# Feasibility Study Update

# Copperwood Project
# Michigan, USA

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# IMPORTANT NOTE

## Cautionary Statement

## Forward-Looking Information

This Technical Report contains "forward-looking information" or "forward-looking statements" that involve a number of risks and uncertainties. Forward-looking information and forward-looking statements include, but are not limited to, statements with respect to the future prices of copper, palladium, platinum, gold and silver; the estimation of Mineral Resources and Reserves; the realization of mineral estimates; the timing and amount of estimated future production; costs (including capital costs, operating costs, and other costs); permitting timelines; timing of the LOM; rates of production; annual revenues, economic analysis, including forecasted annual revenues, cash flows, IRR, NPV, payback period and various other operational, economic and financial metrics; currency exchange rates; levels of employment; requirements for additional capital; government regulation of mining operations; and environmental risks.

Often, but not always, forward-looking statements can be identified by the use of words such as "plans", "expects", or "does not expect", "is expected", "budget", "scheduled", "estimates", "forecasts", "intends", "anticipates", or "does not anticipate", or "believes", or variations of such words and phrases or state that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved.

Forward-looking statements are based on the opinions, estimates and assumptions of contributors to this Technical Report. Certain key assumptions are discussed in more detail herein. Forward looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of the Marathon Project to be materially different from any other future results, performance or achievements expressed or implied by the forward-looking statements.

Such risk factors include, among others: inherent uncertainties with respect to the actual results of current exploration activities, cost estimates, conclusions of economic evaluations and mineral resource and mineral reserve estimates; changes in project parameters, including schedule and budget, as plans continue to be refined; actual results of development activities; future prices of palladium, copper and other metals; possible variations in grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents; labour disputes and other risks of the mining industry; delays in obtaining or renewing governmental approvals; fluctuations in metal prices; shortages of labour and materials, the impact of COVID-19 on the supply chain and other complications facing the economy; inflationary pressures; risks of recession; the situation relating with the war in Ukraine and geopolitical uncertainties; risks of sanctions that may affect supplies of fuel, or the cost thereof, for mining operations; risks associated with pandemics, including any resurgence of the COVID-19 (coronavirus) pandemic; as well as those risk factors discussed or referred to in this Technical Report and in the Company's latest annual information form under the heading "Risk Factors" and other documents filed from time to time by the Company with the securities regulatory authorities in Canada.

There may be other factors than those identified that could cause actual actions, events or results to differ materially from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers are cautioned not to place undue reliance on forward-looking statements. Unless required by securities laws, the authors undertake no obligation to update the forward-looking statements if circumstances or opinions should change.

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Revision #

## Michigan, USA


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# 1. SUMMARY

## 1.1 Introduction

G Mining Services Inc. ("GMS") and other engineering consultants were retained by Highland Copper Company Inc. ("Highland" or the "Company") to produce a Feasibility Study Update (the "FSU" or "Study Update") 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 FSU as disclosed in Highland's press release entitled "*Highland Copper Announces Updated Feasibility Study Results for its fully permitted Copperwood Project in Michigan, USA*" dated March 6, 2023.

The major contributors for the Study and the Report and their respective areas of responsibility are as follows; Since the current report is an update of the 2018 technical report, the 2018 contributors are listed:

- GMS – 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 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 (3) 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 (2) 20-year lease agreements with Keweenaw Minerals, LLC (formerly Keweenaw Land Association Limited) (“KLA”), a 20-year lease agreement with KLA (formerly Sage Minerals Inc.) 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 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 the copper produced from the Copperwood Project; however, upon final closing of the acquisition by Highland of the White Pine Project, Highland granted Osisko a 1.5% NSR royalty on copper from the White Pine North Project and Osisko’s royalty on the Copperwood Project was reduced to 1.5%.

Osisko Stream Royalties Ltd (“Osisko”) also owns a 11.5% royalty on all the produced Silver from the Copperwood Project.

CRI owns approximately 717 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.

The 40 km-long 115 kV line will tie at the existing Norrie substation in Ironwood. Both the transmission line and site main substation will be designed, supplied, built, owned and operated by the Utility company. Power cost rate will be factored to covers these costs.

### 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 (GMS) – Copperwood Deposit, published June 25, 2015, effective date of April 15, 2015.
- Highland (GMS) – 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 1960s. | 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 |

| Company | Activity | Year |
|---|---|---|
| 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 program, 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 (2) 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 (5) core holes for environmental purposes. These drill holes intersected significant copper mineralization. Orvana subsequently completed 82 drill holes in 2009. Orvana U.S. 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 (5) 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 1950s 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 minimize 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 televiewed and subsequently cemented.

GMS 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

GMS 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 Mr. James Purchase, P.Geo. of GMS and Highland representatives. Assay certificates from the 2018 drilling campaign were checked against the database to ensure accuracy. The 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. No additional technical or scientific information has been gathered since the 2018 drilling campaign.

## 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 | 6,800 | - | Highland |
| Head Grade - LoM | % Cu | 1.35 | | Highland |
| | g/t Ag | 3.41 | | Highland |
| Plant Availability (Years 1 & 2) | % | 91.3 | | GMS |
| Plant Availability (Years 3 and after) | % | 95% | | GMS |
| 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 | | Calculation |
| Concentrate Thickener Solids Loading | $t/m^2.h$ | 0.20 | | Consultant |
| Filter Solids Loading | $kg/m^2.h$ | 160 | | Consultant |

**Note[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 (3) 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, GMS 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 0.9% 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 Measured and Indicated Mineral Resources are reported at 54.2 Mt grading an average 1.49% Cu and 3.6 g/t Ag containing 1.78 B lbs Cu and 6.3 M oz Ag using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined. Inferred Mineral Resources are reported at 2.3 Mt grading an average 1.12% Cu and 1.2 g/t Ag containing 56 M lbs Cu and 0.1 M ozs Ag using a cut-off grade of 0.9% Cu.

The Satellite Deposits Inferred Mineral Resources are reported at 76.8 Mt grading 1.09% Cu and 3.6 g/t Ag containing 1.84 billion pounds of copper and 8.9 million ounces of silver using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined.

Table 1.3 reports Mineral Resources for the Copperwood and Satellite Deposits by resource categories. All parameters used in the calculations are also presented in the table’s notes.

The responsible Qualified Person is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Copperwood Mineral Resource Estimate

**Table 1.3: Mineral Resource Estimate - Copperwood Project 0.9% Cu Cut-off Grade February 28th, 2022**

| Deposits | Resource Category | Tonnage (Mt) | Copper Grade (%) | Silver Grade (g/t) | Copper Contained (M lbs) | Silver Contained (M oz) |
|---|---|---|---|---|---|---|
| LCBS | Measured | 27.9 | 1.66 | 4.5 | 1,023 | 4.1 |
| | Indicated | 16.1 | 1.42 | 2.4 | 504 | 1.2 |
| | M + I | 44.0 | 1.57 | 3.7 | 1,527 | 5.3 |
| | Inferred | 2.3 | 1.12 | 1.2 | 56 | 0.1 |
| UCBS | Measured | 0.1 | 0.95 | 4.6 | 2 | 0.0 |
| | Indicated | 10.1 | 1.13 | 3.1 | 253 | 1.0 |
| | M + I | 10.2 | 1.13 | 3.1 | 255 | 1.0 |
| | Inferred | - | - | - | - | - |
| Satellite LCBS | Inferred | 49.7 | 1.1 | 2.5 | 1210 | 3.9 |
| Satellite UCBS | Inferred | 27.1 | 1.1 | 5.7 | 630 | 5.0 |

**Notes on Mineral Resources:*

1. *Mineral Resources are reported using a copper price of US$4.00/lb and a silver price of US$25/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 0.9% 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$4.00/lb Cu price, a sliding scale 5.5% NSR royalty on the Copperwood Project is payable to leaseholders.*
7. *Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.*
8. *A minimum mining thickness of 2m was applied. No additional 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 (2014).*
11. *The qualified person for the estimate is Mr. James Purchase, P.Geo., Consulting Geologist for GMS. The estimate has an effective date of February 28, 2022.*
12. *LCBS: Lower Copper Bearing Sequence.*
13. *UCBS: Upper Copper Bearing Sequence.*
14. *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 Resources that are not mineral reserves do not have demonstrated economic viability.*

## 1.14 Mineral Reserves Estimate

The Mineral Reserves for the Copperwood Project are estimated at 25.7 Mt, at an average grade of 1.45% Cu and 3.91 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$ 69.5/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 minimum mining height or and sloping sections of floor to have a maximum 6º 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 | 18.2 | 1.49 | 4.47 | 597 | 2.6 |
| Probable | 7.5 | 1.34 | 2.56 | 222 | 0.6 |
| **Proven & Probable** | **25.7** | **1.45** | **3.91** | **820** | **3.2** |

**Notes on Mineral Reserves:*

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (Nov. 29, 2019) and CIM Definition Standards for Mineral Resources and Reserves, (May 10, 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 $4.00/lb and a silver price of $25.00/oz.*
4. *Asssuming a long-term copper price $4.00/lb, a sliding scale 4.0% NSR royalty on the Copperwood Project is payable to leaseholders. A 1.5% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd. This also includes an additional 11.5% silver mineral royalty payable to Osisko Stream Royalties.*
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 Carl Michaud, P.Eng., VP, Mining Engineering for GMS. The estimate has an effective date of May 25, 2022*
10. *The numbers may not sum due to rounding; rounding followed the recommendations in NI 43-101.*
11. *The geotechnical parameters of the previous technical report from June 2018 were used in this Feasibility Study update.*

## 1.15 Mining

### 1.15.1 Mining Method

The proposed mining method for the Copperwood Project is room-and-pillar given the relatively sub-horizontal orebody that varies in thickness from 1.6 m to 3.7 m. Based on the orebody thickness, two (2) approaches were selected to carry out the development of the room and pillar: conventional drill and blast, and continuous mining. The drill and blast approach is utilized whenever the orebody thickness is below 3.0 m, whereas the continuous miner will be used in the areas where the orebody thickness is 3.0 m or greater.

The mining method consists of the extraction of a series of entries and crosscuts 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.

### 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 two (2) drifts are excavated, and expand to four (4) drifts at a depth of 35 m. The mine consists of two (2) mining sectors: West and East. The mine development is designed with four (4) 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 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 (2) conveyors.

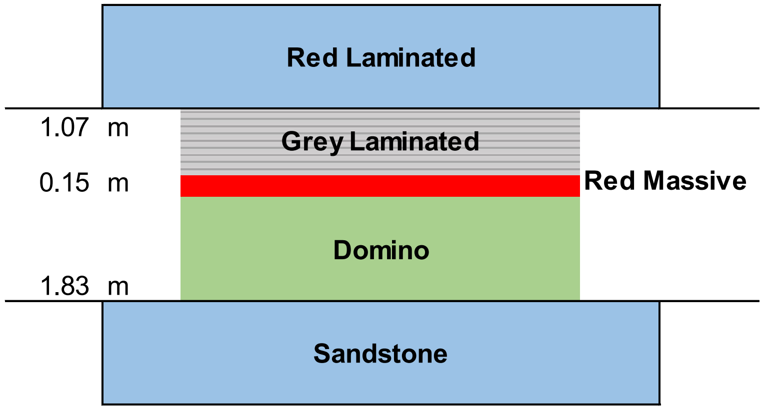
Barrier pillars between the main access and the stopes will be kept in place until the stope area is mined out. These barrier pillars are designed to be recovered, but they will respect Golder's recommended pillar size.

### 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 (3) 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.

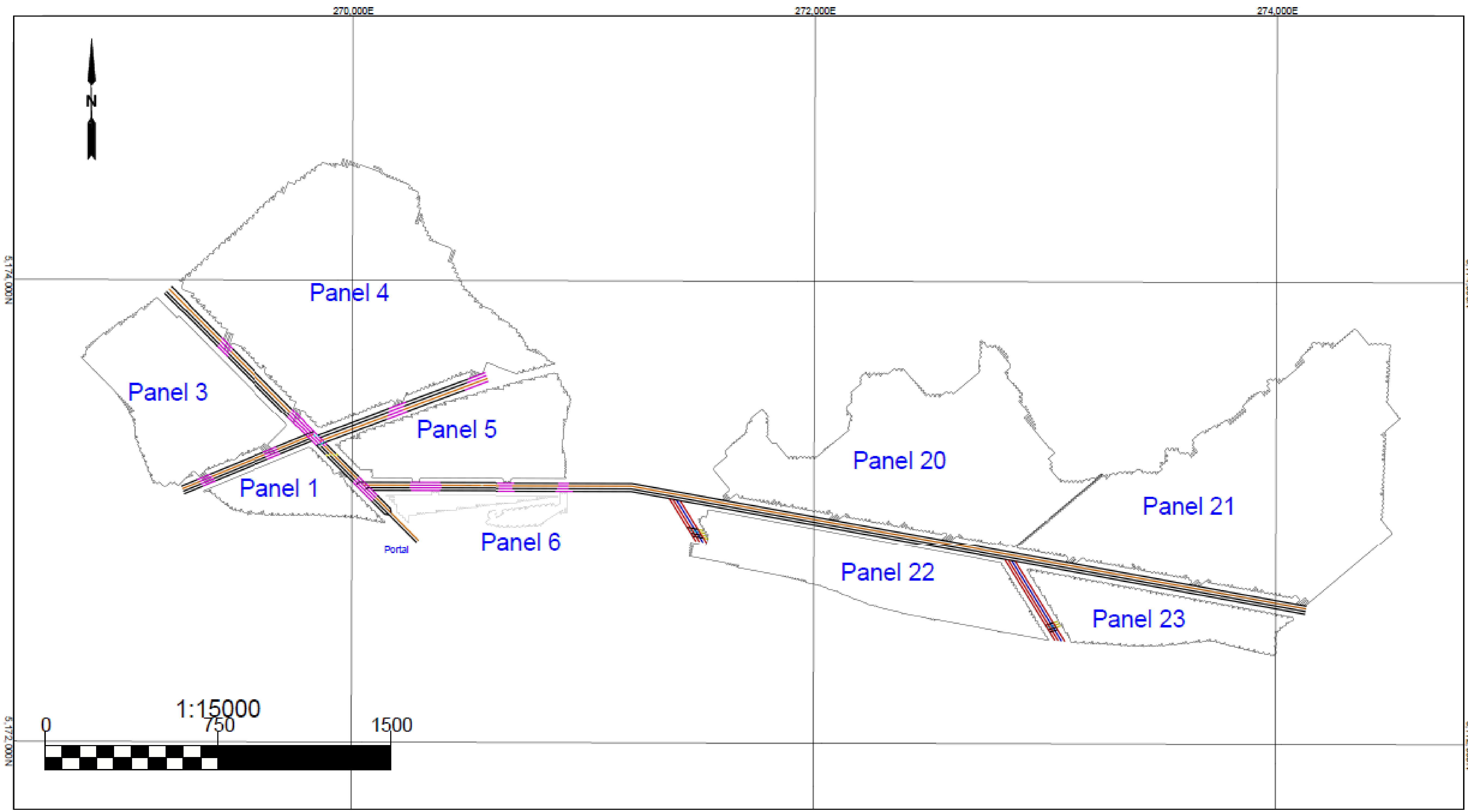
**Table 1.5: Pillar Size Recommendations**

| 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 comprised of two (2) 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 five (5) extraction panels, panel 1, 3,4,5,6, as, detailed in Figure 1.2. The East part is subdivided into four (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**



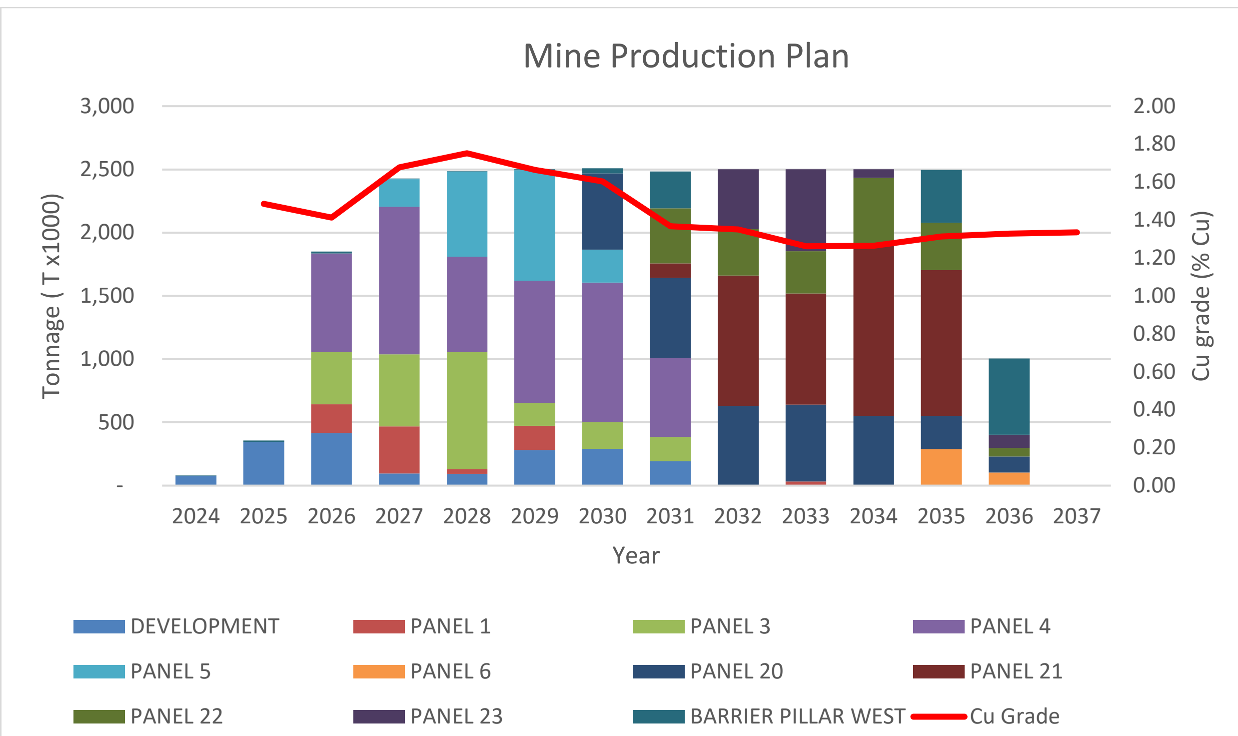
### 1.15.5 Mine Production Schedule

Development mining during the pre-production period is planned to start in Q3 2024 once the box-cut has been completed. Development initially consists of two (2) headings with an advance rate of 5.6 m/d from the box-cut entrance and splits into four (4) headings at which point an additional development team is planned and the advance rate increases to 9.0 m/d. It was assumed that the first six (6) months of pre-production will be excavated by a mining contractor. The rest of the preproduction and production drift development will be excavated by the Owner's mining department. Development mining will be ongoing at different rates 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.

Stopping activities are initiated in Q1 2026 simultaneously with the start of commercial production. Commissioning and plant ramp-up will take place during the first quarter of 2026 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 (662 M lb) with an annual average of 29 kt (64.6 M lb) over the 10.3-year life which includes three (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 1.1 M oz with an annual average of 107 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 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Development Mining** | | | | | | | | | | | | | | | |
| Tonnage | kt | **1,788** | - | 77 | 344 | 414 | 97 | 93 | 281 | 290 | 192 | - | - | - | - |
| Cu Head Grade | %Cu | **1.22** | - | 1.48 | 1.40 | 1.47 | 1.17 | 0.66 | 0.90 | 1.19 | 1.02 | - | - | - | - |
| Ag Head Grade | g/t | **3.54** | - | 5.25 | 4.89 | 5.05 | 2.59 | 1.45 | 1.74 | 2.96 | 2.20 | - | - | - | - |
| Cu Contained Metal | kt | **22** | - | 1.1 | 4.8 | 6.1 | 1.1 | 0.6 | 2.5 | 3.5 | 2.0 | - | - | - | - |
| Ag Contained Metal | K ozs | **204** | - | 13.0 | 54.1 | 67.2 | 8.1 | 4.3 | 15.8 | 27.6 | 13.5 | - | - | - | - |
| **Production Mining** | | | | | | | | | | | | | | | |
| Tonnage | kt | **23,916** | - | 2 | 13 | 1,438 | 2,331 | 2,394 | 2,220 | 2,220 | 2,292 | 2,502 | 2,502 | 2,502 | 2,496 |
| Cu Head Grade | %Cu | **1.46** | - | 1.61 | 1.72 | 1.74 | 1.78 | 1.70 | 1.69 | 1.39 | 1.38 | 1.26 | 1.26 | 1.31 | 1.33 |
| Ag Head Grade | g/t | **3.94** | - | 5.44 | 5.73 | 5.73 | 6.07 | 5.99 | 6.04 | 4.43 | 3.70 | 2.21 | 2.31 | 1.93 | 2.54 |
| Cu Contained Metal | kt | **350** | - | 0 | 0 | 25 | 41 | 41 | 38 | 31 | 32 | 32 | 32 | 33 | 33 |
| Ag Contained Metal | K ozs | **3,029** | - | 0 | 2 | 265 | 455 | 461 | 431 | 317 | 273 | 178 | 186 | 155 | 204 |
| **Total Mining** | | | | | | | | | | | | | | | |
| Tonnage | kt | **25,703** | - | 80 | 356 | 1,852 | 2,428 | 2,487 | 2,502 | 2,510 | 2,484 | 2,502 | 2,502 | 2,502 | 2,496 |
| Cu Head Grade | %Cu | **1.45** | - | 1.48 | 1.41 | 1.68 | 1.75 | 1.66 | 1.60 | 1.37 | 1.35 | 1.26 | 1.26 | 1.31 | 1.33 |
| Ag Head Grade | g/t | **3.91** | - | 5.25 | 4.92 | 5.58 | 5.93 | 5.82 | 5.56 | 4.26 | 3.59 | 2.21 | 2.31 | 1.93 | 2.54 |
| Cu Contained Metal | kt | **372** | - | 1 | 5 | 31 | 43 | 41 | 40 | 34 | 34 | 32 | 32 | 33 | 33 |
| Ag Contained Metal | K ozs | **3,233** | - | 13 | 56 | 332 | 463 | 466 | 447 | 344 | 286 | 178 | 186 | 155 | 204 |

**Table 1.7: Mill Production Schedule Summary**

| Mill Production | | Total | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnage Processed | kt | **25,703** | | 85 | 2,201 | 2,409 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 1,008 |
| Cu Head Grade | % Cu | **1.45** | | 1.43 | 1.64 | 1.75 | 1.67 | 1.60 | 1.37 | 1.35 | 1.26 | 1.26 | 1.31 | 1.33 | 1.34 |
| Ag Head Grade | g/t | **3.91** | | 4.98 | 5.48 | 5.93 | 5.83 | 5.55 | 4.28 | 3.58 | 2.21 | 2.31 | 1.93 | 2.54 | 3.15 |
| Concentrate (dry) | dmt | **1,292** | | 4.2 | 125.2 | 146.6 | 144.7 | 139.2 | 118.9 | 117.2 | 109.6 | 109.8 | 114.0 | 115.4 | 46.8 |
| Concentrate (wet) | wmt | **1,419** | | 4.6 | 137.6 | 161.1 | 159.0 | 153.0 | 130.6 | 128.8 | 120.4 | 120.7 | 125.3 | 126.8 | 51.4 |
| Cu Contained Metal | kt | **372** | | 1 | 36 | 42 | 42 | 40 | 34 | 34 | 32 | 32 | 33 | 33 | 13 |
| Cu Contained Metal | M lbs | **820** | | 2.67 | 79.46 | 93.03 | 91.82 | 88.33 | 75.42 | 74.37 | 69.53 | 69.67 | 72.36 | 73.19 | 29.68 |
| Ag Contained Metal | k ozs | **3,233** | | 14 | 388 | 459 | 469 | 446 | 344 | 288 | 178 | 186 | 155 | 204 | 102 |
| 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 | 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 | 73.4 |
| Cu Metal Production | kt | **320** | | 1.0 | 31.0 | 36.3 | 35.8 | 34.5 | 29.4 | 29.0 | 27.1 | 27.2 | 28.2 | 28.5 | 11.6 |
| Cu Metal Production | M lbs | **705** | | 2.3 | 68.3 | 80.0 | 79.0 | 76.0 | 64.9 | 64.0 | 59.8 | 59.9 | 62.2 | 62.9 | 25.5 |
| Ag Metal Production | k ozs | **2,373** | | 10 | 285 | 337 | 344 | 328 | 253 | 211 | 131 | 137 | 114 | 150 | 75 |
| 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 | 95.76 |
| Ag Payable Rate | % | **47.65** | | 59.38 | 57.57 | 58.04 | 59.41 | 59.02 | 54.61 | 46.48 | 19.07 | 22.41 | 6.50 | 25.84 | 39.78 |
| Cu Payable Metal | kt | **306** | | 1.0 | 29.7 | 34.8 | 34.3 | 33.0 | 28.2 | 27.8 | 26.0 | 26.0 | 27.0 | 27.3 | 11.1 |
| Cu Payable Metal | M lbs | **675** | | 2.2 | 65.4 | 76.6 | 75.6 | 72.7 | 62.1 | 61.2 | 57.3 | 57.4 | 59.6 | 60.3 | 24.4 |
| Ag Payable Metal | k ozs | **1,131** | | 5.9 | 163.9 | 195.6 | 204.3 | 193.4 | 137.9 | 98.2 | 24.9 | 30.6 | 7.4 | 38.8 | 29.8 |

### 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 2.5 Mtpa, or 6,800 mtpd. To achieve this production, a total of 7 to 9 panels must be in production at any given time.

To achieve and maintain an adequate level of production, in drill and blast section 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.

In conventional 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 two (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/min and the average drilling time per round is evaluated at 3.3 h/round.

Explosives will consist of an emulsion mixture. Emulsion is better suited when there is presence of water. A decoupled explosive charge is recommended to pre-split the back. Blasting will be done at shift ends with a period of two (2) hours planned to vent blast fumes.

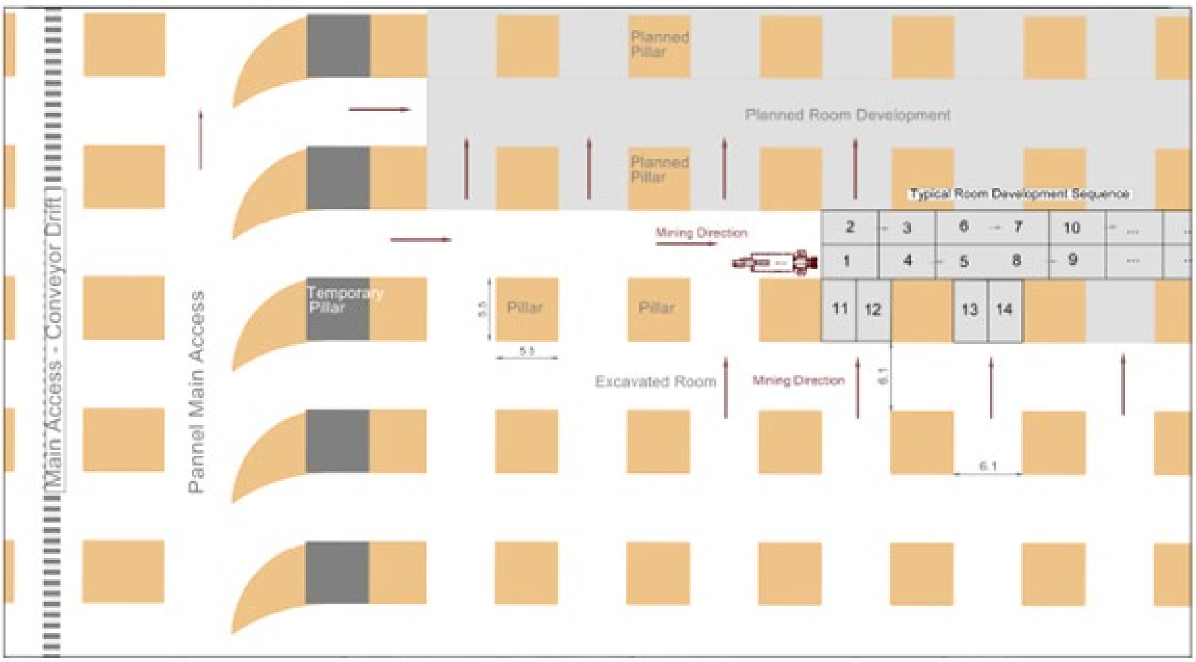
Mucking will be done with 10 t load-haul-dump (“LHD”) units that will load muck at the mine face and transport it to the conveyor loading point established for the production panel. The LHD performance will be a function of dip of the stope and distance. The conveyor loading points will be regularly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

In the continuous operating panels, the continuous miner begins with the cutting of a 3 m-wide section of the ore (room). The continuous miner loads the broken material to the LHD, which hauls the ore from the panel to the feeder breaker. After the broken material produced from the cut section is loaded, the operator backs out of the partially formed room and the rock bolting process, previously described in the conventional operating section, is done and allows the continuous miner to re-enter the room and begin the cutting of an additional 3 m section to produce a wider and final room advance. Figure 1.4 below presents the typical cutting sequence expected in a continuous operating panel.

**Figure 1.4: Continuous Miner Room Development Sequence**




### 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 (2) 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 (2) 5 m exhaust ventilation raises for each side of the mine will be equipped as emergency egresses.

The dewatering system will consist of six (6) 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 300 mtph. 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 (2) 1,200 t crushed ore bins will be equipped with two (2) 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 $m^3$ 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 (3) 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 (6) 18.0 $m^3$ 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 (6) 8 $m^3$ trough cells in series.

The third cleaner flotation cells will consist of six (6) 2 $m^3$ 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:

- Public access road upgrade (County Road 519N)
- Site access roads
- Parking lot
- Plant workshop & Stores
- Reagents storage
- Explosives storage
- Workshop, wash bay and warehouse
- Mine dry
- Mill offices and metallurgical laboratory
- Gatehouse
- Concentrate transload facility
- Administration office
- Assay laboratory
- Fuel storage
- High voltage power line and main substation
- Emergency Site power generation
- Site electrical distribution
- Process Plant Electrical Room
- Underground main Electrical Room at portals

- Other Electrical rooms (Ventilation intake, Tailings, WTP)
- Site Communications network for above ground installation and underground mine
- Potable water treatment
- Surface Water Collection Berm
- Reclaim Water System
- Stream relocations
- Tailings disposal facility in three stages
- Water treatment plant
- Fire water system
- Sewage treatment
- Site Vehicles and Mobile Equipment
- Ore stockpile pad
- Covered box-cut for mine access
- Compressors for underground

## 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$4.00/lb. The silver price has been assumed constant at US$25.00/oz over the Project life.

**Table 1.8: Metal Price Assumptions**

| Metal Price Scenario | Yr -1 (2025) | Yr 1 (2026) | Yr 2 (2027) | Yr 3+ (2028+) |
|---|---|---|---|---|
| Copper (US$/lb) | 4.25 | 4.15 | 4.00 | 4.00 |
| Silver (US$/oz) | 25.00 | 25.00 | 25.00 | 25.00 |

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 to a truck to rail transload facility located in Champion, Michigan, approximately 180 km from site. The truck configuration consists of a 11 axles road train with two (2) covered side-dump trailers and will transport approximately 48 t (53 short tons) per shipment. The location has been chosen due to the costs and mainly because it provides access to the Canadian National Railway (CN) networks.

A summary of the copper concentrate marketing assumptions is found 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

Extensive environmental studies were undertaken to obtain the original Mining Permit issued in 2012, with additional studies commissioned to obtain the Mining Permit Amendment in 2018. In accordance with Michigan's governing regulation Natural Resources and Environmental Protection Act (NREPA) Part 632 Nonferrous Mining, studies describing baseline conditions were conducted characterizing comprehensive environmental and archaeological resources. Encompassed in the 2012 and 2018 Mining Permit and amendment applications, the baseline conditions and environmental, social, and archaeological impacts of the project are thoroughly described.

### 1.19.2 Permitting

Major permits are in place to start construction. Some permits have additional approvals and actions, and Highland Copper is addressing those appropriately. The major environmental permits in place 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

The Part 55 Air Permit to Install is being amended to address on-site power generators not included in the current 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
- As the project continues to develop, there will be routine permit renewals, amendments, and modifications.

## 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$425.1M. The total initial Project capital includes a contingency of US$37.6M, which is 9.7% of the total CAPEX before contingency excluding pre-production revenue of US$33.962M. Net of pre-production revenue, the initial CAPEX is estimated at US$391.2M as presented in Table 1.10. The initial Project CAPEX is spent over a period of 27 months starting in January 2024 and ending in March 2026.

**Table 1.10: Initial Capital Expenditure Summary**

| Initial CAPEX | | US$ k |
|---|---|---|
| 000 - General | | 1,150 |
| 100 - Infrastructure | | 31,779 |
| 200 - Power & Electrical | | 42,460 |
| 300 - Water & TSF Mgmt. | | 46,198 |
| 400 - Mobile Equipment | | 24,932 |
| 500 - Mine Infrastructure | | 51,172 |
| 600 - Process Plant | | 105,502 |
| 700 - Construction Indirects | | 51,028 |
| 800 - General Services & Owner's Costs | | 25,377 |
| 900 - Pre-Production, Commissioning | | 7,888 |
| **Sub-Total Before Contingency** | | **387,487** |
| **Contingency** | 9.7% | **37,645** |
| **Total Incl. Contingency** | | **425,131** |
| **Less: Pre-Production Revenue** | | **(33,962)** |
| **Total Incl. Contingency & Pre-Prod. Revenue** | | **391,170** |

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$269.89 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 | 54.77 | 2.17 | 0.08 |
| Water Treatment Plant | 17.11 | 0.68 | 0.03 |
| Mine Equipment Purchases | 141.56 | 5.61 | 0.21 |
| Mine Development Expenditures | 33.11 | 1.31 | 0.05 |
| Sustaining CAPEX - Other | 23.35 | 0.93 | 0.04 |
| **Total Sustaining CAPEX** | **269.89** | **9.77** | **0.37** |

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

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$2,417M, excluding US$49.83M of NSR, generating during pre-production and treated as a reduction of initial capital expenditures. The average NSR over the LoM is US$3.65/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$48.05/t of ore or US$1.83/lb of payable copper with mining representing 50.0% of the total OPEX, or US$24.02/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 | 2,656 | 105.25 | 4.01 |
| Ag Credits | 27 | 1.09 | 0.04 |
| Revenue | 2,683 | 106.34 | 4.05 |
| Concentrate Transportation Costs | 140 | 5.56 | 0.21 |
| Treatment & Refining Charges | 126 | 4.99 | 0.19 |
| Net Smelter Return | 2,417 | 95.79 | 3.65 |
| Royalties | 136 | 5.37 | 0.20 |
| Mining Costs | 606 | 24.02 | 0.92 |
| Processing Costs | 369 | 14.63 | 0.56 |
| G&A Costs | 102 | 4.03 | 0.15 |
| **Total OPEX** | **1,212** | **48.05** | **1.83** |
| **Operating Cash Flow** | **1,203** | **47.68** | **1.82** |

**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$455.1M for the Copperwood Project. The pre-tax net present value at 8% (“$NPV_{8\%}$”) is estimated at US$221.8M with an 20.0% internal rate of return (“IRR”) and 3.2 y payback period. Similarly, the after-tax $NPV_{8\%}$ is estimated at US$167.6M with an 17.6% IRR and 3.5 y payback period.

The annual cash flow is summarized in Figure 1.5 and a cash flow waterfall for the Copperwood Project is presented in Figure 1.6.

**Figure 1.5: After-Tax Annual Project Cash Flow (with Equity)**

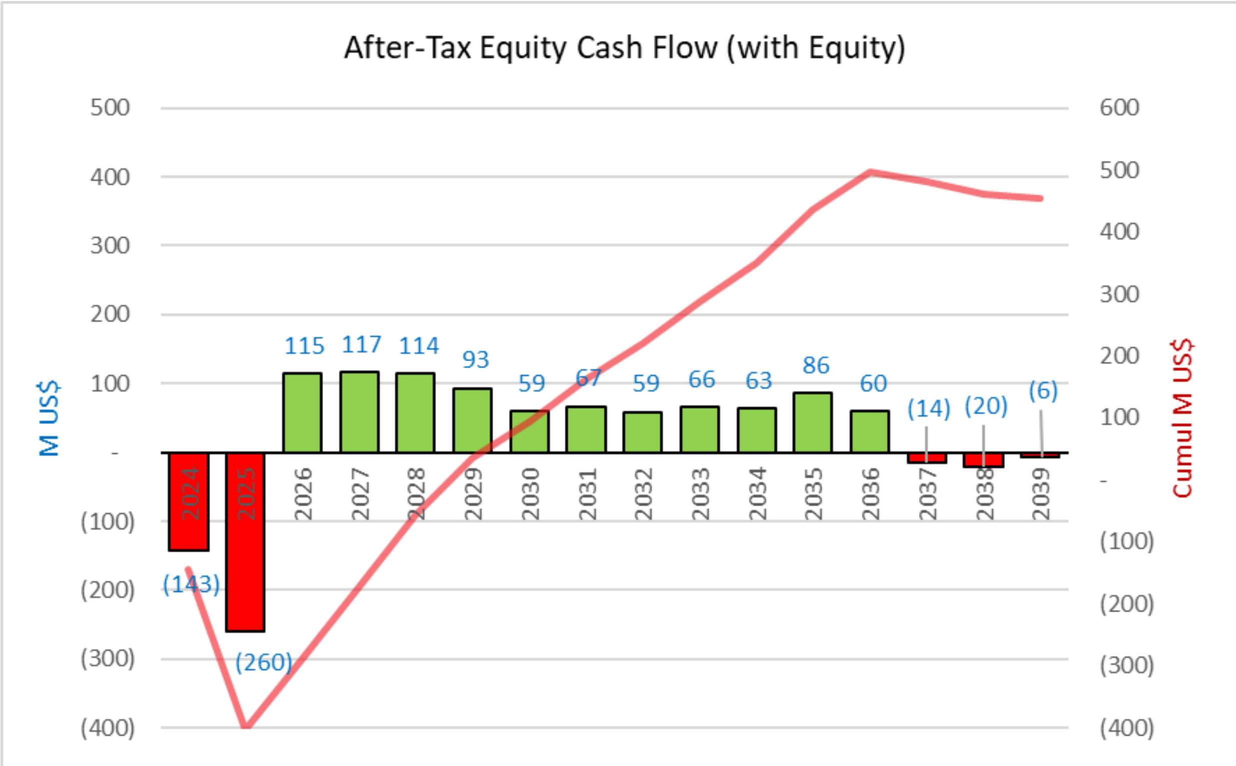
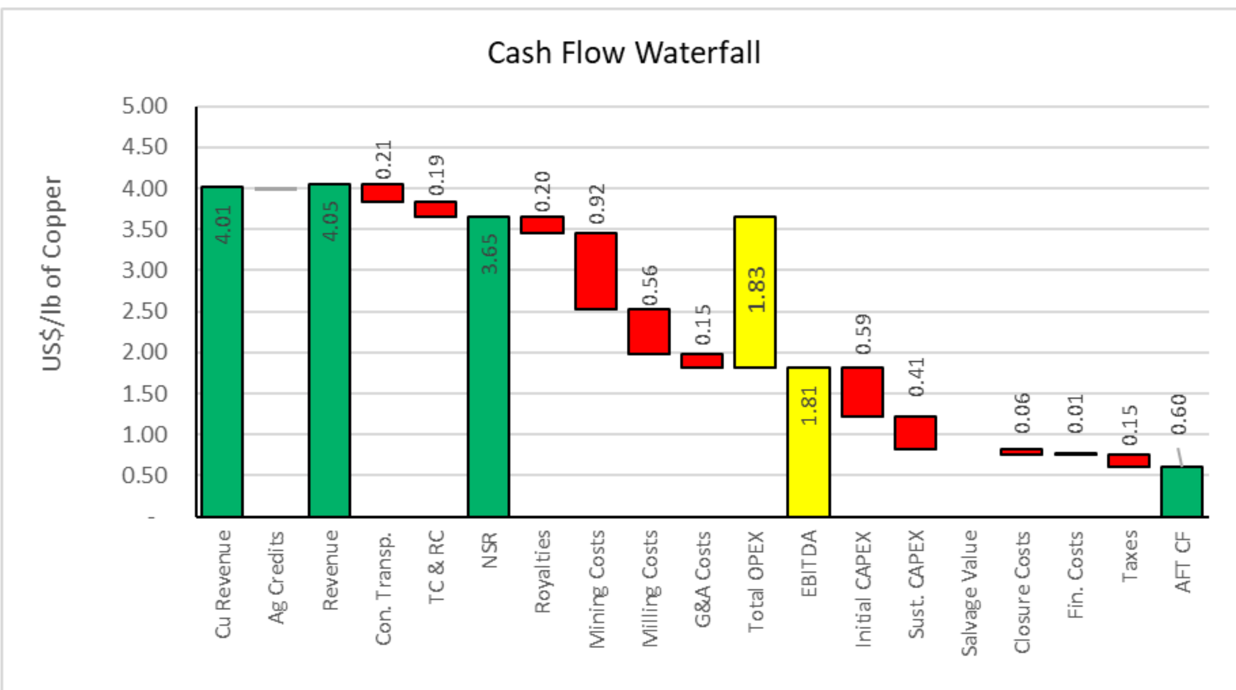



**Figure 1.6: After-Tax Project Cash Flow Waterfall**



## 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 a mixture of “owner managed” and EPCM methodologies. This will result in a mixed management team with both Highland and contracted personnel throughout the construction phase.

The Project team will manage and execute project engineering, procurement of project equipment and material, execute project construction, manage project control, ramp-up staff for start-up and operations, and coordinate the commissioning of the mine and process areas. Certain operations departments will be integrated in the project team early in the process to allow their parallel development and will focus on the project’s operational readiness.

The Project schedule milestones are:

- Start of detailed engineering: Q3 2023.
- Early works groundbreaking: Q3 2023.
- Box-cut completion: July 2024.
- Start of mine development: August 2024.
- TDF Phase 1 construction start: June 2024.
- Start process plant construction: June 2024.
- Powerline commissioning complete: December 2025.
- Plant commission start date: April 2026.

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 25.7 Mt allow for a 10.3 yr LoM (excluding commissioning and ramp-up) based on a 6,800 mtpd nominal milling rate. The mine design criteria are based on a geotechnical assessment completed by Golder in 2018.
- 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 amendment and renewals | Additional mineral reserves |
| Declining metal prices | Ground support design criteria and mining height |
| Ability to attract experienced professionals | Underground tailings disposal |
| Development and construction start date | Metallurgical recovery improvements & optimization of reagents consumption |
| Faults creating offsets to the mineralization | Ore sorting |
| Power line connection to grid | Copper concentrate leaching |
| Local housing and community infrastructure | Financial support of local authorities |

## 1.25 Recommendations

Based on the positive results of the FS, GMS 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 and long lead items 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.
- Review site water balance including construction schedule to optimize the precipitation and run-off water recovery.
- Detailed engineering of the tailings disposal facility and submittal for dam safety permit to construct.
- Initiate routing development of the powerline to site.
- 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

GMS was retained by Highland to lead and coordinate the update of a FS completed in 2018 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 FSU, as disclosed in Highland's press release entitled "*Highland Copper Announces Updated Feasibility Study Results for its Fully Permitted Copperwood Project in Michigan, USA*" dated March 06, 2023.

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

- The effective date of the Current Mineral Resource is February 28, 2022.
- The effective date of the Mineral Reserve is May 25, 2022.
- The effective date of this Report is March 06, 2023.

The project 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 includes the Satellite zones, but the resources estimated on the Satellite zones are not included in the mine plan or economic evaluation.

The FS update scope includes the following main aspects:

- Updated mine engineering, including mine design and production schedule.
- Revised plant design and support infrastructure.
- Power supply options evaluation.

- Surface water collection strategy
- 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 2018 NI 43-101 technical report and references cited in this report. Previous technical reports include Marston and Marston Inc. (now part of Golder) in March 2011 and Golder (now part of WSP) in 2014, in connection with the acquisition of CRI, which only reported historical estimates for the Copperwood Deposit.

GMS has sourced information from previous technical reports and appropriate reference documents as cited in the text and summarized in Section 27 of this Report. GMS 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 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 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 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 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.
- GMS issued a NI 43-101 technical report on 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 technical report had an effective date of April 15, 2015.
- GMS issued a NI 43-101 technical report on July 31, 2018, for Highland as a review of the Copperwood Project resources using then current market conditions and included recommendations of further work. This technical report had an effective date of June 14, 2018.

## 2.3 Qualifications and Experience

The major contributors for the Study and the Report and their respective areas of responsibility are as follows; Since the current report is an update of the 2018 technical report, the 2018 contributors are listed:

- GMS – 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.
- Lycopodium – flowsheet, 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 | Carl Michaud, P. Eng. | GMS | 1, 2, 3, 4, 5, 15, 16, 19, 21.1.4, 21.1.5, 21.1.8, 21.3, 21.4.3, 22, 23, 24, 25, 26 and 27. |
| 2 | James Purchase, P. Geo. | GMS | 6, 7, 8, 9, 10, 11, 12, 14. |
| 3 | Martin Houde, P. Eng. | GMS | 13, 17, 21.4.1 and 21.4.2. |
| 4 | Luc Binette, P. Eng. | GMS | 18, 21.1.1, 21.1.2, 21.1.3, 21.1.6, 21.1.7, 21.1.9 and 21.2. |
| 5 | Andrea K. Martin, P.E. | Foth | 20. |

## 2.4 Site Visits

Mr. James Purchase, P.Geo. met with Highland personnel, between November 6th and 9th, 2017, 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. New drill hole sites were examined, and review was undertaken of new drill core obtained in 2017.

Mr. Carl Michaud, P.Eng and Mr. Pong Mony Khuon, P.Eng of GMS 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, U.S. 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, GMS 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, GMS 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 |
|---|---|
| % | Percent |
| ° | Degrees (Azimuth or Dip) |
| °C | Degrees Celsius |
| 3D | Three Dimensional |
| Actlab | Activation Laboratories Ltd. |
| Ag | Silver |
| AX | AX Size Core; Core Diameter 3.01 cm |
| B lbs | Billion Pounds |
| 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 |
| CFM | Cubic foot per minute |
| Chesbrough | A.M. Chesbrough LLC |
| CIM | Canadian Institute of Mining Metallurgy and Petroleum |
| CIM | Canadian Institute of Mining Metallurgy and Petroleum |
| cm | Centimetre |
| CN | Canadian National |

| Abbreviations | Full Description |
|---|---|
| CoV | Coefficient of variation |
| CPG | Certified Professional Geologist |
| CRI | Copperwood Resources Inc. (formerly known as Orvana Resources U.S. Corp.), is a subsidiary of Highland Copper Company Inc. |
| CRM | Control Reference Material |
| CSA | Canadian Securities Administrators |
| CSF | Confinement Strength Factor |
| Cu | Copper |
| Dmt | Dry metric tonne |
| E | East |
| EIA | Environmental Impact Assessment |
| Eng | Engineering |
| ERP | Enterprise Resource Planning |
| FEMA | Federal Emergency Management Agency |
| Fe-O | Iron Oxide |
| FS | Feasibility Study |
| FSU | Feasibility Study Update |
| ft | Feet |
| G | Billion |
| g | Grams |
| G&A | General & Administration |
| g/t | Grams per Tonne |
| Ga | Billion years |
| GEOID03 | National Geodetic Survey Geoid 03 |
| GLGT | Great Lake Gas Transmission |
| GMS | G Mining Services Inc. |
| Golder | Golder Associates Ltd. |
| ha | Hectares |
| HDPE | High Density Polyethylene |
| Highland | Highland Copper Company Inc. |
| HQ | HQ Size Core; Core Diameter 6.35 cm |

| Abbreviations | Full Description |
|---|---|
| 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 |
| k/t | Kilogram per tonne |
| Kg | Kilogram |
| KLA | Keweenaw Minerals, LLC (formerly Keweenaw Land Association Limited) |
| km | Kilometre |
| $km^2$ | Square kilometre |
| kV | Kilovolt |
| l | Litre |
| LAN | Local Area Network |
| lb | Pound(s) |
| LCBS | Lower Copper Bearing Sequence |
| LCCS | Low Cost Country Sourcing |
| LHD | Load Haul Dump |
| LiDAR | Light Detection and Ranging |
| LLC | Limited Liability Company |
| LoM | Life of Mine |
| Lyco | Lycopodium Limited |
| m | Metre |
| m/d | Metres per day |
| MACRS | Modified Accelerated Cost Recovery System |
| mamsl | Metres above mean sea level |
| MDEQ | Michigan Department of Environmental Quality |
| MDNR | Michigan Department of Natural Resources |
| MDOT | Michigan Department of Transportation |
| METCON | Metcon Research |
| mm | Millimetre |

| Abbreviations | Full Description |
|---|---|
| MST | Nonferrous Metallic Minerals Extraction Severance Tax |
| Mt | Million Tonnes |
| Mtpa | Million tonnes per annum |
| Mtpd | Million tonnes per day |
| mtph | Metric tonnes per hour |
| N | North |
| NAD83 | North American Datum 1983 |
| NAVD88 | North American Vertical Datum 1988 |
| NCNST | North Country National Scenic Trail |
| 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 |
| OK | Ordinary Kriging |
| OPEX | Operating Expenditures |
| Orvana | Orvana Minerals Corp. |
| Osisko | Osisko Gold Royalties Ltd |
| PE | Professional Engineer |
| PMWSP | Porcupine Mountains Wilderness State Park |
| Project | Copperwood Project |
| QA/QC | Quality Assurance / Quality Control |
| QP | Qualified Person |
| R&P | Room and Pillar |
| REI | Resource Exploration Inc |
| S | South |
| Sage | Sage Minerals Inc. |

| Abbreviations | Full Description |
|---|---|
| SG | Specific Gravity |
| SGCN | Michigan Species of Greatest Conservation Need |
| SGS | SGS Lakefield |
| t | Tonnes |
| TC/RC | Transportation Costs & Smelter Conversion Charges |
| TDF | Tailings Dam Facility |
| TDM | Tailings & Water Disposal Management |
| tpa/tpy | Tonnes per annum |
| tpd | Tonnes per day |
| TSF | Tailings Storage Facility |
| UCBS | Upper Copper Bearing Sequence |
| USA | United States of America |
| USD | United States Dollars |
| USFWS | U.S. Fish and Wildlife Service |
| USG | U.S. Gallon |
| USGPM | U.S. Gallon per minute |
| USMR | United States Metals Refining Company |
| UTM | Universal Transverse Mercator |
| W | West |
| WBS | Work Breakdown Schedule |
| WC | Working Capital |
| wt.% | Weight Percent |
| WTP | Water Treatment Plant |
| WWTP | Wastewater Treatment Plant |
| y | Year |
| µm | Micron |

# 3. RELIANCE ON OTHER EXPERTS

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

- Information available to GMS 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 GMS has relied:

- GMS has relied on information prepared by Lycopodium pertaining to the design of the Process Plant;
- GMS has relied on information prepared by Golder pertaining to the Tailings Disposal Facility (TDF) and the Water Balance Model;
- GMS relied on information prepared by Golder regarding underground mining geotechnics and ground support recommendation;
- GMS has relied upon information provided by Highland including lease agreements and legal opinions concerning Highland's mineral and surface rights prepared by Kendricks, Bordeau, Keefe, Seavoy & Larsen, P.C., a Michigan law firm;
- GMS has relied on information supplied by Highland pertaining to the concentrate transportation costs and the selection of a trainload facility location;
- GMS has relied on input from KPMG LLP regarding the taxation model and estimates used to estimate after-tax cash flows in the economic model;
- GMS has relied on geotechnical input from the GeoEngineering North, LLC report; Drilling and sampling activities were supervised by Dr. Stanley Vitton of Michigan Technological University;

- GMS has relied on Coleman Engineering from Michigan for wetland area surveys, surface water drainage design, the water intake design, the County Road 519N upgrade and capital cost estimate pertaining to the water intake and County Road 519N upgrade;
- GMS has relied on construction rates obtained by HCC from local contractors, as pertains to bulk materials and manhour rates; this information was used in the capital cost estimate of the project.

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 GMS.

# 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 $mi^2$ (93 $km^2$) comprising 6 x 6 arrays of 36 sections, named according to distance and direction from a principal meridian and baseline. Sections are 1 $mi^2$ (2.6 $km^2$), 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 1,904 contiguous hectares under three 20-year lease agreements and one 30-year mineral lease agreement as summarized in Table 4.1. The sections, surveyed as part of the Public Lands Survey System, are identified at corners with federal monuments established pursuant to orders and instructions issued by the United States government. These original public land survey corners are now being protected and perpetuated by the State of Michigan under the Corner Recordation Act 74 of 1970 and the State Survey and Remonumentation Act 345 of 1990. 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**

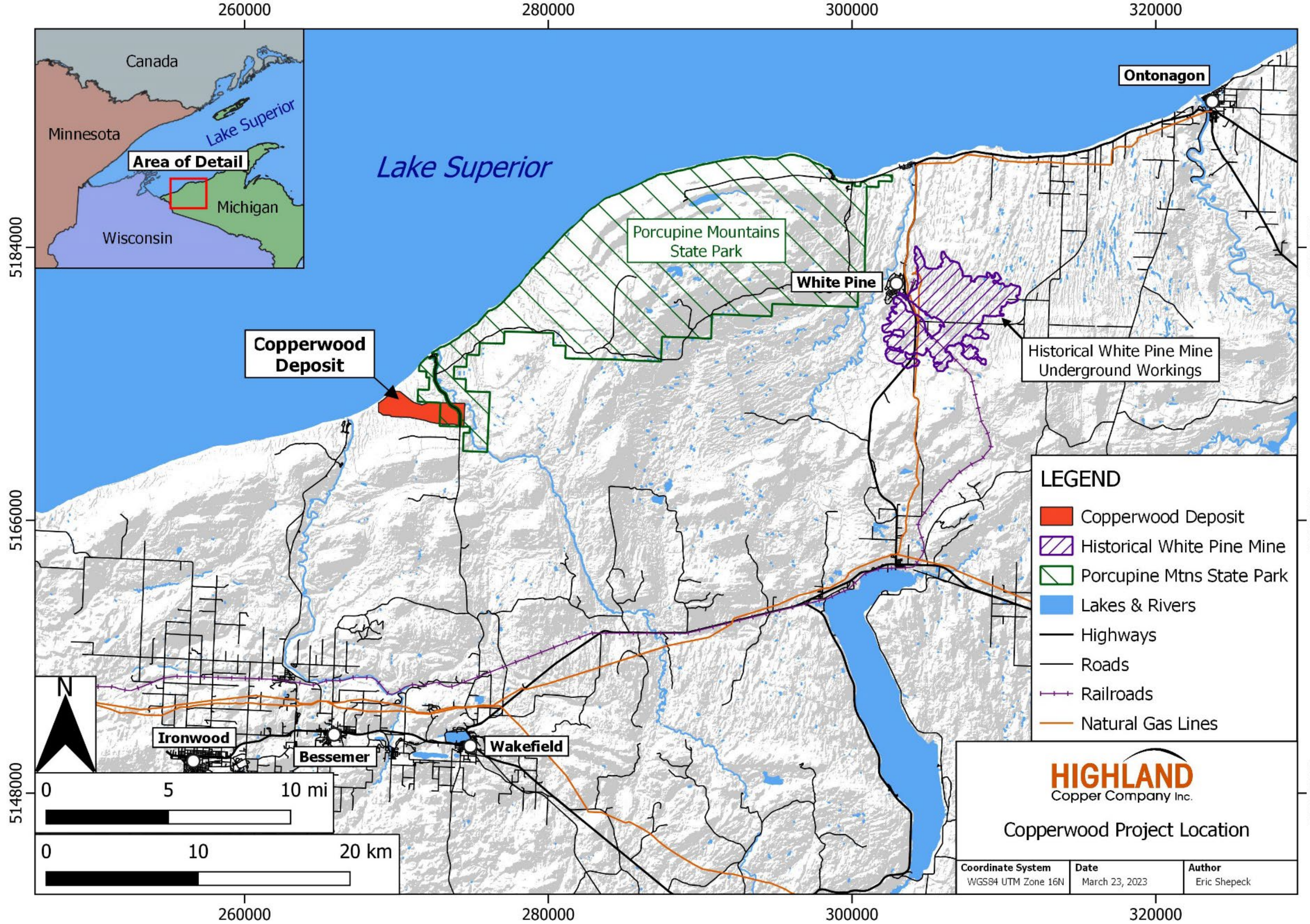


**Table 4.1: Copperwood Mineral Tenure**

| 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 (fractions) | 595 | Option to Lease ending in 2028 |

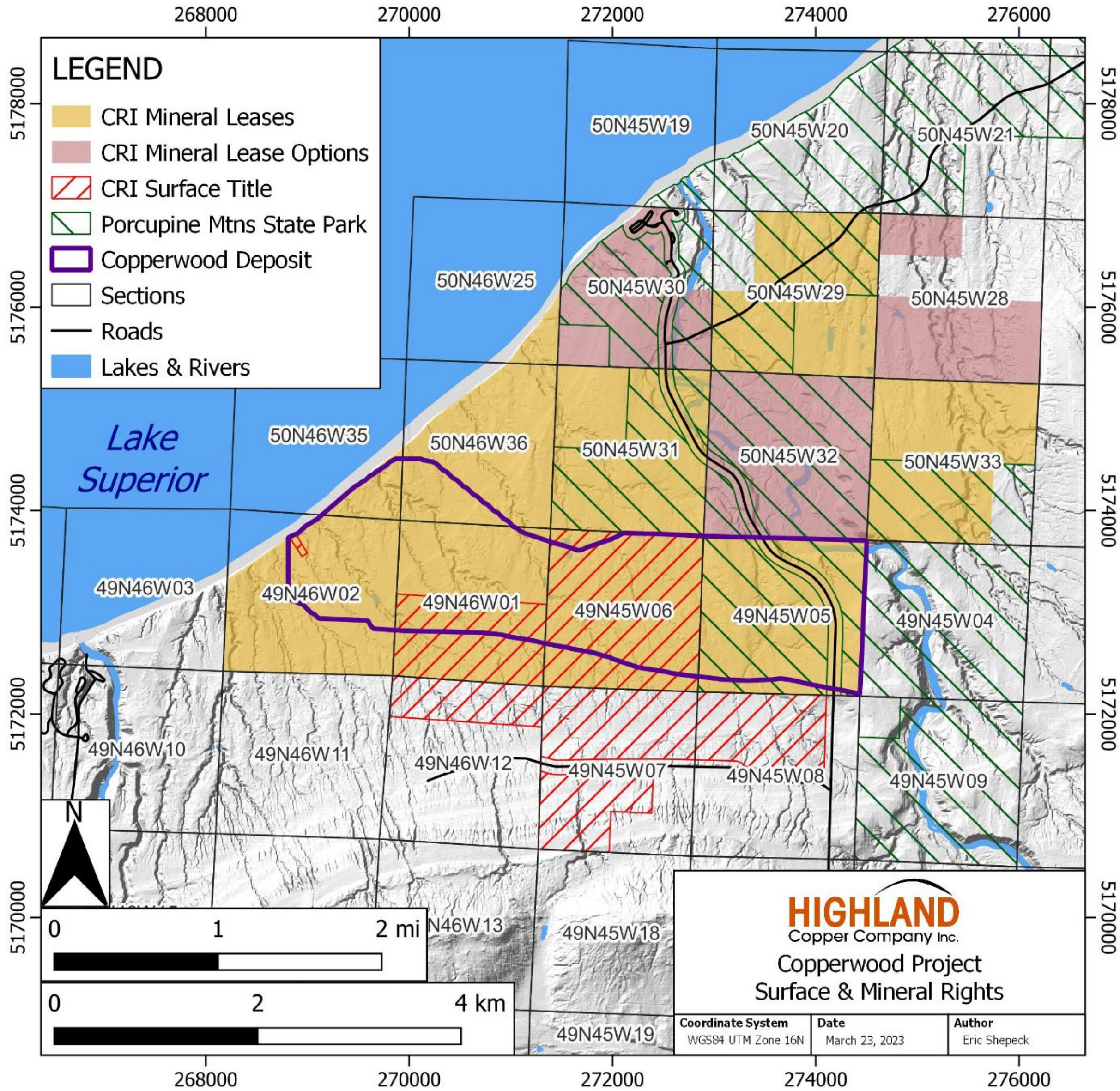
## 4.3 Surface Rights

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 underground 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, Ironwood Township, Gogebic County
- Section 35, Township 50 North, Range 46 West, Ironwood 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, Ironwood Township, Gogebic County
- Section 36, Township 50 North, Range 46 West, Ironwood Township, Gogebic County

Sage assigned its rights, title and interest in this Mining Lease to KLA pursuant to an agreement dated effective September 21, 2021.

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

- 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 4.4.

In addition to the lease payments, CRI must pay to the mineral right owners (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 Mining Lease Payment Schedules**

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

**Table 4.3: Chesbrough 2010 Mining Lease Payment Schedule**

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

**Table 4.4: KLA 2017 Mining Lease Payment Schedule**

| Date | Amount (USD) |
|---|---|
| Commencement Date | 35,000 |
| 1st Anniversary of Commencement Date | 52,500 |
| 2nd Anniversary of Commencement Date | 70,000 |
| 3rd Anniversary of Commencement Date | 87,500 |
| 4th Anniversary of Commencement Date | 105,000 |
| 5th through 10th Anniversaries of Commencement Date | 140,000 |
| 11th through 15th Anniversaries of Commencement Date | 175,000 |
| 16th 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 established on the Producer Price Index – Finished Goods, prepared by the USA Department of Labour. Benchmark prices are initially set at USD 2/lb Cu and USD 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 was part of a lease-option agreement with Sage covering approximately 595 ha located within Wakefield Township, Gogebic County, Michigan, with an effective date of October 16, 2008. Sage transferred its rights, title and interest in the option to lease to KLA pursuant to an agreement dated September 21, 2021. 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 (USD) |
|---|---|
| On Effective Date | 6.18/ha |
| On 1st Through 5th Anniversaries of Effective Date | 6.18/ha |
| On 6th Through 10th Anniversaries of Effective Date | 12.36/ha |
| On 11th Through 15th Anniversaries of Effective Date | 18.53/ha |
| On 16th and Later Anniversaries of Effective Date | 24.71/ha |

CRI has the right to exercise the option to lease at any time before October 16, 2028, 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

Osisko holds a 1.5% NSR royalty on the copper produced from the mineral rights and leases comprising the Copperwood Project.

In 2014, the Company had granted Osisko an option to purchase 100% of future silver production from the Copperwood and White Pine projects for a total consideration of USD 26 million. In 2021, the terms and conditions of this option were modified, and the Company granted Osisko a 3/26th (~11.5%) royalty on future silver production from the Copperwood Project and the White Pine Project in consideration of USD 3 million (the “Initial Payment”). Osisko has the option to acquire the remaining 23/26th royalty on all future silver production from the Copperwood and White Pine projects by paying an additional USD 23 million to the Company within 60 days following the delivery of a feasibility study on the White Pine Project.

## 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 Environment, Great Lakes and Energy (EGLE), formerly known as 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 Natural Resource and Environmental Protection Act (NREPA), Public Act 451 of 1994 as amended. 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 freshwater 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 EGLE. A well is considered abandoned if it is inactive for one year, unless an extension is granted by EGLE 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 are 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 NREPA. 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 EGLE to conduct any necessary reclamation and remediation measures and must be updated at least every three years. Funds to cover the costs for EGLE 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, now known as EGLE:

- April 30, 2012 – Part 632 Mining Permit for Copperwood Project, Upper Peninsula, Michigan, USA.
- February 7, 2013 – Amendment #1 for provision of a no subsidence mine plan.
- December 14, 2018 – Amendment #2 for changes in infrastructure, mining and processing plans in the June 2018 Feasibility Study Report by G Mining Services.
- November 13, 2012 - National Pollutant Discharge Elimination System (NPDES) permits for treated sanitary and process wastewater related to the proposed Copperwood copper mine, Upper Peninsula, Michigan, USA. Permit expiration date was October 1, 2017.
- NPDES Renewal application submitted September 17, 2017, draft permit public noticed July 16, 2022, a new permit was issued effective May 1, 2022, by EGLE with an expiration date of October 1, 2024.
- November 26, 2018– Air Quality Division Permit to Install 180-11.

- April 20, 2020, Extension #1 – 18 months to begin construction.
- December 22, 2021, Extension #2 – additional 18 months to begin construction. Expires May 26, 2023.
- February 28, 2023, Extension #3 – A letter was sent to EGLE requesting a third extension until October 16, 2023 (expiration date of the Wetlands, Lakes and Streams permit).
- October 16, 2018 - Wetlands Part 303, Inland's Lakes and Streams Part 301, and Great Lakes Submerged Lands Part 325 permits for the proposed Copperwood copper mine. Expiration date is October 16, 2023.
- November 9, 2018 – Part 315 Dam Safety Permit.
- Extensions required every two (2) years for a Part 315 Permit to remain active. Extension #1, approved October 9, 2020, that expired November 9, 2022. A second extension was approved by EGLE until October 16, 2023.

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 of 1,700 and 5,050 respectively.

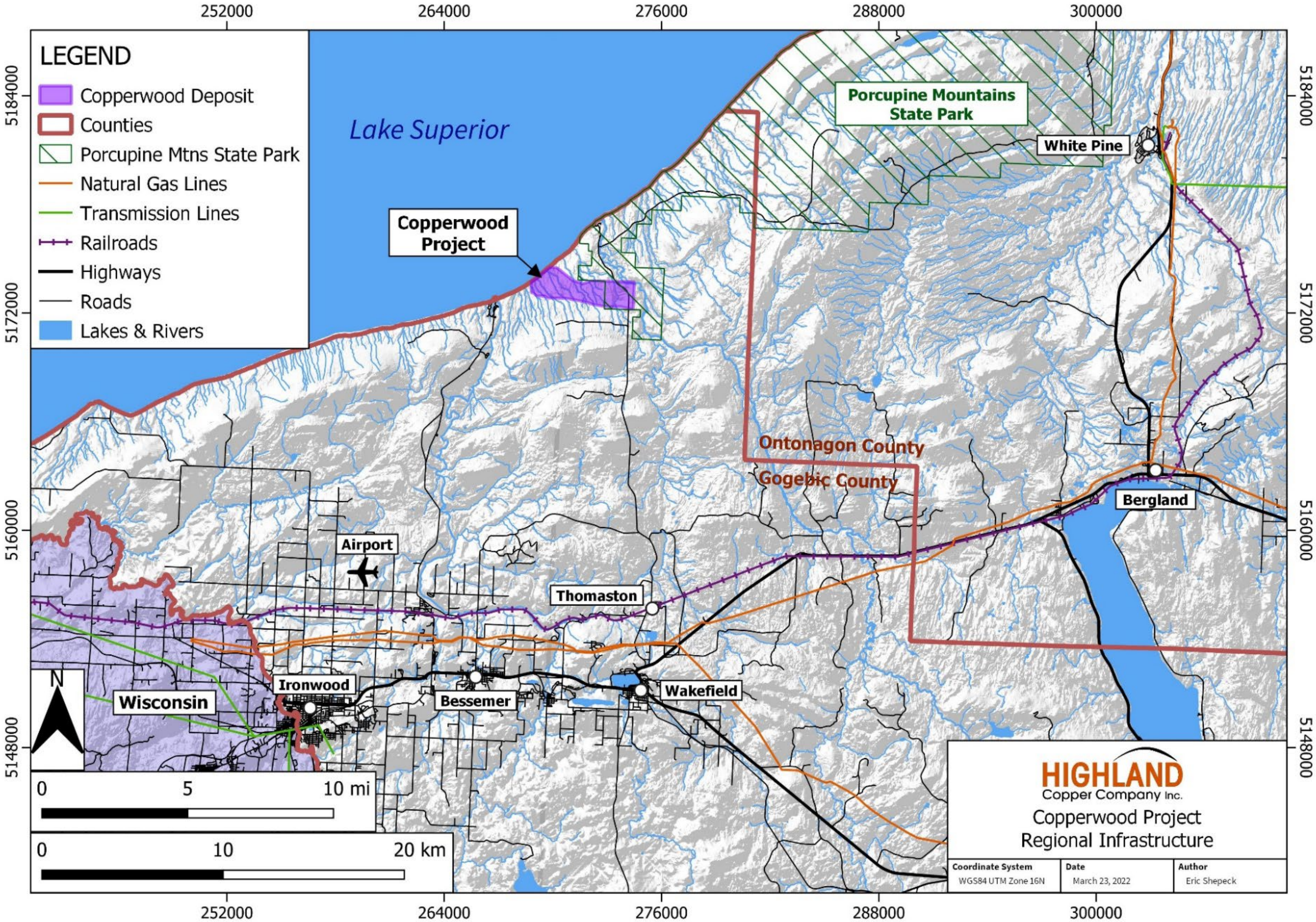
The main access to the Copperwood Project property is by way of the paved north-south County Road 519, which branches off State Highway M-28, just east of Wakefield. The Project property is transected by a series of dirt roads and drill trails allowing access for exploration activities.

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/h. 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. This rail section is currently out of service. Furthermore, Watco Rail Services has completed the acquisition of the rail network which runs through Thomaston from the CN. Major works would be required to recommission the rail network in and around Thomaston. 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 was at 6.4% in Gogebic County in January 2023; 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**

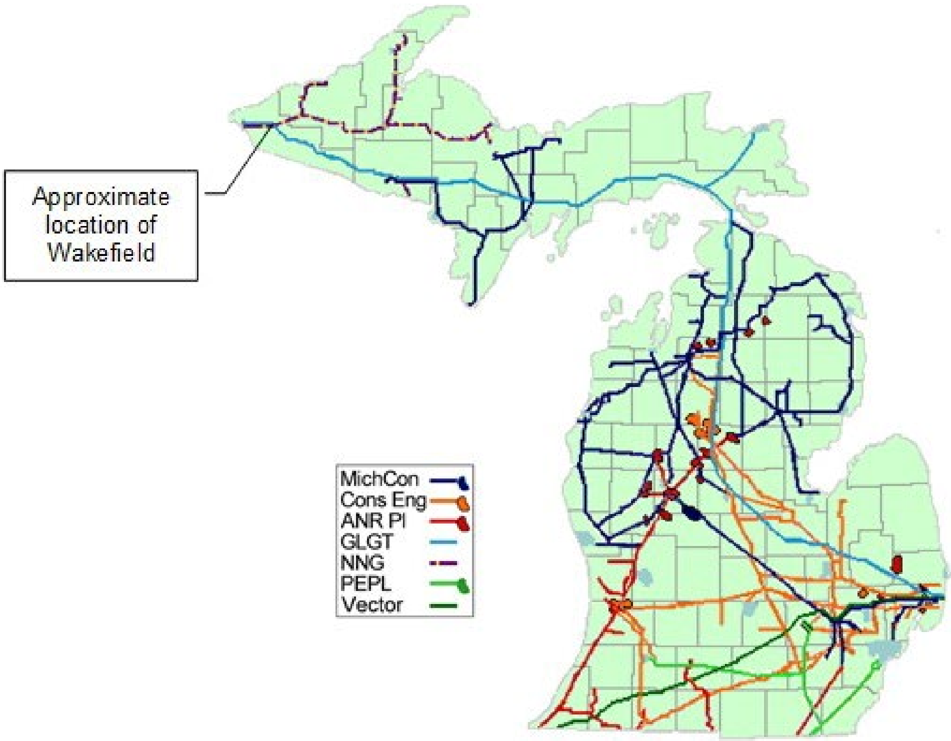
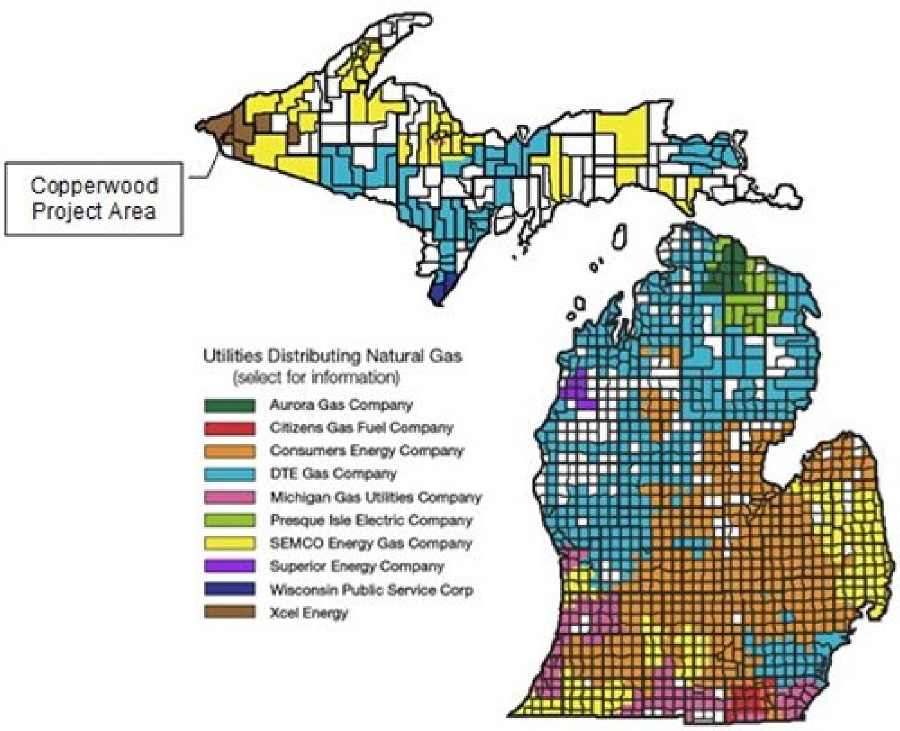



**Figure 5.3: Michigan Gas Utility Service Areas**



There are no aquifers beneath the surface of the property capable of yielding sufficient water for the process plant by themselves. Potable water will come from a well, through a potable water treatment plant. Process water for any planned mining operation will come from the collection of precipitation and run-off water from the full TDF surface. However, during normal course operations, it is planned to recycle water from the tailings disposal facility back to the process plant.

Current site communication is comprised of cell phone services available via a repeater tower at the Indianhead ski resort, located approximately 19 km away.

The Copperwood Project property and surface rights are 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, topsoil, 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. The Porcupine Mountains Wilderness State Park is located to the immediate east and north of the project area.

# 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 1960s. | 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 (5) 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 Copperwood | Drilled 40 holes and 13 wedges for resource estimate, metallurgical and geotechnical studies. | 2017 |
| Highland Copperwood | 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 the Presque Isle Syncline in 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. A total of 135 holes were then completed at 660 or 330 m spacing. Forty-two (42) 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 (3) 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 Copperwood carried out a drilling program comprising of 35 HQ diameter (plus 13 wedges) and five (5) 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 Copperwood completed a drilling program of eight (8) NQ-diameter holes and one (1) wedge as well as finishing one (1) 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. Overall, 14 drill holes were completed in bedrock above the copper-bearing interval and six (6) 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 (8) reject samples of mineralization, hanging wall, and footwall rocks from three (3) historical drill holes. Interpretation of the geochemical test results by Geochimica, Inc. indicates that Copperwood rocks are unlikely to be acid generating. Michigan's nonferrous mining law, however, defines "Reactive" as being susceptible to reacting, dissolving, or otherwise forming a leachate that is or may be harmful to the environment or to human health and safety under conditions that may exist at a mining operation. Based on this definition, ore, tailings and bedrock are reactive as the test program demonstrated that water in contact with them under oxidizing conditions has the potential for certain constituents to reach concentrations too high for discharge to the environment without treatment. In addition to the geochemical test program, 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-acid generating.

## 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 a 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 Copperwood (GMS) – Copperwood Deposit, published June 25, 2015, effective date of April 15, 2015.
- Highland Copperwood (GMS) – Copperwood Deposit, published December 5, 2017, effective date of October 18, 2017.
- Highland Copperwood (GMS) – Copperwood Deposit, effective date of April 30, 2018.

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 (5) 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 |

| Historical Resource Estimates | Tonnage (Mt) | Copper Grade (wt.%) | Copper Cut-off (wt.%) | Minimum Thickness (m) |
|---|---|---|---|---|
| **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 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 Copperwood – GMS Resource Estimate

In April 2015, GMS 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 GMS' Brossard Office, Quebec, Canada. GMS 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/cm$^3$ was used for all rock sequences in the model. Copper and silver grades were estimated using the Ordinary Kriging interpolation method in three (3) successive passes, using ellipse ranges of 175 m, 250 m, and 350 m.

To define resource categories, GMS 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 USD 3.00/lb and a silver price of USD 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 USD 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 USD 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, P. Eng., Senior Advisor, Geology and Resources for GMS. 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 Copperwood – GMS Resource Update October 2017

In October 2017, GMS 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 (2) 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 (1) 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/cm$^3$ was applied to Domino and Red Massive units, and 2.72 g/cm$^3$ 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, GMS 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 USD 3.00/lb and a silver price of USD 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 USD 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, P.Eng.,Senior Advisor, Geology and Resources for GMS. 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.

### 6.4.6 Highland Copperwood – GMS Resource Update April 2018

In 2018, Highland Copperwood completed a drilling program of eight (8) NQ-diameter holes and one (1) wedge, as well as finishing one (1) 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 from Inferred to Indicated category for inclusion in the 2018 feasibility study. The block estimate was completed following similar steps to the October 2017 Mineral Resource, and the tabulation can be found in Table 6.7.

**Table 6.7: Mineral Resource Estimate - Copperwood Project 1.0% Cu Cut-off Grade – April 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 USD 3.00/lb and a silver price of USD 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) *A sliding scale 3.0% NSR royalty on the Copperwood Project is applicable at USD 3.00/lb Cu price with Osisko Gold Royalties Ltd. Upon closing of the acquisition of the White Pine Project, Highland Copper Company will grant Osisko a 1.5% NSR royalty on all metals produced from the White Pine project, and Osisko's royalty on Copperwood will be reduced to 1.5%.*
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., Senior Advisor - Geology and Resources for GMS. The estimate has an effective date of April 30th, 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.*

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# 7. GEOLOGICAL HISTORY 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

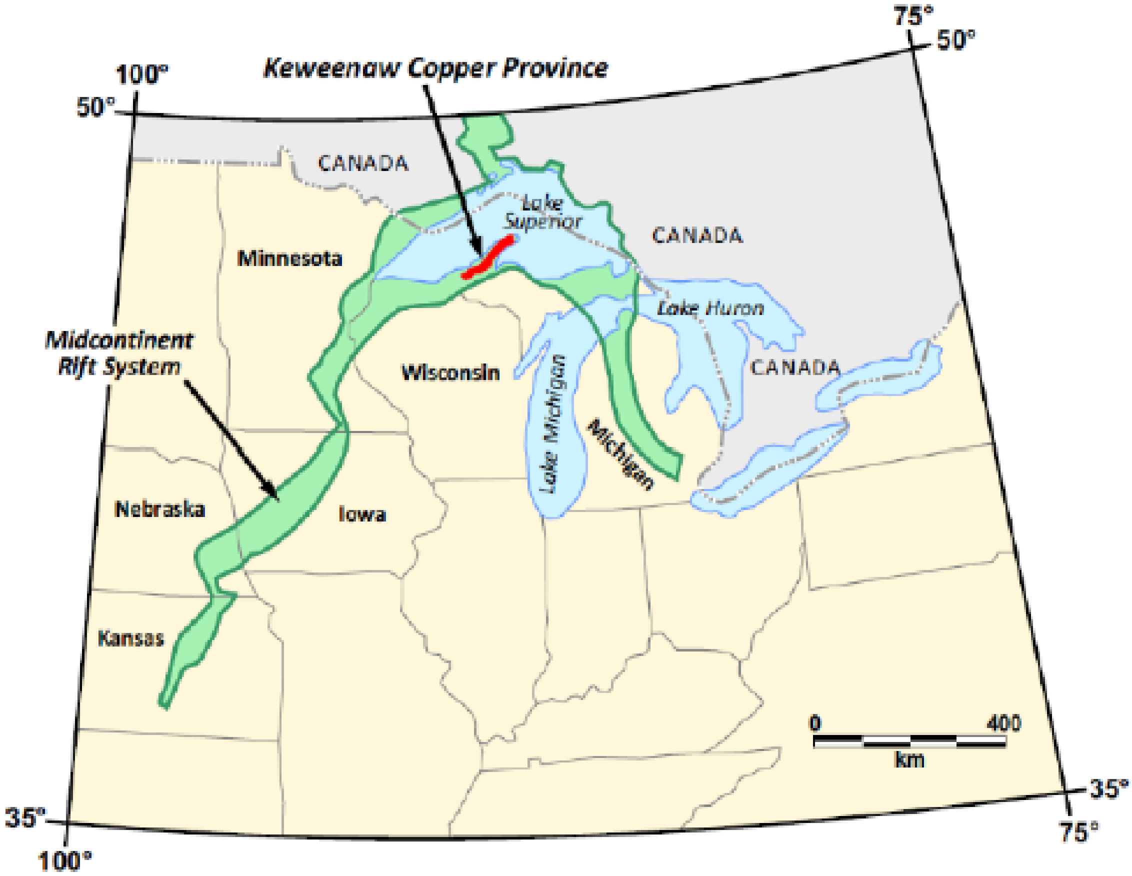
The Copperwood Project area is situated along the southeast flank of the 2,200 km long Mesoproterozoic mid-continent rift system of North America, within the Keweenaw Copper province as shown in Figure 7.1. The rocks of this rift system consist of a package of volcanic and clastic sedimentary rocks that are up to 30 km thick called the Keweenawan 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 Oak Bluff Formation lies 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 historical White Pine mine, as shown in Figure 7.3.

The Copper Harbor Formation is composed of red-brown conglomerates and sandstones with lesser siltstone as 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-grey-to-green-red siltstone and shale with minor carbonate laminates, and sandstone lenses up to 30 m thick. Black to dark-grey 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**

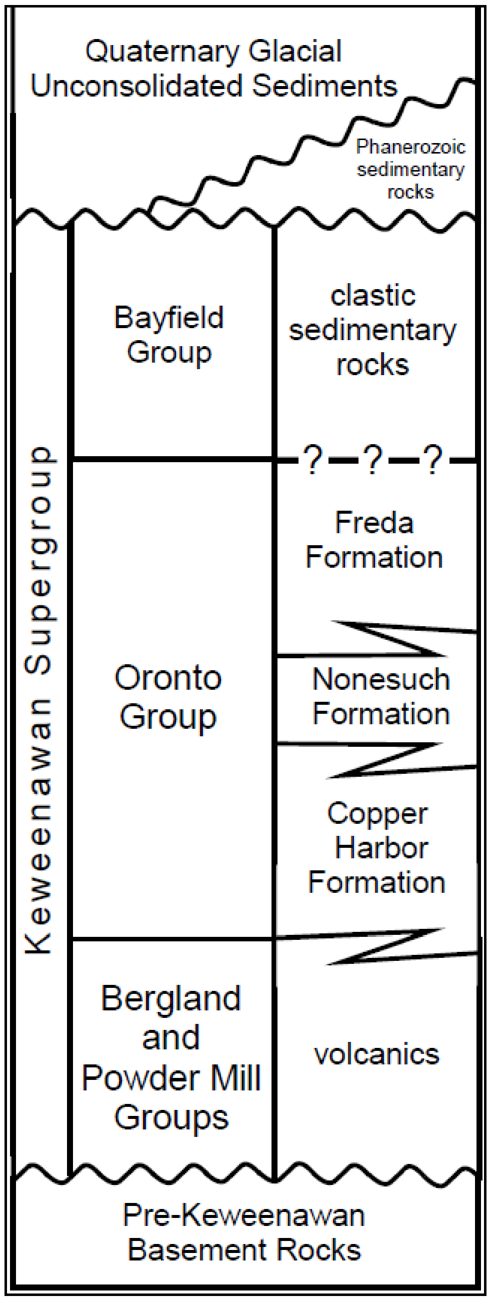
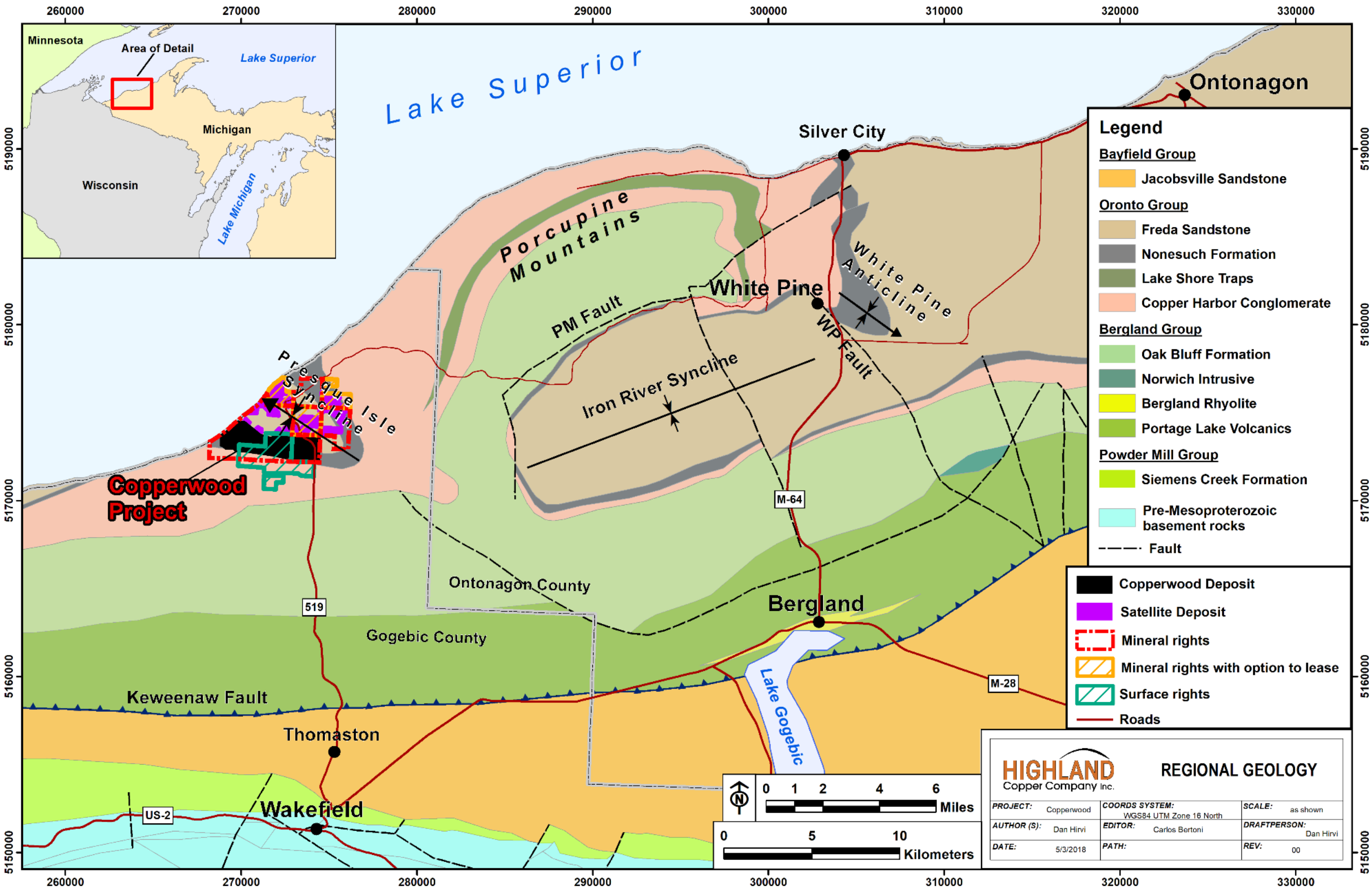


**Figure 7.3: Regional Geology and Project Location**



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 copper mineralization in the volcanic-dominated strata of the Bergland Group and 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 eroded by Pleistocene continental glaciers beginning about two million years ago, uncovering the Precambrian rocks of the western Upper Peninsula.

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 the 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 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 commonly 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’s center, which was a topographic high during 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 grey-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 Lower Copper Bearing Sequence (LCBS). The LCBS is a group of subunits of the Nonesuch Formation that hosts the bulk of the copper and silver mineralization at Copperwood. The Upper Copper Bearing Sequence (UCBS) is a second group of subunits that contains 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. This separation gradually decreases 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 lacking copper mineralization.

### 7.2.1 Lower Copper Bearing Sequence

The LCBS at the Copperwood Deposit is subdivided into the Domino, Red Massive and Grey 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 grey 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 grey 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**

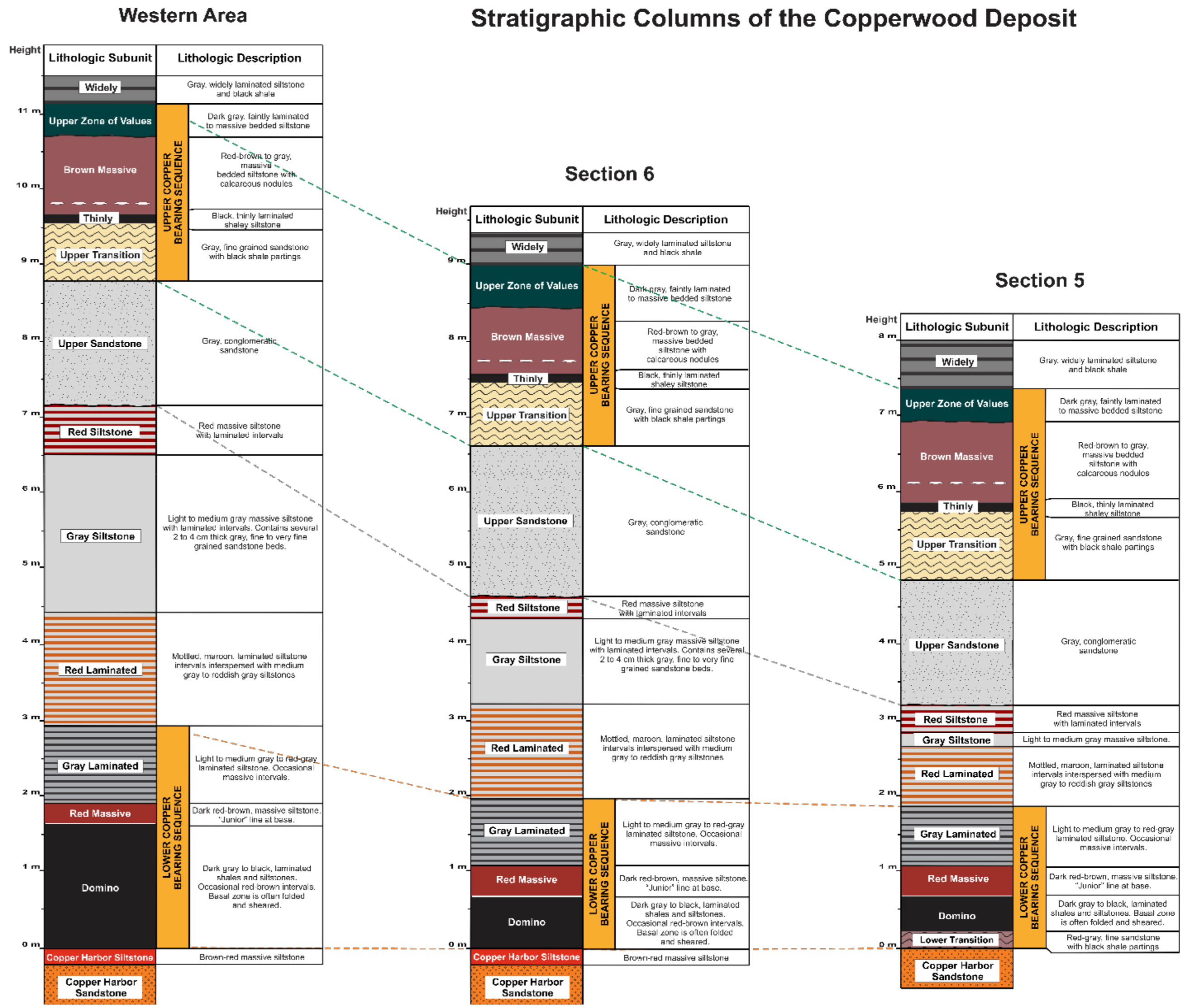
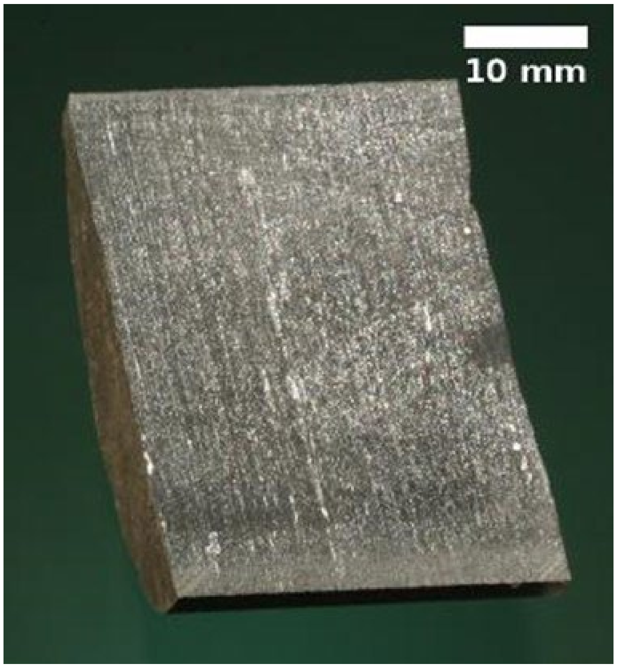


**Figure 7.5: Mineralized Domino Subunit Drill Core Sample**




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-grey 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-grey. The upper contact is placed where the color changes from reddish grey to grey. 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 Grey Laminated subunit contact with the underlying Red Massive is gradational. This subunit consists of light to medium-grey to reddish-grey, laminated to locally massive siltstone. Brownish layers are occasionally present in parts of the Grey Laminated interval. A 10 to 50 cm thick zone of calcareous nodules in grey siltstone occurs in all holes near the base of Grey Laminated. The upper contact is placed where the color changes from dominantly grey to mixed maroon and grey. The transition zone is typically on the order of 0.1 m thick. The Grey 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, Grey 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 Grey Laminated. This subunit is characterized by laminated siltstone with a bimodal color distribution of maroon to red-brown and grey. Typical Red Laminated has mottled or wavy maroon intervals interspersed with medium grey to reddish grey 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 Grey Siltstone and Red Siltstone subunits overlie the Red Laminated. The Grey Siltstone consists of a laminated, light and dark grey siltstone. The Red Siltstone is a red-grey 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 grey siltstones and sandstones, with minor grey conglomeratic, white sandstone and red-brown siltstone lenses.

### 7.2.2 Upper Copper Bearing Sequence

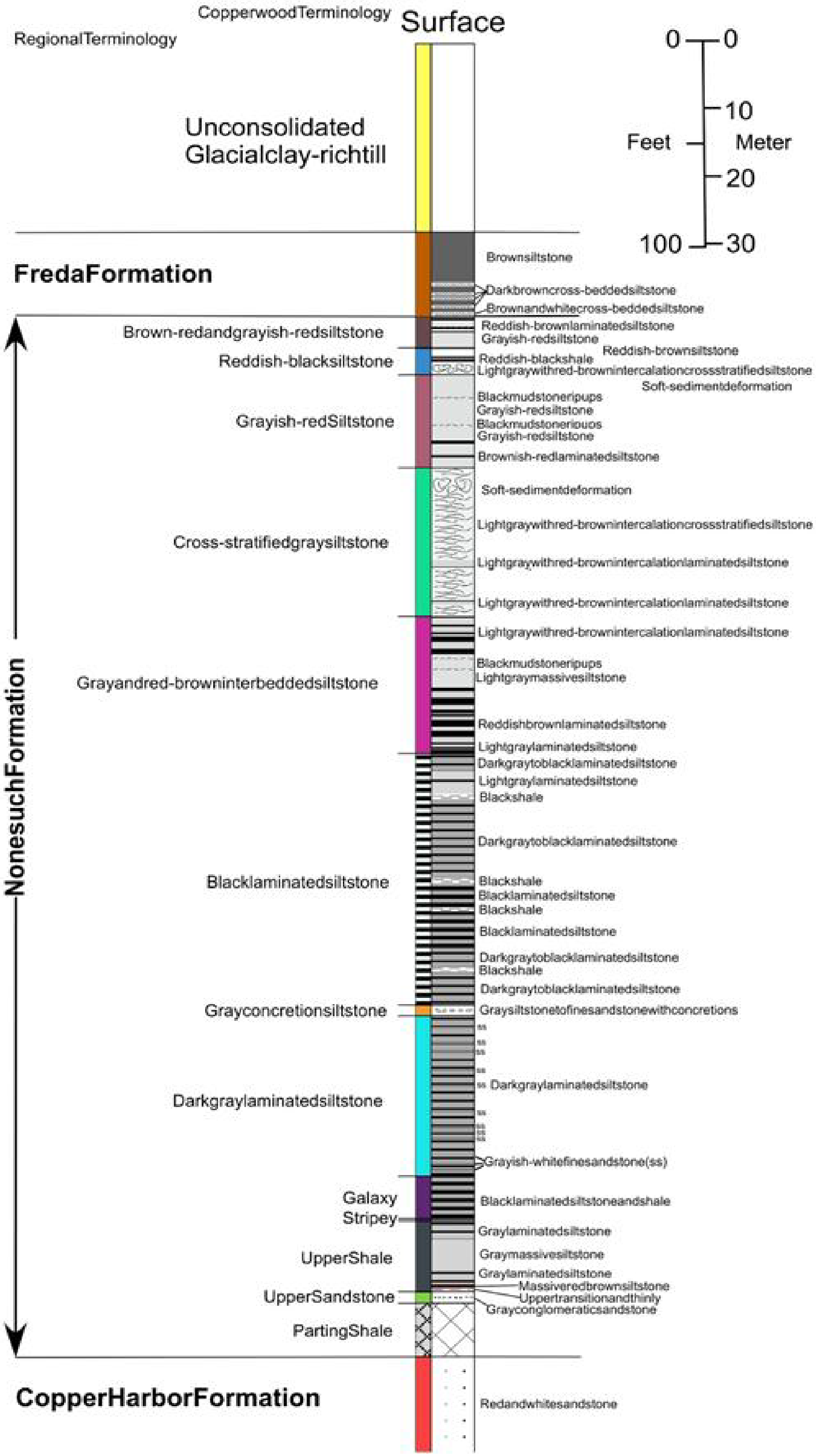
The UCBS, which lies above the Upper Sandstone subunit, is comprised of the following subunits: Upper Transition, Thinly, Brown Massive and Upper Zone of Values.

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 intermittent 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 grey 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. The average vertical displacement is 4.8 m (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. Drill holes CW-09-82 and CW-17-186 are the 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**

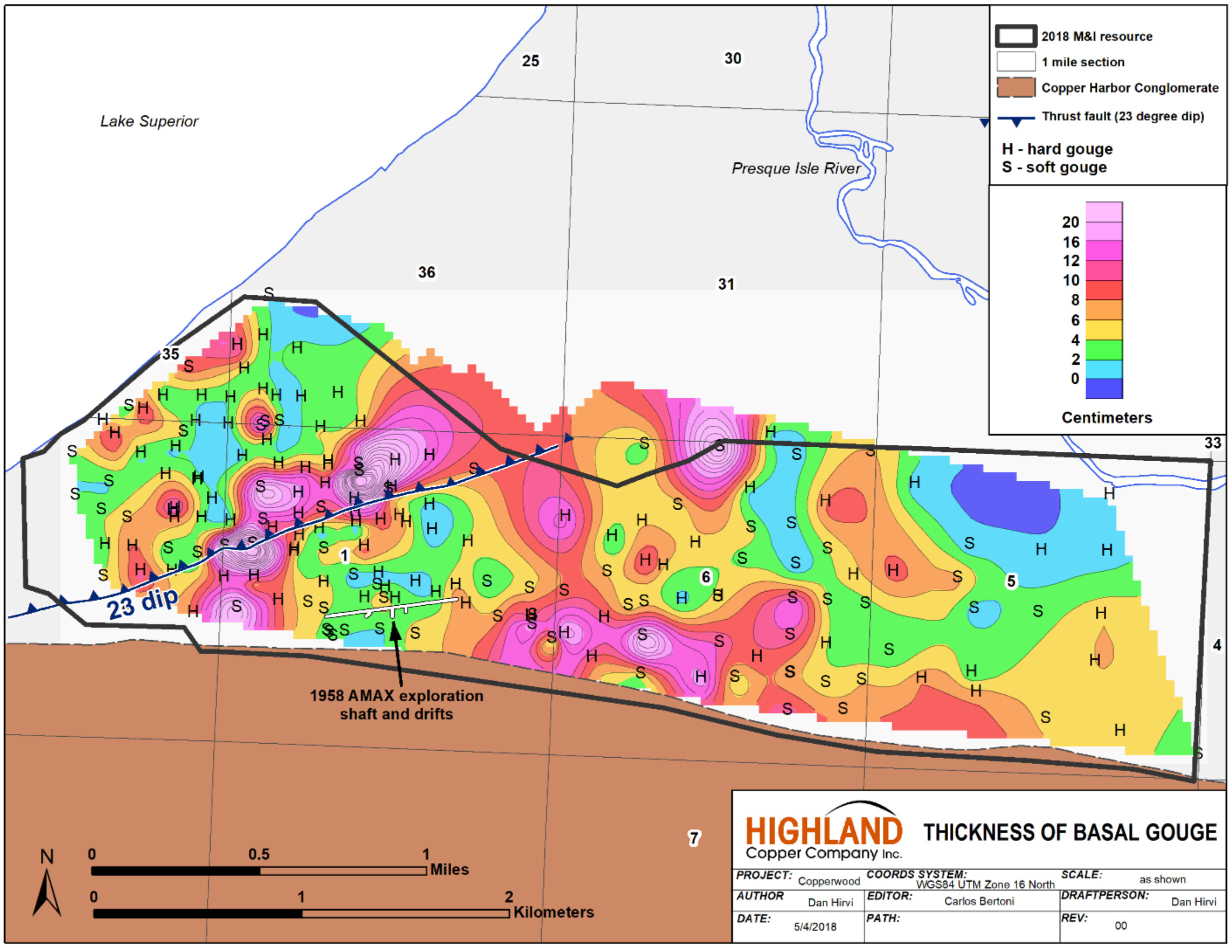


**Figure 7.8: Cross Section Showing the LCBS – Southwest-Northeast Fence Diagram – Western Copperwood Deposit**

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

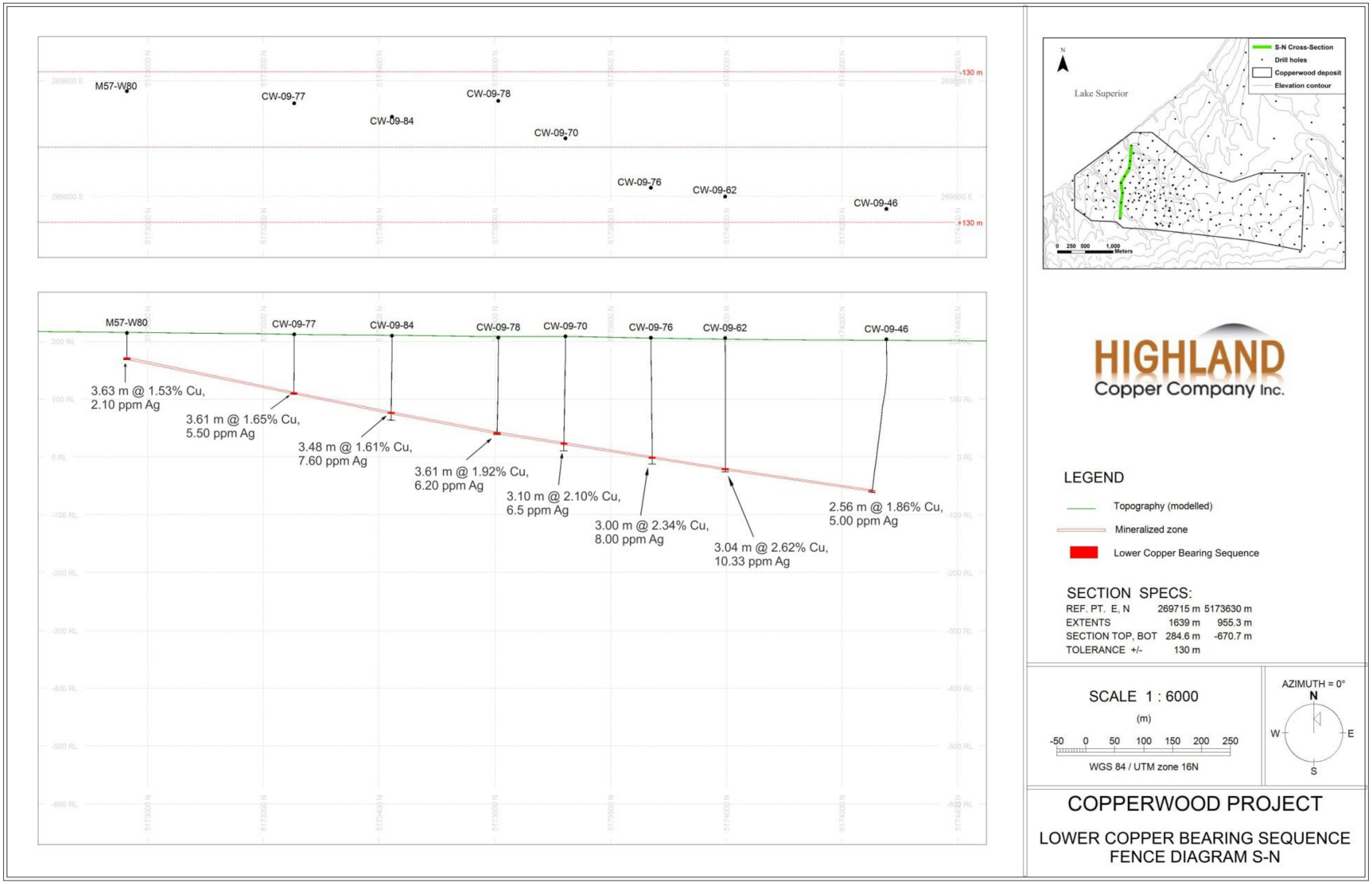


**Figure 7.10: Cross Section Showing the LCBS – South-North Fence Diagram – Central Copperwood Deposit**

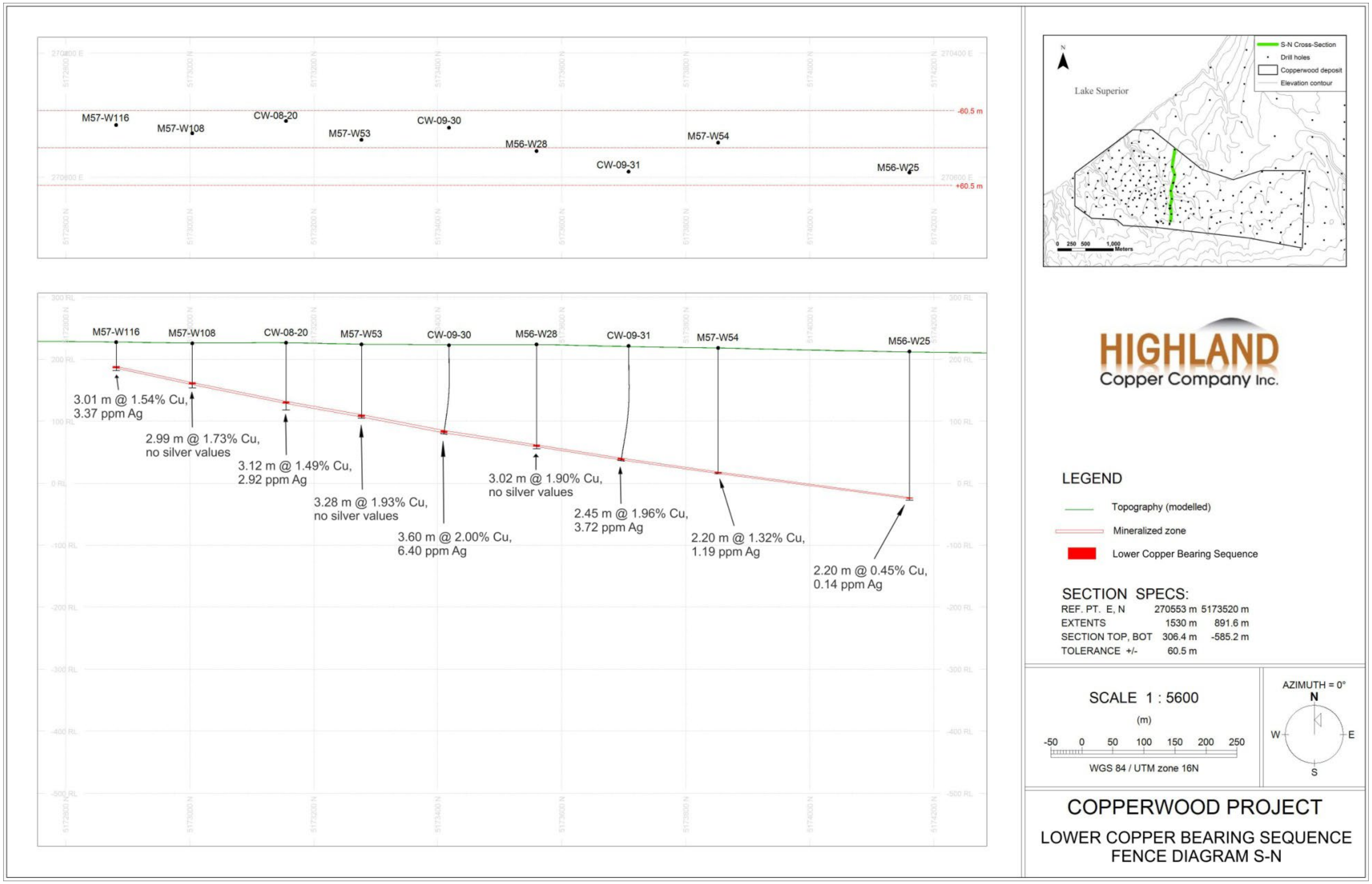


**Figure 7.11: Cross Section Showing the LCBS – South-North Fence Diagram – East Copperwood and Satellite Deposits**

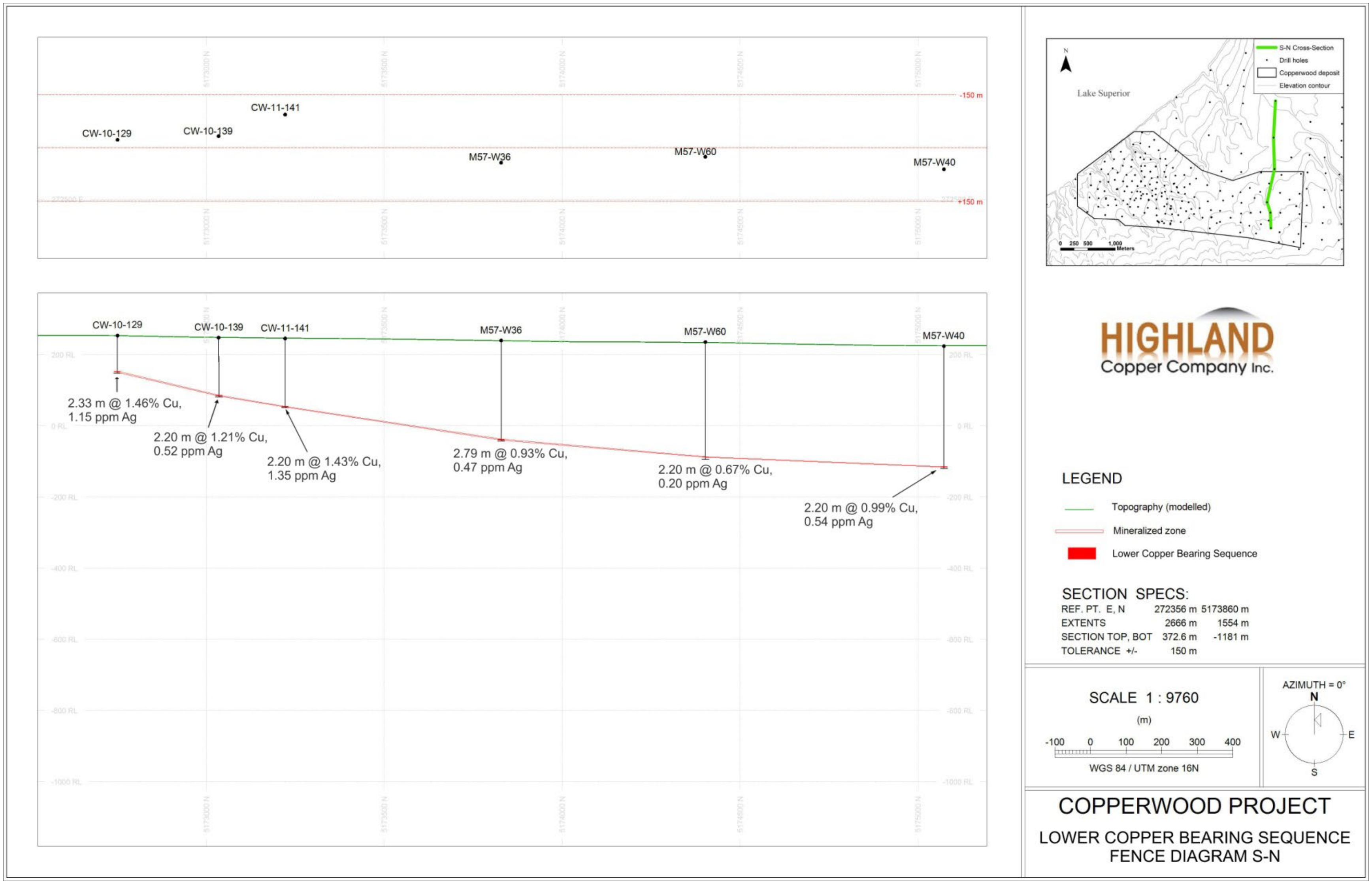
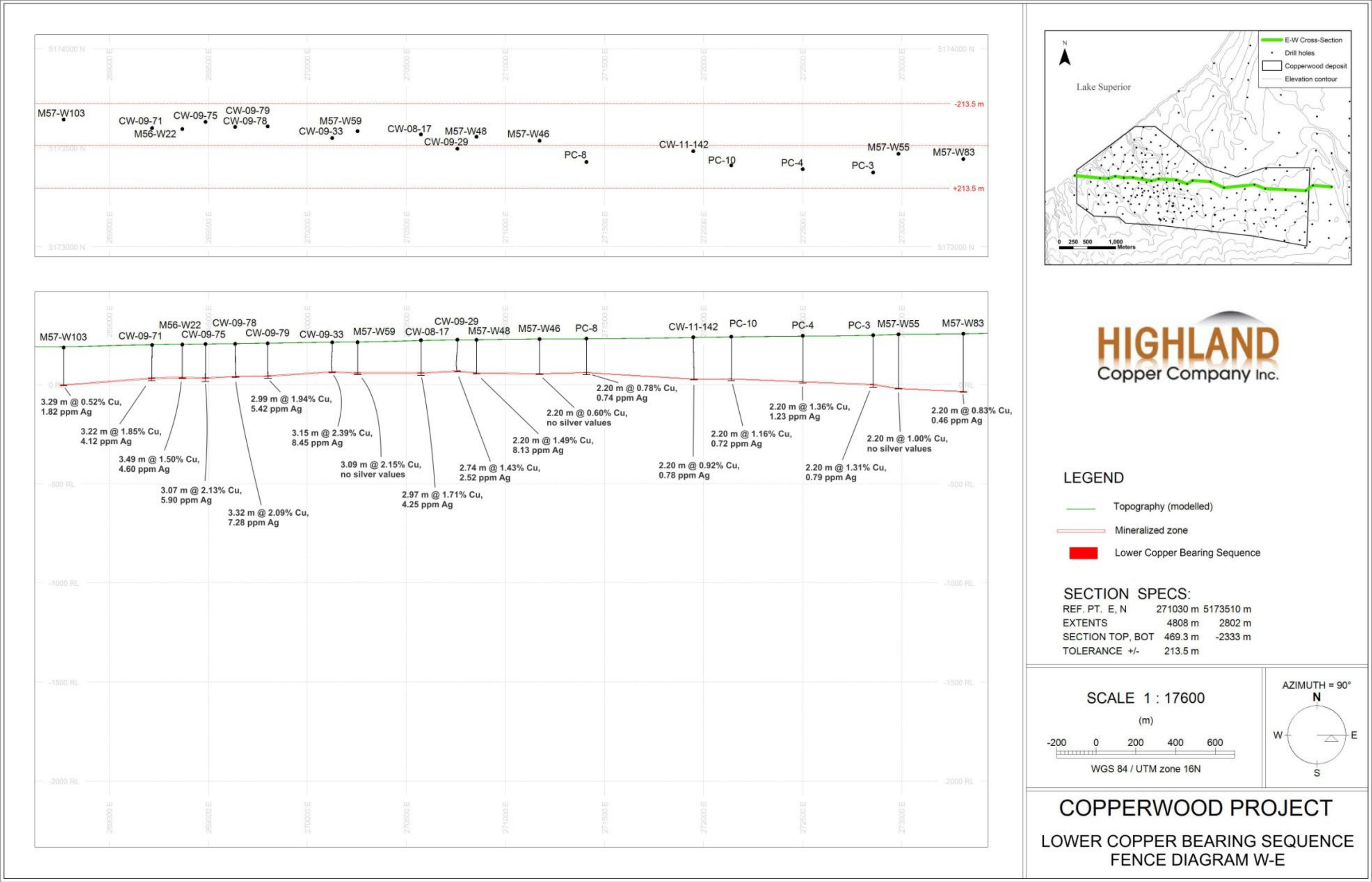


**Figure 7.12: Longitudinal Section Showing the LCBS – West-East Fence Diagram – Copperwood and Satellite Deposits**



## 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 (LCBS), and another located stratigraphically higher (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 Grey Laminated subunits, respectively. The Red Laminated demonstrates a localized 1% increase in copper grades occurring at the base of the unit adjacent to the Grey 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 resulted 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-grey to grey colored Domino subunit has the highest copper grades; the medium to light-grey-colored Grey Laminated has medium copper grades; and the red-brown colored Red Massive has distinctly the lowest copper grades.

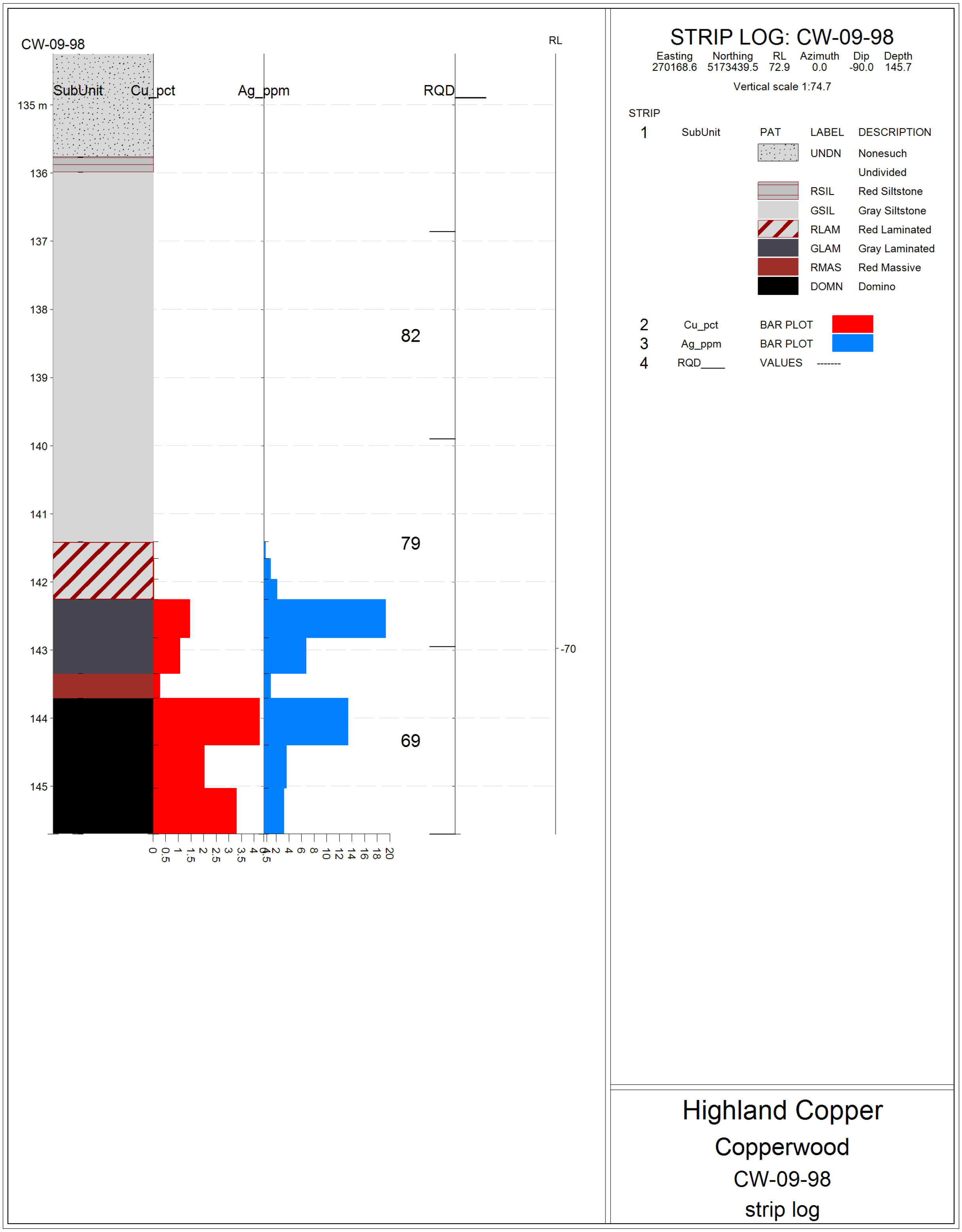
Grade profiles for each of the LCBS units show that there is a natural break in the grade profile, at approximately 1 wt.% copper. The 1 wt.% copper grade is a natural cut-off and is extensively used in Zambian and other African sediment-hosted copper deposits, where most intercepts grade a few tenths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

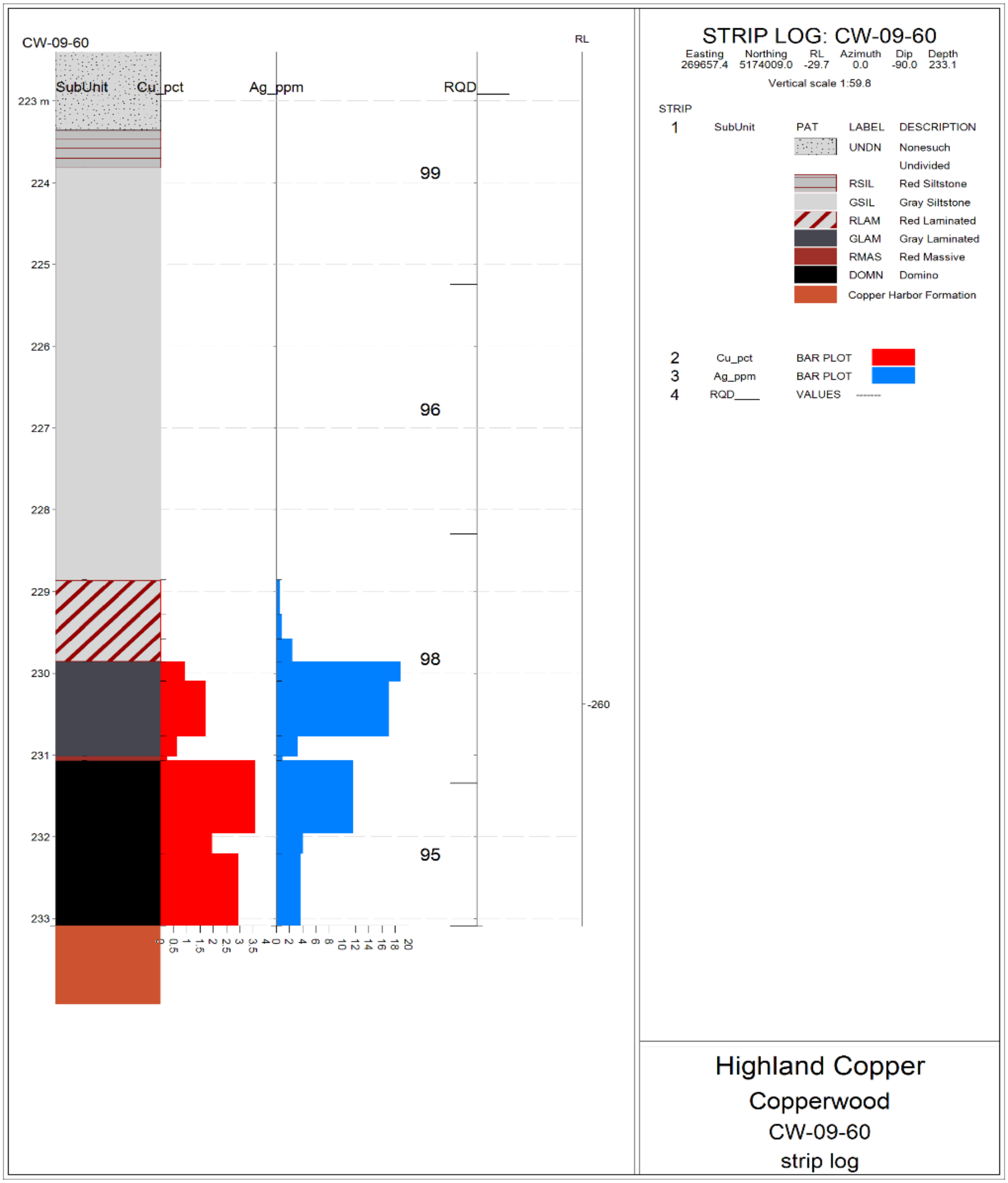
The UCBS hosts the same style of chalcocite mineralization as the LCBS but contains trace to no chalcocite mineralization in the western, thicker part of the deposit. The copper grade gradually increases towards the center of the Presque Isle Syncline, Section 6 contains an UCBS grade of 0.5 to 0.8 wt.% 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 recent drilling programs.

Although the average grades of silver in the Domino and Grey Laminated are of lower economic importance (4-6 g Ag/t), the spatial distribution of silver grades is 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 (Oak Bluff Formation).

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 historical White Pine Mine terminology and its 2014-15 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. As 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 is currently not considered.

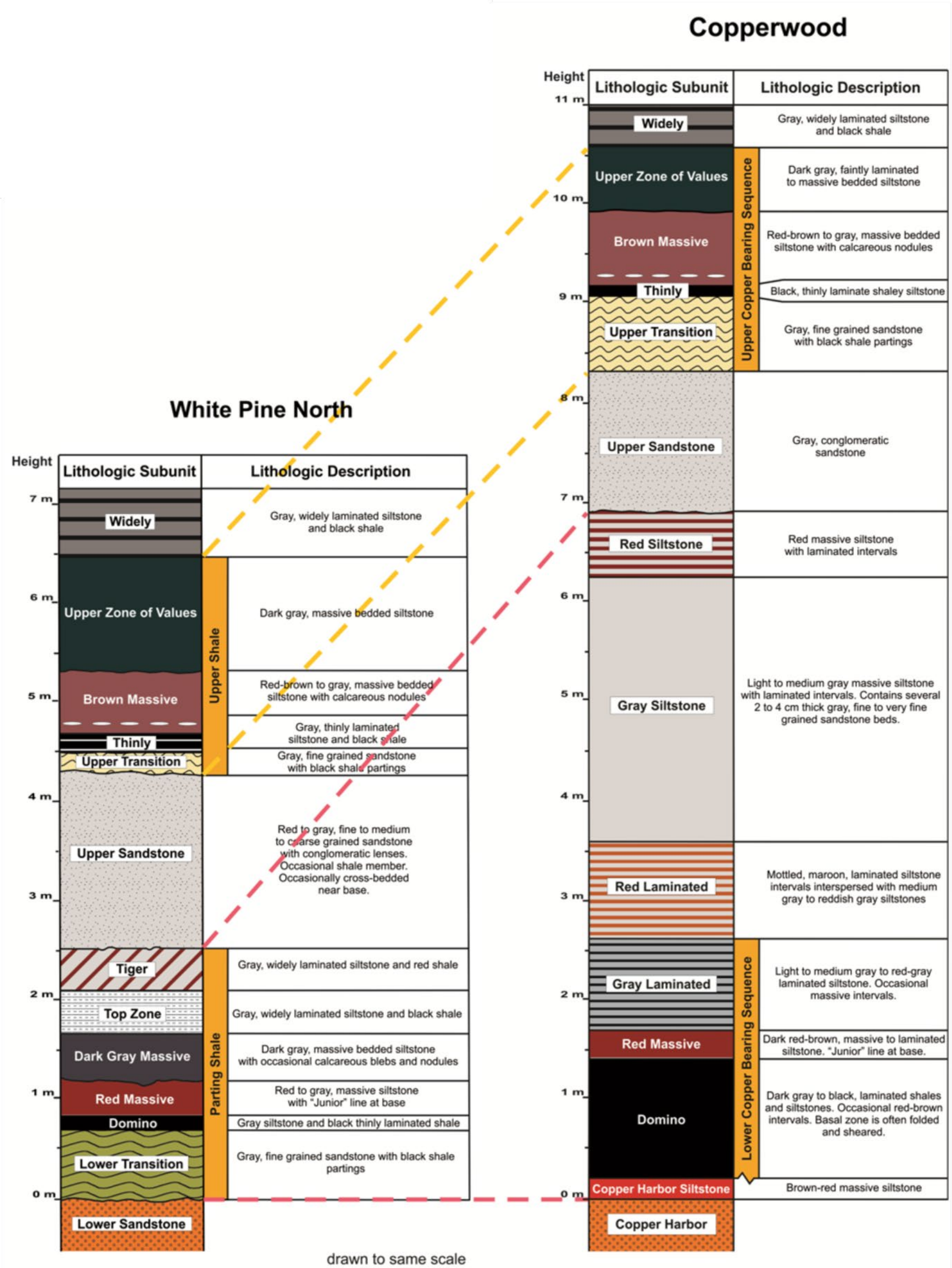
Structurally, there are significant differences between Copperwood and White Pine. The White Pine deposit straddles an anticline and a right-lateral strike-slip fault. Both the southwest and northwest domains of the White Pine deposit contain strike-slip and thrust faults. These faults are interpreted as being generated during the regional late rift compressional event. In contrast, the Copperwood deposit is structurally located on a simple dipping plane and 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 (2) 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 Grey 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 Grey 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 a sediment-hosted stratiform copper deposit. 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 minerals occur as disseminations and veins.

Sediment-hosted deposits have been grouped on the basis of the reductant into three (3) subtypes: reduced facies, red bed copper and Revett Copper. They can also be classified based on basinal setting into two (2) subtypes: Kupferschiefer and redbed. 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 redbeds (albite, potassic feldspar, chlorite, quartz, carbonate minerals, dolomitization, etc.); and, bleaching of red sediments to greenish grey or light grey 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 deposit is 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 redbeds 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 Deposit were performed by various owners, namely Orvana, AMAX, and United States Mineral Refining Company ("USMR"), and Highland Copper.

Discovery of the Presque Isle Syncline was made by mapping of bedrock outcropping in the Presque Isle River. The syncline did not begin appearing on published geological maps until the mid 20th century, though existence of the Nonesuch Shale in the vicinity was likely previously identified. With the opening of the White Pine Mine in 1955, the Nonesuch Formation in the Presque Isle Syncline became the subject of exploration activity. As the vast majority of the syncline is not exposed at surface, drilling was the sole exploration method.

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 subsequently by Highland Copper.

## 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 in 2010. 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 Copper Exploration Program

In 2017, Highland Copper carried out a drilling program comprising of 35 HQ diameter, five (5) 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 surveyed with acoustic televiewing equipment by DGI Geoscience for an improved understanding of the rock’s in situ geotechnical characteristics. An additional hole was optically televiewed by DGI Geoscience in December 2017 for geotechnical studies requested by Golder.

In early 2018, Highland Copper 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 (2) 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 (5) 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, seven (7) primarily for geotechnical purposes, and one 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 Copper 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**

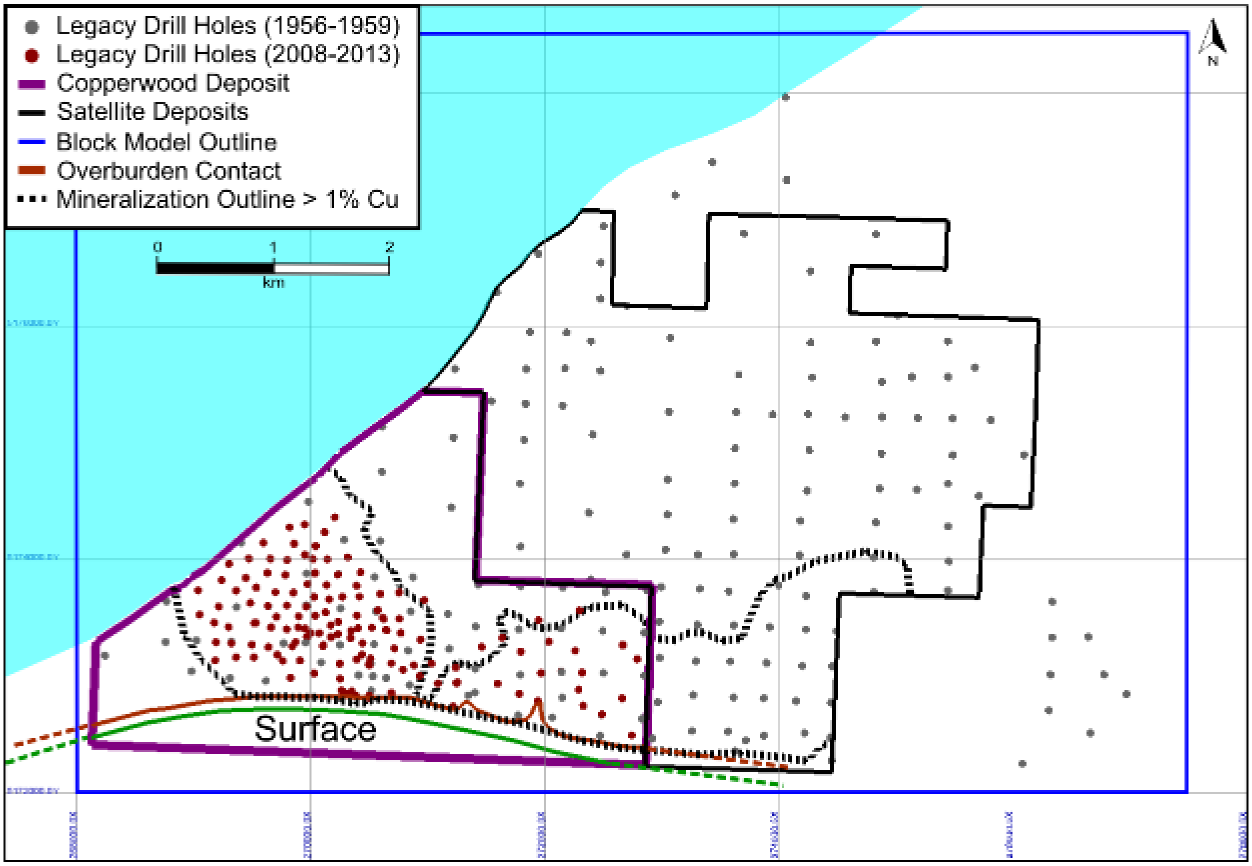
| Company | Period | Core Size | Drill Hole Count | Length (m) | % of Total Drilling |
|---|---|---|---|---|---|
| USMR | 1956 to 1958 | BX & AX | 161 | 34,050 | 49% |
| BMC | 1959 | BX & AX | 23 | 3,998 | 6% |
| Orvana | 2008 | NQ | 6 | 744 | 1% |
| Orvana | 2009 | NQ | 82 | 12,858 | 18% |
| Orvana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

**Note: 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 10º to 15º. Most of the drilling has been vertical; therefore, intercepts are slightly greater than true thicknesses.

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, 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). IDEA Drilling has since been acquired by Timberline Drilling Inc. 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. 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 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 determined which survey points to reject due to casing interference.

### 10.2.3 Core Logging

A Highland Copper 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 (3) 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 (1) 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 Copper 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 (10) 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 Copper’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**

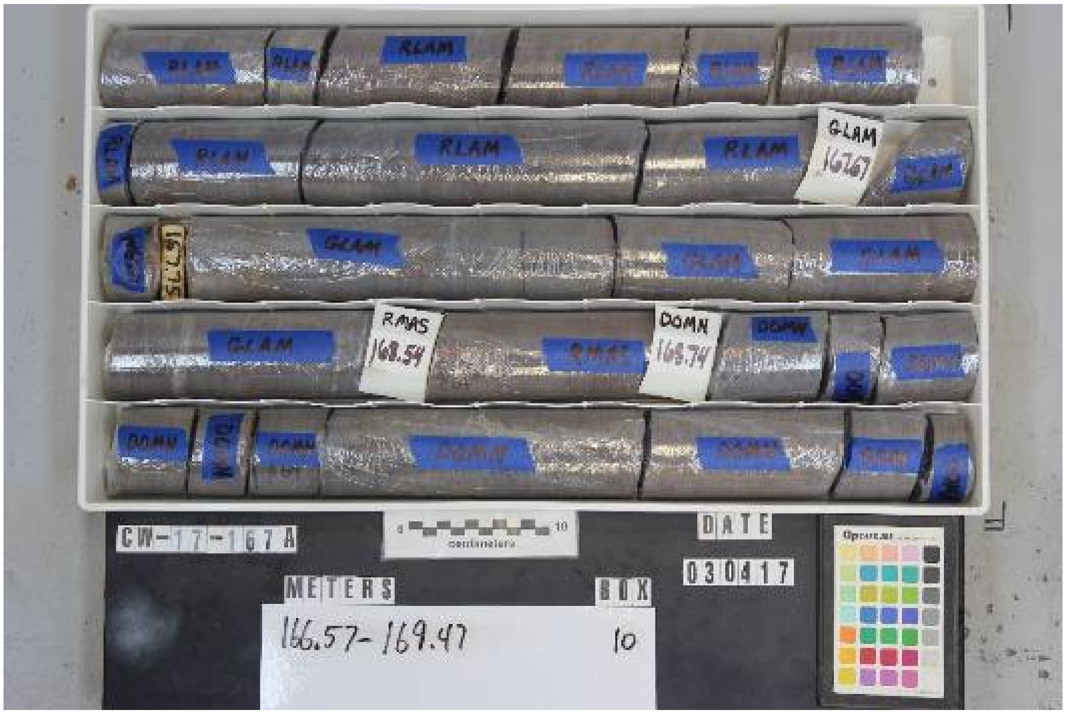



**Figure 10.9: Top of LCBS Showing Marked Intervals for Assay Sampling**

**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 Copper collected 57 specific gravity measurements of which 49 were completed in-house using the water immersion method and eight (8) 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 Copper 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 Copper 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 were analyzed at the Actlabs analytical laboratory in Thunder Bay, Ontario. With the exception of the 2018 assays, the samples were analyzed 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. GMS 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.

### 11.1.2 Quality Control

Highland Copper 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 |

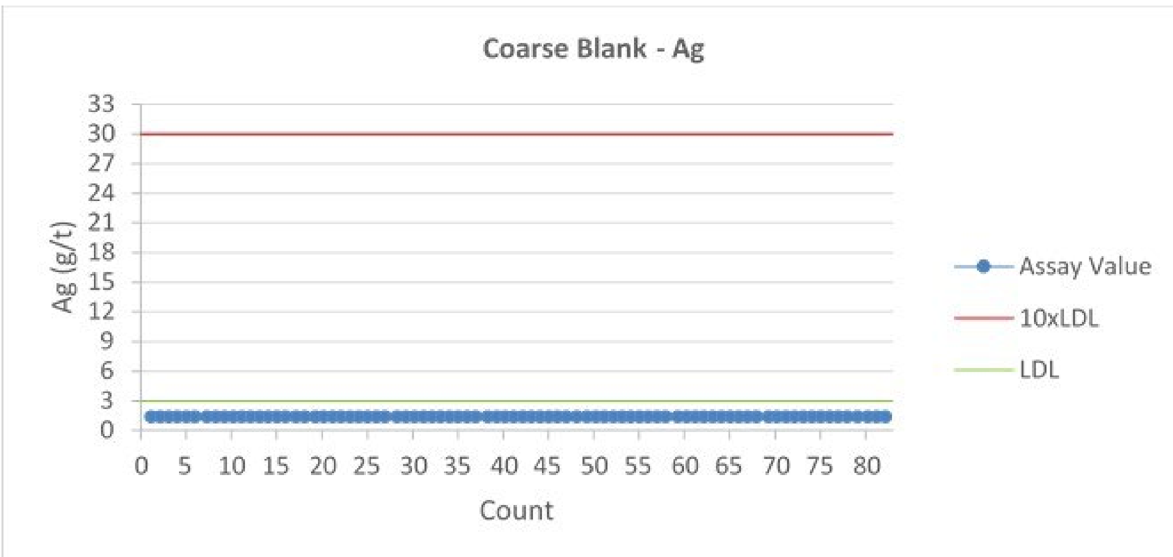
A geologist regularly inserted two (2) standard CRMs, three (3) coarse blanks, and one (1) core duplicate for each drill hole. CRMs with a high Cu wt.%, medium Cu wt.%, and low Cu wt.% were inserted in a high-grade, medium-grade, and low-grade interval, respectively. Coarse blanks were inserted between 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 Copper 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.

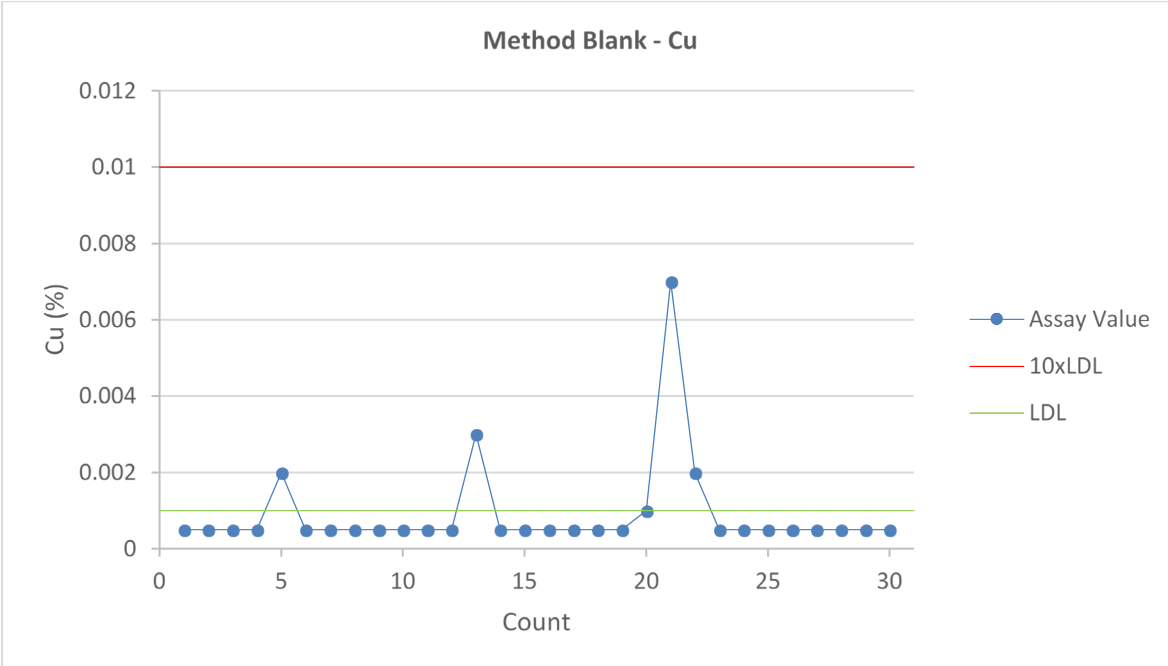
Less than 4% (3 samples) of the coarse blanks show greater values than 0.01% Cu (10 x lower detection limit). All three (3) blanks fall after a previous sample with high-grade Cu (>1% Cu). Two (2) blanks failing the QA/QC and the surrounding primary samples were re-analyzed. The results for both the 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 (1) analysis of greater than 0.01% Cu but was considered within acceptable limits.

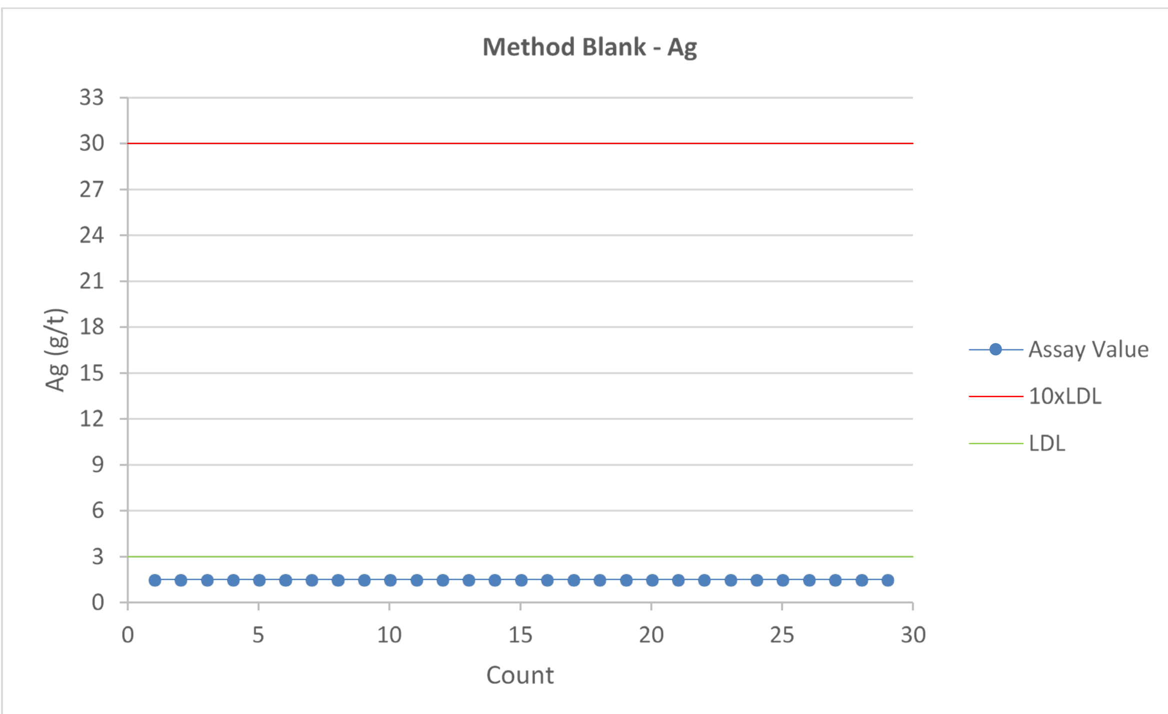
**Figure 11.1: Highland-inserted Blank Material Analytical Results (coarse CRM) for Cu (top) and Ag (bottom)**




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.

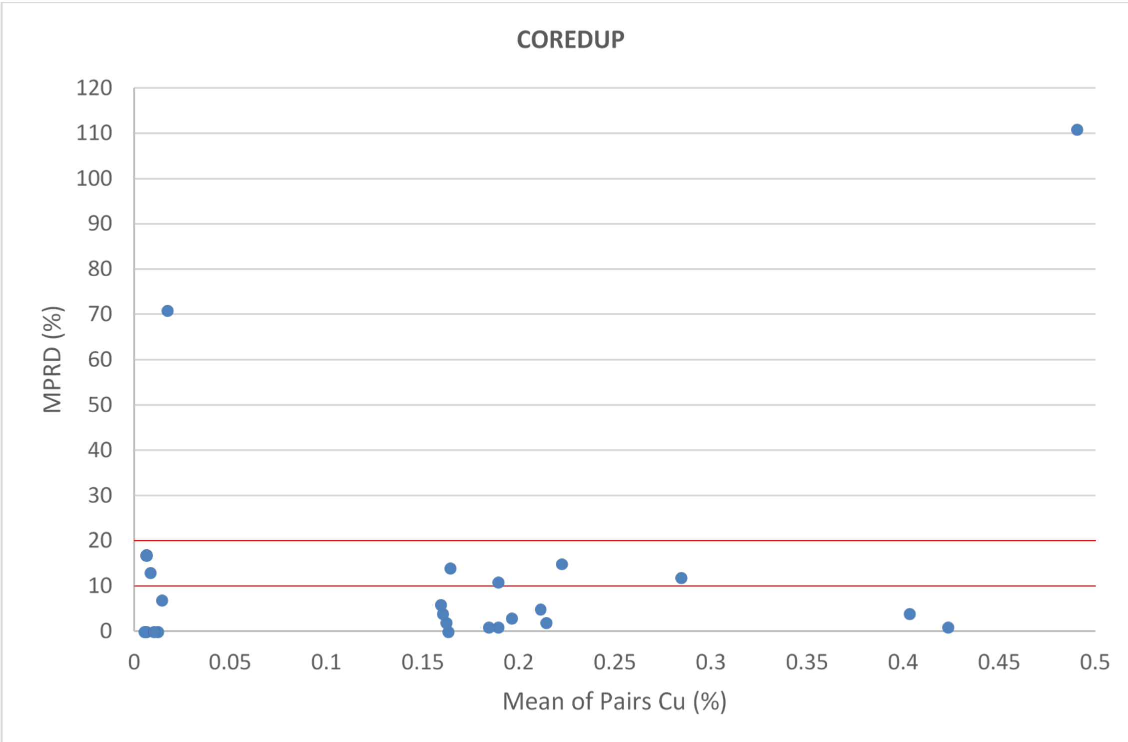
The core duplicates performance is considered to be acceptable reflecting good overall precision and negligible sampling and analytical error (field and laboratory). Two (2) copper core duplicates out of 26 core duplicates have a mean pair relative difference greater than 20% and possibly highlight variability characteristics of the ore deposit. Three (3) silver core duplicates also have a mean pair relative difference greater than 20% and one (1) of the silver duplicates coincident with one (1) of the two (2) 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 (1) 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 (2) 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 (5) 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)**



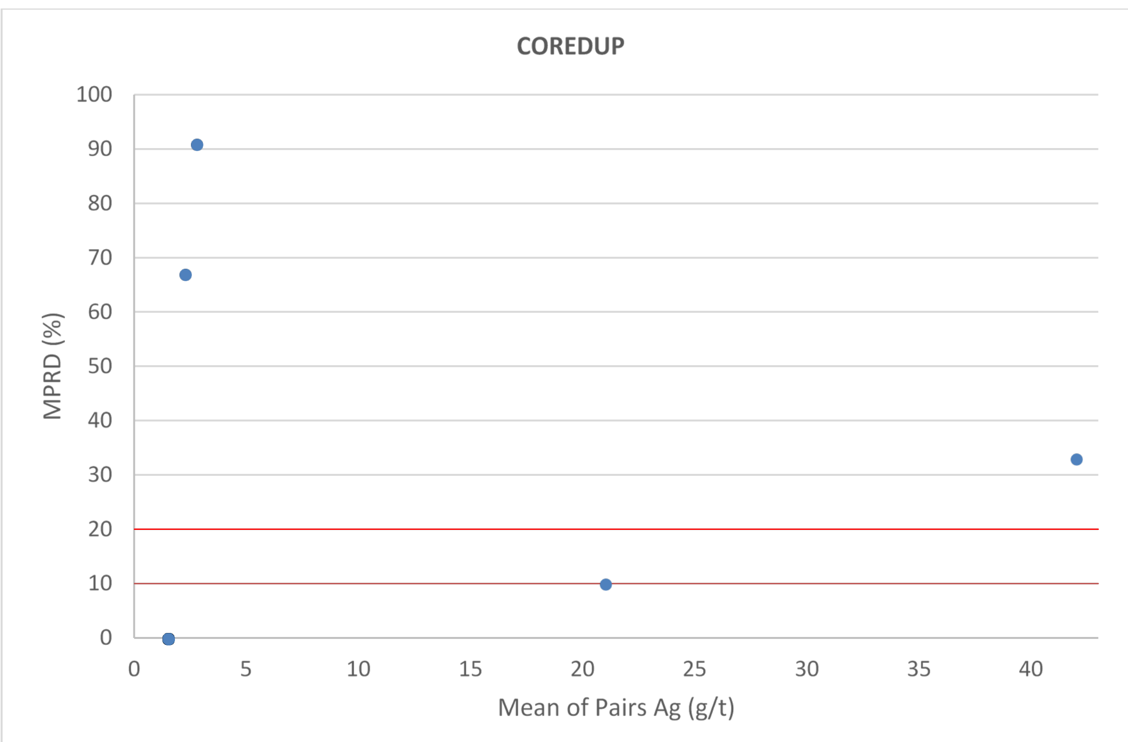
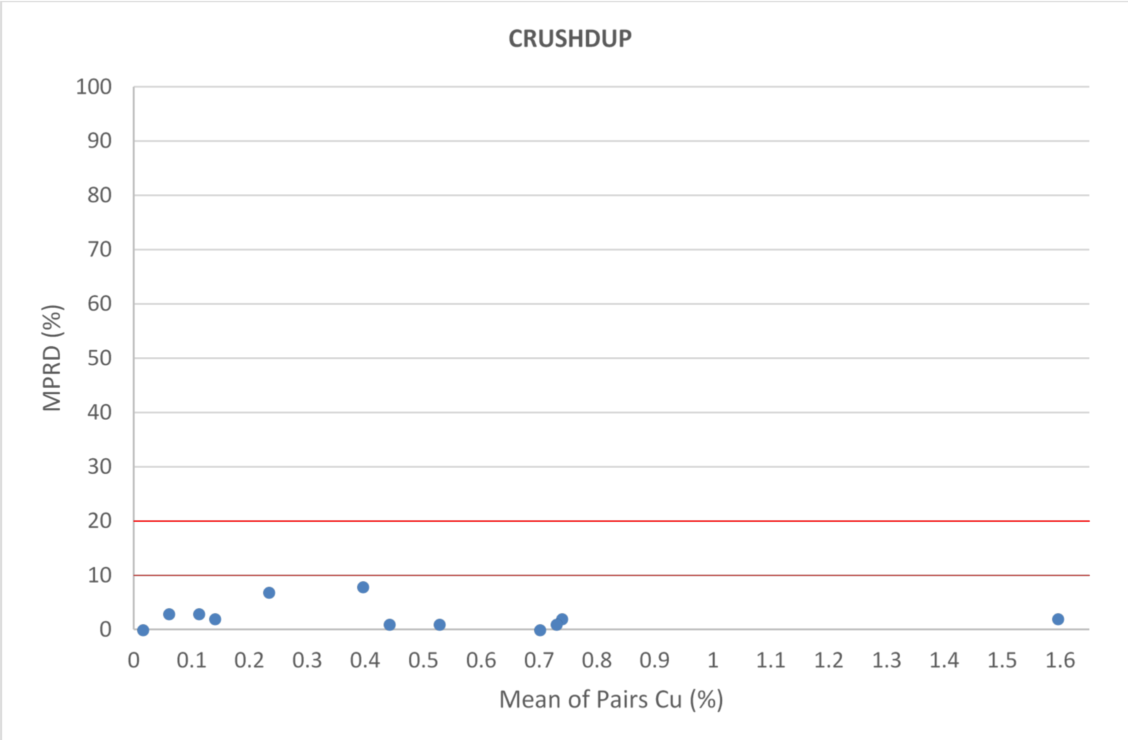



**Figure 11.4: Crush Duplicate Performance for Cu (top) and Ag (bottom)**



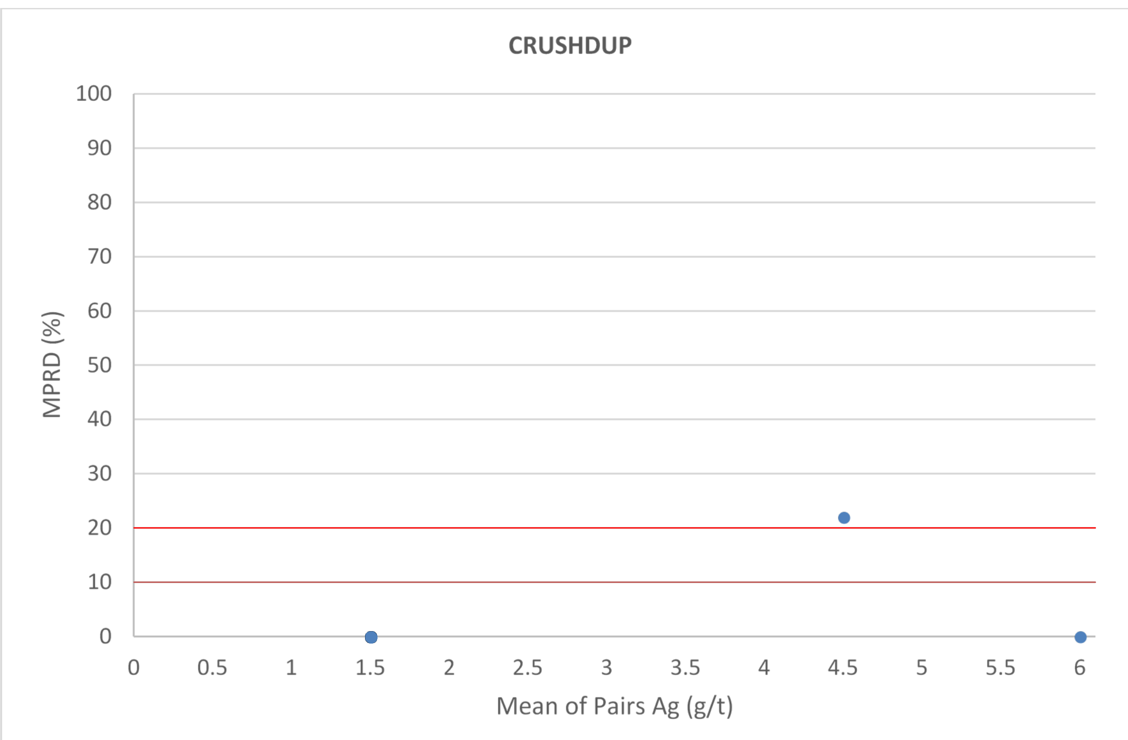
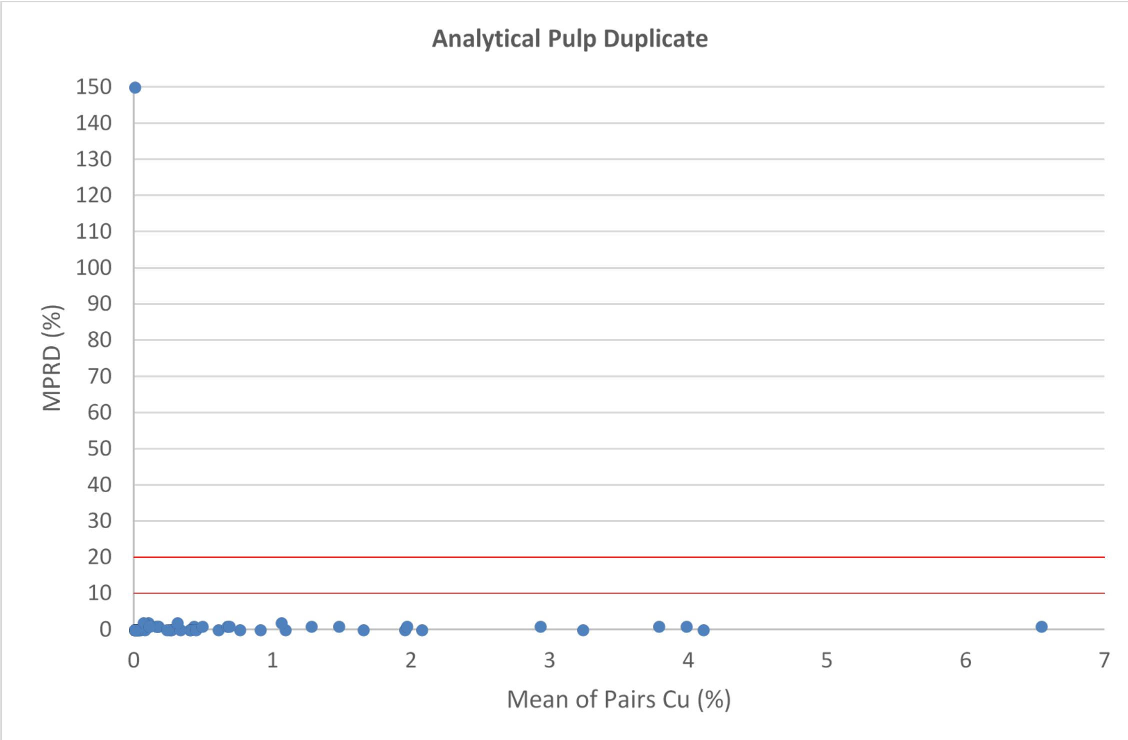
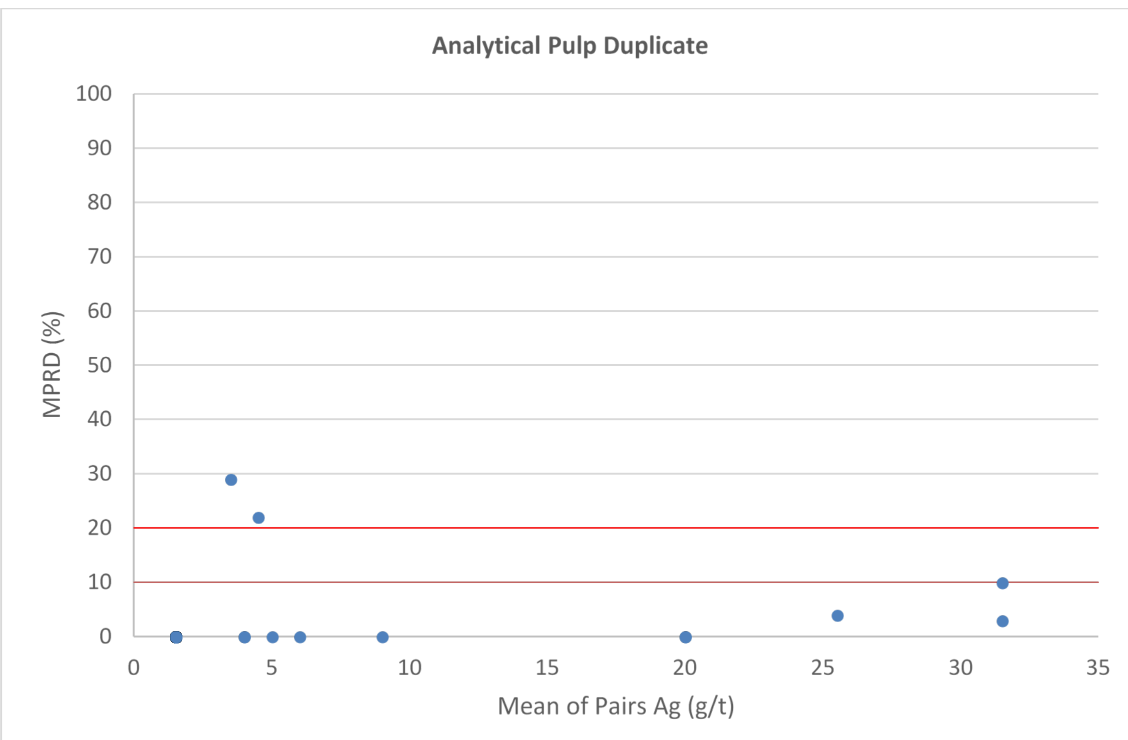



**Figure 11.5: Analytical Pulp Performance for Cu (top) and Ag (bottom)**



## 11.2 Performance of Standards

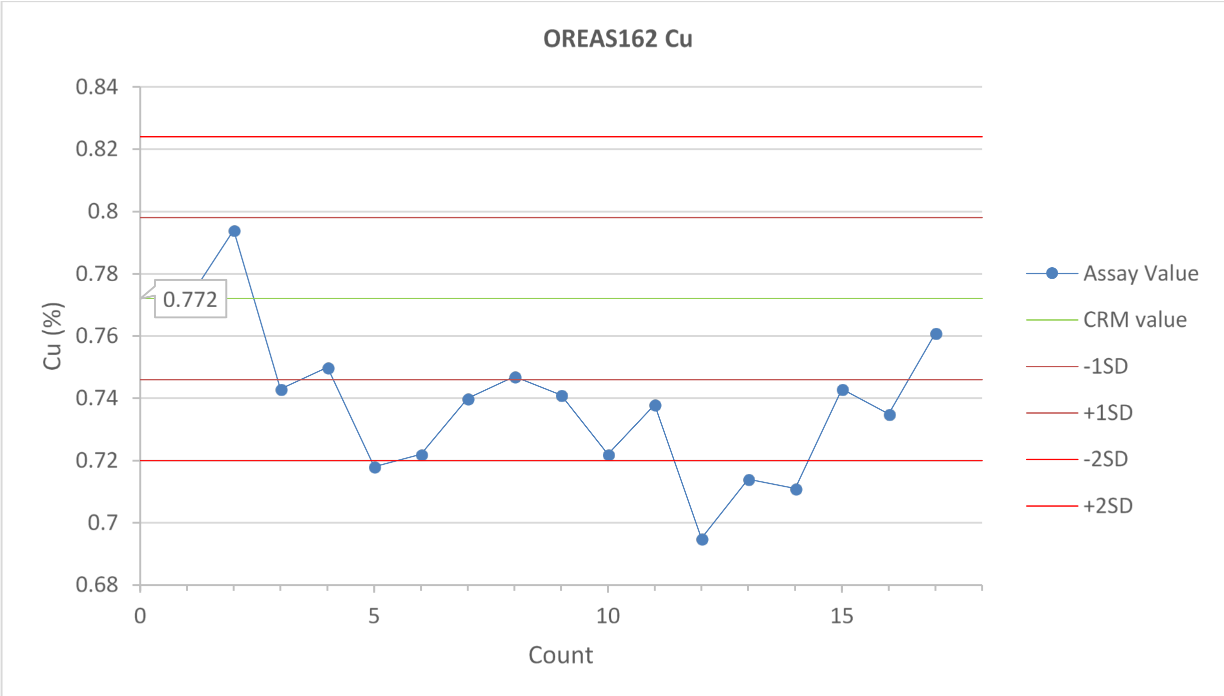
Throughout the analysis of 2017-18 drilling program, standards were inserted at an 10.1% rate. A total of 54 standards were used during the 2017-18 analytical assaying. Three (3) different standards OREAS 162, OREAS 97 and OREAS 930 were used with principle certified values of 0.772% Cu, 6.31% Cu, 2.52% 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 (5) standards out of 54 have analytical values greater than ±2 standard deviations from the certified value for copper and two (2) of these have an analytical value greater than ±2 standard deviations from the certified value for silver. Three (3) 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 (5) standards with analytical values greater than ±2 standard deviations from certified values along with the surrounding primary samples were re-analyzed. 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)**



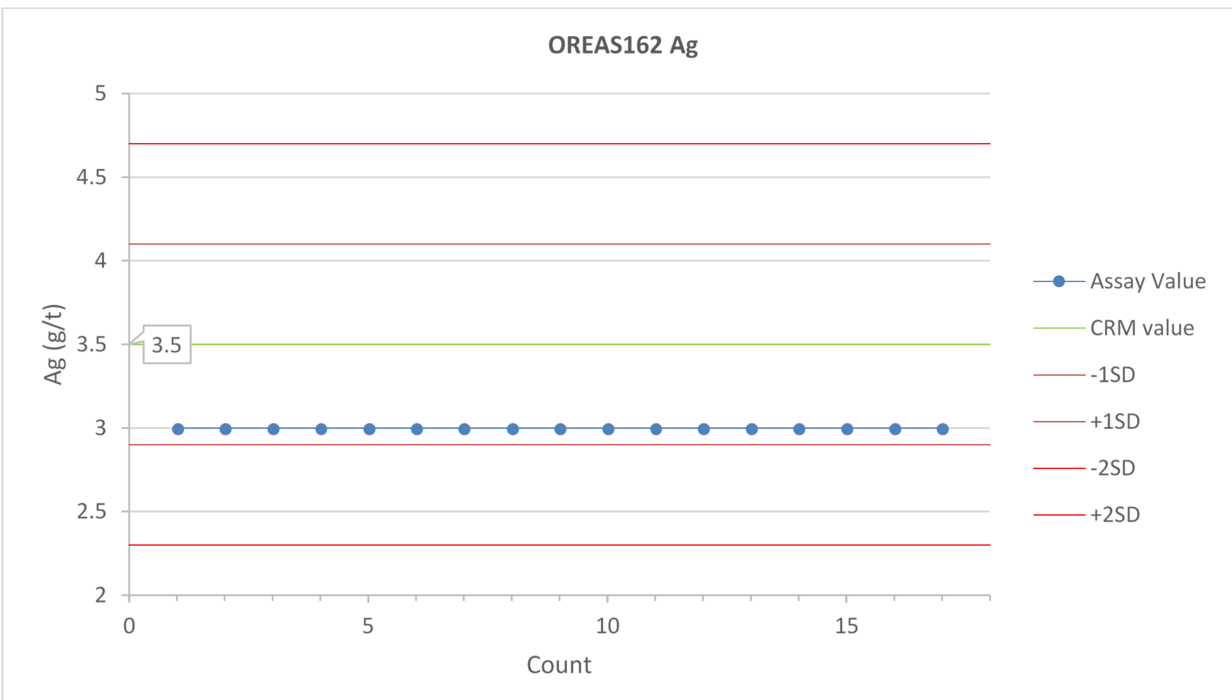
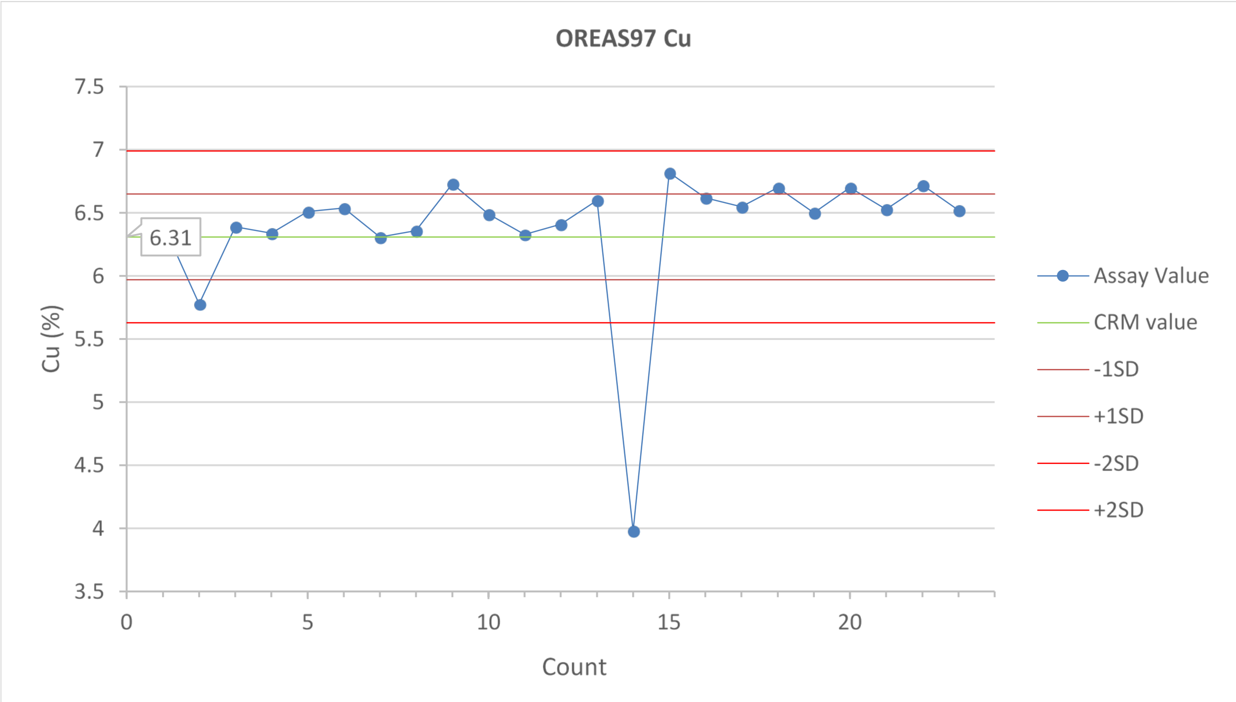



**Figure 11.7: Performance of Control Reference Material OREAS 97 for Cu (top) and Ag (bottom)**



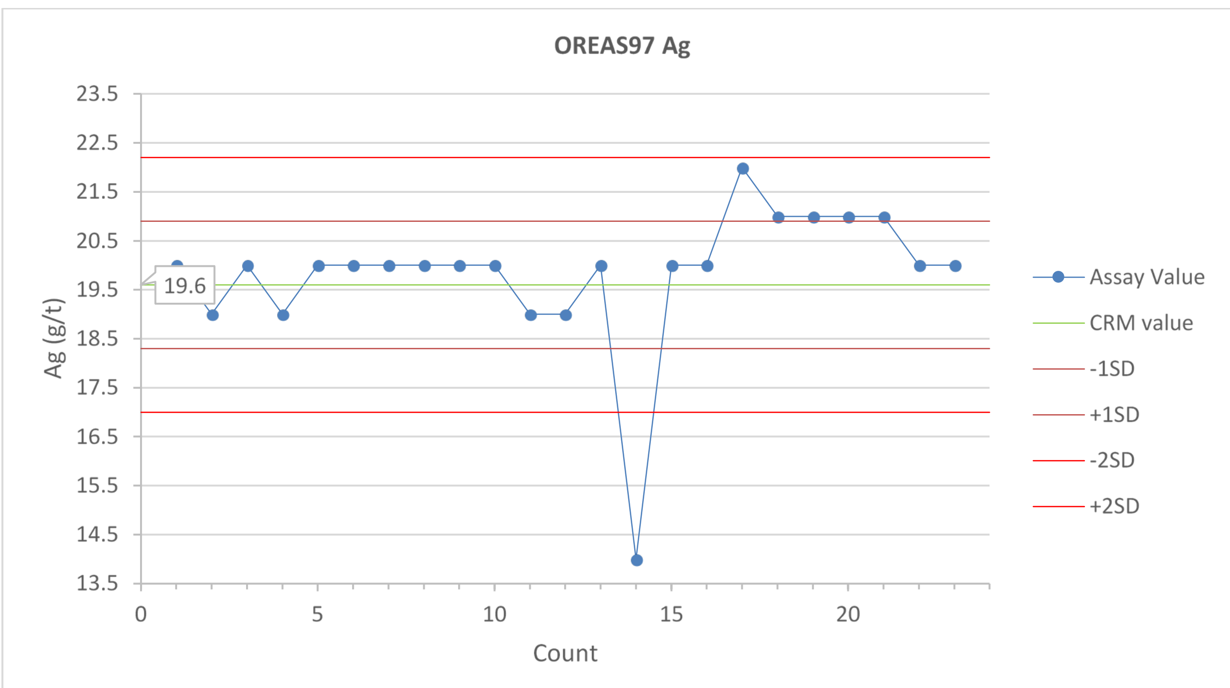
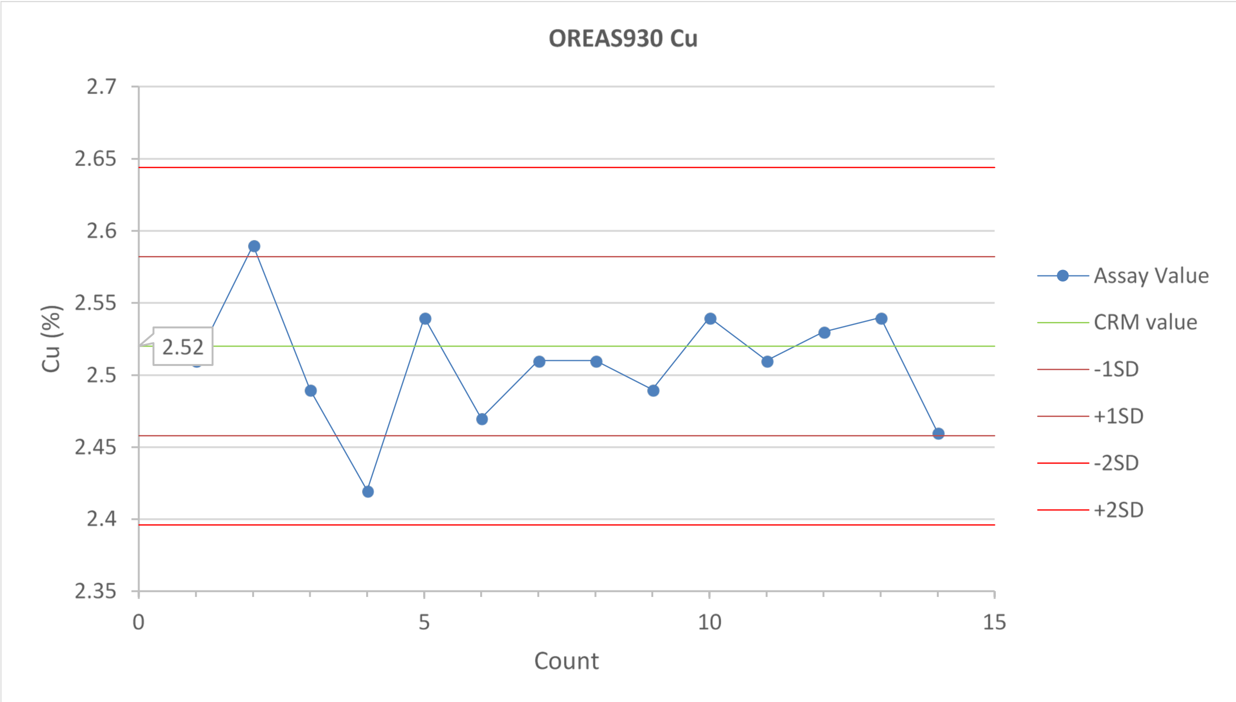
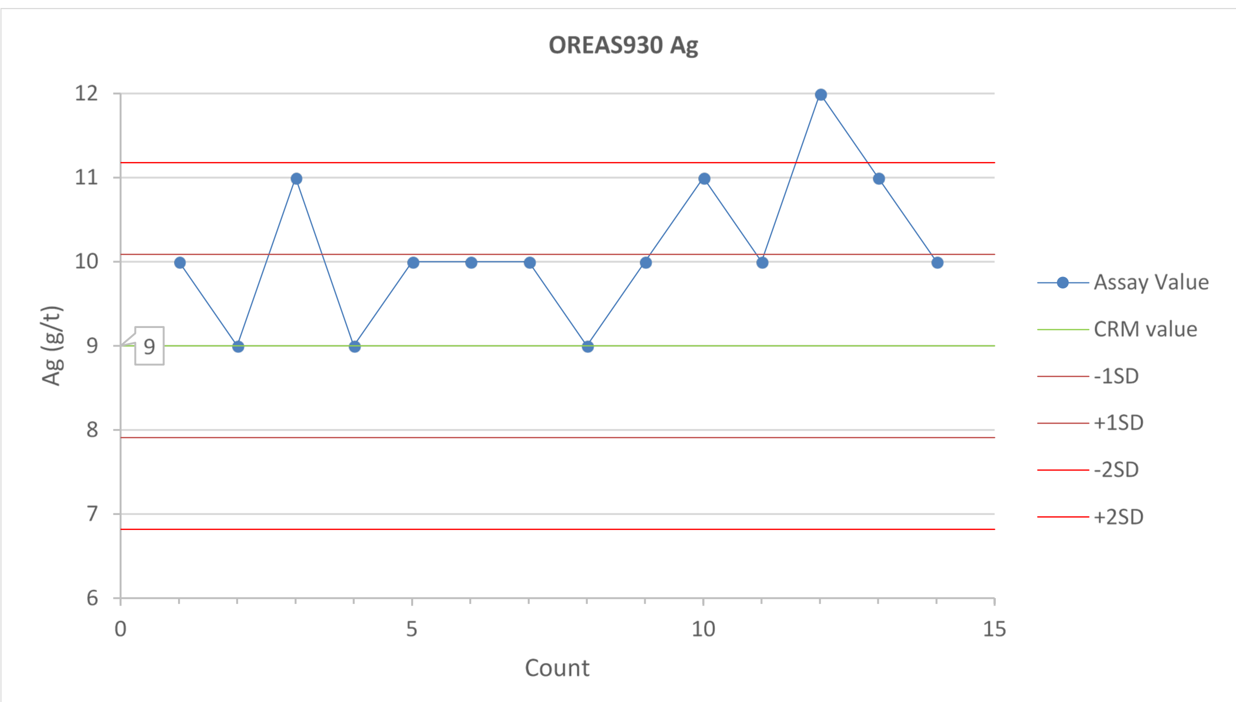



**Figure 11.8: Performance of Control Reference Material OREAS 930 for Cu (top) and Ag (bottom)**

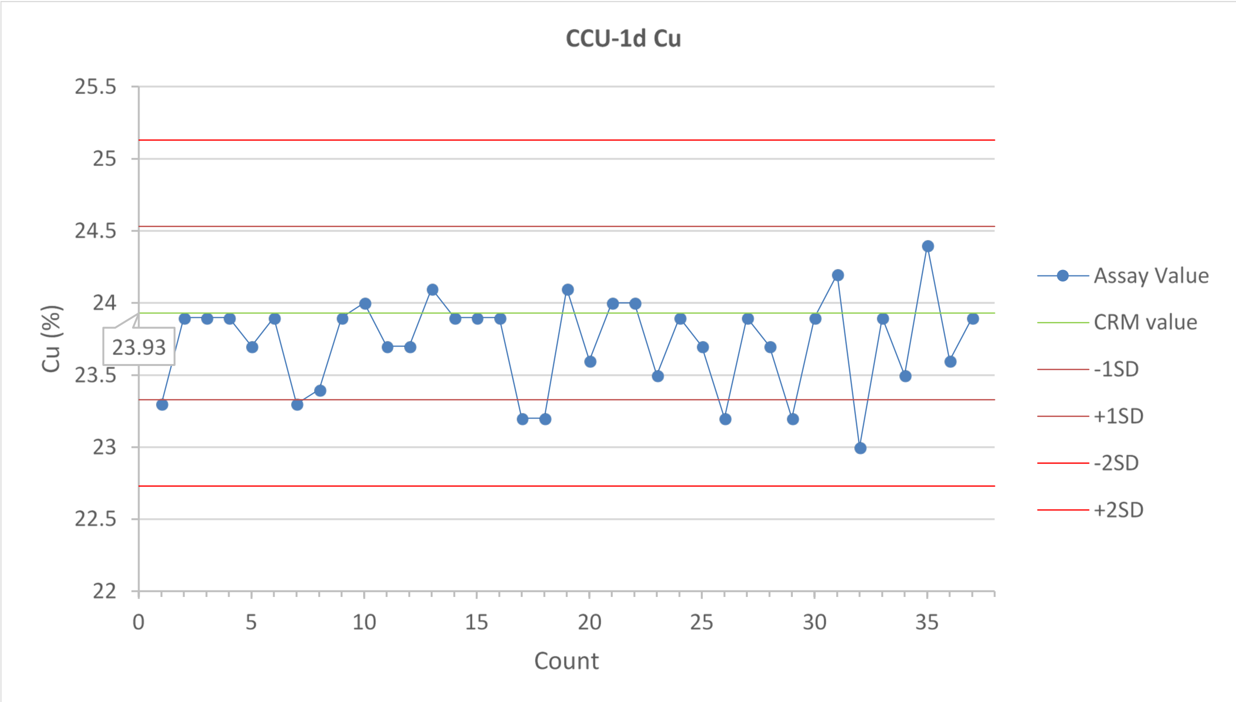


Four (4) different internal laboratory standards were inserted by the Actlabs at a 33.1% rate for Cu ICP-OES and at a 19.6% rate for Ag ICP-OES. The certified standards include CCU-1d, CZN-4 and MP-1b from Natural Resources Canada and OREAS 14P from Ore Research and Exploration Pty Ltd. All four (4) standards were analyzed for copper and three (3) of the standards excluding OREAS 14P were analyzed 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 (5) having values within ±2 standard deviations from the certified value. Initially, two (2) 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 (2) failing standards. The silver internal laboratory standards are within ±2 standard deviations from the certified value with the exception of four (4) standards. The four (4) 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)**



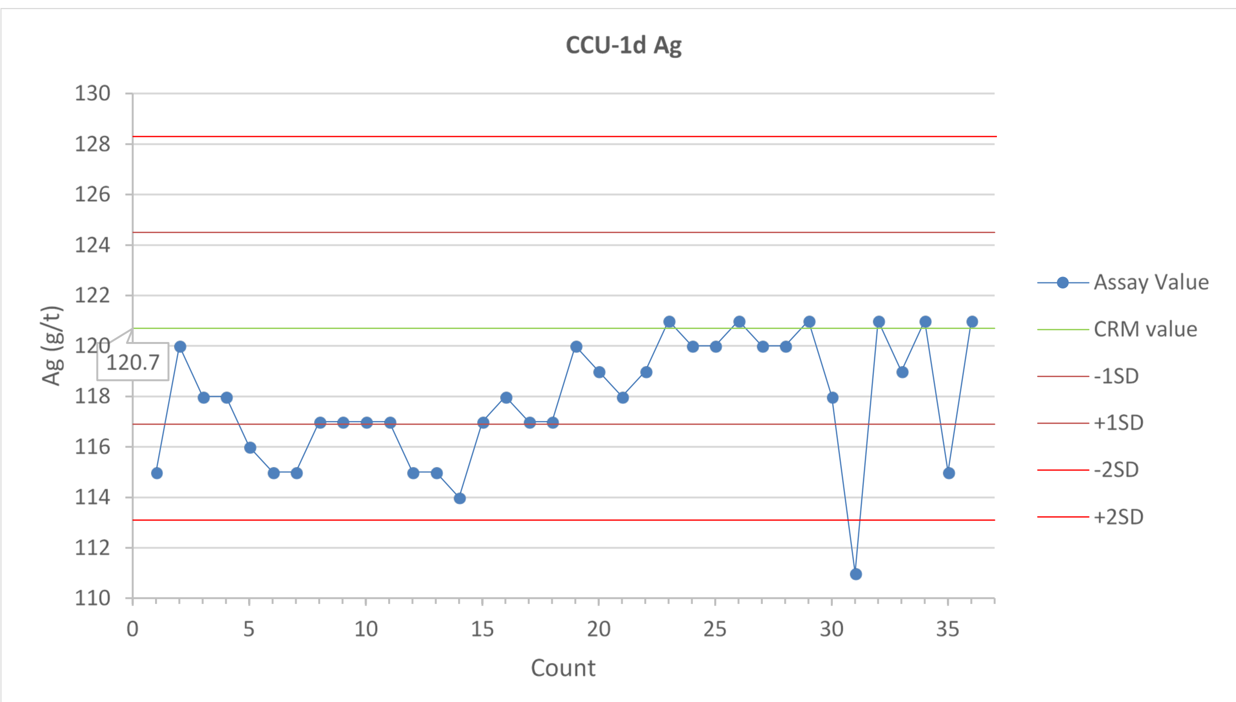
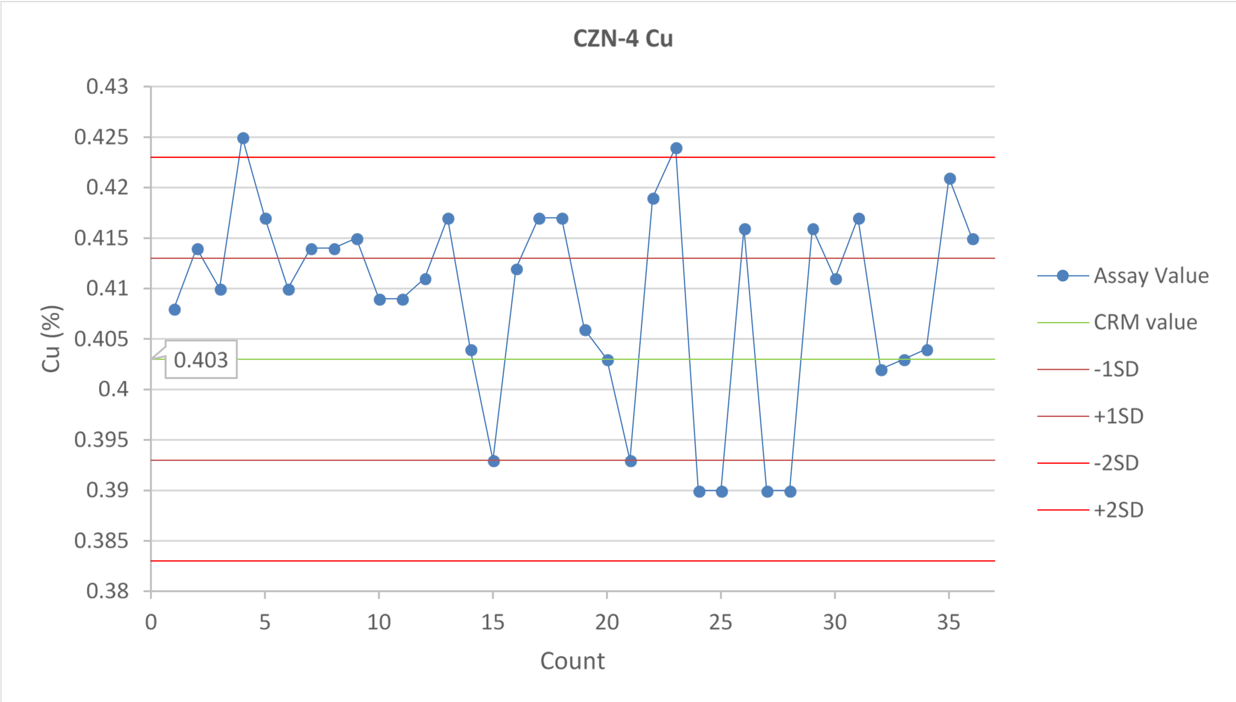



**Figure 11.10: Performance of Control Reference Material CZN-4 for Cu (top) and Ag (bottom)**



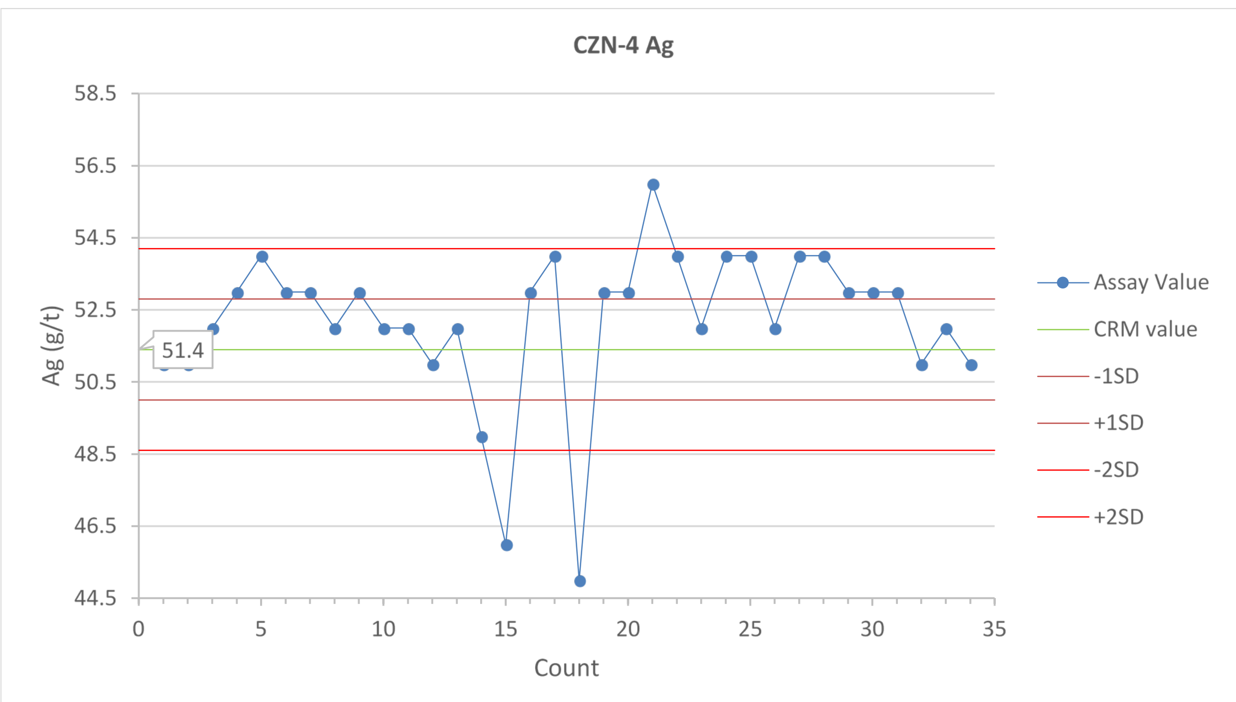
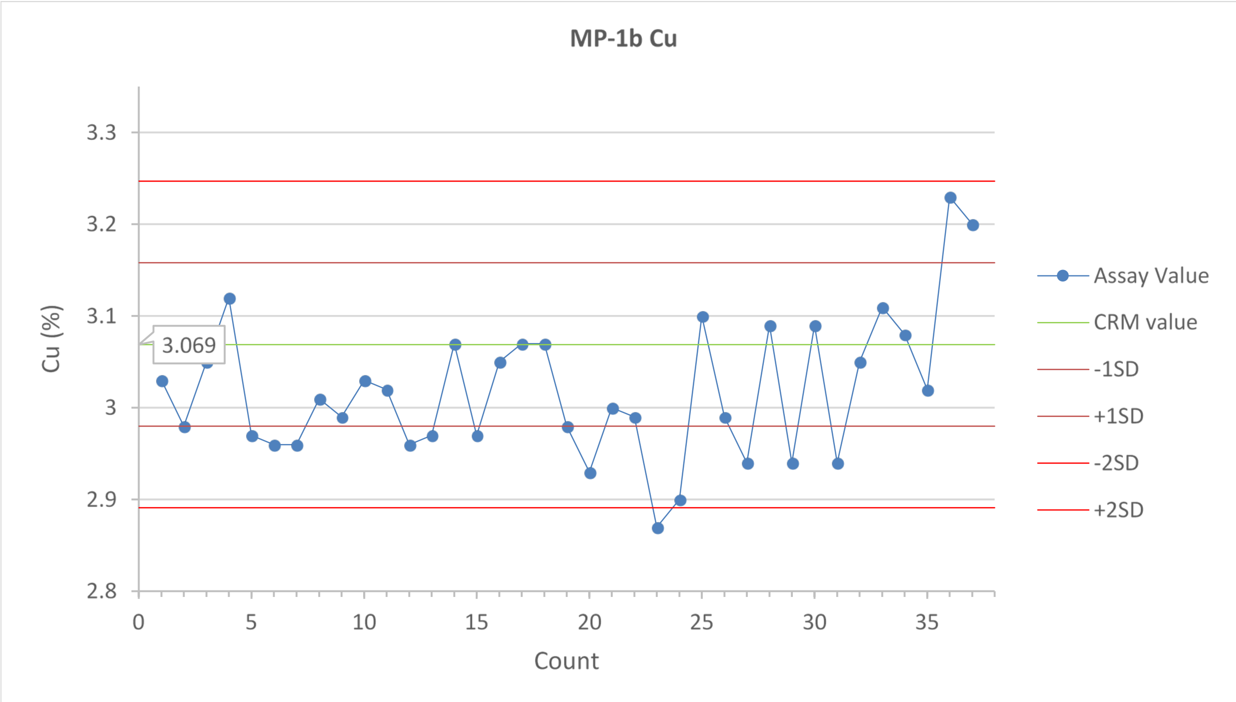



**Figure 11.11: Performance of Control Reference Material MP-1b for Cu (top) and Ag (bottom)**



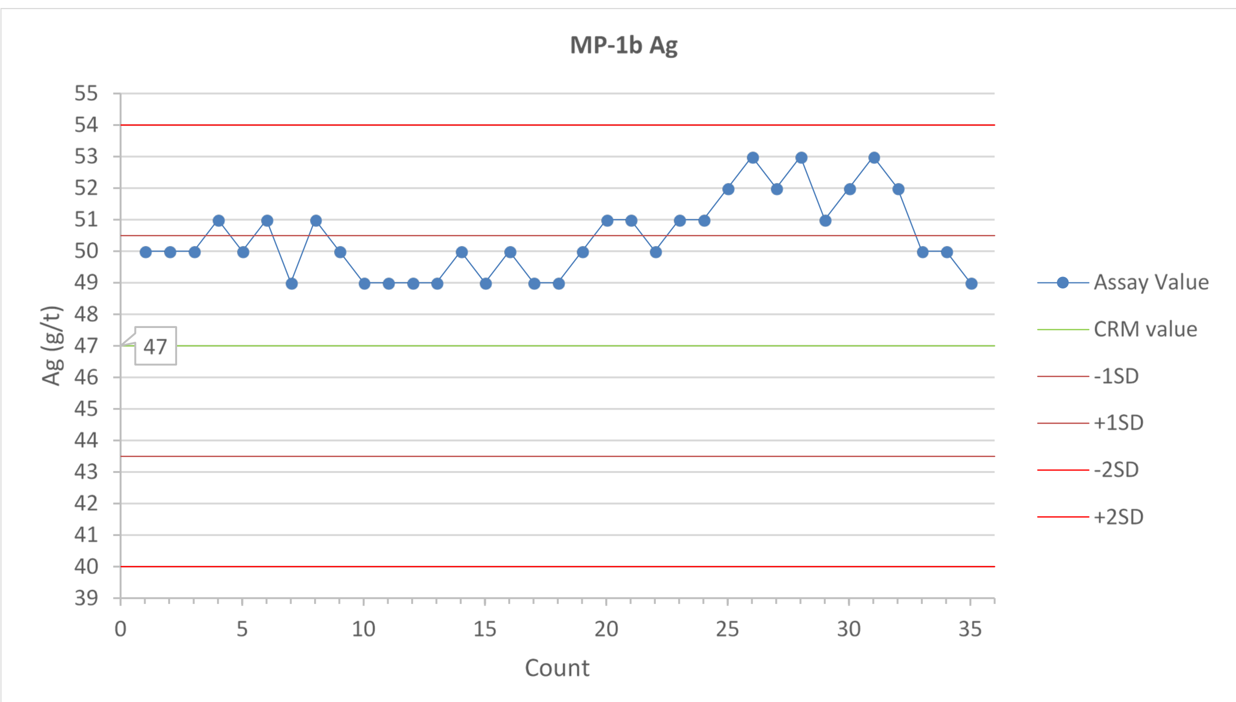
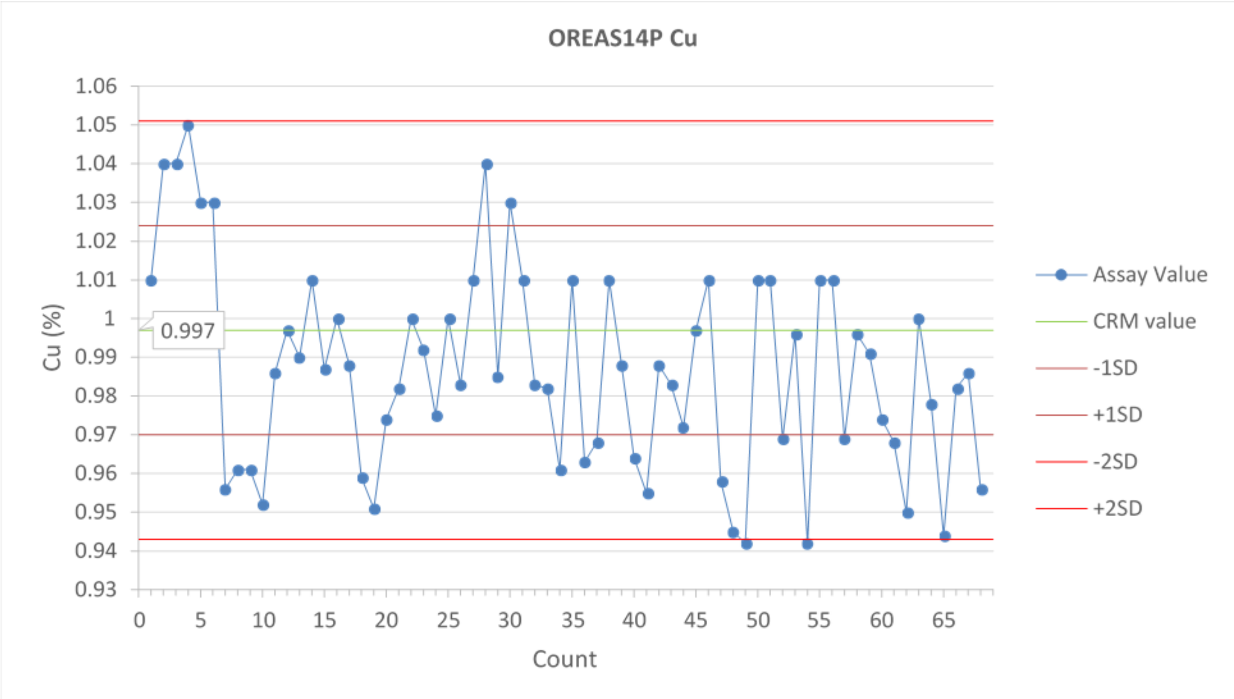



**Figure 11.12: Performance of Control Reference Material 14P for Cu**




## 11.3 <u>Density Procedures</u>

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 an 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.

## 11.4 <u>Security</u>

Highland Copper 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.

## 11.5 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, GMS also concluded that the QA/QC and security protocols established by Orvana and the quality of the results support resource and future reserve estimation.

# 12. DATA VERIFICATION

## 12.1 Database

Drill hole information for the 2018 drilling program at the Copperwood Project was provided to GMS 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. No drilling has been undertaken on the property since the data was transferred to GMS. GMS imported the files into the MS Access database used in the 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 GMS Data Verification

Most of the content in this section is sourced from the NI 43-101 technical report prepared by GMS 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 reviewed during the site visit between November 6th and 9th, 2017. Mr. James Purchase, P.Geo. and Mr. Réjean Sirois, P.Eng., of GMS were present during the site visit. Drill hole collars drilled in 2018 (14 drill holes) were 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.

GMS performed data verification checks of the drill logs, assay certificates, downhole surveys, and additional information sources on site at Highland Copper'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 (5) random laboratory certificates were also directly sent to GMS from Actlabs to compare with Highland Copper's certificates. No error was found.
- GMS 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 errors were found; GMS 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 surveys and verification by Coleman Engineering
- Validation of QA/QC, density, metallurgical and logging procedures with Highland Copper's professional staff. All information pertaining to the aforementioned procedures are rigorously recorded in procedure manuals easily accessible to Highland Copper'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

Mr James Purchase, P.Geo visited numerous drill collars from the 2017 drilling campaign during the site visit between November 6$^{th}$ and 9$^{th}$, 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 – CW17-184**


## 12.4 QA/QC Validation

GMS 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, the QP 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. GMS believes that all data is reliable for use in the statement of Mineral Resources presented in this Report. No additional technical or scientific information has been gathered since the 2018 driling program.

# 13. MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

The update of this feasibility study concerning the mineral processing and metallurgical testing was a technical review of the previous work. No tests were performed during this update. Therefore, this section has largely been reproduced from the previous technical report on this property documented in G Mining Services Inc. et al (2018) and provides a description of metallurgical test work, analysis and interpretation of the test work results completed from 2008 to 2018.

## 13.2 Early Metallurgical Testing (before 2012)

Metallurgical test work 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 test work 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 test work 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 test work, 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.

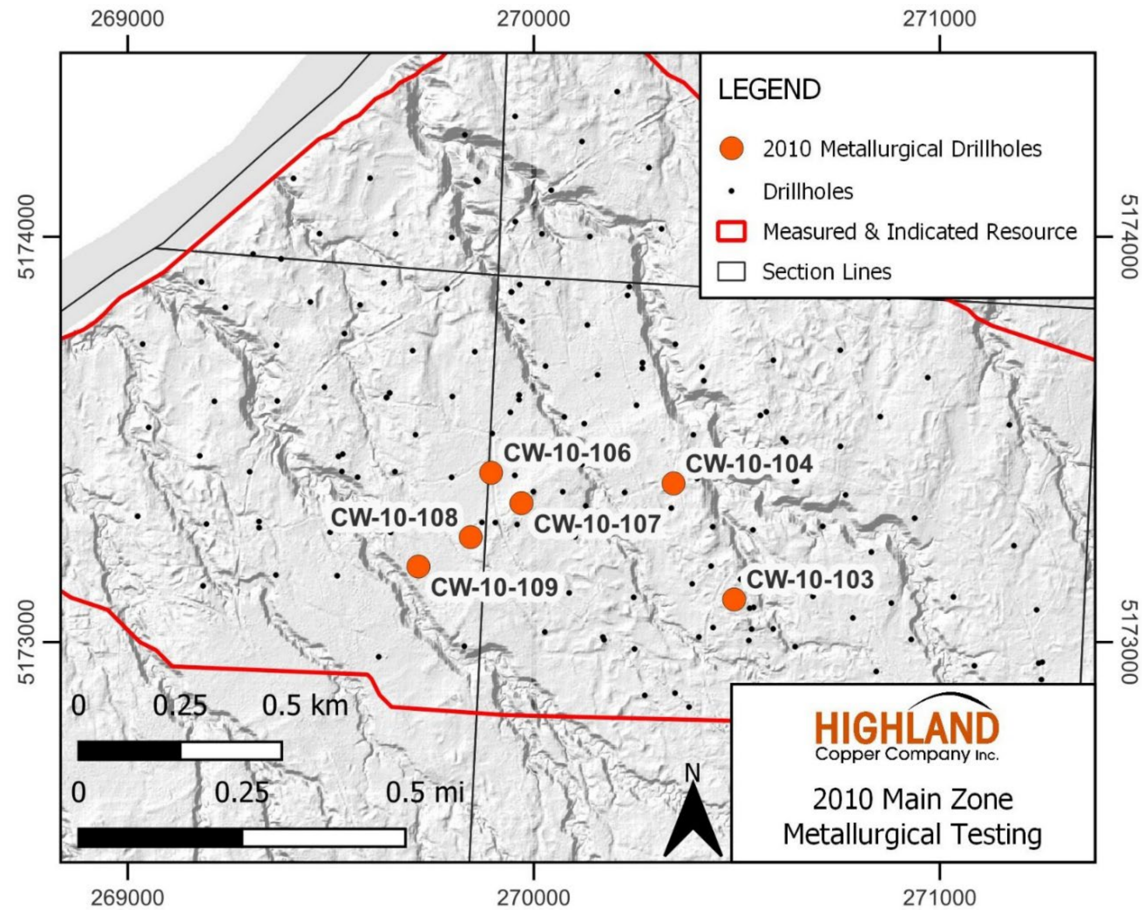
**Table 13.1: Composites No. 4 and No. 5 Assays**

| Sample ID | Assays (%) | | | Sequential Copper Analysis (%) | | | |
|---|---|---|---|---|---|---|---|
| | Cu (%) | Fe (%) | Ag (g/t) | As Cu | CNs C | 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 |
| | CW-10-104 | Main |
| | CW-10-106 | Main |
| | CW-10-107 | Main |
| | CW-10-108 | Main |
| | CW-10-109 | Main |
| CBS5 | CW-10-125 | Section 6 |
| | CW-10-129 | Section 6 |
| | CW-10-133 | Section 6 |
| | CW-10-136 | Section 6 |
| | CW-10-138 | Section 6 |
| | CW-10-139 | Section 6 |
| | CW-10-142 | Section 6 |
| | CW-10-143 | Section 6 |

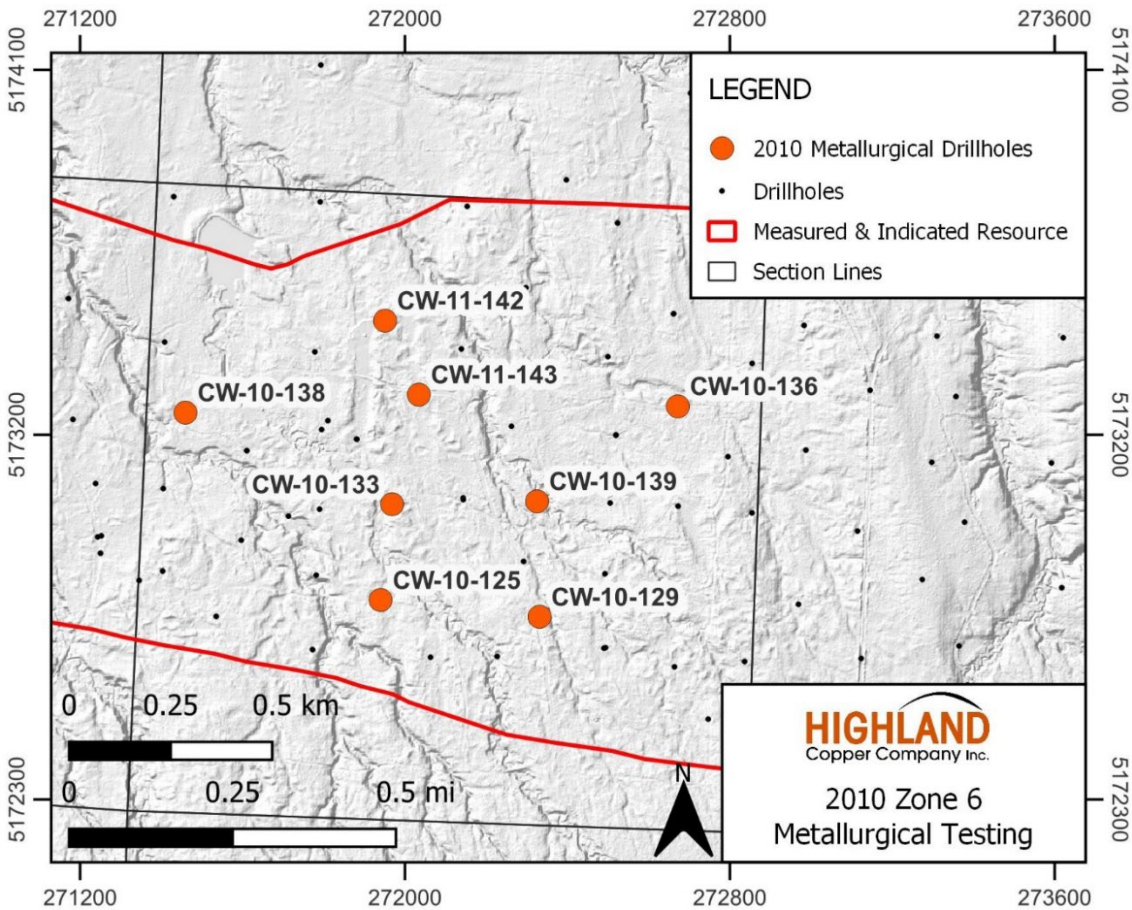
**Figure 13.1: Main Zone Sample Location**




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 Section 6 were more evenly distributed over the Copperwood deposit.

**Figure 13.2: Section 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 for Metallurgical Mass Balance**

| 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 | $S_r$ | Insol. | Cu | Ag | Fe | $S_r$ | 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** |

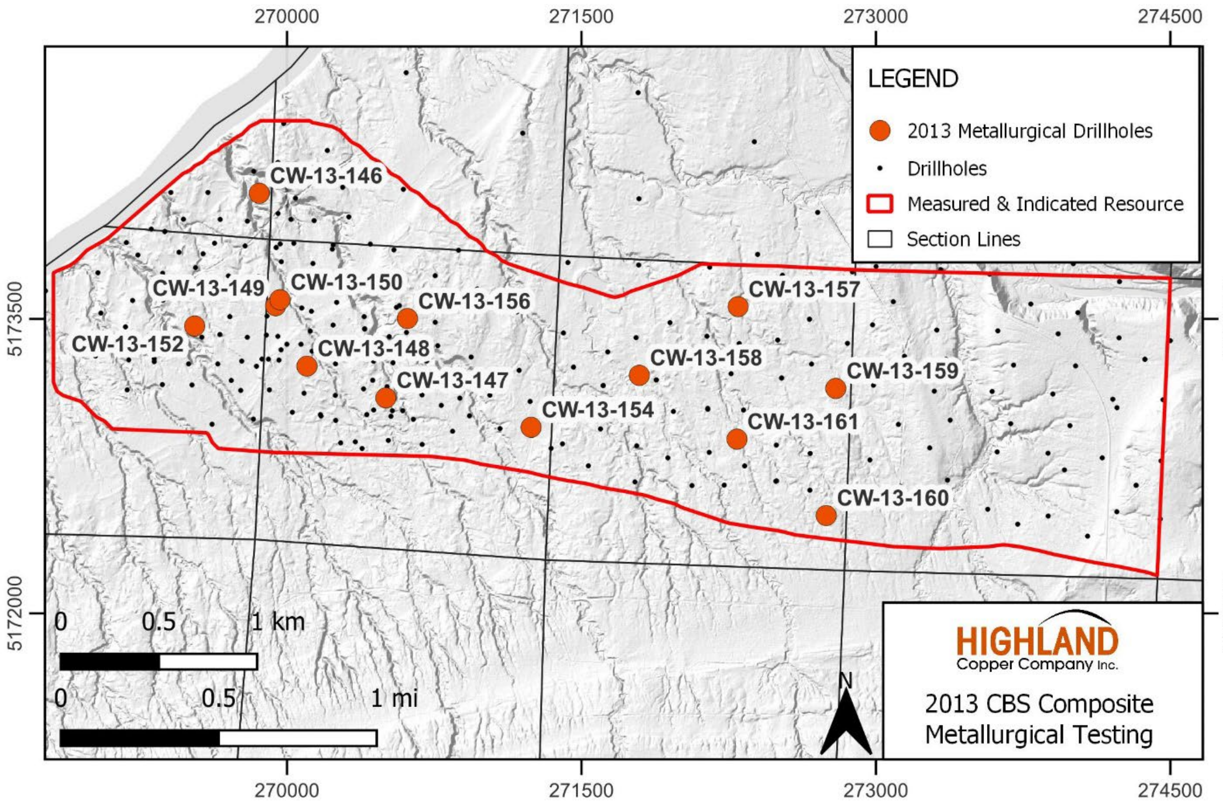
The results from the locked cycle tests suggested an average Cu recovery of 85.5% with a copper concentrate of 23.9% (Main Zone) and 16.6% Cu (Section 6) for the last two (2) cycles (5 & 6). A copper concentrate grade of 24% with 86% Cu recovery has been used for the FS 2012. Additional test work was recommended in the FS 2012 due to limited sampling areas.

## 13.3 2013 Locked Cycle Flotation Tests

Following the 2012 FS recommendations, additional test work was carried out on new drill cores from the Main Zone and Section 6 at SGS Lakefield (CBS composite sample). The main purpose of this test work 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 Section 6.

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

**Figure 13.3: CBS 2 Sample Locations**




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

### 13.3.1 Head Assay

One (1) 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).

### 13.3.2 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**

| NaSH Dosage (g/t) | Cumulative Time (Minute) | Mass Recovery (%) | Cumulative Grade (%) | | | | | Cumulative Recovery (%) | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Cu | Ag (g/t) | Fe | $S_r$ | Insol. | Cu | Ag | Fe | $S_r$ | 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.

#### 13.3.3 Locked Cycle Flotation

Seven (7) 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 (7) tests are summarized in Table 13.6.

**Figure 13.4: Locked Cycle Flowsheet**

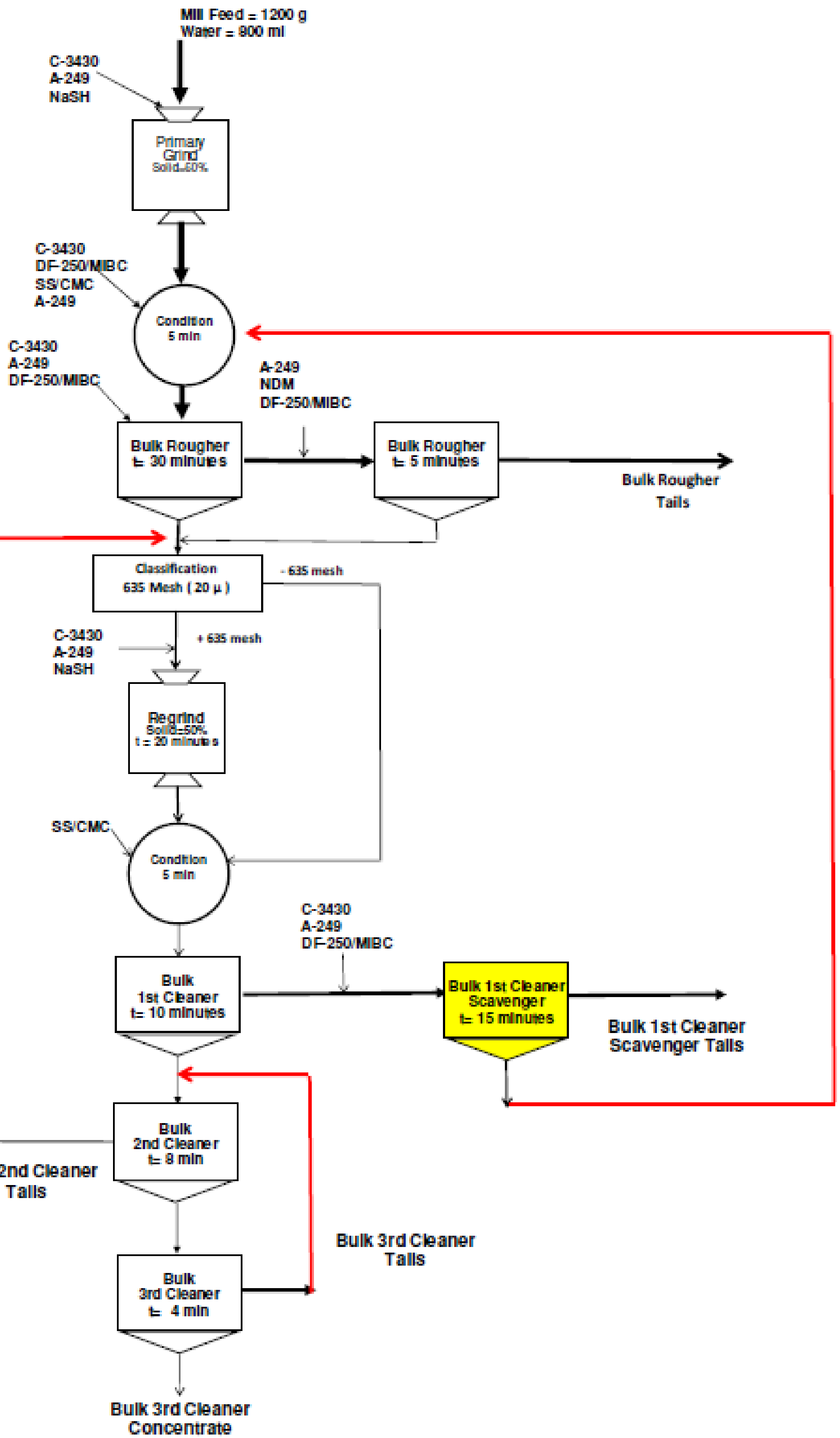


**Table 13.6: CBS2 Locked Cycle Flotation Test Results Summary**

| Cycle | Mass Recovery (%) | Grade (%) | | | | | Recovery (%) | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Cu | Ag (g/t) | Fe | $S_r$ | Insol. | Cu | Ag | Fe | $S_r$ | 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 were 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 performance was lower than the FS 2012.

### 13.3.4 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 (1) 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 test work results are summarized below. For detailed test work 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 (10) 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 |

**Note 1: A x b from DWT.*
**Note 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 |

### 13.3.5 Sample Preparation and Testing Matrix

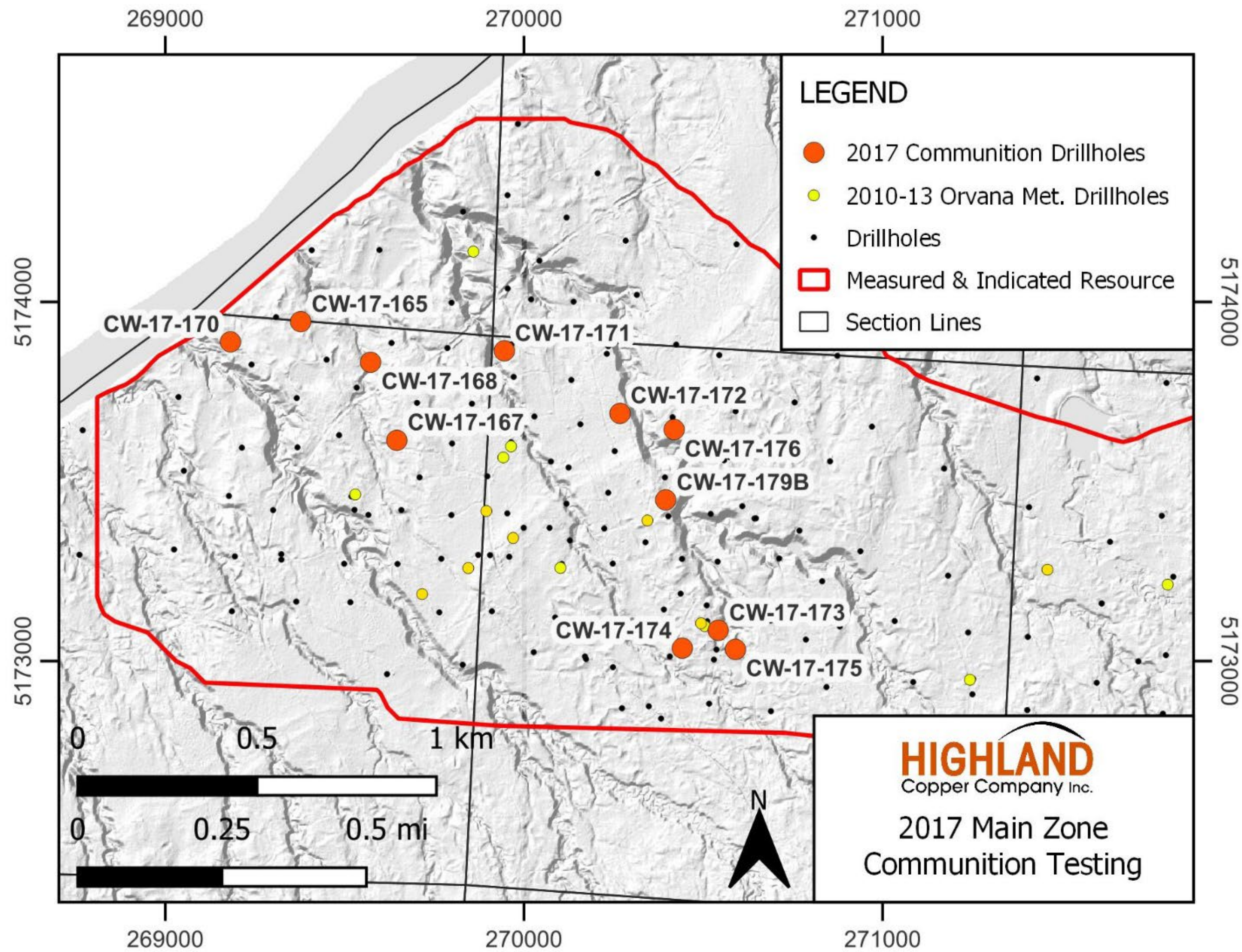
A total of 26 core boxes from 11 drill holes were received at SGS Lakefield on June 22, 2017. These samples were used to generate nine (9) comminution samples. Of these samples, six (6) consisted of drill hole composites representing a blend of the three (3) ore types, while the three (3) other samples were ore type composites. One (1) ore type composite represented the Grey Laminated ore type, made from three (3) holes, while material from three (3) drill holes was combined to make the Grey Laminated + Red Massive composite and the Domino composite. The information for the nine (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 (1) crate with 12 bags of whole PQ core was received. The 12 bags represented material from four (4) drill holes, and each bag contained one (1) ore type sample. A single PQ composite was made by combining the material in nine (9) 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** |

### 13.3.6 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 (8) 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 | $t_a$[1] | Hardness Percentile | DWI (kWh/m³) | $M_{ia}$ (kWh/t) | $M_{ih}$ (kWh/t) | $M_{ic}$ (kWh/t) | SCSE (kWh/t) | Relative Density |
|---|---|---|---|---|---|---|---|---|---|---|---|
| CW-17-185/186/187 PQ Comp | 51.1 | 1.07 | 54.7 | 0.63 | 30 | - | - | - | - | 8.6 | 2.70 |
| CW-17-185/186/187 PQ Comp | *58.2* | *0.84* | *48.9* | *0.47* | - | *5.8* | *17.4* | *12.5* | *6.5* | *9.0* | *2.70* |
| CW-17-165 | *77.5* | *0.51* | *39.5* | *0.37* | - | *7.2* | *20.5* | *15.4* | *7.9* | *10.0* | *2.73* |
| CW-17-167 | *69.0* | *0.61* | *42.1* | *0.39* | - | *6.9* | *19.6* | *14.6* | *7.5* | *9.8* | *2.76* |
| CW-17-170 | *71.0* | *0.55* | *39.1* | *0.37* | - | *7.3* | *20.6* | *15.5* | *8.0* | *10.0* | *2.73* |
| CW-17-173 | *72.8* | *0.58* | *42.2* | *0.40* | - | *6.9* | *19.5* | *14.5* | *7.5* | *9.7* | *2.74* |
| CW-17-176 | *71.9* | *0.57* | *41.0* | *0.39* | - | *7.0* | *19.9* | *14.8* | *7.7* | *9.8* | *2.72* |
| Grey Laminated + Red Massive | *78.9* | *0.42* | *33.1* | *0.31* | - | *8.7* | *23.7* | *18.4* | *9.5* | *10.9* | *2.73* |
| Domino | *68.5* | *0.65* | *44.5* | *0.42* | - | *6.4* | *18.6* | *13.6* | *7.0* | *9.5* | *2.74* |

**Note: SMC results are presented in italics.*
**Note1: The $t_a$ 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 ($t_a$) 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 JK Tech'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.

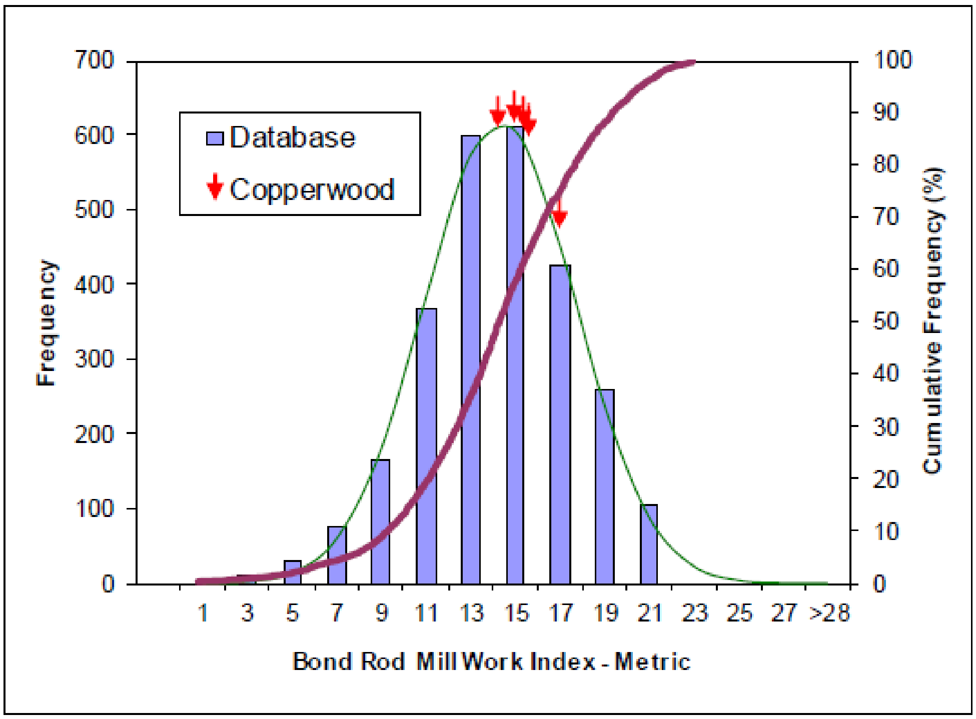
### 13.3.7 Bond Rod Mill Grindability Test

Five (5) 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 (5) samples tested.

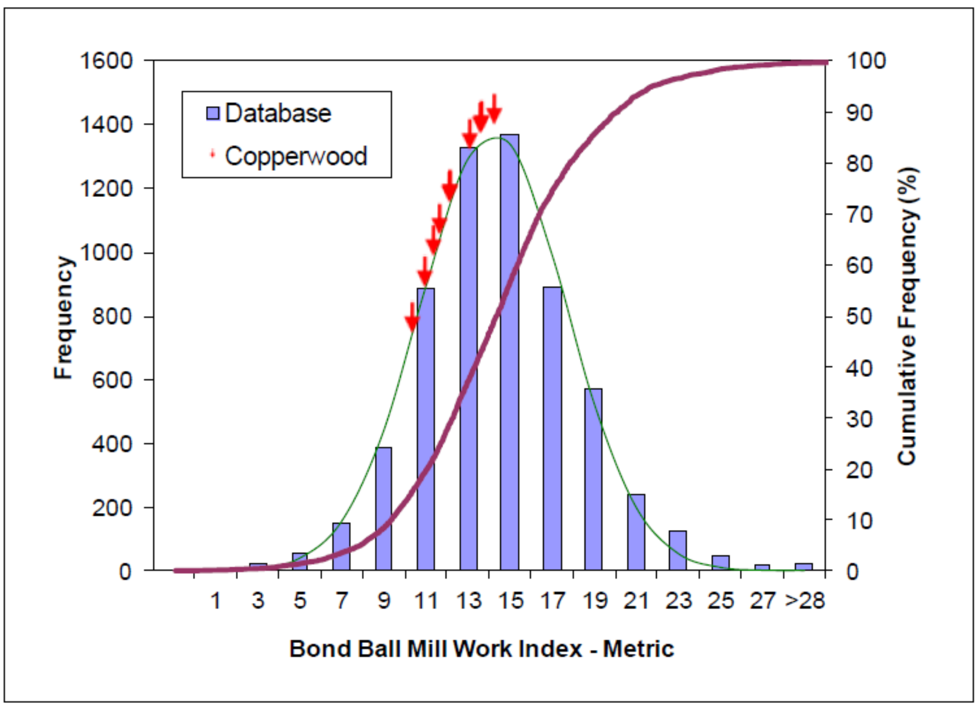
### 13.3.8 <u>Bond Ball Mill Grindability Test</u>

Ten (10) 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 (6) out of ten (10) samples fell in the soft range of hardness of the SGS database, while four (4) samples ("CW-17-167", "CW-17-176", "Grey Laminated", and "Grey

Laminated + Red Massive") were categorized as moderately soft to medium. The attained $P_{80}$ values varied from 42 to 46 microns. The PQ composite was the softest sample among the ten (10) samples tested.

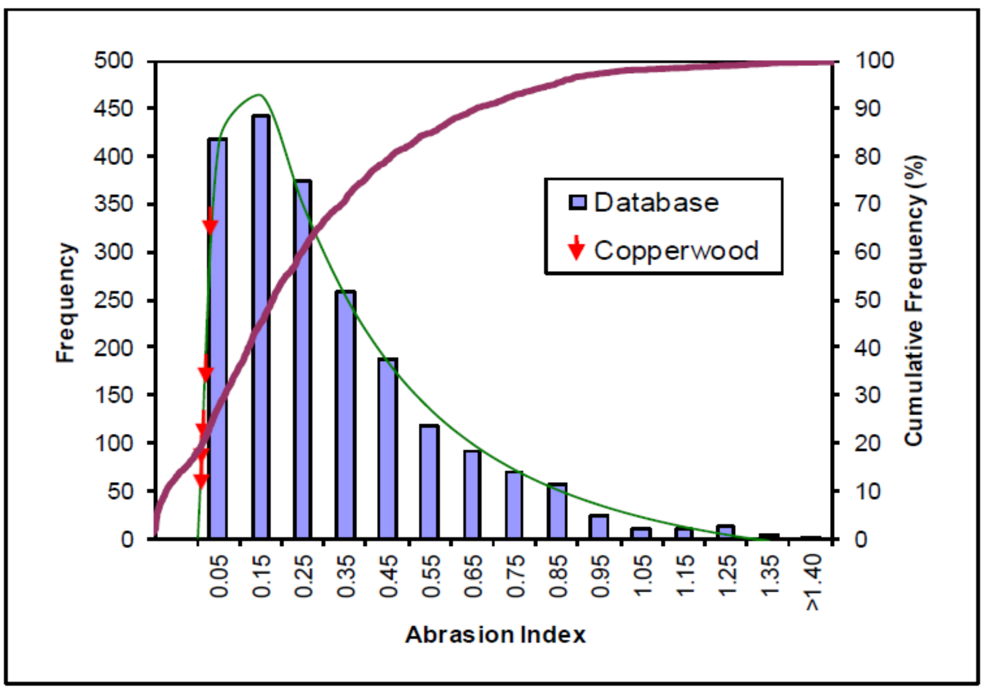
### 13.3.9 Bond Abrasion Test

Six (6) 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.

## 13.4 2017 Grindability Tests – Sections 5 and 6

Additional samples from Section 5 and Section 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 test work results are summarized below. For detailed test work 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 Sections 5 and 6**

| Statistics | 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 |
| CW-17-163-166 | - | - | - | - | 12.6 | - |
| CW-17-169-196 | - | - | - | - | 12.6 | 0.034 |
| CW-17-181-183 | - | - | - | - | 13.1 | 0.093 |
| CW-17-188-195 | - | - | - | - | 12.6 | 0.072 |
| CW-17-189-194 | - | - | - | - | 12.8 | - |
| CW-17-190-192 | - | - | - | - | 14.2 | 0.087 |
| CW-17-191-197 | - | - | - | - | 12.5 | - |

**Note1: The $t_a$ value reported as part of the SMC procedure is an estimate.*

### 13.4.1 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 (6) are from Section 5 and seven (7) are from Section 6.

The information for the 13 comminution samples is summarized in Table 13.16 and the drill hole locations are shown in Figure 13.9.

**Table 13.16: Sections 5 and 6 Test Matrix**

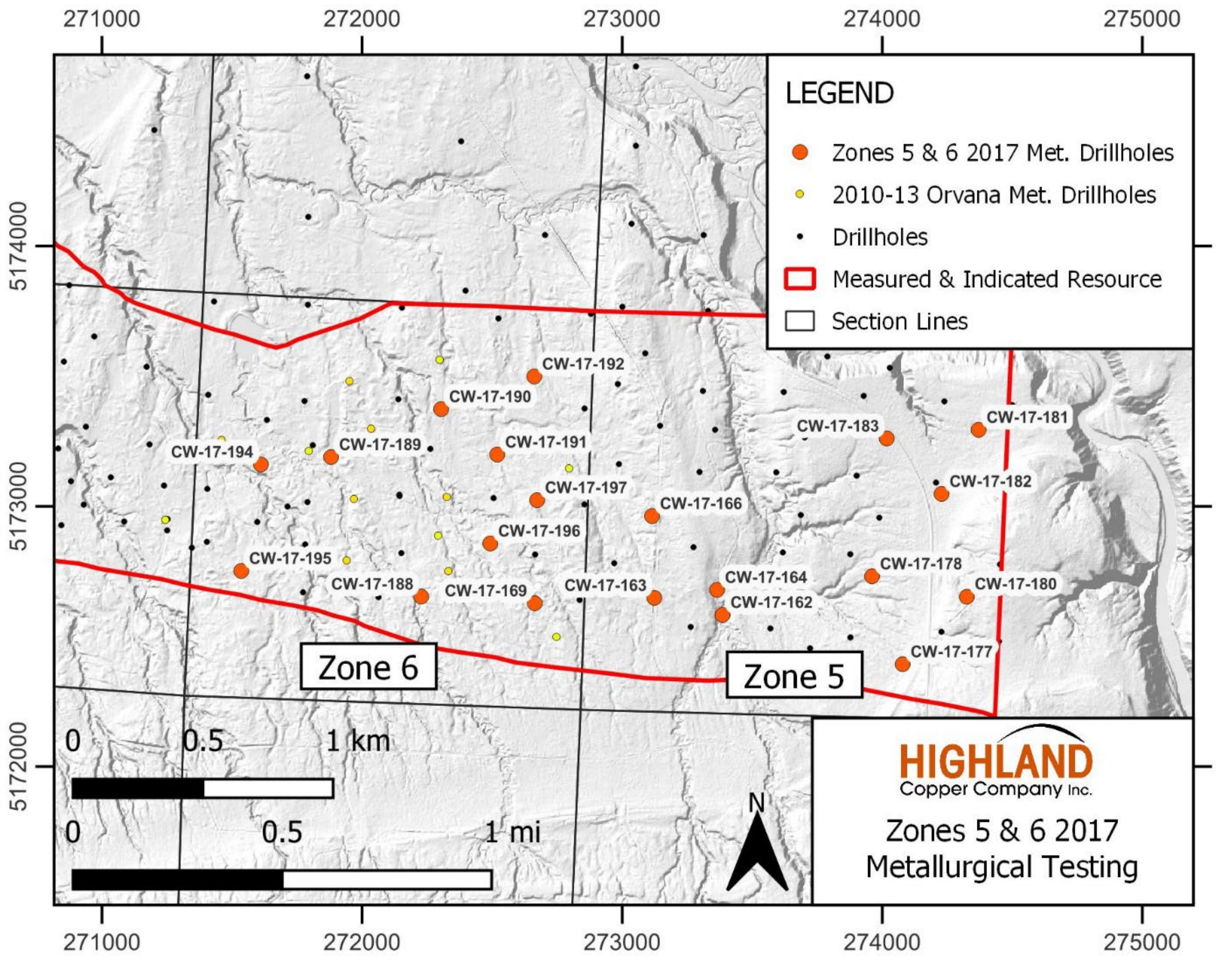
| Sample | Sample ID | SMC | BWI | Head Characterization | Bond Abrasion | Flotation |
|---|---|---|---|---|---|---|
| Sample 1 | CW-17-162 | x | x | x | | x |
| Sample 2 | CW-17-178 | x | x | x | | x |
| Sample 3 | CW-17-182 | x | x | x | | x |
| Sample 4 | CW-17-164 | | | x | | x |
| Sample 5 | CW-17-177 | | | 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 | ½ Core | RLAM to DOMN |
| | 8 | CW-17-164 | Section 5 | ½ Core | CHSA to CHSA |
| Sample 5 | 9 | CW-17-177 | Section 5 | ½ Core | DOMN to CHSA |
| | 10 | CW-17-177 | Section 5 | ½ Core | UPSA to DOMN |

| Sample / Composite | Box No. | Core No. | Core Location | Core Type | Rock Type |
|---|---|---|---|---|---|
| Sample 6 | 11 | CW-17-180 | Section 5 | ½ Core | CHSI to CHSH |
| | 12 | CW-17-180 | Section 5 | ½ Core | RSIL to DOMN |
| Composite 1 | 13 | CW-17-169 | Section 6 | ½ Core | DOMN to CHSA |
| | 14 | CW-17-169 | Section 6 | ½ Core | RLAM to DOMN |
| | 15 | CW-17-196 | Section 6 | ½ Core | DOMN to CHSA |
| | 16 | CW-17-196 | Section 6 | ½ Core | RLAM to DOMN |
| Composite 2 | 17 | CW-17-188 | Section 6 | ½ Core | DOMN to CHSA |
| | 18 | CW-17-188 | Section 6 | ½ Core | RLAM to DOMN |
| | 19 | CW-17-195 | Section 6 | ½ Core | RLAM to DOMN |
| | 20 | CW-17-195 | Section 6 | ½ Core | DOMN to CHSA |
| Composite 3 | 21 | CW-17-190 | Section 6 | ½ Core | DOMN to CHSA |
| | 22 | CW-17-190 | Section 6 | ½ Core | RLAM to DOMN |
| | 23 | CW-17-192 | Section 6 | ½ Core | CHSA to CHSA |
| | 24 | CW-17-192 | Section 6 | ½ Core | RLAM to CHSA |
| Composite 4 | 25 | CW-17-181 | Section 5 | ½ Core | DOMN to CHSA |
| | 26 | CW-17-181 | Section 5 | ½ Core | RSIL to DOMN |
| | 27 | CW-17-183 | Section 5 | ½ Core | UPSA to CHSI |
| Composite 5 | 28 | CW-17-189 | Section 6 | ½ Core | RLAM to DOMN |
| | 29 | CW-17-189 | Section 6 | ½ Core | CHSA to CHSA |
| | 30 | CW-17-194 | Section 6 | ½ Core | DOMN to CHSA |
| | 31 | CW-17-194 | Section 6 | ½ Core | RLAM to DOMN |
| Composite 6 | 32 | CW-17-191 | Section 6 | ½ Core | DOMN to CHSA |
| | 33 | CW-17-191 | Section 6 | ½ Core | RLAM to DOMN |
| | 34 | CW-17-197 | Section 6 | ½ Core | RLAM to CHSA |
| Composite 7 | 35 | CW-17-163 | Section 5 | ½ Core | RLAM to CHSA |
| | 36 | CW-17-166 | Section 5 | ½ Core | RLAM to CHSA |

**Figure 13.9: Section 5 and 6 Samples Location Map**




### 13.4.2 <u>SMC Tests</u>

SMC tests were done on three (3) 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/m³) | $M_{ia}$ (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 |

**Note1: The $t_a$ value reported as part of the SMC procedure is an estimate.*

Section 5 and Section 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.

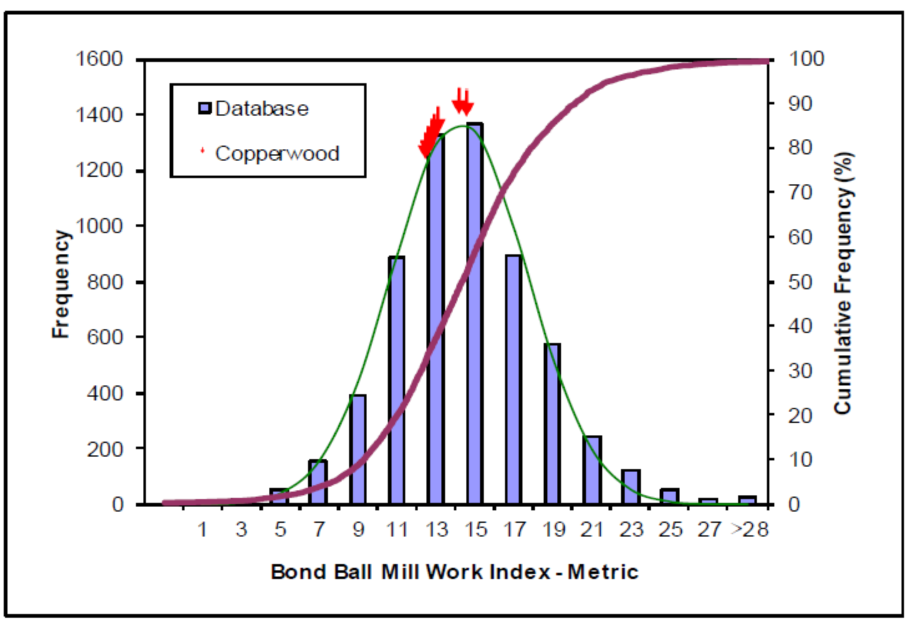
### 13.4.3 Bond Ball Mill Grindability Test

Ten (10) 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 ten (10) samples fell in the moderately soft to medium range of hardness. The attained $P_{80}$ values varied from 43 to 46 microns. All of these results are comparable to the Main Zone results.

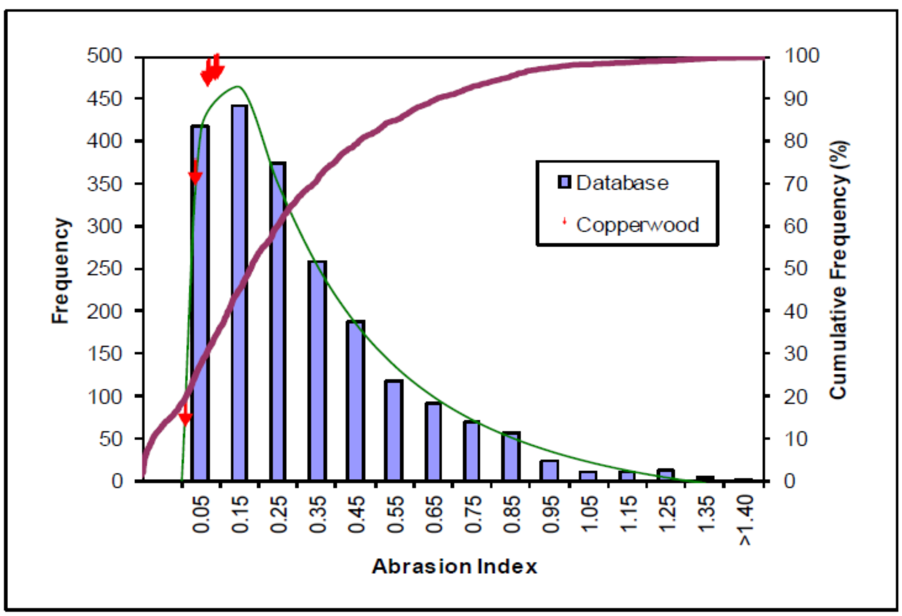
### 13.4.4 Bond Abrasion Test

Seven (7) 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 Section 5 and Section 6 are slightly more abrasive than samples from Main Zone.

### 13.4.5 Special Jar Mill Grindability Test and SMD Lab Test

One (1) 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 (80%) 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 80% 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 (10%) 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 more coarse 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 Test Work 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 |

### 13.4.6 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/cm$^3$. The sample weight received was 6.5 kg, allowing ORC to perform a standard small sample HIGmill™ test. The coefficient of determination from the test work data, denoted as R2 ('R squared') = 0.9948, indicating good accuracy of the results. The grindability signature plot curve has the equation:

SGE = 79,395 x 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.

## 13.5 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 test work results are summarized below. For detailed test work procedures and results, refer to document *"An Investigation into Optimization Flotation Test Work on Material from Copperwood Deposit*", Project No. 16256-002", prepared by SGS Canada Inc, dated May 2018.

### 13.5.1 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

### 13.5.2 Sample Location and Composite Definition

Samples from the Main Zone were collected from five (5) different drill holes. Two (2) composites were collected from each drill holes giving a total of ten (10) 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)**

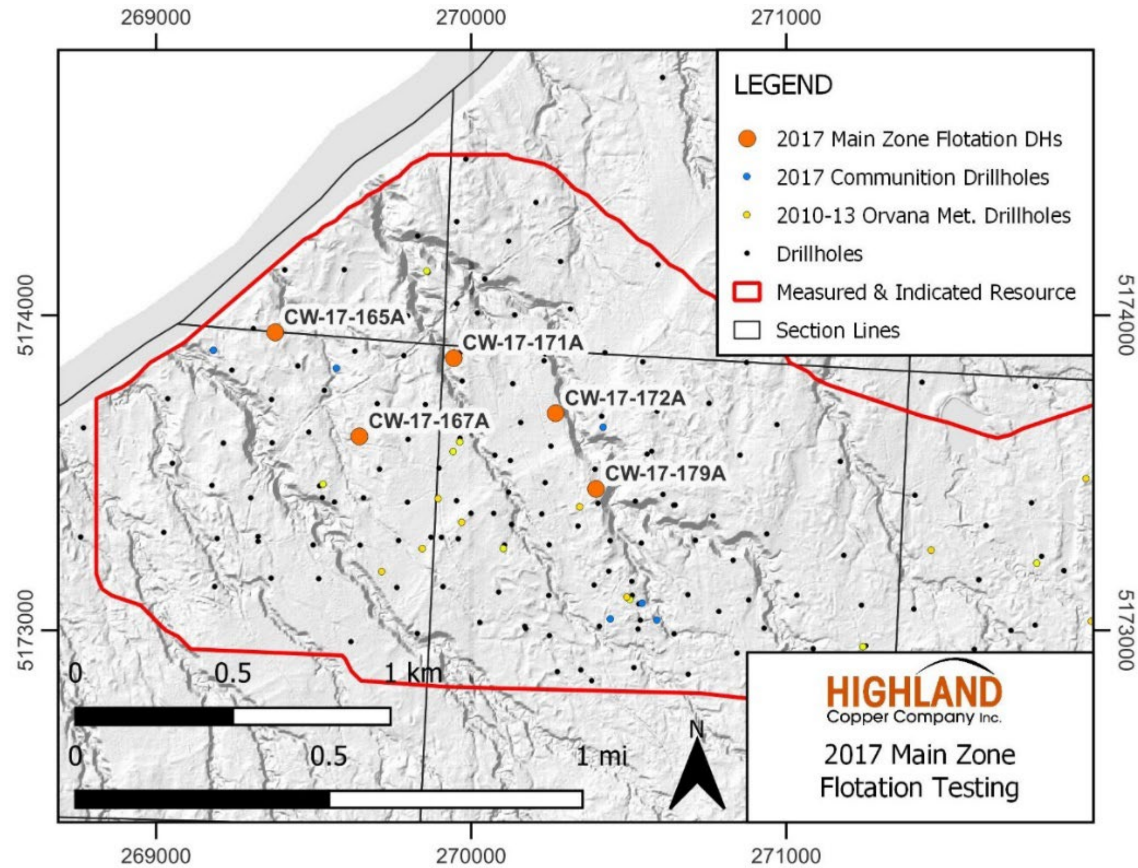



**Table 13.23: Flotation Sample Grinding**

| | 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** | |

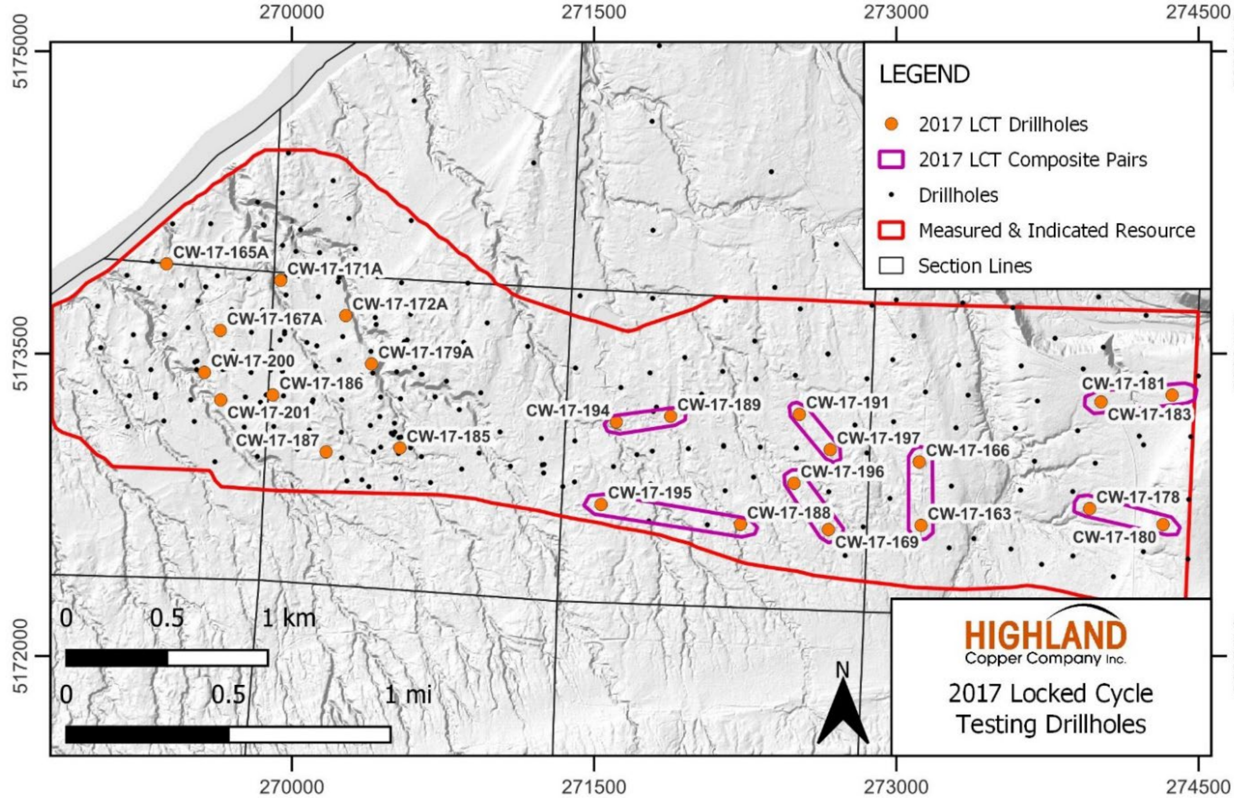
**Note: Refer to for sample location.*

Overall, 77 batch flotation tests were performed. The first 55 batch flotation and the first 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 test work 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 test work, 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 | Section 6 |
| CW-17-200 | Main Zone | CW-17-163-166 | Section 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 | Section 6 | CW-17-167A | Main Zone |
| CW-17-172A | Main Zone | CW-17-191-197 | Section 6 |
| CW-17-188-195 | Section 6 | CW-17-181-183 | Section 5 |
| CW-17-178-180 | Section 5 | - | - |

**Figure 13.13: LCT Samples Drill Holes Map**




### 13.5.3 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 $H_2SO_4$ | % | 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 |

### 13.5.4 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.

### 13.5.5 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 test work 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 test work 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.

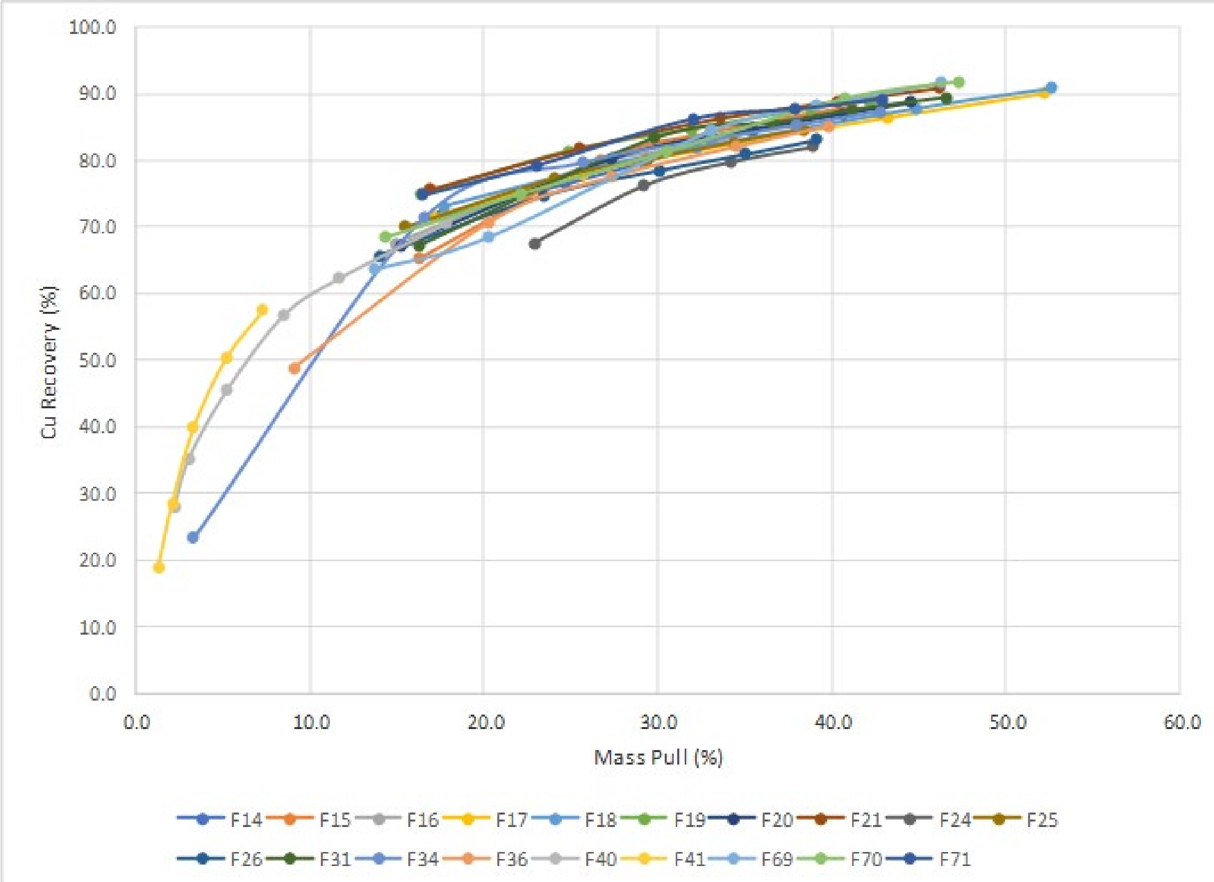
Modifications were made to the flotation time and reagents dosages, which provided higher mass pull and copper recovery (i.e., a mass pull from 40 to 46 % mass pull and a copper recovery of 91-92% - See results from tests F69 to F71). These tests formed the basis for the LCT 8 which was considered the 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 | | NaHS | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 8L | 4L | 53 | 45 | 36 | Nat | 9/10 | Standard | High | Same | 1L | Yes | No | Low | 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 | |

| Test | Objective | Standard Cell | | Primary Grind | | | pH | | Collector g/t | | Cell Size at End | | HIC | | NaHS | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 8L | 4L | 53 | 45 | 36 | Nat | 9/10 | Standard | High | Same | 1L | Yes | No | Low | Standard | High |
| 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**




### 13.5.6 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 is 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.

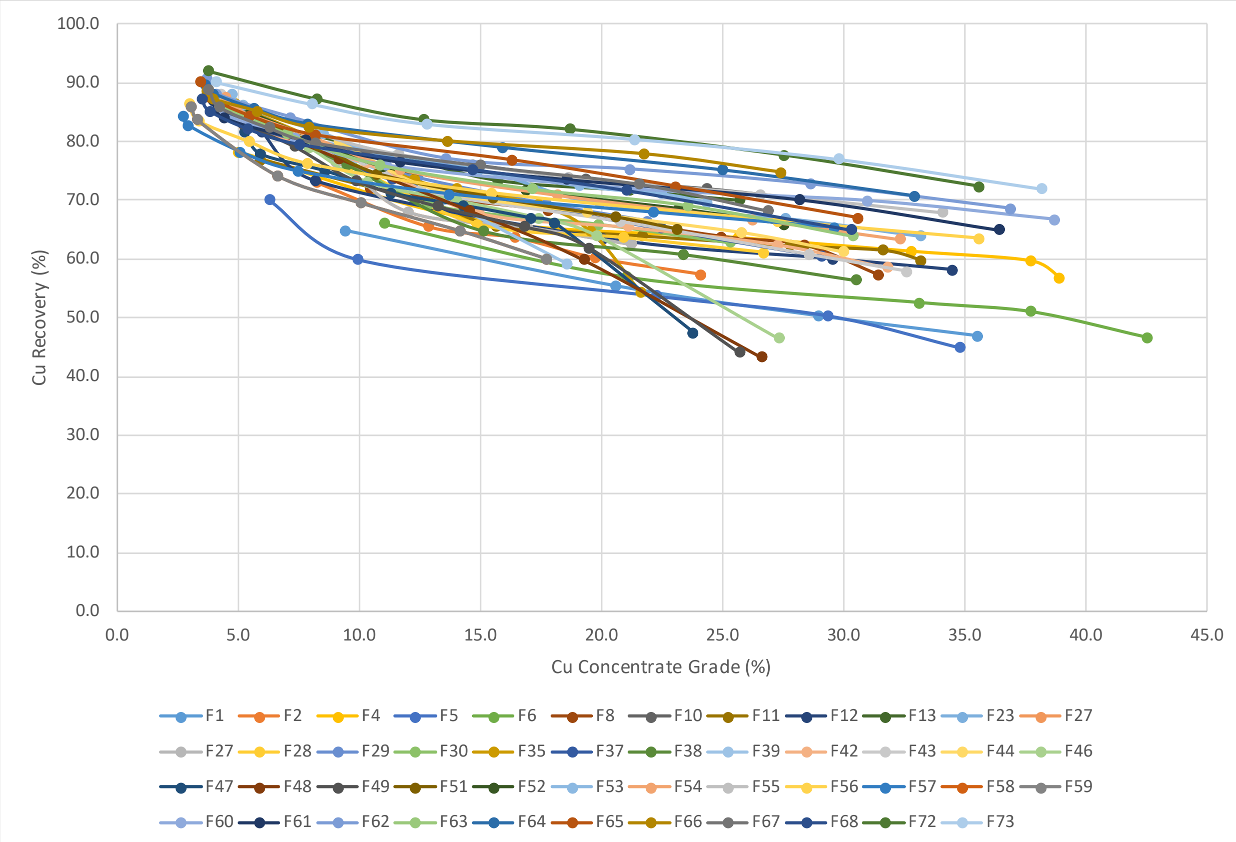
The initial locked cycle tests (LCT-1 to 3) were carried out using the F55 test flowsheet and conditions. The results were promising. Additional cleaner tests were performed using flowsheet and reagents modifications to improve the first cleaner recovery. Tests F56 to F68 improved the cleaner copper 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 form 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 ($P_{80}$) 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 test work carried out by SGS Tucson (formerly Metcon). | x | | | | 20 | | | | | | x | x | x | x | x | x | | NA |
| F4 | To duplicate test work 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 $1^{st}$ 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 $1^{st}$ cleaner kinetic concentrates for potential mineralogical analysis. | x | | | | 13 | | | | | | x | x | x | x | | x | | NA |
| F42 | Repeat F22 and add more $1^{st}$ 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 |
| 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 |

| Test | Purpose | Primary Grind Size ($P_{80}$) 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 | |
| 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 Ro 1 (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**




### 13.5.7 <u>Flash Flotation Tests</u>

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 test work shows 28-30% recovery at one-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.

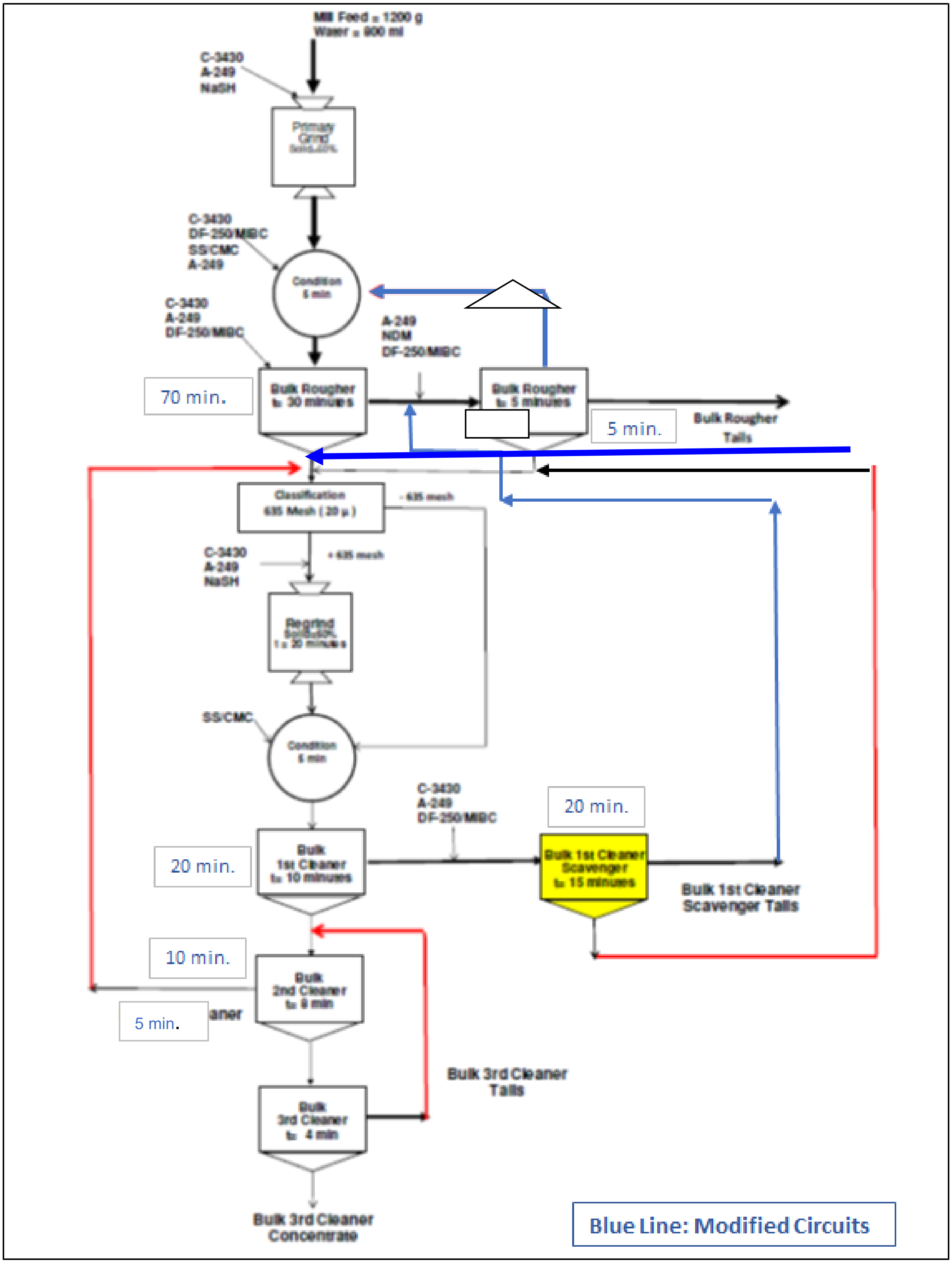
### 13.5.8 <u>Locked Cycle Flotation Tests</u>

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 (8) locked cycle tests combined with optimization batch tests were carried out to develop the selected flowsheet for the variability test work 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 (1) combined final tailing. 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 Test Work No. 8 Block Flow Diagram**



### 13.5.9 Variability Test work

A total of 17 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 (10) composites were used for the variability; 10 from the Main Zone which represents a major part of the Copperwood resource, four (4) from Section 6 and three (3) from Section 5. Table 13.28 shows the overall locked cycle test work results.

**Table 13.28: Overall Locked Cycle Test Work Results**

| LCT No. | Hole No. | Zone | Head Grade | | Concentrate | | Cu (%) | Ag (g/t) |
|---|---|---|---|---|---|---|---|---|
| | | | 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 | Section 6 | 1.00 | 1.30 | 22.5 | 22.0 | 86.9 | 64.1 |
| 15 | CW-17-169-196 | Section 6 | 1.24 | 1.40 | 22.0 | 18.0 | 88.1 | 63.2 |
| 16 | CW-17-163-166 | Section 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 | Section 6 | 1.17 | 2.20 | 24.9 | 37.0 | 88.1 | 68.1 |
| 24 | CW-17-191-197 | Section 6 | 1.15 | 1.10 | 21.9 | 15.0 | 86.3 | 58.4 |
| 25 | CW-17-181-183 | Section 5 | 1.36 | 1.70 | 27.0 | 29.0 | 83.4 | 71.6 |
| 26 | CW-17-178-180 | Section 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** |

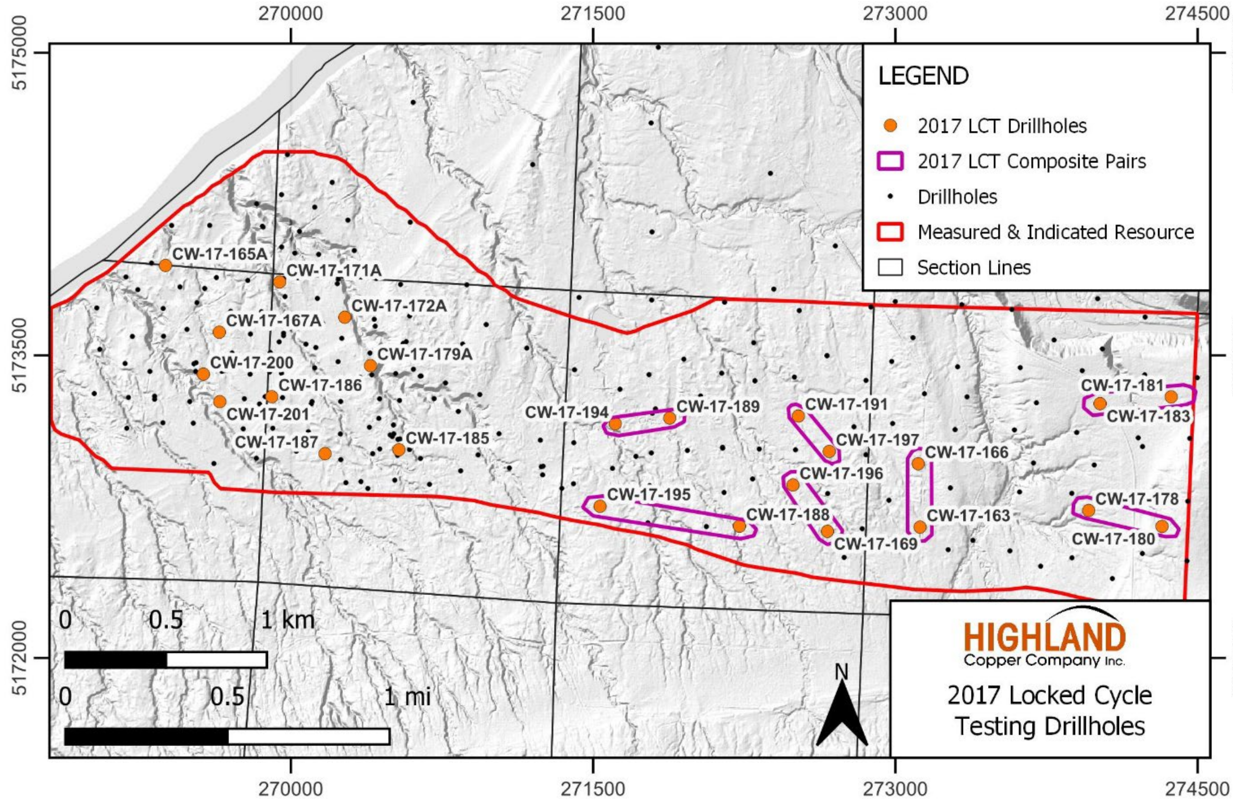
Table 13.29 illustrates the locked cycle test work results by zone.

**Table 13.29: Locked Cycle Test Work 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** | **Section** | 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** | **Section** | 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** | **Section** | 5 | **1.26** | **2.20** | **24.63** | **34.13** | **85.10** | **69.03** |

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

**Figure 13.17: Locked Cycle Test Work Results over the Copperwood Deposit**




### 13.5.10 MLA Analysis

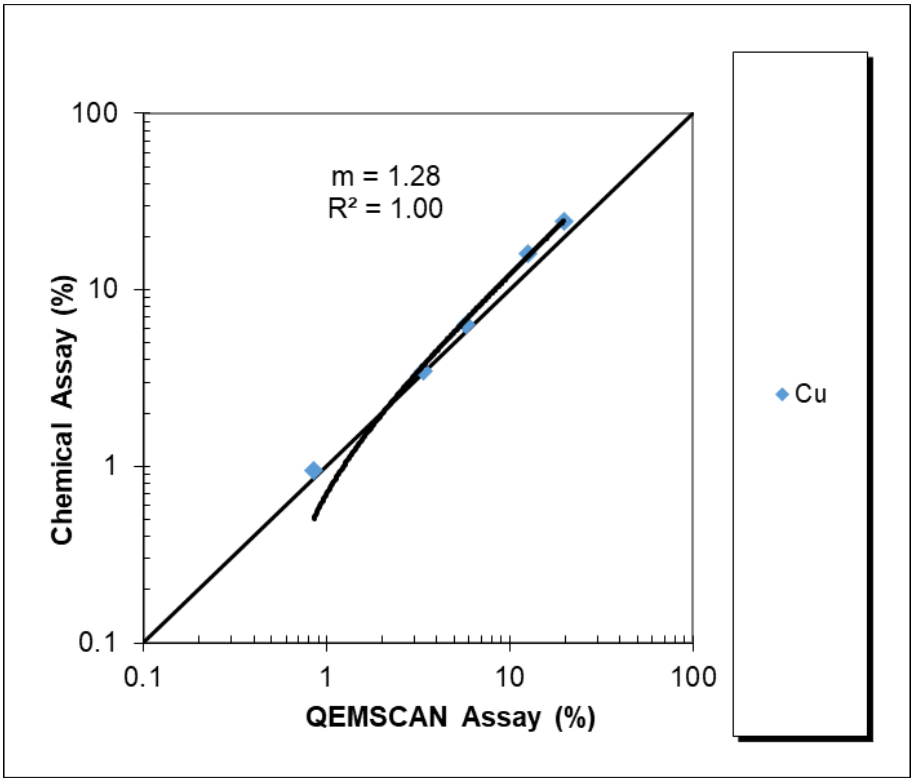
Concentrate and tails produced in Test F39R were sent to undergo mineral liberation analysis (MLA). Copper, molybdenite and overall mineralogy were determined from scanning electron microscope / electron dispersive spectroscopy ("SEM/EDX") data and MLA.

### 13.5.11 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**




### 13.5.12 Modal Mineralogy

Cleaner concentrates 1, 2, 3, cleaner scavenger concentrates 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 Cl Con 1 | F39R Cl Con 2 | F39R Cl Con 3 | F39R Cl Scav. Con | F39R Cl 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**

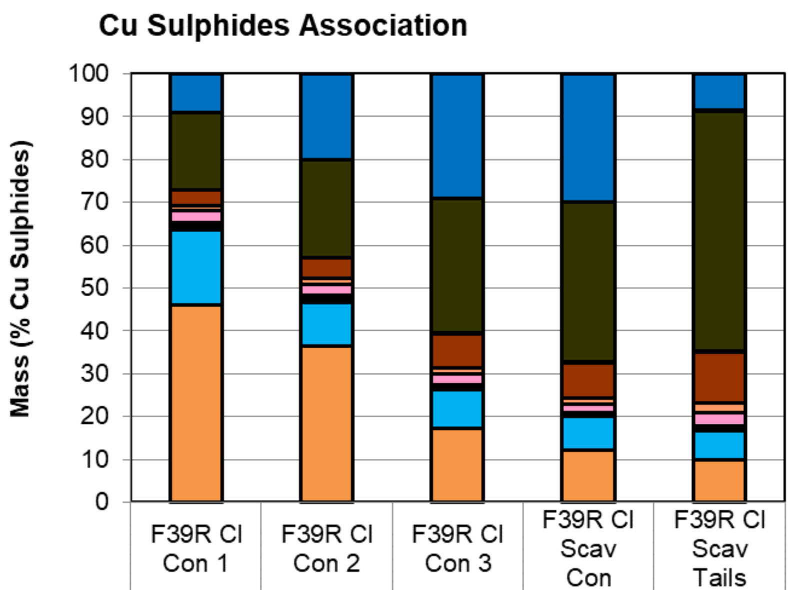
| Sample | | F39R CI Con 1 | F39R CI Con 2 | F39R CI Con 3 | F39R CI Scav. Con | F39R CI Scav. Tails |
|---|---|---|---|---|---|---|
| Calculated ESD Particle Size | | 17 | 17 | 20 | 21 | 18 |
| Mean Grain Size by Frequency (µm) | Chalcocite | 15 | 14 | 12 | 11 | 9 |
| | Fine Cu-Sulph / Sil | 13 | 13 | 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 |

### 13.5.13 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.

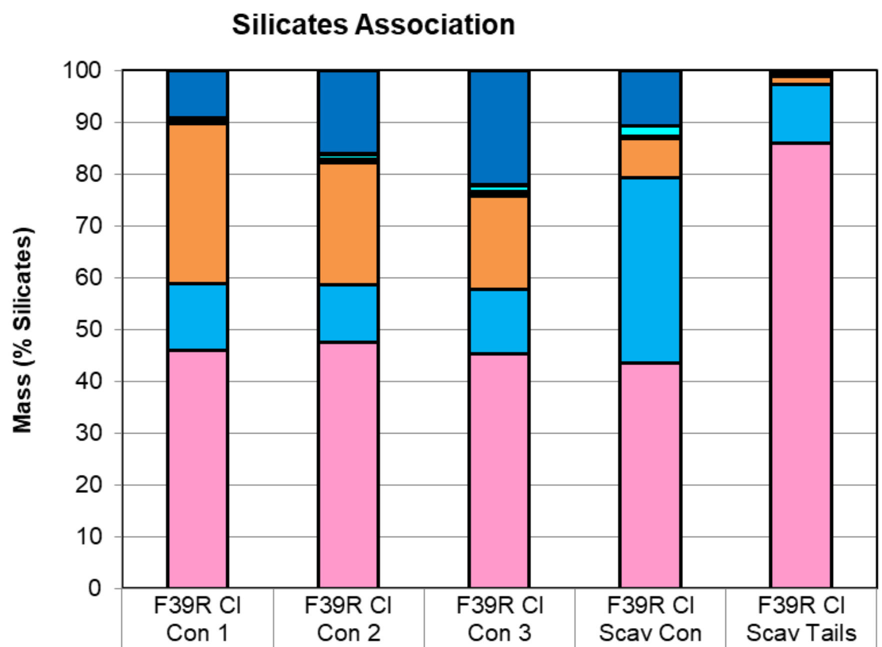
In the final cleaner concentrate (cleaner concentrate 3), copper is predominately carried out in the silicates and complex minerals while the copper is predominately associated with the silicates minerals in the tails.

**Figure 13.19: Copper Sulphides Association**




| | F39R Cl Con 1 | F39R Cl Con 2 | F39R Cl Con 3 | F39R Cl Scav Con | F39R Cl Scav Tails |
|---|---|---|---|---|---|
| Complex | 9.2 | 20.2 | 29.1 | 29.9 | 8.5 |
| CuS:Other | 0.0 | 0.0 | 0.1 | 0.0 | 0.1 |
| CuS:Silicates | 17.9 | 22.7 | 31.2 | 37.4 | 56.0 |
| CuS:Other Silicates | 0.1 | 0.1 | 0.2 | 0.1 | 0.1 |
| CuS:Amphibole/Pyroxenes | 0.1 | 0.1 | 0.1 | 0.1 | 0.2 |
| CuS:Clays/Micas/Chlorite | 3.6 | 4.8 | 8.1 | 8.3 | 11.8 |
| CuS:Feldspars | 1.2 | 1.3 | 1.3 | 1.3 | 2.3 |
| CuS:Quartz | 2.7 | 2.5 | 2.5 | 2.2 | 3.2 |
| CuS:Carbonates | 0.1 | 0.3 | 0.3 | 0.1 | 0.0 |
| CuS:Oxides | 1.5 | 1.4 | 0.8 | 0.4 | 1.1 |
| CuS:Cu-Silicates | 0.1 | 0.1 | 0.0 | 0.0 | 0.0 |
| CuS:Other Sulphides | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Lib Cu Sulphides | 17.6 | 10.1 | 9.1 | 7.9 | 6.9 |
| Free Cu Sulphides | 46.1 | 36.6 | 17.2 | 12.2 | 9.7 |

**Figure 13.20: Silicates Associations**




| | F39R Cl Con 1 | F39R Cl Con 2 | F39R Cl Con 3 | F39R Cl Scav Con | F39R Cl Scav Tails |
|---|---|---|---|---|---|
| Complex | 9.2 | 16.1 | 22.0 | 10.6 | 0.4 |
| Sil:Other | 0.2 | 0.1 | 0.2 | 0.1 | 0.2 |
| Sil:Carbonates | 0.3 | 0.9 | 1.2 | 1.9 | 0.2 |
| Sil:Oxides | 0.7 | 0.7 | 0.7 | 0.4 | 0.3 |
| Sil:Other Sulphides | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Sil:Cu Sulphides | 30.8 | 23.4 | 18.0 | 7.7 | 1.6 |
| Lib Silicates | 12.9 | 11.2 | 12.7 | 35.6 | 11.4 |
| Free Silicates | 46.0 | 47.5 | 45.2 | 43.6 | 85.9 |

### 13.5.14 Copper Deportment

The copper deportment data for the five (5) 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 Deportment of Copper**

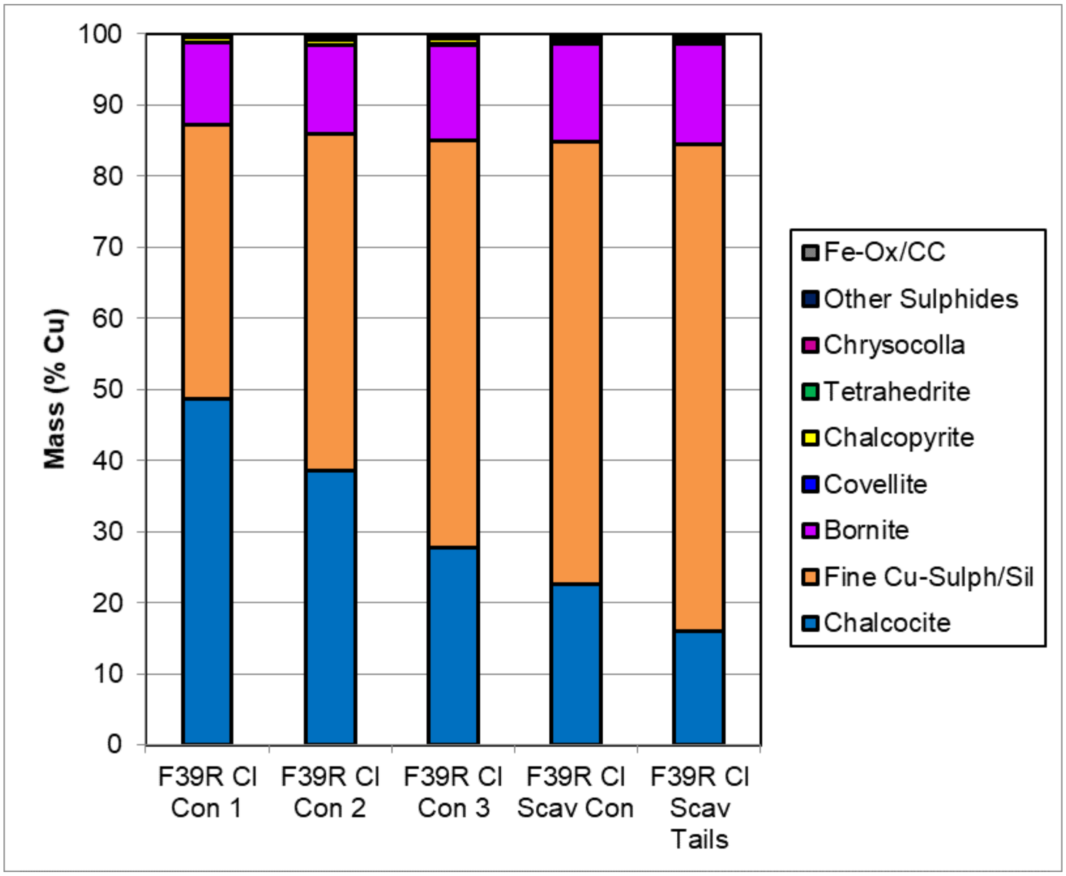
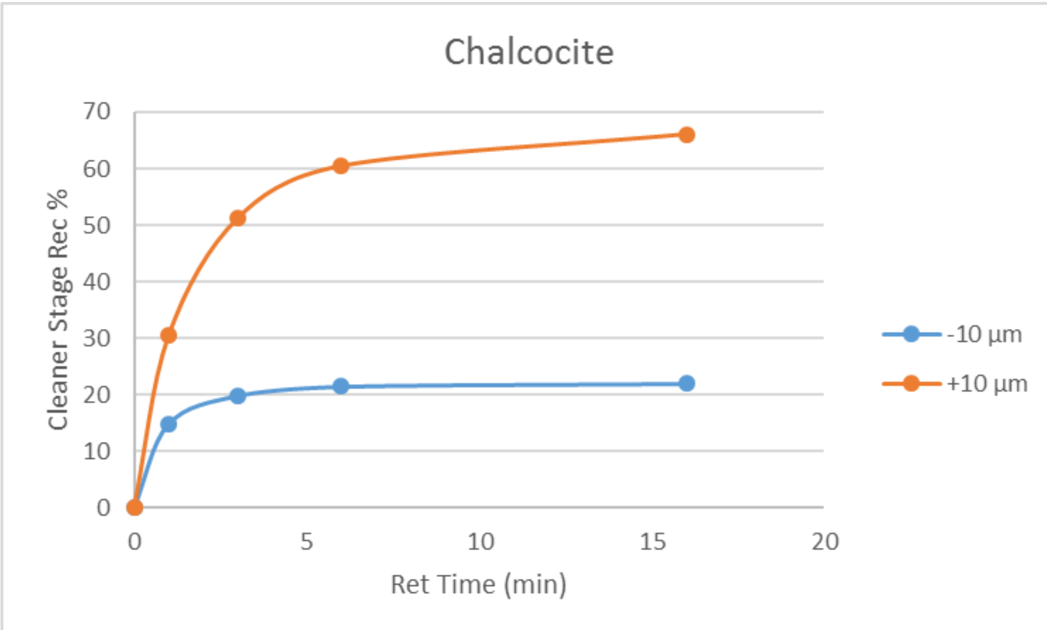



**Table 13.32: Elemental Deportment of Copper**

| Mineral Name | F39R Cl Con 1 | F39R Cl Con 2 | F39R Cl Con 3 | F39R Cl Scav. Con | F39R Cl 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 first 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 three (3) times greater than the recovery of fine particles (Figure 13.22).

**Figure 13.22: Chalcocite Recover vs. Particle Size**




#### 13.5.15 Locked Cycle Concentrate Specifications

Table 13.33 shows full chemical analyses of the locked cycle test work.

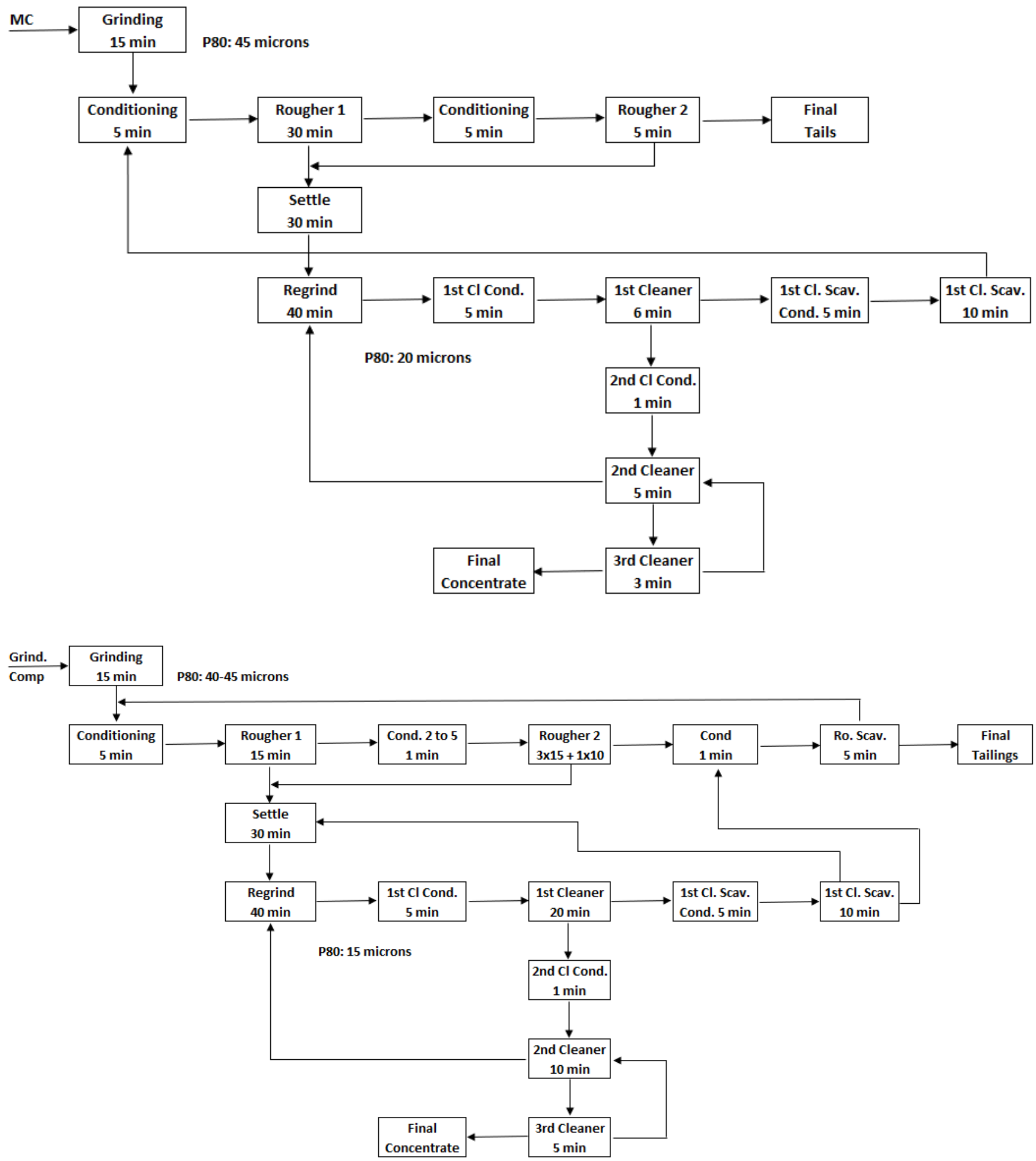
**Table 13.33: Copper Concentrate Specification**

| Locked Cycle Test Zone | LCT-10 3rd Cleaner Conc F | LCT-9 3rd Cleaner Conc F | LCT-12 - 3rd Cl Main Zone | LCT-13 - 3rd Cl Main Zone | LCT-23-24 Comb. 3rd Cl - Section 6 | LCT-25-26 Comb. 3rd Cl Conc F - Section 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 |
| $Al_2O_3$% | 8.30 | 9.34 | 8.81 | 7.93 | 9.07 | 8.63 |
| $Fe_2O_3$% | 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 |
| $K_2O$% | 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 |
| $Cr_2O_3$% | 0.043 | 0.069 | 0.1 | 0.082 | 0.14 | 0.18 |
| $V_2O_5$% | 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 | 6,530 | 7,690 | 7,420 | 5,770 | 6,370 | 7,170 |
| 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 | 2,940 | 2,330 | 99 | 102 | 110 | 144 |
| **Total (%)** | **96.88** | **97.22** | **98.23** | **97.91** | **97.09** | **96.17** |

### 13.5.16 Test work Discussion and Recommendations

Comprehensive test work programs have been carried out on Copperwood ores over the years with variable results. During the last test work 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 test work 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 test work 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 test work 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 test work 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 test work 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 test work, 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 processing 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 a pilot plant campaign to validate and optimize the process flowsheet, retention time, reagents type and addition points.

# 14. MINERAL RESOURCE ESTIMATES

*This section remains similar to the 2018 Technical Report, as the underlying block model remains unchanged based on drilling data provided on April 12th, 2018. No additional drilling has been undertaken on the project since the 2018 technical report. For this Technical Report, the only significant change was a reduction in the cut-off grade from 1.0% Cu to 0.9% Cu due to an increased copper price (USD 3.00/lb to USD 4.00/lb).*

## 14.1 Introduction

GMS 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 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 GMS' opinion, 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. James Purchase, P.Geo., Consulting Geologist of GMS, an independent "Qualified Person" as defined in NI 43-101. Geovia GEMS™ and Leapfrog Geo™ software were 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.

## 14.2 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. GMS 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.

### 14.2.1 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 summarized 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 of increasing length heading northwards depending on the mineralized horizon depth. Figure 14.1 illustrates the grid spacing for the Copperwood Project.

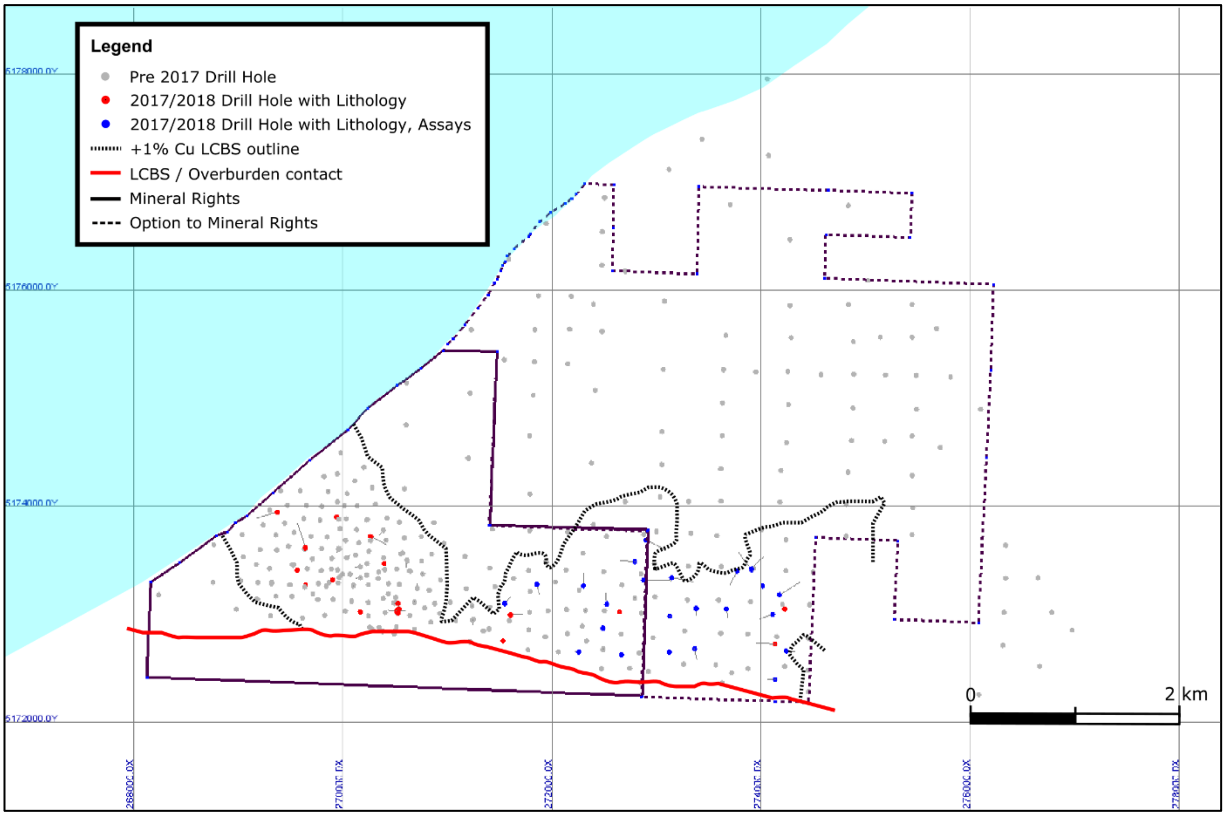
The final drill spacing is judged adequate to develop a reasonable model of the mineralization distribution, and to quantify its volume and quality with a high level of confidence.

**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 |

**Note: 48 drill holes with an additional 14 wedges*

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




### 14.2.2 Data Conditioning

GMS 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 not easily distinguished.

GMS 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). GMS 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.

## 14.3 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).

GMS 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.

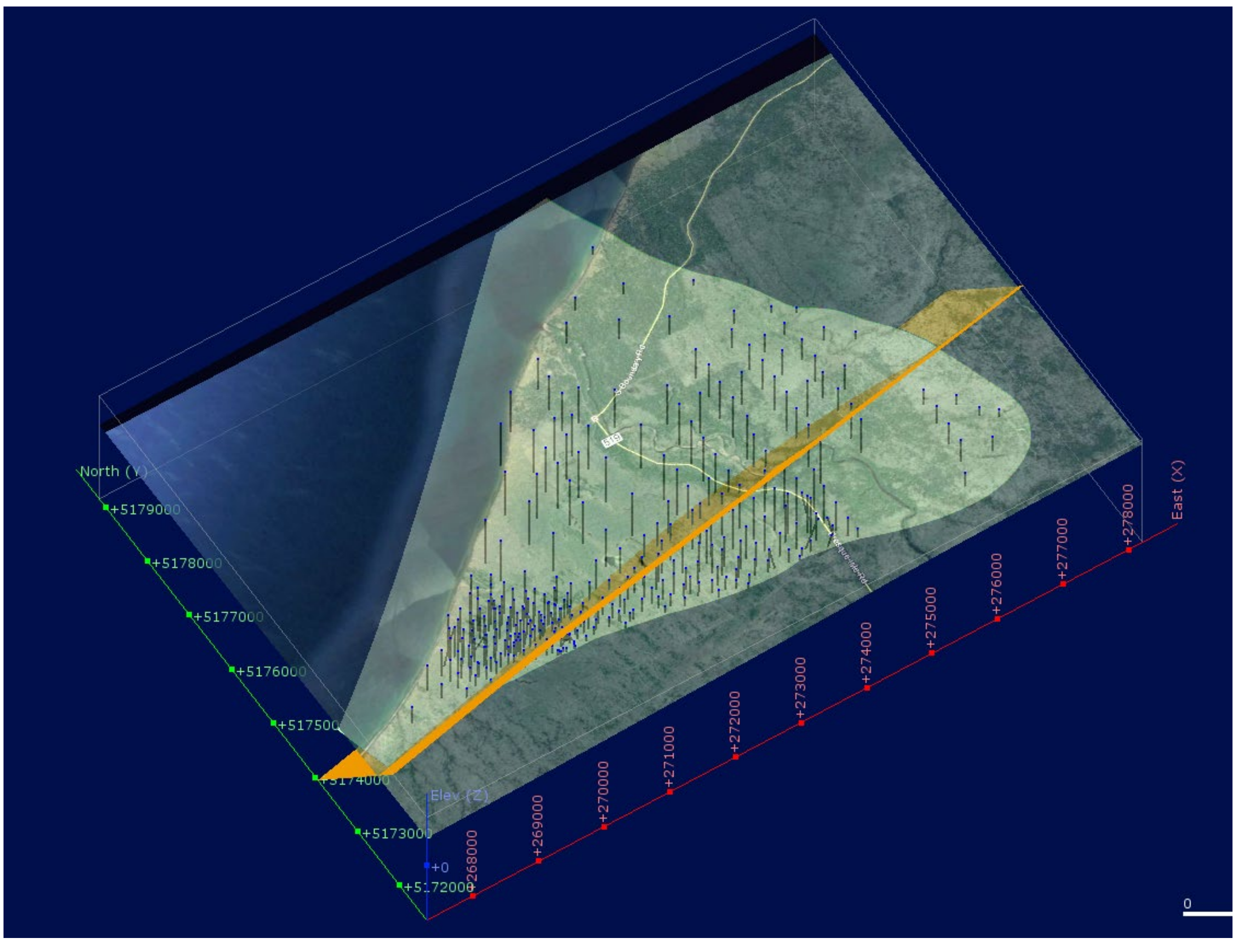
As the lithology units within the LCBS have a strong control on copper grade, no additional lower grade cut-off was applied during modelling of the LCBS. The constraints applied by modelling each unit are considered sufficient to accurately represent mineralisation boundaries.

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, GMS 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.

#### 14.3.1 Structural Model

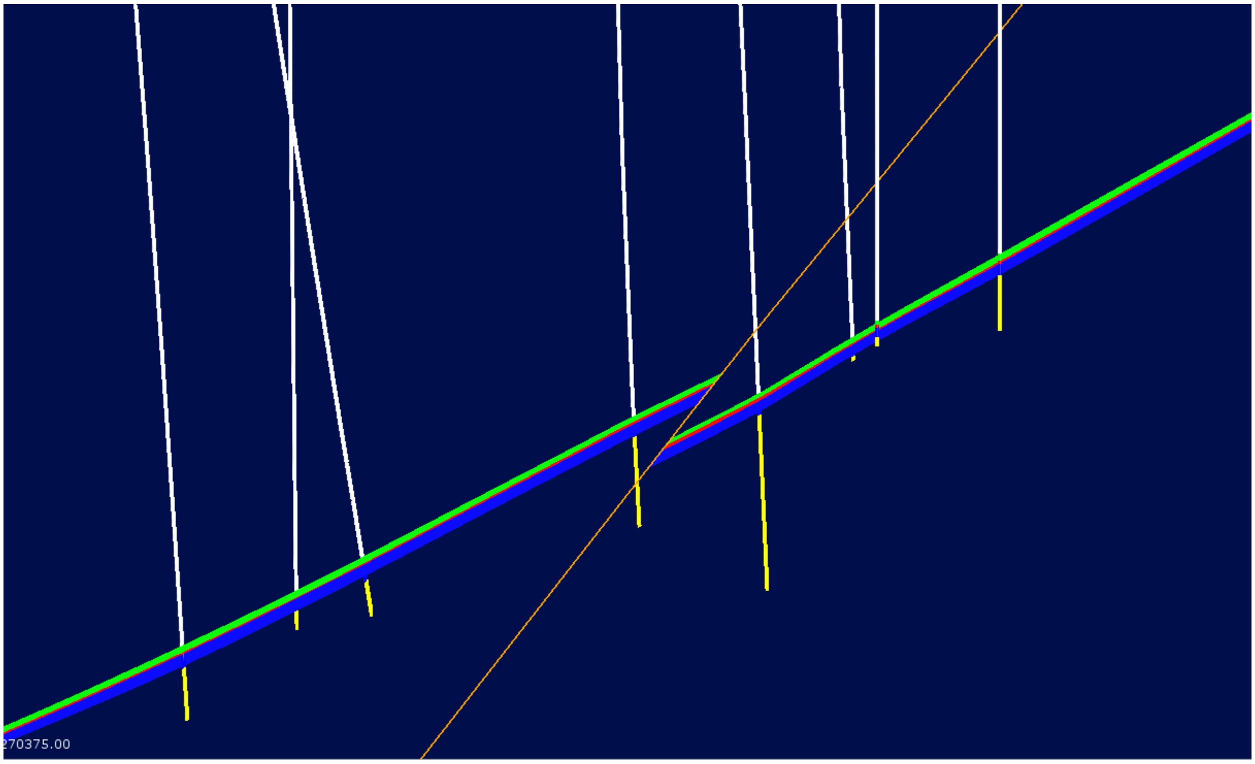
During the 2017 drilling program, a repetition of the LCBS was intersected in CW-17-186, which prompted a review of structural data with the Main Zone of the Copperwood deposit. The review delineated a thrust fault within the extents of 269,500 mE – 271,000 mE and was based off drill core observations from 11 drill holes. The thrust fault strikes around 80° azimuth, with a dip of 20° – 25° to the NNW. GMS 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, it is up to 8 m in some areas (Figure 14.3).

**Figure 14.3: Section 270375 mE Showing Displacement of UCBS (vertical exaggeration x 3)**




### 14.3.2 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**

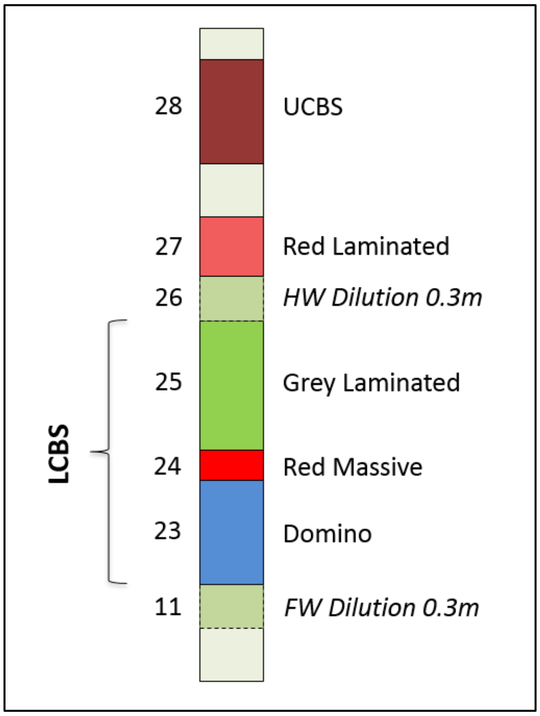



**Table 14.2: Average Vertical Thicknesses of the LCBS Units**

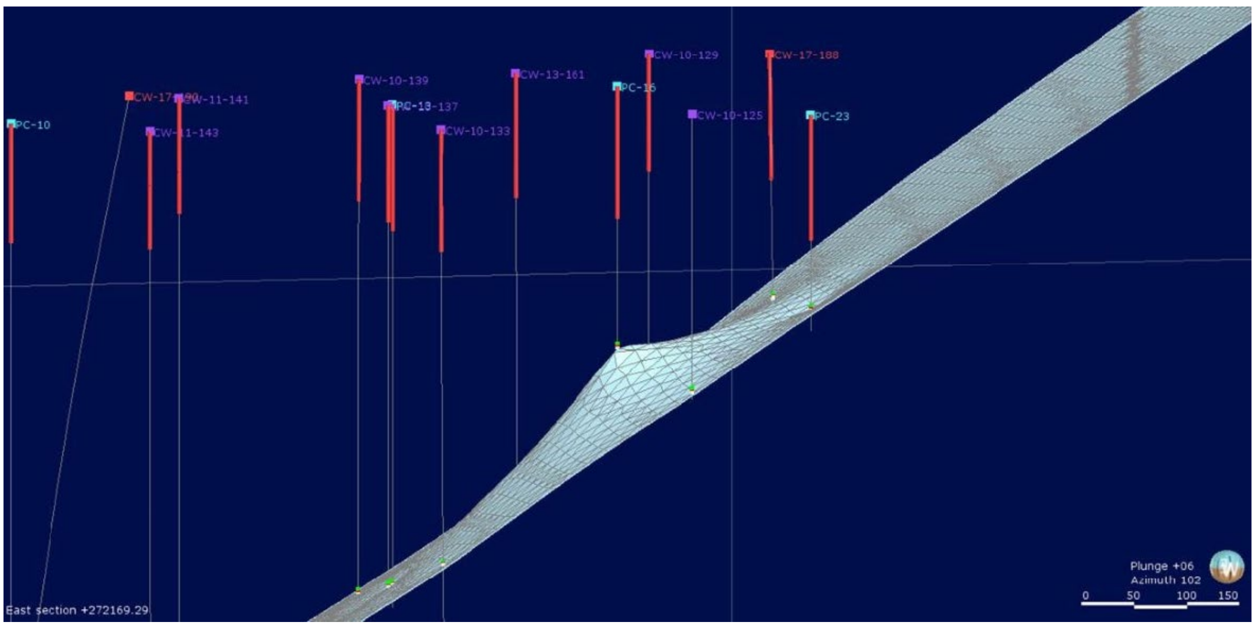
| Lithology (Code) | Average Thickness (m) |
|---|---|
| Gray Laminated (25) | 1.21 |
| Red Massive (24) | 0.36 |
| Domino (23) | 1.09 |
| LCBS (2345) | 2.66 |

Two (2) 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 GMS applied 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 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 GMS that PC 16 requires further confirmation. Consequently, 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 LCBS Interpretation**




### 14.3.3 <u>Weathering Wireframes</u>

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.

### 14.3.4 <u>Topography Surface</u>

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.

## 14.4 Statistical Analysis

### 14.4.1 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 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 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. GMS 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 LCBS (left) and UCBS (right)**

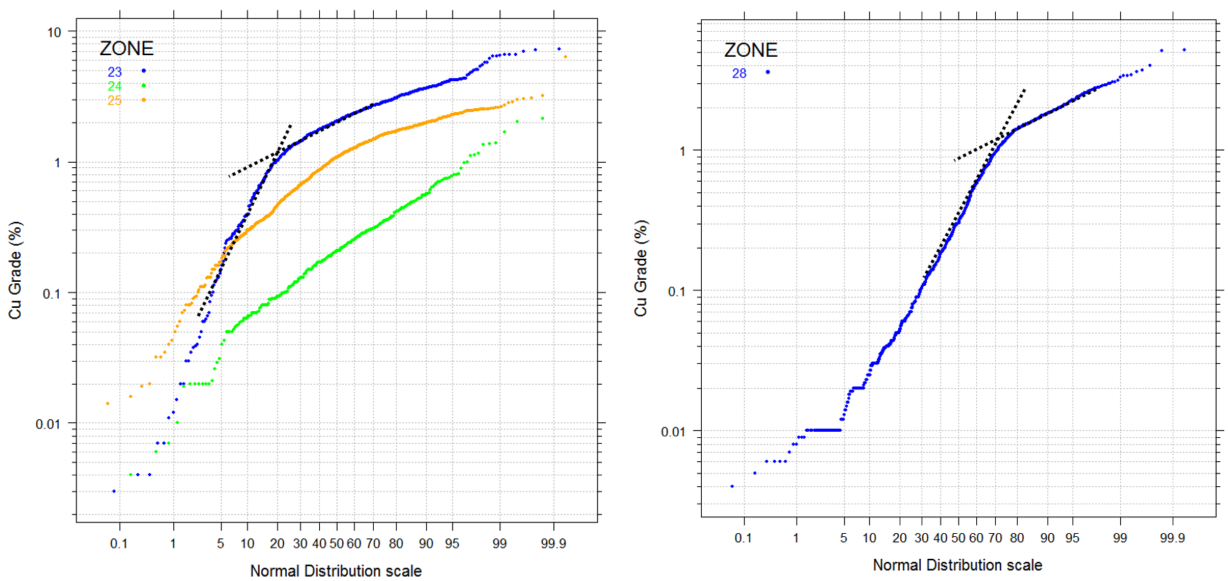
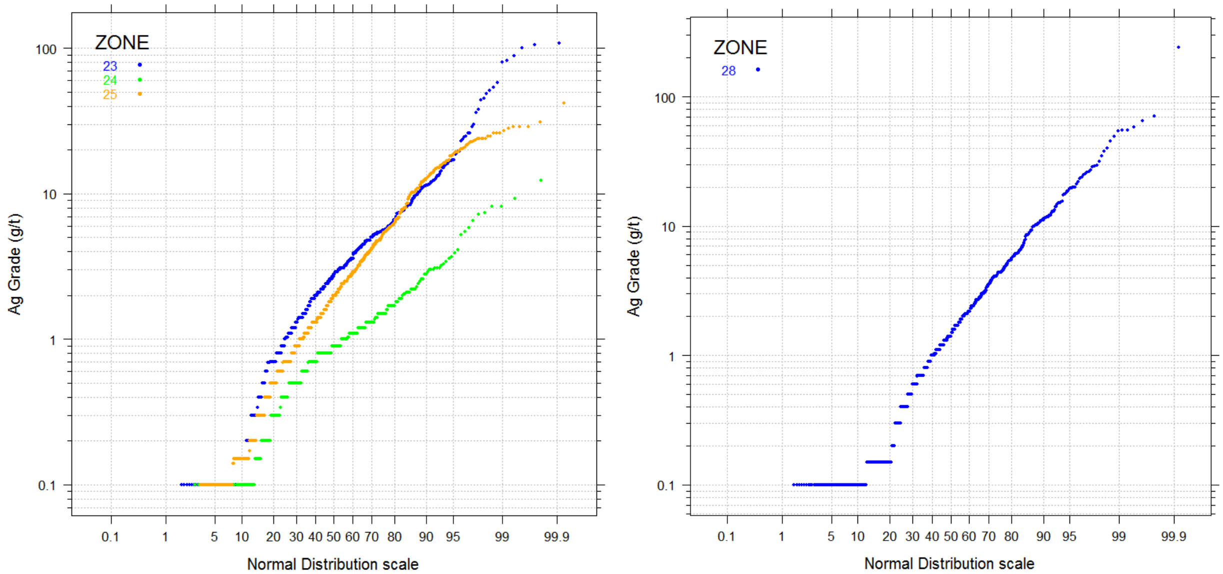


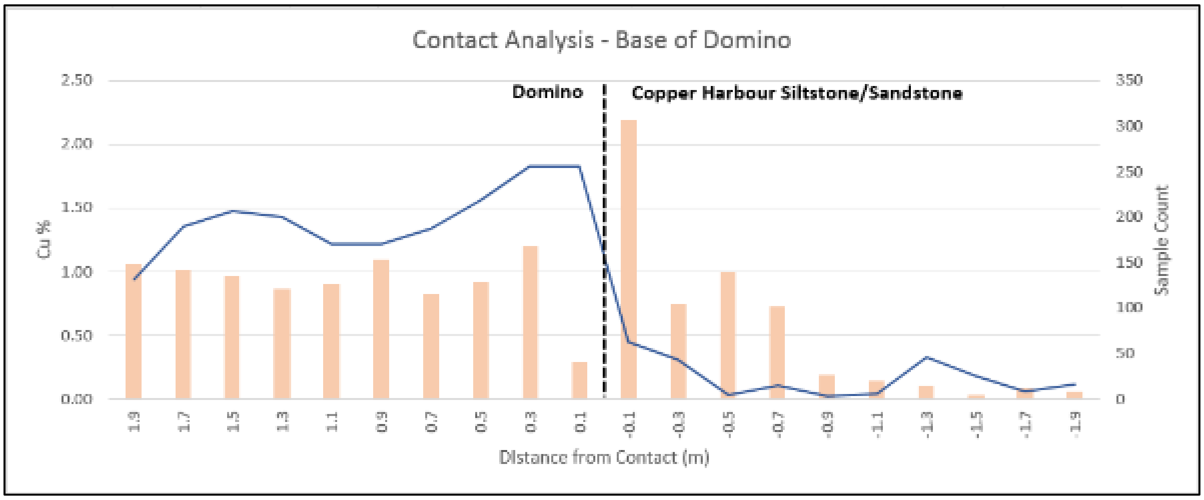
**Figure 14.7: Overlaid Cumulative Probability Graphs of Ag g/t Raw Assays for units of LCBS (left) and UCBS (right)**



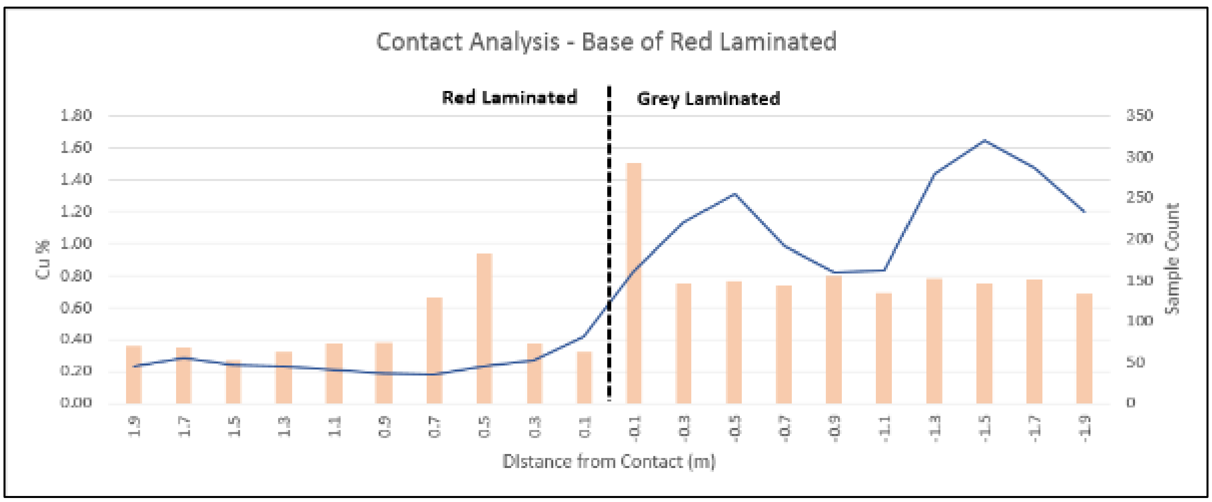

### 14.4.2 <u>Contact Analysis</u>

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 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.

### 14.4.3 Compositing

Drill holes intervals were flagged in 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 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 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).

### 14.4.4 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 14.6 present the statistics calculated from the copper and silver composites.

**Table 14.5: Copper Composites Statistics**

| 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: Silver Composites Statistics**

| 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. GMS 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 LCBS (left) and UCBS (right)**

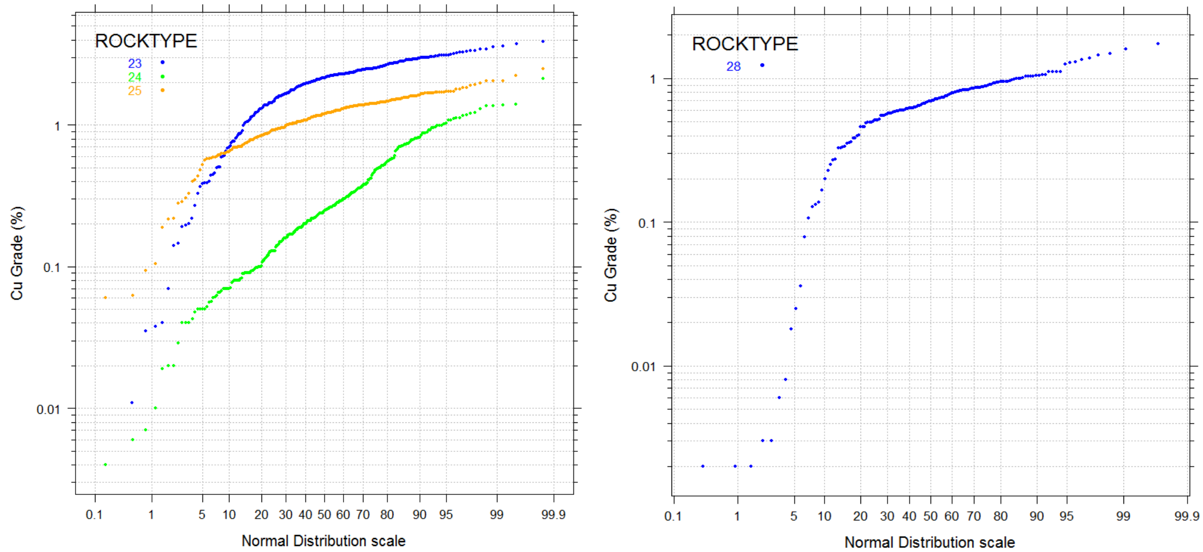
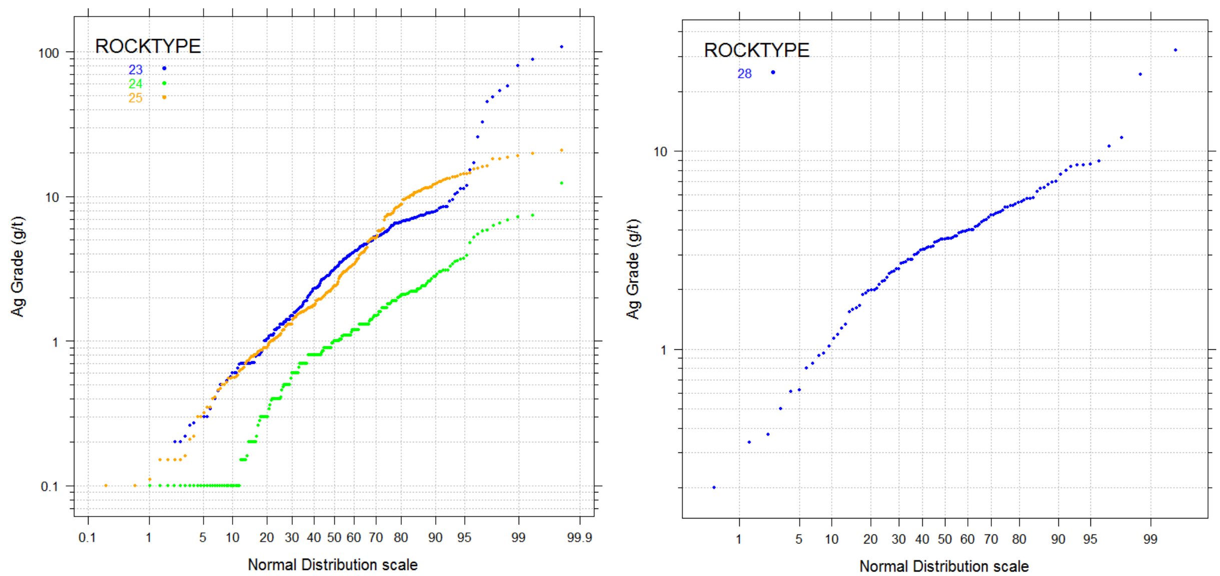



**Figure 14.10: Overlaid Cumulative Probability Graphs of Ag g/t Composites for Units of LCBS (left) and UCBS (right)**

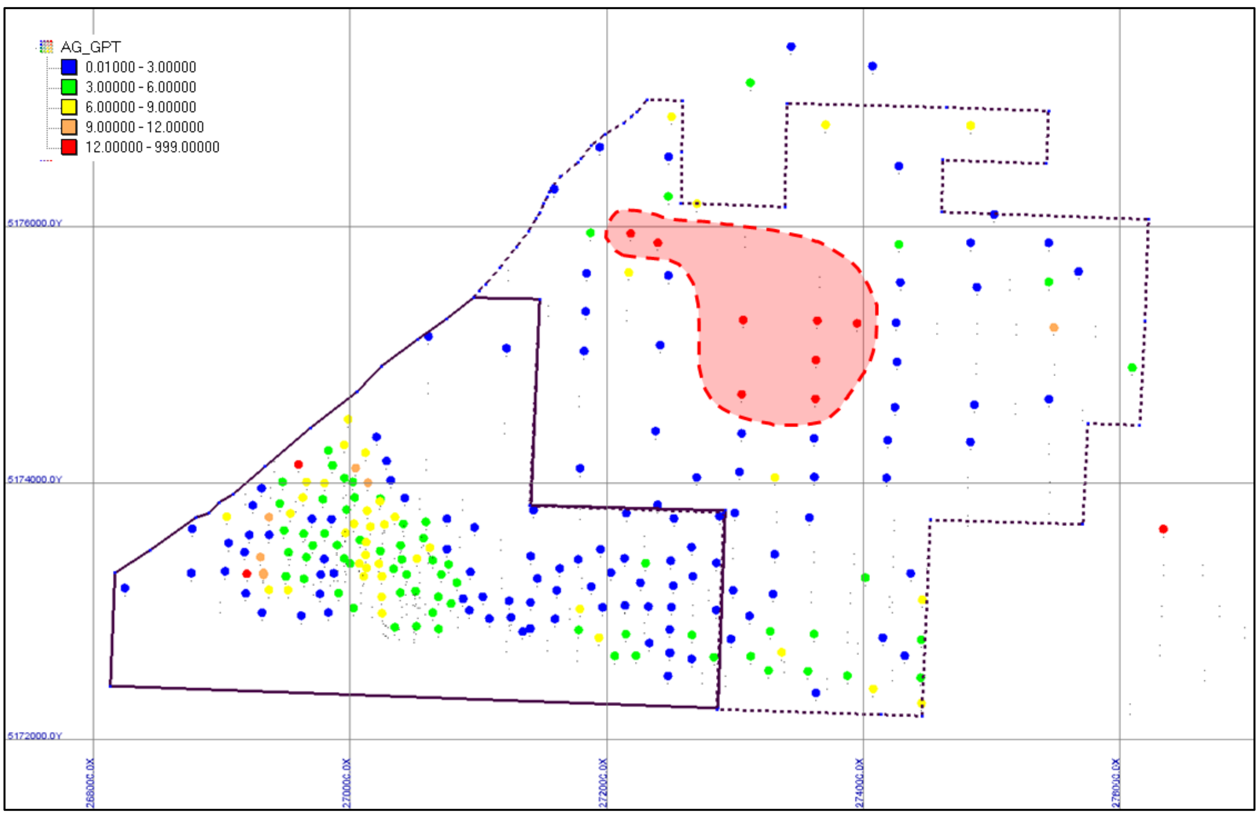


The silver outliers (> 10 g Ag/t) of the Domino unit were further examined to investigate their spatial distribution, and their potential impact on the estimation of the Copperwood deposit. Figure 14.11 shows that the outliers are spatially limited to a zone in the northern extents of the sparsely drilled 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, GMS 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 (2) 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 Coloured by Ag with Leasing Outlines.**




**Note the Sub-population in the Northern Area (within the sparse drilling).*

## 14.5 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 14.8 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/cm$^3$ for the LCBS. The range of the density data is minimal, where the minimum and maximum values were respectively 2.62 g/cm$^3$ and 2.79 g/cm$^3$. 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 Specific Gravity Measurements Presented by Years of Collection for 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 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 |

**Note: Determined from all density samples within the UCBS solid wireframe*

**Table 14.9: Specific Gravity Averages Used in Resource Estimation**

| Lithology | Specific Gravity (g/cm³) |
|---|---|
| Overburden | 2.20 |
| Domino | Q112.70 |
| Red Massive | 2.70 |
| Grey Laminated | 2.72 |
| UCBS | 2.69 |

## 14.6 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.

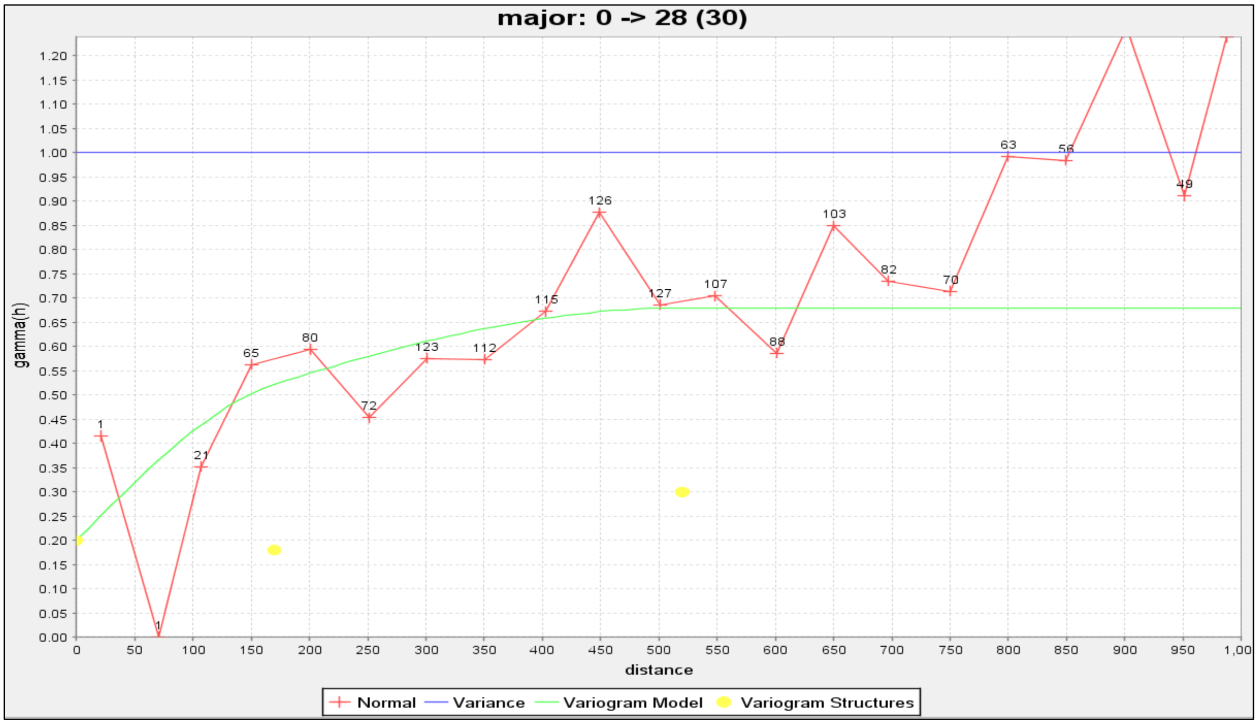
A series of variograms was generated from the composites of each unit every 5 degrees azimuth and 5 degrees dip increments. The spread angle was set to 30 degrees, with a bandwidth of 250 m. A lag distance of 50 m was applied. Only composites selected between 268,000 mE and 275,000 mE, and 5,172,000 mN and 5,174,500 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 Copper and Silver Composites of Zone**

| Element | Rock Codes | Nugget | Ranges of Influence (m) | | | | | | | | Rotation | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 1st Structure | | | | 2nd 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 Grey Laminated Subunit of LCBS**




## 14.7 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 counter clockwise 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/cm$^3$. Overburden blocks were assigned a density of 2.2 g/cm$^3$.

**Table 14.12: Rock Codes Used in 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 named 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 |

## 14.8 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 |

| Profile Name | Element Estimated | Pass | Sample | | | Ellipses Name | Semi-Variogram Name |
|---|---|---|---|---|---|---|---|
| | | | Min | Max | Max per Hole | | |
| 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 |
| 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 | - |

| Profile Name | Element Estimated | Pass | Sample | | | Ellipses Name | Semi-Variogram Name |
|---|---|---|---|---|---|---|---|
| | | | Min | Max | Max per Hole | | |
| 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 | | | |

## 14.9 Classification and Resource Reporting

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"). The "CIM Definition Standards on Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM council on May 10, 2014, provides standards for the classification of Mineral Resources and Mineral Reserve 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:

A "Measured Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resources. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

An "Indicated Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probably Mineral Reserve.

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

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration, Confidence in the estimate is insufficient to allow the meaningful application of technical economic parameter or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

More specifically, the classification of blocks is affected using the following criteria:

- Quality and reliability of drilling and sampling data.
- Distance between sample points (drilling density).
- Confidence in 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, GMS 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**

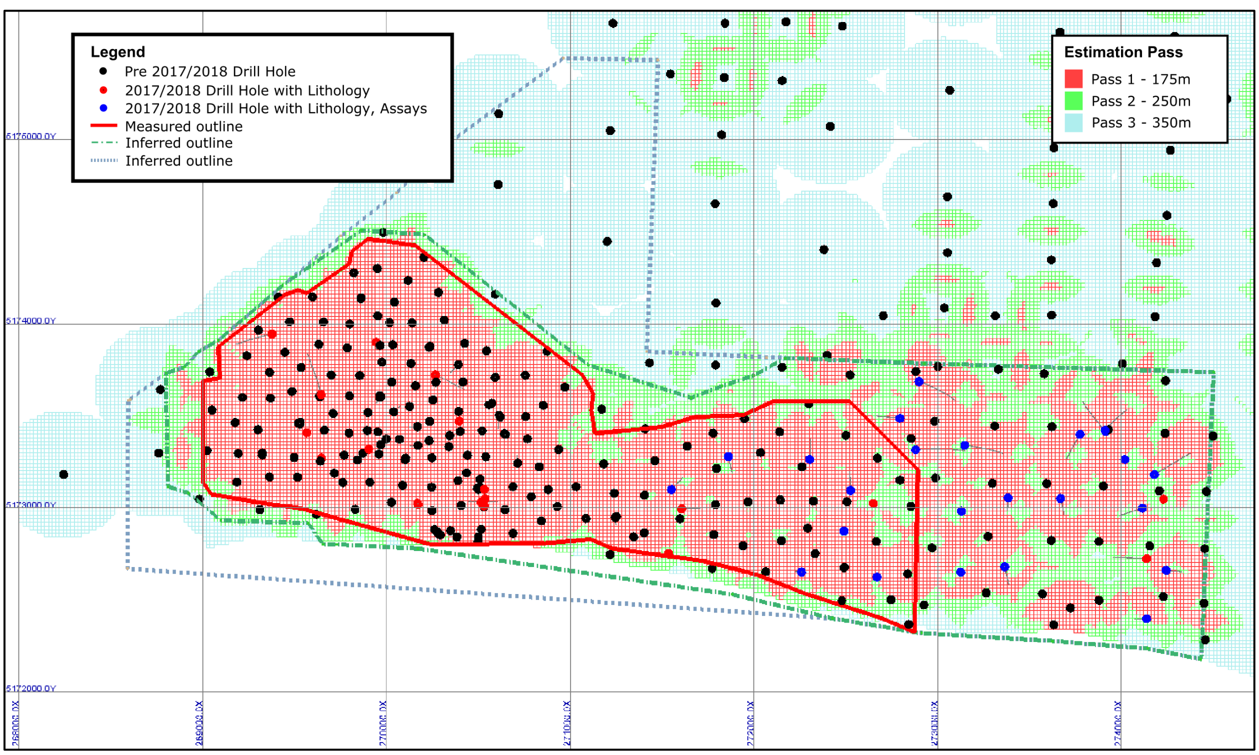


Figure 14.14 shows the previous resource categories applied by GMS for the April 2018 Mineral Resource estimate, compared to Figure 14.15 which shows the resource categories from the October 2017 Mineral Resource estimate. No change to the resource category was made in 2023. Measured Resources constitute 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: Current Resource Categories – Same as April 2018**

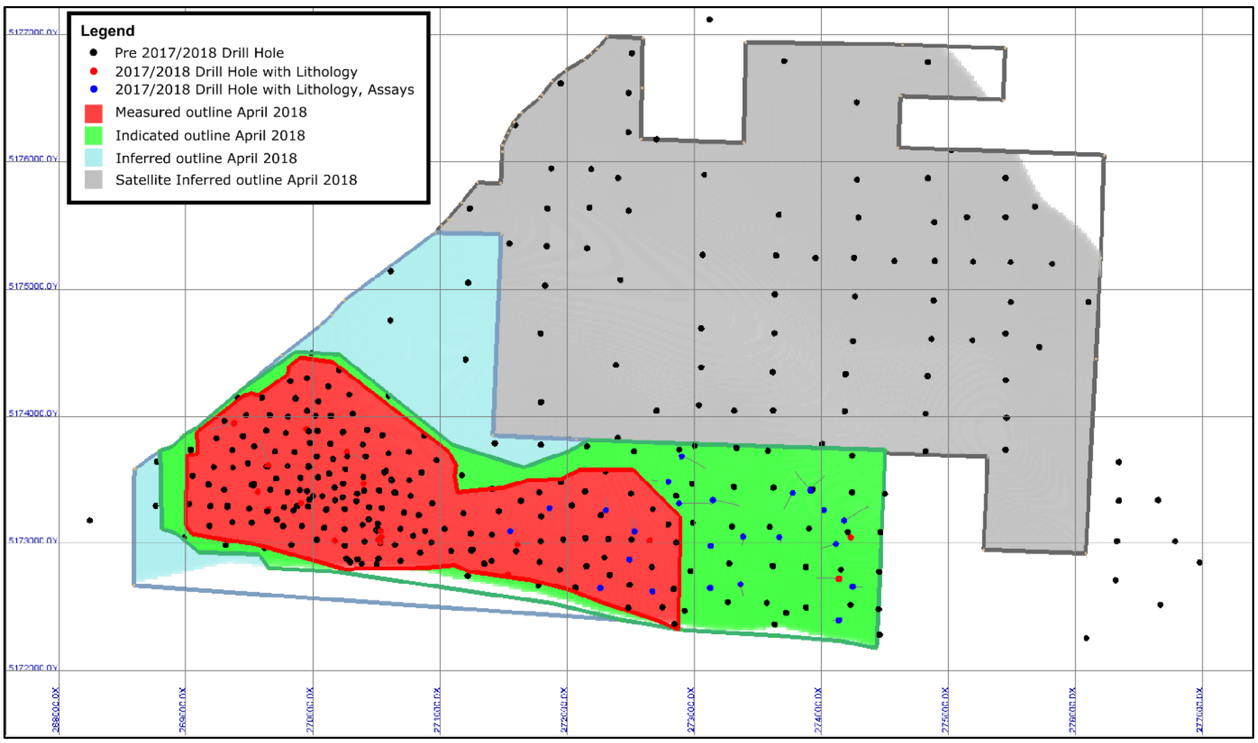
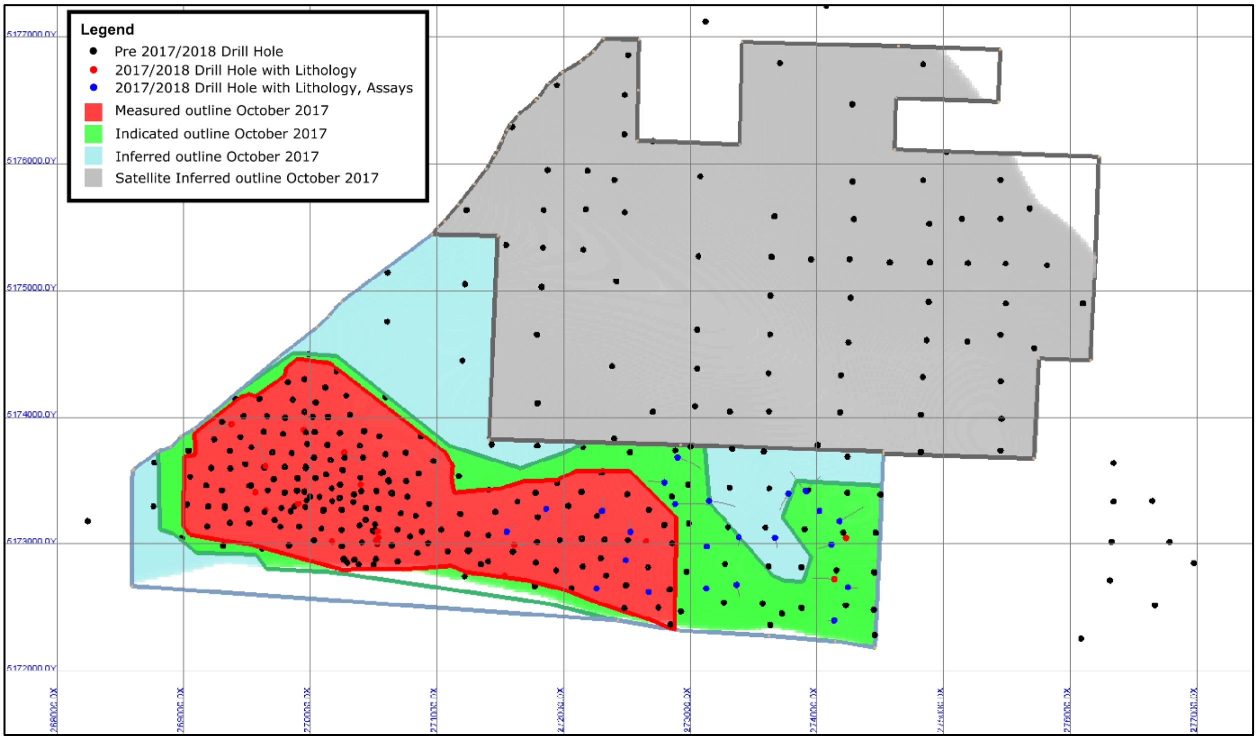


**Figure 14.15: Previous Resource Categories – October 2017**




## 14.10 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).

### 14.10.1 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.

### 14.10.2 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 268,000 mE – 275,000 mE, and 5,172,000 mN – 5,174,500 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 |

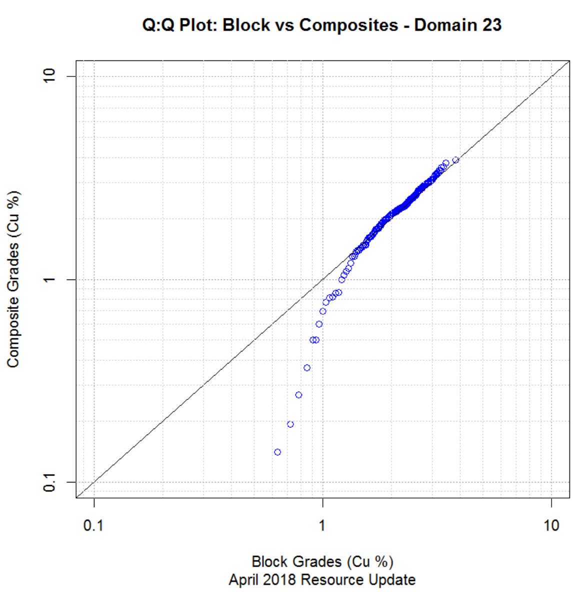
In general, the reconciliation of grade between the composites and blocks is good (less than 10% difference in mean grades). Silver grade reconciliation for rock code 25 (Grey Laminated) are adversely affected by a localised area of higher composite grades, hence the blocks appear underestimated in the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

### 14.10.3 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 Domino (23) and Grey Laminated (25) Subunits of the LCBS.**



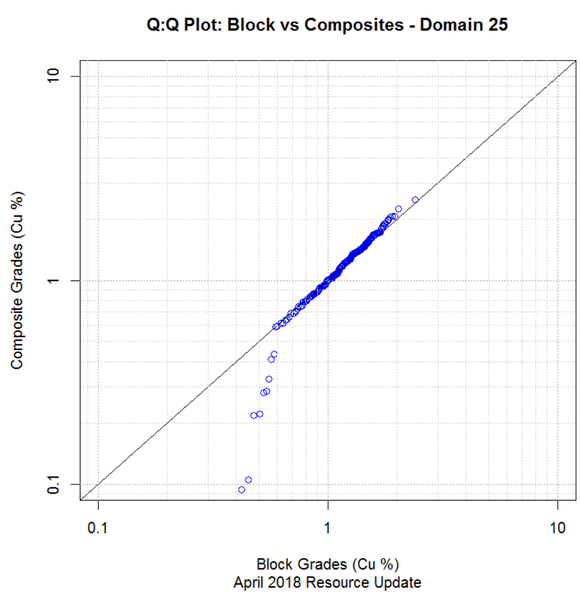
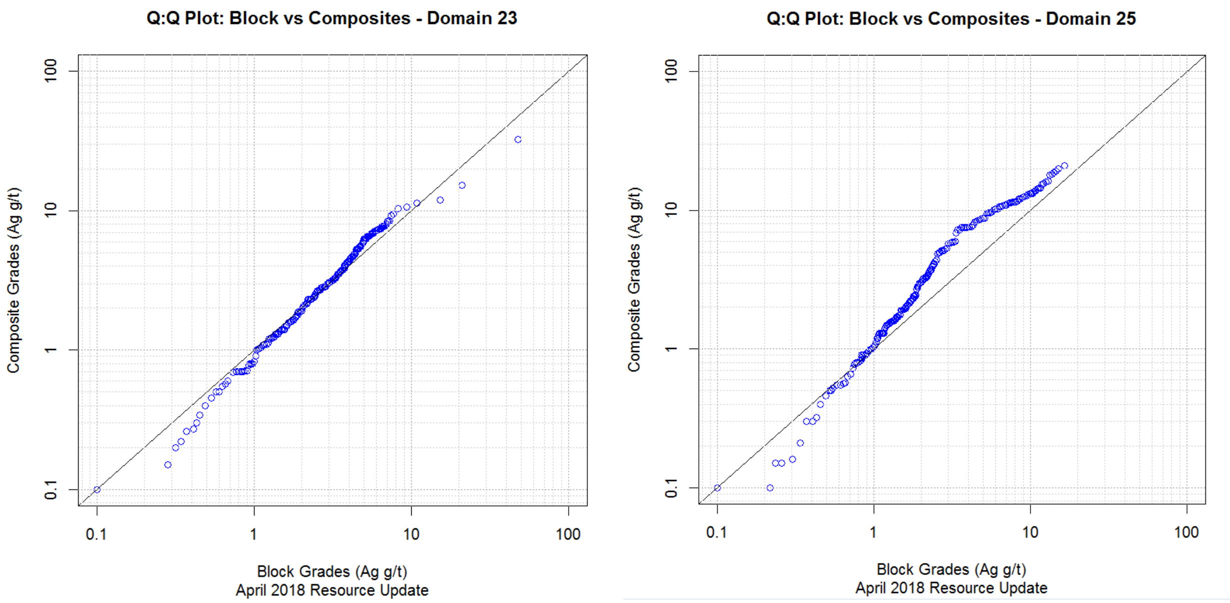



**Figure 14.17: Quantile:Quantile Plots of Ag g/t Distributions for Domino (23) and Grey Laminated (25) Subunits of the LCBS**



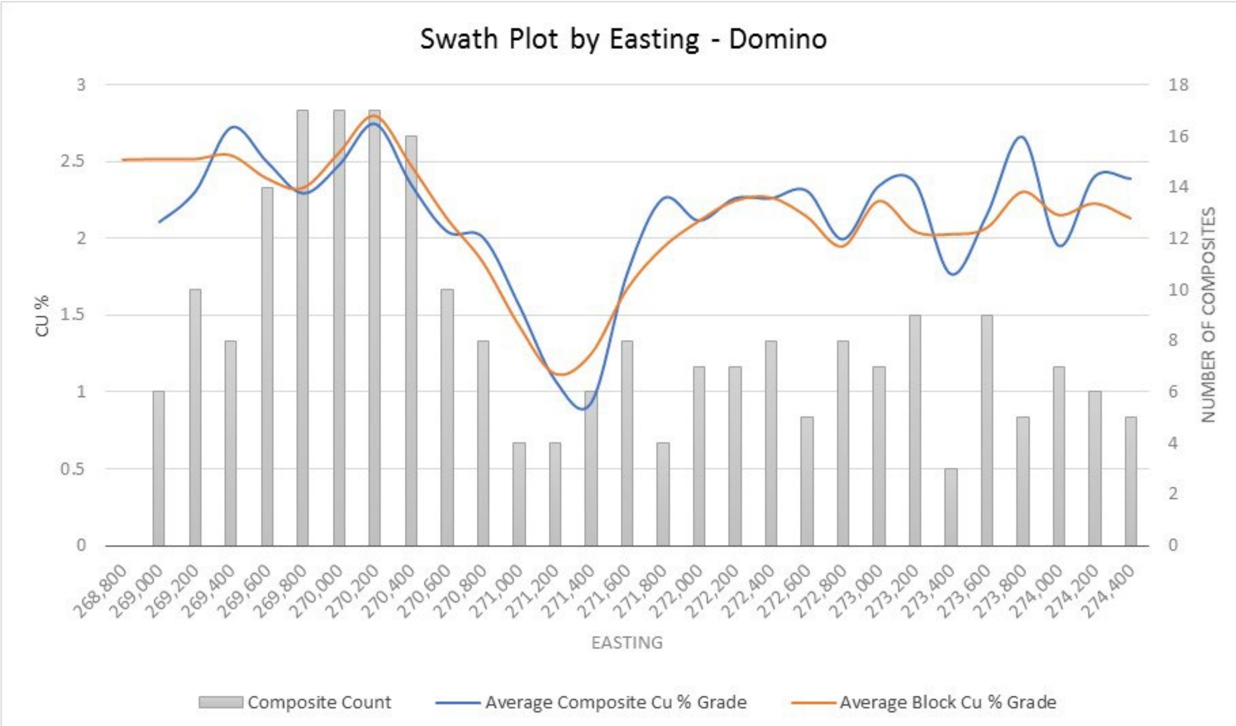

### 14.10.4 Local Statistical Validation - Swath Plots

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 Domino (23) by Easting**




### 14.10.5 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.

## 14.11 Minimum Thickness and RPEEE

To satisfy the requirements to demonstrate Reasonable Prospects for Eventual Economic Extraction ("RPEEE"), the minimum mining height for the purposes of reporting a Mineral Resource has been set at 2 m. GMS applied the following procedure to add grade dilution to areas of the LCBS which are less than 2 m.

The 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 (rock code 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**

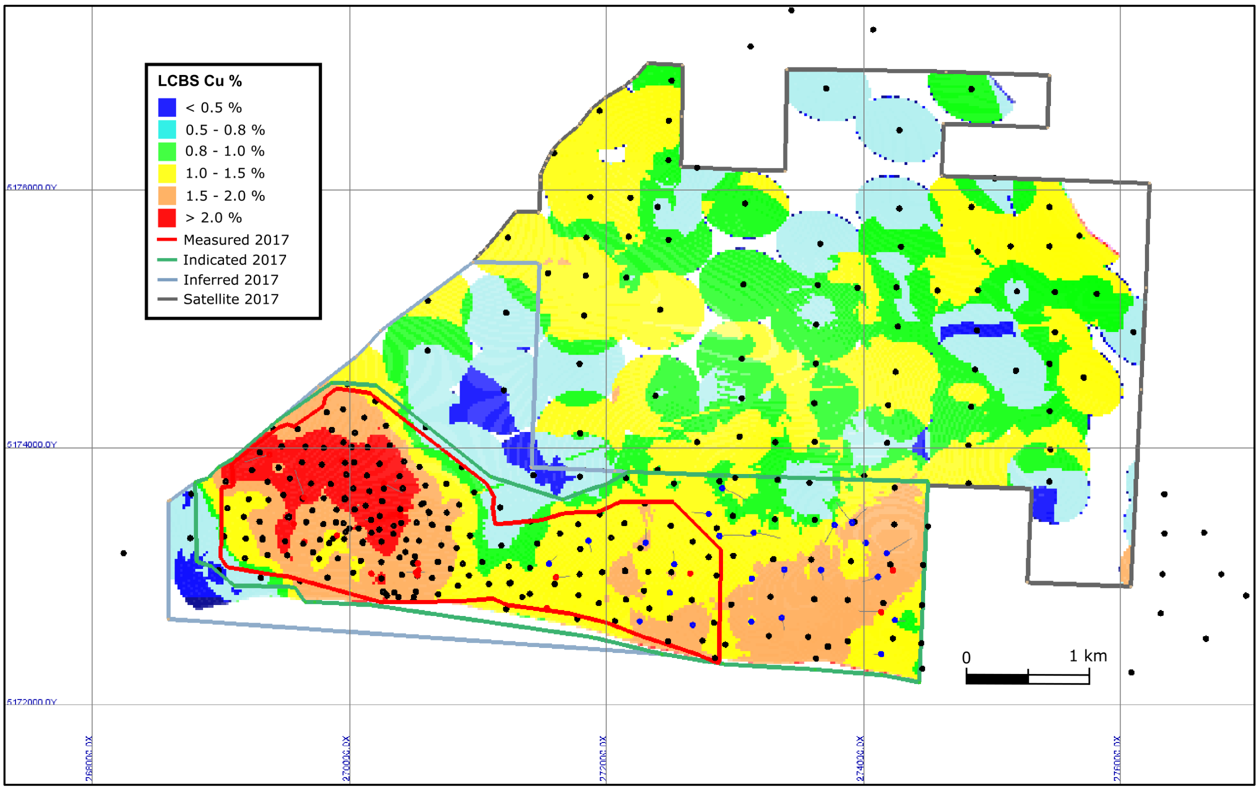
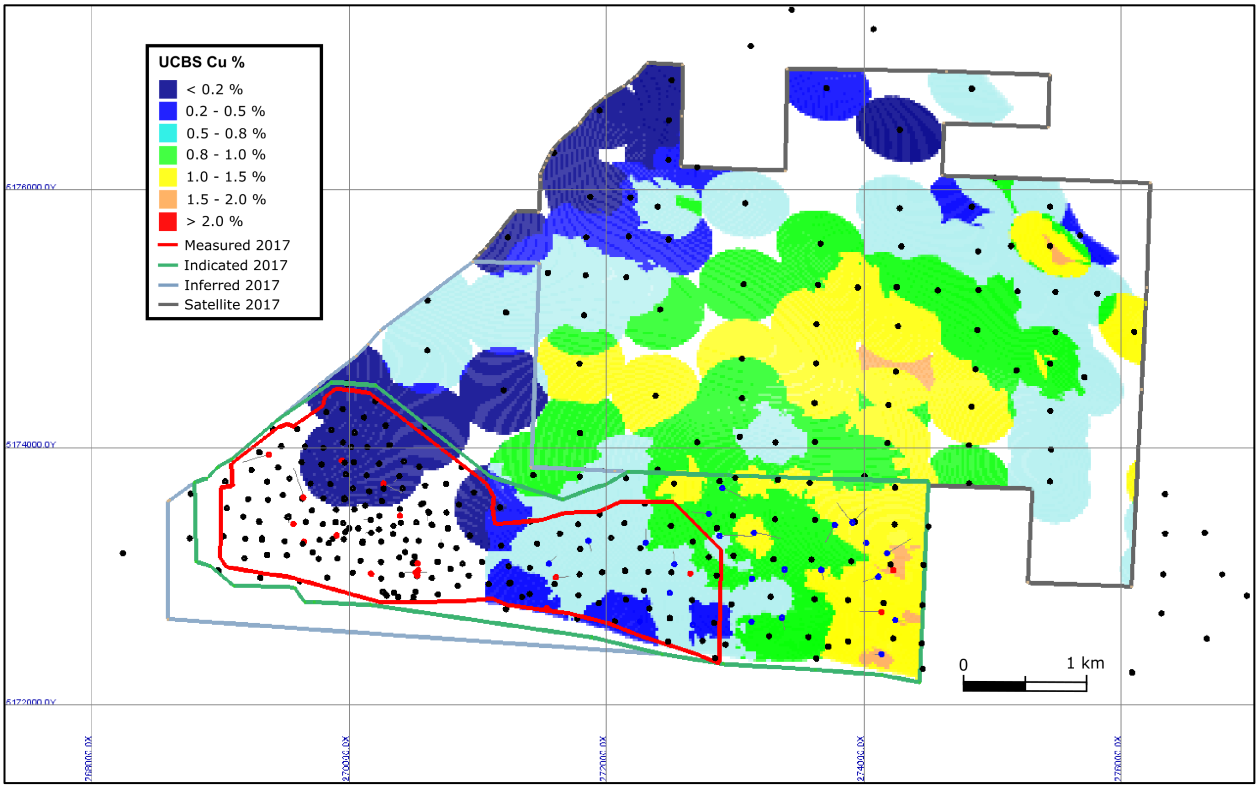


**Figure 14.20: Copper Grade Distribution in the UCBS (2 m minimum thickness) with Mineral Resource Classification**




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 0.9% Cu. The UCBS is not sampled or logged above the Main Zone of the Copperwood deposit.

## 14.12 Mineral Resource Statement

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

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, GMS 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.

### 14.12.1 Underground Optimization Parameters

The following conceptual mining parameters were considered:

- An NSR sliding scale royalty is applicable and equivalent to 5.5% at a USD 4.00/lb copper price.
- No mining loss and no additional mining dilution was considered at this stage for Mineral Resources.
- Mineral Resources are reported using a copper price of USD 4.00/lb and a silver price of USD 25/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 0.9% Cu was used to report the Mineral Resources.
- Operating costs are based on a processing plant located at the Copperwood site.

### 14.12.2 Mineral Resource Tabulation

The Copperwood Deposit Total Measured and Indicated Mineral Resources are reported at 54.2 Mt grading an average 1.49% Cu and 3.6 g/t Ag containing 1.78 B lbs Cu and 6.3 Moz Ag using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined. Inferred Mineral Resources for the Copperwood Deposit are reported at 2.3 Mt grading an average 1.12% Cu and 1.2 g Ag/t containing 56 Mlbs Cu and 0.1 Moz Ag using a cut-off grade of 0.9% Cu.

The Satellite Deposits Inferred Mineral Resources are reported at 76.8 Mt grading 1.09% Cu and 3.6 g Ag/t containing 1.84 billion pounds of copper and 8.9 million ounces of silver using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined.

Table 14.18 reports the Mineral Resources for the Copperwood and Satellite Deposits by resource categories. All parameters used in the calculations are presented in the table’s notes.

The responsible Qualified Person is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Copperwood Mineral Resource Estimate.

**Table 14.18: Mineral Resource Estimate - Copperwood Project**
**0.9% Cu cut-off Grade – February 28th, 2022**

| Deposits | Resource Category | Tonnage (Mt) | Copper Grade (%) | Silver Grade (g/t) | Copper Contained (M lbs) | Silver Contained (M oz) |
|---|---|---|---|---|---|---|
| LCBS | Measured | 27.9 | 1.66 | 4.5 | 1,023 | 4.1 |
| | Indicated | 16.1 | 1.42 | 2.4 | 504 | 1.2 |
| | M + I | 44.0 | 1.57 | 3.7 | 1,527 | 5.3 |
| | Inferred | 2.3 | 1.12 | 1.2 | 56 | 0.1 |
| UCBS | Measured | 0.1 | 0.95 | 4.6 | 2 | 0.0 |
| | Indicated | 10.1 | 1.13 | 3.1 | 253 | 1.0 |
| | M + I | 10.2 | 1.13 | 3.1 | 255 | 1.0 |
| | Inferred | - | - | - | - | - |
| Satellite LCBS | Inferred | 49.7 | 1.11 | 2.5 | 1210 | 3.9 |
| Satellite UCBS | Inferred | 27.1 | 1.05 | 5.7 | 630 | 5.0 |

**Notes on Mineral Resources:*

1. *Mineral Resources are reported using a copper price of US$4.00/lb and a silver price of US$25/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 0.9% 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$4.00/lb Cu price, a sliding scale 5.5% NSR royalty on the Copperwood Project is payable to leaseholders.*
7. *Measured, Indicated and Inferred Mineral Resources have a drill hole spacing of 175 m, 250 m and 350 m, respectively.*
8. *A minimum mining thickness of 2 m was applied. No additional 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 (2014).*
11. *The qualified person for the estimate is Mr. James Purchase, P.Geo., Consulting Geologist for GMS. The estimate has an effective date of 28th February 2022.*
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.*

## 14.13 Underground Mineral Resources Sensitivity

### 14.13.1 LCBS

Table 14.19, Table 14.20 and Table 14.21 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 (1.5 Mt) is gained when using a cut-off grade of 0.8% instead of 0.9% Cu for Measured and Indicated Resources. On the contrary, in the satellite deposits, a significant proportion (13.0 Mt) for the LCBS grades between 0.8% and 0.9% 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 14.19: 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 | 7.2 | 1.6 |
| 1.5% | 25.1 | 1.83 | 1,008 | 5.0 | 4.0 |
| 1.0% | 42.2 | 1.60 | 1,488 | 3.8 | 5.2 |
| 0.9% | 44.0 | 1.57 | 1,527 | 3.7 | 5.3 |
| 0.8% | 45.5 | 1.55 | 1,554 | 3.7 | 5.4 |

**Table 14.20: 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 | 7.7 | 0.0 |
| 1.0% | 1.6 | 1.18 | 43 | 1.6 | 0.1 |
| 0.9% | 2.3 | 1.12 | 56 | 1.2 | 0.1 |
| 0.8% | 3.2 | 1.04 | 74 | 0.9 | 0.1 |

**Table 14.21: 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 | 0.4 | - |
| 1.0% | 34.4 | 1.17 | 888 | 2.3 | 2.5 |
| 0.9% | 49.7 | 1.11 | 1,210 | 2.5 | 3.9 |
| 0.8% | 62.7 | 1.05 | 1,456 | 2.8 | 5.6 |

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

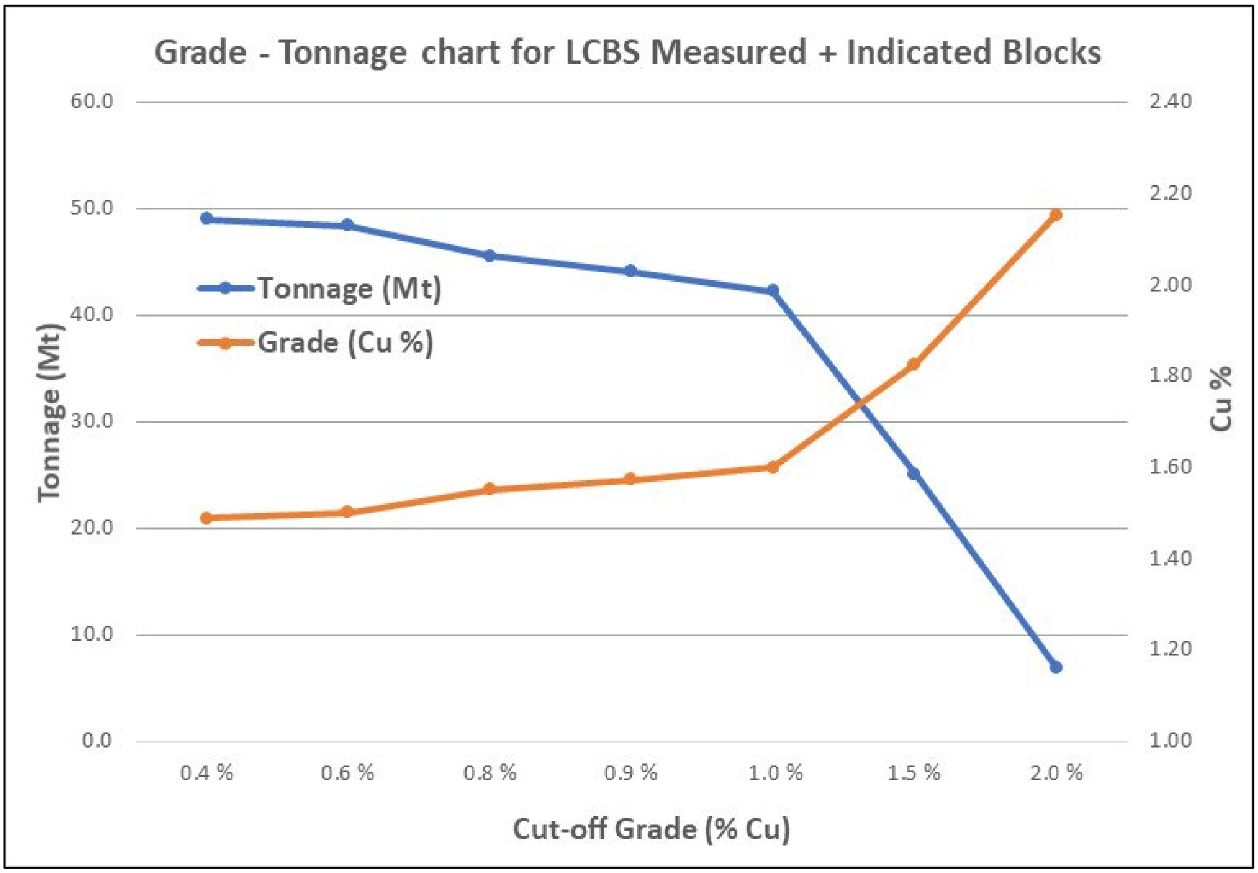


**Figure 14.22: Grade-Tonnage Curve of Inferred Resources for LCBS at the Copperwood Deposit**

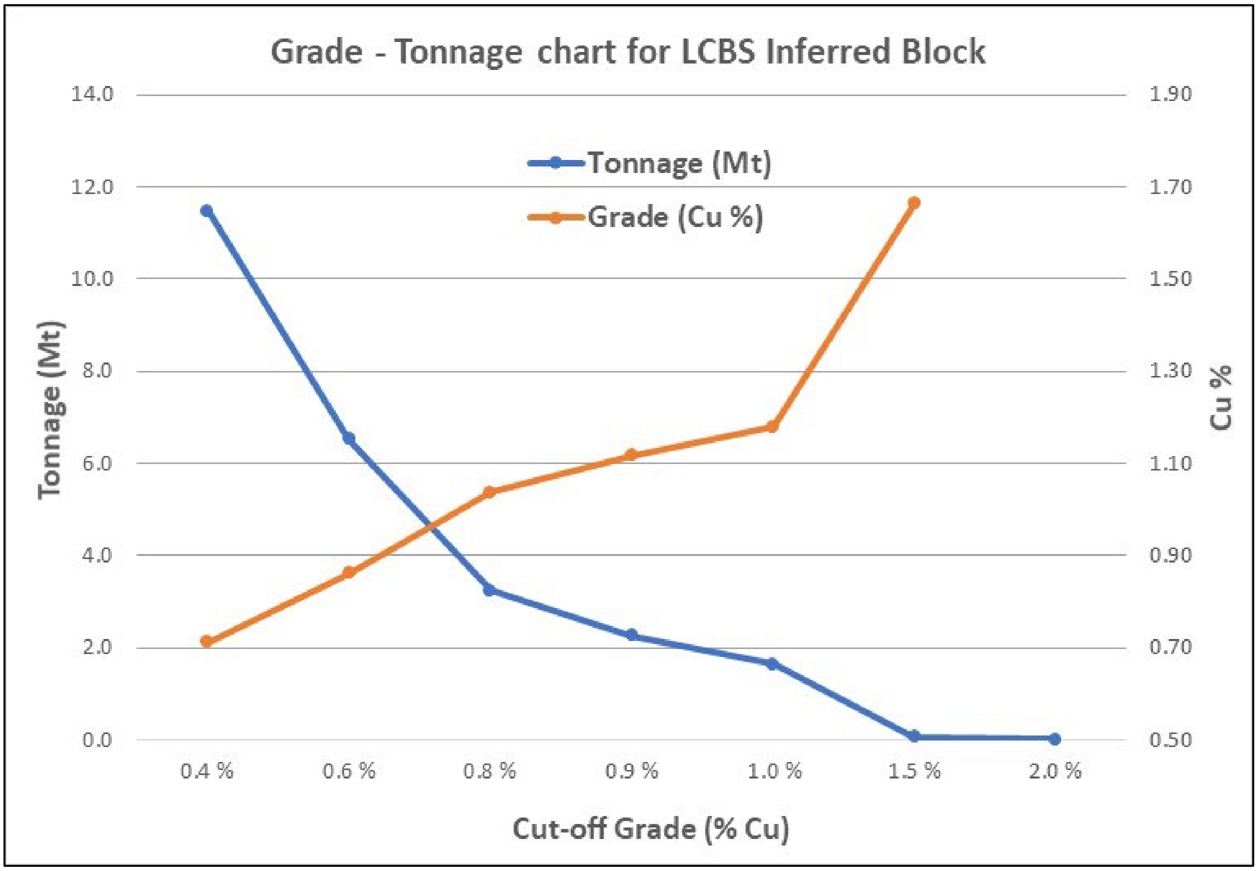
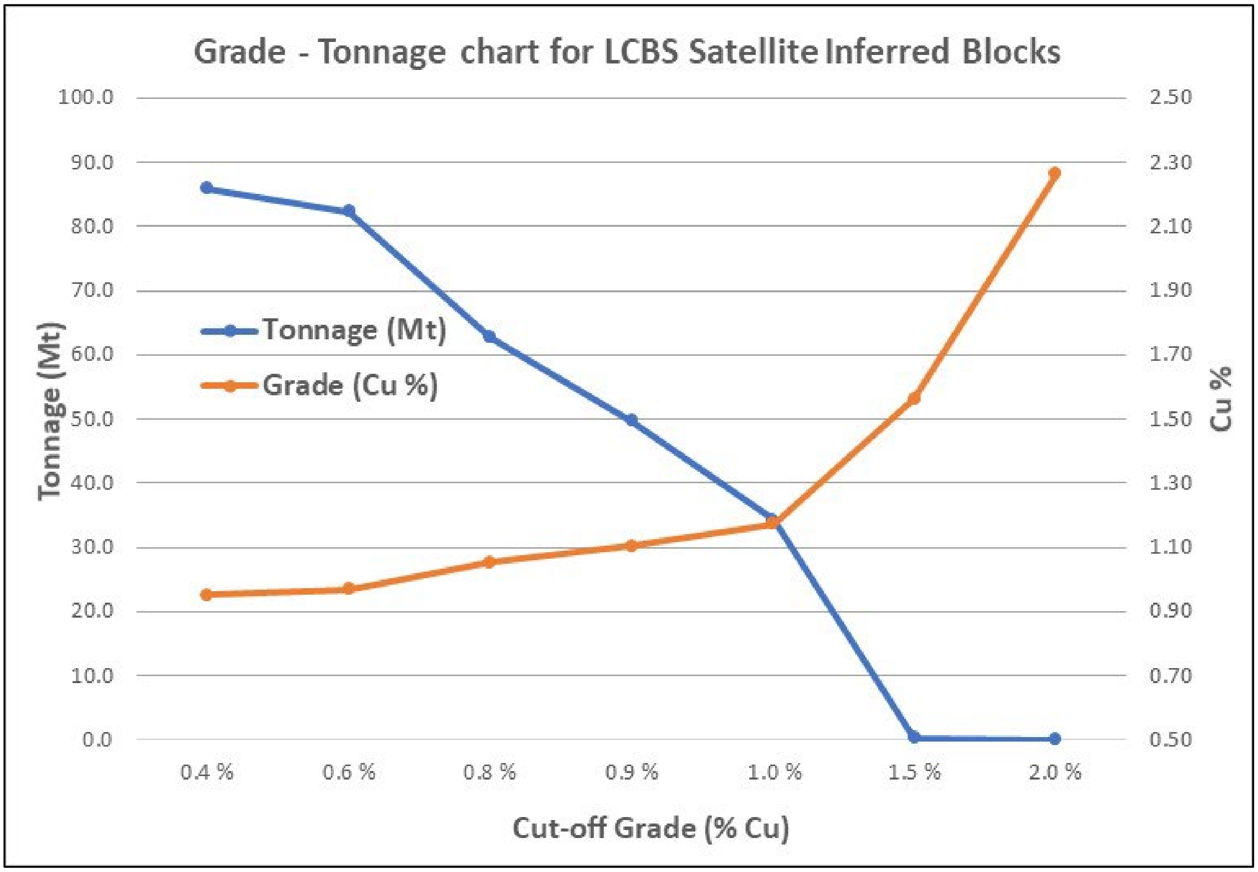


**Figure 14.23: Grade-Tonnage Curve of Inferred Resources for the LCBS at the Satellite Deposits**




### 14.13.2 UCBS

Table 14.22, Table 14.23 and Table 14.24 summarize the sensitivity of the constrained underground Mineral Resources of the UCBS 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 Inferred satellite deposits, a significant proportion (6.4 Mt) for the UCBS grades between 0.8% and 0.9% Cu.

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

**Table 14.22: 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 | 4.0 | 0.1 |
| 1.0% | 7.1 | 1.21 | 189 | 3.3 | 0.7 |
| 0.9% | 10.2 | 1.13 | 255 | 3.1 | 1.0 |
| 0.8% | 13.7 | 1.06 | 319 | 3.0 | 1.3 |

**Table 14.23: 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% | - | - | - | - | - |
| 0.9% | - | - | - | - | - |
| 0.8% | 0.5 | 0.87 | 10 | 2.6 | 0.0 |

**Table 14.24: 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 | 3.7 | 0.1 |
| 1.0% | 15.5 | 1.12 | 384 | 5.9 | 3.0 |
| 0.9% | 27.1 | 1.05 | 630 | 5.7 | 5.0 |
| 0.8% | 33.5 | 1.01 | 749 | 5.2 | 5.6 |

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

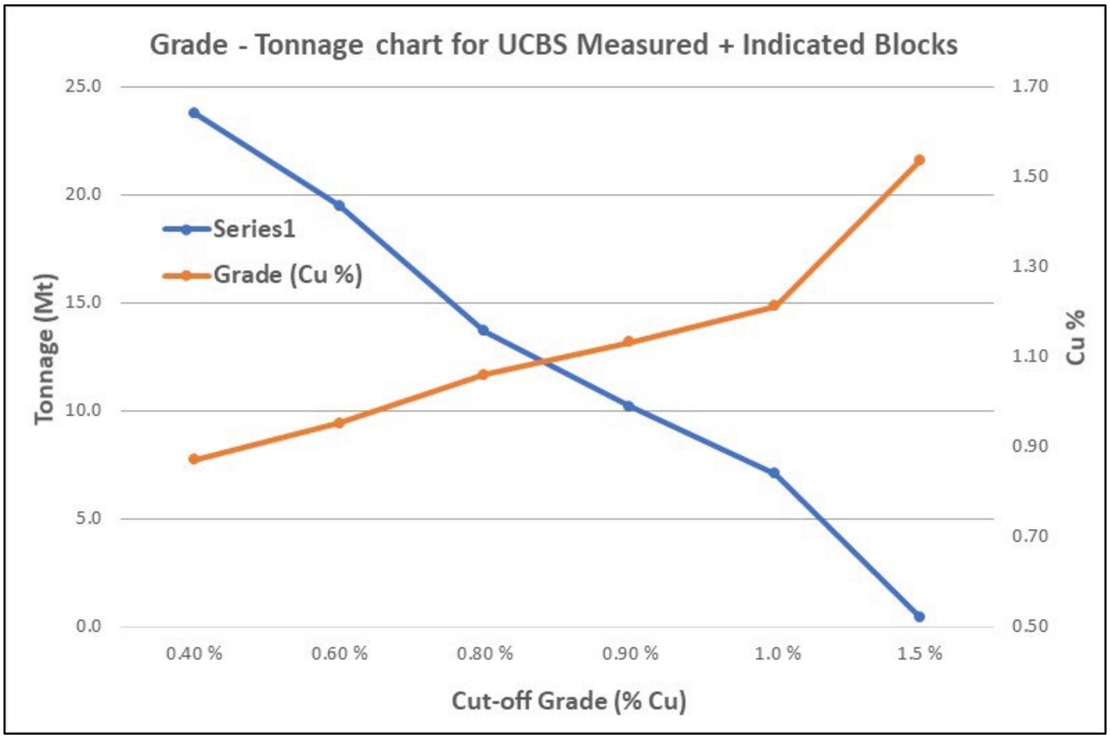



**Figure 14.25: Grade-Tonnage Curve of Inferred Resources for UCBS at the Copperwood Deposit**

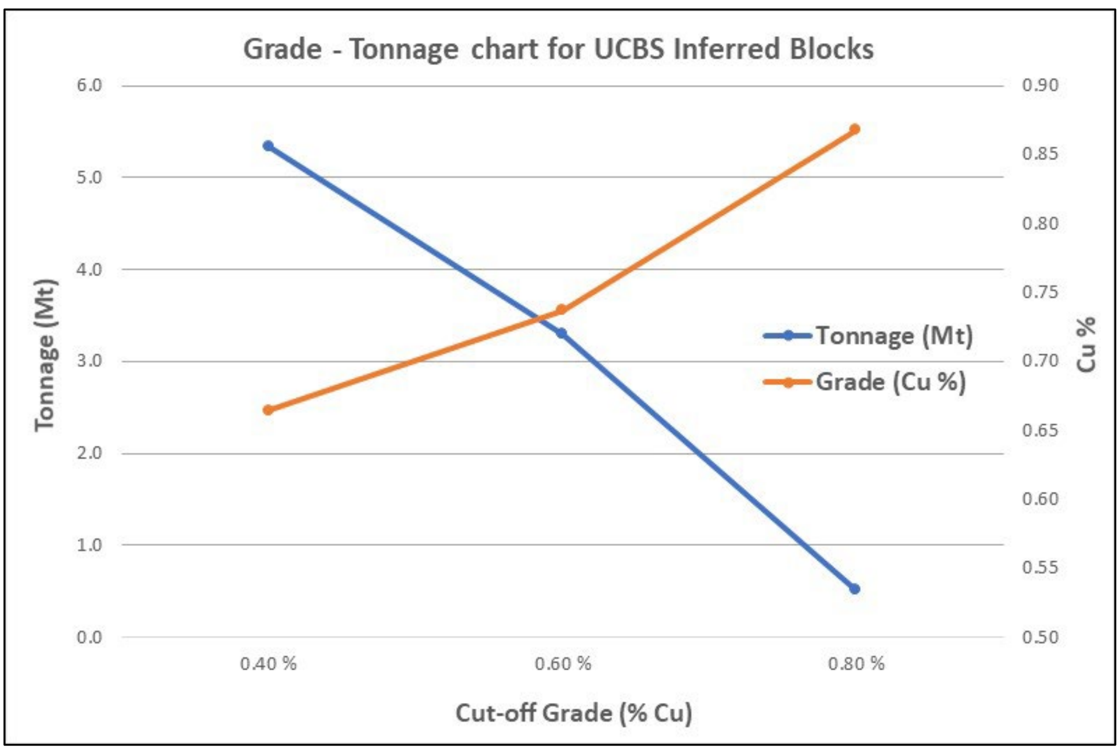
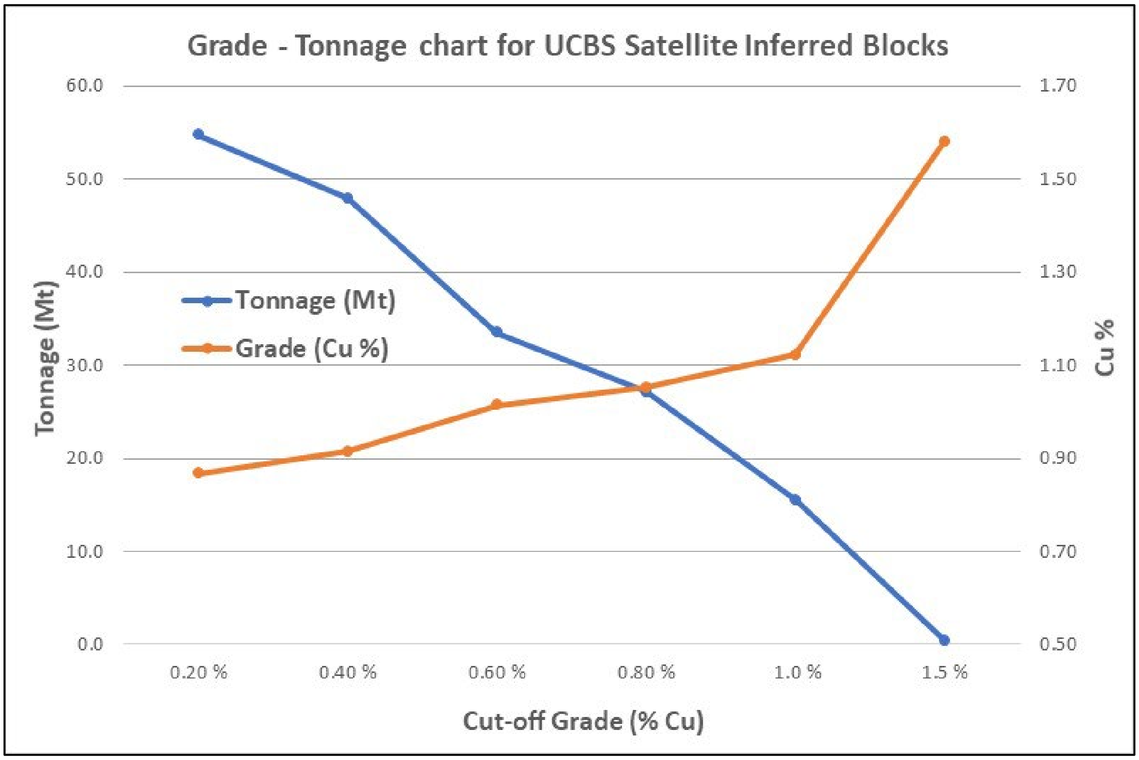


**Figure 14.26: Grade-Tonnage Curve of Inferred Resources for the UCBS at the Satellite Deposits**



# 15. MINERAL RESERVE ESTIMATES

The Mineral Reserves for the Copperwood Project are estimated at 25.7 Mt, at an average grade of 1.45% Cu and 3.91 g/t Ag, as summarized in Table 15.1. The Mineral Reserve estimate was prepared by GMS. The resource block model was also generated by GMS.

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**

| Reserve by Category | Tonnes (Mt) | Cu Grade (%) | Ag Grade (g/t) | Cu Contained (M lbs) | Ag Contained (M oz) |
|---|---|---|---|---|---|
| Proven | 18.2 | 1.49 | 4.47 | 597 | 2.61 |
| Probable | 7.5 | 1.34 | 2.56 | 222 | 0.62 |
| **Proven & Probable** | **25.7** | **1.45** | **3.91** | **820** | **3.23** |

**Notes:*

1) *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (Nov 29th,2019) and CIM Definition Standards for Mineral Resources and Reserves, (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 USD 4.0/lb and a silver price of USD 25.00/oz.*
4) *Assuming a long-term copper price $4.00/lb, a sliding scale 4.0% NSR royalty on the Copperwood Project is payable to leaseholders. A 1.5% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd. This also includes an additional 11.5% silver mineral royalty payable to Osisko Stream Royalties.*
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 Carl Michaud, P.Eng., VP, Mining Engineering for GMS. The estimate has an effective date of May 25, 2022*
10) *The numbers may not sum due to rounding; rounding followed the recommendations in NI 43-101.*
11) *The geotechnical parameters of the previous technical report from June 2018 were used in this feasibility update.*

## 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 2020.3) 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 (2) 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) has been established, taking into account the cut-off grade in Section 15.2, the minimum mine height and the mine dilution.

All the tonnage outside of this zone is removed because it does not meet the economic criteria for room and pillar stoping. An exception is made for the development tonnage required between two (2) mining areas; this tonnage is then included in the reserves. The Mineral Reserve is net of all the pillars, including those in the mine production panels, the Lake Superior 30 m offset, a crown pillar providing a 25 m vertical of rock above openings, and a 15 m barrier pillar around the historical test mine openings.

An ore thickness 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 at ground level to ensure a minimum height of 2.1 m in jumbo stopes or 3 m in continuous miner stopes. 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 have been carried out, the drift and stope design are complete, taking into account the efficiency, equipment limit 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+I to Mining Reserves Proven and Probable**

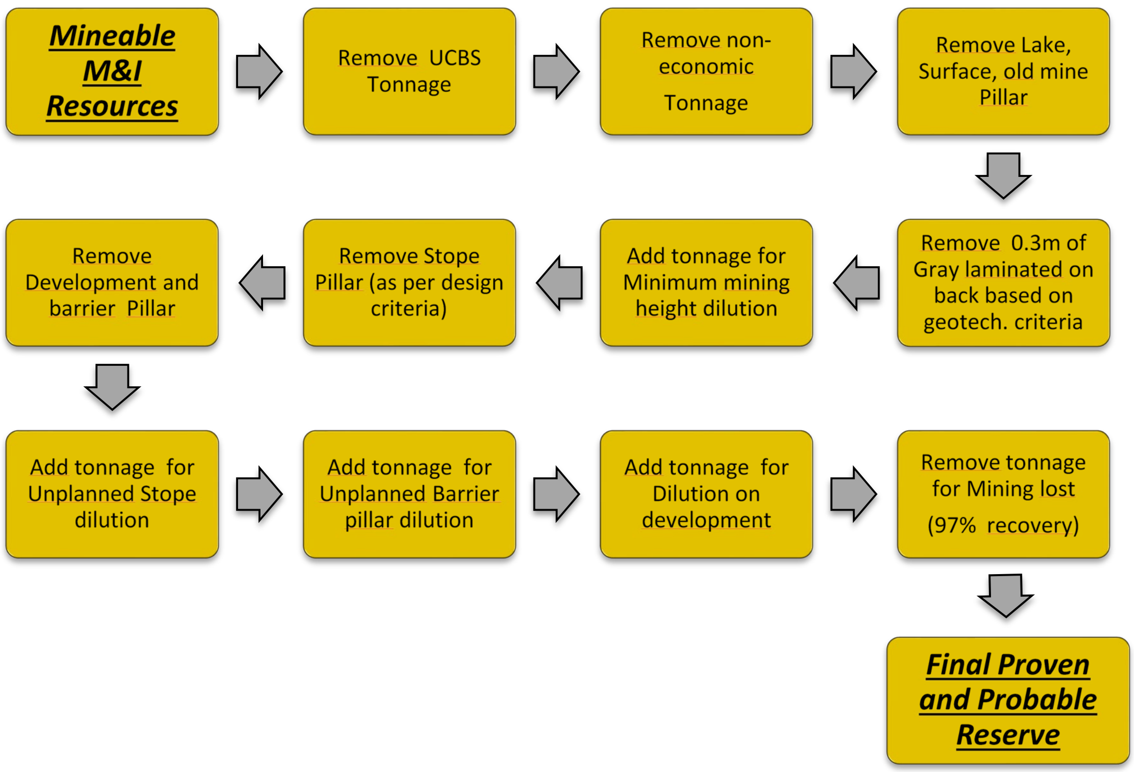
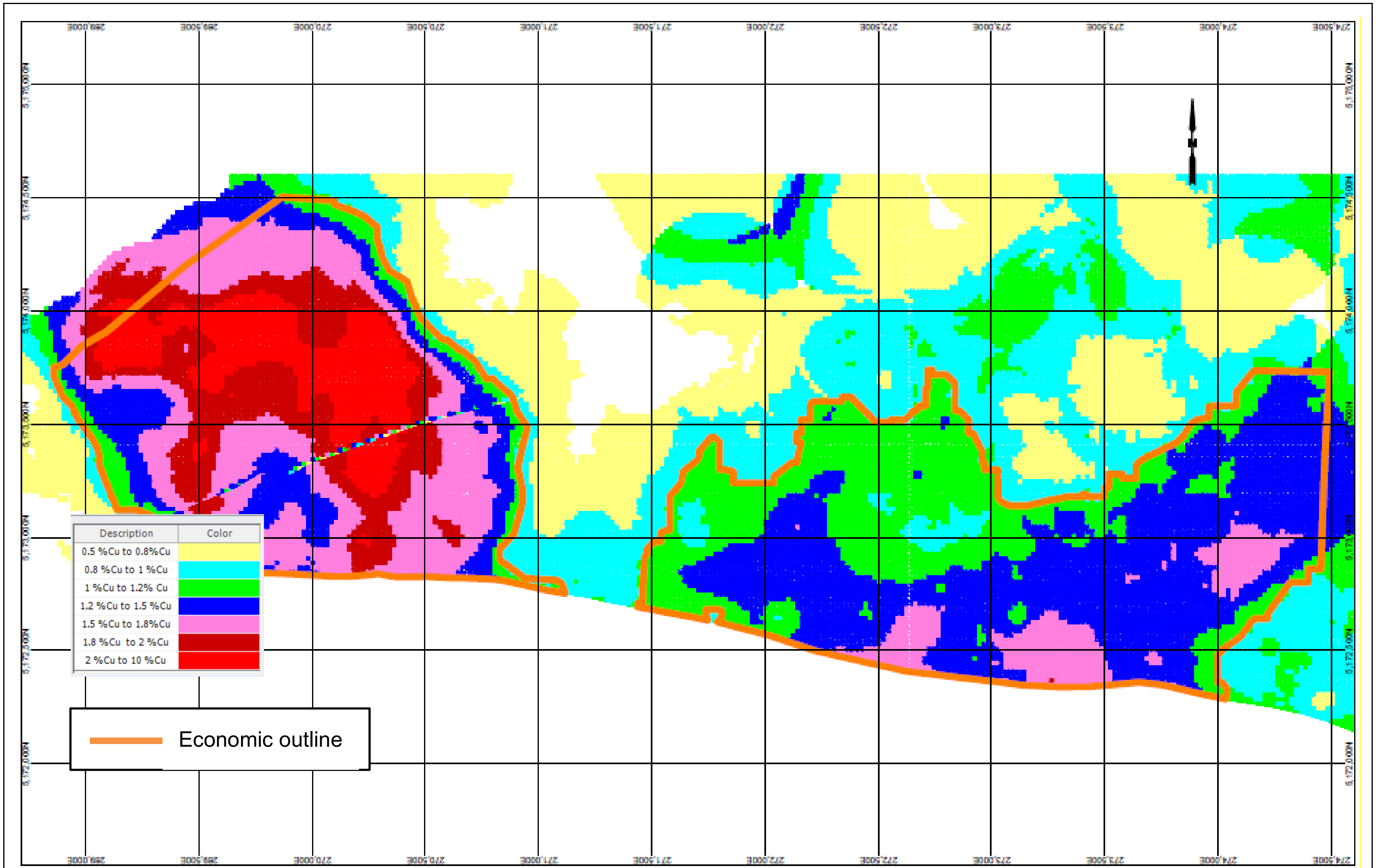


**Figure 15.2: Project Economic Outline**




| | | | |
|---|---|---|---|
| Mining Services | 7900 W. Taschereau Blvd. Suite D-200<br>Brossard, Qc J4X 1C2 | **Title: Project Economic Outline** | **Author: Carl Michaud, ing**<br>**Date: 25/02/2023** |

### 15.2 Cut-Off Grade

The cut-off grade is the mineral grade that must be extracted from a mine in order for the operating costs of the mine to be covered by the revenue generated from the sale of the mineral. In other words, the cut-off grade is the concentration level of mineral below which it would not be profitable to extract the mineral from the mine. If the mineral grade is below the cut-off grade, the extraction of the mineral would not generate sufficient revenue to cover the production costs.

Factors that influence the cut-off grade include operating costs, mineral prices, available quantities of mineral, and available technologies for mineral extraction. It should be noted that the cut-off grade can vary over time depending on changes in production costs and mineral prices. Additionally, the cut-off grade can vary depending on the type of mineral and extraction methods used.

To calculate the portion of measured and indicated resources that can be converted into reserves, the economic part of the resource needs to be identified. To achieve this, a cut-off grade including 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.

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 cut-off grade, the dilution is also 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 6,800 tpd 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 (USD) |
|---|---|---|
| Metal Prices | | |
| Copper | $/lb | 4 |
| Silver | $/oz | 25 |
| Process Recovery | | |
| Copper | % | 86 |
| Silver | % | 73.4 |
| Effective Payable Rate | | |
| Copper | % | 96 |
| Silver | % | 54 |
| Silver Credit Net of Refining | $/t ore | 0.7 |
| Operating Costs First Estimation | | |
| Processing | $/t ore | 15.00 |
| G&A | $/t ore | 5.00 |
| Sustaining CAPEX | $/t ore | 5.5 |
| UG Mining Costs | $/t ore | 27.50 |
| Royalties | $/t ore | 5.5 |
| TC/RC | $/t ore | 5 |
| Transportation | $/t ore | 6 |
| Total | $/t ore | 69.50 |
| Cut-Off Grade | %Cu | 1 |

**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 (2) types of dilutions anticipated in the Copperwood Project:

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

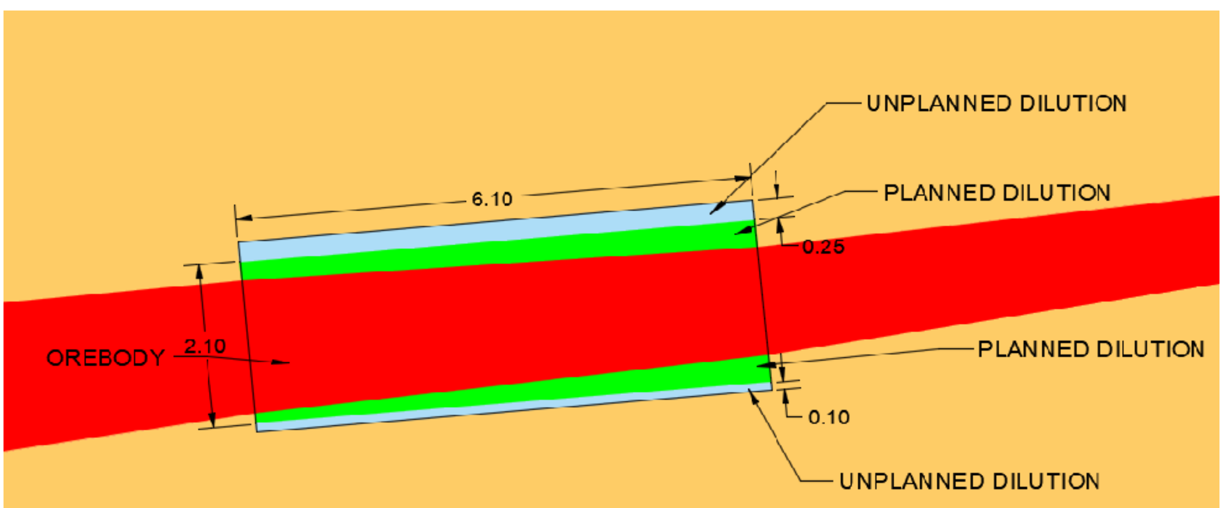
The dilution grade assigned depends on whether the material is inside the block model or not. If the material is inside the block model, the grade value of the block model is given. Should the material be outside of the block model, a grade of 0.15 % Cu and 1.5 g/t Ag is specified. This represents the 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.

### 15.3.1 Internal Dilution (planned dilution)

Planned dilution is the part of the dilution included in the stope design. Two (2) scenarios can create dilution in the mining rooms with regards to the Copperwood Project. In the first scenario, this dilution is added to the floor to reach a minimum mining height of 2.1 m in production panels with Jumbo Drill and a minimum mining height of 3 m for the main access development and production panels with Continuous miner (Road header). In the second scenario, the dilution is added to the floor so as to not exceed 6° of side dip. Figure 15.3 shows how dilution is applied for a drilling jumbo chamber with a floor with a lateral inclination of less than 6°.

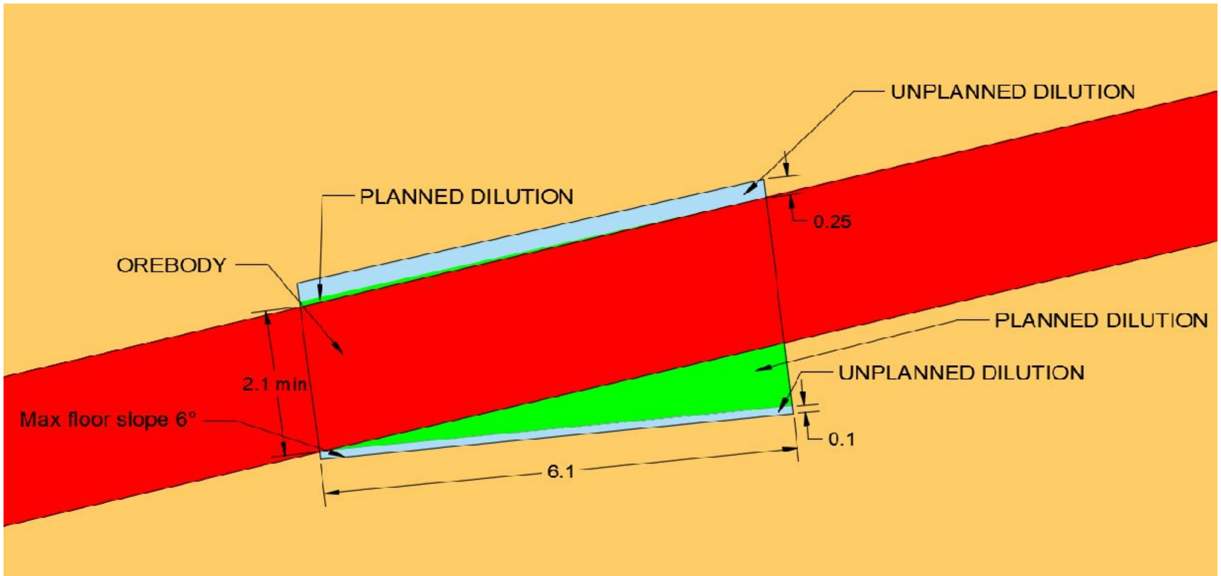
Figure 15.4 shows how dilution is applied for a drilling jumbo chamber with a floor with a lateral inclination of more than 6°. Both approaches are similar for continuous miners, except that the minimum height would be 3 m instead of 2.1 m.

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




| | | | |
|---|---|---|---|
| GMining Services | 7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2 | Title: Typical Room and Pillar Shape (maximum floor slope <6°) | Author: Carl Michaud, P.Eng. Date: 25/05/2018 |

**Figure 15.4: Typical Room and Pillar Shape (floor slope >6°)**




| | | | |
|---|---|---|---|
| Mining Services | 7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2 | Title: Typical Room and Pillar Shape (floor slope >6°) | Author: Carl Michaud, P.Eng. Date: 25/05/2018 |

### 15.3.2 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.

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.

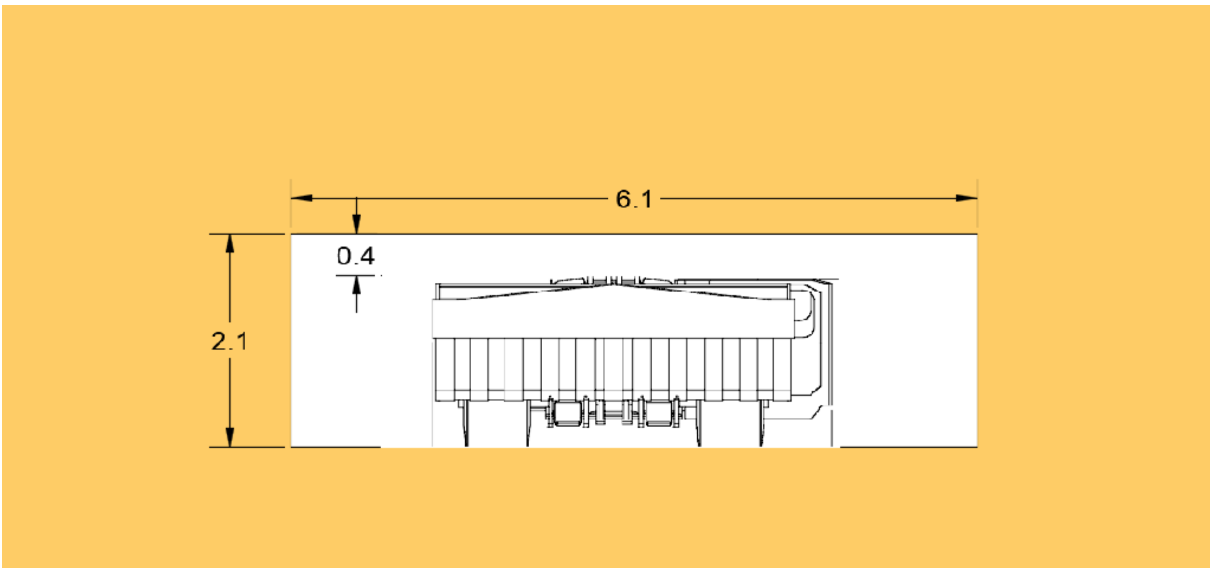
## 15.4 Minimum Mining Height

The minimum mining height primarily depends on the orebody's geometry and the selection of mining equipment. Two (2) types of equipment are currently planned for the Copperwood project, namely low-profile Jumbo drills and a continuous miner (roadheader).

For the rooms and pillars mined with the Jumbo drill, the minimum height before dilution is 2.1 m. In the case of rooms and pillars mined with a continuous miner (roadheader), the minimum height is 3.0 m.

This minimum height allows the production equipment to move easily and the drilling equipment, as well as the continuous miner, to have enough space for the operation.

**Figure 15.5: Minimum Mining 2.1 m Height Stope**




| | | | |
|---|---|---|---|
| **Mining Services** | **7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2** | **Title: Minimum Mining Height Jumbo Operation** | **Author: Carl Michaud, P.Eng. Date: 25/05/2018** |

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) | Ore zone Height (m) | Mining Recovery | Dilution | | |
|---|---|---|---|---|---|---|
| | | | | Total | Planned | Unplanned |
| West Zone | 11.17 | 2.80 | 69.8% | 14.3% | 1.8% | 12.6% |
| East Zone | 11.35 | 1.76 | 74.0% | 46.4% | 30.5% | 15.9% |
| Barrier Pillar | 1.40 | 2.49 | 65.0% | 20.3% | 8.3% | 12.0% |
| **Total Room-and-Pillar** | **23.91** | **2.26** | **71.7%** | **30.8%** | **16.6%** | **14.2%** |
| **Development** | **1.79** | | **65.0%** | **70.4%** | **58.7%** | **11.7%** |
| **Total Ore** | **25.70** | **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.
- Cost assumptions, particularly cost escalation.
- Geological complexity and continuity.
- Dilution and recovery factors.
- Geotechnical assumptions concerning rock mass stability.
- Hydrogeological assumptions concerning water seepage.
- License with third parties.
- Cut-off NSR estimations.
- Capital and operating cost assumptions.
- Geological complexity and resource block modelling.
- Stope 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 are 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 room-and-pillar given the relatively sub-horizontal orebody that varies in thickness from 1.6 m to 3.7 m. Based on the orebody thickness, two (2) approaches were selected to carry out the development of the room and pillar: conventional drill and blast, and continuous mining. The drill and blast approach is utilized whenever the orebody thickness is below 3.0 m, whereas the continuous miner will be used in the areas where the orebody thickness is 3.0 m or greater.

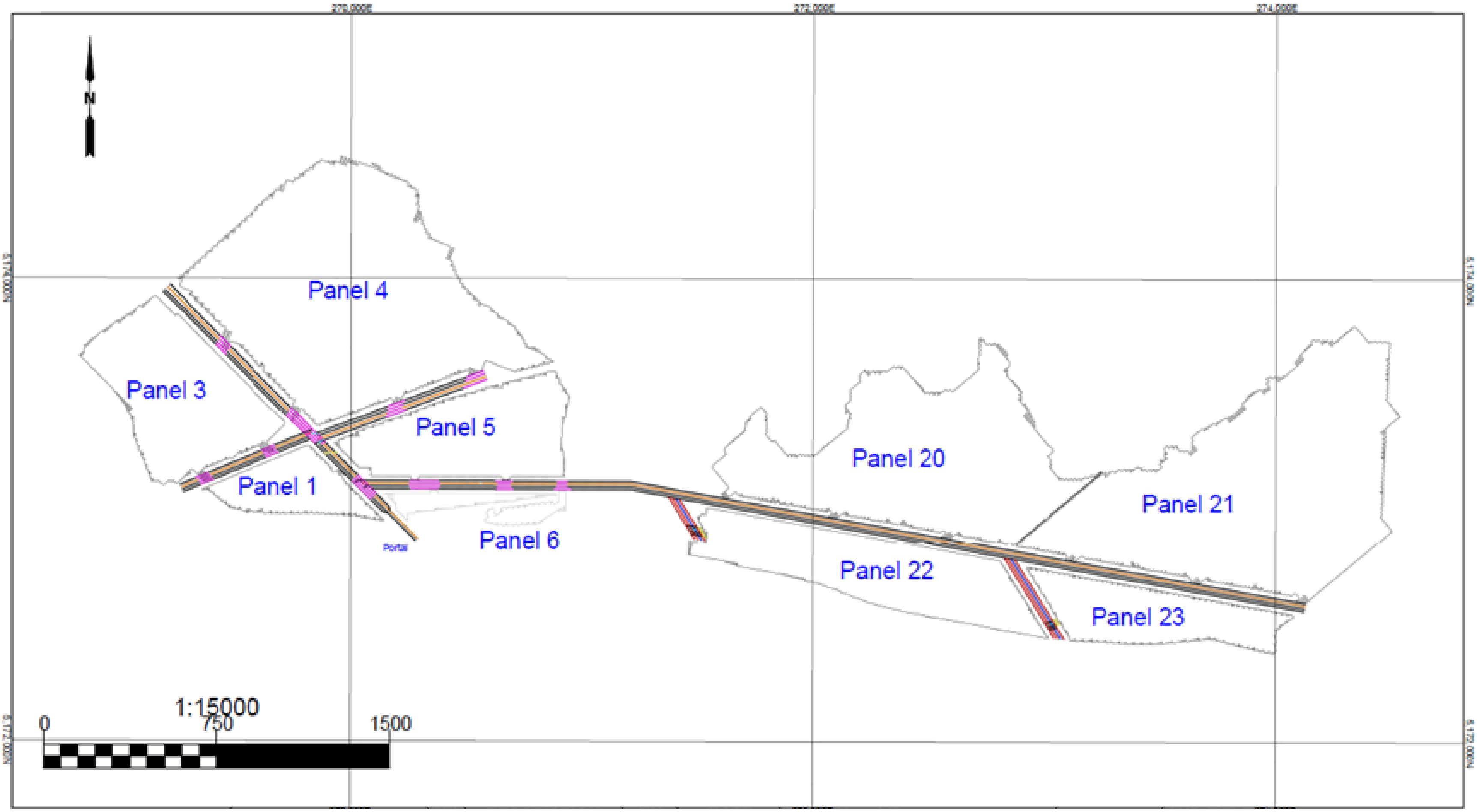
The mining method consists of the extraction of a series of entries and crosscuts in the ore leaving pillars in place to support the back. The entries cross cuts and pillars are sized using a geotechnical analysis of the rock, and experience from other mines sharing similar ground conditions.

The mining equipment for the project consists of a low-profile two (2) boom electro-hydraulic jumbo used for drilling in areas under three (3) metres back height. For higher drifts and stopes, the ore will be excavated using a continuous miner of the Roadheader type. A one (1) boom low-profile electro-hydraulic bolter is being considered for ground support installation. A low-profile, 10-tonne (6 $m^3$) capacity LHD planned for removing ore from the face and transporting broken ore to a loading point. At the loading point, a feeder breaker will reduce the size of larger particles of ore, which will be placed on a conveyor belt and transported to crushed ore storage bins at the surface, from which the mill will be fed.

The mine is comprised of two (2) 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 five (5) extraction panels as detailed in Figure 16.1. The East part is subdivided into four (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 in 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 Plan View**



## 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 1950s. 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 (2) 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 (1) 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 (1) of the more notable observations was that significant noise occurred in the four (4) to eight (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 4 to 8 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.

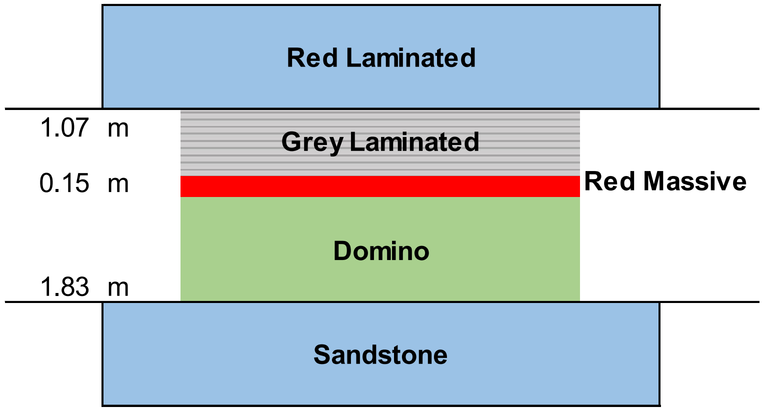
There are several reports and published papers from the time that suggest that these failures may be the result of poor-quality bolting, high horizontal stresses, and the use of water during drilling of bolts that may have led to weakening of the RLAM. It is not clear, based on current knowledge, whether 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**

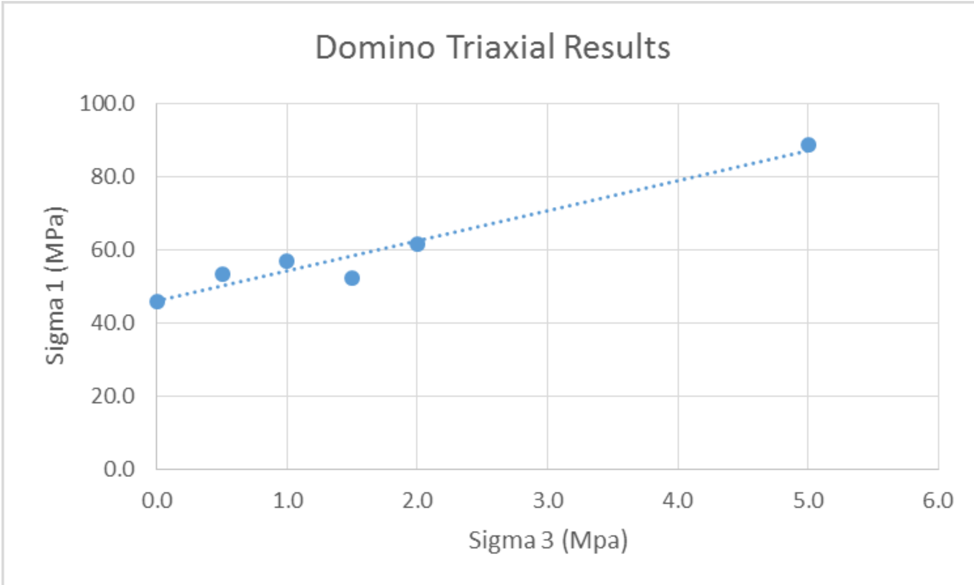
| Subunit | West Orebody | | East Orebody | |
|---|---|---|---|---|
| | Number of Tests | Average UCS (psi) | Number of Tests | Average UCS (psi) |
| RLAM | 17 | 8,550 | 16 | 10,600 |
| GLAM | 16 | 8,550 | 19 | 12,750 |
| RMAS | 11 | 10,600 | 5 | 12,900 |
| DOMN | 16 | 6,700 | 7 | 7,800 |

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 eight (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 <u>Rock Structure</u>

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 (2) 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.

At this time, there is no evidence 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 (1) 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 |

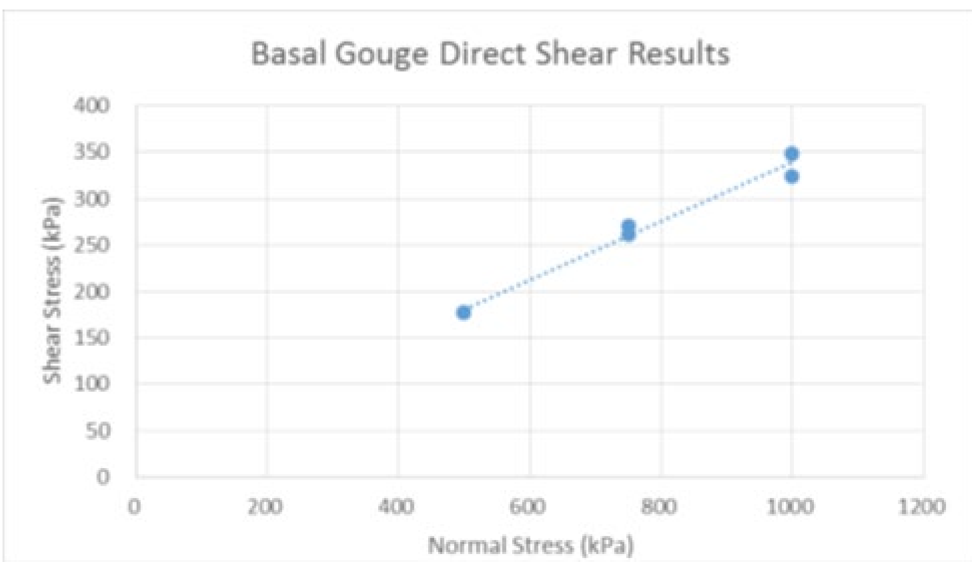
### 16.2.8 Basal Gouge

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 (2) samples of gouge were tested in direct with slickensides fabric aligned with the contact shear at confining stresses between 500 and 1,000 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 versus 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 (9) 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 |
| Depth (m) | 0-91 | 91-183 | 183-274 | 91-183 | 183-274 | 91-183 | 183-274 | 0-122 | 0-122 |
| GLAM | 3.5 | | | 3.0 | 2.0 | 3.5 | 3.5 | 3.0 | 2.5 |
| RMAS | 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 Lake Superior 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 16.8.4 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's main access from the glacial overburden will be minimal.

## 16.3 Mining Method Selection

Based on geotechnical information and mineralization geometry, a room-and-pillar mining method is selected for the Copperwood deposit. The principle of this method is to dig horizontal drift in the mineral layer, leaving intact ore pillars to support the roof of the mine. These pillars are left in place to form a grid of pillars and chambers, hence the name of the method. The drift can be aligned to form a network of underground passages, allowing miners, equipment, and materials to be transported inside the mine. The mining design was based on a mining rate of approximately 2.5 Mt/year. Two (2) approaches are planned for the extraction of the chambers. In the case of lower-height chambers (-3 m), the approach with low-profile drilling jumbos is recommended. In the case of chambers greater than 3 m, a continuous miner of the road header type is planned.

The underground access and infrastructure development were designed to support the mining method and size based on mining equipment and production rate requirements. Table 16.5 shows the selected mining approach by activities.

**Table 16.5: Mining Approach by Activities**

| Development Activities | Mining Approach | |
|---|---|---|
| Main Access Drifts | Conventional development (Jumbo Drill and Blast) | Continuous Miner |
| 1N | 5% | 95% |
| 3W | 5% | 95% |
| 4E | 5% | 95% |
| 5 W | 5% | 95% |
| 22S | 5% | 95% |
| 23S | 5% | 95% |
| **Stope Panels** | | |
| West Panels | | |
| PANEL 1 | 15% | 85% |
| PANEL 3 | 15% | 85% |
| PANEL 4 | 32% | 68% |
| PANEL 5 | 6% | 94% |
| PANEL 6 | 100% | 0% |
| East Panels | | |
| PANEL 20 | 100% | 0% |
| PANEL 21 | 100% | 0% |
| PANEL 22 | 100% | 0% |
| PANEL 23 | 100% | 0% |
| Barrier Pillars | | |
| West | 100% | 0% |
| East | 100% | 0% |

### 16.3.1 Development Design

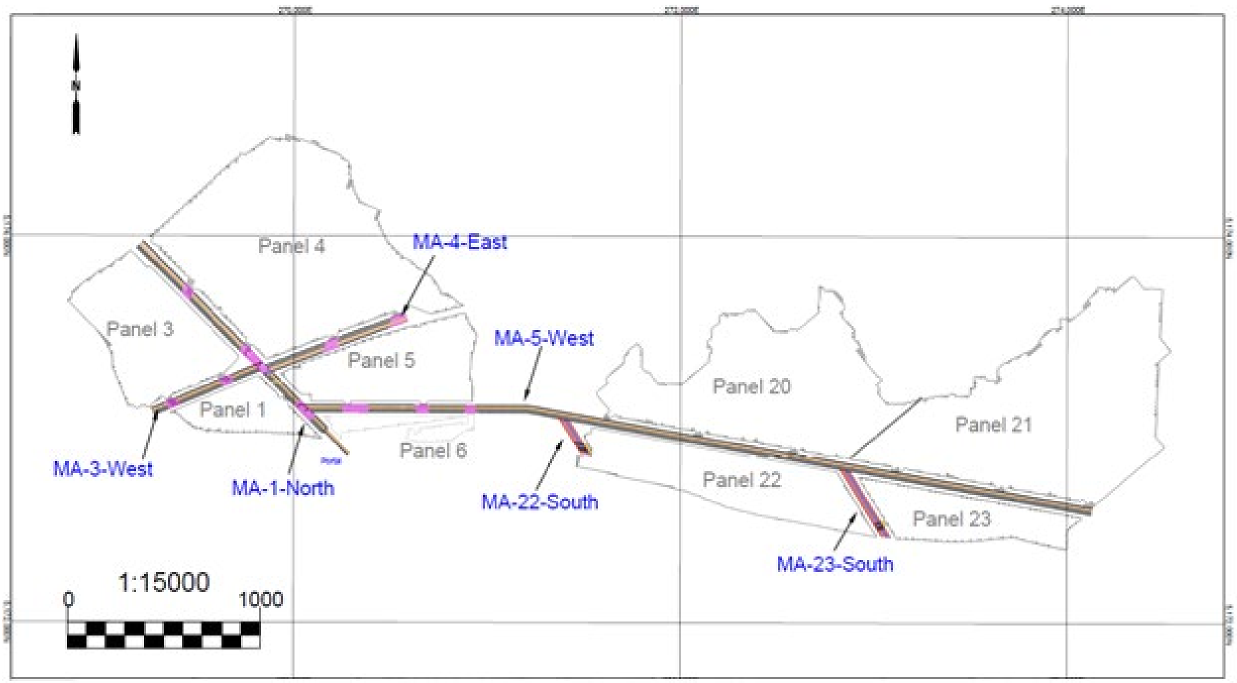
#### 16.3.1.1 Main Access Drift

The mine will be accessed through a circular multiplate style box cut. Only two (2) drifts will be excavated from the surface portal to a depth of 35 m, at which point the two (2) main drifts will be split to create a total of four (4) separate drifts. The four (4) drifts in main access include 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 surface portal. If waste is encountered during development, it will be stored in a closed underground excavation. All drifts will have a width of 6.1 m, and their height will vary from a minimum of 3 m to a maximum of 6 m. The backs of the drifts will follow the deposit's geology to allow for better resource recovery. However, the floor of the drifts will be relatively flat to allow for equipment traffic. The height at the intersections of the ore conveyor drifts will be set at 6 m to allow for the installation of a transfer point between the panel conveyor and the main conveyor. If a main drift intersects a conveyor drift, the height of the intersection 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 production mining has been completed in this area. These barrier pillars are designed to be recovered and will respect Golder's recommendations for pillar size in this area of the mine.

The mine is divided into two (2) main sections, the East section, and the West section. The West section is given priority for exploitation as it has higher grades and better heights. Therefore, starting with this section will optimize the profitability of the project. The first access (MA-1) starts from the south of the West zone and runs through the end to the northeast. The position of this access allows the working faces in the West section to begin quickly. From this access, two (2) other series of drifts (MA-4 and MA-3) are excavated perpendicular to MA-1 and will follow the thrust fault parallelly, one (1) in the northeast direction (MA-4) and the other towards the southwest (MA-3). These two (2) new accesses will allow for the development and production of panels 3 and 4, while respecting the recommended mining directions. From the MA-1 access, the MA-5 access is excavated to develop the eastern part of the mine and panel 5 in the western part of the mine. From the MA-5 drift, two (2) other secondary access (MA-22 South and MA-23 South) will be excavated to allow access to panels 22 and 23.

The main accesses are shown in Figure 16.5 below.

**Figure 16.5: Development Design - Main Accesses Plan View**




As these accesses are over 3 m high, it is planned that they will be excavated mainly with continuous miners. However, in the case of sections with steep turns (Ex connection between 2 drifts), jumbos will be used. It is expected that 95% of the excavation will be done with continuous miners. As for stoping, the ore will be removed using LHD loaders and transported behind the face. In the case of development, it will be loaded into low-profile 30-tonne trucks and transported to the surface or onto a feeder breaker if the surface is too far away. For ground support, low-profile, single-boom bolters will be used.

### 16.3.2 Stope Entry

To access the mining production panel, three (3) 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 return air. The width of these drifts will be 6.1 m and the height will be the same as the production panel.

### 16.3.3 Intake Ventilation Raise

In addition to the drift, three (3) 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 deep. The raise will provide fresh air for the production period and allow a second emergency exit for the mine.

### 16.3.4 Exhaust Ventilation Raise

Two (2) 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 portion of the mine and will be raise bored 4 m in diameter and 85 m deep. The second exhaust raise will be in the eastern part of the mine near the boundary with the Porcupine Mountains Wilderness State Park and will be raise bored 5 m diameter and 160 m deep.

### 16.3.5 Stope Design

The orebody is divided into nine (9) main panels. The western part of the mine includes panels 1, 3, 4, 5, 6 and the eastern part includes panels 20 to 23. The thrust fault located in the western horizon splits panel 4 from panel 5 and panel 1 from panel. 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. Panel 6 includes historical mining where stopes collapsed. In the eastern horizon, panels 20 and 21 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. Figure 16.6 and Figure 16.7 present the panel division for the western section and the eastern section of the mine, respectively. Table 16.6 and Table 16.7, respectively, present the mine design summary and the stope pillar size.

**Figure 16.6: Panel Division – Western Section**

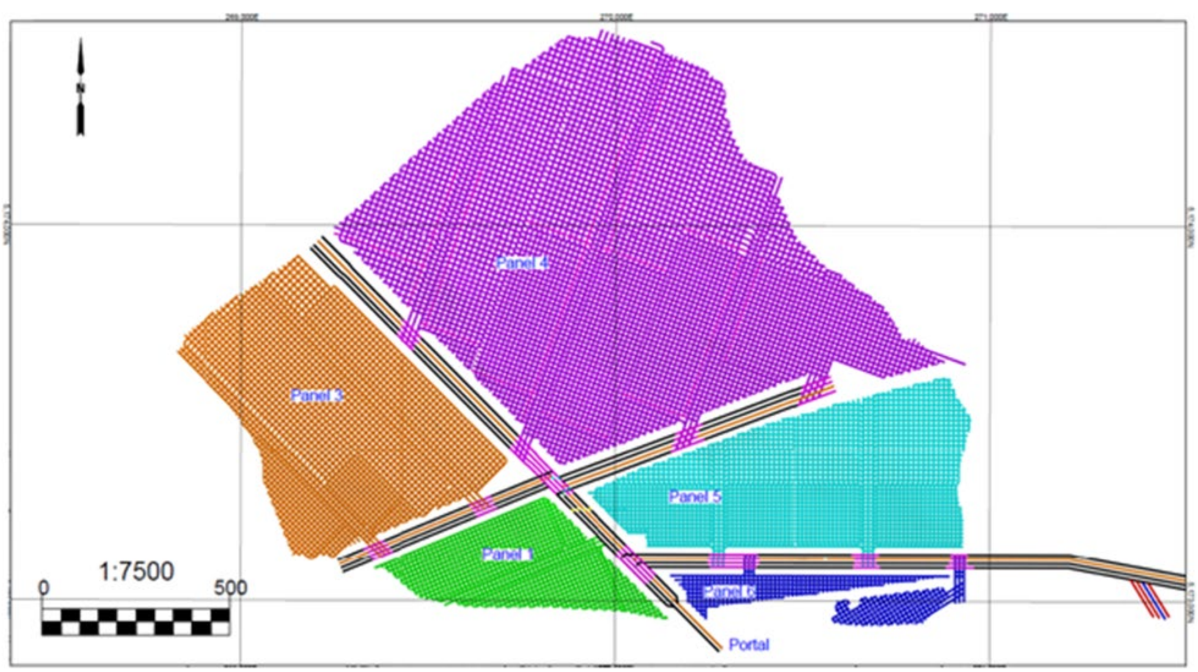


**Figure 16.7: Panel Division – Eastern Section**

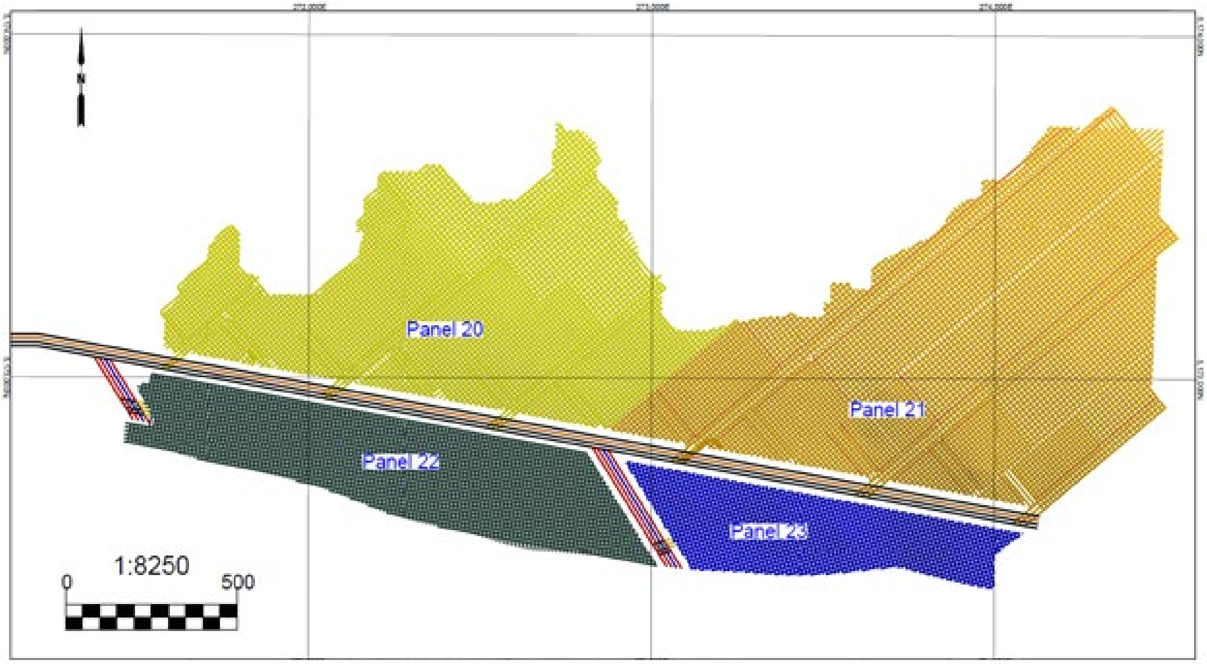



**Table 16.6: Mine Design Summary**

| LoM Physicals | |
|---|---|
| Ore Tonnes Development (Mt) | 1.787 |
| Cu Grade % | 1.22 |
| Ag Grade (g/t) | 3.54 |
| Stope Production (Mt) | 23.91 |
| Cu Grade % | 1.46 |
| Ag Grade (g/t) | 3.94 |
| Total Underground Production (t) | 25.70 |
| Cu Grade % | 1.45 |
| Ag Grade (g/t) | 3.91 |
| Waste Tonnes (Mt) | 0.15 |
| **Development** | **Metres** |
| Lateral Development | 31,139 |
| Ventilation Raise 5 m | 308 |
| Ventilation Raise 4 m | 85 |
| Rock Breaker Excavation (waste – tonnes Mt) | 0.13 |

**Table 16.7: 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 | | 274 | 7.6 | 69% |
| | 21 | 183 | 6.1 | 75% |
| | | 274 | 7.6 | 69% |
| | 22 | 122 | 4.9 | 80% |
| | 23 | 122 | 5.2 | 79% |

## 16.4 Mine Operations

### 16.4.1 Stoping

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 return air. Between the stopes and the main access, a barrier pillar is kept for protecting the main access. From the stope accesses, the panel operation begins with the drilling and blast method or the continuous miner excavation.

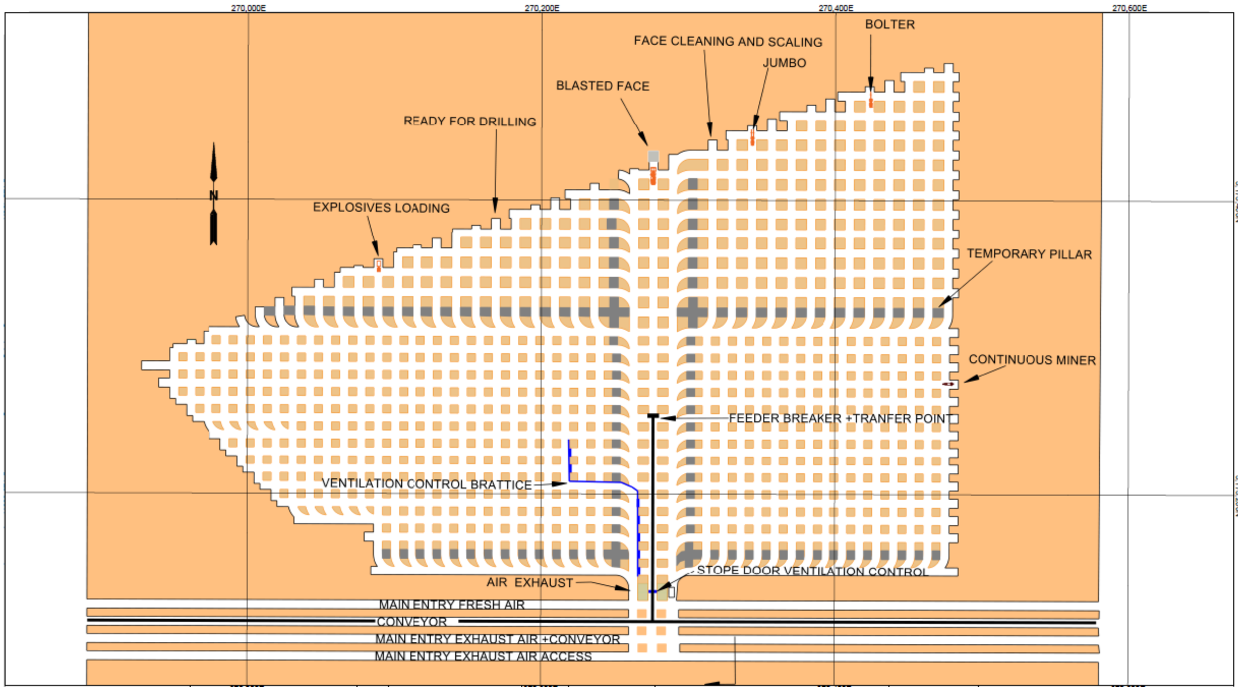
#### 16.4.1.1 Drill and Blast operation

To achieve and maintain an adequate level of production, in drill and blast section 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.

In conventional 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 two (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/min 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.

Figure 16.8 shows the configuration of the production panel.

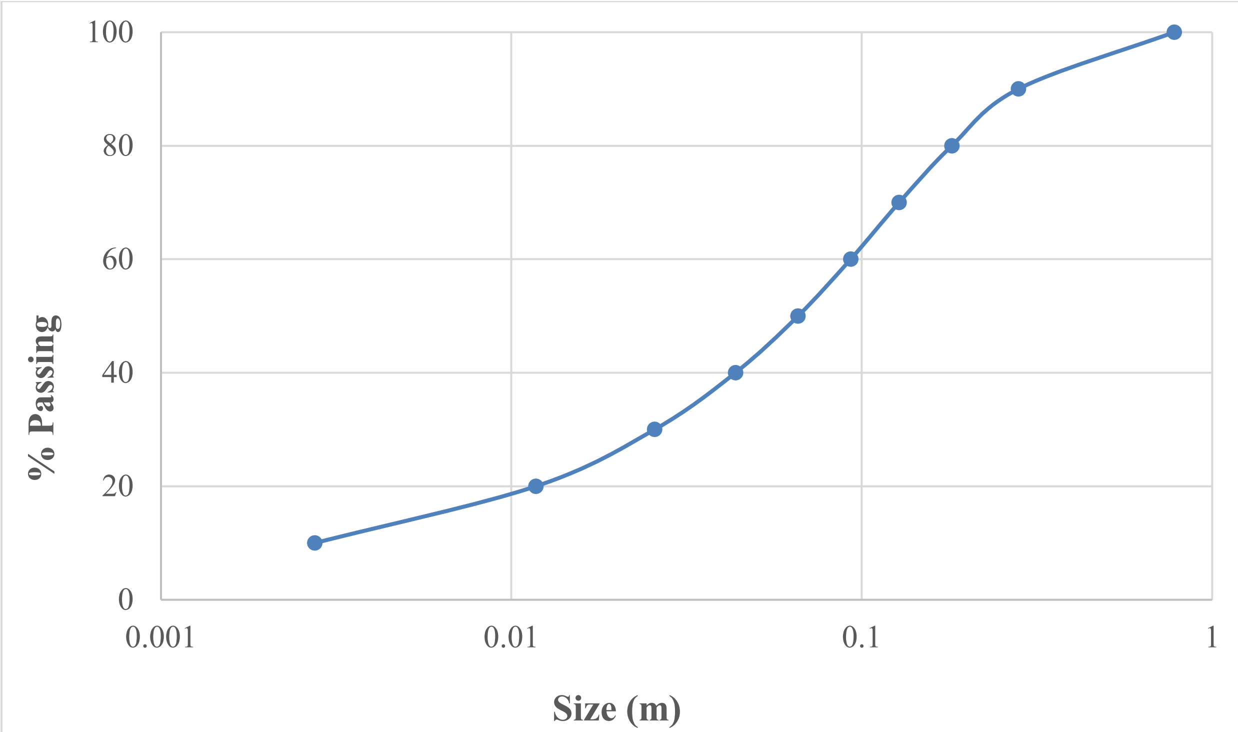
**Figure 16.8: Room and Pillar Stope Configuration**




Blasting crews will load the rounds with explosives and initiate blasts at the end of each shift. Explosives will consist of an emulsion mixture. Emulsion is better suited in the 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 emulsion as explosives.

**Figure 16.9: Particle Size Distribution Red Massive with Emulsion Explosives**




The blasting of the loaded round will be performed at the end of every shift. A period of two (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.

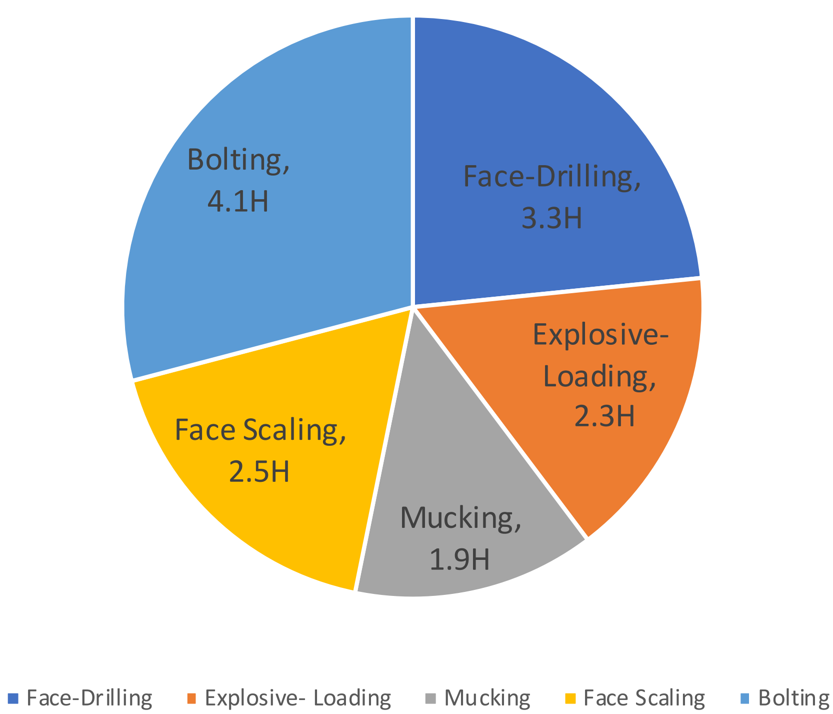
The third mining activity is to muck the blasted ore from the face and to transport it with a low-profile 10t LHD. The performance of the LHD is a function of the dip of the stope and the distance between the face heading and the rock breaker. The LHD performance will vary from 3.9 km/h at 17% (loaded) 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 economically justify 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 10 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. In order to proceed, 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. Some 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/min.

The total round cycle time is estimated at an average of 14.1 h/round. The breakdown is shown in Figure 16.10 below.

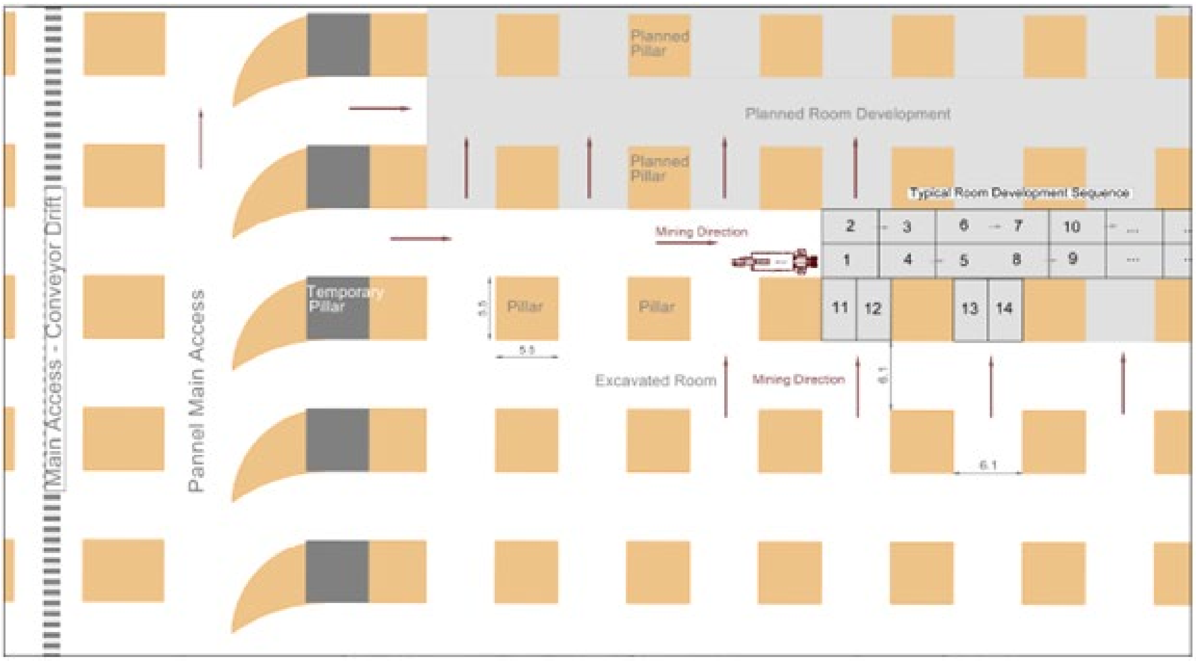
**Figure 16.10: Round Cycle Time**




#### 16.4.1.2 <u>Continuous Mining Operation</u>

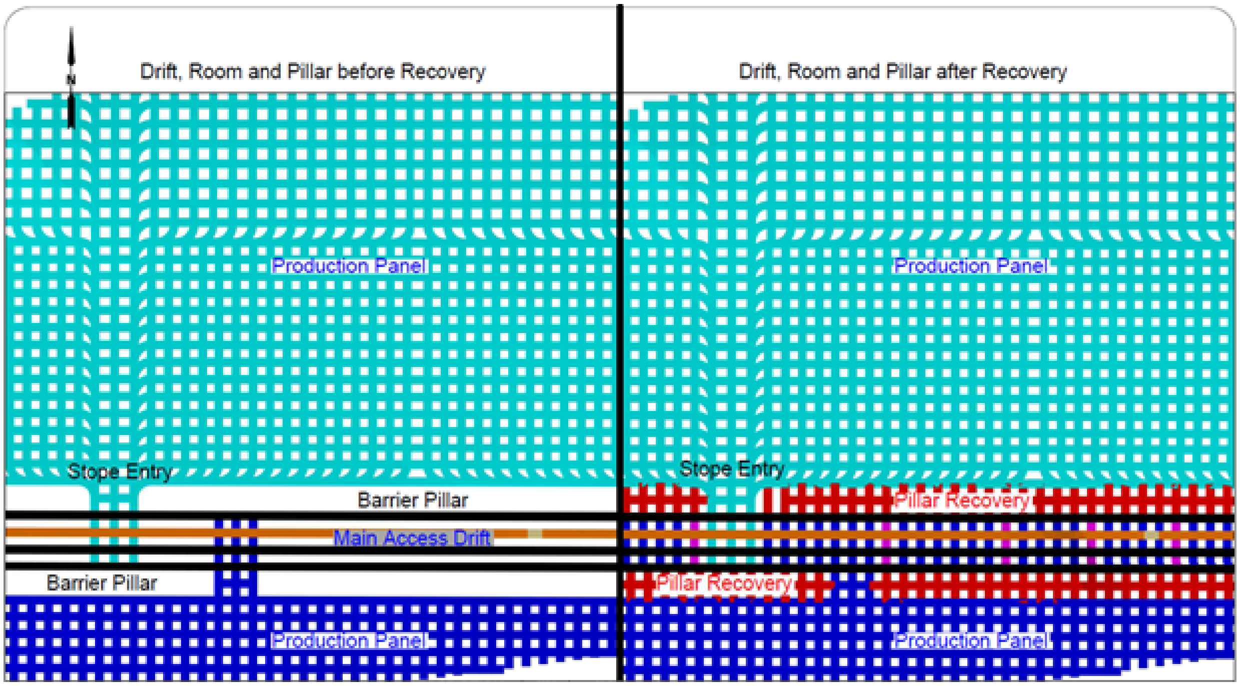
In the continuous operating panels, the continuous miner begins with the cutting of a 3 m-wide section of the ore (room). The continuous miner loads the broken material to the LHD, which hauls the ore from the panel to the feeder breaker. After the broken material produced from the cut section is loaded, the operator backs out of the partially formed room and the rock bolting process, previously described in the conventional operating section, is done and allows the continuous miner to re-enter the room and begin the cutting of an additional 3 m section to produce a wider and final room advance. Figure 16.11 below presents the typical cutting sequence expected in a continuous operating panel.

**Figure 16.11: Continuous Miner Room Development Sequence**




Once mining in an area is complete, the maximum amount of ore will be recovered from the barrier and drift pillars as illustrated in Figure 16.12.

**Figure 16.12: Drift, Room and Pillar before Recovery**




### 16.4.1.3 <u>Mining Parameters</u>

The basic operational assumptions are summarized as follows:

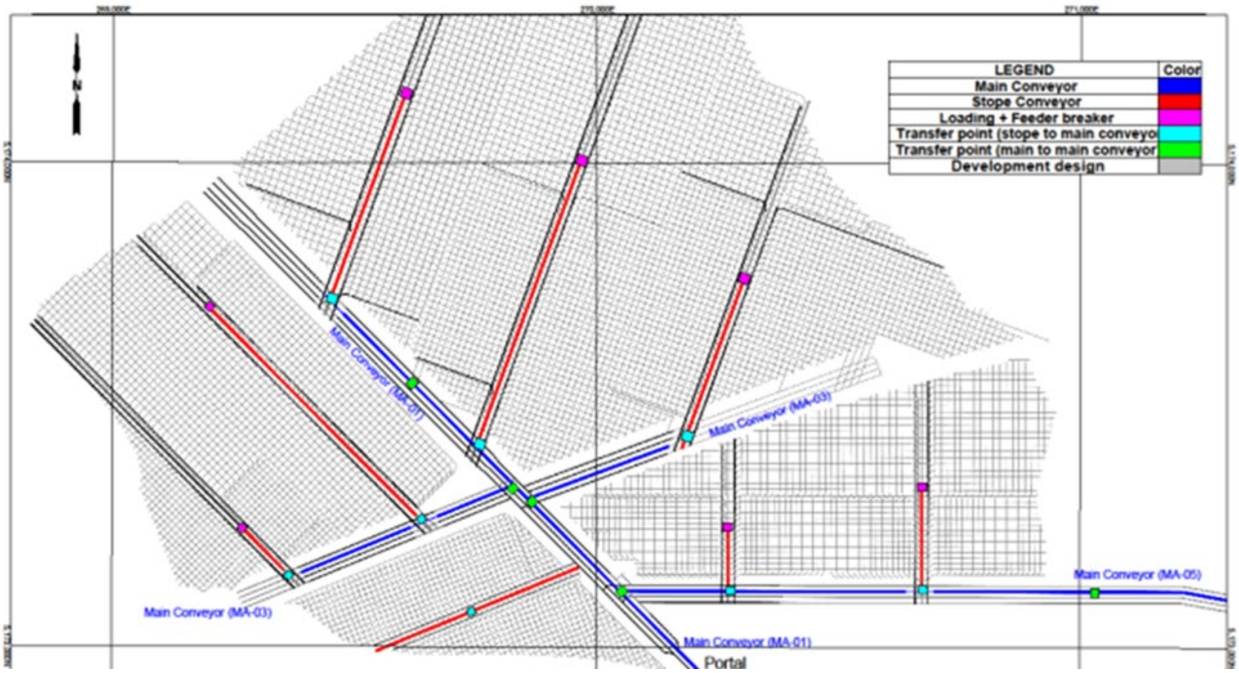
- Minimum mining height 2.1 m for Jumbo Room and 3 m for continuous miner (limited by the equipment).
- Maximum mining height 6.0 m.
- Average mining height 2.5 m.
- Average mining height, western sector 2.81 m.
- Average mining height, eastern sector 2.21 m.
- Cut-off grade 1.0% Cu. %
- Annual production – 2.5 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 10 m.

- Fresh air raise 5 m diameter.
- East exhaust air raise 5 m diameter.
- West exhaust air raise 4 m diameter.
- 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 feeder-breaker loading point. The broken ore from the stope will be mucked by a low-profile LHD to a stope feeder breaker. The feeder breaker reduces large rocks to a diameter that allows them to be loaded onto a conveyor belt. 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 (1) main conveyor is 1,200 m. After this distance, a second is installed and interrelated between them. In the West section, two (2) main underground conveyors are required. In the East section, four (4) main conveyors are required. Each of these conveyors is equipped with 500 HP electric motor. The main conveyor transports the ore to the surface.

**Figure 16.13: West Section Conveyor Arrangement**




### 16.4.3 Mining Equipment

Table 16.8 shows the equipment requirements to support the planned 6,800 mtpd nominal production rate.

**Table 16.8: Mine Equipment Requirements**

| Mobile Equipment | |
|---|---|
| Continuous Miner | 4 |
| Low-Profile 2 Booms Jumbo Drill | 6 |
| Low Profile 1 Boom Electric-Hydraulic Bolter | 14 |
| Low Profile LHD 10 Mt | 9 |
| Low Profile LHD 8 Mt | 2 |
| Scaler | 3 |
| Development Truck | 4 |
| Lube Trucks | 1 |
| Flat Bed Trucks | 2 |
| Scissor Lift | 5 |
| Grader | 1 |
| Tractor - Underground | 11 |

| | |
|---|---|
| **Mobile Equipment** | |
| ATV - Underground | 8 |
| Explosive trucks | 2 |
| Cable Bolt Drill Stope Mate Drill | 1 |
| **Ore Handling System** | |
| Loading Point+ Rock Breaker | 5 |
| Main Conveyor 1,200 m – 500 HP | 8 |
| Stope Conveyor 500 m – 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 1,250 HP | 2 |
| 50 MBTU Natural Gas-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

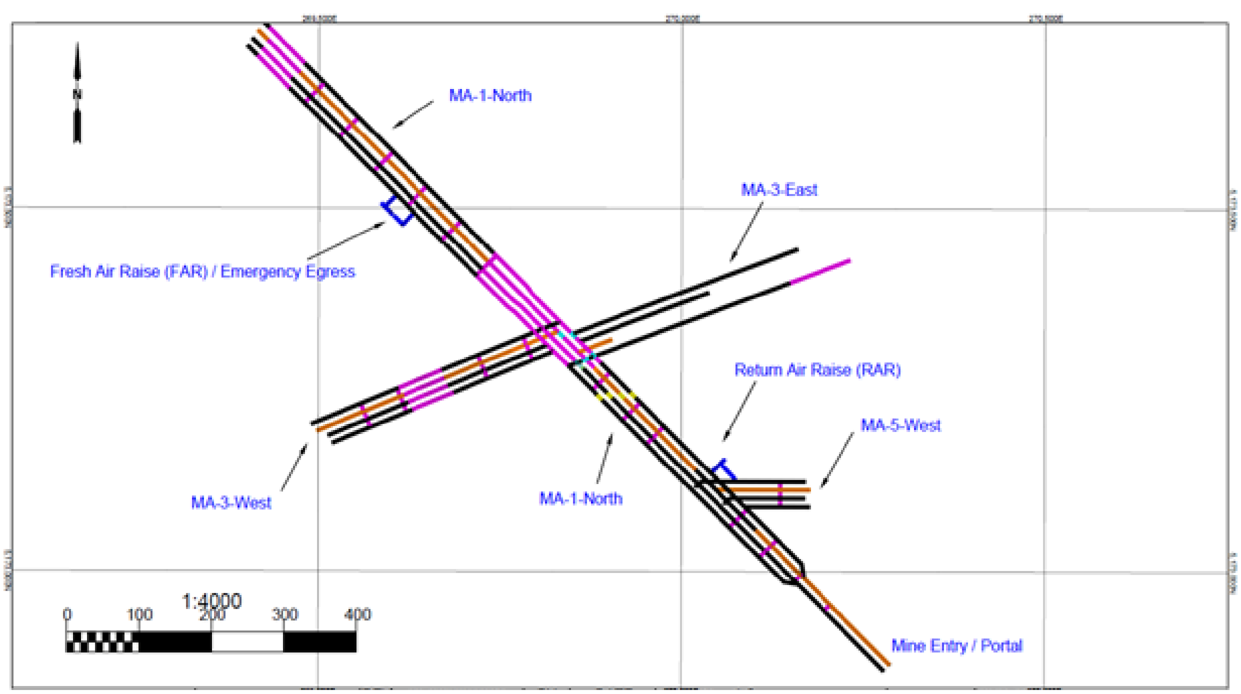
Development will be divided into two (2) periods: a pre-production development period (from the beginning to the 21st month) and a production period (from the 21st month to the end).

### 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 systems, dewatering systems, compressed air and water supply).
- Develop sufficient production panels to support the mine production rate.

The Figure 16.14 below shows the development completed during the pre-production period.

**Figure 16.14: Pre-Production Development Period**



### 16.5.2 Production

It was assumed that the first six (6) months of pre-production will be excavated by a mining contractor. The rest of the preproduction and production drift development will be excavated by the Owner's mining department.

The owner approach is preferred to reduce development. Once the portal excavation is completed, development of the main access drifts will begin. The production of the two (2) main access drifts from the portal will be 5 m/d. Once the main access drifts divide into four (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 three (3) teams. From February 2025, the number of development teams will be reduced to two (2) and subsequently only one (1) will be remaining in June 2025. Drift development will be completed in 2026.

Excavation 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 21 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 2.5 M t/yr. To achieve this annual production, seven (7) to nine (9) 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.

In April 2026, the mining of the first stope, from which the mineral will be sent to the process plant, will begin. Before August 2026, 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 establish the production from the western part of the mine as this zone has better grade and taller rooms allowing for higher productivity. Table 16.9 below presents the ramp up summary.

**Table 16.9: Ramp-Up Summary**

| Tonnage | Pre-Production | | | | | | | | Production Ramp-Up | | | | | Production | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2024 | | | | 2025 | | | | 2026 | | | | 2027 | | | |
| | Q1 | Q2 | Q3 | Q4 | Q1 | Q2 | Q3 | Q4 | Q1 | Q2 | Q3 | Q4 | Q1 | Q2 | Q3 | … |
| Development (tonnes) | - | - | 26,873 | 50,450 | 87,148 | 78,226 | 97,115 | 81,236 | 106,226 | 138,934 | 88,622 | 80,199 | 24,278 | 24,553 | 24,362 | 23,758 |
| Stoping (tonnes) | - | - | 273 | 1,912 | 3,260 | 999 | 4,618 | 3,635 | 208,154 | 344,550 | 435,000 | 449,990 | 576,000 | 584,550 | 585,000 | 585,000 |
| **Total Tonnes** | - | - | 27,146 | 52,362 | 90,409 | 79,224 | 101,733 | 84,871 | 314,380 | 483,484 | 523,622 | 530,189 | 600,278 | 609,103 | 609,362 | 608,758 |
| Tonnage / days | - | - | 295 | 569 | 1,005 | 871 | 1,106 | 923 | 3,493 | 5,313 | 5,692 | 5,763 | 6,670 | 6,693 | 6,623 | 6,617 |

The Jumbo 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 tpd and can reach up to 1,285 tpd for a 3.9 m high stope. The Continuous miner productivity is estimated at 1,800 tpd independent of stope height.

The limit of stope production is the productivity of the jumbo drill and the continuous miner's advance rate. Table 16.10 below presents the productivity rate per mining panel.

**Table 16.10: 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 and slowly introduces tonnage from the eastern portion of the orebody in 2030. The copper grade drops in value in 2030 and levels around 1.30%; whereas it averages 1.69% from 2024 to 2029. In 2027-2028 and from 2030, pillar recovery occurs where main drifts are no longer useful. Table 16.11 and Figure 16.15 below present the mine production schedule summary and the production (panel mining) sequence respectively.

**Table 16.11: Mine Production Schedule Summary**

| Mine Production | | Total | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Development Mining | | | | | | | | | | | | | | | | |
| Tonnage | kt | 1,788 | - | 77 | 344 | 414 | 97 | 93 | 281 | 290 | 192 | - | - | - | - | - |
| Cu Head Grade | %Cu | 1.22 | - | 1.48 | 1.40 | 1.47 | 1.17 | 0.66 | 0.90 | 1.19 | 1.02 | - | - | - | - | - |
| Ag Head Grade | g/t | 3.54 | - | 5.25 | 4.89 | 5.05 | 2.59 | 1.45 | 1.74 | 2.96 | 2.20 | - | - | - | - | - |
| Cu Contained Metal | kt | 22 | - | 1.1 | 4.8 | 6.1 | 1.1 | 0.6 | 2.5 | 3.5 | 2.0 | - | - | - | - | - |
| Ag Contained Metal | K ozs | 204 | - | 13.0 | 54.1 | 67.2 | 8.1 | 4.3 | 15.8 | 27.6 | 13.5 | - | - | - | - | - |
| Production Mining | | | | | | | | | | | | | | | | |
| Tonnage | kt | 23,916 | - | 2 | 13 | 1,438 | 2,331 | 2,394 | 2,220 | 2,220 | 2,292 | 2,502 | 2,502 | 2,502 | 2,496 | 1,004 |
| Cu Head Grade | %Cu | 1.46 | - | 1.61 | 1.72 | 1.74 | 1.78 | 1.70 | 1.69 | 1.39 | 1.38 | 1.26 | 1.26 | 1.31 | 1.33 | 1.34 |
| Ag Head Grade | g/t | 3.94 | - | 5.44 | 5.73 | 5.73 | 6.07 | 5.99 | 6.04 | 4.43 | 3.70 | 2.21 | 2.31 | 1.93 | 2.54 | 3.15 |
| Cu Contained Metal | kt | 350 | - | 0 | 0 | 25 | 41 | 41 | 38 | 31 | 32 | 32 | 32 | 33 | 33 | 13 |
| Ag Contained Metal | K ozs | 3,029 | - | 0 | 2 | 265 | 455 | 461 | 431 | 317 | 273 | 178 | 186 | 155 | 204 | 102 |
| Total Mining | | | | | | | | | | | | | | | | |
| Tonnage | kt | 25,703 | - | 80 | 356 | 1,852 | 2,428 | 2,487 | 2,502 | 2,510 | 2,484 | 2,502 | 2,502 | 2,502 | 2,496 | 1,004 |
| Cu Head Grade | %Cu | 1.45 | - | 1.48 | 1.41 | 1.68 | 1.75 | 1.66 | 1.60 | 1.37 | 1.35 | 1.26 | 1.26 | 1.31 | 1.33 | 1.34 |
| Ag Head Grade | g/t | 3.91 | - | 5.25 | 4.92 | 5.58 | 5.93 | 5.82 | 5.56 | 4.26 | 3.59 | 2.21 | 2.31 | 1.93 | 2.54 | 3.15 |
| Cu Contained Metal | kt | 372 | - | 1 | 5 | 31 | 43 | 41 | 40 | 34 | 34 | 32 | 32 | 33 | 33 | 13 |
| Ag Contained Metal | K ozs | 3,233 | - | 13 | 56 | 332 | 463 | 466 | 447 | 344 | 286 | 178 | 186 | 155 | 204 | 102 |

**Figure 16.15: Panel Sequence**

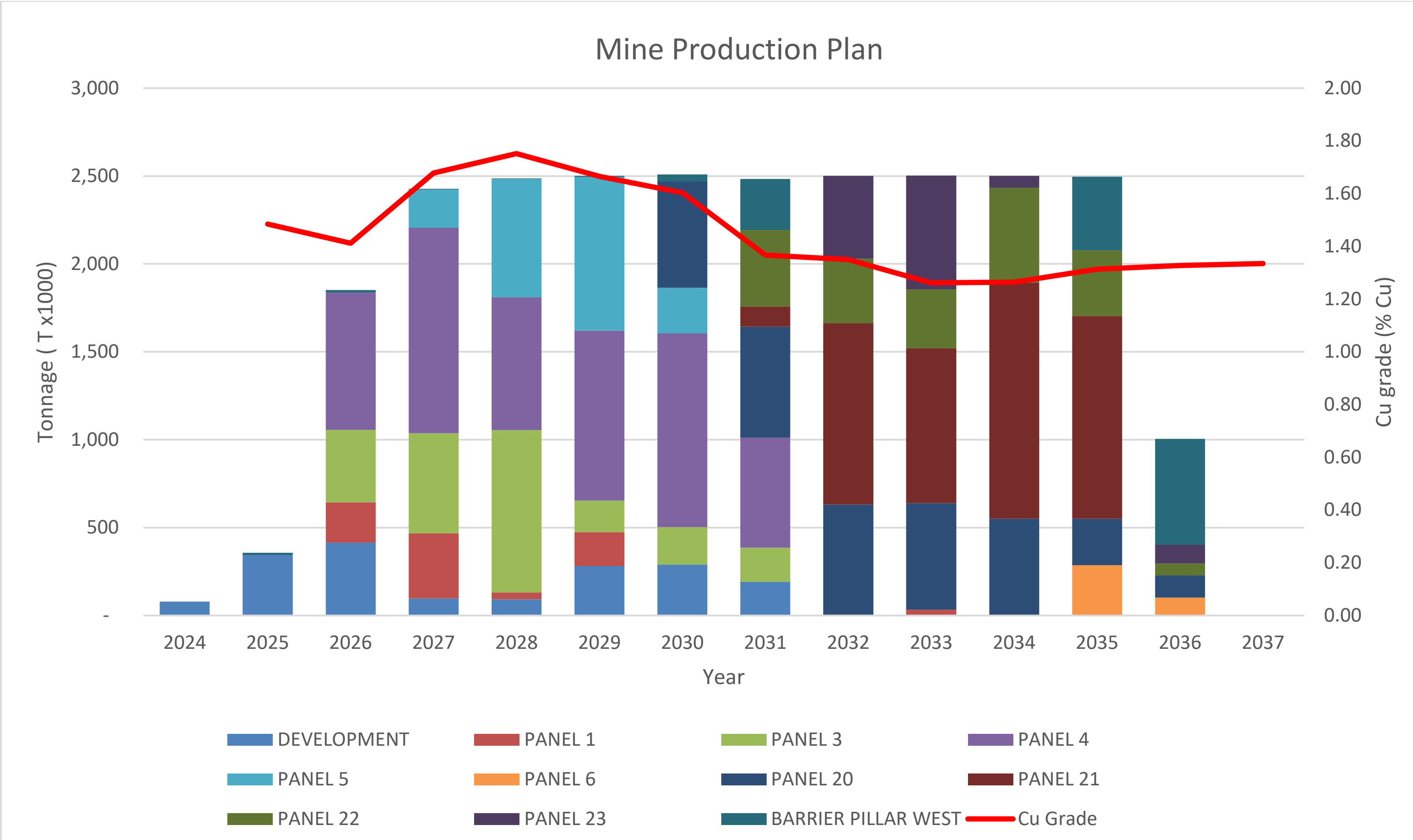


**Table 16.12: Operating Shift Assumptions**

| 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 Workplace | 30 |
| Workplace Inspection | 15 |
| Equipment Inspection / Set-up | 15 |
| Lunch (includes travel time) | 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 & Continuous Miner Availability | 85% |
| Jumbos & Continuous Miner Available Hour | 6,120 |
| Utilization % | 62% |
| Jumbos & Continuous Miner Operating Hour | 3,881 |

## 16.7 Manpower and Working Schedule

Labour levels are estimated based on the production schedule and equipment requirements to reach a production level of 2.5 Mt/yr. To achieve the level of productivities used in this Study, the workforce must be a mix of skilled labour with an experienced management team. The mine work schedule is based on working two (2) 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. The Table 16.13, Figure 16.16 and Table 16.14 present the different schedules for the underground mining operation and summarize the manpower requirement over the life of mine.

**Table 16.13: Production Working Schedule**

| Grade | Job Title | Rotation Schedule | Worked Hours |
|---|---|---|---|
| **Technical Services** | | | |
| Staff | Chief Mine Engineer | 5 On/2 Off | 2,080 |
| Staff | Long-Term Planning Engineer | 5 On/2 Off | 2,080 |
| Staff | Short-Term Planning Engineer | 5 On/2 Off | 2,080 |
| Staff | Project Engineer | 5 On/2 Off | 2,080 |
| Staff | Senior Geotechnical Engineer | 5 On/2 Off | 2,080 |
| Staff | Mine Technician | 5 On/2 Off | 2,080 |
| Staff | Geotech. Technician | 5 On/2 Off | 2,080 |
| Staff | Senior Surveyor | 5 On/2 Off | 2,080 |
| Staff | Surveyor | 7 On/7 Off | 2,080 |
| **Geology** | | | |
| Staff | Chief Geologist | 5 On/2 Off | 2,080 |
| Staff | Senior Geologist | 5 On/2 Off | 2,080 |
| Staff | Geologist | 5 On/2 Off | 2,080 |
| Staff | Geology Technician | 7 On/7 Off | 2,080 |
| **Mine Operations** | | | |
| Staff | Mine Manager | 5 On/2 Off | 2,080 |
| Staff | Mine Ops. Superintendent | 5 On/2 Off | 2,080 |
| Staff | Mine Ops. Foreman | 7 On/7 Off | 2,080 |
| Staff | Mine Ops. Trainer | 5 On/2 Off | 2,080 |
| Staff | Mine Secretary | 5 On/2 Off | 2,080 |
| Hourly Class 1 | Jumbo Operator | 7 On/7 Off | 2,080 |
| Hourly Class 1 | Bolter Operator | 7 On/7 Off | 2,080 |
| Hourly Class 1 | LHD Operator | 7 On/7 Off | 2,080 |
| Hourly Class 1 | Truck Operator | 7 On/7 Off | 2,080 |
| Hourly Class 1 | Grader Operator | 7 On/7 Off | 2,080 |
| Hourly | Blaster | 7 On/7 Off | 2,080 |

| Grade | Job Title | Rotation Schedule | Worked Hours |
|---|---|---|---|
| Hourly | Blaster Helper | 7 On/7 Off | 2,080 |
| Hourly | Conveyor Service | 7 On/7 Off | 2,080 |
| Hourly Class 1 | Feeder Breaker Operator | 7 On/7 Off | 2,080 |
| Hourly | Laborer | 7 On/7 Off | 2,080 |
| **Mine Maintenance** | | | |
| Staff | Mine Maint. Superintendent | 5 On/2 Off | 2,080 |
| Staff | Mine Maint. Foreman | 7 On/7 Off | 2,080 |
| Staff | Mine Maint. Planner | 5 On/5 Off | 2,080 |
| Staff | Mine Maint. Trainer | 5 On/2 Off | 2,080 |
| Staff | Mechanical Engineer | 5 On/2 Off | 2,080 |
| Staff | Electrical Engineer | 5 On/2 Off | 2,080 |
| Staff | Mine Maintenance Clerk | 7 On/7 Off | 2,080 |
| Hourly | Mechanic | 7 On/7 Off | 2,080 |
| Hourly | Electrician | 7 On/7 Off | 2,080 |
| Hourly | Welder / Machinist | 7 On/7 Off | 2,080 |
| Hourly | Fuel & Lube Technician | 7 On/7 Off | 2,080 |
| Hourly | Tyreman | 7 On/7 Off | 2,080 |
| Hourly | Tool crib Attendant | 7 On/7 Off | 2,080 |
| Hourly | Maint. Helper | 7 On/7 Off | 2,080 |

No allowance has been made for absenteeism, sickness, snow days, or dumped shifts. Holidays and vacation expenses are covered in the fringe benefit allowance.

**Figure 16.16: Mine Manpower Requirements**

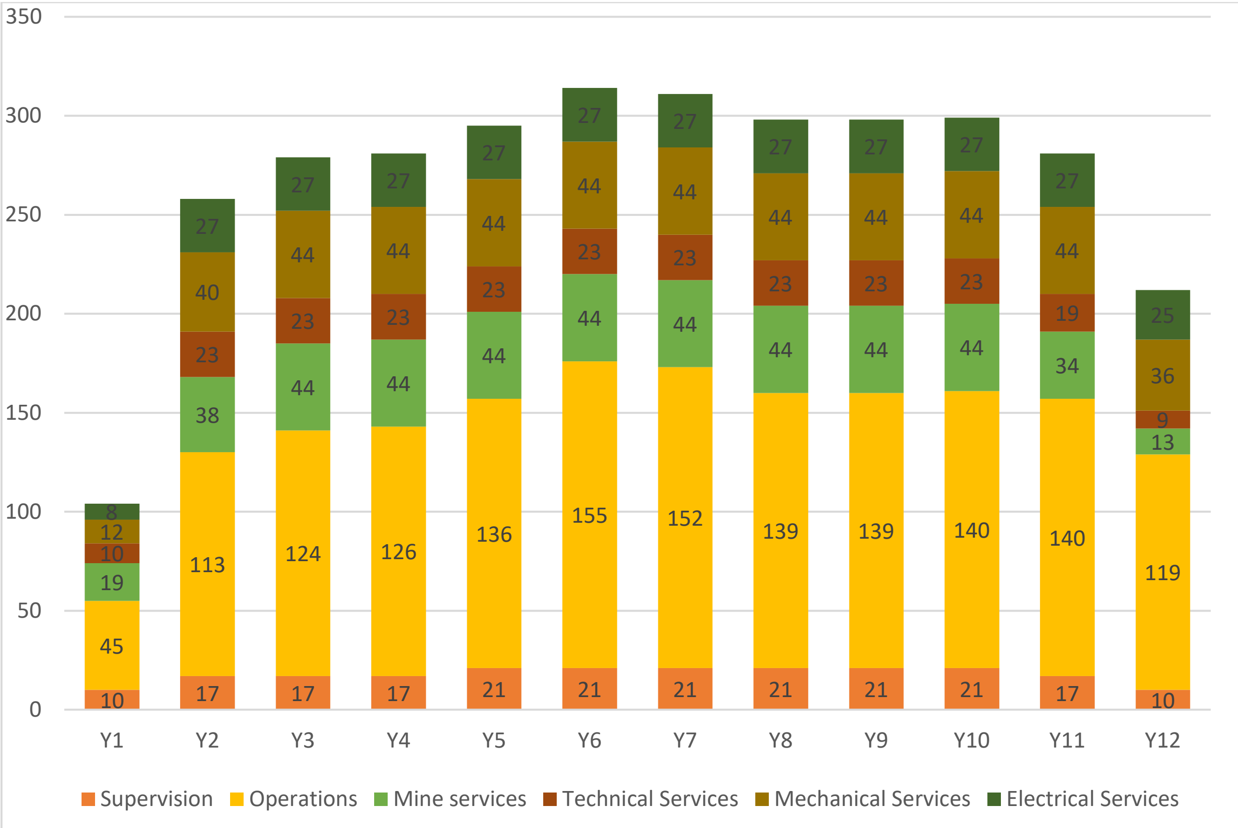



**Table 16.14: Mine LoM Manpower Requirements**

| Mine Manpower by Year | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mine Supervision** | | | | | | | | | | | | |
| Mine Manager | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Mine Ops. Superintendent | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Mine Secretary | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Mine Ops. Foreman | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |
| Mine Ops. Foreman | 8 | 12 | 12 | 12 | 16 | 16 | 16 | 16 | 16 | 16 | 12 | 8 |
| Mine Ops. Trainer | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| **Mine Operation** | | | | | | | | | | | | |
| Jumbo Operator | 1 | 2 | 4 | 4 | 4 | 16 | 20 | 20 | 20 | 20 | 20 | 20 |
| Continuous Miner Operator | 8 | 16 | 16 | 16 | 16 | 16 | 8 | 0 | 4 | 0 | 0 | 0 |
| Blaster | 16 | 32 | 32 | 32 | 32 | 40 | 40 | 40 | 40 | 40 | 40 | 26 |
| Bolter Operator | 8 | 36 | 44 | 44 | 44 | 48 | 48 | 52 | 48 | 52 | 52 | 48 |

| Mine Manpower by Year | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| LHD Operator | 4 | 19 | 25 | 27 | 25 | 26 | 25 | 27 | 27 | 28 | 28 | 25 |
| Truck Operator | 10 | 16 | 3 | 4 | 15 | 21 | 11 | 0 | 0 | 0 | 0 | 0 |
| **Mine Services** | | | | | | | | | | | | |
| Grader Operator | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Feeder Breaker Operator | 0 | 4 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 4 |
| U/G Constructions Maintenance | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 4 | 0 |
| Material Handling | 2 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 4 | 4 |
| Ventilation Crew | 0 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 | 2 |
| Conveyor Serviceman | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 0 |
| Labour - Lunch room, tool crib, etc. | 0 | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 |
| Lamps-Dry | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |
| Drill Bits Sharpener, Tool Crib, etc. | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| **Technical Services** | | | | | | | | | | | | |
| Chief Mine Engineer | 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 | 0 | 0 |
| Short-Term Planning Engineer | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 |
| Project Engineer | 0 | 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 |
| Mine Technician | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 | 0 |
| Geotech. Technician | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |
| Senior Surveyor | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Surveyor | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Chief Geologist | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Senior Geologist | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Geologist | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |
| Geology Technician | 1 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 1 |
| **Mechanical services** | | | | | | | | | | | | |
| Mine Maint. Superintendent | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Mine Maint. Foreman | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Mine Maint. Foreman | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| Mine Maint. Planner | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 0 |

| Mine Manpower by Year | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mechanical Engineer | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 0 |
| Mechanic | 7 | 26 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 |
| Mechanics - Fixed equipment | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 1 |
| Maint. Helper | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| **Electrical Services** | | | | | | | | | | | | |
| Mine Maint. Superintendent | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Mine Maint. Foreman | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| Mine Maint. Foreman | 0 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |
| Electrical Engineer | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Electrician | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Electricians | 5 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 |
| Electricians | 0 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 |
| **Total** | **106** | **266** | **279** | **282** | **295** | **326** | **311** | **298** | **298** | **299** | **281** | **212** |

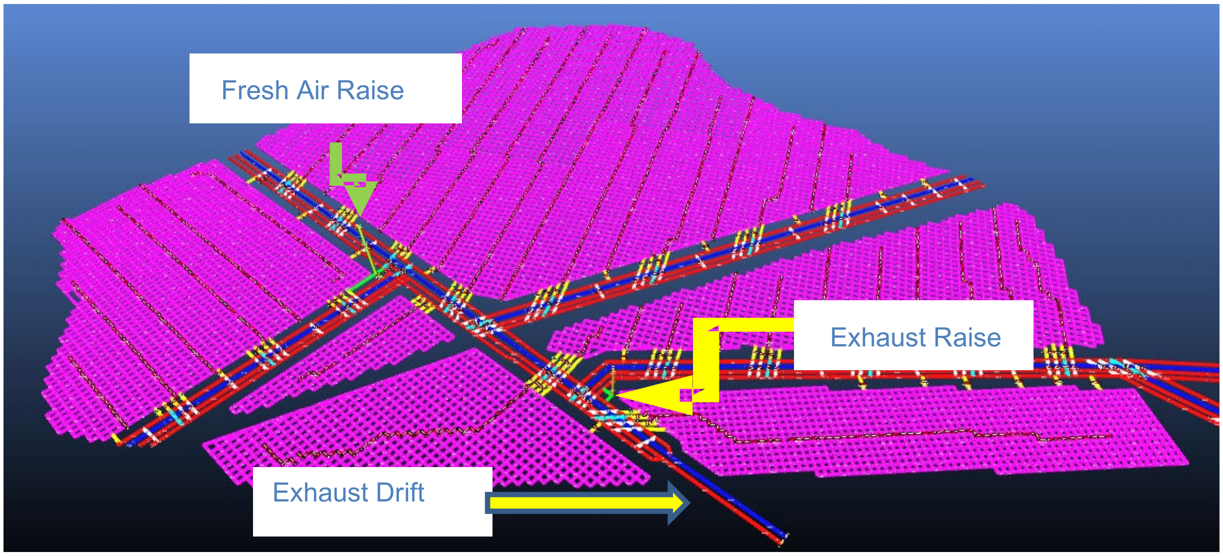
## 16.8 Mine Services

### 16.8.1 Ventilation

During the pre-production period, air requirement will be supplied through two (2) 300 HP 1.4 m diameter parallel Van axial surface fans. The two (2) fans will be installed on a metallic stand then connected with vent tubes directed to the portal. The two (2) fans in parallel will generate approximately 55 $m^3/s$ each at 2.5 kPa of water gauge. These two (2) 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 (2) other drifts.

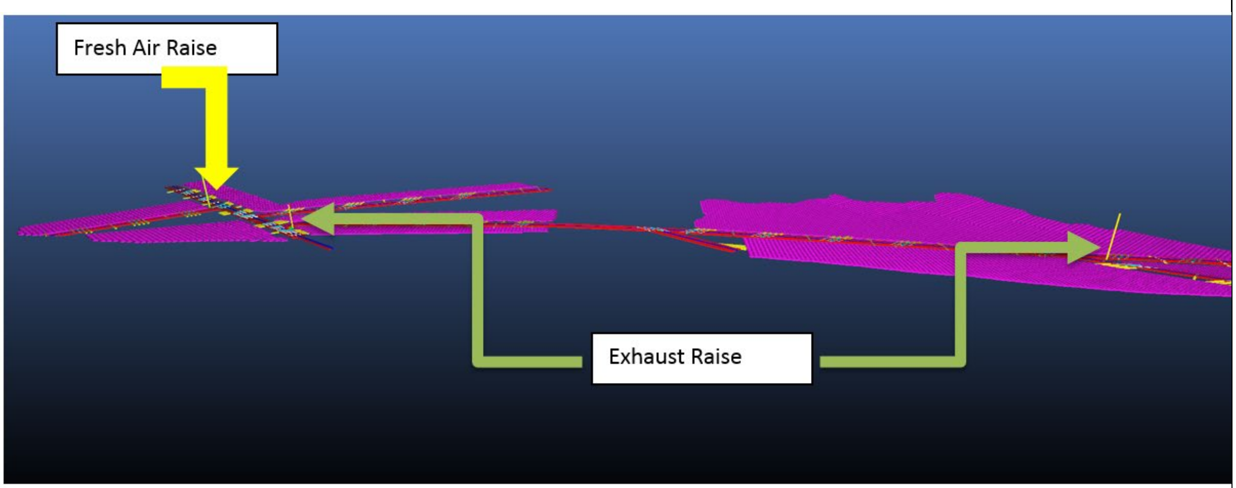
The ventilation system will consist of a push system whereby two (2) 1,250 HP 2.60 m diameter parallel main fans will be installed at surface providing approximately 200 $m^3/s$ each at 3.34 kPa. The two (2) main fans will be installed and provide heated air through a 5 m ventilation raise and air will 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 16.17: Ventilation Layout during Production Period (western side)**




| | | | |
|---|---|---|---|
| Mining Services | **7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2** | **Title: Ventilation Layout during Production Period (western side)** | **Author: C. Michaud,** P.Eng **Date: 15/06/2022** |

**Figure 16.18: Ventilation Layout during Production Period (eastern side)**




| | | | |
|---|---|---|---|
| Mining Services | **7900 W. Taschereau Blvd. Suite D-200 Brossard, Qc J4X 1C2** | **Title: Ventilation Layout during Production Period (eastern side)** | **Author: C. Michaud,** P.Eng **Date: 15/06/2022** |

### 16.8.2 Water Supply

Water is required underground for drilling, dust control and fire protection. Water will be distributed underground using a 6 in (15.2 cm) Schedule 40 steel pipe in 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% |
| Continuous Miner | 50 | 50% |
| 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

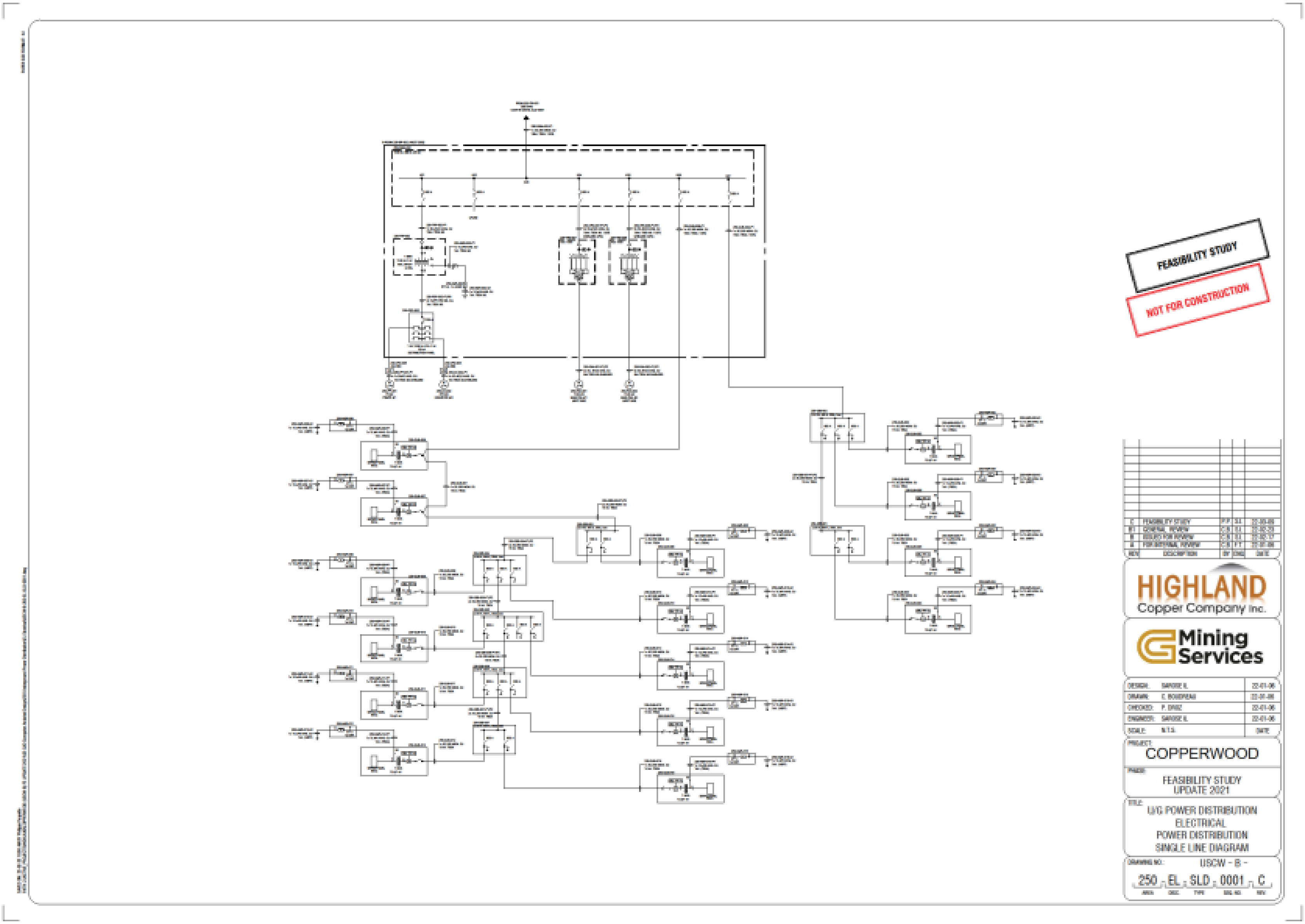
Major electrical power consumption in the mine will be required for the following equipment:

- 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 480 V for the electrical needs of the operation.

Figure 16.19 shows the power distribution single line diagram.

**Figure 16.19: Power Distribution Single Line Diagram**



#### 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’’. Water will be cleaned and sent to the event pond 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 (2) main pumping stations, P1 and P2, have 12.0 $m^3$ and 9.0 $m^3$ water tanks, equipped with agitators to prevent mud from settling at the bottom. The four (4) other pumping stations will have a 3.5 $m^3$ tank without agitators.

The Copperwood dewatering system consists of six (6) permanent pumping stations (Figure 16.20). 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 P6 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 16.20: Mine Dewatering Pumping Network**

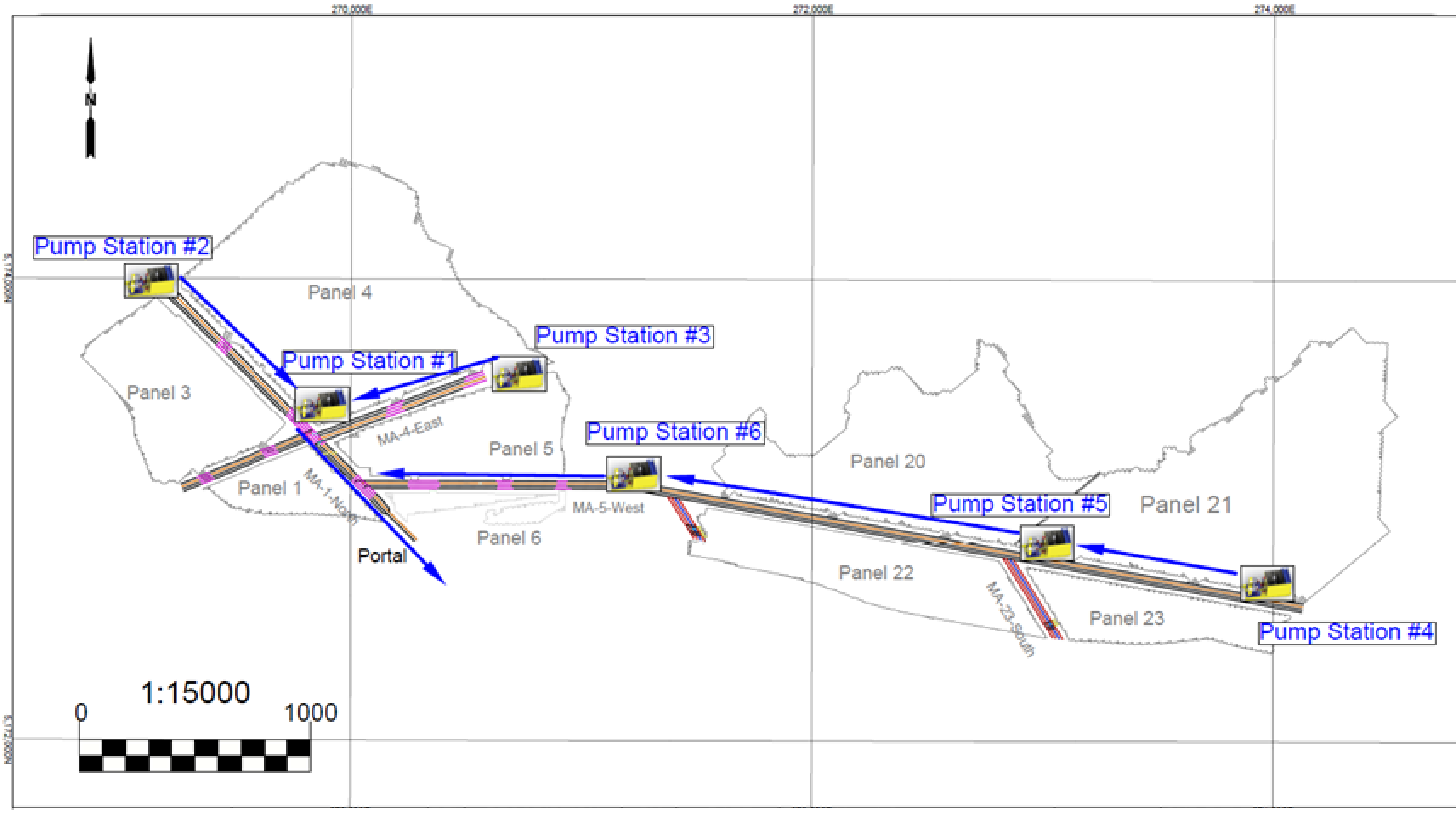
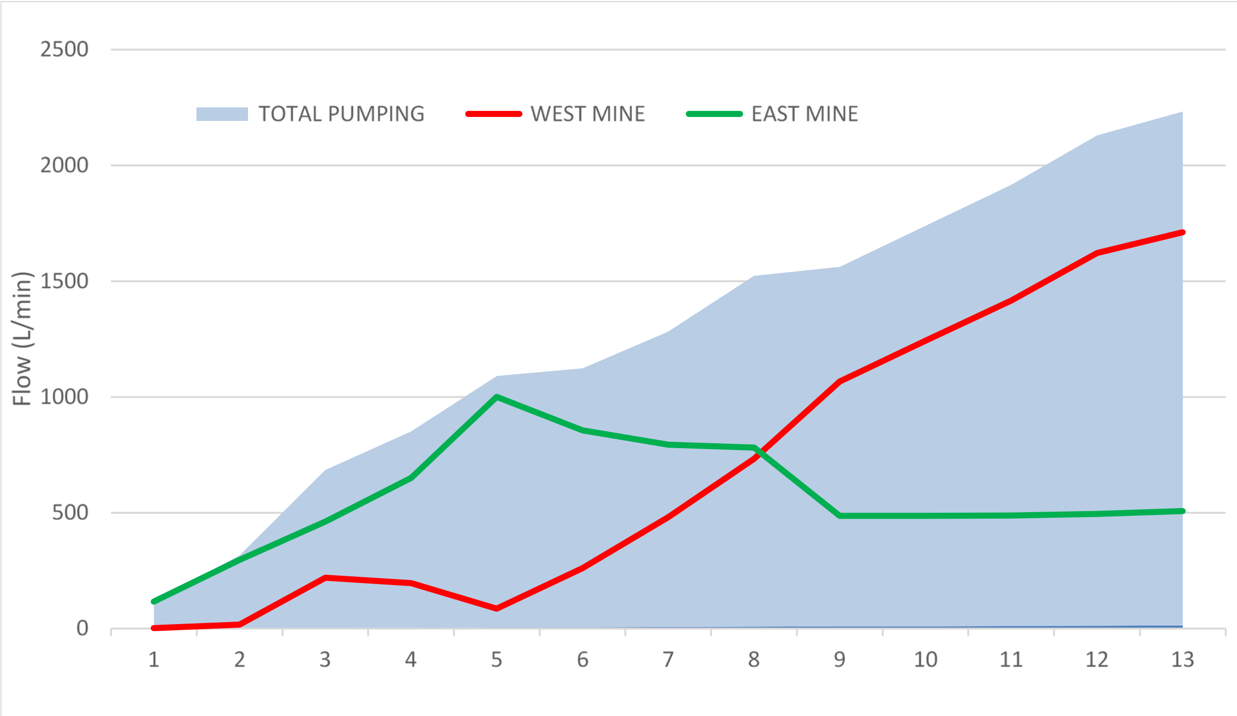


**Figure 16.21: Pumping Requirement over LoM in L/min Operating @ 100%**




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 (8) 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 small pumps for dewatering development work, to handheld drills, and 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 (2) 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, explosives will be stored in permanent magazines and accessories (detonators) will be stored in a separate magazine, both located at the surface. Once panel rooms become available, an underground explosive and detonator magazine will be installed. The Study includes a provision for two (2) underground explosives, one (1) 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. Emulsion 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 ATVs 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 (2) service LHDs 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 must, at least once every twelve (12) months or when changes occur, be instructed in the escape and evacuation plans and procedures and fire warnings signals in effect at the mine. Pursuant to *Regulation 57,4361*, mine evacuation drills shall be held every six (6) months for each shift. All exercises and instruction records will be kept for at least one (1) 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 (2) permanent and two (2) 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

The broken ore will be watered after blasting and during loading. Additionally, with continuous miners (roadheaders), systems are equipped to spray water on the rock to reduce the heat of the carbide tips and also to reduce dust.

# 17. RECOVERY METHODS

## 17.1 Introduction

This section is based to the previous technical report on the Copperwood Project made by Lycopodium and G Mining in June 2018. Comprehensive test work programs, described in the chapter 13, have been completed on samples from mineralized zones including repeatability of the results and provide a good understanding of the flotation circuit and the associated process plant design. The current process plant design is lean and fit to purpose.

No technical modification was made during this feasibility update therefore this section has largely been reproduced from the previous technical report.

## 17.2 Process Design

The process plant design for the Copperwood Project is based on a metallurgical flowsheet designed to produce copper concentrate. The flowsheet consists of equipment, process systems, and technologies widely used in the mining 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 optimize constructability of the processing facility.

The key project design criteria for the plant are:

- Nominal throughput of 300 metric tonne per hour (mtph) ore.
- Process plant availability of 91.3% is planned for the first two (2) years and 95% in the years after. This availability will be achieved during the third year of operation through good asset management, 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.

### 17.2.1 Selected Process Flowsheet

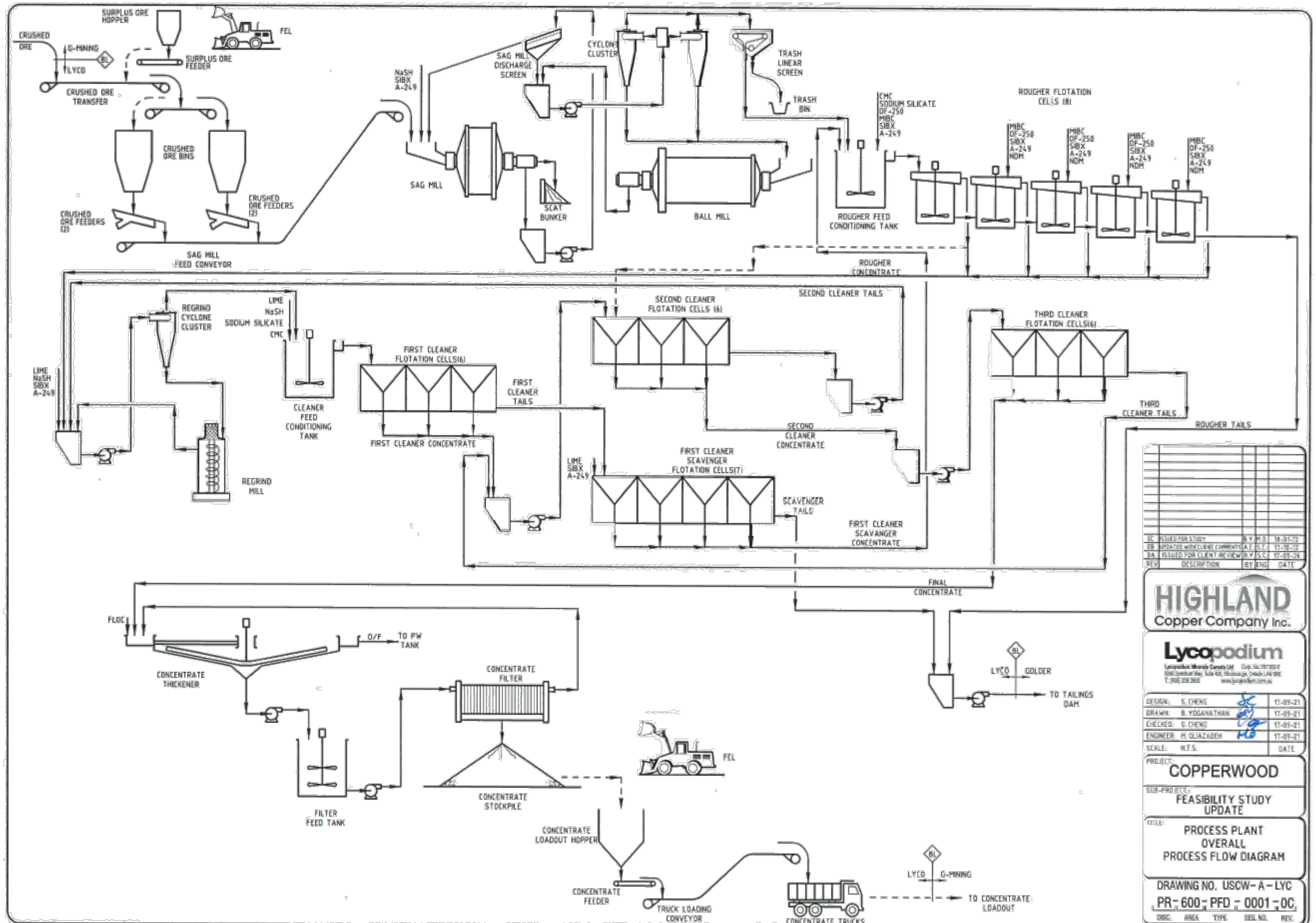
Design documents for this study have been prepared by incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical test works.

The process plant has been designed for a throughput of 300 mtph (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
- Services (air, water, reagents)
- 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**



### 17.2.2 Key Process Design Criteria

The key process design criteria listed in Table 17.1 form the basis of the detailed process design criteria and the process equipment. Process parameters selected are based on preliminary metallurgical test work carried out at SGS Lakefield ("SGS") dated August 28th, 2018, and earlier work conducted at Metcon 2011 with consideration of the design head grade. Ongoing optimization metallurgical test work 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 | mtph | 300 | Highland |
| Head Grade - LoM | % Cu | 1.35 | Highland |
| Head Grade - Design | % Cu | 2.2 | Highland |
| | g/t Ag | 3.41 | Highland |
| Bond Crusher Work Index (CWi) | kWh/t | 20.3 | Consultant |
| Bond Ball Mill Work Index (BWi) | kWh/t | 16.2 | Test work |
| SMC Axb[1] | | 34.5 | Consultant |
| Bond Abrasion Index (Ai) | g | 0.014 | Test work |
| Concentrator Feed Size ($F_{80}$) | mm | 150 | Test work |
| Grind Size ($P_{80}$) | µm | 45 | Test work |
| Rougher Residence Time - Laboratory | min | 50 | Test work |
| Cleaner 1 Residence Time - Laboratory | min | 6 | Test work |
| Cleaner 1 Scavenger Residence Time - Laboratory | min | 10 | Test work |
| Cleaner 2 Residence Time - Laboratory | min | 5 | Test work |
| Cleaner 3 Residence Time - Laboratory | min | 3 | Test work |
| Regrind Mill Product Size ($P_{80}$) | µm | 20 | Test work |
| Concentrate Production Rate | t/h | 15.1 | Calc |
| Target Concentrate Grade | % Cu | 24.7 | 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.*

#### 17.2.2.1 Comminution

Design parameters for the comminution circuit were sourced from test work conducted at various laboratories from 2010 and 2018. Orway Mineral Consultants carried out ore characterization and comminution modelling based on this test work.

Major observations and conclusions from the ore characterization were as follows:

- The comminution test work 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 test work for these zones has been analyzed separately but the grinding characteristics have revealed small difference from the CW zone parameters.
- The grinding characteristics of the three geologic subunits 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 for each of the Domino and Grey Laminated ore types indicates that Domino may be the softer ore in the CBS blend.
- The results for Section 5 and Section 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 Main 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 85$^{th}$ 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 Ai 0.014 and it can be considered as soft ore.
- The hardness of the ore and the electrical cost could make the HPGR circuit interesting with potential savings on the OPEX costs and an opportunity study has been realized. The annual production of Copperwood does not favor the selection of the HPGR and therefore it was not retained.

#### 17.2.2.2 Flotation Circuit

The flotation circuit configuration, residence times, reagent addition rates and concentrate mass recoveries have been selected based on the metallurgical test work conducted at SGS in 2018 and earlier work conducted at Metcon in 2011 with consideration of the design head grades.

## 17.3 General Process Description

The process plant has been designed for a throughput of 300 mtph (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

## 17.4 Crushed Ore Reclaim

Crushed ore from the underground mine will be conveyed to a crushed ore transfer conveyor equipped with a weight scale. 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 onto the SAG mill feed conveyor. This conveyor will also be equipped with a weight scale for measuring and controlling the SAG mill feed rate.

A surplus ore feeding system, comprised of a mobile hopper / feeder and conveyor, will allow ore material to be fed to the crushed ore bin via a front-end loader from the ore stockpile when required.

## 17.5 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 operating in a closed circuit with a screen and a ball mill operating in a 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 a 12% to 15% ball charge. Ore will be fed to the SAG mill at a controlled rate, nominally 300 dry mtph, and water will be 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 9.00 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 flash flotation. Space has been identified in the building for future installation of a flash flotation circuit if required.

## 17.6 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 $m^3$ 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.

Space has been allocated in the area to allow the rougher concentrate from the last three rougher flotation cells to be collected in a pump box and pumped to the second cell of the rougher flotation circuit in the future, if required. Also, space has been considered for future expansion or circuit reconfiguration (e.g., recycling cleaning concentrate or tailings) following the optimization flotation test work results.

## 17.7 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 pH modifier (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.

## 17.8 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.

Regrind cyclone overflow will gravitate to the cleaner conditioner tank. NaSH, pH modifier 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 $m^3$ 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 $m^3$ trough cells in series. A pH modifier, an A-249 and an 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 (6) 8 $m^3$ trough cells in series. A pH modifier and an 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 (6) 2 $m^3$ 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.

## 17.9 Concentrate Thickening and Filtration

Final concentrate at 15.1 mtph 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 $m^3$/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 by filter feed pumps (1 operating, 1 standby). The filter for 35 mtph (235 $m^2$ area) 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 pump. Approximately 9.9 $m^3$/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 weight scale.

## 17.10 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 Disposal Facility (“TDF”).

## 17.11 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 detailed 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 use in the grinding, flotation and thickener areas via the process water pump.

## 17.12 Reagents

### 17.12.1 Frother (MIBC/D-250)

An MIBC and a 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. An MIBC and a D-250 will be dosed neat, without dilution in a 1:1 weight ratio. An MIBC and a 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.

### 17.12.2 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.

### 17.12.3 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. An 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.

### 17.12.4 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.

### 17.12.5 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.

### 17.12.6 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.

### 17.12.7 Aeroflot 249 (A-249)

An 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. The 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.

### 17.12.8 pH modifier (Hydrated Lime)

Flotation performs at natural pH around 8. Thus, the addition of a pH modifier will be required if the process water becomes too acidic. Given this low addition when needed, a hydrated lime system was selected as a means of mitigation.

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.

### 17.12.9 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.

## 17.13 Services and Utilities

### 17.13.1 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.

#### 17.13.2 High and Low-pressure Air

High pressure air at 700 kPa (g) will be provided by two (2) high pressure air compressors to supply 1,000 $Nm^3/h$ each, 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 (2) low pressure blowers at 12,600 $Am^3/h$. Cleaner flotation air will be supplied by two (2) low pressure blowers at 7,600 $Am^3/h$.

# 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 519N)
- Site access roads
- Parking lot
- Plant workshop & Stores
- Reagents storage
- Explosives storage
- Workshop, wash bay and warehouse
- Mine dry
- Mill offices and metallurgical laboratory
- Gatehouse
- Concentrate transload facility
- Administration office
- Assay laboratory
- Fuel storage
- High voltage power line and main substation
- Emergency Site power generation
- Site electrical distribution
- Process Plant Electrical Room
- Underground main Electrical Room at portals
- Other Electrical rooms (Ventilation intake, Tailings, WTP)
- Site Communications network for above ground installation and underground mine
- Potable water treatment

- Surface Water Collection Berm
- Reclaim Water System
- Stream relocations
- Tailings disposal facility in three stages
- Water treatment plant
- Fire water system
- Sewage treatment
- Site Vehicles and Mobile Equipment
- Ore stockpile pad
- Covered box-cut for mine access
- Compressors for underground

## 18.2 Public Access Road

The Project is accessed via the existing County Road 519 North ("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 the Gogebic County Road Commission ("GCRC"), the road has seasonal limits on truck weight during spring thaw conditions (around the end of April). Highland Copper Company Inc. will work with the GCRC to upgrade CR 519 to better handle the increase in heavy 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"). The GCRC will be responsible to supervise and manage the engineering and the design of the road, and the execution of the construction works during road improvement.

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**

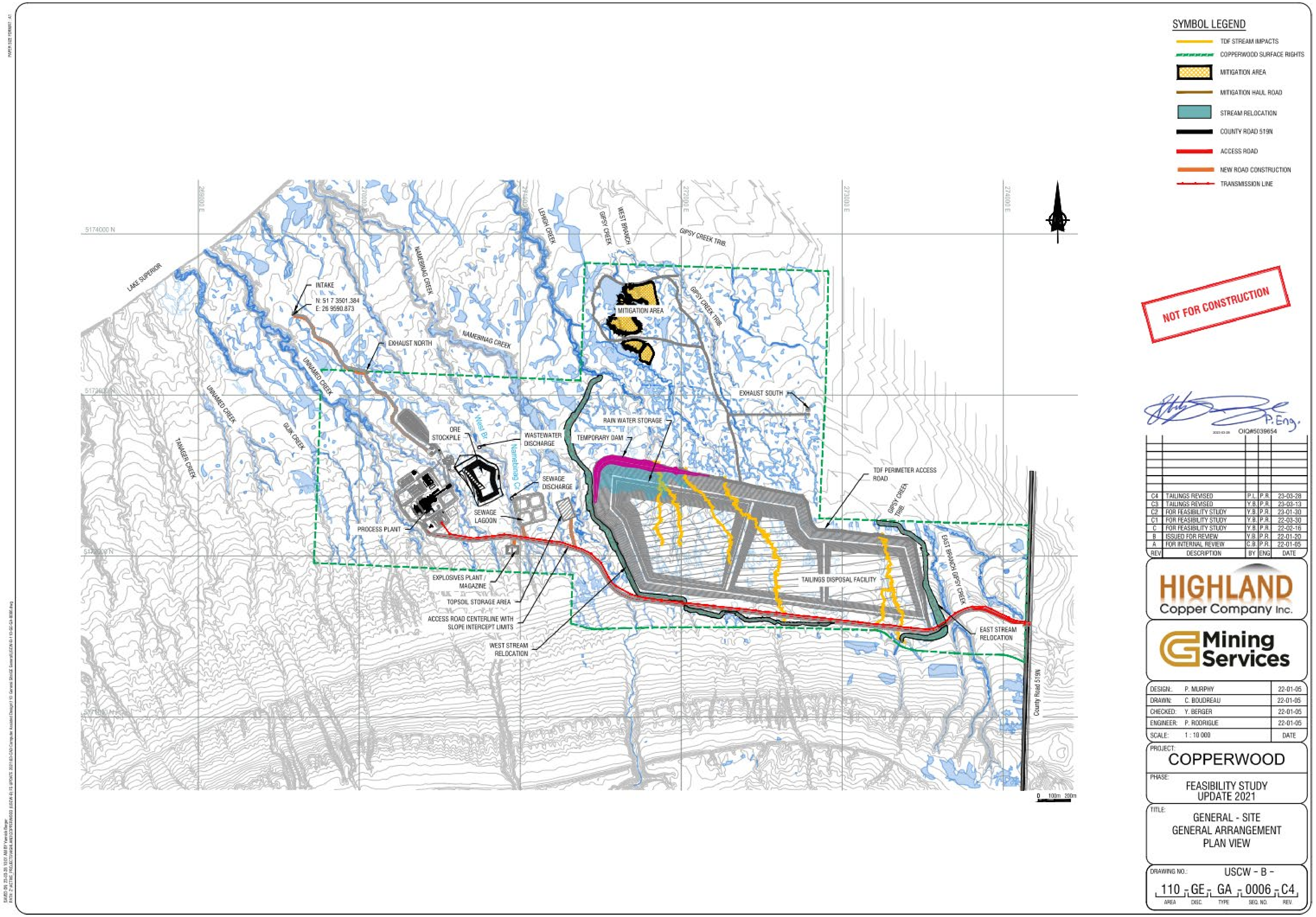
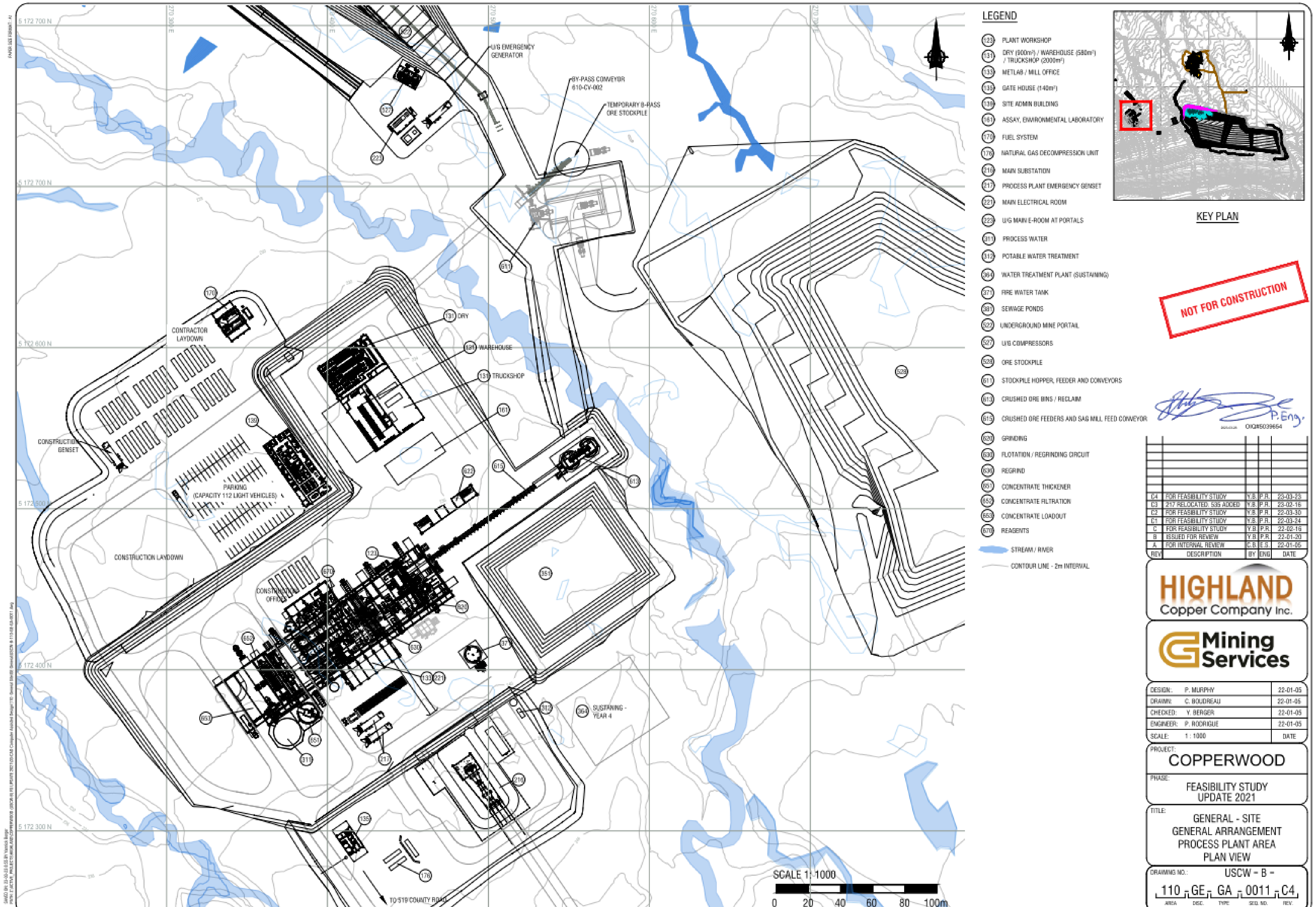


**Figure 18.2: Copperwood Project Plant Site Area General Arrangement**



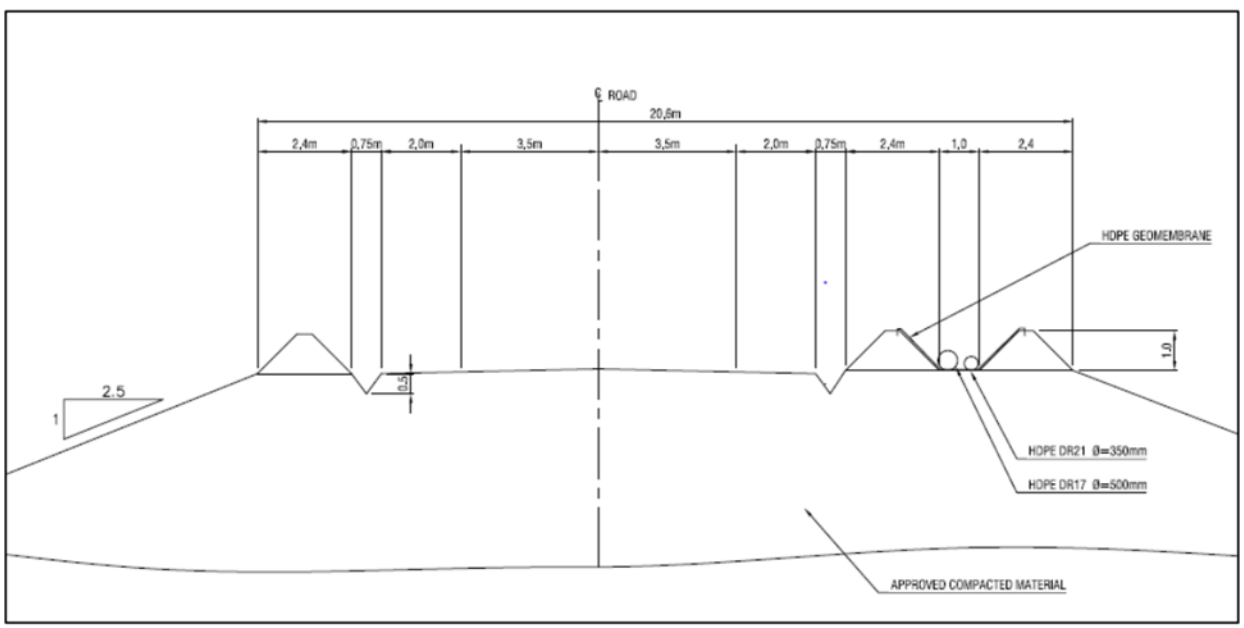
## 18.3 Site Roads

### 18.3.1 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 up to the processing area. The distance between the site entrance and the mill site gate house is approximately 4.1 km. 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**



### 18.3.2 Site Roads

Site roads will allow movement of vehicles and personnel from the various support buildings and infrastructure located away from the process plant area and the main access road. The site roads will use existing trails as much as possible to reduce clearing and negatively impacting wetlands. The existing trails 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 explosive magazine access road and the road between the box cut and the site. The total length of the newly constructed site roads is approximately 1.2 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 one side ditch. The typical ROW for site roads is approximately 11 m.

## 18.4 Parking Lot

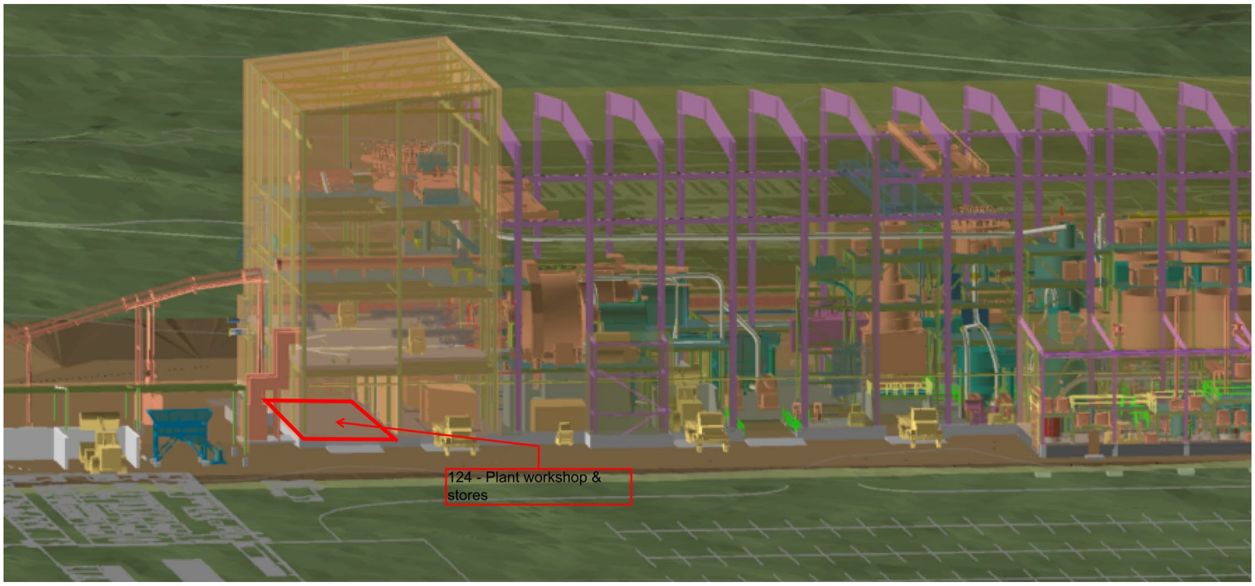
An employee parking lot is planned on the north-west side of the process plant. This parking will have approximately 112 light-vehicle parking spots covering an area of 4,900 square metres. The parking will be built with tuff material excavated from the box cut over a geotextile fabric to improve drainage. No other granular material is expected to be used.

## 18.5 Workshop / Storage

### 18.5.1 Plant Workshop & Stores

The Plant’s workshop & stores will be located under the grinding operating floor in the northern corner of the process plant building. The floor space is expected to be 225 square metres. A hoist and an allowance for mechanical equipment is included in the feasibility.

**Figure 18.4 Plant Workshop & Stores Location**



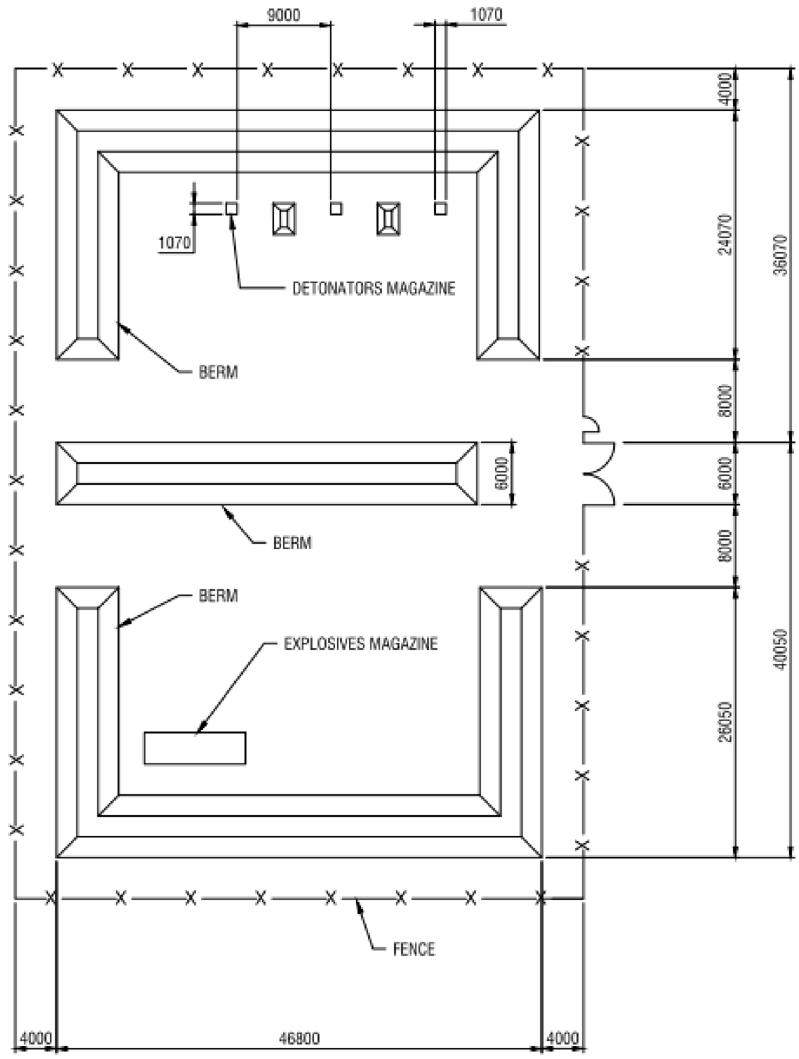

### 18.5.2 Reagent Storage

Reagents required to support the processing operations will be stored under the same roof as the reagent mixing and dosing equipment. The reagents building will provide an area of approximately 160 square metres for storage. An additional containerized storage zone will be available at the construction laydown area once the project's construction is completed.

### 18.5.3 Explosive Magazine

The explosive magazine will be located on the south side of the main access road. The dimensions of the explosive magazine are 76 m x 55 m. The design includes protective berms that will ease the traffic in and out of the storage facilities. The explosive material will be stored in a container designed to satisfy safety requirements and will provide a week's worth of explosives storage.

**Figure 18.5 Surface Explosive Magazine Plan View**



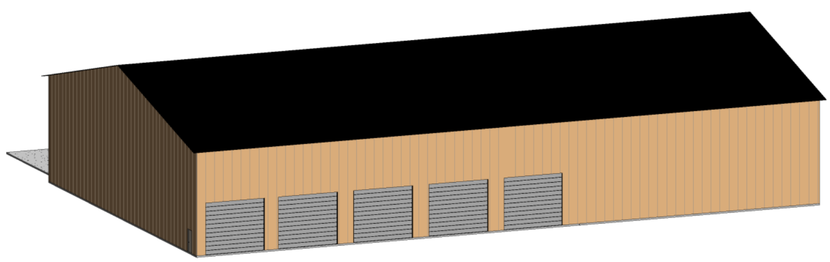

## 18.6 Support Infrastructure

### 18.6.1 Workshop & Warehouse

The workshop and warehouse are in the same pre-engineered steel structure and located in the northwest part of the process plant area. This building also includes the mine dry, described in the next sub-section. The building will be 62.5 m long x 36 m wide x 10 m at low eaves. The workshop will be used primarily to support the maintenance of heavy-duty vehicles and has a floor area of 38.1 m long x 18 m wide. The workshop will have five (5) separate bays which will each be equipped with a 6 m wide by 5 m high roll-up door. A 10-tonne overhead crane will be included and will be available for four (4) bays. The workshop will be equipped with a lube distribution system, including ancillary equipment. The following fluid will be stored in the lube containers, and dispensed to their corresponding location: engine oil, hydraulic / transmission oil, differential / final drive oil, coolant and grease. One (1) bay will be used for washing purposes and the remaining bays may be used for welding or general maintenance. Water used to clean vehicles in the workshop is to be considered as contact water and will be collected and sent to the event pond.

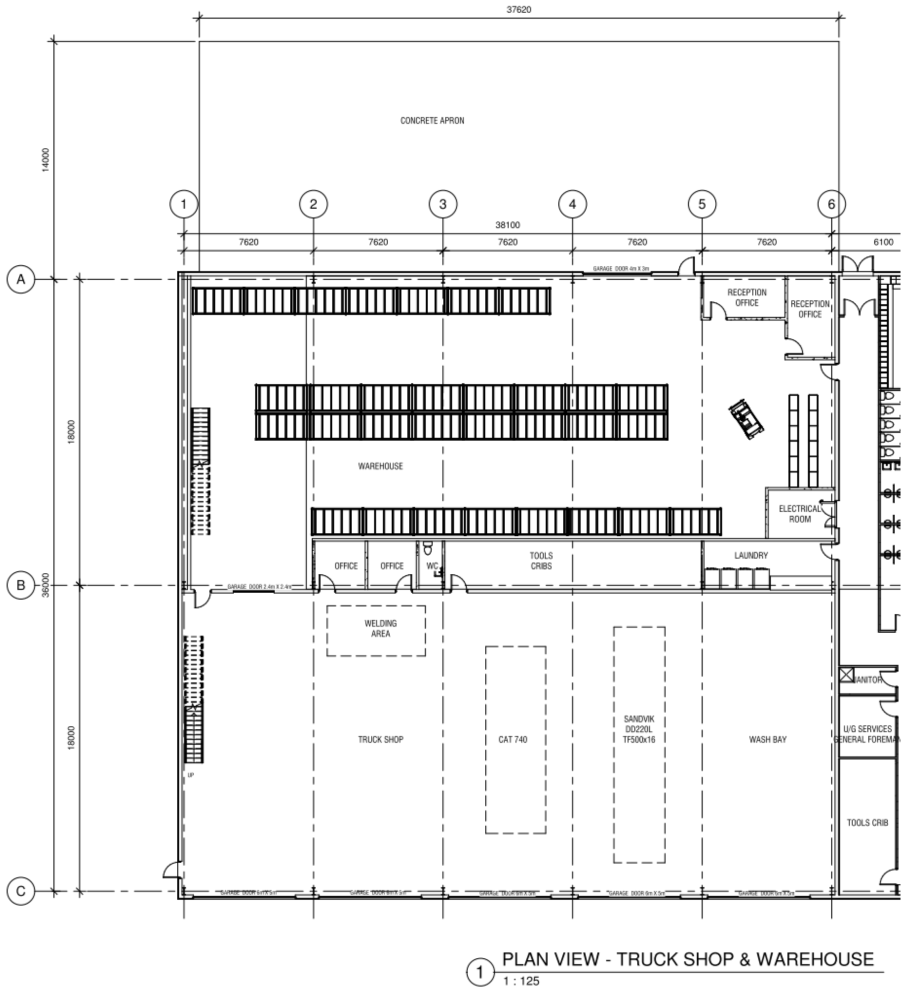
The warehouse will have racking to store spare parts and consumables. The warehouse's interior dimensions will be 38.1 m x 18 m. The warehouse and truck shop will both have concrete aprons to better handle heavy vehicle traffic.

**Figure 18.6: Maintenance Shop and Warehouse**



2 OUTSIDE 3D VIEW

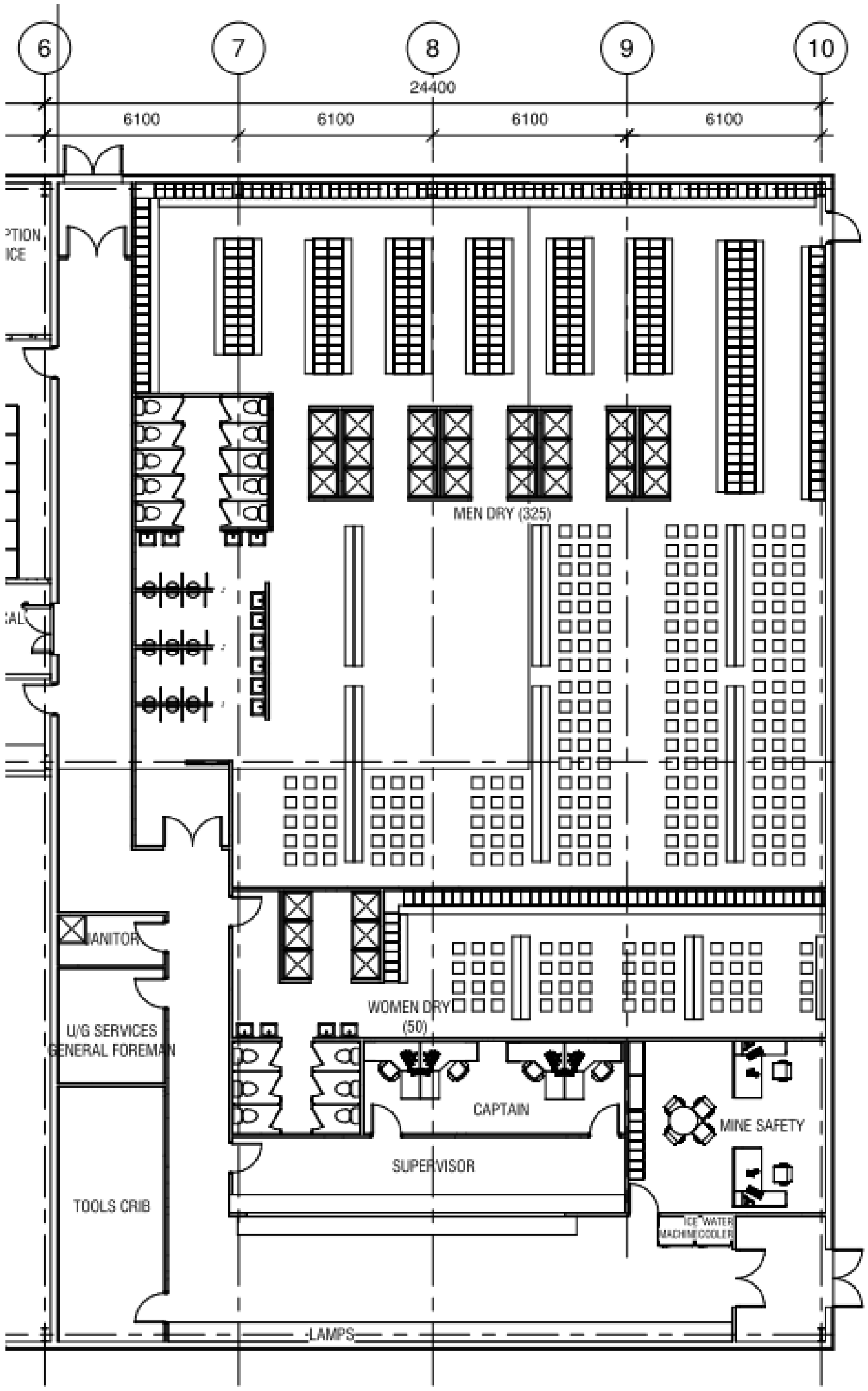
**Figure 18.7: Truck Shop and Warehouse Plan View**




## 18.6.2 Mine Dry

The mine dry will be adjacent to the truck shop and warehouse. The dry will serve as the locker room for 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 an extension of the truck shop and warehouse pre-engineered steel-clad building.

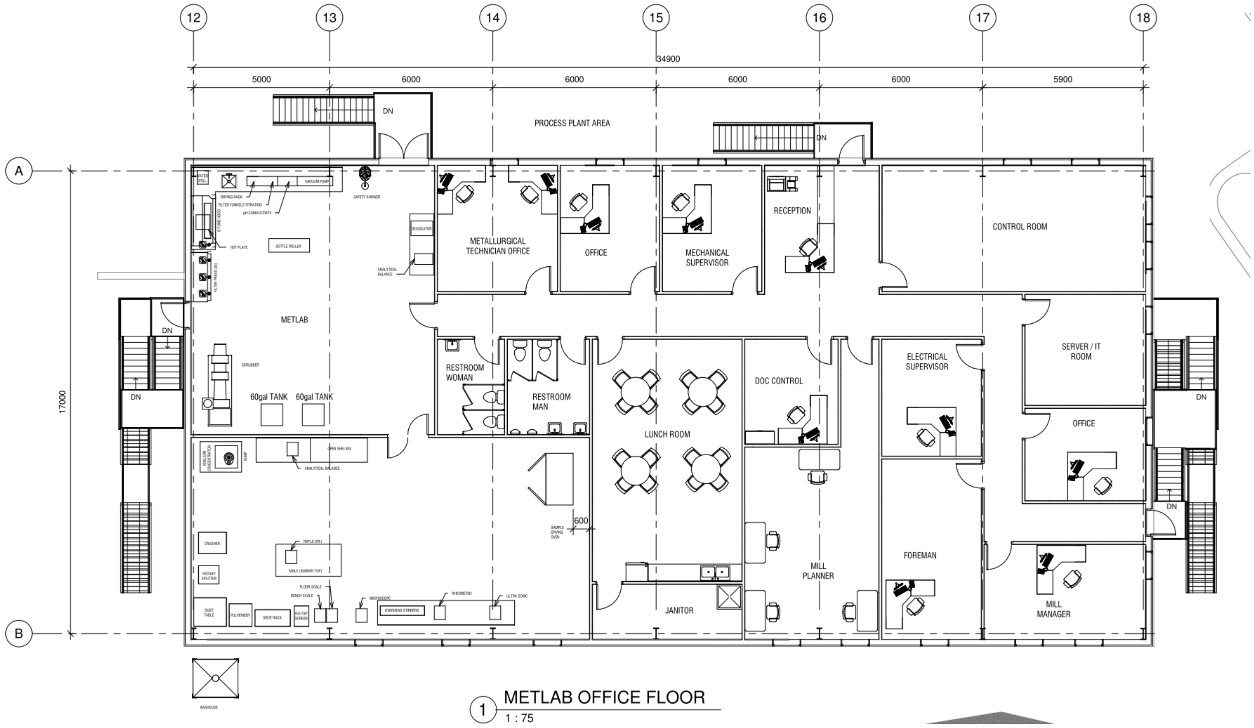
**Figure 18.8: Plan View – Dry**




### 18.6.3 <u>Met Lab and Mill Offices</u>

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. It will be located on the second floor of the Process plant electrical room. The control room will be installed on the same floor as the mill offices and will have a view of the processing equipment. A lunchroom for the office and lab worker is included in this building.

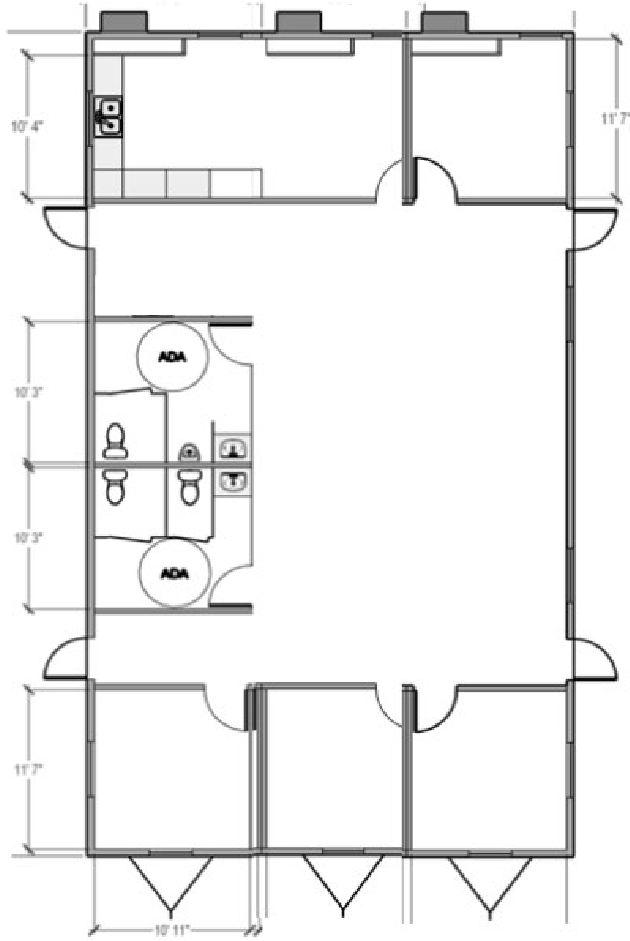
**Figure 18.9 Met Lab and Mill Office Plan View**




### 18.6.4 <u>Gate House</u>

Access to site will be controlled at the gatehouse located adjacent to the main access road, south of the process area. All traffic coming to or leaving the process area will be controlled at 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 square metres. Vehicle access will be controlled by a boom gate. The gate house will be a prefabricated, modular building.

**Figure 18.10: Gatehouse Plan View**




### 18.6.5 Off-Site Transload Facility

The transload facility will be located at a rail siding in Champion, Michigan approximately 180 km from site. Following guidance from the rail transportation study from Concept Consulting LLC (2017), the location has been chosen due to the costs and mainly because it provides access to the Canadian National Railway (CN) networks. The operator of the short line between Champion and Ishpeming is Mineral Range Railroad. Rail cars will be loaded at the transload facility, then moved from Champion to Ishpeming. From there, the rail cars will be moved by locomotive owned and operated by the CN.

Copper concentrate shipments will arrive at the transload facility via over-the-road trucks. These side-dump haul trucks will drive into the transload facility building and dump the concentrate on a concrete slab, before exiting the building. Copper concentrate will then be loaded into rail cars with a front-end loader. The storage building will be fully enclosed to ensure the control of air quality with sufficient air changes, and to control water content. Roll-up style doors will be installed to allow equipment to move in and out of the concentrate storage building and to regulate airflow through the building. Each haul truck carries a concentrate payload of around 46 metric tonnes, a weight that is limited by the Michigan DOT to a maximum gross vehicle weight of 151,400 lb (11-axles truck configuration).

**Figure 18.11: Transload Building Cross-Section**

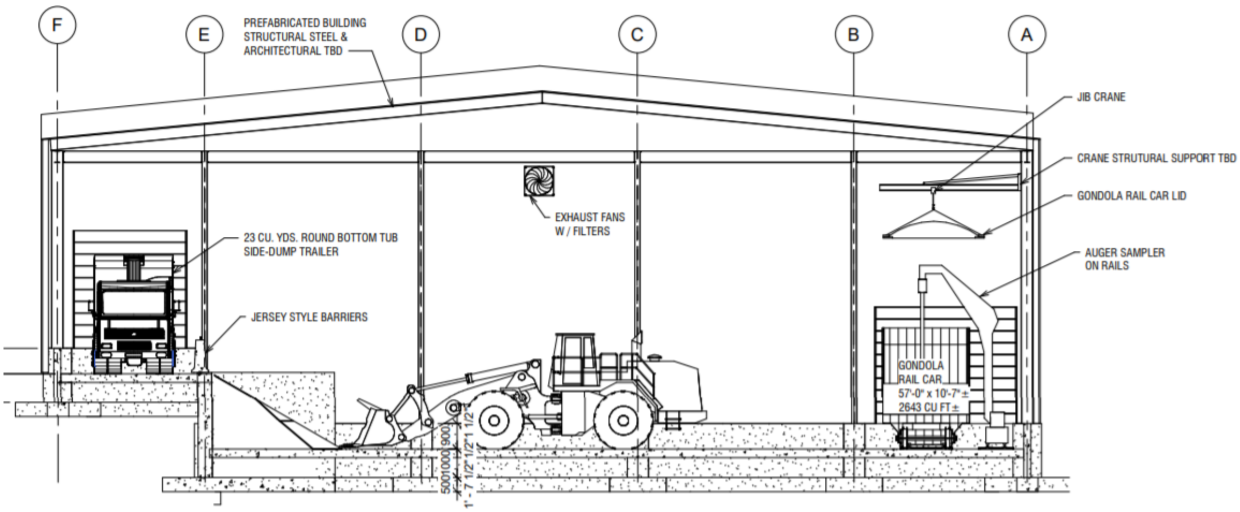


**Figure 18.12: Transload Building Plan View**

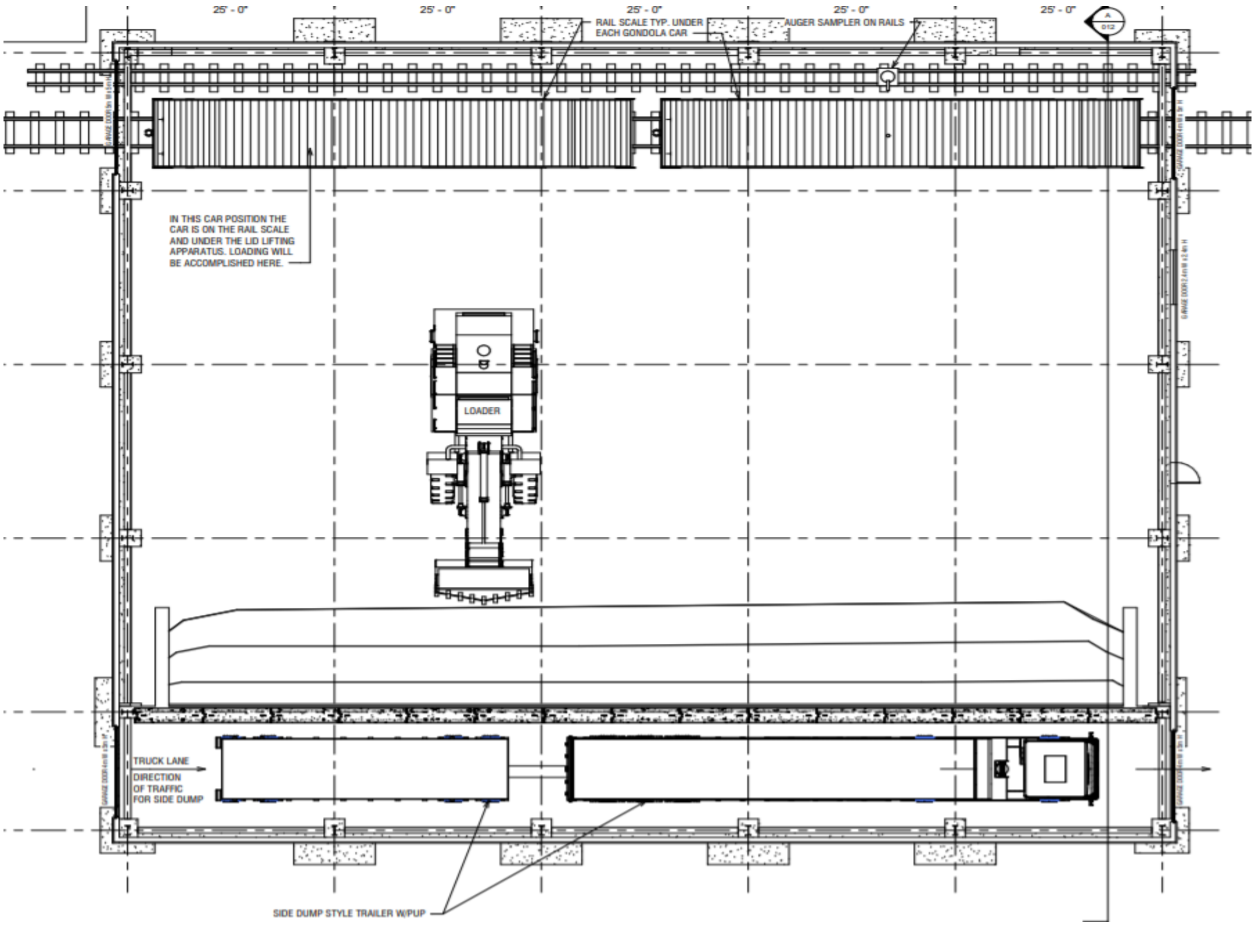
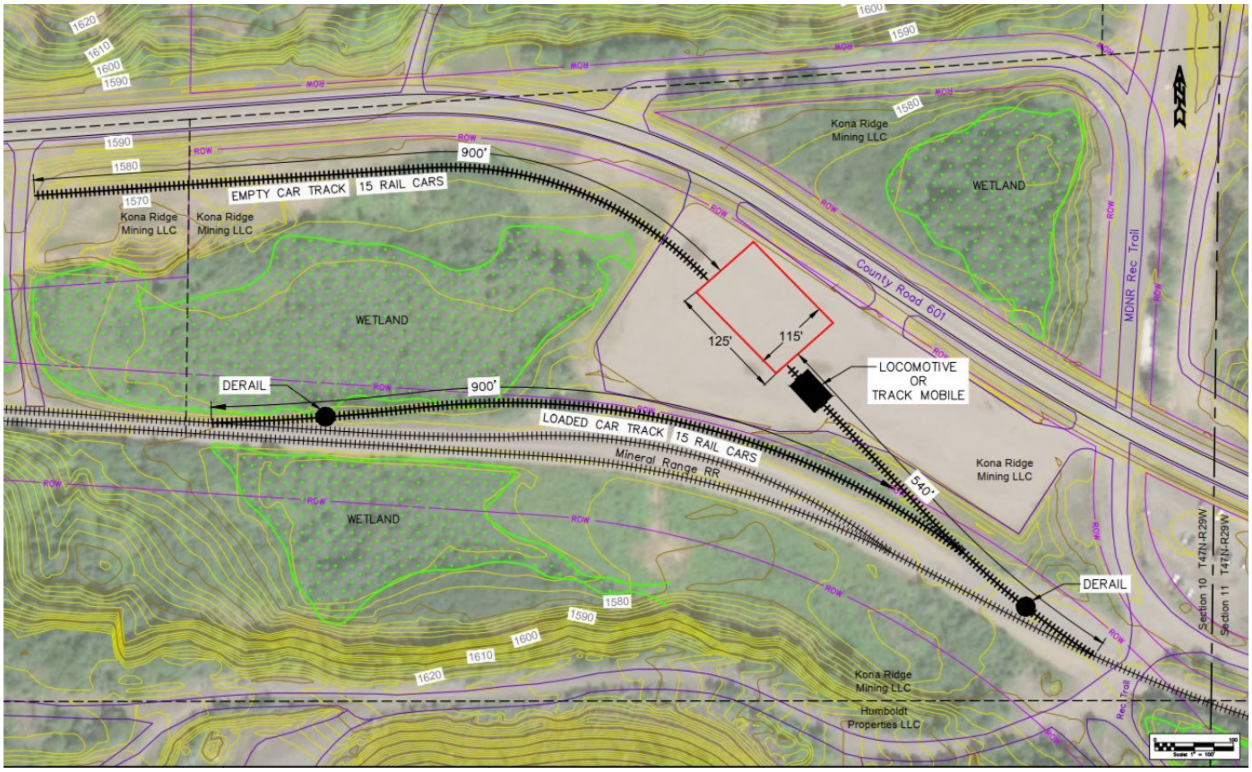


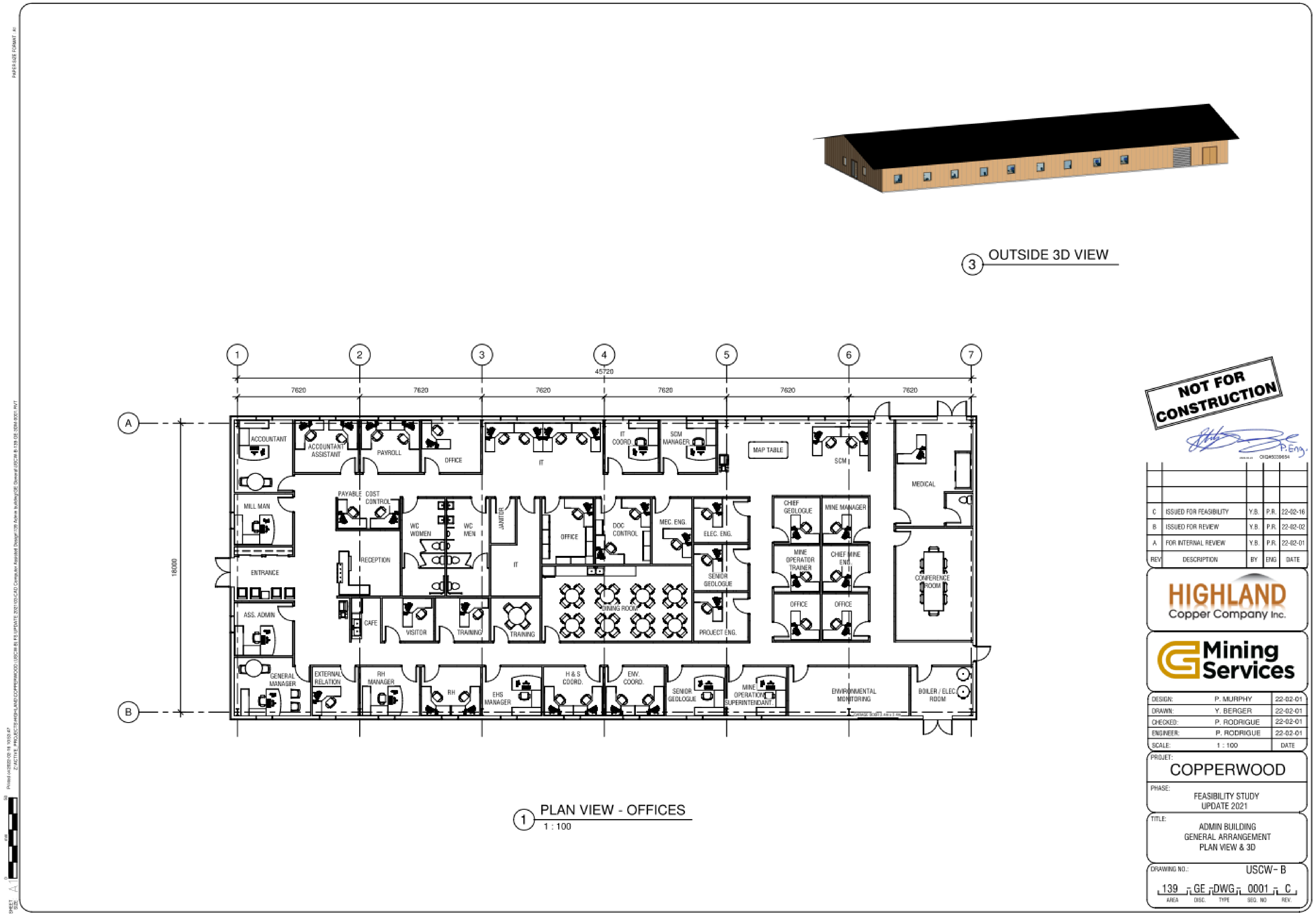
**Figure 18.13: Champion, MI Transload Facility Key Plan**




### 18.6.6 Administration Building

The administration building will be located on site on the north-west side of the process plant, between the parking lot and the warehouse/Workshop. The dimensions of this building are 46 m long x 18 m wide. It is a structural steel stick-built building. The building includes a medical center, a conference room, a dining room for 32 people and office spaces for approximately 38 people, including staff from the Mine, Accounting, Human Resources, Environmental, and Health and Safety departments.

**Figure 18.14: Administration Building General Arrangement**



## 18.7 Assay & Environmental Laboratory

The assay lab will be located on site on the south-east side of the workshop / warehouse building. It will be managed by a third-party, who will setup their own prefabricated modular building with all the necessary equipment. The modular building will be sitting on concrete blocks.

## 18.8 Diesel Fuel Storage

A fuel storage facility for mining, mine support equipment and light vehicles will be built. The fuel storage is located south of the mine entrance. The facility will include two (2) 10,000 l self-contained tanks for diesel with unloading and two distribution pumps. Equipment will be installed on concrete bases. Fuel will be filtered to remove water content and particle at delivery, and it will be filtered again to remove particles when pumped to vehicles. A concrete containment and two (2) aprons for the fuel stations, one (1) for the mining fleet and one for light vehicles as well as for the fuel unloading. An oil-water separator pit will be included in the design to separate any spill from the tanks or during the operations. A containerized electrical room will be required.

**Figure 18.15: Fuel Systems & Storage Plan View**

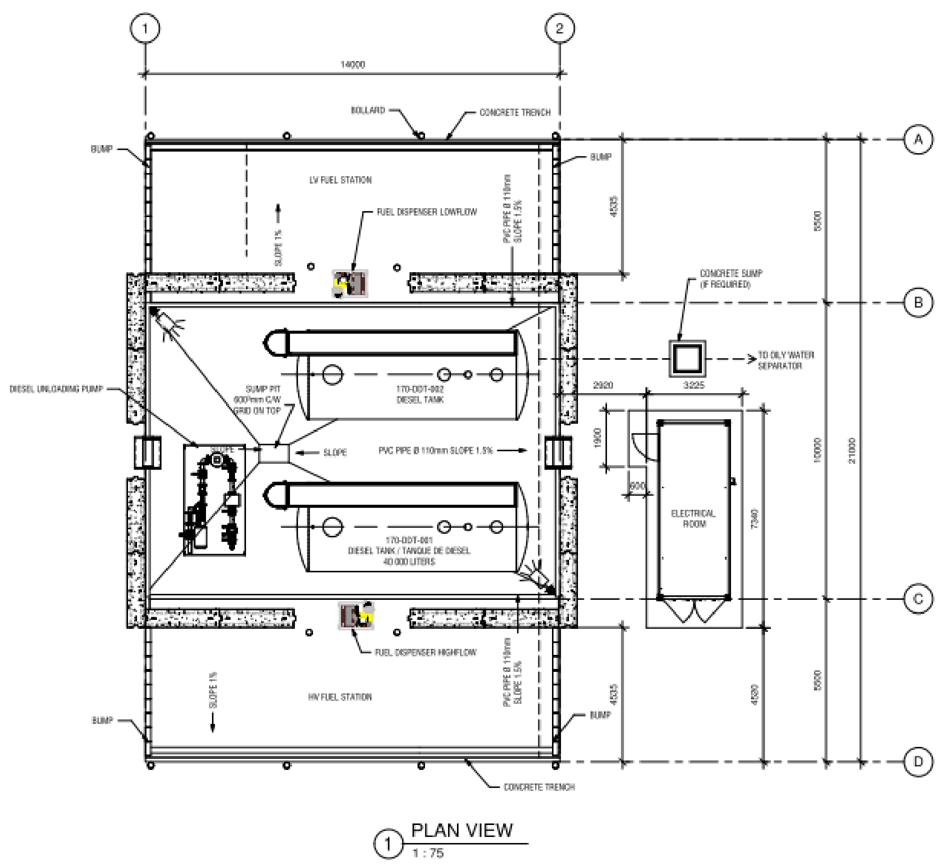
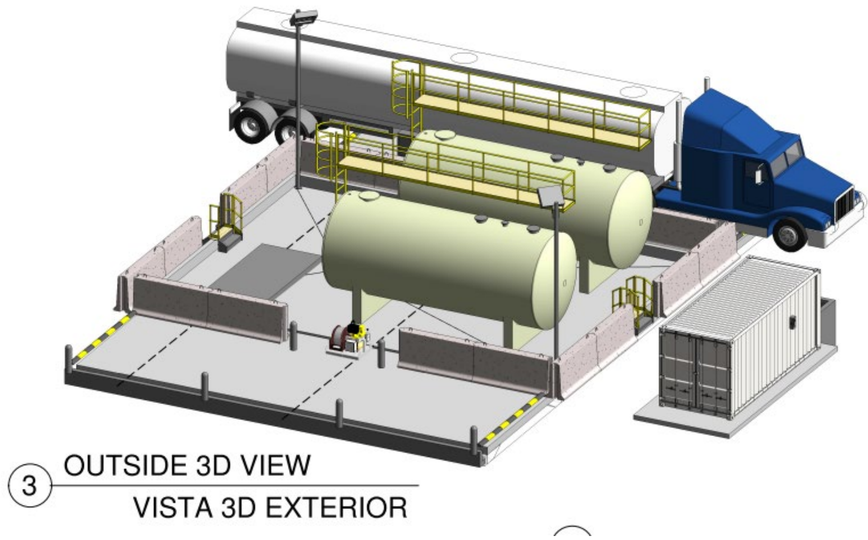


**Figure 18.16: Fuel System & Storage 3D View**



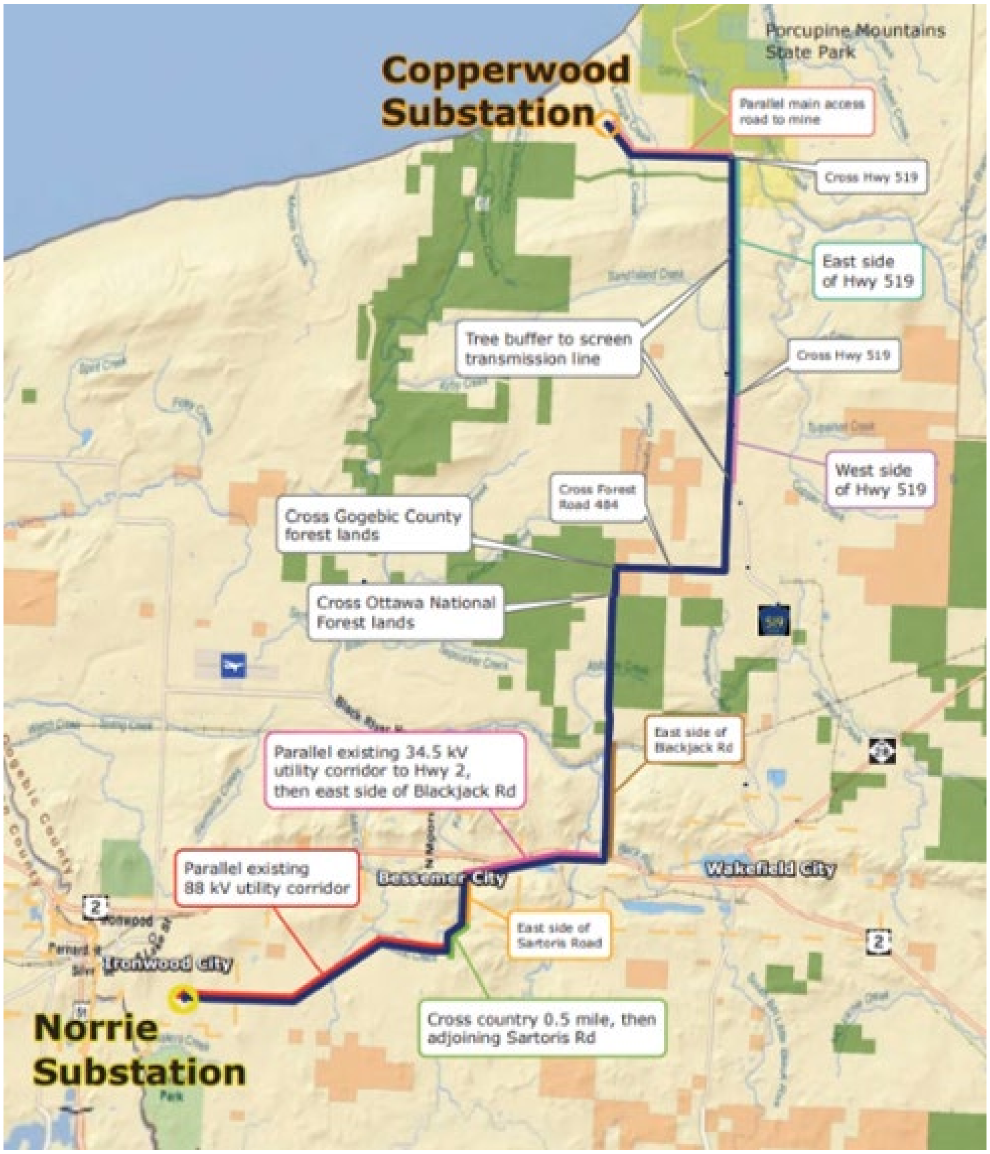

## 18.9 Power and Electrical

### 18.9.1 High Voltage Power Line and Main Substation

The feasibility study in 2018 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 40 km long 115 kV line will tie at the existing Norrie substation in Ironwood. Both transmission line and site main substation will be designed, supplied, built, owned and operated by the Utility company. Power cost rate will be factored to covers these costs.

The powerline proposed route is shown below:




The main substation and the power line design was based on the following loads:

| Load | MW |
|---|---|
| Installed | 42.0 |
| Peak | 24.9 |
| Running (mean) | 20.4 |

From the main substation, the electrical distribution will be at 13.8 kV to supply the process plant and site buildings.

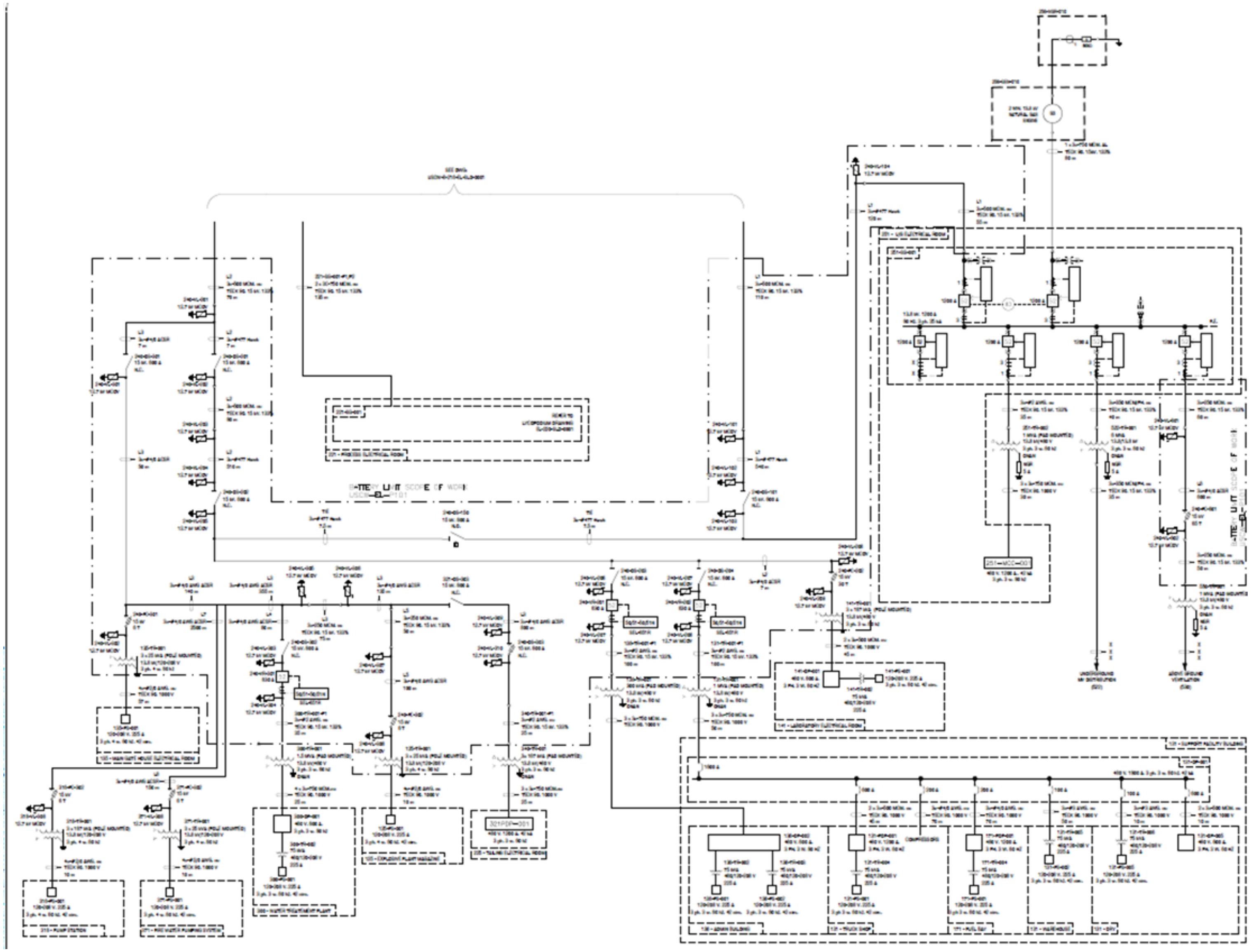
Power available from the grid is limited at 21 MVA for the first five (5) years of operation. Additional power production will be supplier by the site natural gas emergency generators to shave the peaks on grid. Generator will be used mainly during winter season.

### 18.9.2 Site Electrical Distribution

The electrical distribution on site will be done by a mix of overhead lines and underground cables for an approximate total length of 6 km. Two (2) redundant distribution lines will energize the portal underground mine area and ventilation air intake.

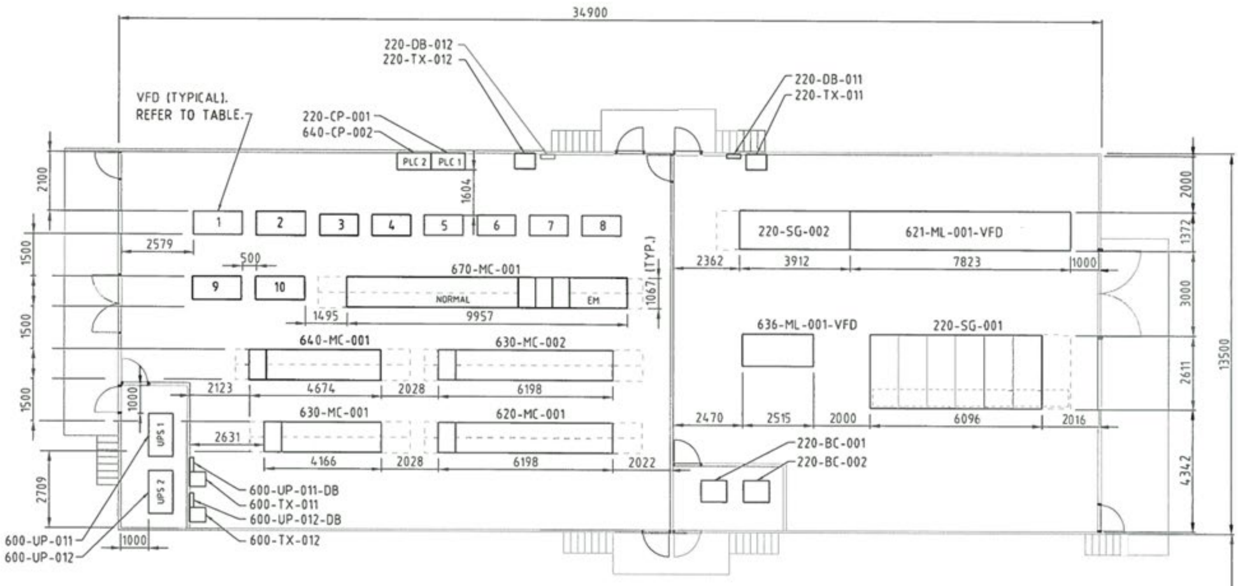
:

Aerial distribution will also provide power to site infrastructures:



### 18.9.3 Process Plant Electrical Room

The process plant electrical room is designed into two sections to house medium voltage equipment (13.8 kV, 6.9 kV and 4.16 kV) and low voltage equipment (480 V and 120/240 V). Dedicated rooms will be built to store batteries and UPS.




Main equipment included in the MV side:

| Equipment Number | Description |
|---|---|
| 220-SG-001 | 13.8 kV MV Switchgear. Incoming from main substation and 13.8 kV distribution for process plant. |
| 220-SG-002 | 6.9 kV synchronous switchgear for ball and sag mill starting and operation. |
| 621-ML-001VFD | Main 6.9 kV variable frequency drive for ball mill starting and sag mill starting and operation. |
| 636-ML-001-VFD | Regrind 4.16 kV variable frequency drive. |

Main equipment included in the LV side:

| Equipment Number | Description |
|---|---|
| 600-MC-001 | Process critical load for emergency back-feed.<br>480 V, 2,000 A, 42 kA. |
| 620-MC-001 | Ore handling and grinding MCC.<br>480 V, 2,000 A, 42 kA. |
| 630-MC-001 | Flotation MCC1.<br>480 V, 2,000 A, 42 kA. |
| 630-MC-002 | Flotation MCC2.<br>480 V, 2,000 A, 42 kA. |
| 640-MC-001 | Regrinding and tailings MCC.<br>480 V, 2,000 A, 42 kA. |
| 670-MC-001 | Reagents and services MCC.<br>480 V, 2,000 A, 42 kA. |
| Standalone VFD (10) | Standalone 480 V VFD for process motors over 200 kW. |
| 220-CP-001<br>640-CP-002 | Control panel hosting PLC1 for E-room equipment control and PLC2 for process control. |

Nine (9) vegetal oil-type distribution transformers used at the process plant will be installed outdoors at 10 m from the process plant electrical room. Each transformer will be separated by a 2-hours fire rated walls and installed with an oil spill containment system.

### 18.9.4 Main Underground Electrical Room at Portal

The main electrical room for the underground mine will be located above ground, near the mine portal. Major equipment located inside the electrical room include a medium voltage 13.8 kV switchgear and a low voltage 480 V MCC to energize the nearby conveyors and surface air compressors. The underground network will be insulated from the surface network by a 10 MVA, 13.8 kV-13.8 kV isolation transformer.

One (1) main e-room will be located underground to distribute power to the different zones. Portable mine substations will be fed by a 13.8 kV network and step-down power at 480 V to the different equipment.

### 18.9.5 Other Electrical Rooms

Small electrical rooms will be installed to support the operations of remote services. Each electrical room will be fed from the 13.8 kV overhead line. Remote electrical rooms include distribution panels and standalone starting devices. Remote services include the reclaim water pumping station, fuel storage facility and support buildings, such as the truck shop and dry buildings.

### 18.9.6 Emergency Power

Located near the main substation, two reciprocating emergency generators of 2 MW running on natural gas will be connected to the main switchgear in the process electrical room. In case of power failure, the critical loads will be kept running using emergency generators. Critical control devices like PLC will be powered by UPS. A third diesel generator will be located at the mine portal for emergency underground distribution.

Generators will also be used in normal operation mode to shave the peak load required during winter seasons. This is relevant for the first five (5) years of operation due to a limitation of power available from the utility grid. Once the utility company completes its grid upgrades, the generators will be used only for emergency events.

### 18.9.7 Telecommunication Network Infrastructure

In order to operate efficiently, modern mines require a comprehensive telecommunications network, comprising multiple communication systems. The Project will require a range of solutions, including fibre, microwave, and satellite backhaul solutions for internet access. Preliminary discussions with the utility company for the deployment of fibre have already commenced to ensure a 1 Gbps connection at the mine.

To support construction communication, the use of Cellular on Wheels (COW) has been proposed. This solution is often used by wireless service providers or emergency services to provide temporary communication at a special event or incident. Given that permanent tower installations can take up to 18 months, a COW will be the quickest way to enable wireless communications at the mine site. Additionally, to ensure a connection offsite, a microwave link from the COW to a nearby SBA tower located at Copper Peak will be required.

To facilitate communication both indoors and outdoors, Land Mobile Radio (LMR) and LTE will be used, with LMR and Wi-Fi being essential for above-ground operations. The LTE core in the main admin building will be connected to other onsite buildings via fibre. A Distributed Antenna System (DAS) will be used to provide indoor wireless coverage. Furthermore, two (2) 200 ft towers will be erected on the mine property to provide cellular service on the site's surface. Any radio and antenna equipment purchased for the construction phase can be reused for the permanent outdoor cellular service.

The underground mine will require fibre to support IoT devices, equipment, and communication. A connection to the main fibre node in the admin building will provide the underground fibre network. Radiating coaxial cables will be permanently installed in two of the four main drifts for permanent LTE communication. After completion, mined panels will be sealed, and reusable telecom equipment transferred to new panels prior to sealing. The use of picocell technology for underground wireless service will simplify installation and equipment reuse in subsequent panels. A single radio will be used for each antenna, connected by fibre cable to a junction near the panel entrance.

To reduce CAPEX expenses as well as maintenance and support costs related to equipment's "on premise" software licenses, cloud-based software applications, including the Enterprise Resource Planning software (ERP), are preferable. The proper implementation of the ERP system is essential for the success of the Project’s mining operations. After careful evaluation of various options, Microsoft Business Central has been identified as the most suitable ERP solution. The implementation process will be executed in four (4) phases to ensure a smooth integration with the company's operations as it progresses towards both construction and production phases. This approach will enable HCC to streamline its business processes and enhance operational efficiency.

## 18.10 Water Management

### 18.10.1 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 On-Stream-Analysis water.

### 18.10.2 Potable Water Treatment

A containerized water treatment plant will provide potable water to services such as offices, safety showers, and the mine dry. Raw water will be pumped from a well to the water treatment equipment housed in a containerized building. Potable water storage tanks and distribution pumps will also be housed in the containerized building. The nominal capacity of 3.6 $m^3$/h.

### 18.10.3 Surface Water Recovery

Lake Superior was planned as the initial water intake to support the processing plant during the first few years of the project. A pumping station was designed to pump the water from a wet well located near the lake to the mine site through a 2.6 km long 8” diameter HPE pipe.

Due to multiple factors, such as the challenge and uncertainty of permitting, as well as relatively high capital costs, it has been decided to evaluate the use of precipitation and run-off water from the TDF area as the main water supply source for the mine site.

Coleman Engineering Company produced a study which demonstrates that for a minimal precipitation scenario (20 years low precipitation scenario), a total of 635.9 k $m^3$ of water can be collected within the 11 months between the construction of TDF Stage 1 and the pre-commissioning phase. In the average precipitation scenario, a total of 855.5 k $m^3$ of water can be collected over the same period.

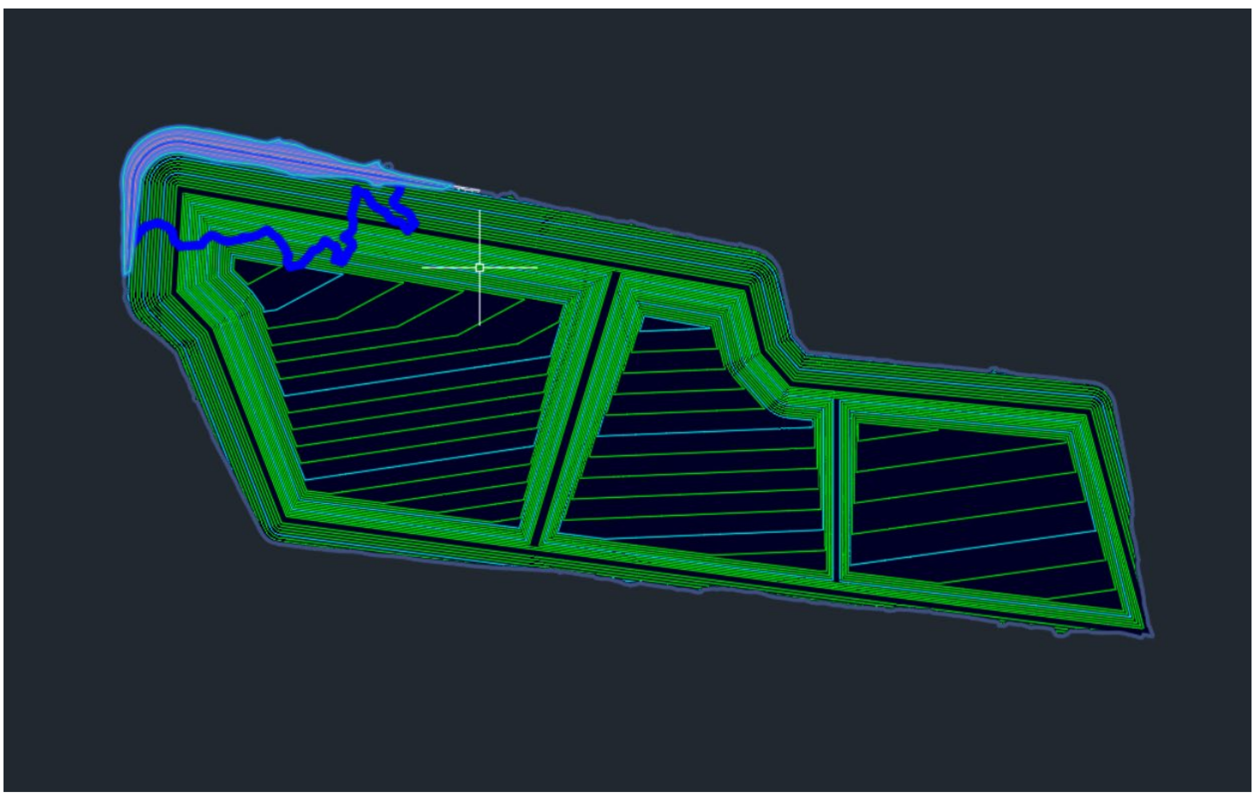
The targeted volume of 283.9 k $m^3$ of water available in the TDF before the pre-commissioning phase is then largely achieved for both scenarios (minimal and average precipitation) allowing for some unforeseen delays with the completion of TDF Stage 1 or an earlier than expected pre-commissioning phase.

The surface water collection strategy consists of a temporary water dam with a capacity of 56.8 $m^3$ of water, built inside the footprint of TDF stage 3. To minimise rework and reduce costs as much as possible, the temporary dam structure will be reused for the permanent TDF stage 3 dam. The temporary dam is located at the lowest point of the drainage area to ensure precipitation / run-off water are effectively captured; water is then pumped from the temporary water dam into the TDF stage 1 by a dual pump system installed on barges.

Water available in TDF 1 is to be used as the main water supply source for the mine site. Later in the life of mine when TDF 1 is at full capacity, water from the temporary water dam will be transferred to TDF stage 2. Temporary water dam will no longer be used once TDF stage 3 is completed. Figure 18.17below shows the temporary water dam designed to collect precipitation / run-off water over the drainage area of TDF stages 2 and 3.

**Figure 18.17: Surface Water Collection Temporary Dam Preliminary Design**

**Figure 18.18: Surface Water Collection with Temporary Dam within the TDF Stage 3 Footprint**



### 18.10.4 Reclaim Water System

To support the water requirements of the process plant, a reclaim water pumping system will be installed at the tailings disposal facility. The reclaim system pumping capacity is designed at 550 $m^3/h$. One (1) floating barge with an access platform and two vertical turbine pumps, (1 operating / 1 standby), control valves, anchor system), one (1) access platform, and a trolley beam and hoist are planned to support pumping and maintenance activities (1 shuttle boat).

### 18.10.5 Stream Relocation

Before the construction of the TDF stages, upstream natural creeks will be diverted to convey their water away from the future infrastructure. A natural channel stream diversion system will be constructed along the future TDF perimeter areas. Prior to the TDF works beginning, connections between the upstream existing creeks and the natural channel stream diversions will be made. Stream channels located in the footprint of the TDF infrastructure will be permanently blocked.

## 18.11 Tailings Disposal Facility

### 18.11.1 General Arrangement and Development

The tailings disposal facility (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 material. The principles for optimizing the TDF to the proposed design were:

- Mostly balanced cut and fill within the TDF's footprint
- A staged construction of facility
- Storage capacity for the planned mine production and estimated process and TDF water balance

For conservatism, the TDF was designed to store the full volume of tailing slurry produced by processing the entire ore. 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 18.19, Figure 18.20, and Figure 18.21.

### 18.11.2 <u>Embankment Configuration</u>

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.

- 3:1 Exterior (downstream) embankment slope (see Figure 18.21).
- 2:1 Interior (upstream) embankment slope (see Figure 18.21).

The configuration and dimensions of the embankment are shown on Figure 18.22 and Figure 18.23.

**Figure 18.19: Tailings Disposal Facility – Stage 1**

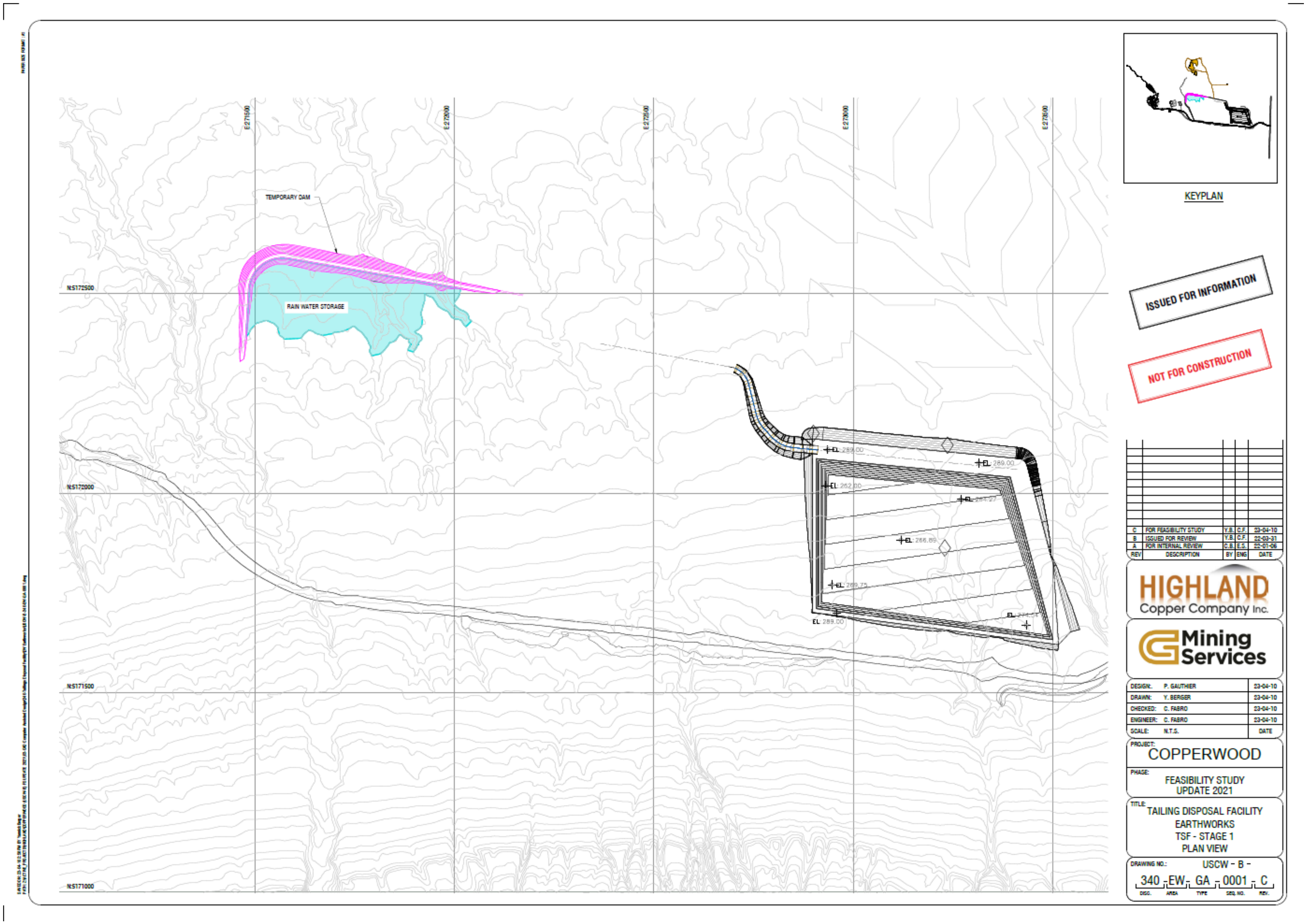


**Figure 18.20: Tailings Disposal Facility – Stage 2**

**Figure 18.21: Tailings Disposal Facility – Stage 3**

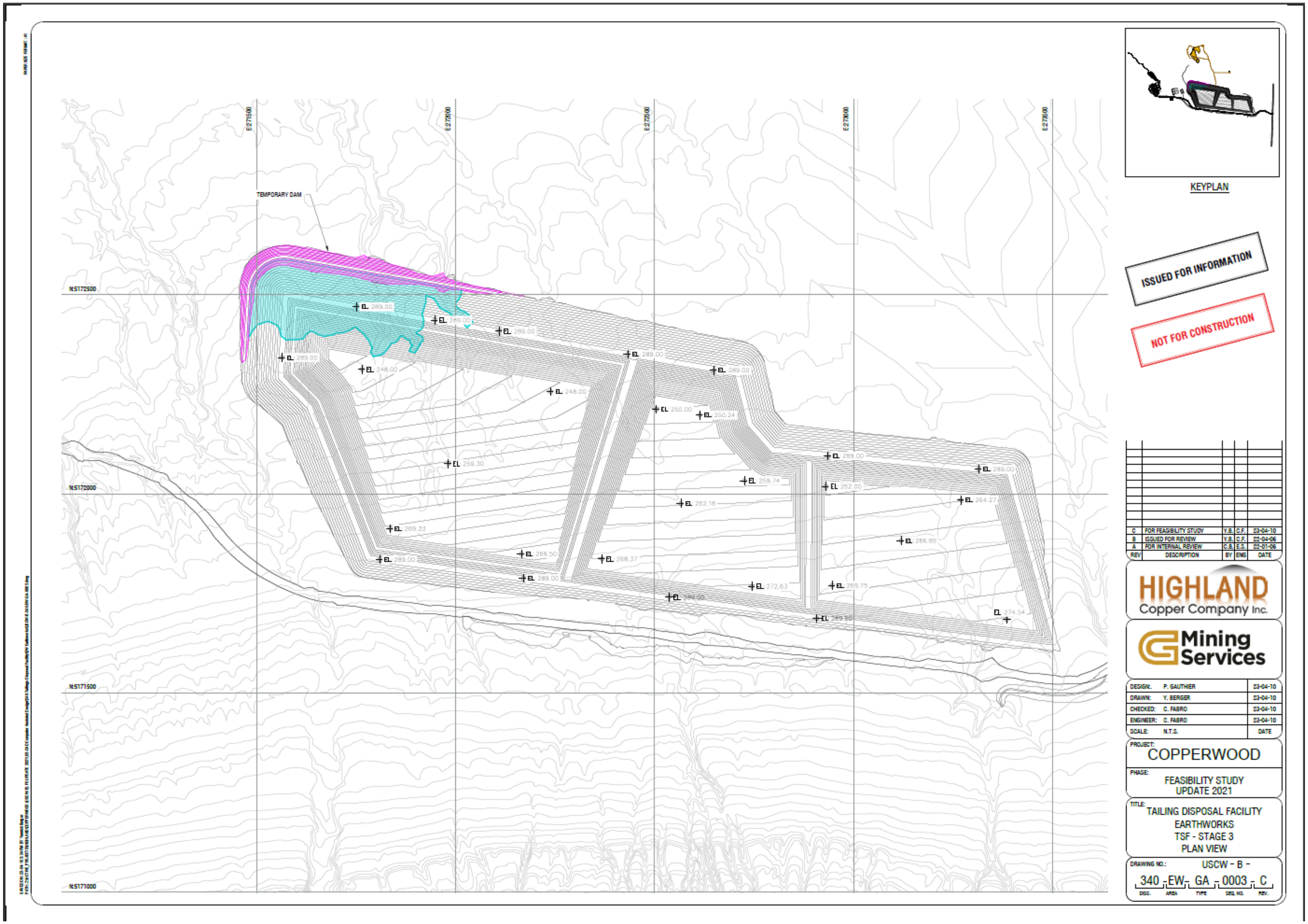


**Figure 18.22: Embankment and Basin Details (1 of 2)**

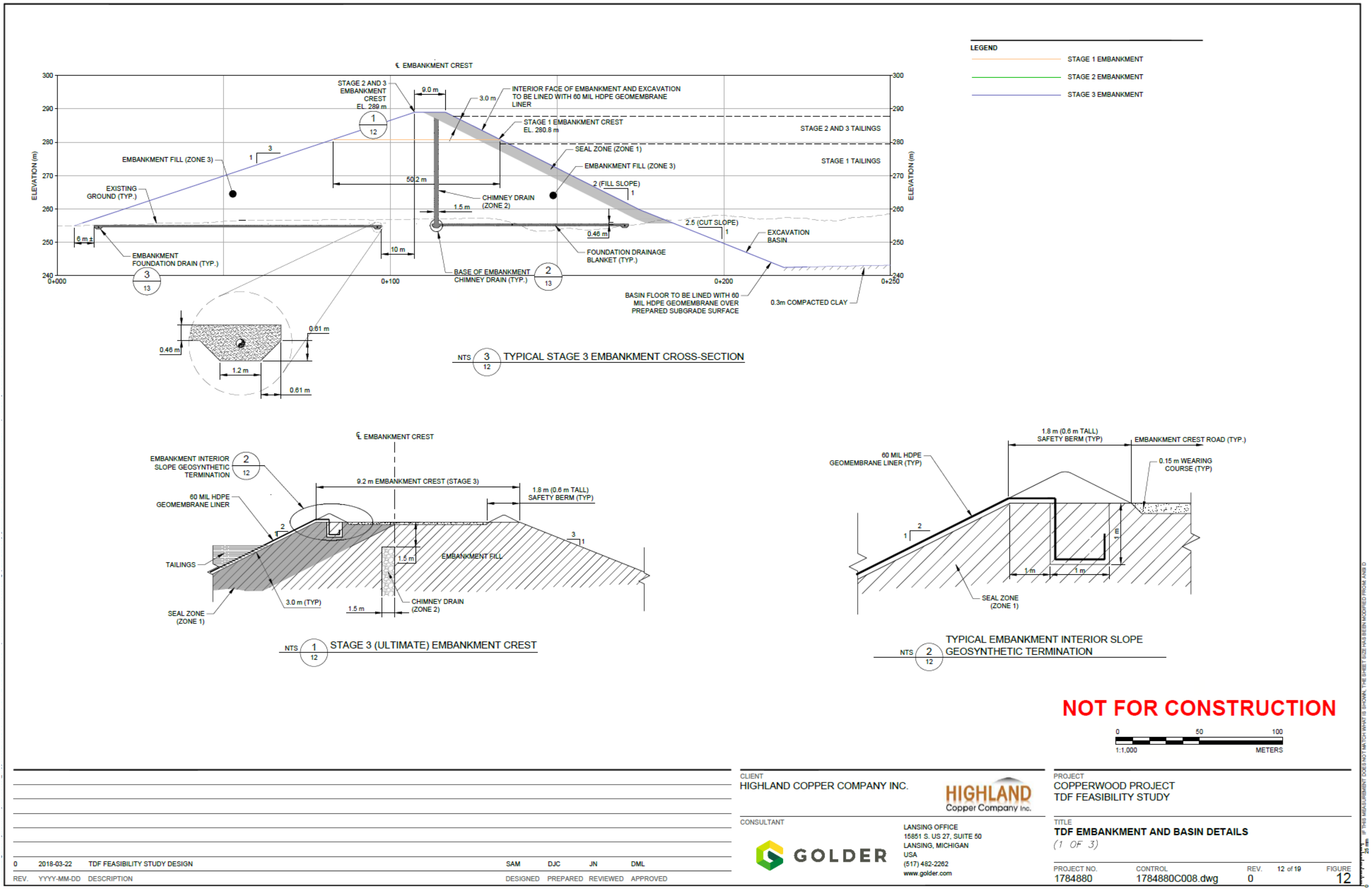
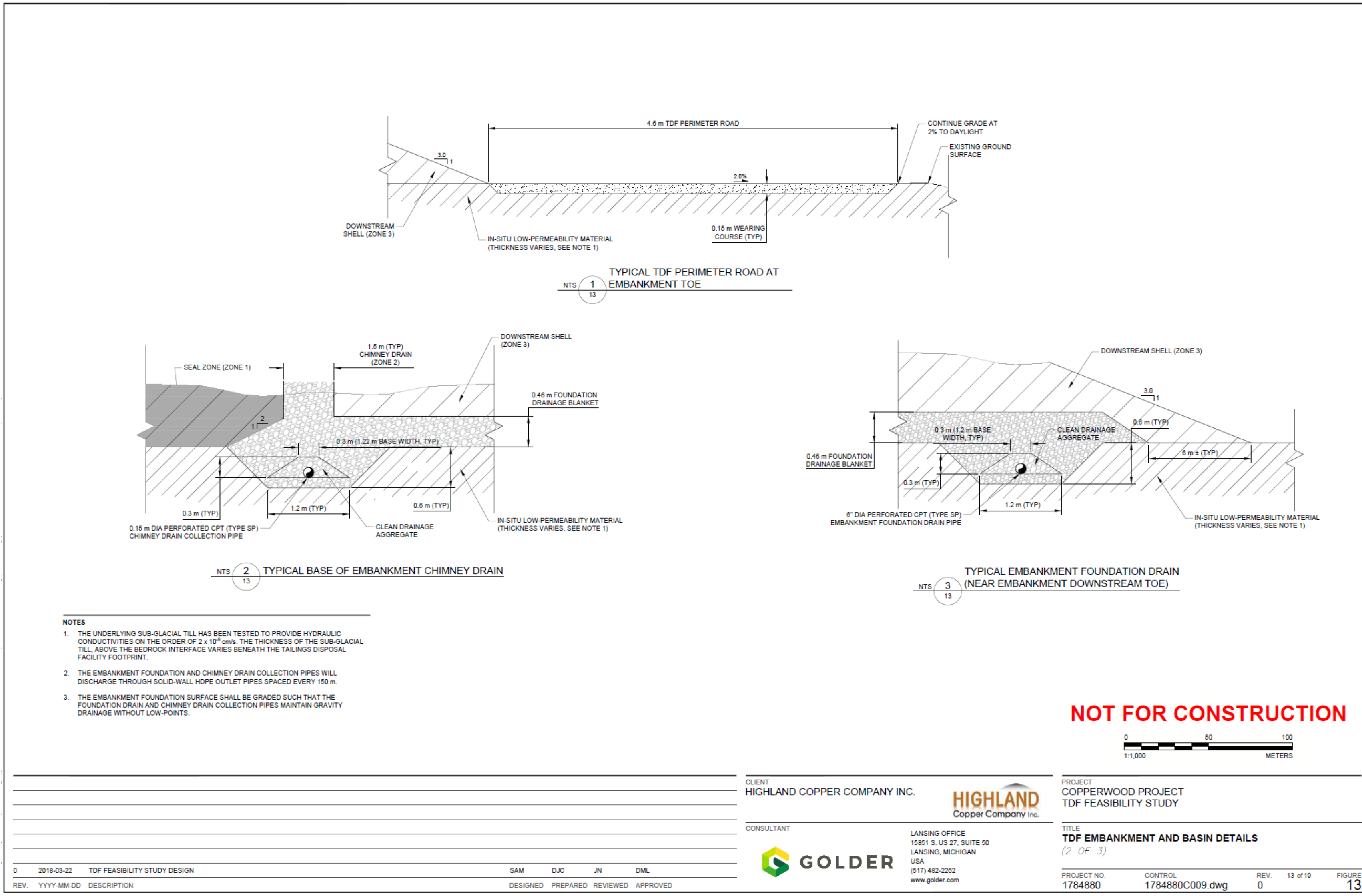


**Figure 18.23: Embankment and Basin Details (2 of 2)**



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 18.22 and Figure 18.23. The embankment will be constructed directly on top of 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 fine grained and clayey, 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 wetter than 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 each material type.

The excavation within the basin for fill material will vary from 0 to 20 m, generally increasing 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 18.22 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 well logs, the glacial till will provide an impermeable unit over the bedrock.

### 18.11.3 Decant System and Tailings Management

Tailings from the process plant will be pumped as a slurry to the TDF through an overland tailings pipeline and discharged within the basin. A tailings disposal 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 process plant during the first four (4) years of operations, and once the water balance becomes positive, decant water will also be pumped to the water treatment plant prior to discharge to the environment. A barge-mounted pumping system will be used and periodically moved within the TDF basins over the life of mine. The initial barge location will be where the basin elevation is the lowest (northwest) and as the tailings are discharged move towards the south.

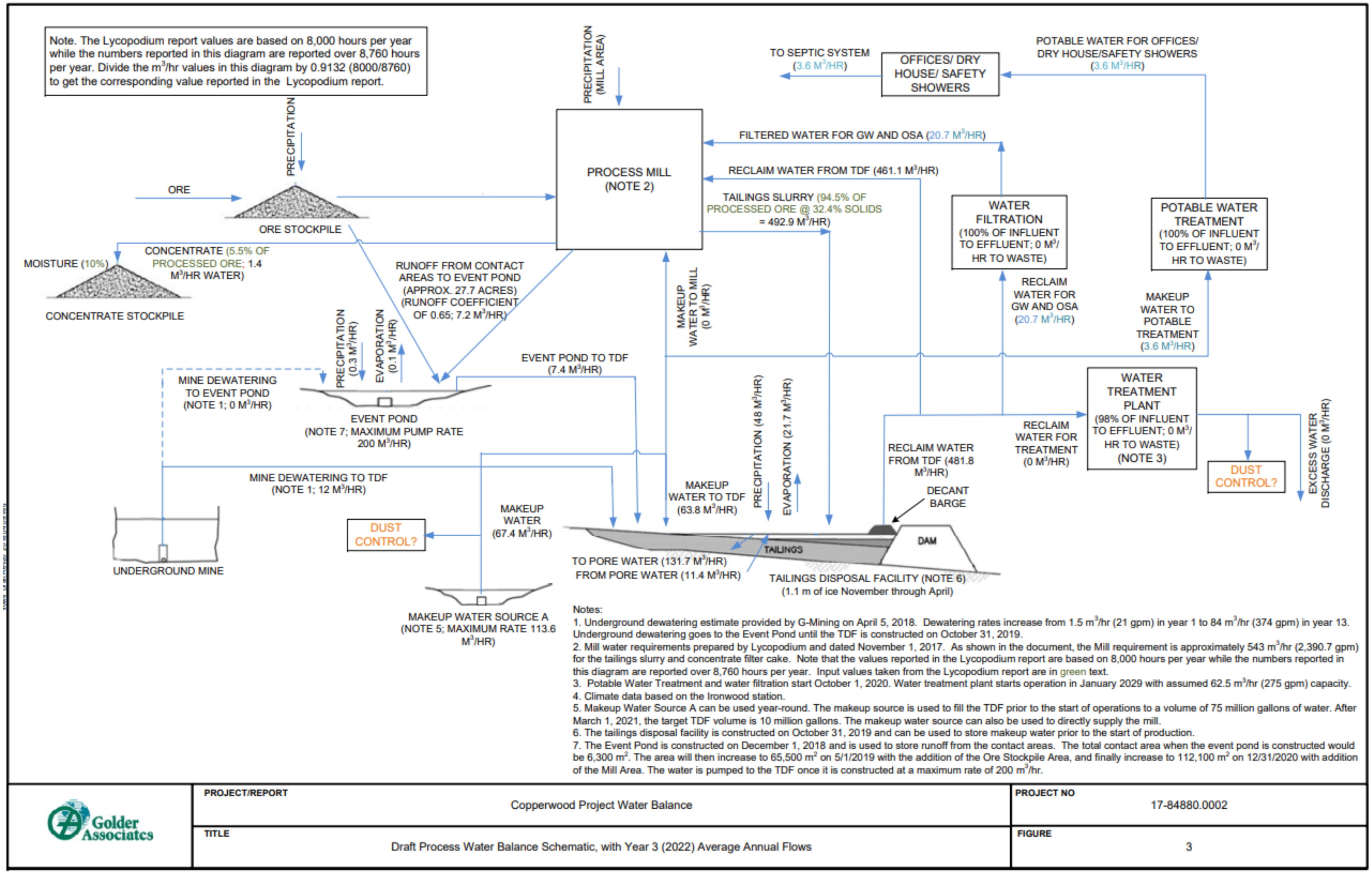
#### 18.11.4 Water Treatment Plant

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 precipitation and run-off water collected from the TDF footprint will be used to build and maintain inventory in the TDF.

The total volume of water considered in the water balance includes direct precipitation, run-on from the TDF liner area, and water pumped from the event pond at the mill site. Once the process plant ramps up production, the precipitation and run-off water collection will be required to maintain a total TDF pond volume (frozen and unfrozen water) of 283,906 m$^3$. After startup is completed and the plant reaches full production, the precipitation and run-off water collection will continue to be used as needed to maintain a total TDF pond volume (frozen and unfrozen water) of 37,854 m$^3$. Precipitation and run-off water collection will not be required once the WTP is operational, as the project will then have a positive water balance. 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 18.24 shows the water balance schematic.

**Figure 18.24: Process Water Balance Schematic, with Year 3 Average Annual Flows**




*Source: Copperwood Project – Tailings Disposal Facility and Water Balance, Golder Associates Inc. June 2018.*

The water balance model inputs include the following:

- Production schedule
- Climate data and site conditions
- Underground mine dewatering
- 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

### 18.11.5 Water Treatment Plant Design

A feasibility design of the WTP was completed by Golder in 2012. As a part of the Feasibility Study performed in 2018 and, in relation to the updated water balance evaluation, Golder reviewed their design to verify its adequacy to the mine and water management plan.

### 18.11.6 Influent Design Basis

The Influent Design Basis (IDB) model made by Golder includes water quality, water quantity and the treatment requirements for TDF water discharge compliance. 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 at a design capacity of 79.5 $m^3$/hr discharging in the Namebinag Creek at a permitted point north of the mine portal. 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 tailing's slurry, underground mine water and precipitation from different sources. The 2012 model was reviewed and updated in 2018. At this stage, it was assumed that the 2018 data is still valid.

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.

Chemicals of Potential Concerns (COPCs) include arsenic, barium, boron, copper, mercury, selenium, silver, strontium, vanadium, total suspended solids, and total dissolved solids.

The treatment goals from the 2012 Study were reviewed and based on the following:

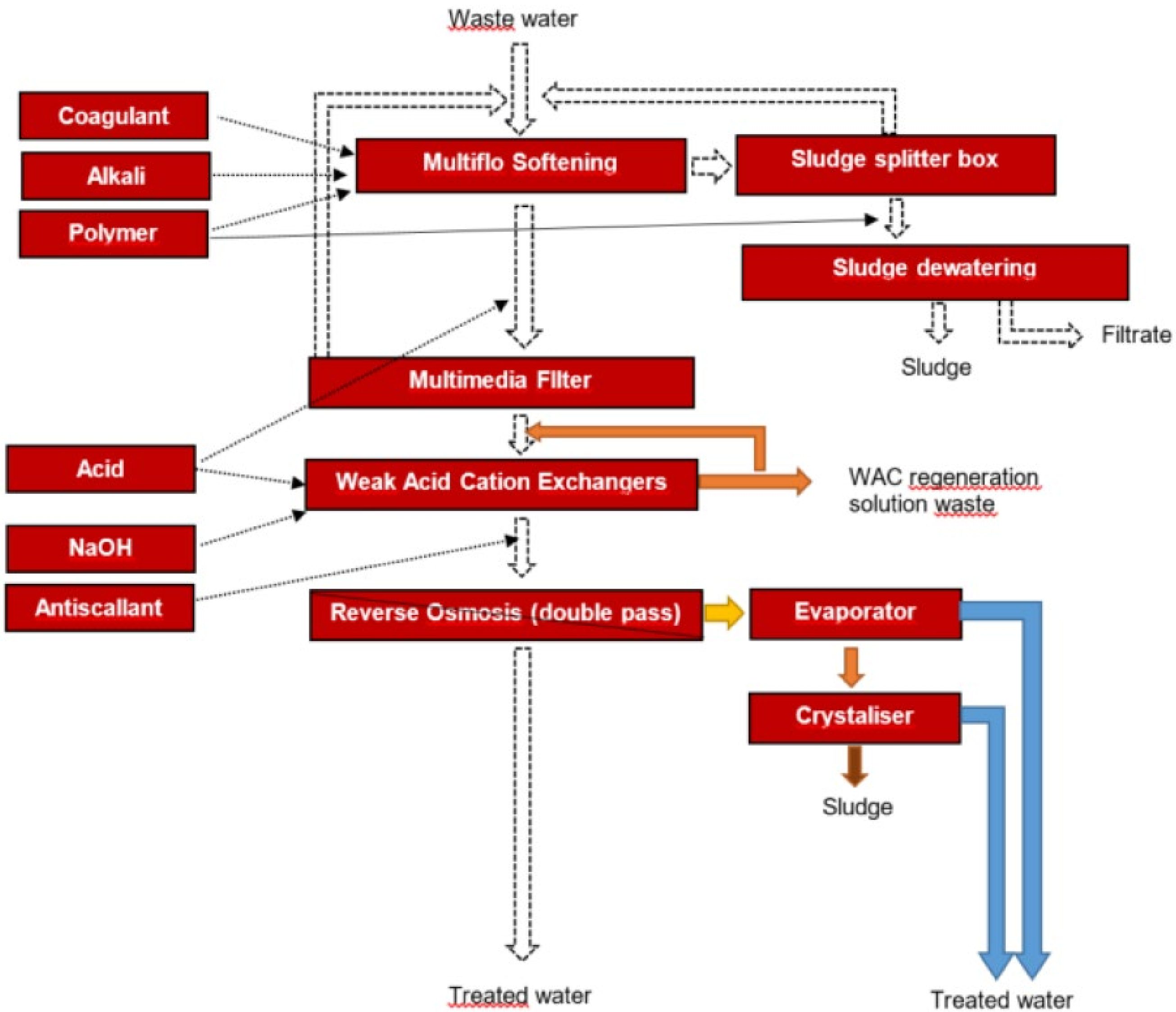
- The Michigan Department of Environmental Quality Rule 57 Water Quality Values (October 21, 2016).
- The National Pollutant Discharge Elimination System (NPDES) permit expiring October 1, 2024, for the Copperwood Underground Copper Mine.

### 18.11.7 <u>Water Treatment Plant Process</u>

Based on the IDB and requirements, the treatment system will include:

- pH and hardness adjustment
- Multiflo Softening
- Sludge Dewatering
- Multimedia Filter
- Weak Acid Cation Exchanger
- Reverse Osmosis
- Evaporation / Crystallization
- Chemical feed and storage

**Figure 18.25: Process Flow Diagram of Water Treatment Plant**




*Source: Veolia*

#### 18.11.8 Fire Protection

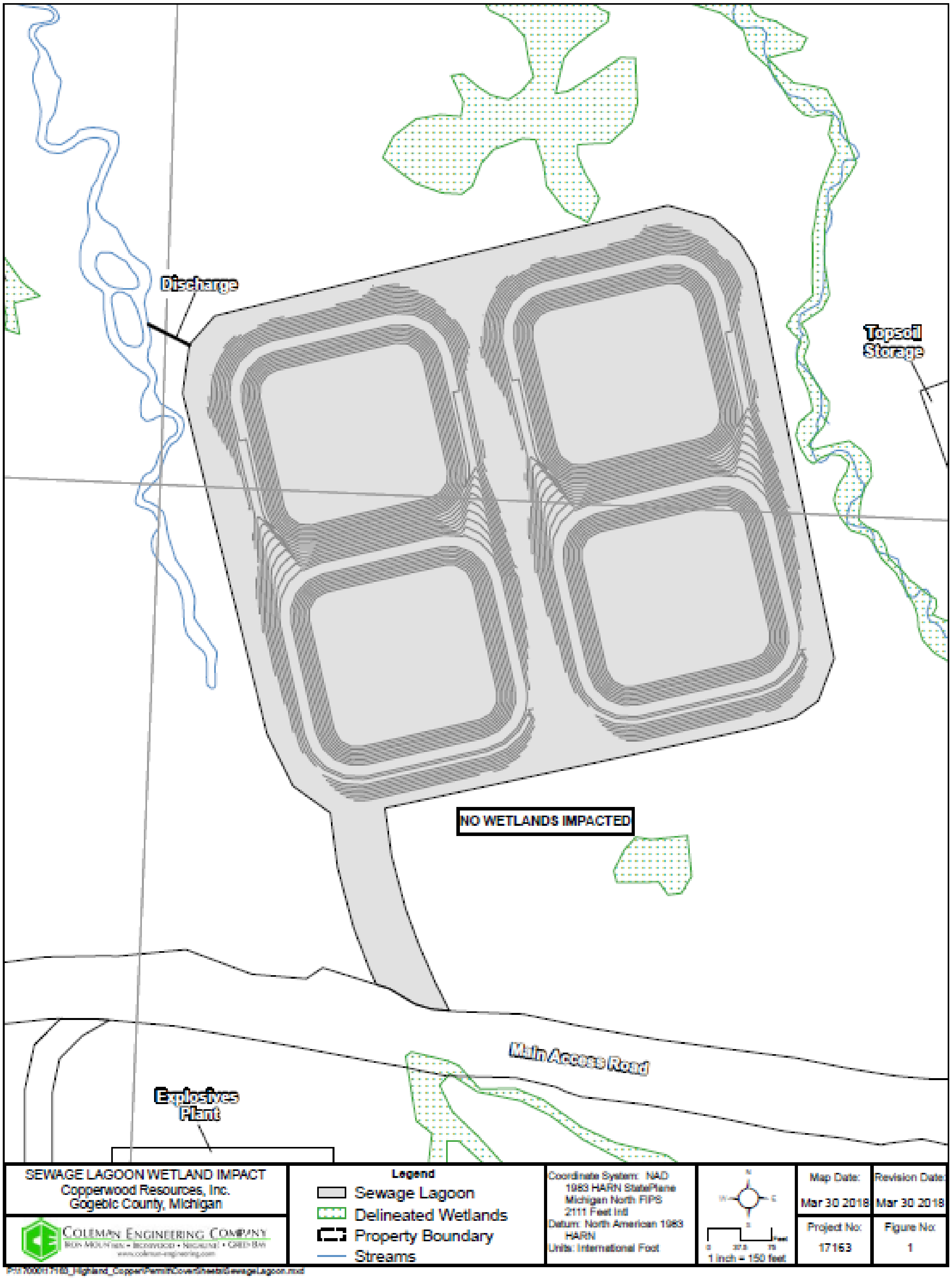
Water for emergency fire extinguishing will be stored in a 100,000 USG tank East of the Process Plant. Fire pumps will provide the proper water flow and pressure as stipulated in the North American codes (NFPA). The fire pumping equipment will include one diesel pump, one electrical pump, and one jockey pump to maintain pressure within the fire protection distribution network.

The distribution network will be supported using buried HDPE pipes. Each building will have its own fire water distribution system. To support fire protection around the property, fire hydrants will also be installed.

#### 18.11.9 Sewage Treatment

Sewage treatment will be handled using stabilization ponds. The stabilization ponds are based on Orvana Copperwood Lagoon System Evaluation done by Coleman Engineering in 2012, which they updated in March 2020 following some comments from Highland Copper. The ponds effluent water discharge will drain into the west branch Namebinag Creek via a pipeline to a discharge point approved by the Michigan Department of Environment, Great Lakes, and Energy (EGLQ). Two ponds will be constructed during construction and two additional ponds at year-4 of operation.

**Figure 18.26: Process Flow Diagram of Water Treatment Plant**



#### 18.11.10 Natural Gas network

Natural gas is required for the site generators (peak shaving and emergency power) and the gas burner to heat the underground mine. Skid type natural gas decompression station will be installed at the entrance of the mine site. Decompression station will transfer CNG to usable pressure natural gas to equipment. From this station, a pipeline will be brought to emergency generators, mine portal and mine vent raise.

Compressed Natural Gas will be trucked from nearest CNG installation to site by a logistic company that owns the trucks and decompression station. Logistic decompression station renting rate will be added to natural gas market price.

### 18.12 Mine Infrastructure

#### 18.12.1 Ore Stockpile Pad

The ore stockpile pad is located 200 m southeast 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. Once construction is completed, a stacker conveyor will be used to divert ore to the stockpile. Ore from the stockpile pad will be fed to the processing facility by a front-end loader. Ore will be dumped into a hopper / feeder system and discharged on the ore bins feed conveyor. The stockpile will reach its maximum capacity at the end of the mine development period, and just before the ramp up of production of the process plant.

The pad is approximately 65,000 $m^2$ 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. The geomembrane will be covered by 300 mm of fine grain material to protect the integrity of the membrane. Water from the ore stockpile must be managed as contact water. It will be recovered and pumped to the event pond and/or to the process plant. 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 northwest corner of the stockpile where it will be pumped to the event pond and ultimately to the TDF or the water treatment plant later in the life of the mine.

#### 18.12.2 Box-Cut

The box-cut entrance is located approximately 300 m northeast of the mill area. It is located close to support buildings such as the mine dry, maintenance shop and warehouse. The ore stockpile is located just southwest of the box-cut. The design of the box cut will consist of a ramp approximately 250 m long with a 15% slope. It provides access to the mine portal and underground mine. The box-cut will be excavated at a minimum of 15 m into the fresh rock, and 2 x 6.7 m diameter round multi-plates culverts will be placed, and then backfilled for water management. One tunnel will be used by underground mobile equipment and one tunnel will house the main conveyor moving ore to the surface.

**Figure 18.27: Box-Cut Design**

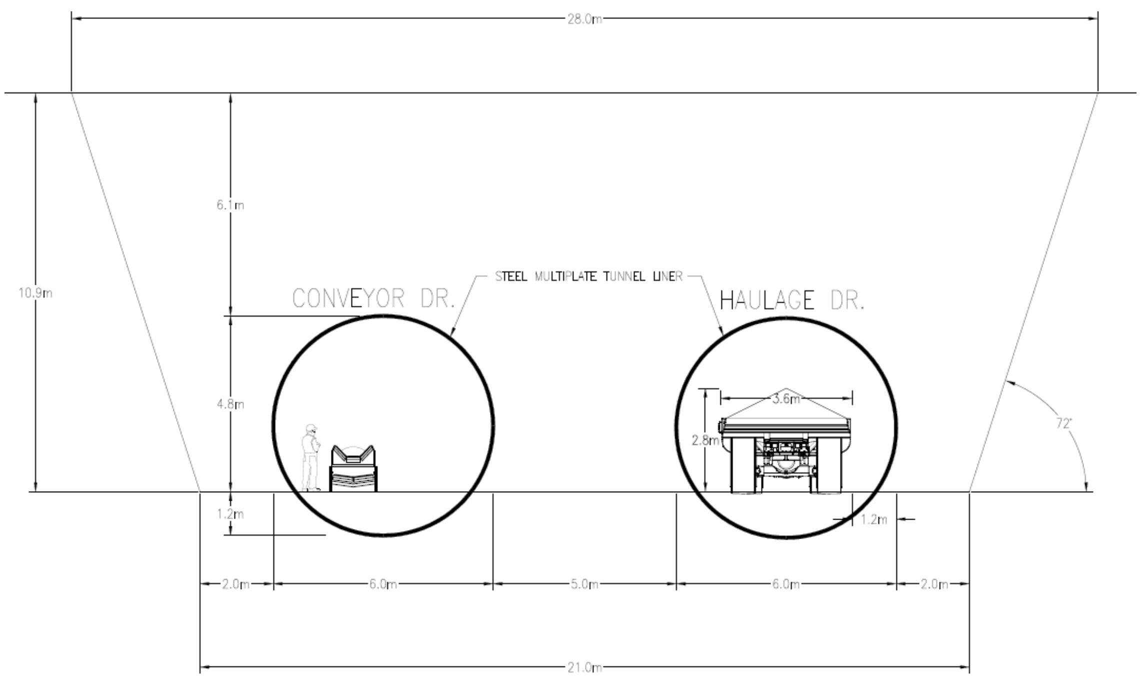
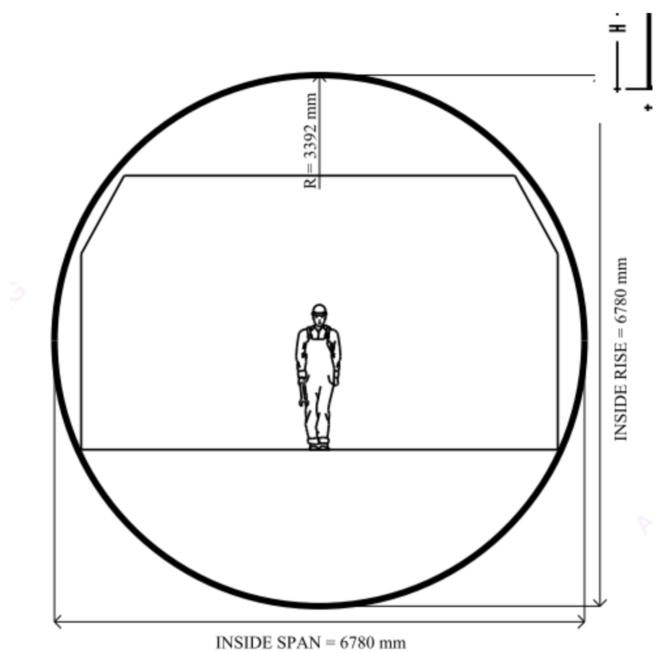



**Figure 18.28: Circular Multiplate Dimensions for Box-Cut**



### 18.12.3 Underground Compressors

Underground compressed air requirements will be supported by four (4) screw compressors each equipped with an air dryer (3 Running – 1 Standby) located in a shelter close to the mine portal. The dimensions of the compressor building will be 15 m x 8 m. Two (2) vertical air receiving tanks will be located outside of the building. Each compressor will have a 1,501 CFM@125 PSI capacity.

**Figure 18.29: Underground Compressors Building Plan View.**

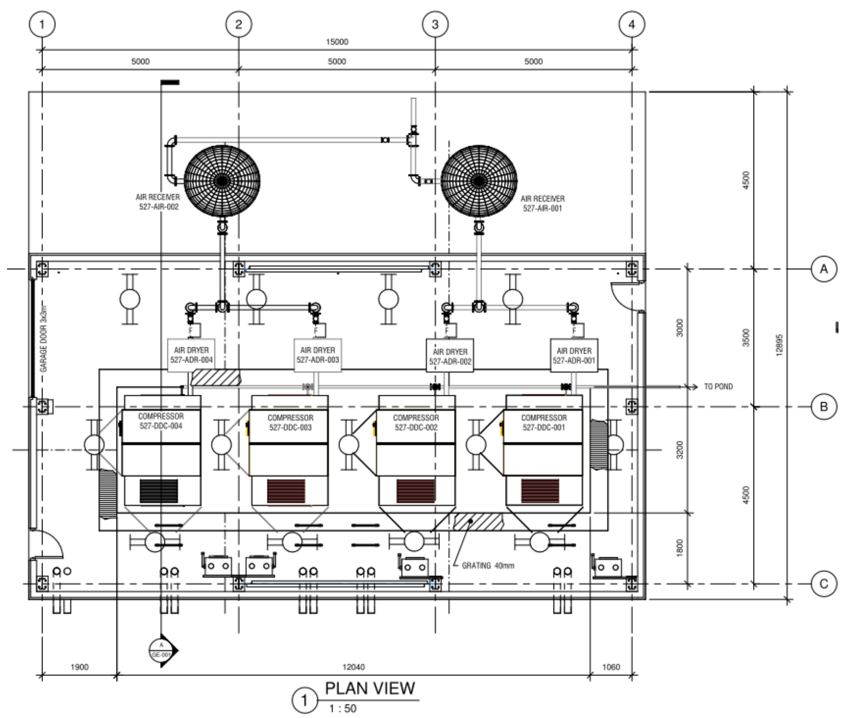
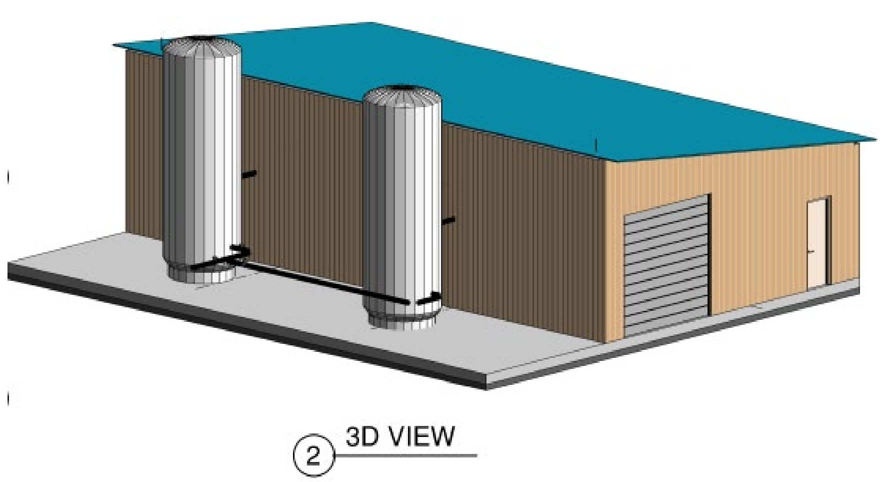



**Figure 18.30: Underground Compressors 3D View**



## 18.13 Construction Facilities

### 18.13.1 Contractor Laydown

The contractor laydown will be located on the northwest side of the process plant platform to the north of the parking lot. The main construction office, contractors' offices, along with services (power, communications) will be installed in the contractor's laydown during the construction period. Eight (8) to ten (10) contractors should be active at the same time on the Project and sufficient space is available for offices and shops. It represents around 5,800 square metres. In the early phases of the construction, portions of the parking lot could be used temporarily.

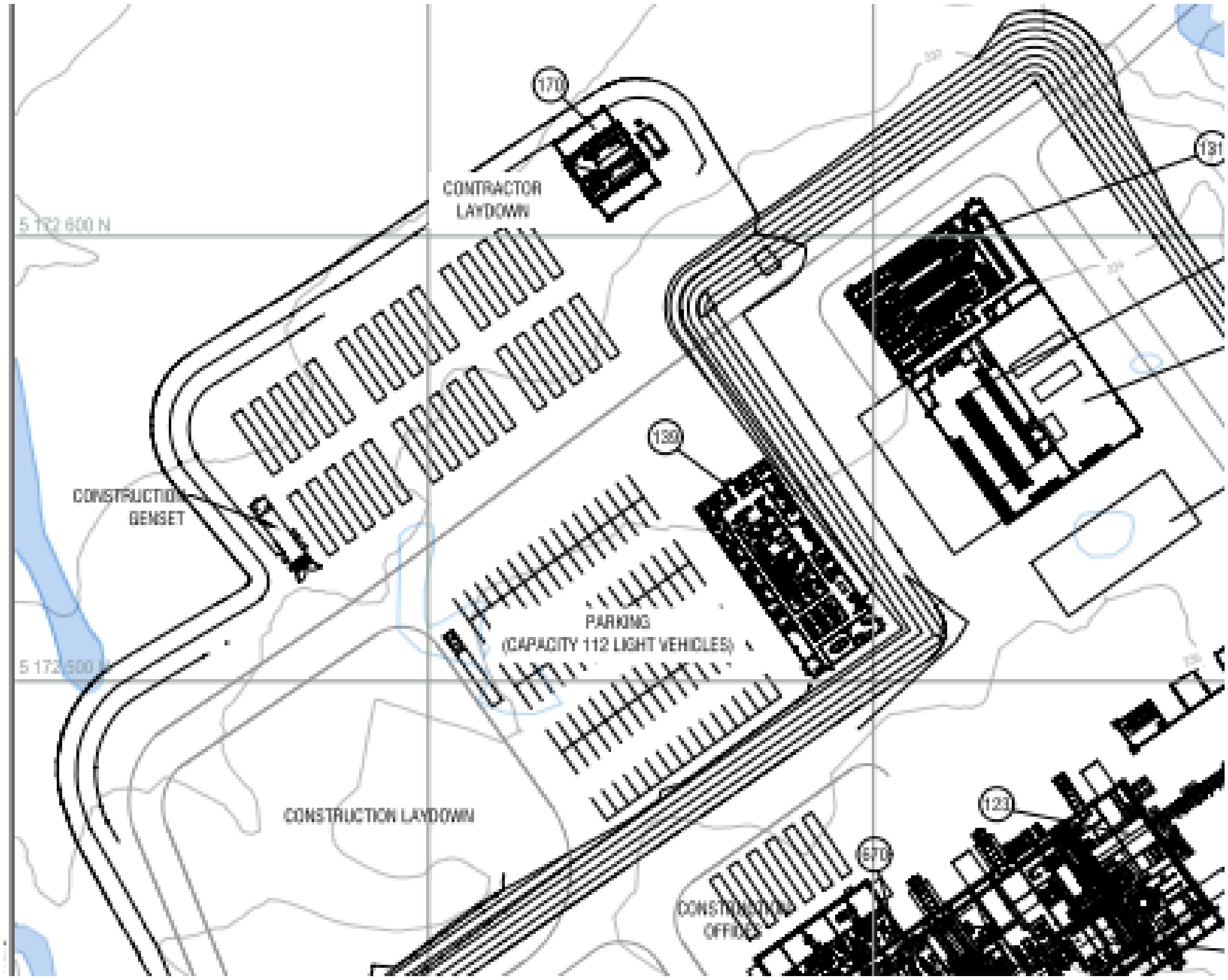
### 18.13.2 Construction Offices

The construction offices are containerized and located just north of the process plant. Ten (10) stand-alone 50' trailers 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 can be purchased to allow for general office space over the life of the mine. Additional offices will be required for the contractors. As these are temporary, they will be located adjacent to the construction office to facilitate electrical connections.

### 18.13.3 Construction Laydown

The common construction laydown area will be located west of the parking lot. The laydown will be managed by the construction management team and used to store material and equipment during the construction period. The laydown area will be approximately 6,100 square metres. Containers, as well as a covered shelter will be located at the construction laydown to provide shelter for weather-sensitive equipment and material.

**Figure 18.31: Construction Facilities Location**



# 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$4.00/lb. The silver price has been assumed constant at US$25.00/oz over the Project life.

**Table 19.1: Metal Price Assumptions**

| Metal Price Scenario | Yr -1 (2025) | Yr 1 (2026) | Yr 2 (2027) | Yr 3+ 3(2028+) |
|---|---|---|---|---|
| Copper (US$/lb) | 4.25 | 4.15 | 4.00 | 4.00 |
| Silver (US$/oz) | 25.00 | 25.00 | 25.00 | 25.00 |

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$4.90/lb seen in March 2022 and a low of US$1.95/lb in beginning-2016. The 10-year historical price for silver is similarly presented in Figure 19.2.

**Figure 19.1: 10-Year Historical Copper Prices**


*Source: www.macrotrends.net*

**Figure 19.2: 10-Year Historical Silver Prices**


*Source: www.macrotrends.net*

## 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 maritime freight.

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$65/dmt of concentrate, RC = $0.065/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 (6) 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 |

| Concentrate Analysis | Minimum | Maximum | Expected (Average) |
|---|---|---|---|
| Cl g/t | 0 | 300 | 135.0 |
| F% | 0.038 | 0.046 | 0.042 |
| SiO2% | 32.6 | 40.2 | 36.4 |
| Al2O3% | 7.93 | 9.34 | 8.7 |
| Fe2O3% | 11.3 | 14.4 | 13.5 |
| MgO% | 2.76 | 3.51 | 3.1 |
| CaO% | 0.59 | 0.85 | 0.7 |
| K2O% | 1.75 | 2.16 | 1.9 |
| TiO2% | 0.88 | 1.04 | 1.0 |
| MnO% | 0.11 | 0.15 | 0.12 |
| Cr2O3% | 0.043 | 0.180 | 0.102 |
| V2O5% | 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 | 5,770 | 7,690 | 6,825.0 |
| 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 |

| Concentrate Analysis | Minimum | Maximum | Expected (Average) |
|---|---|---|---|
| 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

In 2017, Concept Consulting LLC conducted a study on concentrate transportation. The assumptions made by Concept in 2017 were reviewed in 2022 and were updated based on discussions with local trucking companies, and rail operators. Final delivery point is still considered as the Horne smelter, in Rouyn-Noranda.

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 Champion, Michigan approximately 180 km from site. The truck configuration consists of a 11 axles road train with two (2) covered side-dump trailers and will transport approximately 48 t (53 short tons) per shipment. The location has been chosen due to the costs and mainly because it provides access to the Canadian National Railway (CN) networks. The operator of the rails between Champion and Ishpeming is Mineral Range. Wagons will be loaded at the transload facility, then moved from Champion to Ishpeming. From there, wagons will be moved by locomotive owned and operated by the CN. The CN is a Class 1 railroad 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 transload facility is described in the Section 18.

The concentrate transportation costs are estimated at US$98.89/t of concentrate which includes trucking, transload operations, 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 | 26.25 |
| Transload Operations | 4.41 |
| CN Rail Transportation | 59.78 |
| MRR Rail Transportation | 1.35 |
| Gondola Lease Costs | 5.20 |
| Lid Rental | 1.42 |
| Lease Property | 0.48 |
| **Total Transport Cost** | **98.89** |

### 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, PERMTTING AND SOCIAL OR COMMUNITY IMPACT

## 20.1 Environmental Studies and Material Impacts

Extensive environmental studies were undertaken to obtain the original Mining Permit issued in 2012, with additional studies commissioned for the Mining Permit Amendment application of 2018. In accordance with Michigan's governing regulation Natural Resources and Environmental Protection Act (NREPA) Part 632 Nonferrous Mining, studies describing baseline conditions were conducted characterizing the following:

- Groundwater
- Surface Water
- Wetlands
- Geology
- Hydrogeology
- Aquatics and Fisheries
- Geochemical Characteristics
- Geotechnical Characteristics
- Terrestrial Survey and Habitat
- Archaeology

The Project's baseline setting and potential impact on the environment are described in detail in the Mining Permit Application (Orvana 2011), Mining Permit Application Copperwood Volume II (Environmental Impact Assessment Amendment [EIAA]), and Copperwood Project Environmental Assessment (Highland Copper Company Inc., 2018). The studies are summarized in the earlier FS as well, (G Mining, 2018) so will not be repeated here. The resulting environmental impact description is summarized in this section.

### 20.1.1 Topography, Drainage, Surface Water (Streams)

The project will significantly change the area within its footprint by disturbing and constructing infrastructure and impervious surfaces. A large portion of the footprint, considered the contact area, collects all contact water to route to treatment prior to discharge to the west branch of Namebinag Creek. Outside the contact area, storm water will be routed to adjacent streams. The change in storm water flow is a slight increase from baseline flows.

Construction of the Tailings Disposal Facility (TDF) necessitates the fill and re-routing of the upper headwaters of Gipsy Creek and Lehigh Creek. Flow from these two (2) creeks as well as surface water to the south of the TDF will be diverted around the TDF into new natural stream design diversion channels.

Post operations, infrastructure will be removed, and impervious surfaces will be restored to pre-mining conditions. The exception is the TDF, which will permanently remain. Under the closure plan, the TDF will be capped and vegetated such that drainage flows similarly to pre-mining conditions.

### 20.1.2 Water Balance

Minimal impacts on the existing site water balance are attributed to the Project. This considers mine dewatering effects on groundwater discussed in the next section and facility impact on surface water in the subsequent section.

### 20.1.3 Groundwater

A variety of studies and tests have characterized the groundwater, summarized below:

- Unconsolidated glacial overburden deposits range from 10 feet to 140 feet in thickness.
- Three (3) primary lithostratigraphic units may be present within the project area including a lower portion of coarser materials, an upper portion of finer grained materials, and lateral, inconsistent layers of silty and sandy till and sand.
- Hydraulic conductivity results and groundwater sampling events indicate the upper unit transmits very limited quantities of water; the lateral inconsistent (middle) layer has hydraulic conductivities roughly two (2) orders of magnitude greater than the upper unit and one (1) order of magnitude greater than the underlying unit. The limited thickness and lateral extent of the middle unit; however, does not provide a significant pathway for groundwater flow.
- Groundwater flow within the units is very slow and in the context of other characteristics, the conclusion is that there are non-useable aquifers beneath the site. Studies show that groundwater travel time from the site to Lake Superior should not change dramatically from current conditions. Groundwater within many parts of the bedrock is brine.
- Groundwater within the overburden is recharged predominantly through precipitation, with spring and fall influencing the recharge. Most precipitation falling on the surface runs off into the surface water system or is lost to evapotranspiration.

- The groundwater model indicates that given the system characteristics of minimal groundwater presence and movement, groundwater drawdown and cone of depression will likely be limited and minimal.

The Project impacts on groundwater will likely be limited to reducing the groundwater infiltration from pervious areas on site. Additional potential impact mechanisms from Project activities include contaminant seepage from leaching from the underground mine walls, leakage from liner deficiencies (TDF and ore stockpile), subsidence effects, and spills. Each mechanism has been addressed with estimates of maximum potential and commensurate mitigation measures, summarized below.

During operations, there may be limited constituents seeping into the groundwater from leaching in the excavated mine walls. Mine water inflow will collect in sumps to be used in operations or treated prior to discharge. Upon mine closure, the mine will be flooded with fresh water, therefore preventing ongoing oxygenated leaching.

To protect groundwater quality, engineered liners or impervious pads will be installed under all structures containing potentially reactive materials – tailings, ore, and the ore stockpile. The TDF will have a rigorous design in accordance with NREPA Parts 315 and 632 permits. The requirements include liner system performance standards, leak detection system, testing, and construction quality assurance certification. This approach provides the TDF with the long-term leakage integrity appropriate to protect groundwater. Monitoring during operations and post-closure continuously informs system integrity.

Subsidence can potentially occur upon mine excavation. Should subsidence become pronounced, it may result in opening pathways for surface water to infiltrate into the mine and surrounding host area more effectively. Flooding may occur as well as introduction of surface contaminants into the groundwater, if present. Potential surface subsidence was studied and modeled for the Project, with results indicating that the maximum surface subsidence could be approximately 0.1 feet (3 cm). This points to minimal anticipated subsidence. The Mining Permit as amended contains requirements for subsidence monitoring, with a subsidence monitoring plan that must be approved by EGLE before mining operations.

Potential spills are present at practically all industrial facilities. Federal and state requirements, including the implementation of Spill Prevention Control and Countermeasures Plan, Pollution Incident Prevention Plan, and Stormwater Pollution Prevention Plan maintain inspection, management practices, monitoring, inspections, and reporting to minimize the potential for spills or contaminated runoff to enter the environment.

### 20.1.4 Surface Water – Lakes, Streams and Wetlands

Surface water within the Project footprint will be impacted by spatial interference and by redirecting natural water flows. The Wetland Permit allows wetland and stream impacts: excavation or fill of approximately 57.52 acres of wetlands and hydrologically impacting 3.29 wetland acres. To accommodate the TDF, 16,557 lineal feet of sections of Lehigh Creek, West and East Branches of Gipsy Creek will be abandoned. Approximately 3,800 lineal feet of stream channel will be constructed to relocate a portion of middle Branch Gipsy Creek and approximately 9,900 lineal feet of stream channel constructed to divert water around the south and west sides of the TDF. Mitigation for the permitted wetland impacts include preserving 717 acres of high-quality wetland and an on-site creation of 18.3 acres of wetlands.

Wetland and stream impacts from mine dewatering was modeled in support of the Mining Permit and described in Orvana (2011). The glacial till present has very low hydraulic conductivity with no surface springs, seeps or other groundwater expressions observed in the area. Isolated wetlands between the on-site streams appear to be perched above the water table and will not be significantly impacted as the mine develops.

The estimated worst-case potential impacts were as follows to wetlands and streams adjacent to the site. The impacts are small and will not likely be noticed in the natural variability of the stream flows. Impacts will cease upon cessation of operations.

- Approximately 13 acres of riparian wetlands may experience a decrease in groundwater discharge. This groundwater discharge impact will be temporary as the mine dewatering will end at cessation of operations. The estimated change in water volume discharged to the area is very small due to the low hydraulic conductivity of the glacial till.
- For streams within the footprint, flow may be reduced. The maximum estimated decrease in average estimated stream flow due to mine dewatering:
  - Gijik Creek at SW-M: 0.08%
  - Unnamed1 Creek at SW-N: 0.09%
  - Namebinag Creek (East Branch) at SW-P: 0.06%
  - Lehigh Creek at SW-Q: 0.02%.

Three (3) unnamed creeks flowing to Lake Superior are present directly southwest of the Project. The Unnamed Creek listed above is the one closest to the site and may be named Linja Creek, to be confirmed.

To supply water needed for ore processing at the mill, precipitation and run-off water from the TDF area will be collected and managed in the TDF. Section 18.10.3 provides more detail. Using this method reduces the overall impact on site water balance compared to accessing a supply from Lake Superior (an option noted in G Mining (2018)) or from groundwater. Precipitation will always enter the facility boundary (although in a highly variable manner); however, by using the large area needed for tailings management as a reservoir, water naturally entering the site can be used by the facility. All excess water will be treated prior to discharge.

### 20.1.5 Surface Water Quality

Potential impacts to surface water may originate from sediment mobilization during construction and operations, spills, deficiencies in contact, septic, and storm water management, liners, and water treatment. The probability that any impacts will occur is minimized by implementing management plans and maintaining permit and regulatory compliance. More detail on permits is provided in Section 20.5.

### 20.1.6 Geochemical Characteristics

The geochemical characteristics of potentially reactive materials (ore, waste rock, tailings) were studied and presented in the original permit application (Orvana, 2011). Static and humidity cell testing were conducted in accordance with standard industry practices. Mineralogy of overburden, rock, and ore were found to have limited variation within a particular geologic unit. The testing and mineralogy concluded the general lack of acid-generating sulfide minerals and the presence of acid-neutralizing calcite. That said, the material storage facilities (TDF, ore piles, etc.) will be lined and the leachate collected will be treated prior to discharge.

### 20.1.7 Mine-Induced Subsidence

Minimal to no subsidence of the overlying bedrock and overburden are indicated in a comprehensive geotechnical report (Golder, 2018). This supports the premise that the ground surface will not be affected and impact features such as streams and wetlands. The Mining Permit requires subsidence monitoring and reporting throughout operations.

### 20.1.8 Aquatic Flora and Fauna

The mine entrance, ore stockpile, and mill will reduce the drainage to local streams. Baseline studies identified limited aquatic communities primarily due to relatively seasonal intermittent water flows, accumulations, and beaver impoundments. Wetlands on the site are primarily supported by surface water runoff and most have no standing water during the summer. With the poor quality of aquatic macroinvertebrate community, very limited fish habitat, and seasonal water presence, the Project is anticipated to have limited impact on aquatic plants and animals. Monitoring is required throughout operations.

### 20.1.9 Terrestrial Flora and Fauna

Construction of the mine facilities will remove approximately 410 acres of mixed forest community. During operations, the TDF may attract birds and bats. The facility will implement a mitigation plan to prevent wildlife from exposure to detrimental materials. Upon closure, a portion of the site will return to forest coverage and the 350-acre TDF will be closed with a grass cover. This represents a small change within the region.

Plant and animal species of special concern have been studied and documented elsewhere. The impacts to terrestrial wildlife are anticipated to be minimal. To address unanticipated encounters with listed plant species within the Project Area, appropriate permits and relocation will be pursued. In contrast, the operation will implement a monitoring and response plan for preventing the spread of invasive species.

### 20.1.10 Air Quality

Impacts are expected during facility construction, operations, and reclamation. Particulate matter is primarily generated from vehicle traffic and material handling activities. Maintaining compliance with the air permit, the malfunction abatement plan, and the fugitive dust control plan is the most effective method to minimize air quality impacts.

### 20.1.11 Cultural and Archaeological Resources

No cultural or historical resources have been identified within the project boundaries or in the surrounding area. This conclusion is based on consultation with Ottawa National Forest Heritage Program Manager / Forest Archaeologist; an anthropologist / Archaeologist retained by ORUSC; and interaction with the Lac Vieux Desert (LVD) Band of Chippewa. The LVD have not indicated the presence of cultural or historical resources in the Project Area. Phase I Archaeological Surveys were completed over the Project site (AVD, 2009), with additional efforts undertaken in support of the Mining Permit Amendment Application (AVD, 2012, 2018).

The area has been used for commercial timber production and contains old railroad grades and forest communities. A test mine was identified (ca. 1950s); however, attributed with no historical significance. It is anticipated that the Project will have no impact on cultural, historical, and archaeological resources. Should mining activity unearth archaeological, historic, or cultural artifacts, relevant activities shall be immediately suspended, and the Oil, Gas, and Minerals Division (OGMD) of Michigan Department of Environment, Great Lakes, and Energy (EGLE)[1] will be notified.

### 20.1.12 Impacts to Surrounding Public and Recreational Areas

The original and subsequent EIA reports consider the land use in areas surrounding the Project. This included evaluating potential Project aesthetics, acoustic, seismic impacts, and impacts to parks and recreational areas in the vicinity. The studies and evaluations indicate minimal impacts to these resources from the Project.

## 20.2 Mine Waste Management

Mine waste refers to tailings and waste rock. As tailings are generated in the mill, they will be transported via slurry pumping to the TDF. The TDF will be built in three (3) stages using downstream construction methods and at full build-out, will cover approximately 320 acres. The base will be lined and constructed over a 0.3 m clay layer, preventing seepage to the groundwater and facilitating leachate collection.

TDF design, construction, and operation will be based on sound engineering practices with rigorous monitoring to maintain structural integrity and demonstrate environmental compliance. Additional details can be found in Section 18.11.

Waste rock management needs are minimal at the site. Except for the box cut and removal of an existing waste rock pile on the north side of the facility, mining will generate very little waste rock. Any waste rock excavated during the box cut will be stored in the ore stockpile building, disposed in the TDF, or otherwise managed and disposed of according to waste management regulations.

## 20.3 Environmental Monitoring

Monitoring a variety of media throughout construction, operations, and closure is the foundation of identifying Project-related impacts on the environment. With a broad data set of baseline conditions, ongoing monitoring is the basis of identifying and evaluating actual impacts. The Mining and Wetland Permits contain several conditions addressing updating baseline data, summarized in Section 20.5.

[1] EGLE was previously named Michigan Department of Environmental Quality (MDEQ). The agency name changed in 2019.

Once Project activities begin, monitoring data evaluation will indicate environmental changes and whether they are attributable to the Project. Actions may be taken to mitigate or remediate impacts or modify operations. Monitoring is addressed in all permits including the Mining Permit and includes rigorous reporting requirements. Detailed plans are needed in place, several of which require agency approval prior to commencement of certain activities. Monitoring includes data collection and evaluation of:

- Surface Water – quality and flow / presence;
- Groundwater – quality and flow / presence;
- Wetlands – a variety of metrics;
- Flora and Fauna;
- Facility Water quality and flows;
- Integrity of liners, berms, dam structures, and basins;
- Air quality;
- Geochemical changes in waste materials.

## 20.4 Water Management During and After Operations

Water management is more fully described in Section 18. Figure 18.23 shows the Process Water Balance Schematic. From the environmental perspective, facility water to manage includes:

- Contact water to be stored in the Event Pond:
    - Mine water as the mine is developed.
    - Storm water from within the contact area.
    - Leachate collected from ore, concentrate, waste rock, tailings piles.
    - Mill water and excess water from the tailings basin.
- Water from the Event Pond will be transferred to the TDF as needed, which can supply the Process Mill as needed.

Facility water demands will increase as mining operations continue. As described in Section 18, precipitation and run-off will be collected in the TDF for use in operations. Once mining and milling commence, the facility water balance will be affected by water exiting the system through concentrate, precipitation and evaporation, and other entering and exiting water flows. Excess water in the system, beyond which is desirable to store, will be treated in the wastewater treatment plant (WWTP) prior to discharge to the West Branch Namebinag Creek. It is anticipated that the WWTP will start treating and discharging to the creek at the beginning of Year 5 into operations.

## 20.5 Permit Requirements and Status

Significant permits held by Copperwood are in place to commence construction, and operations and are listed in Table 20.1. Comments are provided for each entry regarding the permit status. Additional less significant permits and approvals will be needed; however, they are not listed because the level of effort to obtain approval is not highly consequential.

Project power will be accessed from the grid with a new connection to the Norrie substation located in Ironwood. The 40 km-line and site main substation will be designed, supplied, built, owned, and operated by the Utility company. For the first five (5) years of operation prior to full grid connection, three (3) natural gas generators will operate, supplied by compressed natural gas delivered by truck. Once grid power is available, the generators will be used for emergency and/or supplemental power. The generators will be incorporated into the air permit in 2023, prior to current air permit expiration.

**Table 20.1: Status of Significant Project Permits**

| Permit / Issuing Agency | Issue and Expiration Dates | Comments |
|---|---|---|
| Mining Permit<br>MP 01 2012 | Issued April 30, 2012.<br>Amended: February 7, 2013, and December 14, 2018; no expiration. | This permit extends through operations and post-closure. Other state permits must be in place and active to support the Mining Permit. An amendment will be sought to authorize mining east of the Porcupine Mountains Wilderness State Park. |
| Air Permit – Permit to Install 180-11A / EGLE | Issued Nov 26, 2018.<br>Three (3) extensions have been granted, to October 16, 2023. | This air permit enables site activities prior to its expiration. A renewal application containing information supporting the power plant (Section 18.9).will be submitted prior to October 16, 2023. |
| NPDES Permit<br>MI0058969 / EGLE | A renewed permit has been issued, expiring October 1, 2024. | A renewal application must be submitted by April 4, 2024. |
| Wetland Permit<br>WRP013721 | Issued Oct. 16, 2018.<br>Expires: October 16, 2023. | Permitted activities include fill and excavation of wetlands on site. Activities are underway at the site. |
| Dam Safety Permit<br>WRP013851 v. 1 / EGLE | Issued: Nov. 9, 2018.<br>Expired: Nov. 9, 2020.<br>Extension Request to: Nov. 9, 2022. | Permit issued for conceptual approval only. Final engineering design plans and specifications must be approved by WRD Dam Safety Program. |

| Permit / Issuing Agency | Issue and Expiration Dates | Comments |
|---|---|---|
| Water Intake Structure Permit (Section 10 Federal Rivers and Harbors Act) / USACE | Application: March 26, 2018. | This permit is no longer needed. |
| Power Supply Permit - undetermined | Power plant permitting being addressed under Air Permit. Power line installation – TBD. | Natural gas generators on site will need to be incorporated into current air permit. Power line installation to Norrie substation, Ironwood MI will be permitted by utility. |

### 20.5.1 Summary of Construction Readiness Permit Requirements

The Mining Permit, Dam Safety Permit, and Wetland Permit have requirements that Highland Copper Company inc. will need to address prior and during construction. This section provides high level summary of those known permit conditions; it is not comprehensive.

#### 20.5.1.1 Financial Assurance

The Mining Permit and Wetland Permit have financial assurance requirements that must be in place to address third party reclamation and environmental protection; and project specific wetland establishment and responsibilities. Satisfactory financial assurance must be in place prior to site activities.

#### 20.5.1.2 Monitoring Updates

Several environmental monitoring plans must be prepared and approved before an update of environmental baseline data can proceed. The plans address surface and groundwater, aquatics, and subsidence. Additional surface water stations, groundwater monitoring wells, and other control or reference stations are required with commensurate baseline data. These control stations will be needed to evaluate whether changes over time near the Project are attributable to the Project or natural variability.

#### 20.5.1.3 Construction and Installation Requirements

Engineering and design reports will need to be submitted for approval. These include the following:

- Tailings Disposal Facility engineering design and specification package must be approved by EGLE before proceeding with TDF construction.
- Before tailings, ore, waste rock, and overburden can be placed, design certifications of liners, covers, and leachate collection systems must be submitted for approval.

- Wastewater Treatment Plant engineering design and specification package must be approved by EGLE before proceeding with WWTP construction.
- Design plans for all aboveground storage tanks containing flammable or combustible materials must be approved prior to installation. Once installed, the tanks must be inspected and approved.
- Records, Reports, Plans, and Notifications: a variety of notices, plans, and submittals are necessary before commencing certain activities.
- Wetland Related Requirements: certain agreements are stipulated according to the Wetland Permit including those directly supporting the mitigation of associated wetlands. With the establishment of a created wetland site and preservation of additional wetlands, easements are needed and additional local improvements.

## 20.6 Bond Requirements

The general provisions of NREPA Part 632 Nonferrous Metallic Mineral Mining rules define financial assurance as an assurance instrument or statement of financial responsibility provided by an operator to ensure compliance with the act, rules, permit conditions, instructions, or orders of the department (R 425.102(n)). Specific requirements on financial assurance are provided in both the Mining and Wetlands Permits.

### 20.6.1 Mining Permit

Financial assurance must be established before site activities can commence. Financial assurance extends through post-closure monitoring. Regular revisions are accommodated.

Financial assurance applies to all mining and reclamation operations subject to the Mining Permit. It is intended to cover the cost of administering and the hiring of a third party to implement reclamation and environmental protection should the mine owner be unable to do so. Subject to adjustment prior to commencement of construction, the current financial assurance at the closure stage of the Project is US$38 million. The establishment of financial assurance at the initiation of activities can be negotiated to be consistent with progress at the site. Mechanisms for partial release can be exercised as the site is reclaimed. Several financial assurance instruments are satisfactory, subject to agency approval.

### 20.6.2 Wetland Permit

The permittee shall provide a surety bond or letter of credit to EGLE for approximately US$4.7 million. These financial resources are to ensure that the stream mitigation is completed, created wetlands constructed, the stewardship agreement and endowment are complete, the conservation easements are recorded, baseline conditions are documented, and other mitigation actions are performed. Additional bonding will likely be part of additional wetland permitting needed for powerline installation should this alternative be pursued.

## 20.7 Potential Social or Community Requirements and Plans

The Project is in the western part of Michigan's Upper Peninsula (UP), just north of the city of Wakefield. The Wisconsin border is approximately 20 miles west of the Project. The region, including the UP and northern Wisconsin, is rural, sparsely populated, with modest economic activity including logging, farming, and tourism. The region is historically connected to mining, with the active Eagle Mine and Humboldt Mill to the east and the closed White Pine Mine (also owned by Highland Copper) to the east nearby.

At the time of the original permit applications (Orvana, 2011), a study was prepared for Orvana by the Labovitz School of Business and Economics describing the Economic Impact of the proposed Copperwood Project, (Labovitz, 2011). Although a decade has passed since the report was issued, the conclusions of the report remain valid: a significant positive economic impact is anticipated from the Project. This includes several hundred direct jobs on the Project and an additional several hundred indirect and induced jobs (those needed to support the population growth stemming from those directly employed). An increase in local tax revenues will positively impact the community, providing additional resources for local needs. Public information will be issued via press releases, local correspondence, and stakeholder engagement and continue throughout operations. The economic impacts are anticipated to extend into surrounding counties including those in northern Wisconsin.

## 20.8 Expected Mine Closure and Costs

Reclamation takes place to the degree possible during operations such establishing vegetation and erosion controls in disturbed areas and removing selected equipment as appropriate. Ongoing reclamation supports environmental protection and reduces the scope of mine closure. A closure plan is included in the permit application and the Mining Permit addresses specific reclamation and closure requirements. At the end of mining operations, mine closure begins in earnest and addresses such activities as:

- Box cut reclamation.
- Equipment and materials evacuation from the mine workings. At the appropriate time, the mine will be flooded with fresh water.
- Demolition and/or removal of the plant, structures, and equipment at the surface.

- Reclamation of the TDF: an engineered cover system will be installed. TDF drainage will be monitored, collected, and treated until flows diminish.
- WWTP removal at the appropriate time.
- Revegetation of disturbed areas.
- Abandonment of monitoring wells in accordance with EGLE requirements.
- Long-term monitoring and reporting of environmental conditions.

The intent of closure is to establish a self-sustaining ecosystem similar to pre-mining conditions while maintaining environmental protections and land uses. Reduction in financial assurance amounts can be accomplished as mine closure goals are accomplished. With the updated FS report, mine closure costs have been updated, see Section 21.4.

# 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 USD 425.1M and has an accuracy within a range of -10% / +15%., in line with a AACE Class 3 estimate. A summary of the capital expenditures is presented in Table 21.1.

Labour 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 GMSs 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 labour 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
- Labour rates based on quotations received from contractors, labour 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 | 31,778,937 |
| 200 – Power and Electrical | 42,459,628 |
| 300 – Water | 46,198,148 |
| 400 – Mobile Equipment | 24,932,209 |
| 500 – Mining | 51,172,467 |
| 600 – Process Plant | 105,502,312 |
| 700 – Construction Indirects | 51,028,250 |
| 800 – General Services | 25,377,352 |
| 900 – Pre-production, Start-up, Commissioning | 7,887,547 |
| 990 – Contingency | 37,644,730 |
| **Grand Total** | **425,131,435** |

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 Q2-2022 with a few updates done between Q3 and Q4 2022 (Reagents, Structural Steel. Underground Mining Ground support steel).

The estimates include earthworks, construction material, equipment, and labour. Earthworks will be performed by regional contractors when possible.

### 21.1.1 <u>Infrastructure</u>

A CAPEX summary for infrastructure is presented in Table 21.2. The detailed description of infrastructure and roads are presented in Section 18.

**Table 21.2: Infrastructure CAPEX**

| Capital Expenditures | USD |
|---|---|
| **110 – Roads** | **8,305,708** |
| 111 – Main Access Road | 2,584,930 |
| 112 – Site Roads | 343,866 |
| 114 – Fencing | 178,390 |
| 117 – Employee Parking Lot | 477,843 |
| 119 – Road 519 improvements | 4,720,679 |
| **120 – Workshops / Storage** | **993,186** |
| 123 – Plant Workshop & Stores | 435,091 |
| 124 – Reagents Storage Building | 17,730 |
| 125 – Explosives Plant / Magazine | 540,365 |
| **130 – Support Buildings** | **21,348,933** |
| 131 – Workshop, Warehouse, Lunchrooms & Dry Building | 10,014,274 |
| 133 – Mill Office (Construction Office) / Met Lab / Control Room | 3,372,752 |
| 135 – Main Gatehouse | 545,800 |
| 138 – Off-Site Facilities – Transload Building & facilities | 4,726,866 |
| 139 – Administrative Building (considered on site) | 2,689,240 |
| **160 – Laboratories** | **50,984** |
| 161 – Assay, Environmental Laboratory | 50,984 |
| **170 – Fuel Systems** | **1,038,396** |
| 170 – Fuel Systems Storage | 291,630 |
| 171 – Mining Equipment Fuel Storage | 746,766 |
| **180 – Other Facilities** | **41,730** |
| 183 – Topsoil Storage Area | 41,730 |
| **Grand Total** | **31,778,937** |

### 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 infrastructure is detailed in Section 18 of this Report.

**Table 21.3: Power Supply and Electrical Capital Expenditures**

| Area | USD |
|---|---|
| **210 – Main Power Generation** | **8,309,349** |
| 216 – Main Sub-Station | 451,181 |
| 217 – Emergency Power Generation (Surface) | 7,858,168 |
| **220 – Process Plant Electrical Rooms** | **13,992,404** |
| 221 – Process Plant E-Room | 9,171,473 |
| 223 – U/G Main E-Rooms at Portal | 4,170,292 |
| 224 – U/G Heating Ventilation Intake E-Room | 180,152 |
| 225 – Tailings E-Room | 470,487 |
| **240 – Site Power Distribution** | **1,563,380** |
| 241 – Site Powerlines | 1,563,380 |
| **250 – U/G Power Distribution** | **6,296,258** |
| 251 – U/G Sub-Station | 2,126,094 |
| 255 – U/G Distribution | 4,170,164 |
| **260 – IT & Site Communications (surface)** | **8,383,846** |
| 261 – IT & Site Communications (surface) | 8,383,846 |
| **270 – U/G Communications Network** | **2,193,331** |
| 270 – U/G Communications Network (Mining Eq. Monitoring) | 350,000 |
| 271 – U/G Communications Network | 1,843,331 |
| **280 – Automation Network** | **323,000** |
| 281 – Automation Network | 213,000 |
| 282 – Process Monitoring System | 110,000 |
| **290 – IT Network & Fire Detection** | **1,398,060** |
| 293 – Fire Detection Network | 1,263,260 |
| 295 – Server Room | 134,800 |
| **Grand Total** | **42,459,628** |

### 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 (2) other phases are planned for construction and will be commissioned for Y3 and Y5, respectively, and therefore are included in sustaining expenditures. Capital costs include earthworks, concrete, structure steel, mechanical, electrical and instrumentation equipment and labour.

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 fire water estimate includes the fire pumps, the distribution network within the processing plant and the other buildings (truckshop, admin building, lab).

A CAPEX summary for water is presented in Table 21.4.

**Table 21.4: Tailings & Water Capital Expenditures**

| Area | USD |
|---|---|
| **310 – Raw Water Supply & Potable Water** | **6,472,054** |
| 311 - Process Water | 2,768,291 |
| 312 - Potable Water Treatment | 1,088,945 |
| 317 - Surface Water Recovery | 2,614,817 |
| **320 – Reclaim Water** | **5,789,084** |
| 321 - Reclaim Water System | 3,013,126 |
| 322 - Reclaim Pipeline | 1,772,884 |
| 322 - Gland Water | 1,003,074 |
| **330 – Water Management** | **1,410,471** |
| 331 - Water Management Surface | 1,410,471 |
| **340 – Tailings Disposal Facility** | **23,319,704** |
| 341 - TDF Roads | 42,625 |
| 342 - TDF Main Dams | 18,370,893 |
| 343 - TDF Water Pumphouse & Seepage Tank | 200,000 |
| 344 - TDF Basin | 223,828 |
| 346 - TDF Pipeline | 4,482,358 |

| Area | USD |
|---|---|
| **350 – Surface Water Management** | **4,637,331** |
| 351 - Ponds | 672,044 |
| 352 - Mitigation Area Works | 1,897,033 |
| 354 - Stream Relocation | 2,068,254 |
| **370 - Fire Water** | **3,356,310** |
| 371 - Pumping System & Reservoir | 1,424,568 |
| 372 - Fire Water Distribution | 1,931,742 |
| **380 – Domestic Sewage** | **1,213,196** |
| 381 - Sewage Treatment System | 1,213,196 |
| **Grand Total** | **46,198,148** |

### 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 | USD |
|---|---|
| **410 – U/G Mining Equipment & Maintenance** | **21,752,588** |
| 412 – U/G Mining Equipment | 16,096,050 |
| 414 – U/G Support Equipment | 5,656,538 |
| **420 – Construction Vehicles and Equipment** | **321,621** |
| 422 – Light Vehicles and Other Equipment | 321,621 |
| **430 – Surface Mobile Equipment** | **2,858,000** |
| 431 – Surface Mobile Equipment | 2,858,000 |
| **Grand Total** | **24,932,209** |

### 21.1.5 Mine Infrastructure

Mine infrastructure CAPEX includes the portal excavation, installation of multi-plate culverts, and backfill. Hauling starts outside of the ramp to the ore stockpile. Mine development includes labor, Ground Support, and consumables to complete the drifts to reach the 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.

Mine Development costs during pre-production are also parts of the Mine Infrastructure CAPEX.

A summary of the CAPEX for mine infrastructure is presented in Table 21.6.

**Table 21.6: Mine Infrastructure Capital Expenditures**

| Area | USD |
|---|---|
| **510 – Surface Mine Infrastructure** | **1,596,053** |
| 512 – Haul Road | 268,340 |
| 517 – Ore Stockpile Pad | 1,327,713 |
| **520 – U/G Mine Infrastructure** | **24,892,392** |
| 522 – Portal (Box-cut) | 6,670,386 |
| 526 – Level Development | 17,212,213 |
| 527 – Underground compressors | 956,551 |
| 529 – U/G Mine Refuge / Lunchroom | 53,243 |
| **530 – Ventilation Raise & Escapeways** | **5,165,817** |
| 531 – Collar & Excavation | 2,403,344 |
| 533 – Power Supply / HVAC | 1,976,514 |
| 535 – Building, Gas supply and civil works | 785,959 |
| **550 – U/G Mine Dewatering System** | **750,750** |
| 551 – U/G Mine Dewatering System | 750,750 |
| **570 – U/G Explosives Storage** | **17,500** |
| 571 – U/G Explosive Storage Facility | 17,500 |

| Area | USD |
|---|---|
| **580 – U/G Conveying / Crushing System** | **18,749,955** |
| 581 – Feeder breakers and Primary conveyors | 12,785,824 |
| 583 – Main conveyor to Surface | 5,964,131 |
| **Grand Total** | **51,172,467** |

### 21.1.6 Process Plant and Related Infrastructure

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 labour.

Quantities for the earthwork, concrete, structure, piping, electrical, instrumentation and architecture material take-offs were estimated by GMS. The unit rates for material were estimated by GMS. 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 601. Scope includes the haul ramps to access the feed hopper and finishes at the tailings pumps. All related plant auxiliary services and reagents are also included.

**Table 21.7: Processing Capital Expenditures**

| Processing Capital Costs | USD |
|---|---|
| **600 – General** | **11,009,253** |
| 601 – Process Plant Building | 10,245,488 |
| 603 – Buried Services | 763,766 |
| **610 – Ore Handling** | **14,637,589** |
| 610 – Ore Handling Area General | 2,308,221 |
| 611 – Stockpile Hopper, Feeder and Conveyor | 3,365,702 |
| 613 – Crushed Ore Bins / Reclaim | 6,252,136 |
| 615 – Crushed Ore Feeders & Sag Mill Feed conveyor | 2,711,531 |
| **620 – Grinding** | **29,802,289** |
| 620 – Grinding Area General | 9,054,996 |
| 621 – Grinding & Cyclopak | 20,228,008 |

| Processing Capital Costs | USD |
|---|---|
| 622 – Media Storage | 519,286 |
| **630 – Flotation / Regrind Circuit** | **25,231,017** |
| 630 – Flotation / Regrind Circuit General | 6,475,655 |
| 631 – Conditioning Tank | 2,160,845 |
| 632 – Rougher Cells | 4,868,399 |
| 633 – Scavenger / 1st Cleaner Cells | 4,481,844 |
| 634 – 2nd Cleaner Cells | 1,815,357 |
| 635 – 3rd Cleaner Cells | 1,087,705 |
| 636 – Cyclone & Regrind | 4,341,213 |
| **640 – Tailings** | **3,045,745** |
| 642 – Flotation Tailings | 3,045,745 |
| **650 – Copper Concentrate Filtration, Thickening & Handling** | **8,303,008** |
| 650 – Cu Concentrate Filtration, Thickening & Handling General | 583,552 |
| 651 – Cu Concentrate Thickening | 1,786,599 |
| 652 – Cu Concentrate Filtration | 4,569,190 |
| 653 – Load-Out, Packaging Concentrate | 1,363,667 |
| **670 – Reagents** | **6,945,580** |
| 670 – Reagents General | 2,161,412 |
| 671 – Lime Circuit | 1,213,713 |
| 672 – MIBC | 595,280 |
| 673 – PAX | 1,155,805 |
| 674 – NaHS | 370,656 |
| 675 – Na2SiO3 | 667,149 |
| 676 – Flocculant | 707,056 |
| 677 – Anti-Scalant | 74,508 |
| **680 – Plant Services** | **6,527,829** |
| 680 – Plant Services General | 3,976,182 |
| 681 – Compressed Air | 1,021,214 |
| 682 – Low Pressure Compressed Air | 1,530,434 |
| **Grand Total** | **105,502,312** |

### 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 equipment rentals, site power generation and fuel.

Construction Indirect Costs are presented in Table 21.8.

**Table 21.8: Construction Indirect Capitals**

| Construction Indirects | USD |
|---|---|
| **710 – Engineering, CM, PM** | **30,977,982** |
| 711 – Site CM Staff and Consultants | 8,438,651 |
| 715 – External Engineering | 13,343,583 |
| 716 – Surveying | 1,852,861 |
| 717 – QA/QC | 2,180,533 |
| 718 – Commissioning and Vendor's Rep | 320,000 |
| 719 – Induction / Travel / Visas / Working Permits | 4,842,355 |
| **720 – Construction Facilities & Services** | **20,050,267** |
| 722 – Construction Temporary Services | 8,926,119 |
| 723 – Concrete Batch Plant | 216,714 |
| 726 – Construction Offices | 1,115,717 |
| 727 – Construction Tools / Consumables | 3,957,578 |
| 728 – Construction Temp Power Distribution | 1,838,864 |
| 729 – Construction Equipment Rentals | 3,995,275 |
| **Grand Total** | **51,028,250** |

### 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 (General Management)
- Supply Chain

- HR & Training
- Health and Safety
- ESR
- Security
- IT
- Accounting and Finance

All freight is estimated based on a percentage of the Equipment and Material costs. As of Q2-2022, this is in line with 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 | USD |
|---|---|
| **810 – Departments** | **12,710,292** |
| 811 – General Administration | 1,126,161 |
| 812 – Supply Chain | 1,100,175 |
| 813 – HR & Training | 1,318,885 |
| 814 – ESR | 498,195 |
| 815 – Health & Safety | 1,941,606 |
| 816 – Security | 510,753 |
| 818 – IT | 3,773,242 |
| 819 – Accounting & Financing | 2,441,275 |
| **820 – Logistics / Taxes / Insurance** | **12,073,060** |
| 821 – Freight | 11,469,410 |
| 822 – Customs, Taxes & Duties | 362,190 |
| 823 – Insurance (Freight) | 241,460 |
| **830 – Pre-Production Operating Expenses** | **594,000** |
| 832 – International Travel | 339,000 |
| 834 – Roads Maintenance / Snow Removal | 255,000 |
| **Grand Total** | **25,377,352** |

### 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 | USD |
|---|---|
| **950 – Process Plant Pre-Prod. & Commissioning** | **6,587,155** |
| 955 – Process Plant Mgmt. and Training | 3,240,578 |
| 956 – Process Plant Commissioning | 1,446,000 |
| 958 – First Fill | 572,259 |
| 959 – Commissioning Spares | 1,328,318 |
| **960 – Capital Spares / Stay in business Spares** | **1,300,392** |
| 961 – Capital Spare Parts | 380,000 |
| 962 – Capital Spares (Major Components) for Mining Equipment Fleet | 920,392 |
| **Grand Total** | **7,887,547** |

A 9.7% contingency on all costs was included for a total of USD 37.6M.

## 21.2 Sustaining Capital

Sustaining capital of USD 269.9M 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. Beginning of Stage 2 construction occurs in Y2 while Stage 3 construction starts in Y4.

The effluent water treatment plant is constructed in Y5 to be operational since Stage 3 of the TDF is used for tailings disposal that same year, 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 | 54.77 | 2.17 | 0.08 |
| Water Treatment Plant | 17.11 | 0.68 | 0.03 |
| Mine Equipment Purchases | 141.56 | 5.61 | 0.21 |
| Mine Development Expenditures | 33.11 | 1.31 | 0.05 |
| Others (IT, Electrical) | 23.35 | 0.93 | 0.04 |
| **Total Sustaining CAPEX** | **269.89** | **9.77** | **0.37** |

**Table 21.12: Sustaining Capital Costs**

| Sustaining CAPEX (USD M) | Total | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mine Equipment** | | | | | | | | | | | | |
| Jumbo Drills | **4.33** | - | - | - | - | 3.46 | 0.87 | - | - | - | | |
| Bolters | **13.42** | 8.05 | 1.79 | 0.89 | - | - | 1.79 | 0.89 | - | - | | |
| LHD 10t | **6.73** | 3.85 | 1.92 | 0.96 | - | - | - | - | - | - | | |
| LHD 8t | **0.77** | 0.77 | - | - | - | - | - | - | - | - | | |
| Scaler | **1.74** | 1.74 | - | - | - | - | - | - | - | - | | |
| Dev. Truck | **0.92** | 0.92 | - | - | - | - | - | - | - | - | | |
| Loading Point & Rock Breaker | **2.74** | 1.83 | 0.91 | - | - | - | - | - | - | - | | |
| Lube Trucks | **0.42** | - | - | - | - | - | - | 0.42 | - | - | | |
| Flatbed Trucks | **0.50** | 0.50 | - | - | - | - | - | - | - | - | | |
| Scissor Lift | **0.84** | 0.84 | - | - | - | - | - | - | - | - | | |
| Grader | **0.40** | 0.40 | - | - | - | - | - | - | - | - | | |
| Tractor & ATV | **1.13** | 0.86 | - | - | - | 0.27 | - | - | - | - | | |
| Cable Bolt Drill | **0.11** | 0.11 | - | - | - | | - | - | - | - | | |
| Equipment Major Components | **54.48** | 4.50 | 5.86 | 5.97 | 6.12 | 6.55 | 6.36 | 6.36 | 6.30 | 6.47 | | |
| Continuous Miner | **9.67** | 9.67 | - | - | - | - | - | - | - | - | - | - |
| **Sub-Total Mine Equipment** | **98.21** | **34.04** | **10.48** | **7.82** | **6.12** | **10.28** | **9.01** | **7.68** | **6.30** | **6.47** | | |
| **Material Handling** | | | | | | | | | | | | |
| Conveyor Purchases | **38.47** | 12.48 | 9.89 | 6.34 | - | 2.40 | 2.40 | - | - | 4.95 | | |
| Conveyor & Rock Breaker Moves | **4.88** | 0.04 | 0.56 | 0.54 | 0.54 | 1.61 | 0.54 | 0.54 | 0.54 | - | | |
| **Sub-Total Materials Handling** | **43.35** | **12.52** | **10.45** | **6.87** | **0.54** | **4.01** | **2.94** | **0.54** | **0.54** | **4.95** | | |

| Sustaining CAPEX (USD M) | Total | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Sustaining Mining Capital | 141.56 | 46.57 | 20.93 | 14.70 | 6.65 | 14.29 | 11.95 | 8.21 | 6.83 | 11.42 | | |
| Mining Development | 33.11 | 10.08 | 2.15 | 2.03 | 6.21 | 7.42 | 5.21 | - | - | - | | |
| Tailings Facility Expansion | 54.77 | 9.19 | 9.88 | 15.36 | 20.34 | - | - | - | - | - | | |
| Water Treatment Plant | 17.11 | - | - | - | - | 17.11 | - | - | - | - | | |
| Other Sustaining Capex | 23.35 | 4.00 | 7.13 | 0.23 | 5.77 | 0.84 | 5.38 | - | - | - | | |
| Total Sustaining CAPEX | 269.89 | 69.84 | 40.10 | 32.32 | 38.97 | 39.65 | 22.55 | 8.21 | 6.83 | 11.42 | | |

### 21.3 Closure Costs and Salvage Value

The closure costs are estimated to USD 37.15M net of USD 20.86M 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 2037 to 2039).

**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 | 29.69 | 205,000 | 6,086 |
| TDF Disposal Area Reclamation Stage 2 | sq.m | 29.69 | 295,000 | 8,759 |
| TDF Disposal Area Reclamation Stage 3 | sq.m | 29.69 | 471,118 | 13,987 |
| **Sub-Total** | | | 971,118 | **28,832** |
| Site Closure & Reclamation | | | | |
| Place and Compact Soil Cover | cu.m | 2.00 | 201,957 | 404 |
| Place and Hydroseed Topsoil | sq.m | 2.15 | 2,330,000 | 5,010 |
| Structural Steel Demolition | tonnes | 600 | 2,871 | 1,722 |
| Concrete Demolition | tonnes | 8.00 | 40,190 | 322 |
| Concrete Disposal | tonnes | 2.00 | 40,190 | 80 |
| Modular Building Removal | sq.m | 50 | 230 | 11 |
| Mechanical Pipelines | lot | 574,138 | 1 | 574 |
| Electrical Distribution | lot | 574,138 | 1 | 574 |
| Tank Removal and Disposal | lot | 116,181 | 1 | 116 |

| Closure Cost Estimate | Unit | Unit Price | Qty | Cost ($k) |
|---|---|---|---|---|
| Admin. Support | % | 15% | 1 | 1,322 |
| **Sub-Total** | | | | **10,136** |
| General / Reclamation | lot | 9,722,100 | 1 | 9,722 |
| Salvage Value | | | | |
| 120 – Workshops / Storage | lot | 248,296 | -1 | (248) |
| 130 – Support Facilities | lot | 5,337,233 | -1 | (5,337) |
| 160 – Laboratory | lot | | -1 | - |
| 210 – Main Power Generation | lot | | -1 | - |
| 220 – Process Plant Electrical Rooms | lot | 4,897,341 | -1 | (4,897) |
| 310 – Raw Water Supply & Potable Water | lot | 1,618,545 | -1 | (1,619) |
| 400 – Mobile Equipment | lot | 4,192,459 | -1 | (4,192) |
| 430 – Surface Mobile Equipment | lot | 635,924 | -1 | (636) |
| 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** | | | | **(20,863)** |
| Post Closure Monitoring (DCF 5%) | lot | 3,924,449 | 1 | 3,924 |
| MDEQ Admin Oversight | % | 5.0% | 1.00 | 5,393 |
| **Total Cost** | | | | **37,145** |

## 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 five (5) years in

Table 21.15. The OPEX by year is presented in Table 21.16. The LoM unit operating cost is estimated at USD 1.83/lb of payable copper and lower at USD 1.56/lb for the first five (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 | 136 | 5.37 | 0.20 | 11.2% |
| Mining | 606 | 24.02 | 0.92 | 50.0% |
| Processing | 369 | 14.63 | 0.56 | 30.4% |
| General and Administration | 102 | 4.03 | 0.15 | 8.4% |
| **Total** | **1,212** | **48.05** | **1.83** | **100%** |

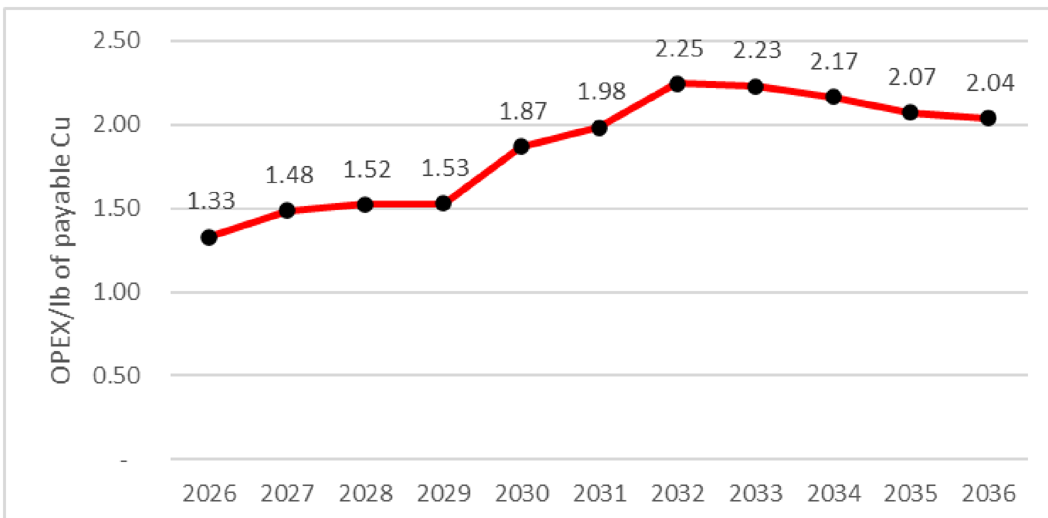
**Table 21.15: First 5-year Operating Cost Summary**

| First 5-Year OPEX | Total Cost ($M) | Unit Cost ($/tonne milled) | Unit Cost ($/payable lb) | % |
|---|---|---|---|---|
| Royalties | 71 | 6.09 | 0.21 | 13.5% |
| Mining | 239 | 20.35 | 0.70 | 45.0% |
| Processing | 172 | 14.71 | 0.50 | 32.5% |
| General and Administration | 48 | 4.11 | 0.14 | 9.0% |
| **Total** | **531** | **45.26** | **1.55** | **100%** |

**Table 21.16: Annual Operating Costs**

| OPEX Summary ($M) | Total | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2035 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Royalties | 135.59 | 11.78 | 15.90 | 15.74 | 15.13 | 12.81 | 12.49 | 11.43 | 11.47 | 11.82 | 12.07 | 4.95 |
| Mining | 606.05 | 28.43 | 51.76 | 52.33 | 48.98 | 57.06 | 63.82 | 70.74 | 70.09 | 70.95 | 66.55 | 25.34 |
| Processing | 369.09 | 26.89 | 35.72 | 36.75 | 36.75 | 36.36 | 36.27 | 36.27 | 36.27 | 36.27 | 36.27 | 15.26 |
| G&A | 101.66 | 7.30 | 10.34 | 10.34 | 10.34 | 9.88 | 8.90 | 10.12 | 10.12 | 10.12 | 9.93 | 4.29 |
| Total | **1,212.39** | **74.39** | **113.73** | **115.16** | **111.20** | **116.11** | **121.48** | **128.55** | **127.94** | **129.15** | **124.83** | **49.85** |
| Unit Cost ($/t milled) | **48.06** | **41.13** | **47.21** | **46.06** | **44.48** | **46.45** | **48.59** | **51.42** | **51.18** | **51.66** | **49.93** | **49.43** |
| Unit Cost ($/pay. lb Cu) | **1.83** | **1.33** | **1.48** | **1.52** | **1.53** | **1.87** | **1.98** | **2.25** | **2.23** | **2.17** | **2.07** | **2.04** |

**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 (10) 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 | **19.82** | 0.79 | 0.03 | 3.3% |
| Production Drilling (Incl. Continuous Miner) | **65.77** | 2.61 | 0.10 | 10.9% |
| Blasting | **78.66** | 3.12 | 0.12 | 13.0% |
| Stope Piping, Scaling & Serv. | **36.91** | 1.46 | 0.06 | 6.1% |
| Ground Support | **154.93** | 6.14 | 0.23 | 25.6% |
| Hauling | **44.89** | 1.78 | 0.07 | 7.4% |
| Mine Services and Const. | **64.38** | 2.55 | 0.10 | 10.6% |
| Mechanical Maintenance | **54.46** | 2.16 | 0.08 | 9.0% |
| Electrical Maintenance | **58.37** | 2.31 | 0.09 | 9.6% |
| Technical Services | **27.86** | 1.10 | 0.04 | 4.6% |
| **Total Mining Cost** | **606.05** | **24.02** | **0.92** | **100%** |

The four (4) main costs for mining are ground support, blasting, production drilling (which include the continuous miner operating costs) and Mine services. These four (4) main costs represent 60% of all mining costs.

**Table 21.18: Annual LoM Mining OPEX**

| Mining Costs ($M) | Total | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine Supervision | **19.82** | 1.22 | 2.02 | 2.02 | 1.87 | 1.86 | 1.94 | 2.10 | 2.10 | 2.10 | 1.93 | 0.66 |
| Production Drilling (incl. continuous miner) | **65.77** | 1.83 | 3.50 | 3.46 | 3.13 | 5.49 | 8.07 | 9.13 | 9.14 | 9.13 | 9.09 | 3.80 |
| Blasting | **78.66** | 1.84 | 3.48 | 3.38 | 3.02 | 6.73 | 9.60 | 11.57 | 11.48 | 11.56 | 11.57 | 4.43 |
| Stope Piping, Scaling & Serv. | **36.91** | 1.70 | 3.29 | 3.32 | 3.12 | 3.29 | 3.60 | 4.31 | 4.26 | 4.40 | 4.12 | 1.50 |
| Ground Support | **154.93** | 7.21 | 14.28 | 14.35 | 13.27 | 13.99 | 15.14 | 17.53 | 17.27 | 17.93 | 17.40 | 6.56 |
| Hauling | **44.89** | 2.61 | 4.18 | 4.45 | 4.96 | 5.92 | 5.01 | 4.04 | 4.03 | 4.19 | 4.02 | 1.48 |
| Mine Services and Const. | **64.38** | 3.98 | 6.84 | 6.90 | 6.39 | 6.45 | 6.72 | 7.22 | 7.02 | 6.68 | 4.75 | 1.43 |
| Mechanical Maintenance | **54.46** | 2.80 | 5.51 | 5.51 | 5.09 | 5.14 | 5.28 | 5.79 | 5.74 | 5.78 | 5.52 | 2.30 |
| Electrical Maintenance | **58.37** | 3.43 | 5.79 | 6.06 | 5.49 | 5.53 | 5.69 | 6.04 | 6.04 | 6.17 | 5.59 | 2.54 |
| Technical Services | **27.86** | 1.80 | 2.88 | 2.89 | 2.67 | 2.66 | 2.77 | 3.00 | 3.00 | 3.00 | 2.55 | 0.64 |
| **Total Mining Cost** | **606.05** | **28.42** | **51.77** | **52.34** | **49.01** | **57.06** | **63.82** | **70.73** | **70.08** | **70.94** | **66.54** | **25.34** |
| **Unit Cost ($/t milled)** | **24.03** | 15.72 | 21.49 | 20.94 | 19.60 | 22.82 | 25.53 | 28.29 | 28.03 | 28.38 | 26.62 | 25.13 |
| **Unit Cost ($/payable lb Cu)** | **0.91** | 0.51 | 0.68 | 0.69 | 0.67 | 0.92 | 1.04 | 1.24 | 1.22 | 1.19 | 1.10 | 1.04 |

**Note: Excludes costs during pre-production which are included in the initial CAPEX. Continuous Miner in use until Y5 of operation.*

### 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 (7) categories: labour, reagents, grinding media, liners, maintenance supplies, operating 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 44.6% of cost or USD 6.53$/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 item.

The process plant manpower comprises 52 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 mainly planned from the utility company grid, complemented by natural gas generators. These gensets will be used in normal operation mode to shave the peak load required in winter season for the first five (5) years of operation due to a limitation in power available on the utility grid. Thus, the indicative electricity price will be USD 0.083/kWh in the first five (5) years and then USD 0.0716/kWh through the rest of the life of mine for interruptible service with the main substation provided. The power consumption at 6,800 mtpd is estimated at 53.66 kWh/t milled (Table 21.23).

**Table 21.19: Process Operating Cost Summary**

| Mill OPEX | LoM Cost ($M) | Avg. Cost ($M/y) | $/t ore | $/lb | % |
|---|---|---|---|---|---|
| Mill Labour | 53.82 | 5.13 | 2.13 | 0.081 | 14.6% |
| Reagents | 164.74 | 15.69 | 6.53 | 0.249 | 44.6% |
| Grinding Media | 15.14 | 1.44 | 0.60 | 0.023 | 4.1% |
| Liners | 3.72 | 0.35 | 0.15 | 0.006 | 1.0% |
| Maintenance Supplies | 13.13 | 1.25 | 0.52 | 0.020 | 3.6% |
| Operating Supplies | 14.72 | 1.40 | 0.58 | 0.022 | 4.0% |
| Power | 103.82 | 9.89 | 4.11 | 0.157 | 28.1% |
| **Total Mill OPEX** | **369.09** | **35.15** | **14.63** | **0.558** | **100.0%** |

**Table 21.20: Annual LoM Processing OPEX**

| Mill OPEX ($M) | Total | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mill Labour | **53.82** | 3.53 | 4.71 | 4.71 | 4.71 | 4.71 | 5.77 | 5.77 | 5.77 | 5.77 | 5.77 | 2.62 |
| Reagents | **164.74** | 11.85 | 15.73 | 16.32 | 16.32 | 16.32 | 16.32 | 16.32 | 16.32 | 16.32 | 16.32 | 6.58 |
| Grinding Media | **15.14** | 1.09 | 1.45 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 0.61 |
| Liners | **3.72** | 0.27 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.18 |
| Maintenance Supplies | **13.13** | 0.96 | 1.28 | 1.28 | 1.28 | 1.28 | 1.28 | 1.28 | 1.28 | 1.28 | 1.28 | 0.64 |
| Operating Supplies | **14.72** | 1.06 | 1.41 | 1.46 | 1.46 | 1.46 | 1.46 | 1.46 | 1.46 | 1.46 | 1.46 | 0.60 |
| Power | **103.82** | 8.12 | 10.79 | 11.12 | 11.12 | 10.74 | 9.58 | 9.58 | 9.58 | 9.58 | 9.58 | 4.03 |
| **Total Mill OPEX** | **369.09** | **26.89** | **35.72** | **36.75** | **36.75** | **36.36** | **36.27** | **36.27** | **36.27** | **36.27** | **36.27** | **15.26** |
| **Unit Cost ($/t milled)** | **14.63** | **14.87** | **14.83** | **14.70** | **14.70** | **14.55** | **14.51** | **14.51** | **14.51** | **14.51** | **14.51** | **15.13** |
| **Unit Cost ($/pay. Lb Cu)** | **0.56** | **0.48** | **0.47** | **0.49** | **0.51** | **0.59** | **0.59** | **0.63** | **0.63** | **0.61** | **0.60** | **0.62** |

**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 (USD/t) |
|---|---|---|---|---|---|---|---|
| Sodium Hydrosulphide (NaHS) | 1,089.3 | g/t | 1,123 | USD/t | 2,624 | t/y | 1.22 |
| Sodium Isobutyl Xantante (C-3430) | 316.7 | g/t | 3,189 | USD/ | 763 | t/y | 1.01 |
| Methyl Isobutyl Carbinol (MIBC) | 32.8 | g/t | 2,753 | USD/ | 79 | t/y | 0.09 |
| Dowfroth 250 (D-250) | 74.8 | g/t | 4,951 | USD/ | 180 | t/y | 0.37 |
| Alkylaryl Dithiophosphate (A-249) | 334.6 | g/t | 7,154 | USD/ | 806 | t/y | 2.39 |
| n-Dodecyl Mercaptan (NDM) | 89.3 | g/t | 8,789 | USD/ | 215 | t/y | 0.78 |
| Sodium Silicates | 98.4 | g/t | 683 | USD/ | 237 | t/y | 0.07 |
| Carboxymethyl Cellulose Sodium | 137.8 | g/t | 4,070 | USD/ | 332 | t/y | 0.56 |
| Hydrated Lime | - | g/t | 235 | USD/ | - | t/y | - |
| Flocculant | 0.5 | g/t | 4,802 | USD/ | 1.1 | t/y | 0.00 |
| Anti-Scalant | 9.7 | L/h | 2,753 | USD/$m^3$ | 23.4 | $m^3$/y | 0.03 |
| **Total** | | | | | | | **6.53** |

**Table 21.22: Grinding Media and Liner Consumption**

| Grinding Media & Liners | Dosage | | Consumable Pricing | | Media & Liner Consumption | | Unit Cost (USD/t |
|---|---|---|---|---|---|---|---|
| SAG Mill Grinding Media | 164 | g/t | 1,494 | USD/t | 395 | t/y | 0.25 |
| Ball Mill Grinding Media | 248 | g/t | 1,394 | USD/t | 597 | t/y | 0.35 |
| Regrind Mill Grinding Media | 5 | g/t | 1,857 | USD/t | 12 | t/y | 0.01 |
| SAG Mill Liner | 31 | g/t | 624,000 | USD/set | 0.43 | set/y | 0.11 |
| Ball Mill Liner | 38 | g/t | 178,000 | USD/set | 0.53 | set/y | 0.04 |
| **Total** | | | | | | | **0.75** |

**Table 21.23: Average Annual Mill Power Consumption by Area**

| Mill Power by Area | 6,000 mtpd Power (kWh/t) | 6,800 mtpd Power (kWh/t) |
|---|---|---|
| Crushed Ore Conveying, Storage & Reclaim | 0.31 | 0.27 |
| Grinding Circuit | 36.09 | 36.02 |
| Rougher Flotation | 2.85 | 2.51 |
| Regrind Circuit | 5.18 | 5.16 |
| Cleaner Flotation | 2.36 | 2.11 |
| Concentrate Dewatering | 1.82 | 1.70 |
| Tailings | 1.42 | 1.25 |
| On Stream Analyzer | 1.12 | 1.10 |
| Reagents Storage and Handling | 0.10 | 0.09 |
| Plant Services | 2.10 | 2.03 |
| Buildings and Power | 1.51 | 1.33 |
| **Total** | **54.86** | **53.58** |

### 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 whole site. 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 USD 1.62M/y starting in Q4-2031 and up to the end of the mine life.

The G&A labour includes 40 people whose total labour cost represents 34.2% 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 USD 9.68M or USD 4.03/t of ore.

**Table 21.24: General Management and Administration Cost Summary**

| G&A OPEX by Department | LoM Cost ($M) | Avg. Cost ($M/y) | $/t ore | $/lb | % |
|---|---|---|---|---|---|
| General Management | 4,459 | 425 | 0.18 | 0.007 | 4.4% |
| Finance & Accounting | 5,906 | 562 | 0.23 | 0.009 | 5.8% |
| Supply Chain | 7,301 | 695 | 0.29 | 0.011 | 7.2% |
| Information Technology | 11,784 | 1,122 | 0.47 | 0.018 | 11.6% |
| Human Resources | 9,551 | 910 | 0.38 | 0.014 | 9.4% |
| Health, Safety & Environment | 25,110 | 2,391 | 1.00 | 0.038 | 24.7% |
| Surface Support | 24,892 | 2,371 | 0.99 | 0.038 | 24.5% |
| Insurance | 12,659 | 1,206 | 0.50 | 0.019 | 12.5% |
| **Total G&A Costs** | **101,662** | **9,682** | **4.03** | **0.154** | **100.0%** |

**Table 21.25: Annual LoM G&A OPEX**

| G&A Cost ($M) | Total | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| General Management | **4.46** | 0.31 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.19 |
| Finance & Accounting | **5.91** | 0.35 | 0.60 | 0.60 | 0.60 | 0.60 | 0.60 | 0.60 | 0.60 | 0.60 | 0.60 | 0.13 |
| Supply Chain | **7.30** | 0.48 | 0.73 | 0.73 | 0.73 | 0.73 | 0.73 | 0.73 | 0.73 | 0.73 | 0.73 | 0.26 |
| Information Technology | **11.78** | 0.75 | 1.17 | 1.17 | 1.17 | 1.17 | 1.17 | 1.17 | 1.17 | 1.17 | 1.17 | 0.53 |
| Human Resources | **9.55** | 0.69 | 0.96 | 0.96 | 0.96 | 0.96 | 0.96 | 0.96 | 0.96 | 0.96 | 0.86 | 0.34 |
| Health, Safety & Environment | **25.11** | 1.18 | 1.74 | 1.74 | 1.74 | 1.74 | 2.15 | 3.36 | 3.36 | 3.36 | 3.28 | 1.43 |
| Surface Support | **24.89** | 2.60 | 3.46 | 3.46 | 3.46 | 3.00 | 1.62 | 1.62 | 1.62 | 1.62 | 1.62 | 0.81 |
| Insurance | **12.66** | 0.93 | 1.24 | 1.24 | 1.24 | 1.24 | 1.24 | 1.24 | 1.24 | 1.24 | 1.24 | 0.62 |
| **Total G&A Costs** | **101.66** | **7.30** | **10.34** | **10.34** | **10.34** | **9.88** | **8.90** | **10.12** | **10.12** | **10.12** | **9.93** | **4.29** |
| **Unit Cost ($/t milled)** | **4.03** | **4.04** | **4.29** | **4.13** | **4.13** | **3.95** | **3.56** | **4.05** | **4.05** | **4.05** | **3.97** | **4.26** |
| **Unit Cost ($/payable lb Cu)** | **0.15** | **0.13** | **0.13** | **0.14** | **0.14** | **0.16** | **0.15** | **0.18** | **0.18** | **0.17** | **0.16** | **0.18** |

**Note: Excludes costs during pre-production which are included in the initial CAPEX*

# 22. ECONOMIC ANALYSES

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 conducted in real terms (i.e., without inflation factors) in Q1 2023 US dollars, with no project financing assumptions but with financing for mining production equipment provided by certain manufacturers. 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 (2025 = US$4.25/lb, 2026 = US$4.15/lb, 2027 = US$4.00/lb) with a long-term price of US$4.00/lb (2028). The silver price is kept constant at US$25.00/oz.

### 22.1.2 Fuel

The reference diesel fuel price used for estimating operating costs is US$0.73/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 | LoM (Incl. pre-production) | |
|---|---|---|
| | US$/gal. | US$/L |
| Retail Diesel Fuel Price | 3.54 | 0.93 |
| Less: Federal Excise Tax | -0.29 | -0.08 |
| Less: State Tax | -0.30 | -0.08 |
| Less: Prepaid Sales Tax | -0.18 | -0.05 |
| **Off-Highway Diesel Fuel Price** | **2.77** | **0.73** |

### 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$/$CAD | 0.8 |
| US$/Euro | 1.25 |

## 22.2 Metal Production and Revenue

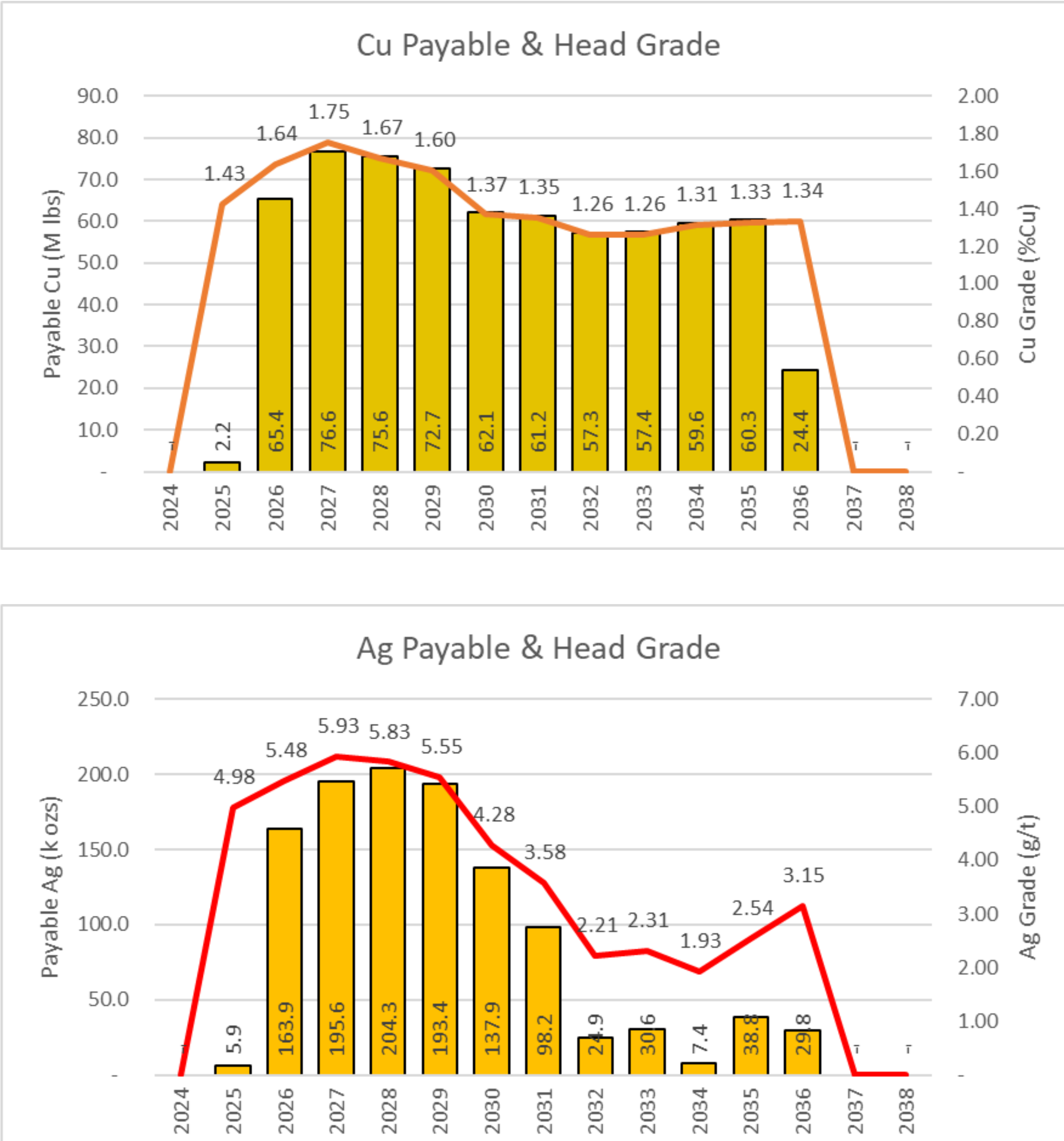
Payable copper produced over the Project life is 306 kt (675M lb) (including 6 kt (13M lb) during pre-production) with annual average of 29,4 kt (64.8 M lb) over the 10.3-year operation life. The average payable copper rate is 95.8%, which includes the 0.2% concentrate loss. Payable silver production over the LoM is 1.13 M oz (including 0.34 M oz during pre-production). with an annual average of 109 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 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnage Processed | kt | **25,703** | 85 | 2,201 | 2,409 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 |
| Cu Head Grade | % Cu | **1.45** | 1.43 | 1.64 | 1.75 | 1.67 | 1.60 | 1.37 | 1.35 | 1.26 | 1.26 | 1.31 | 1.33 |
| Ag Head Grade | g/t | **3.91** | 4.98 | 5.48 | 5.93 | 5.83 | 5.55 | 4.28 | 3.58 | 2.21 | 2.31 | 1.93 | 2.54 |
| Concentrate (dry) | dmt | **1,292** | 4.2 | 125.2 | 146.6 | 144.7 | 139.2 | 118.9 | 117.2 | 109.6 | 109.8 | 114.0 | 115.4 |
| Concentrate (wet) | wmt | **1,419** | 4.6 | 137.6 | 161.1 | 159.0 | 153.0 | 130.6 | 128.8 | 120.4 | 120.7 | 125.3 | 126.8 |
| Cu Contained Metal | kt | **372** | 1 | 36 | 42 | 42 | 40 | 34 | 34 | 32 | 32 | 33 | 33 |
| Cu Contained Metal | M lbs | **820** | 2.67 | 79.46 | 93.03 | 91.82 | 88.33 | 75.42 | 74.37 | 69.53 | 69.67 | 72.36 | 73.19 |
| Ag Contained Metal | K ozs | **3,233** | 14 | 388 | 459 | 469 | 446 | 344 | 288 | 178 | 186 | 155 | 204 |
| 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 | **320** | 1.0 | 31.0 | 36.3 | 35.8 | 34.5 | 29.4 | 29.0 | 27.1 | 27.2 | 28.2 | 28.5 |
| Cu Metal Production | M lbs | **705** | 2.3 | 68.3 | 80.0 | 79.0 | 76.0 | 64.9 | 64.0 | 59.8 | 59.9 | 62.2 | 62.9 |
| Ag Metal Production | K oz. | **2,373** | 10 | 285 | 337 | 344 | 328 | 253 | 211 | 131 | 137 | 114 | 150 |
| 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 | % | **47.65** | 59.38 | 57.57 | 58.04 | 59.41 | 59.02 | 54.61 | 46.48 | 19.07 | 22.41 | 6.50 | 25.84 |
| Cu Payable Metal | kt | **306** | 1.0 | 29.7 | 34.8 | 34.3 | 33.0 | 28.2 | 27.8 | 26.0 | 26.0 | 27.0 | 27.3 |
| Cu Payable Metal | M lbs | **675** | 2.2 | 65.4 | 76.6 | 75.6 | 72.7 | 62.1 | 61.2 | 57.3 | 57.4 | 59.6 | 60.3 |
| Ag Payable Metal | K ozs | **1,131** | 5.9 | 163.9 | 195.6 | 204.3 | 193.4 | 137.9 | 98.2 | 24.9 | 30.6 | 7.4 | 38.8 |
| Operating periods | Yr. | **10.8** | 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, Q1-2026 is part of 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 six (6) months where the tonnage ramps-up from 900 to 4,290 mtpd in between the last quarter of 2025 and the first quarter of 2026. From Q2-2026, commercial operations are declared with an average milling rate of 6,600 mtpd for seven (7) quarters and then a final increase to steady state throughput of 6,800 mtpd in the second quarter of 2028. The operations period lasts 10.3 years (when excluding the pre-production period) based on the currently defined mineral reserves being depleted in Q3-2036.

**Table 22.4: Mill Commissioning and Ramp-Up**

| Mill Commissioning and Ramp-Up | | Days | Tonnage (t/month) | Max Mill Rate (t/d) | % Nameplate |
|---|---|---|---|---|---|
| Pre-Prod Quarter 1 | Q4-2025 | 92 | 28,333 | 924 | 13.59% |
| Pre-Prod Quarter 2 | Q1-2026 | 90 | 128,700 | 4,290 | 63.09% |
| **Total Pre-Prod.** | **Q4-2025 to Q1-2026** | **182** | **78,517** | **4,290** | **63.09%** |
| Operation Quarter 1 | Q2-2026 | 91 | 200,200 | 6,600 | 97.06% |
| Operation Quarter 2 | Q3-2026 | 92 | 202,400 | 6,600 | 97.06% |
| Operation Quarter 3 | Oct-26 | 92 | 202,400 | 6,600 | 97.06% |
| **Total Operations Yr. 1** | **Q2 to Q4-2026** | **275** | **201,667** | **6,600** | **97.06%** |
| **Total Operations Yr. 2** | Q4-2025 | **365** | **200,750** | **6,600** | **97.06%** |
| **Total Operations Yr. 3+** | | **365** | **206,833** | **6,800** | **100%** |

## 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$425.1M. The total initial Project capital includes a contingency of US$37.6M which is 9.7% of the total CAPEX before contingency excluding pre-production revenue of US$33.962M. Net of pre-production revenue, the initial CAPEX is estimated at US$391.2M as presented in Table 22.5. The initial Project CAPEX is spent over a period of 27 months starting in January 2024 and ending in March 2026.

**Table 22.5: Initial Capital Expenditure Summary**

| Initial CAPEX | US$ k |
|---|---|
| 000 - General | 1,150 |
| 100 - Infrastructure | 31,779 |
| 200 - Power & Electrical | 42,460 |
| 300 - Water & TSF Mgmt. | 46,198 |
| 400 - Mobile Equipment | 24,932 |
| 500 - Mine Infrastructure | 51,172 |
| 600 - Process Plant | 105,502 |
| 700 - Construction Indirects | 51,028 |
| 800 - General Services & Owner's Costs | 25,377 |
| 900 - Pre-Production, Commissioning | 7,888 |
| **Sub-Total Before Contingency** | **387,487** |
| **Contingency 9.7%** | **37,645** |
| **Total Incl. Contingency** | **425,131** |
| **Less: Pre-Production Revenue** | **(33,962)** |
| **Total Incl. Contingency & Pre-Prod. Revenue** | **391,170** |

### 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 $269.89M with the breakdown presented in Table 22.6.

**Table 22.6: Sustaining Capital Expenditure Summary**

| Sustaining CAPEX | LoM ($M) | $/t Ore | $/lb Cu Payable |
|---|---|---|---|
| Tailings Disposal Facility Expansions | 54.77 | 2.17 | 0.08 |
| Water Treatment Plant | 17.11 | 0.68 | 0.03 |
| Mine Equipment Purchases | 141.56 | 5.61 | 0.21 |
| Mine Development Expenditures | 33.11 | 1.31 | 0.05 |
| Sustaining CAPEX - Other | 23.35 | 0.93 | 0.04 |
| **Total Sustaining CAPEX** | **269.89** | **9.77** | **0.37** |

**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 2037 to 2038, with an overall estimate of US$37.1M net of salvage value.

**Table 22.7: Closure and Reclamation Cost Estimate by Stage**

| Closure Cost Estimate | Cost ($k) |
|---|---|
| TDF Reclamation | 28,832 |
| Site Closure & Reclamation | 10,136 |
| General Reclamation | 9,722 |
| Salvage Value | (20,863) |
| Post Closure Monitoring | 3,924 |
| MDEQ Admin Oversight | 5,393 |
| **Total Cost** | **37,145** |

#### 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$8M 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$2,417M, excluding US$49.83M of NSR, generating during pre-production and treated as a reduction of initial capital expenditures. The average NSR over the LoM is US$4.05/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$48.05/t of ore or US$1.83/lb of payable copper with mining representing 50.0% of the total OPEX, or US$24.02/t of ore. A summary of operating cash flow and operating costs is presented in Table 22.8.

**Table 22.8: Operating Cost & Summary**

| Operating Cash Flow | LoM (US$M) | US$/t Ore | US$/lb Cu Payable |
|---|---|---|---|
| Cu Revenue | 2,656 | 105.25 | 4.01 |
| Ag Credits | 27 | 1.09 | 0.04 |
| Revenue | 2,683 | 106.34 | 4.05 |
| Concentrate Transportation Costs | 140 | 5.56 | 0.21 |
| Treatment & Refining Charges | 126 | 4.99 | 0.19 |
| Net Smelter Return | 2,417 | 95.79 | 3.65 |
| Royalties | 136 | 5.37 | 0.20 |
| Mining Costs | 606 | 24.02 | 0.92 |
| Processing Costs | 369 | 14.63 | 0.56 |
| G&A Costs | 102 | 4.03 | 0.15 |
| Working Capital | 0 | 0.00 | (0.00) |
| **Total OPEX (including royalties)** | **1,212** | **48.05** | **1.83** |
| **Operating Cash Flow** | **1,203** | **47.68** | **1.82** |

**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 | 612 | 24.25 | 0.92 |
| Processing | 369 | 14.63 | 0.56 |
| G&A | 102 | 4.03 | 0.15 |
| Offsite Costs (transport, TC/RCs) | 266 | 10.55 | 0.40 |
| By-Product credits | (27) | (1.09) | (0.04) |
| **C1 Cost** | **1,316** | **52.14** | **1.99** |

| LoM Costs | Total Cost (US$M) | Unit Cost $/tonne milled) | Unit Cost ($/payable lb) |
|---|---|---|---|
| Depreciation and Closure | 698 | 27.67 | 1.05 |
| Royalty Costs | 136 | 5.27 | 0.20 |
| Interest cost (3rd party debt – Eq. financing) | 3 | 0.13 | 0.01 |
| **C3 Cost** | **2,153** | **85.31** | **3.25** |

**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 | 239 | 20.35 | 0.70 |
| Processing | 172 | 14.71 | 0.50 |
| G&A | 48 | 4.11 | 0.14 |
| Offsite Costs (transport, TC/RCs) | 138 | 11.73 | 0.40 |
| By-Product Credits | (22) | (1.85) | (0.06) |
| **C1 Cost** | **575** | **49.05** | **1.68** |
| Depreciation and Closure | 264 | 22.54 | 0.77 |
| Royalty Costs | 71 | 5.88 | 0.20 |
| Interest Cost (3rd party debt – Eq. financing) | 3 | 0.13 | 0.01 |
| **C3 Cost** | **914** | **77.96** | **2.67** |

## 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$34.8M.

### 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$62.7M.

### 22.5.3 Royalties

The owners of the mineral rights (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$4.01/lb the royalty rate is 4.0% NSR. Lease payments are deductible from the royalty payments. Over the LoM, this royalty represents a cost of US$98.64M.

Under a transaction with Osisko Gold Royalties, Osisko is to receive a 1.5% NSR royalty which is fixed regardless of copper price. Over the LoM, the Osisko royalty represents a cost of US$36.6M.

Additionally, the Silver Stream royalty with Osisko, which represents a total cost of US$3.2M over the LoM.

## 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 22.2. A cash flow waterfall for the Project is summarized in Figure 22.3.

The undiscounted after-tax cash flow is estimated at US$455.7 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 | 552.6 | 455.1 |
| NPV 8% | $M | 221.8 | 167.6 |
| IRR | % | 20.0% | 17.6% |
| Payback | yr. | 3.2 | 3.5 |

**Figure 22.2: After-Tax Annual Project Cash Flow (with Equity)**

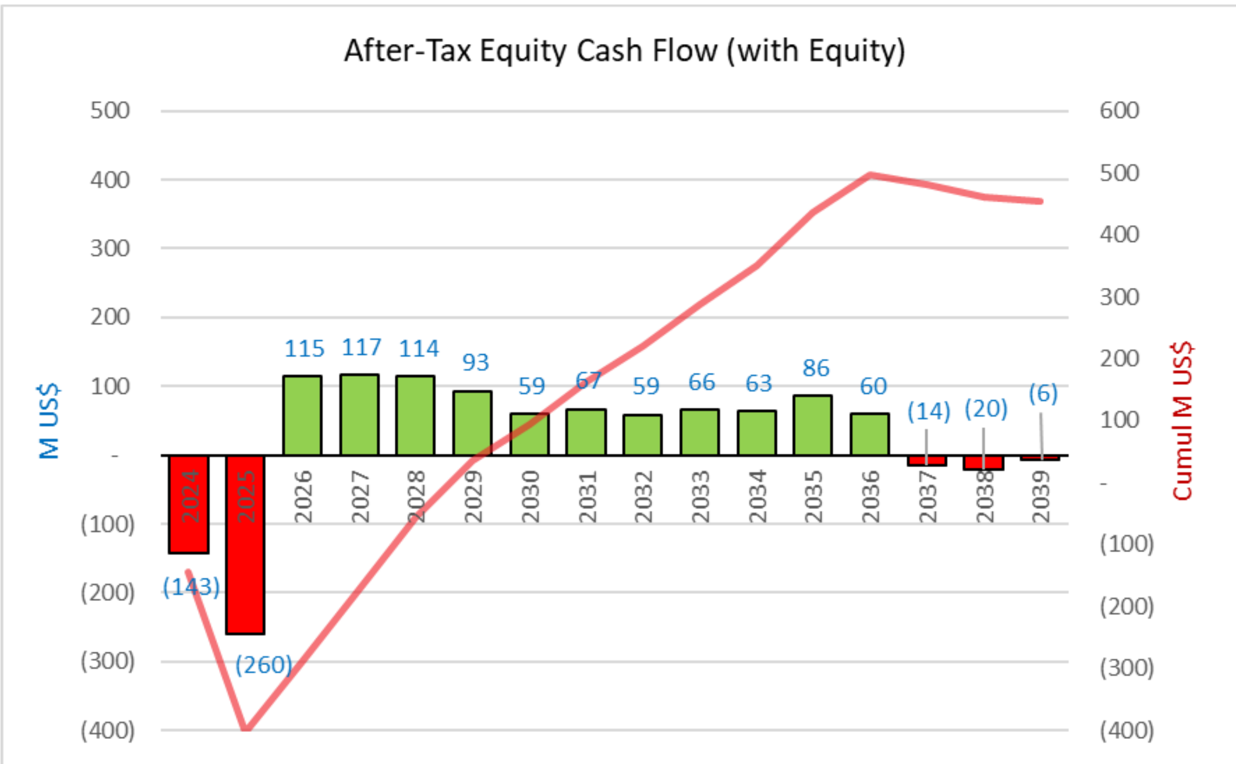


**Figure 22.3: After Tax Project Cash Flow Waterfall**

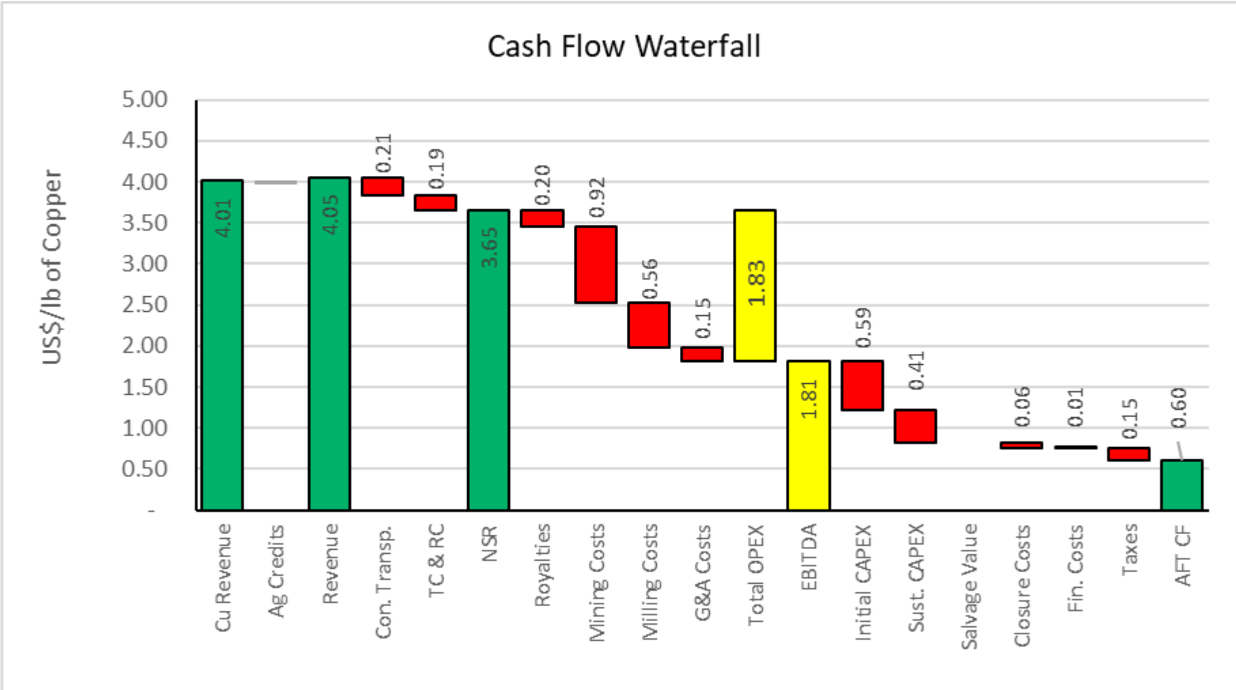


**Table 22.12: After-Tax Annual Cash Flow Summary**

| Cash Flow ($M) | Total | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Revenue (Cu + Ag) | **2,683** | - | - | 230 | 311 | 308 | 296 | 252 | 247 | 230 | 230 | 239 | 242 | 99 | - | - | - |
| Con. Transp. Costs | **(140)** | - | - | (12) | (16) | (16) | (15) | (13) | (13) | (12) | (12) | (13) | (13) | (5) | - | - | - |
| TC / RCs | **(126)** | - | - | (10) | (15) | (14) | (14) | (12) | (12) | (11) | (11) | (11) | (11) | (5) | - | - | - |
| Net Smelter Return | **2,417** | - | - | 208 | 280 | 277 | 267 | 227 | 223 | 207 | 207 | 215 | 218 | 89 | - | - | - |
| Royalties | **(136)** | - | - | (12) | (16) | (16) | (15) | (13) | (12) | (11) | (11) | (12) | (12) | (5) | - | - | - |
| Mining Costs | **(612)** | - | - | (34) | (52) | (52) | (49) | (57) | (64) | (71) | (70) | (71) | (67) | (25) | - | - | - |
| Processing Costs | **(369)** | - | - | (27) | (36) | (37) | (37) | (36) | (36) | (36) | (36) | (36) | (36) | (15) | - | - | - |
| G&A Costs | **(102)** | - | - | (7.30) | (10.34) | (10.34) | (10.34) | (9.88) | (8.90) | (10.12) | (10.12) | (10.12) | (9.93) | (4.29) | - | - | - |
| Total Operating Costs | **(1,218)** | - | - | (80) | (114) | (115) | (111) | (116) | (121) | (129) | (128) | (129) | (125) | (50) | - | - | - |
| Working Capital | - | (4) | (5) | (34) | 0 | 4 | (2) | 10 | 1 | (1) | 1 | (3) | 2 | 31 | - | - | - |
| Operating Cash Flow | **1,199** | (4) | (5) | 93 | 167 | 166 | 153 | 121 | 103 | 77 | 80 | 83 | 95 | 70 | - | - | - |
| Initial CAPEX | **(391)** | (140) | (279) | 28 | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sustaining CAPEX | **(270)** | - | - | (70) | (40) | (32) | (39) | (40) | (23) | (8) | (7) | (12) | - | - | - | - | - |
| Closure & SV | **(37)** | - | - | - | - | - | - | - | - | - | - | - | - | - | (11) | (20) | (6) |
| MLA Receipts / Disbursements | **(4)** | 0 | 15 | 17 | (5) | (10) | (10) | (7) | (3) | | | | | | | | |
| Taxes | **(98)** | - | - | - | (5) | (9) | (11) | (15) | (10) | (9) | (7) | (8) | (9) | (10) | (4) | - | - |
| Project AFT Cash Flow (w/o equity) | **399** | (143) | (269) | 68 | 117 | 114 | 93 | 59 | 67 | 59 | 66 | 63 | 86 | 60 | (14) | (20) | (6) |
| Cumul. AFT Cash Flow (w/o equity) | | (143) | (413) | (344) | (228) | (113) | (20) | 39 | 106 | 165 | 231 | 294 | 380 | 440 | 426 | 406 | 399 |
| Project AFT Cash Flow (with equity) | 455 | (143) | (260) | 115 | 117 | 114 | 93 | 59 | 67 | 59 | 66 | 63 | 86 | 60 | (14) | (20) | (6) |
| Cumul. AFT Cash Flow (with equity) | | (143) | (404) | (289) | (172) | (58) | 36 | 95 | 162 | 220 | 287 | 350 | 436 | 496 | 482 | 461 | 455 |

**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 parameter 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$168.0M to US$451.7M. Similarly, a decrease of 20% in copper price or copper grade reduces the $NPV_{8\%}$ to -US$111.8M.

**Table 22.13: Pre-Tax Sensitivity Results**

| Variance | Before-Tax Results | | | |
|---|---|---|---|---|
| | $NPV_{0\%}$ (M$) | $NPV_{8\%}$ (M$) | IRR (%) | Payback (yrs.) |
| Metal Price Sensitivities | | | | |
| 20% | 1080.1 | 542.4 | 34.2% | 2.0 |
| 10% | 816.3 | 382.1 | 27.4% | 2.5 |
| 0% | 552.6 | 221.8 | 20.0% | 3.2 |
| -10% | 297.3 | 66.6 | 11.9% | 4.7 |
| -20% | 40.3 | -89.6 | 1.9% | 9.1 |
| Initial Capital Cost Sensitivities | | | | |
| 20% | 467.6 | 143.6 | 14.7% | 4.0 |
| 10% | 510.1 | 182.7 | 17.2% | 3.6 |
| 0% | 552.6 | 221.8 | 20.0% | 3.2 |
| -10% | 595.1 | 261.0 | 23.4% | 2.8 |
| -20% | 637.6 | 300.1 | 27.5% | 2.4 |
| Operating Cost Sensitivities | | | | |
| 20% | 334.6 | 95.2 | 13.7% | 3.9 |
| 10% | 443.6 | 158.5 | 17.0% | 3.6 |
| 0% | 552.6 | 221.8 | 20.0% | 3.2 |
| -10% | 661.6 | 285.2 | 22.8% | 2.9 |
| -20% | 770.6 | 348.5 | 25.4% | 2.7 |

**Table 22.14: After-Tax Sensitivity Results**

| Variance | After-Tax Results | | | |
|---|---|---|---|---|
| | $NPV_{0\%}$ (M$) | $NPV_{8\%}$ (M$) | IRR (%) | Payback (yrs.) |
| Metal Price Sensitivities | | | | |
| 20% | 915.8 | 451.3 | 31.3% | 2.1 |
| 10% | 685.4 | 309.5 | 24.7% | 2.6 |
| 0% | 455.1 | 167.6 | 17.6% | 3.5 |
| -10% | 231.7 | 30.0 | 9.9% | 5.3 |
| -20% | -1.4 | -112.3 | 0.0% | 9.8 |
| Initial Capital Cost Sensitivities | | | | |
| 20% | 381.3 | 96.0 | 12.7% | 4.4 |
| 10% | 418.2 | 131.8 | 15.0% | 3.8 |
| 0% | 455.1 | 167.6 | 17.6% | 3.5 |
| -10% | 491.9 | 203.4 | 20.8% | 2.9 |
| -20% | 528.8 | 239.2 | 24.6% | 2.5 |
| Operating Cost Sensitivities | | | | |
| 20% | 259.0 | 52.6 | 11.4% | 4.4 |
| 10% | 357.5 | 110.3 | 14.6% | 3.7 |
| 0% | 455.1 | 167.6 | 17.6% | 3.5 |
| -10% | 552.6 | 224.8 | 20.4% | 3.0 |
| -20% | 650.1 | 282.1 | 23.0% | 2.8 |

**Figure 22.4: After-Tax NPV$_{8\%}$ Sensitivity**

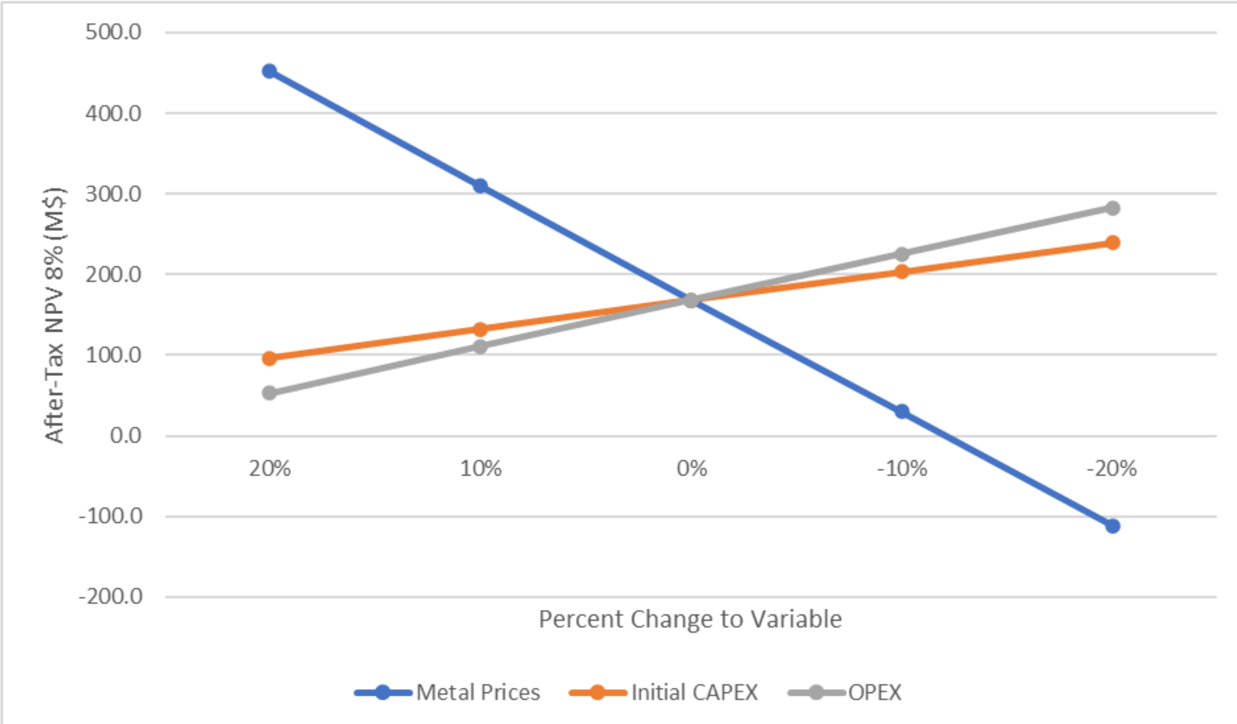
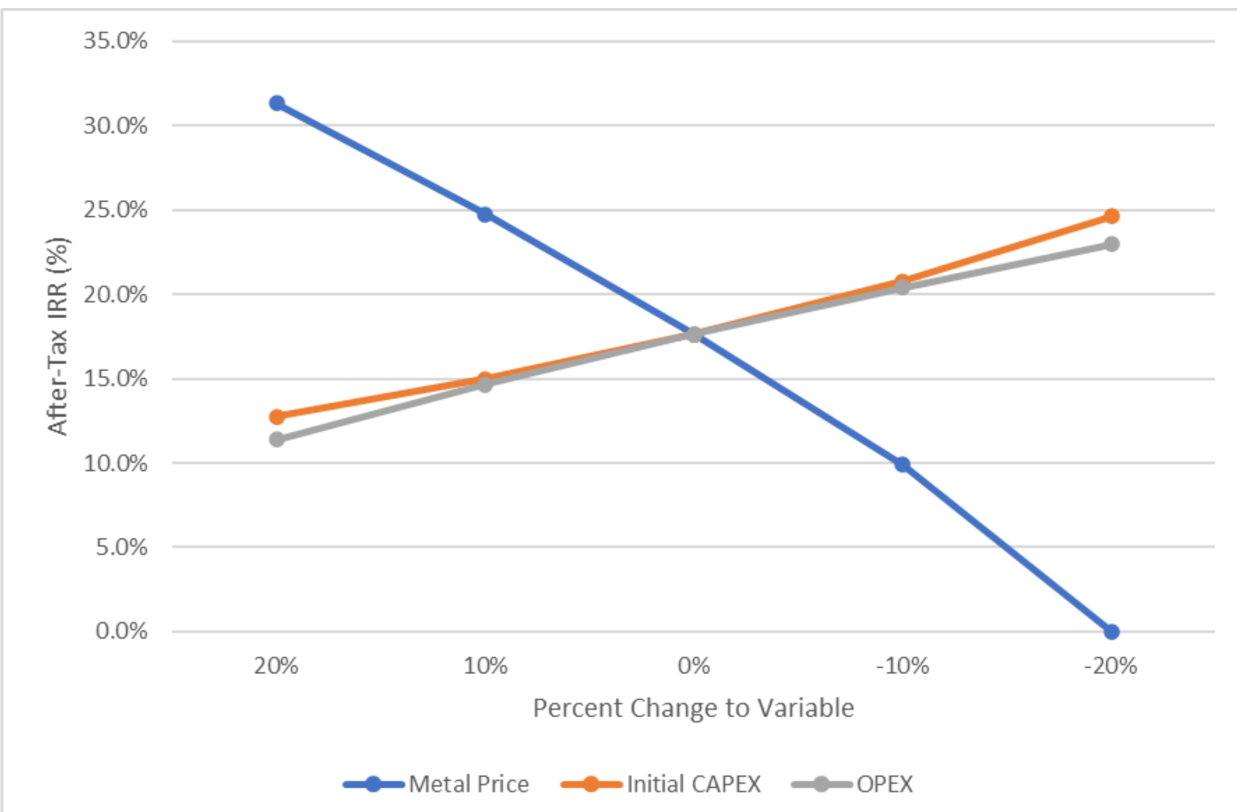



**Figure 22.5: After-Tax IRR Sensitivity**



# 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 a mixture of “owner managed” and EPCM methodologies. This will result in a mixed management team with both Highland and contracted personnel throughout the construction phase.

The Project team will manage and execute project engineering, procurement of project equipment and material, execute project construction, manage project control, ramp-up staff for start-up and operations, and coordinate the commissioning of the mine and process areas. Certain operations departments will be integrated in the project team early in the process to allow their parallel development and will focus on the project’s operational 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.

As part of the early works, the preconstruction phase will involve clearing and grubbing, permitted stream and wetland impacts, and stream and wetland mitigation projects.

Once the project enters construction, the priority activities will be to complete site preparation (clearing, grubbing), excavation of the box cut, access road development, Tailings Disposal Facility (“TDF”) phase 1, and temporary power access. Reputable third-party consultants and engineers will be hired for engineering and QA/QC services.

Detailed engineering will also be a priority early on to de-risk the project in the months leading to full scale construction. Early detailed engineering will focus on the long lead items, as well as the power supply strategy, the mine development, and the process facility. The engineering of the TDF and the main access road will also be initiated early on to be ready for construction in the early stage once construction is initiated.

## 24.2 Project Development Organization

The Project implementation team will be made up of several teams working together and sharing information and resources to lead the project to completion.

The first group that will be active in the early stages of the project execution will be the engineering team. This group will oversee all technical aspects of the project, including management of engineering packages, selection of equipment and material, review of vendor data, and support during pre-production, commissioning, and delivery to the operations group. The engineer team will act on behalf of the best interest of the Owner and will consist of professionals from different technical backgrounds.

The Mine Group includes the Operations, Maintenance, Engineering, and Geology departments. The Mine Group will be in charge of overseeing the development of the mine, while managing employees and contractors. It will also manage the procurement of mining equipment and material.

The Mill Group personnel will integrate the technical team during the detailed engineering of the process facility and will eventually take charge of the commissioning and process plant pre-production activities.

The General Services Group is established early in the project development phase and will support procurement and logistics activities during the engineering and procurement phase of the project. The group includes general administration and management, 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. 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 Construction team will have the responsibility to manage on-site activities, coordination of contractors, supervision of progress, management of construction contracts, implementation of corporate HSE policies, and ensure the quality of products delivered to the Owner. The construction team will be located on site and will be made up of a group of professionals with different technical backgrounds to support all aspects of construction.

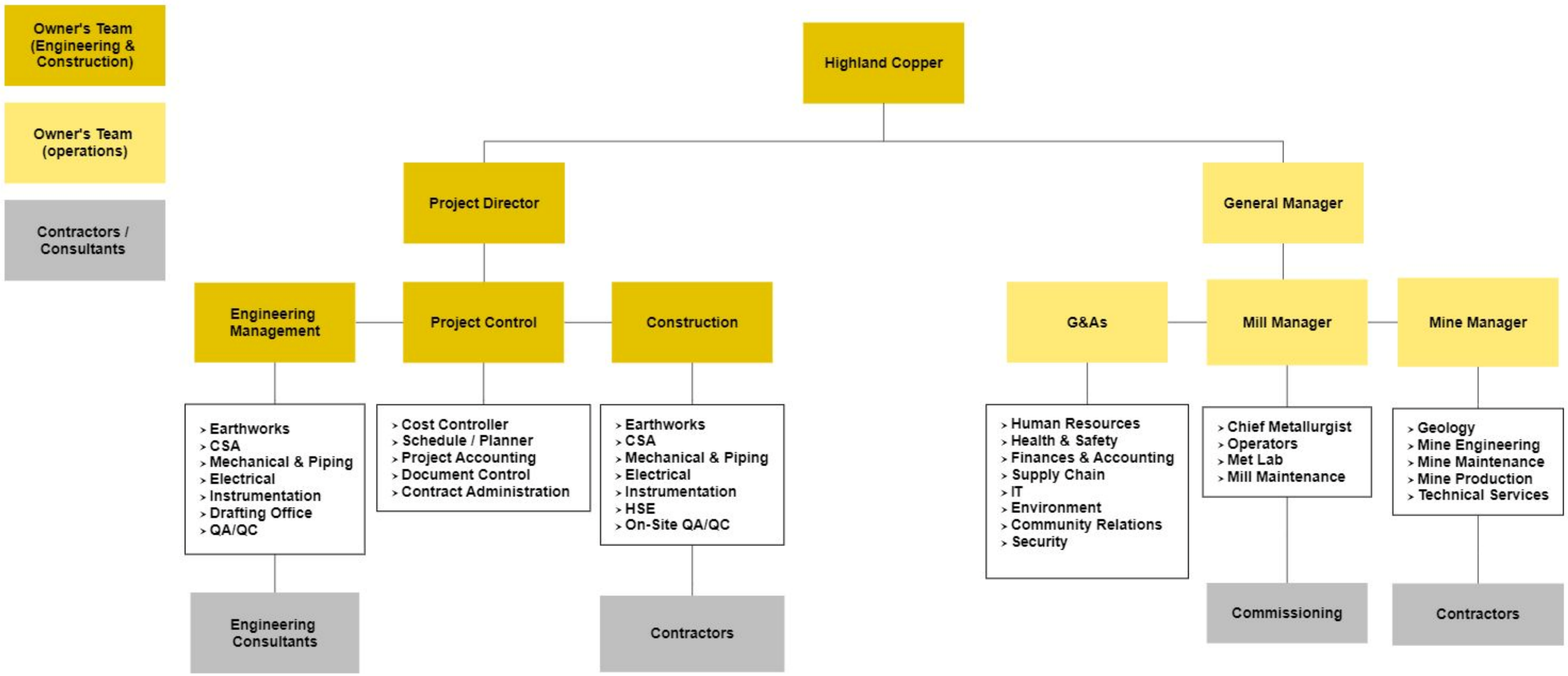
The utility company will be responsible for the engineering, easements negotiations and agreements, construction and commissioning of the power line from Norrie, the tie-ins at Norrie substation and at the Copperwood site main substation. The engineering and construction group will support and coordinate the utility's engineering and construction team.

Procurement and logistics will be supported by all groups. Equipment and or material will be requested by one of the team (Technical, Mine, Mill, G&A, Construction). Bid documents will be assembled and sent out to selected bidders. Bidders will be selected based on their expertise and capabilities in producing the deliverable on budget, and on time. The bid process will be managed by the supply chain team, which operates under the General Services Group. The technical aspects, bid reviews, and final selection of the vendor will be a joint effort between the requester and the supply chain team. Purchase orders will be issued by Highland or by the Owner’s representative. Once a purchase order is placed, the supply chain group will manage expediting, and transportation to site.

The ultimate project authority lies with Highland. The Project Director will be appointed by Highland and share responsibility for all steps in the process required to reach commercial production.

Principal organizational relationships are outlined in Figure 24.1

**Figure 24.1: Project Team Organization for Engineering and Construction Phases**



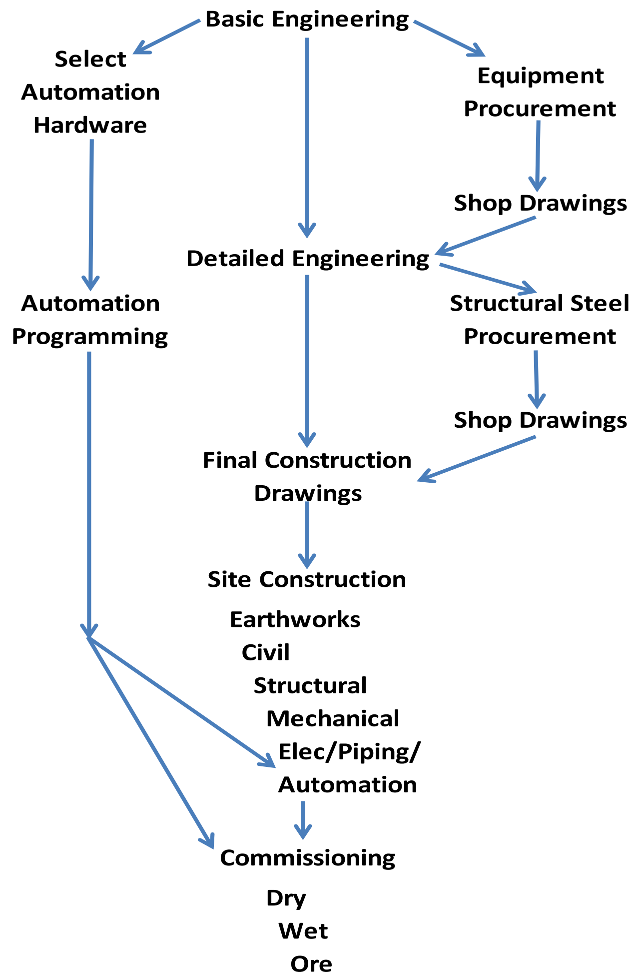
## 24.3 Project Engineering

Project engineering will be divided into engineering packages. Each package will have its own scope of work, and deliverables. The major engineering packages are listed as follow:

- Process Facility
- Underground Mine and Infrastructure
- Tailings Disposal Facility (TDF)
- Access Road
- Powerline
- Site Power Distribution
- Road 519
- Support Buildings

Each engineering package will be awarded to consultancy firms with experience in delivering the expected deliverables. The workflow of each engineering package will resemble the diagram shown in Figure 24.2.

**Figure 24.2: Engineering Process**




The project management team will also look to maximize knowledge and know-how of manufacturers and contractors to provide design-built services. Basic engineering for support buildings, or simple structure buildings, will be used to support the procurement of design-built packages.

## 24.4 Early Works

This Study has identified activities to be developed as early as possible to meet the Project's schedule. On-site works are required during the summer season of 2023 to complete wetland and stream impacts. Works planned on site for 2023 include:

- Clearing & Grubbing
- Wetland and stream impacts
- Wetland creation project

- Stream Relocation project

These activities will be initiated during pre-construction. The project team will manage the engineering of the wetland and stream mitigation projects and support the bid process to hire earthworks contractors to support the construction works. QA/QC services will be provided by a 3rd party, while on-site management will be supported by Highland team.

Once the project enters full construction, early works activities include:

- Clearing & Grubbing
- Excavation of Box-Cut
- Main Access Road
- Process Plant Pad
- TDF Stage 1

### 24.4.1 <u>Pre-Constructruction Detailed Engineering</u>

Highland will initiate detailed engineering activities to achieve the following objectives:

- Be ready to initiate procurement of long-lead items
- De-risk and optimization of technical aspects
- Initiate development of the powerline
- Achieve permitting milestones

Long lead items are listed below:

- Primary, Secondary and Regrind Mills
- Press Filter
- Flotation Circuit
- Road Headers
- Generators

Engineering activities planned during the early stage will focus on:

- Process Facility
- Mine Planning and Development
- Power supply

## 24.5 Procurement and Logistics

Procurement of equipment and material will involve various groups that are part of the project execution team. Equipment, material, and services will be contracted through a competitive bidding strategy. Each package will consist of a scope of work and a list of deliverables. A bidders list will be prepared by the requester and the supply chain group. The mining group will support the procurement packages of mine equipment and services. The engineering group will support the procurement of process equipment packages, electrical equipment and material, buildings and structures. The construction group will support the procurement of installation packages and the hiring of contractors.

Once proposals are received, the project management and supply chain teams will review all bids, produce a bid evaluation (commercial and technical) and recommend a vendor. A vendor may be recommended for technical, commercial, schedule reasons, or a blend of these three reasons. For service contracts, local firms and labor, as well as environmental and safety considerations will be included in the analysis.

Quality Assurance/Quality Control ("QA/QC") services (during construction and manufacturing) would be provided by consultancy firms. Work being executed on-site by contractors will be supervised by a QA/QC group located on site. Manufacturing of goods and equipment will be supervised by a QA/QC group working with the technical team. In some cases, manufacturing locations will be visited for visual inspections. Documents pertaining to QA/QC (certificates of conformity, corrective measures taken, internal test plans, logbooks and measurements, etc.) will be required from vendors and contractors, and copies will be archived on Highland's data management system.

Construction material (structural steel, concrete material, piping, etc.) will be procured in North America. Mechanical and electrical equipment may come from abroad. Equipment and material will be procured through competitive bidding. The project team will consider the country of origin in its decision-making strategy. Local sources will be preferred when it is technically and economically feasible.

## 24.6 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:

- SharePoint 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.
- The project scheduling software will be either MS Project or Primavera P6.

## 24.7 Quality and Design Standards

The Project's detailed engineering will use relevant Michigan design codes and standards using qualified and proven manufacturers.

Health and safety standards will comply with all relevant OSHA and MSHA regulations and conform to Highland's requirements.

## 24.8 Quality Management

QA/QC of all construction activities will be performed by a suitably accredited third-party engineering firm under the direct supervision of the Resident Engineer of the Project. All QA/QC documentation will be archived on Highland's 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 will also be archived on Highland's data management system for archival and review purposes.

## 24.9 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 will be managed 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 will be used to support commissioning activities.

## 24.10 Project Schedule

The construction and pre-production development schedule leading to commercial production is 27 months, consisting of two months for initial mobilization of key personnel and equipment and 25 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 always subject 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.

As described in the previous sections, the project schedule involves pre-construction and construction activities.

Pre-construction includes tasks that will be initiated and /or completed prior to the construction decision and completion of financing.

Pre-construction activities includes:

- Site preparation and site impacts
- Wetland mitigation project
- Development of Powerline
- Process Plant Engineering
- Infrastructure Engineering
- Underground Mine Engineering

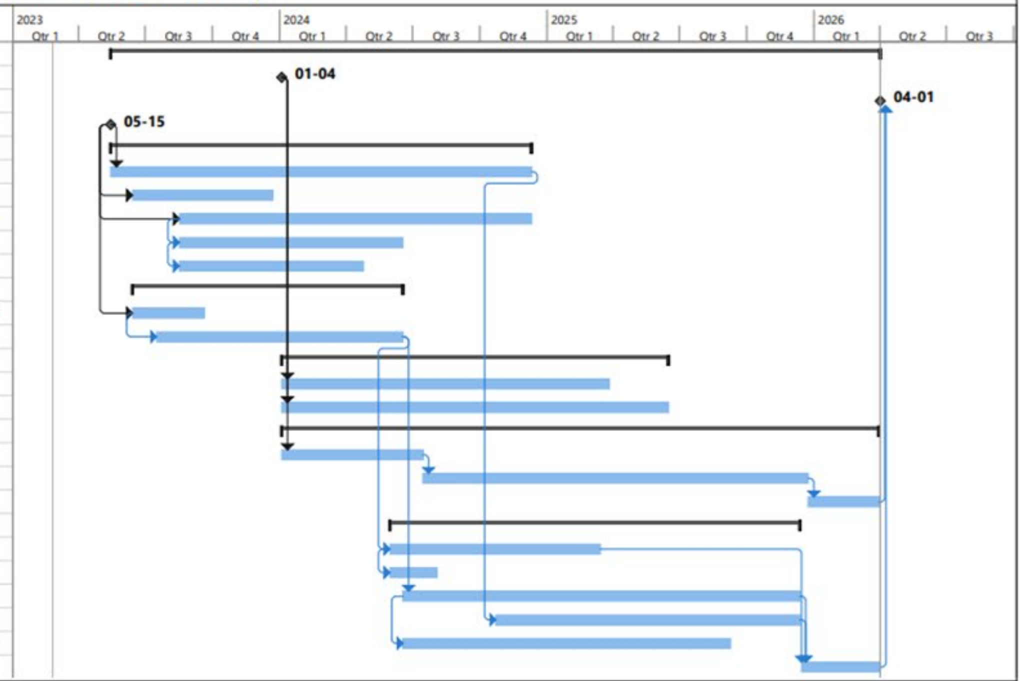
Once construction starts, detailed engineering packages will be expanded to cover all aspects of engineering, including power distribution, instrumentation, programming, as well as concrete design, piping routing, etc. Procurement of long lead items will be initiated, and other packages for equipment and material will be issued for bid.

The early activities during the construction period will focus on site preparation, clearing & grubbing, the development of the boxcut to expedite the level development of the underground mine, and the construction of the tailings disposal facility (TDF) Stage 1. The construction of the process facility pad and main access road will be prioritized during the first summer season.

## Figure 24.3: Project Level 1 Schedule

### USCW - Copperwood Project

| ID | Task Name | Start | Finish |
|---|---|---|---|
| 1 | **Milestones** | **2023 May 15** | **2026 April 01** |
| 2 | Construction Decision | 2024 January 04 | 2024 January 04 |
| 3 | Commercial Production | 2026 April 01 | 2026 April 01 |
| 4 | Engineering Kick-Off | 2023 May 15 | 2023 May 15 |
| 5 | **Engineering** | **2023 May 15** | **2024 December 10** |
| 6 | Powerline Route Development & Engineering | 2023 May 15 | 2024 December 10 |
| 7 | TDF Engineering | 2023 June 14 | 2023 December 22 |
| 8 | Processing Facility | 2023 August 17 | 2024 December 10 |
| 9 | Mine | 2023 August 17 | 2024 June 17 |
| 10 | Support Infrastructure | 2023 August 17 | 2024 April 24 |
| 11 | **Early works** | **2023 June 14** | **2024 June 17** |
| 12 | Site Prepartion | 2023 June 14 | 2023 September 19 |
| 13 | Site Mitigation Projects | 2023 July 17 | 2024 June 17 |
| 14 | **Procurement** | **2024 January 04** | **2025 June 16** |
| 15 | Mining Equipment | 2024 January 04 | 2025 March 27 |
| 16 | Process Equipment | 2024 January 04 | 2025 June 16 |
| 17 | **Mine Development** | **2024 January 04** | **2026 March 31** |
| 18 | Box Cut | 2024 January 04 | 2024 July 15 |
| 19 | Level Development | 2024 July 15 | 2025 December 24 |
| 20 | Production Ramp Up | 2025 December 24 | 2026 March 31 |
| 21 | **Construction** | **2024 May 31** | **2025 December 12** |
| 22 | Tailings Disposal Facility | 2024 May 31 | 2025 March 14 |
| 23 | Surface Water Recovery Infrastructure | 2024 May 31 | 2024 August 02 |
| 24 | Processing Facility | 2024 June 18 | 2025 December 12 |
| 25 | Powerline | 2024 October 23 | 2025 December 12 |
| 26 | Infrastructure | 2024 June 18 | 2025 September 08 |
| 27 | **Commissioning** | **2025 December 15** | **2026 April 01** |




Project: USCW - Copperwood Project
Date: 2023 March 31

| | | | | |
|---|---|---|---|---|
| Task | Project Summary | Manual Task | Start-only | Deadline |
| Split | Inactive Task | Duration-only | Finish-only | Progress |
| Milestone | Inactive Milestone | Manual Summary Rollup | External Tasks | Manual Progress |
| Summary | Inactive Summary | Manual Summary | External Milestone | |

In order to execute the schedule shown in Figure 24.3, the following activities will require continued monitoring:

- Development (routing, permiting, engineering, procurement, construction) of the powerline
- Underground mine development and ramp-up
- Process facility construction and commissioning

# 25. INTERPRETATION AND CONCLUSIONS

## 25.1 Conclusions

### 25.1.1 Geology and Mineral Resources

GMS has prepared a mineral resource estimate update for the Copperwood Project based on the original drilling database used for the April 2018 Mineral Resource upgrade. 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. James Purchase, P.Geo., Consulting Geologist for GMS, 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:

- GMS 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. GMS 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 metres, 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 2.0 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 5.5% (4.0% NSR royalty on the Copperwood Project payable to leaseholders and 1.5% NSR royalty on the Copperwood Project payable to Osisko Gold Royalties Ltd.) at USD 4.00/lb;
  2) No mining loss / dilution;
  3) Copper price of USD 4.00/lb and a silver price of USD 25/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 0.9% Cu; and
  7) Operating costs based on an operating plant at Copperwood.
- The Copperwood Deposit Total Measured and Indicated Mineral Resources are reported at 54.2 Mt grading an average 1.49% Cu and 3.6 g/t Ag containing 1.78 Blbs Cu and 6.3 Moz Ag using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined. Inferred Mineral Resources for the Copperwood Deposit are reported at 2.3 Mt grading an average 1.12% Cu and 1.2 g Ag/t containing 56 Mlbs Cu and 0.1 Moz Ag using a cut-off grade of 0.9% Cu.
- The Satellite Deposits Inferred Mineral Resources are reported at 76.8 Mt grading 1.09% Cu and 3.6 g Ag/t containing 1.84 billion pounds of copper and 8.9 million ounces of silver using a lower cut-off grade of 0.9% Cu for the LCBS and UCBS combined.

- 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.

GMS 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. GMS 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

GMS 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, P.Eng, Manager of Underground Mine Engineering with GMS, 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. A combination of continuous miner and Jumbo is the most optimal approach for this type of mining operation. Room dimensions are 6.1 m wide with height depending on the LCBS mineralization thickness. For the rooms and pillars mined with the Jumbo drill, the minimum height before dilution is 2.1 m. In the case of rooms and pillars mined with a continuous miner (road header), the minimum height is 3.0 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 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 USD 69.5/t of ore which assumes a USD 4.00/lb copper price. The Proven and Probable Mineral Reserves are estimated at 25.7 Mt with an average copper grade of 1.45% and silver grade of 3.91 g/t for 820 M lb of contained copper and 3.23 M oz of silver. The Mineral Reserve estimate includes planned dilution to respect the minimum mining height of 2.1 or 3.0 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 two-boom jumbos, mucking with ten (10) 10 t and two (2) 8 t LHDs and ground support installed with eighteen 1-boom electric-hydraulic bolters. Material handling consists of 12 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 $m^3$/s of fresh air delivered from a 5 m vent raise with two 4 m exhaust raises for each side of the mine.
- The production plan is developed to supply the mill at a nameplate capacity of 6,800 mtpd with the best available grade coming from the west mine followed by lower grade from the east mine. During the pre-production period, the main conveyor drifts are excavated, and a stockpile of ore is generated reaching a maximum amount of 350 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 ore samples 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 300 mtph with a planned availability of 95% in the third year. 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 (2) 1,200 t bins equipped with two (2) pan feeders to reclaim material to feed the SAG mill feed conveyor.
- Grinding circuit includes a 7.92 m diameter x 4.21 m EGL with a 5,500-kW motor. The ball mill will have a 5.8 m diameter x 9.0 m EGL with a 5,500-kW motor.
- Rougher flotation will consist of eight (8) 130 $m^3$ 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 (3) stages of closed-circuit cleaning. The first cleaner consists of six (6) 8 $m^3$ cells in series. The second cleaner consists of six (6) 8 $m^3$ cells in series and the third cleaner six (6) 2 $m^3$ 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 (235 $m^2$) with a target moisture of 9%.

### 25.1.4 <u>Infrastructure</u>

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 519N upgrade under responsibility of the Michigan Department of Transportation.
- Site access road (4.1 km) from the entrance of CR 519N.

- 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 m$^2$ area with HDPE liner).
- Surface water recovery of the precipitation and run-off water of the whole TDF area as the main water supply source of the project.
- Sewage treatment using stabilization ponds.
- Fuel storage (20,000 l).
- Gatehouse to control site access.
- Explosives depot.
- Truck shop (5 bays including one wash bay), warehouse (37 m x 18 m) and related offices.
- Mine dry for 325 workers.
- Metallurgical laboratory and mill offices.
- Transload facility for concentrate handling (located in Champion, MI).
- Administration offices
- Assay laboratory
- Tailings disposal facility constructed with cut and fill approach in three (3) stages with HDPE liner.
- Effluent water treatment plant for 350 USGPM constructed in Y5.
- Event pond and ditches for surface water management at mill site.

### 25.1.5 Environmental and Permitting

Major permits are in place to start construction. Some permits have additional approvals and actions, and Highland Copper is addressing those appropriately. The major environmental permits 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

The Part 55 Air Permit to Install is being amended to address on-site power generators not included in the current 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 U.S. Bureau of Alcohol, Tobacco, and Firearms
- Storage tank permits
- Mine Safety and Health Administration registration
- As the project continues to develop, there will be routine permit renewals, amendments, and modifications.

### 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 USD 425M. The total initial project capital includes a contingency of USD 37.6M, which is 9.7% of the total CAPEX before contingency, and excludes pre-production revenue of USD 33.96M. Net pre-production revenue, the initial CAPEX, is estimated at USD 391.2M.
- 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 USD 269.89M.
- The NSR for the Project during operations is estimated at USD 2,417M excluding USD 49.8M of NSR generating during pre-production and treated as pre-production revenue. The average NSR over the LoM is USD 3.65/lb of payable copper.
- The average OPEX over the LoM is USD 48.05/t of ore or USD 1.83/lb of payable copper with mining representing 50.0% of the total OPEX, or USD 24.02/t of ore.

- The undiscounted after-tax cash flow is estimated at USD 455.1M for the Project. The pre-tax $NPV_{8\%}$ is estimated at USD 221.8M with a 20.0% IRR and 3.2 y payback period. Similarly, the after-tax $NPV_{8\%}$ is estimated at USD 167.6M with a 17.6% IRR and 3.5 y 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 amendments and renewals
- Ability to attract experienced professionals
- Declining metal prices
- Development or construction schedule
- Faults creating offsets in the mineralization
- Powerline connection to grid
- Local housing and community infrastructure

**Table 25.1: Copperwood Project Risks**

| Risk | | Explanation / Potential Impact | Possible Risk Mitigation |
|---|---|---|---|
| GR1 | Permit amendments and renewals | Renewal of air permit, acquisition of dam safety permit to construct, amendment of mining permit to mine Section 5. | A quality permit application and continued discussions with regulators are 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.<br>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 benefit from the first spring and summer seasons to achieve key objectives. The critical activities in the Project schedule areis the mine development and the development of the powerline. | Route development of the powerline to be initiated prior to project moving into construction. Early works when project will move into construction will be focused on box-cut excavation to initiate mine development as soon as possible. |
| 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 | Powerline 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.<br>Another mitigation is to either bring Natural Gas to site and build a power plant at site or build a power plant next to existing Natural Gas Pipeline and built a power line to site. |
| PR4 | Local housing and community infrastructure | Copperwood will create close to 380 jobs in the area. The capacity of local communities to provide adequate housing and public services to allow the relocation of all employees may not be up to speed with the development of the project. | Discussion with local authorities to support the project. Temporary housing for workers. |

#### 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
- Ground support design criteria and mining height optimization
- Underground tailings disposal
- Oresorting
- Metallurgical recovery improvements & reagents optimization
- Financial support of local authorities
- 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 30 m 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 | 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. |
| PO3 | 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 (USD 20.3M) and the associated closure and reclamation costs. Additional environmental characterization and impact assessments would be required as well as additional permitting efforts. |

| Opportunity | | Explanation | Potential Benefit |
|---|---|---|---|
| PO4 | Metallurgical Recovery Improvements & Reagents Optimization | The copper recovery may be further optimized by concentrate grade and reagent optimization.<br>Additional characterization might be done specifically for areas where results were lower.<br>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. |
| PO5 | Concentrate Leaching | Concentrate ferric sulfate leaching including technology developed by FLSmidth was investigated as a replacement alternative in part or in totality of the production of a copper concentrate with its associated transport, smelter treatment and refining costs.<br>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, 80-90ºC, acid ferric sulfate process modified for leaching copper from primary and secondary copper sulfide concentrates.<br>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 (CSTRs). 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.<br>Additional testwork is required a full trade-off study comparing with the current processing scheme designed to produce a copper concentrate. |
| P06 | Ore Sorting | Copperwood has undergone some preliminary testwork with an OEM to determine the viability of ore sorting using the ore from the LCBS. X-Ray Transmission (XRT) proved to be a method which could adequately sort the ore. | The potential benefit is tied to a reduction of material going through the process while still recovering a signification portion of the copper. Further testwork with larger quantities of ore is required to determine its true economic benefit.<br>Additionally, sorting could potentially allow Copperwood to mine sections of the UCBS which is separated by a layer of waste. Being able to separate the waste from the ore could reduce the dilution in these areas and render them economically viable. |
| P07 | Financing from local authorities | Company to engage with the state of Michigan to support the development of local infrastructure. | Better economic return for the communities, additional source of funding for the project. |

# 26. RECOMMENDATIONS

## 26.1 Project Recommendations

Based on the results of the Feasibility Study, GMS 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 and long lead items 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.
- Review site water balance including construction schedule to optimize the precipitation and run-off water recovery.
- Detailed engineering of the tailings disposal facility and submittal for dam safety permit to construct.
- Initiate routing development of the powerline to site.

## 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 Y-5 of operations | | 1,800 |
| Consider relogging the Grey Laminated – Red Laminated contact in the historical drill holes from the 1950s to ensure a consistent interpretation. GMS 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. GMS 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 (3,500 m of total drilling, 10-15 drill holes) if warranted. | Pre-Mine Development | 10 | 1,050 |
| 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. 3,500 m of total drilling (15 drill holes) to determine an Inferred Mineral Resource. | If and when mineral rights are acquired | | 1,050 |
| **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 Continuous 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 |

| Work Program Description | Timing of Program | Estimated Cost US$ k) | |
|---|---|---|---|
| | | Core | Discretionary |
| **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 consumption optimization. | During or before detailed Eng. | 200 | |
| 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. Testwork to verify the suitability of a flash flotation in the grinding area. | Once in development ore | | 400 |
| **Initial Project and Detailed Engineering** | | | |
| Initiate detailed engineering and permitting of power line with power provider. | Post FS completion | 750 | |
| 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** | | **1,195** | **4,600** |

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