.88	2.80
Royalties	85	3.39	0.13
Mining Costs	531	21.26	0.82
Processing Costs	308	12.31	0.47
G&A Costs	72	2.88	0.11
Working Capital	0	0.00	0.00
Total OPEX (including royalties)	996	39.84	1.53
Operating Cash Flow	826	33.03	1.27

Note: Ore tonnage and payable copper unit costs during operations period only

Table 22.9: Life-of-Mine C1 & C3 Cost Summary

LoM Costs	Total Cost (US\$M)	Unit Cost \$/tonne milled)	Unit Cost (\$/payable lb)
Mining	531	21.26	0.82
Processing	308	12.31	0.47
G&A	72	2.88	0.11
Offsite Costs (transport, TC/RCs)	243	9.72	0.37
By-product credits	(17)	(0.67)	(0.03)
C1 Cost	1,137	45.50	1.75
Depreciation and Closure	429	17.18	0.66
Royalty Costs	85	3.39	0.13
C3 Cost	1,651	66.06	2.54

Table 22.10: First 5-Year C1 & C3 Cost Summary

First 5-Year Costs	Total Cost (M \$)	Unit Cost (\$/tonne milled)	Unit Cost (\$/payable lb)
Mining	219	19.30	0.65
Processing	140	12.34	0.42
G&A	29	2.52	0.09
Offsite Costs (transport, TC/RCs)	130	11.45	0.39
By-product credits	(13)	(1.18)	(0.04)
C1 Cost	504	44.43	1.50
Depreciation and Closure	159	14.00	0.47
Royalty Costs	45	3.95	0.13
C3 Cost	707	62.38	2.11

22.5 Taxes and Royalties

22.5.1 Income Tax

Income for tax purposes is defined as metal revenues minus operating expenses, royalties, Michigan severance tax, reclamation and closure expenses, depreciation and depletion. Depreciation is calculated using the Modified Accelerated Cost Recovery System (“MACRS”) method and the unit of production method in accordance with the current U.S. Internal Revenue Service (“IRS”) regulations. The federal income tax rate based on new tax reform is 21%. There is no state income tax which is exempt under the Michigan Nonferrous Metallic Minerals Extraction Severance Tax Act. The estimated federal tax paid over the Project life is US\$32.5 M.

22.5.2 Michigan Severance Tax

The Nonferrous Metallic Minerals Extraction Severance Tax Act (“MST”), PA 410 of 2012, as amended, levies a specific tax on certain nonferrous metallic minerals for mineral producing properties in the state of Michigan. The tax levied on the eligible mine owner is the Minerals Severance Tax and includes exemption from property taxes levied in this state, taxes levied under part 2 of the Income Tax Act, PA 281 of 1967, Sales tax as levied under PA 167 of 1933, and Use tax as levied under PA 94 of 1937.

The minerals Severance Tax is 2.75% of gross income from mining or the net smelter return, less third-party royalty payments. Over the LoM, the Severance Tax represents US\$47.8 M.

22.5.3 Royalties

The owners of the mineral rights (Sage, KLA and Chesbrough) are entitled to a sliding scale royalty ranging from 2% to 4% NSR between a copper price of US\$2.00/lb and US\$4.00/lb. At the base case price of US\$3.10/lb the royalty rate is 3.1% NSR. Lease payments are deductible from the royalty payments. Over the LoM, this royalty represents a cost of US\$58.4 M.

Under a transaction with Osisko Gold Royalties, Osisko is to receive a 1.5% NSR royalty which is fixed regardless of copper price. The 1.5% royalty rate assumes that the White Pine transaction is finalized prior to beginning of commercial production as otherwise the rate would be 3.0%. Over the LoM, the Osisko royalty represents a cost of US\$27.8 M.

22.6 Economic Model Results

The economic model results are presented in terms of NPV, IRR, and payback period in years for recovery of the initial CAPEX. These economic indicators are presented on both pre-tax and after-tax basis. The NPV is presented both undiscounted (NPV_{0%}) and using a discount rate of 8% (NPV_{8%}). The annual cash flow is summarized in Table 22.12 and graphically in Figure 1.4. A cash flow waterfall for the Project is summarized in Figure 22.3.

The undiscounted after-tax cash flow is estimated at US\$316 M for the Project. The economic results on a before-tax and after-tax basis are presented in Table 22.11.

Table 22.11: Economic Results Summary

Economic Results Summary	Unit	Before-Tax Results	After-Tax Results
NPV 0%	\$M	396.3	316.0
NPV 8%	\$M	162.1	116.8
IRR	%	21.1%	18.0%
Payback	yrs	2.9	3.2

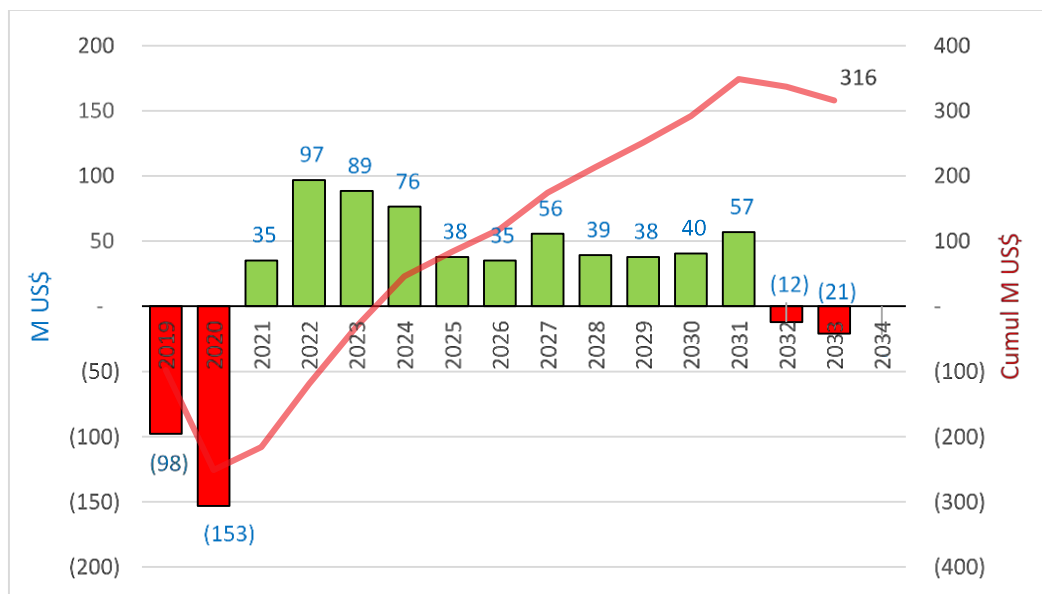
Figure 22.2: After-Tax Annual Project Cash Flow

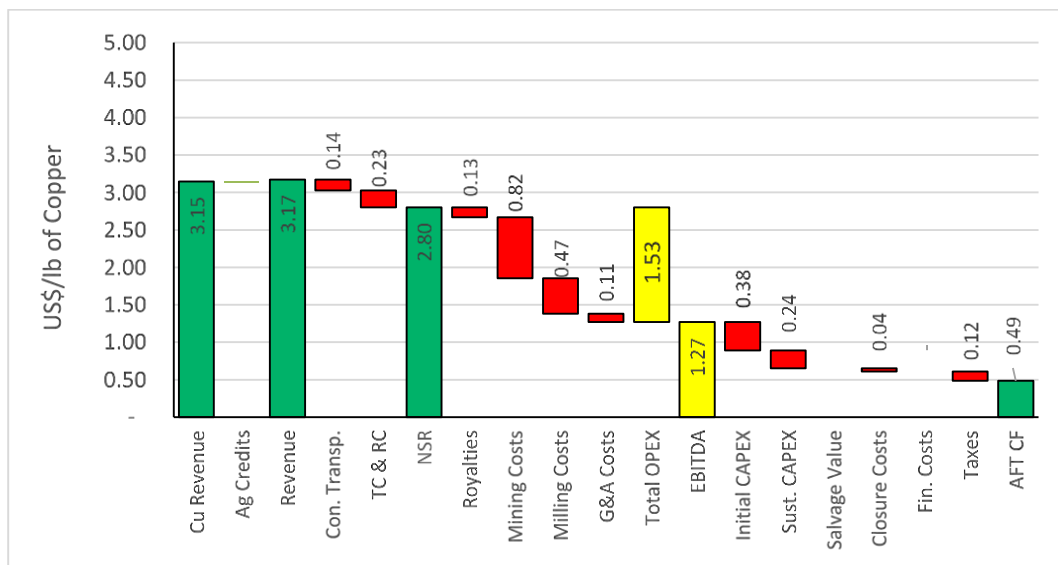
Figure 22.3: After Tax Project Cash Flow Waterfall

Table 22.12: After-Tax Annual Cash Flow Summary

Cash Flow (\$M)	Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Revenue (Cu + Ag)	2,065	-	-	173	261	235	226	189	175	183	168	167	169	118	-	-
Con. Transp. Costs	(94)	-	-	(7)	(11)	(11)	(10)	(9)	(8)	(8)	(8)	(8)	(8)	(5)	-	-
TC / RCs	(149)	-	-	(12)	(19)	(18)	(18)	(14)	(13)	(13)	(11)	(11)	(12)	(8)	-	-
Net Smelter Return	1,821	-	-	154	230	207	198	166	155	162	149	148	150	105	-	-
Royalties	(85)	-	-	(8)	(11)	(10)	(9)	(8)	(7)	(7)	(7)	(7)	(7)	(5)	-	-
Mining Costs	(531)	-	-	(23)	(47)	(50)	(50)	(48)	(51)	(55)	(57)	(56)	(57)	(36)	-	-
Processing Costs	(308)	-	-	(21)	(30)	(30)	(30)	(30)	(30)	(30)	(30)	(30)	(29)	(20)	-	-
G&A Costs	(72)	-	-	(4.51)	(6.01)	(6.01)	(6.01)	(6.01)	(7.36)	(7.81)	(7.81)	(7.81)	(7.59)	(5.07)	-	-
Total Operating Costs	(996)	-	-	(56)	(94)	(95)	(95)	(92)	(95)	(100)	(101)	(101)	(101)	(66)	-	-
Working Capital	(0)	(4)	-	(31)	(1)	3	2	6	(2)	2	1	(0)	(1)	25	-	-
Operating Cash Flow	826	(4)	-	66	135	115	104	80	57	63	49	47	48	64	-	-
Initial CAPEX	(245)	(94)	(153)	3	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	(156)	-	-	(34)	(34)	(17)	(15)	(31)	(15)	(1)	(3)	(3)	(2)	(0)	-	-
Closure & SV	(28)	-	-	-	-	-	-	-	-	-	-	-	-	-	(7)	(21)
Taxes	(80)	-	-	-	(4)	(9)	(13)	(11)	(8)	(6)	(7)	(6)	(6)	(6)	(5)	-
Project AFT Cash Flow	316	(98)	(153)	35	97	88	76	38	35	56	39	38	40	57	(12)	(21)
Cumul. AFT Cash Flow		(98)	(251)	(216)	(119)	(31)	46	84	119	174	214	252	292	349	337	316

Notes:

- Pre-production revenue included in investment capital offsetting pre-production costs.
- Taxes include federal income tax and Michigan Severance Tax.

22.7 Sensitivity Analysis

The sensitivity analysis of the economic model was tested with respect to metal prices, initial CAPEX and OPEX for each case. The value of each parameters was raised and lowered 20% to evaluate the impact of such changes on the NPV and IRR. The pre-tax sensitivity results are presented in Table 22.13 and the after-tax sensitivity results in Table 22.14.

The after-tax NPV of the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices or metal grades. For example, a 20% increase in copper price or copper grade increases the NPV_{8%} from US\$116.8 M to US\$318.8 M. Similarly, a decrease of 20% in copper price or copper grade reduces the NPV_{8%} to -US\$89.2 M.

Table 22.13: Pre-Tax Sensitivity Results

Variance	Before-Tax Results			
	NPV _{0%} (M\$)	NPV _{8%} (M\$)	IRR (%)	Payback (yrs)
Metal Price Sensitivities				
20%	781.9	389.5	35.4%	2.0
10%	589.6	276.1	28.6%	2.3
0%	396.3	162.1	21.1%	2.9
-10%	201.6	47.2	12.3%	3.9
-20%	5.5	-68.4	0.4%	9.9
Initial Capital Cost Sensitivities				
20%	341.3	112.5	15.9%	3.6
10%	368.8	137.3	18.3%	3.2
0%	396.3	162.1	21.1%	2.9
-10%	423.8	186.8	24.3%	2.6
-20%	451.3	211.6	28.2%	2.3
Operating Cost Sensitivities				
20%	212.9	58.5	13.5%	3.6
10%	304.6	110.3	17.5%	3.2
0%	396.3	162.1	21.1%	2.9
-10%	487.9	213.9	24.4%	2.7
-20%	579.6	265.6	27.4%	2.5

Table 22.14: After-Tax Sensitivity Results

Variance	After-Tax Results			
	NPV _{0%} (M\$)	NPV _{8%} (M\$)	IRR (%)	Payback (yrs)
Metal Price Sensitivities				
20%	655.1	318.8	31.9%	2.1
10%	486.1	218.1	25.3%	2.5
0%	316.0	116.8	18.0%	3.2
-10%	145.6	15.4	9.5%	5.2
-20%	-31.8	-89.2	0.0%	10.5
Initial Capital Cost Sensitivities				
20%	266.1	70.2	13.2%	3.9
10%	290.8	93.3	15.4%	3.5
0%	316.0	116.8	18.0%	3.2
-10%	341.4	140.4	21.1%	2.8
-20%	366.8	164.0	24.7%	2.5
Operating Cost Sensitivities				
20%	150.7	22.8	10.3%	4.2
10%	233.5	69.8	14.4%	3.6
0%	316.0	116.8	18.0%	3.2
-10%	398.6	163.9	21.3%	2.9
-20%	481.2	210.9	24.3%	2.6

Figure 22.4: After-Tax NPV8% Sensitivity

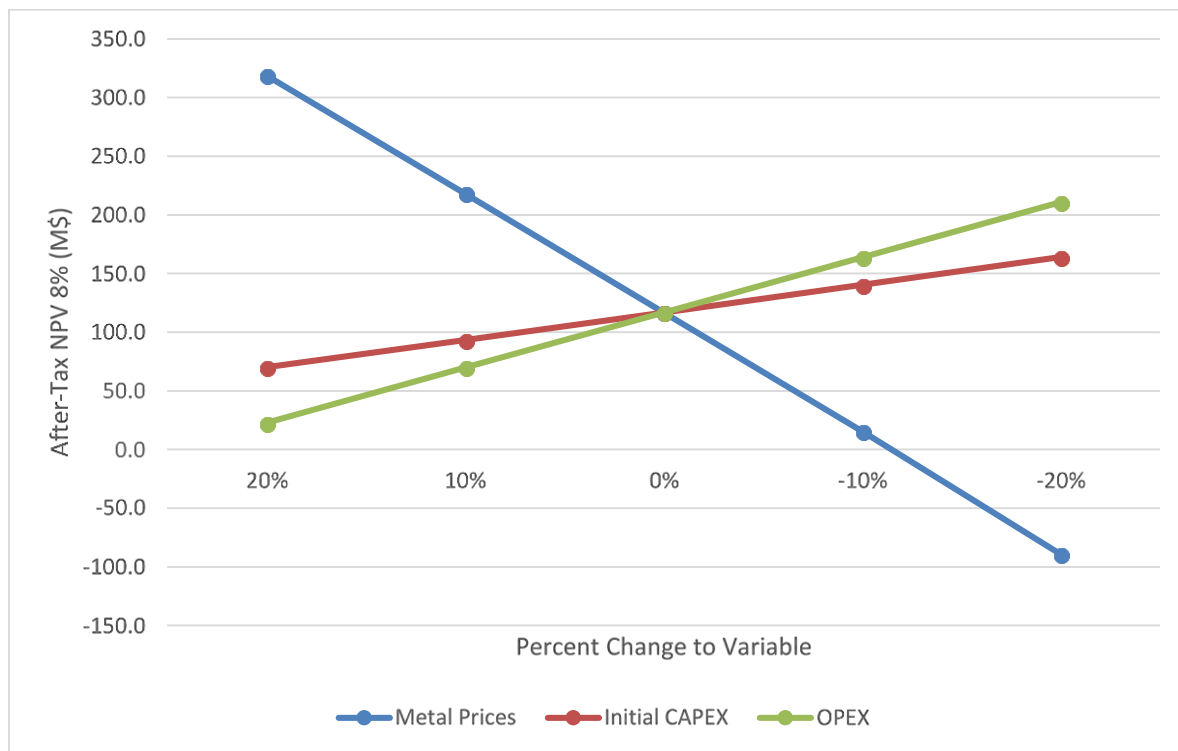
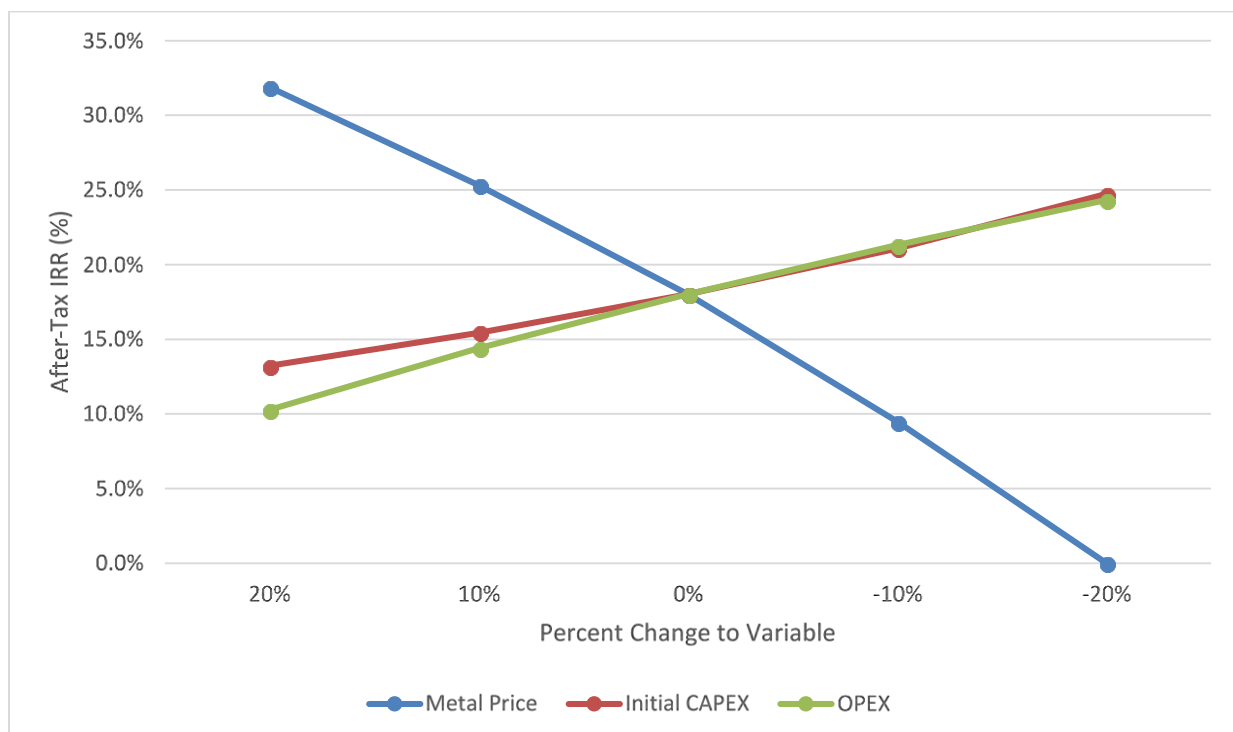


Figure 22.5: After-Tax IRR Sensitivity



23. ADJACENT PROPERTIES

There are no other mineral exploration or development projects adjacent to the Copperwood Project area.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

The Copperwood Feasibility Study has been completed under the assumption that the execution strategy will incorporate some aspects of an “owner managed” style as well as some aspects of an EPCM style. This will result in a mixed management team with both Highland and contracted personnel throughout the construction phase. Highland mandated GMSI to complete the Feasibility Study in order to leverage its experience in implementing similar execution strategies. GMSI teams specializing in mining, engineering, and construction have developed other major projects like Rosebel, Essakane, Iamgold located in Burkina Faso, and, most recently, the Newmont Merian Project in Suriname.

The Project team will manage and execute the project engineering, procure project items, execute project construction, provide project control, staff for start-up and operation, and commission mine and process areas. Certain operation departments will be integrated in the project team early in the process to allow their parallel development and will focus on the project readiness.

Due to the site’s location and relative proximity to qualified contracting operations, most of the on-site labor services in the construction phase will be provided by third party contractors. Third-party hiring shifts a portion of the risks to the contractors in exchange for the party’s markup.

As part of the early works, the first construction phase will involve clearing and grubbing, excavation of the box cut, road development, first Tailings Disposal Facility (“TDF”) phase, and temporary power access. Reputable third-party consultants and engineers will be used for design and QA/QC work to reduce the risks to Copperwood’s critical and specialized components such as the power line engineering and easements process, long lead items for the process plant, TDF, water treatment, and environmental issues. Specifications established by these firms will be approved by the team in charge of each task.

The overall result of this style of management will be the placement of experienced and skilled personnel in their respectable positions which will result in overseeing a qualified workforce and taking advantage of the developed industries near the site.

24.2 Project Development Organization

The Project implementation team is composed of a construction-engineering group, led by a temporary organization, and an operations group. The operations group consists of a mining group, mill group and general services group which are created earlier to support execution activities or to begin pre-production

planning activities. The engineering, construction and operations groups report to the Project director and work together on reducing capital costs through the planning of equipment use, staff and employees, Project commissioning, and Project start-up.

The Mine Group includes the Operations, Maintenance, Engineering and Geology departments. The Mine Group will receive its equipment to start the mining activities by Q2 2019 and will progressively build its initial fleet.

The Mill Group personnel initially reviews and contributes to the mill detailed engineering and procurement activities performed by the process engineer. The group will subsequently expand to monitor the mill construction activities, particularly piping and mechanical installation, electrical installation, instrumentation and process control. It will also be involved in recruiting and training the mill workforce and prepare the inventory of parts and supplies ahead of production. Finally, the mill department will participate in the mill commissioning activities and take responsibility for the process plant with the start of ore commissioning.

The General Services Group is established early in the project development phase, and initially provides services to the Mine Group and the Engineering and Construction Group. It is logical to progressively organize and activate the General Services Group as part of the Project and therefore avoid duplication and dislocation when moving into the production phase. These services include general administration, finance and accounting, supply chain, human resources and training, security, social and environmental management, transportation, camp management, health and safety, surface support and IT as well as communications. Some external contributors / contractors will also provide support, such as freight forwarders. The General Services Group will recruit heavily in the local labor pool and will be an important service provider to the mine and construction activities.

The Engineering and Construction Group has the overall responsibility of the engineering and management, equipment and construction material procurement and construction of the processing facilities, main camp, administration and mine offices, mine workshop and warehouse, and temporary construction power.

All capital equipment and construction material will be tendered by the Engineering and Construction group with the technical assistance of the Process Plant Engineer. Purchase orders will be issued by the Copperwood project team for or on behalf of Highland, the Project's owner. Specifications are established by the various engineering firms retained for the Project and accepted by the Owner's representatives.

Certain specialized construction mandates and services will also be tendered, such as grinding mill installation supervision. These activities will be controlled through contracts. These contracts will be based on the documents approved by HCC, with addenda specific to the Copperwood Project.

Once the Project has been commissioned, the Project Director will turn the project completely over to the Highland/Copperwood General Manager of operations who will assume responsibility of the Mine, Mill and General Services Group and the Construction Group will be phased out as the remaining construction activities are completed. The Project Director will remain on the Project assisting the Operations team until the name plate throughput for the process plant is achieved.

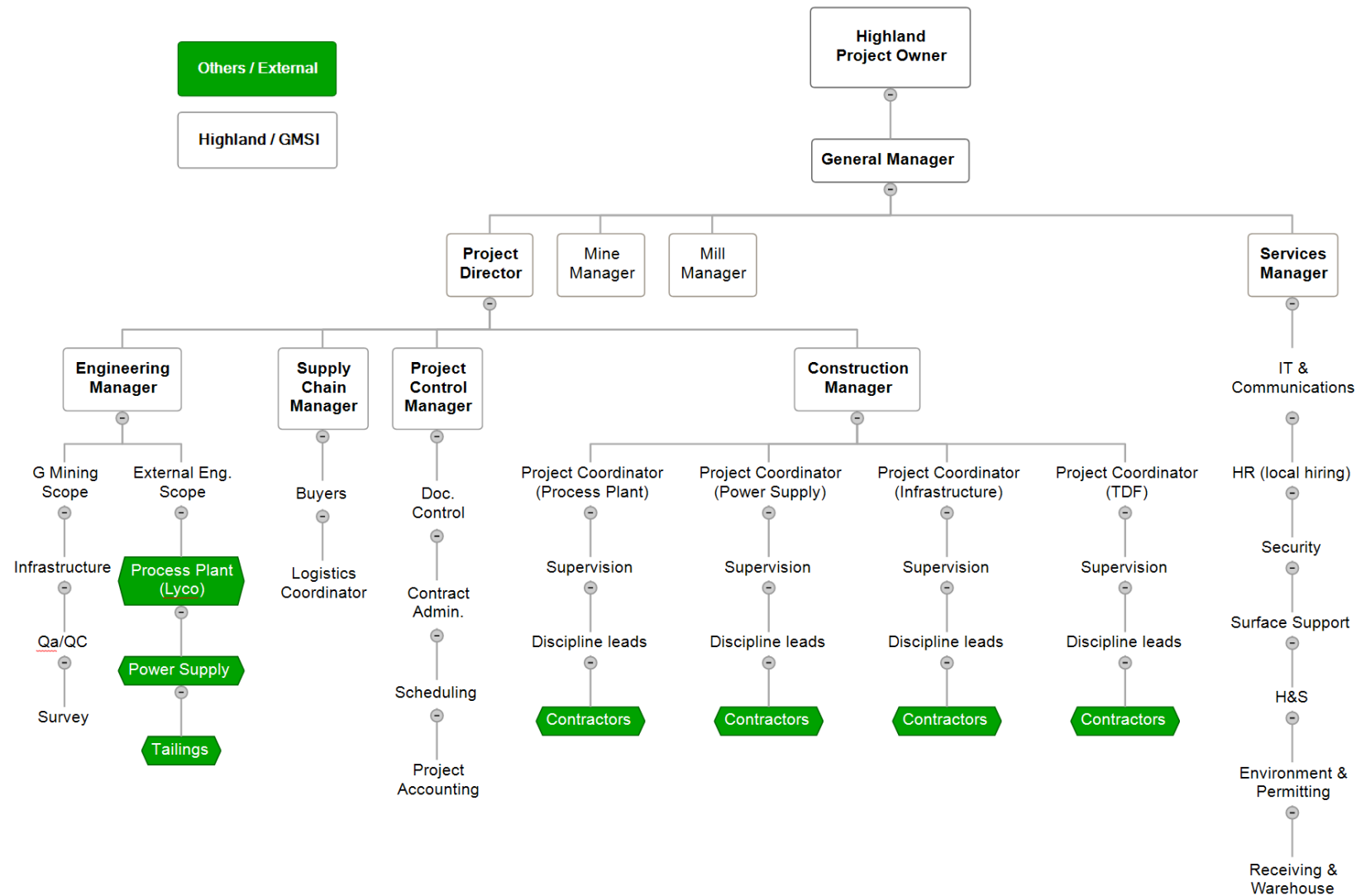
The ultimate project authority lies with Highland. The Project Director shares responsibility for all steps in the process required to reach commercial production.

The engineering and construction group is involved in the development of the detailed engineering, procurement, and construction of all items in relation to on-site and off-site infrastructure including; buildings, site preparation, roads, buildings and TDF construction.

The utility company will be responsible of the engineering, easements negotiations and agreements, construction and commissioning of the power line from Norrie, the tie-ins in Norrie substation and the Copperwood site main substation. The engineering and construction group will support and coordinate the utility's engineering and construction team.

In general, the mixed team for engineering and construction can be broken down into five categories; engineering firm employees on site, engineering firm employees at head office, staff employed by Highland on site, contractors and consultants, and Highland employees. Principal organizational relationships are outlined in Figure 24.1

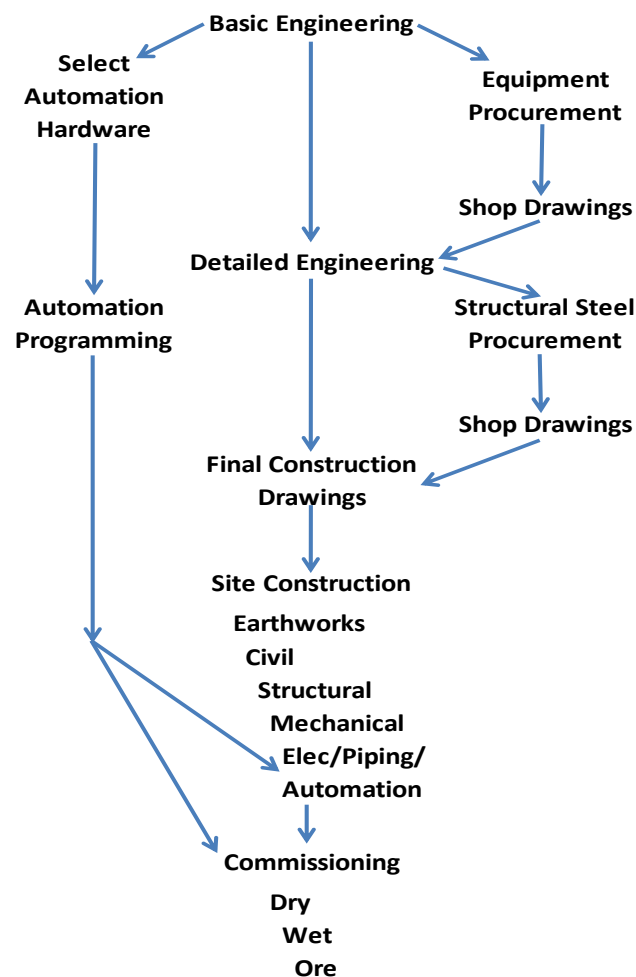
Figure 24.1: Project Team Organization for Engineering and Construction Phases



24.3 Project Engineering

The plan is to use a fast track engineering process to achieve the planned completion date. This process is shown in Figure 24.2. Some project activities will happen in parallel to reduce the overall timeline. Detailed engineering will be prioritized to fit the staged schedule leading with the early works prior to the delivery of major equipment. In general, the order of construction will follow the order of earthworks, civil, structural, mechanical, and electrical/piping/automation, respectively. Drawings will be completed as far in advance as possible, with others being delivered as-needed.

Figure 24.2: Engineering Process



Some areas of engineering will be subcontracted to specialized firms. The actual plan is to subcontract these engineering areas as follows:

- TDF by specialized firm;
- Powerline and main substation by utility company;
- Process plant by process engineering firm;
- Lake Superior water intake and main access road by local engineering firm;
- Road 519 upgrade by MDOT;

All other infrastructure engineering will be performed by engineering firm at their head office.

Basic engineering is finished with the completion of principal trade-offs, plant site alternatives, TDF options, and process specifications. The process flowsheets have been finalized and much of the design criteria and equipment specifications are complete. However, detailed engineering will need to commence earlier. This phase is called “Early Works”.

24.4 Early Works

This Study has identified activities to be developed as early as possible to meet the Copperwood Project schedule.

The mining group needs to be mobilized to begin ore extraction as soon as possible. This activity is clearly in the critical path of the Project. Therefore, mining engineering will confirm the mine design and define mining and support to mining equipment. Contracts will have to be tendered for this equipment. The engineering and the construction for the box-cut, mine portal and related infrastructure such as ore stockpile and contact water management need to be in place as the extraction shall begin.

Some detailed engineering activities for the process plant need to start establishing the parameters and datasheets for all long-term delivery process equipment such as grinding mills. The detailed engineering of the process plant building needs to start earlier since the present schedule requires the completion of this building envelop in 2019.

Engineering by the utility company will also be part of the early works. A schedule has been provided by the utility company and engineering is requested in 2018.

Some of these contracts will be awarded following Highland’s approval of partial or full funds for the Project.

24.5 Detailed Engineering/Procurement

The Copperwood site has been mapped using LIDAR. The general arrangement uses the topographic information from the LIDAR in the form of UTM coordinates to locate infrastructure.

Multiple soil geotechnical studies have been conducted on the Copperwood site. The two most recent studies analyzed the soil underneath the foundation locations in the process area and the soil underneath the tailings facility. Consistent values for permeability, soil type, strata thickness, density, and bearing capacity have been found throughout the site. The soil analyses were performed by Coleman Engineering and Soils Engineering Testing. These studies provide the required information for detailed engineering for foundations, the TDF as well as excavations. The geotechnical investigations of foundations of infrastructure were performed by Dr. Stanley Vitton, PhD, PE and Associate Professor at Michigan Tech University. Dr. Vitton coordinated the site investigations, laboratory tests and wrote the final report.

Rock geotechnical studies have been conducted for metallurgical work as well as pillar design. The metallurgical analysis was performed by SGS to determine the physical properties of the rock

Hydrogeological and hydrological studies were conducted by AECOM for Orvana to determine water inflows to mine infrastructure. Results of both studies are discussed in:

- Hydrology;
- Geochemistry.

Site visits have been conducted to determine availability of nearby resources in categories including; skilled labor, aggregate, concrete, and building supplies.

24.6 Capital Procurement and Logistics

The Engineering and Construction Group will purchase all equipment and import materials required for construction which permits direct control over the procurement budget and schedule. The team follows a standard bidding and evaluation process taking into account the total delivered cost including freight to site to obtain the lowest total cost or best value (considering operating cost and life cycle considerations). The team coordinates logistics and assists suppliers in complying with project freight and transportation procedures. Freight forwarding is managed dynamically to minimize the freight transit times and maximize consolidation of shipments.

Low cost country sourcing ("LCCS") will be considered for low risk construction material and equipment, where practical, to improve the Project capital expenditures.

In order to minimize the duration of project construction, engineering and construction are performed on an owner managed, fast-track basis. The fast-track strategy compresses the project schedule by running design and construction phases in close sequence.

The construction team will consist of experienced construction project managers and superintendents working with a skilled team of disciplined engineers and construction supervisors. In some instances, the team will have the benefit of having participated in the detailed engineering phase of the Project. The Project is managed with a focus on safety, cost, schedule and quality to support the overall project goals.

The construction work teams will incorporate experienced expatriate and national trade supervisors working with predominantly national construction laborers. Skills improvement and safety will be emphasized throughout the construction period; when possible, the best available candidates can transfer to the operations team on conclusion of the construction period. This results in a large contribution of national workers to the overall construction effort, with significant skills development and opportunities for continued employment in the operating phase.

Materials and equipment are purchased by Highland and issued to construction teams in a timely manner. The Owner's team manages all risks and opportunities related to material supply, construction labor organization and prioritization of tasks. This provides full flexibility to organize work in such a way that the best available labor resources are allocated to the most urgent or complex construction tasks. Underperforming staff or contractors can be reassigned to less critical work or terminated.

Quality Assurance/Quality Control ("QA/QC") of earthworks and concrete is performed by a suitably accredited third-party engineering firm, and all QA/QC documentation is posted to the Aconex data management system for archival and review purposes.

Structural steel is procured from qualified suppliers, most likely from sources in North America, and shipped directly to site. Structural steel is procured as material and erect.

Mechanical / electrical and automation equipment is supervised by the Owner's construction discipline teams. Specialized contractors will be retained for all installation by discipline. Some critical equipment may require specialized supervision such as the grinding mills or other complex or high value equipment. Vendors' representatives are expected for QA/QC on the assembly and installation of critical machinery, and for commissioning assistance or warranty inspections for critical and high value machinery.

All automation equipment is pre-programmed in the Montreal project office instrumentation laboratory by the automation staff and pre-tested. Once installed, automation equipment requires only routine de-bugging and last-minute modifications which greatly reduces commissioning delays. In general, there is no need to perform any automation programming during startup and commissioning since it has all been completed in the lab months prior to startup.

Preferred piping materials are high density polyethylene (“HDPE”) except when corrosiveness, operating pressures or other specific design requirements preclude its use. Generally, HDPE pipe for mineral processing plants have the best abrasion resistance and life cycle cost benefits compared to steel or other materials. There will also be substantial use of rubber lined carbon steel pipe where HDPE cannot be used. The majority of pipe connections will be made using Victaulic connections – which facilitates site fabrication, reduces on site labor requirements, and greatly simplifies maintenance once in production. Equipment will generally be procured with Victaulic flanges for commonality. Piping is fabricated and installed using local hires managed by the construction piping staff.

24.7 Project Controls

The Project will be managed and controlled with the assistance of an earned-value project control methodology. The following software tools are used to support the project execution:

- Aconex is a data management service that provides the sharing of all relevant project data and information, such as drawings and specifications, with all project stakeholders – the Owner and Owner’s project development team, engineers, consultants, suppliers, auditors, insurers, and construction contractors. Aconex is also used to manage all documentation related to procurement – bid documents, proposals, technical documentation and manuals. Access to the Aconex services is managed with a flexible system of access controls and protocols, such that each project stakeholder can only access or upload data pertaining to their scope of project involvement.
- The project scheduling software is Primavera P6. It will integrate the Enterprise Resources Planning (“ERP”) chosen by Copperwood, in order to perform standard budget variance and earned value progress reports.

24.8 Quality and Design Standards

The Copperwood Project component’s detailed engineering will be designed based on the relevant Michigan design codes and standards using qualified and proven manufacturers.

Health and safety standards will comply with all relevant OSHA and MIOSHA regulations and also conform to Highland's requirements.

Tailings and water management dams and dykes will be designed with conservative design factors.

24.9 Quality Management

QA/QC of all construction activities is performed by a suitably accredited third-party engineering firm under the direct supervision of the Resident Engineer of the Project. All QA/QC documentation is posted to the Aconex data management system for archival and review purposes.

QA/QC of welding for critical structures (e.g.: fuel tanks) will also be performed by a suitably accredited inspection firm. All QA/QC documentation is posted to the Aconex data management system for archival and review purposes.

The process equipment will be subject to vendor verifications and factory acceptance testing programs included in the Project procurement plan with consideration to an overall process equipment risk analysis.

24.10 Commissioning

As project areas are mechanically completed, commissioning activities begin immediately. There are three basic stages of commissioning checks – dry, wet and ore commissioning. Dry commissioning checks verify the correct installation of equipment, and the proper connections to all interfaces – electrical, instrumentation, and piping. Wet commissioning verifies the integrity of tankages and piping connections as well as proper equipment functionality. Ore commissioning is a final verification of the process in stages, beginning with ore receipt followed by grinding.

Commissioning checklists are continually updated and uploaded to Aconex by the site commissioning team as commissioning progresses. Commissioning of high value or complex process equipment is supported by vendor representatives who will also provide specialized operations and maintenance training to the operations staff.

The automation team is on-site as process equipment installation begins, with the entire plant automation system having been pre-assembled and bench tested, (which significantly reduces the commissioning time). As equipment is installed, input/output interfaces are verified, controls are tested, and automation drawings are updated to as-built-drawings.

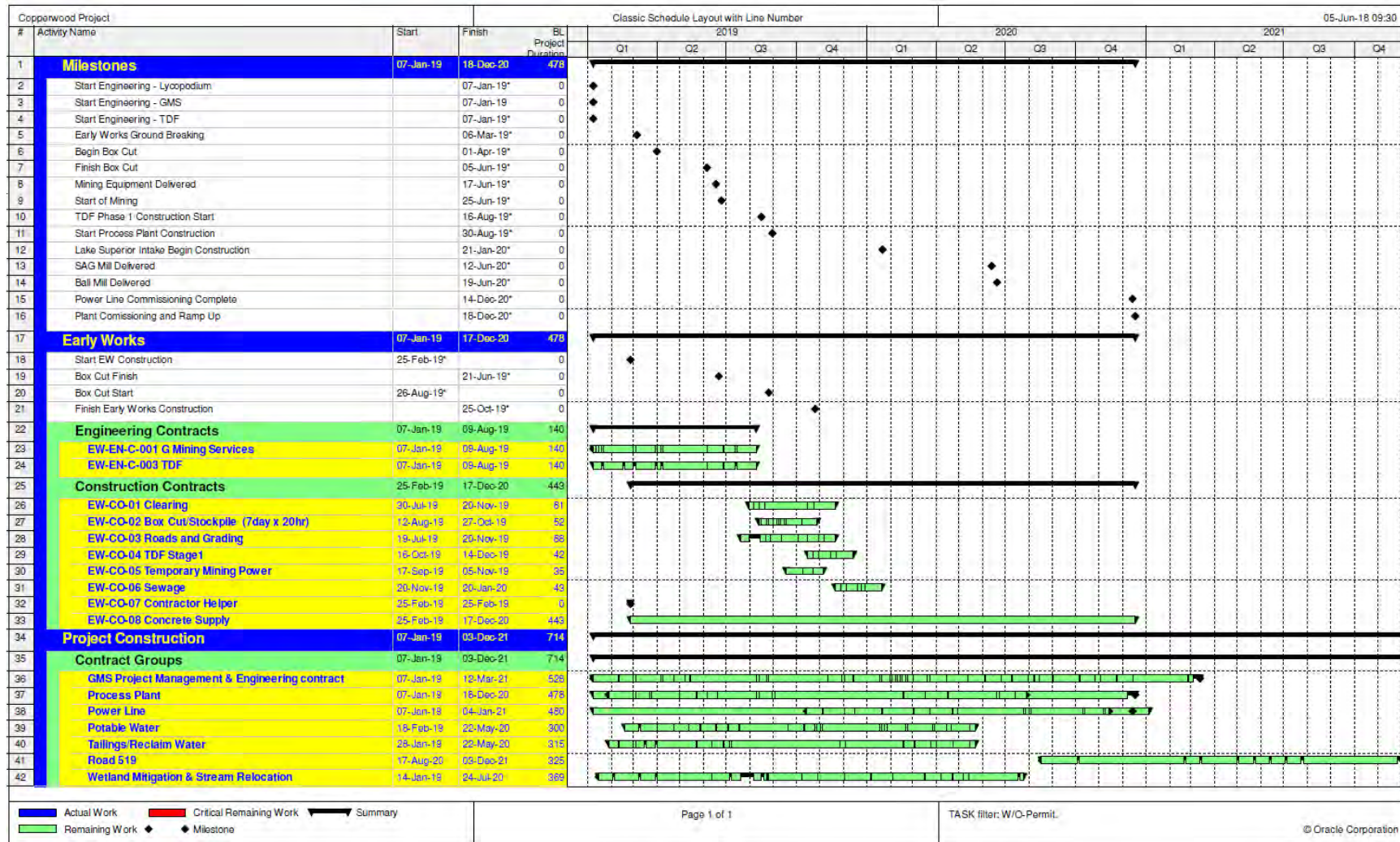
Equipment technical documentation and checklists are available on Aconex for the entire plant at the end of commissioning, so the operation team has all necessary information to ensure a smooth transition from construction to operations.

24.11 Project Schedule

The construction and pre-production development schedule leading to commercial production is 25 months, consisting of two months for initial mobilization of key personnel and equipment and 21 months of on-site construction activities from the start of mining. The project Level 1 schedule is summarized in Figure 24.3.

Highland notes that the timeline of activities described in this Report and completion of such activities is subject at all times to matters that are not within the exclusive control of Highland. These factors include the ability to obtain, on terms applicable to Highland, financing and required permits.

Figure 24.3: Project Level 1 Schedule



25. INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

25.1.1 Geology and Mineral Resources

GMSI has prepared a mineral resource estimate update for the Copperwood Project based on the original drilling database used for the October 2017 Mineral Resource upgrade, with additional drilling data collected in 2018. The resource estimate was prepared in accordance with CIM Standards on Mineral Resources and Reserves (adopted May 10, 2014) and is reported in accordance with the NI 43-101. The mineral estimate was prepared under the supervision of Mr. Réjean Sirois, Eng. GMSI, Vice President, Geology and Resources, an independent QP. Geovia GEMS™ and Leapfrog GEO™ software was used to facilitate the resource estimation process.

The main conclusions of the Mineral Resource estimate of the Copperwood Project are as follow:

- GMSI conducted meetings on the Copperwood Project in 2014, 2015 and 2017, and has reviewed the available data used in the Mineral Resource estimate, including drill logs, assay certificates, downhole surveys and additional supporting information sources. GMSI concludes that the drill hole database could be used with confidence in the Mineral Resource estimate;
- The Mineral Resource estimate is based on a database derived from 366 diamond drill holes (with 14 additional wedges) totaling 70,105 meters, drilled by four companies between 1956 and 2018;
- The resources were estimated for each unit of the LCBS (Domino, Red Massive and Grey Laminated), and the UCBS was modelled as a single unit with a minimum thickness of 2.2 m;
- The statistical analysis of the copper and silver assays revealed that the use of grade capping was not necessary;
- The uncapped raw assays were composited to produce a single composite per unit, per drill hole. The statistical analysis of the copper and silver composites revealed that the use of grade capping was not necessary;
- The variography study based on the zone composites highlighted a near horizontally isotropic distribution of copper and a low nugget effect on copper and silver grades. The semi-variogram models indicated ranges of between 350 m and 500 m, corresponding to the maximum distance of grade continuity between pairs of composites;

- The block size dimension (20 m X 20 m X 2.5 m) was based on the drilling pattern, the anticipated room-and-pillar mining scenario, the complexity of modelling each geological unit and the minimum mining height of 2.0 m;
- The resources were interpolated using the Ordinary Kriging method. Three cumulative passes defined by different degrees of confidence in geological and grade continuity were utilized for block-grade estimation;
- The resources were classified in Measured, Indicated and Inferred Mineral Resources, mostly based on the interpolation passes, but also by delineating groups of blocks of similar interpolation pass;
- The model was validated using many global and local validation methods, including descriptive statistics, swath plots, Q:Q plots and visual methods;
- The grade-tonnage curves for the Measured and Indicated Resources of the Copperwood Deposit do not show a significant degree of sensitivity to cut-off grades, unlike the Satellite Deposits, which tend to show a rapid increase in copper content with decreasing cut-offs grades (between 0.8% and 1.0% Cu);
- An underground room-and-pillar mining scenario is judged to be the most adapted to the geometry and dip of the LCBS, as well as to the tonnage of the deposits;
- The following conceptual mining parameters were used to calculate block values: 1) A NSR sliding scale royalty equivalent to 3% at US\$3.00/lb, 2) No mining loss/dilution, 3) Copper price of US\$3.00/lb and a silver price of US\$18/oz, 4) Recovery of 86% for copper and 73.5% for silver, 5) A payable rate of 96.5% for copper and 90% for silver, 6) A cut-off grade of 1.0% Cu and 7) Operating costs based on an operating plant at Copperwood;
- Copperwood Deposit total underground Measured & Indicated Mineral Resources are reported at 49.3 M tonnes grading an average 1.54% Cu and 3.76 g/t silver containing 1.68 billion pounds of copper and 5.9 M oz of silver using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined. Inferred Mineral Resources are reported at 1.6 M tonnes grading an average 1.18% copper and 1.55 g/t silver containing 43 M pounds of copper and 0.1 M ounces of silver using a cut-off grade of 1.0% Cu;
- The Satellite Deposits total underground R&P Inferred Mineral Resources are reported at 49.9 Mt grading 1.15% Cu and 3.42 g/t silver containing 1.27 billion pounds of copper and 5.5 M ounces of silver using a cut-off grade of 1.0% Cu for the LCBS and UCBS combined;
- The changes observed between the October 2017 and the 2018 Mineral Resources in this Technical Report can be attributed to the upgrade of Section 5 into the Indicated category;

- Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio- political, marketing, or other relevant issues.

GMSI concludes that the resource evaluation reported in the present Report is a reasonable representation of the global mineral resources found in the Copperwood Project at the current level of sampling. GMSI believes that there are no significant risks or uncertainties associated with the Project's Mineral Resource estimate or its potential economic viability.

25.1.2 Mining and Mineral Reserves

GMSI has estimated the Mineral Reserves in accordance with CIM Standards and reported them in accordance with NI 43-101. The Mineral Reserve estimate was prepared under the supervision of Mr. Carl Michaud, Eng, Manager of Underground Mine Engineering with GMSI who is an independent QP.

The main conclusions on the mining and mineral reserve estimation are as follows:

- A room-and-pillar mining is best adapted to the geometry of the orebody being relatively flat dipping over a large area with excellent lateral continuity. Room dimensions are 6.1 m wide with height depending on the LCBS mineralization thickness with an imposed minimum height of 2.1 m;
- Golder's geotechnical recommendations are based on geotechnical investigations, rock mass characterization and numerical modelling. The recommendations establish pillar dimensions as a function of depth for the east and west mine and a crown pillar requirement of 25 m. Pillars in the east mine are 4.9 m x 4.9 m at a depth of 122 m and increase to 7.6 m x 7.6 m at a depth of 275 m. Pillar dimensions in the west mine range from 5.5 m x 5.5 m at a depth of 90 m and increase to 9.4 m x 9.4 m at a depth of 275 m. The theoretical mining recovery is a function of pillar widths which are in turn a function of depth and room height and range between 63% to 80%;
- A 30 m step-back from Lake Superior was applied for the current design and a 10 m offset around old mine workings completed in the 1950s;
- Rock mass characterization from drill core suggests good rock quality with uniaxial compressive strengths ranging from about 50 to 90 Mpa. A basal gouge was characterized at the base of the mining column (base of Domino) and was accounted for in the pillar dimensions. A 30 cm of gray laminated is left in the back as a preferred unit to the red laminated;
- The Mineral Reserves estimate is based on a cut-off-grade of 1% copper or an NSR of about US\$48/t of ore which assumes a US\$3.00/lb copper price. The Proven and Probable Mineral Reserves are estimated at 25.4 Mt with an average copper grade of 1.43% and silver grade of

3.83 g/t for 802 M lb of contained copper and 3.1 M oz of silver. The Mineral Reserve estimate includes planned dilution to respect the minimum mining height of 2.1 m and unplanned dilution of 0.25 m in the back and 0.1 m from the floor. A 3% mining loss allowance is provisioned;

- Mine equipment selection requires low-profile equipment. Drilling will be done with a fleet of 10 two-boom jumbos, mucking with ten 10 t and two 8 t LHDs and ground support installed with eighteen 1-boom electric-hydraulic bolters. Material handling consists of twelve rock breaker loading stations that feed onto 42 in secondary conveyors located in the stopes which transfer to the main conveyors which transport the ore to the ore storage bins at surface;
- Mine ventilation requires 400 m³/s of fresh air delivered from a 5 m vent raise with two 4 m exhaust raises for each side of the mine;
- The mining cycle is approximately 14.1 h per round including 3.3 h for face drilling, 2.3 h for explosives loading, 1.9 h for mucking, 2.5 h for scaling and 4.1 h for bolting. Mining productivity is a function of room height and varies from 950 mtpd per panel at 2.1 m up to 1,285 mtpd per panel at 3.9 m;
- The production plan is developed to supply the mill at a nameplate capacity of 6,600 mtpd with the best available grade coming from the west mine followed by lower grade from the east mine. Approximately 7 panels are in production at any given time to achieve the required mining rate. During the pre-production period, the main conveyor drifts are excavated, and a stockpile of ore is generated reaching a maximum amount of 572 kt which is drawn down while the mine is ramping up in production allowing the mill to feed to capacity;

25.1.3 Metallurgical Testing and Mineral Processing

Comprehensive metallurgical testwork programs have been completed on Copperwood ores over the years. During the latest testwork program in 2017 and 2018, the main objective was to evaluate the process performance selected in the 2012 Orvana Feasibility Study, to improve performance and verify the variability of the ore over the deposit.

Some of the observations and conclusions are as follows:

- Alternative reagents were examined, but finally, the reagents used in the Metcon testwork appeared to deliver better performance which principally made use of sodium hydrosulphide (NaHS) as well as others;
- The major modifications consisted of finer primary grind of 40 microns, finer regrind of 15 microns, recirculation of the first cleaner scavenger concentrate to regrind and recirculation of the first cleaner tailings to rougher scavenger;

- The flotation time for most circuits increased: closing the first cleaner circuit with recirculation of the first cleaner scavenger concentrate to regrind with the same conditions appeared to increase the copper recovery;
- Variability testwork showed that copper recovery varies from 77% up to about 90% with a concentrate grade from 20% up to 29% copper. The overall average copper recovery was 86% with a weighted average copper concentrate grade of 24.7%;
- The process plant flowsheet and design are based on the testwork program with a nominal throughput of 6,600 mtpd with a planned availability of 91.3%. Lycopodium engineered the process plant;
- The overall flowsheet includes crushed ore reclaim, grinding and classification, rougher flotation, rougher concentrate regrind, cleaner flotation using three stages of cleaning, concentrate thickening and filtration and tailings pumping;
- Crushed ore is conveyed from the underground mine into two 1,200 t bins equipped with two pan feeders to reclaim material to feed the SAG mill feed conveyor;
- Grinding circuit includes a 7.92 m diameter x 3.75 m EGL with a 4,500 kW motor. The ball mill will have a 5.8 m diameter x 9.0 m EGL with a 5,000 kW motor;
- Rougher flotation will consist of eight 130 m³ forced air tank cells in series;
- Rougher concentrate and second cleaner tailings will report to the regrind cyclone feed pump box. The regrind mill will be a vertical mill;
- Cleaner flotation will consist of three stages of closed circuit cleaning. First cleaner consists of six 18 m³ cells in series. The second cleaner consist of six 8 m³ cells in series and the third cleaner six 2 m³ cells in series;
- Final concentrate will be pumped to a 16 m diameter high-rate thickener. Thickened concentrate will be pumped in batch to the concentrate filter press (1,500 mm x 1,500 mm x 40 mm) with a target moisture of 9%.

25.1.4 Infrastructure

The Copperwood Project requires several infrastructure elements to support the mining and processing operations.

The infrastructure planned for the project includes the following:

- County Road 519 upgrade under responsibility of the Michigan Department of Transportation;

- Site access road (4.1 km) from the entrance of CR 519;
- Grid power connection requiring 25 mi of 115 kV line between the Norrie substation in Ironwood and main substation at Copperwood under the responsibility of the utility company;
- Site electrical distribution at 13.8 kV;
- Communications infrastructure (fiber optic link and LTE communications network);
- Covered box cut for the mine entry (250 m long ramp at 15%);
- Ore stockpile pad at surface (65,000 m² area with HDPE liner);
- Water in-take in Lake Superior with a capacity of 500 USGPM for fresh water make-up and potable water supply;
- Sewage treatment using stabilization ponds;
- Fuel storage (10,000 l);
- Gatehouse to control site access;
- Explosives depot;
- Truck shop (5 bays including one wash bay), warehouse (20 m x 25 m) and related offices;
- Mine dry for 375 workers;
- Metallurgical laboratory and mill offices;
- Transload facility for concentrate handling (located in Park Falls);
- Administration and assay laboratory (located in Wakefield or Ironwood);
- Tailings disposal facility constructed with cut and fill approach in three stages with HDPE liner;
- Effluent water treatment plant for 275 USGPM constructed in 2025;
- Event pond ditches for surface water management at mill site.

25.1.5 Environmental and Permitting

To start construction and begin operation of this project, a plethora of permits must be obtained and agreed upon between Highland and the Regulators, on both state and federal levels. The major environmental permits required include:

- Part 632 Non-Ferrous Metallic Mining Permit;
- Part 31 National Pollutant Discharge Elimination System Permit;
- Part 55 Air Permit to Install;
- Part 301 Inland Lakes and Streams Permit;
- Part 303 Wetland Permit;
- Part 315 Dam Safety Permit;
- Part 325 Bottomlands Permit;
- Section 10 US Army Corps of Engineers Water Intake Permit.

Other minor and local permits are also required to start construction and mine operation that include:

- Local building and zoning permits;
- Explosives handling permit from the US Bureau of Alcohol, Tobacco, and Firearms;
- Storage tank permits;
- Mine Safety and Health Administration registration.

25.1.6 CAPEX, OPEX and Economic Analysis

- The CAPEX for project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be US\$275 M. The total initial project capital includes a contingency of US\$22.9 M, which is 9.1% of the total CAPEX before contingency, and excludes pre-production revenue of US\$30.36 M. Net of pre-production revenue, the initial CAPEX is estimated at US\$244.6 M;
- Sustaining capital expenditures during operations are required for additional mine equipment purchases, mine development work, tailings storage expansion for Stages 2 and 3, and the WTP. The total LoM sustaining CAPEX is estimated at US\$156.5 M;
- The NSR for the Project during operations is estimated at US\$1,822 M excluding US\$30.36 M of NSR generating during pre-production and treated as pre-production revenue. The average NSR over the LoM is US\$2.80/lb of payable copper;
- The average OPEX over the LoM is US\$39.84/t of ore or US\$1.53/lb of payable copper with mining representing 53.4% of the total OPEX, or US\$21.26/t of ore;

- The undiscounted after-tax cash flow is estimated at US\$316 M for the Project. The pre-tax NPV8% is estimated at US\$162.1 M with an 21.1% IRR and 2.9 yr payback period. Similarly, the after-tax NPV8% is estimated at US\$116.8 M with an 18.0% IRR and 3.2 yr payback period.

25.2 Risks and Opportunities

The risks and opportunities identification and assessment process is iterative and has been applied throughout the Feasibility Study.

Like all projects, there remains risks and opportunities that could affect the economic results of the Project. Many of the risks and opportunities are general to mining projects and some are specific to the Project which typically need additional information, testing or engineering to confirm assumptions and parameters.

25.2.1 Risks

The risks for the Project that are general or specific include:

- Permit acquisition or delays;
- Ability to attract experienced professionals;
- Declining metal prices;
- Development or construction start date;
- Faults creating offsets in the mineralization;
- Power line connection to grid;
- Reduction in grant for County Road 519 upgrade.

Table 25.1: Copperwood Project Risks

Risk		Explanation / Potential Impact	Possible Risk Mitigation
GR1	Permit acquisition or delay	The ability to secure all the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits in a timely fashion could delay the Project.	A quality permit application and continued discussions with regulators is required.
GR2	Ability to attract experienced professionals	The ability of the Project to attract and retain competent, experienced professionals is a key success factor for the Project. High turnover or the lack of appropriate technical and management staff at the Project could result in difficulties meeting Project goals.	The early search for, and retention of, professionals may help identify and attract critical people.
GR3	Declining metal prices	Declining metal prices during the mine development process could have a negative impact on the profitability of the operation, especially in the critical first years.	Begin construction when the outlook is good for price improvement and have mitigating strategies, such as hedging to address the risk of a downturn.
PR1	Development or construction schedule	The timing of the construction start is important to avoid the winter season. The critical activity in the Project schedule is the box-cut excavation to set-up the underground mine entry. Also of importance, is the TDF construction but it is not part of the critical path.	Cell #1 of the TDF was minimized to reduce the initial capital cost and also remove it from the critical path during the construction period.
PR2	Faults creating offsets in the mineralization.	One fault has been identified and modelled. Intercepting faults is very difficult given their vertical nature. This could generate additional difficulty during mining to properly follow the copper-bearing seam resulting in additional costs and/or lower productivities.	To mitigate the risk during mining it is planned to have a heading in the panel mined in advance of the other headings to anticipate any faults and required offsets to be implemented with the other headings.
PR3	Power line connection to grid	Utility company is responsible for engineering, permitting and constructing the power line connecting the Project to the main grid. Any delays in this process could delay commissioning and start-up of the process plant.	An early award of detailed engineering and permitting of the power line will reduce the risk of potential delays. Another mitigation is to identify large rental generators in the event of delay to initiate commissioning of the process plant except for possibly the grinding mills.
PR4	Reduction in grant for County Road 519 upgrade	It is planned to receive a grant from the Michigan Department of Transportation for the road upgrade which reduces the capital investment for the Project.	There is risk that the amount of the grant change in the future.

25.2.2 Opportunities

The Copperwood Project has several opportunities that have not been incorporated in the current Feasibility Study which would require further engineering, technical information or modifications to current permitting applications.

The significant project opportunities identified are as follows:

- Additional mineral reserves;
- Mining with a continuous miner or mobile miner;
- Ground support design criteria and mining height optimization;
- Underground tailings disposal;
- Metallurgical recovery improvements;
- Copper concentrate leaching.

Table 25.2: Copperwood Project Opportunities

Opportunity		Explanation	Potential Benefit
PO1	Additional Mineral Reserves beneath Lake Superior	The Project has the potential to add additional mineral reserves with the most attractive location being the extension of the orebody beneath Lake Superior. The current permitting application leaves an artificial 30m buffer with the Lake Superior boundary.	The mineralization at this location is deemed to be higher grade (in nature and would extend mining of the West side of the orebody (i.e. Main Zone) and defer the mining of lower grade mineralization on the East side (i.e. Section 5 and Section 6). Additional directional drilling and drilling from the lake would be required to extend the geological information. The objective is to demonstrate the viability of the mining method and the lack of ground subsidence and in due course amend the mining permit.
PO2	Continuous Miner.	Mining with a continuous miner has been investigated and presents an interesting alternative to conventional room-and-pillar mining. Epiroc has developed a Mobile Miner specifically developed for continuous hard rock mining applications which provides efficient advance rates, easier scheduling and safe work environments. For the Copperwood application the Mobile Miner 22H was identified which is specifically designed for low seam or low-profile mining as low as 2.2 m. The mechanical excavation of rock significantly reduces the risk of falling rocks compared to drill and blast methods and will make it easier to follow the orebody resulting in less dilution from overbreak.	The range of motion of the bolting arms has blind spots and limited range which makes it impossible to install bolts according to the current geotechnical design criteria established for the Project. Custom engineering and modifications to the machine are required to overcome the current limitations. For the current Feasibility Study, the product was not completely suitable and proven in practice to be included as the primary equipment for the mine. Continued work on addressing the bolting limitations and the introduction of a test mine panel will be envisaged going forward to confirm productivities and costs.
PO3	Ground Support Design Criteria Improvements and Mining Height	The ground support design criteria and mining height are somewhat interrelated. The ground support design criteria require 1.83 m (8.0 ft) bolts and the additional clearance of the bolting machine which in part dictated a minimum mining height of 2.1 m.	Should shorter bolts be acceptable there is the opportunity to further optimize the mining height of certain panels. This could reduce the amount of internal or planned dilution and therefore increase the head grade. Adjustments to the mining height design criteria could result in additional mineral reserves especially from mineralization located on the periphery of the current mine design.

Opportunity		Explanation	Potential Benefit
PO4	Underground Tailings Disposal	The current mining sequence starts in the Main Zone to the West due to the higher grade and then continues towards the East in Section 5 and Section 6.	There is an opportunity to initiate underground tailings disposal once activities have ceased in the West and all mining operation have relocated to the East side. This opportunity would result in less tailings disposal on surface and could be adjusted such that tailings disposal cell #3 not be required. This would reduce the sustaining capital cost associated with the last cell (US\$28M) and the associated closure and reclamation costs. Additional environmental characterization and impact assessments would be required as well as additional permitting efforts.
PO5	Metallurgical Recovery Improvements	The copper recovery may be further optimized by concentrate grade and reagent optimization. Additional characterization might be done specifically for areas where results were lower. In a next stage of testwork the impact of desliming would be worthwhile.	The potential benefit is a direct increase in metal production and therefore revenues which would increase the economics of the Project.
PO6	Concentrate Leaching	Concentrate ferric sulfate leaching including technology developed by FLSmidth was investigated has a replacement alternative in part or in totality of the production of a copper concentrate with its associated transport, smelter treatment and refining costs. The FLSmidth Rapid Oxidative Leach ("ROL") process is in the early development stage and is considered a revolutionary technology now being jointly developed and commercialized with BASF. The leach process technology is an atmospheric, 80-90°C, acid ferric sulfate process modified for leaching copper from primary and secondary copper sulfide concentrates. An enabling feature of this mechano-chemical technology is the incorporation of inter-stage Stirred Media Reactors ("SMRt") within a series of Continuous Stirred Tank Reactors (CSTR's). The technology integrates directly with existing SX/EW plants.	Preliminary testwork with Copperwood concentrates using conventional ferric leaching technology or Rapid Oxidation Leach (ROL), from either rougher or intermediate final flotation stages, showed excellent copper dissolution rates between 96-99% in less than 6-8 hours. Additional testwork is required a full trade-off study comparing with the current processing scheme designed to produce a copper concentrate.

26. RECOMMENDATIONS

26.1 Project Recommendations

Based on the results of the Feasibility Study, GMSI recommends that the Copperwood Project move forward to the next phase which would include the following:

- Secure Project financing;
- Initiate critical detailed engineering to support critical item purchases;
- Finalize and implement an early works program in anticipation of construction release;
- General detailed engineering of process plant and other project components;
- Implement an ERP to facilitate project management and controls;
- Project construction.

26.2 Recommended Work Programs

A series of recommended work programs have been proposed to reduce risks or evaluate further opportunities for the Project.

The timing of these work programs is variable due to project schedule with some costs viewed as core to the current project and others discretionary in nature as these relate to opportunities not factored into the current Study.

The work programs categorized as core are currently part of the initial capital cost estimate, but it is recommended that they be approved prior to full project release to reduce schedule risk.

Table 26.1: Recommended Work Programs

Work Program Description	Timing of Program	Estimated Cost US\$ k)	
		Core	Discretionary
Geology and Mineral Resources			
Infill resource drilling at Copperwood Deposit (Section 5 area) to upgrade current Indicated Mineral Resources to Measured category. 6,000 m of total drilling (20-30 drill holes with varying depths, direct drilling costs only)	Before Yr-5 of operations		600
Consider relogging the Grey Laminated – Red Laminated contact in the historical drill holes from the 1950's to ensure a consistent interpretation. GMSI noticed small inconsistencies regarding the accuracy of this contact, which may be a result of logging practise changes over the years. The contact is difficult to pick as it is transitional in nature, however it is often mineralised. Estimated at 100 - 120 drill holes to relog.	Pre-Mine Development	10	
Consider undertaking a structural review of the Copperwood Deposit to confirm and refine the current interpretation of the thrust fault (T1). This thrust fault displaces the LCBS and UCBS in the western portion of the deposit and adds uncertainty to the mine plan in regard to its exact location. The likelihood of further reverse thrust faults and displacements at Copperwood is high, as the current drill-spacing and orientation does not allow for further definition of these subtle structures. GMSI recommends reviewing the N-S oriented drilling sections to identify unexpected deviations in stratigraphy that could be indicative of a fault displacement, and consider definition drilling (3,500m of total drilling, 10-15 drill holes) if warranted.	Pre-Mine Development	10	350
Consider exploring the area east of Sector 5, where the UCBS and LCBS converge and the grade of the UCBS improves dramatically. There is the potential to add significant tonnage to the Copperwood Deposit, and the life of the mine. 3,500m of total drilling (15 drill holes) to determine an Inferred Mineral Resource.	If and when mineral rights are acquired		350
Mining, Mineral Reserves and Geotechnical			
Undertake test work to determine the directions and intensity of the principal regional stress. These tests must be done from the Main access drift as soon as this drift is far enough from the surface for the test to be representative. This test will improve the geotechnical/rock mechanics modelling.	Once initial development in place		100
Plan and initiate a test mine with the Epiroc Mobile Miner to finalize and validate trade-offs (productivity, CAPEX, OPEX) versus conventional room and pillar mining. A cost sharing approach with the equipment manufacturer would be envisaged.	Once initial development in place		Manufacturer support
Metallurgy and Mineral Processing			
Additional metallurgical testwork programs to verify impact of a desliming stage.	During or before detailed Eng.	50	
Process optimization. Additional characterization of areas with lower metallurgical recoveries. Reagent optimization.	During or before detailed Eng.	100	
Validation and production of copper concentrate and tailings for additional characterisation (suppliers or engineering firm)	During or before detailed Eng.	200	200
A pilot plant campaign to validate and optimise the process flowsheet, retention time and reagents type and addition points could be beneficial.	Once in development ore		400
Initial Project and Detailed Engineering			
Initiate detailed engineering and permitting of power line with power provider.	Post FS completion	400	
Box-cut detailed engineering to finalize culvert design and purchase orders.	Post FS completion	125	
Implement early works program to put in place project controls and operating systems to support construction activities (ex: ERP with job cost modules, etc.).	Post FS completion	300	
Total Cost		995	2,000

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