NI 43-101 TECHNICAL REPORT

Nevada Copper Corp., Pumpkin Hollow Project, Open Pit and Underground Mine Prefeasibility Study

Pumpkin Hollow

Nevada, USA

For Nevada Copper Corp.

Effective Date: January 21, 2019

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This technical report on the Pumpkin Hollow Project is submitted to Nevada Copper Corp. and is effective as of January 21, 2019.

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/s/ Robert McKnight, P.Eng. Robert McKnight, P.Eng. Nevada Copper Corp. Date Signed: April 16, 2019	Market Studies & Contracts, Economic Analysis, Transport & Shipping
/s/ Edward Minnes, P.E. Edward Minnes, P.E. Golder Associates Ltd. Date Signed: April 16, 2019	Introduction, Open Pit Mining Method, Open Pit Capital & Operating Costs, Open Pit Mineral Reserves
/s/ Maurice Mostert, FSAIMM Maurice Mostert, FSAIMM Mining Plus Canada Consulting Ltd. Date Signed: April 16, 2019	Underground Mineral Reserve, Underground Mining Method, Underground Geotechnical, Underground Infrastructure and Underground Capital & Operating Costs
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/s/ Rex Bryan, RM SME Rex Bryan, RM SME Tetra Tech Inc. Date Signed: April 16, 2019	Geological Setting and Mineralization, Deposit Type, Drilling, Exploration, Drilling, Sample Preparation Analyses and Security, Data Verification, Underground Mineral Resources

/s/ Ronald Turner, AusIMM Ronald Turner, AusIMM Golder Associates Ltd. Date Signed: April 16, 2019	Open Pit Deposit Type, Open Pit Mineral Resources
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/s/ Guy Roemer, P.E. Guy Roemer, P.E. Tetra Tech Inc. Date Signed: April 16, 2019	Water Balance

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Item 1.0 SUMMARY

1.1 General

This Technical Report (Report) describes the Pumpkin Hollow Project, consisting of approximately 17,500 acres of contiguous mineral rights near Yerington, Nevada (the Pumpkin Hollow Property or the Property) controlled indirectly by Nevada Copper Corp. (together with its subsidiaries, Nevada Copper) and its advancement based on a phased development approach of the underground and open pit deposits as stand-alone projects, referred to herein as the Underground Project and the Open Pit Project.

This Report describes the stand-alone Underground Project based on a prefeasibility study (PFS) completed in 2017. The construction of the stand-alone Underground Project is underway. This Report also describes the stand-alone Open Pit Project, based on a PFS completed in 2019. Nevada Copper intends to advance the evaluation of the stand-alone Open Pit Project with further engineering and study at this time.

In 2017, Nevada Copper retained Sedgman Canada Limited (Sedgman) and Mining Plus Canada Consulting Limited (Mining Plus) to complete a PFS for a 5,000 short tons per day (stpd) project that evaluates a potential underground copper mine, process plant and associated infrastructure, accessing the Eastern Area underground deposits. The study was completed on November 30, 2017, with an amended and restated version of the applicable technical report being filed on SEDAR on January 9, 2018. The results of the 2017 study for the Underground Project (2017 Technical Report), remain unchanged and are described in this Technical Report, with the current effective date of January 21, 2019. Nevada Copper decided on August 28, 2018, to commence construction of the stand-alone Underground Project. with construction commencing shortly thereafter. The construction is continuing at the effective date of this Report, and first production is forecast for late 2019.

On December 21, 2017, Nevada Copper Corp., Nevada Copper Inc., (NCI) and Triple Flag Mining Finance Bermuda Ltd. (Triple Flag) entered into a metals purchase and sale agreement (the Stream Agreement). The Stream Agreement is a corporate-level financing arrangement entered into as part of a larger corporate financing package. Under the Stream Agreement, Triple Flag committed to pay US$70 million (the Stream Deposit) to Nevada Copper for the future delivery by Nevada Copper of gold and silver during the life of the Underground Project. The amount of gold and silver to be delivered is to be determined with reference to 90% of the gold and silver production equivalent from the Underground Project, calculated based on a fixed ratio of 162.5 ounces of gold and 3,131 ounces of silver for each 1 million pounds of copper in concentrate produced. Nevada Copper will receive an ongoing payment of 10% of the spot price for each ounce of gold and silver delivered to Triple Flag, and while the Stream Deposit is outstanding, the difference between the cash price and the spot price will be applied against the outstanding balance of the Stream Deposit. NCI intends to sell, at market-based prices, all of its concentrate produced from the Underground Project (including copper, gold and silver) under offtake agreements with third party offtakers. NCI will need to use a portion of the proceeds received from these offtakers (or its own funds) to acquire gold and silver credits from sources other than the Underground Project to deliver to Triple Flag under the Stream Agreement.

Nevada Copper has a one-time option on March 31, 2020 to reduce the amount of gold and silver to be delivered under the Stream Agreement with such reduced amount to be determined with reference to 55% of the gold and silver production from the Underground Project (based on the fixed ratios noted above) by making a payment of US$36 million to Triple Flag, subject to certain adjustments. Nevada Copper received the full amount of the Stream Deposit on September 6, 2018 following the announcement of the decision to proceed with construction of the Underground Project.

In 2018, Nevada Copper retained Golder Associates Ltd. (Golder) and Sedgman to complete a PFS for a stand-alone Open Pit Project that evaluates a potential open pit copper mine, process plant and associated infrastructure, with a Phase I at 37,000 stpd, with a Phase II expansion to 70,000 stpd, accessing the Western Area Deposits. The PFS was completed on February 15, 2019, with the Technical Report being filed on SEDAR on April 16, 2019. The results of the PFS for the stand-alone Open Pit Project (Open Pit PFS) have an effective date of January 21, 2019.

1.2 Introduction

1.2.1 Underground

This Report discloses the prefeasibility information regarding the advancement of the Pumpkin Hollow Property through to a project that mines the Eastern Area at 5,000 stpd using underground mining techniques. This stand-alone project (separate to the Open Pit Project) is referred to as the Underground Project in this Report.

The board of Nevada Copper made the decision that the Underground Project was fully funded and made a formal construction decision on August 28, 2018. Construction activities on the Underground Project have advanced as described in this Report.

1.2.2 Open Pit

This Report discloses the prefeasibility study information regarding the development of the Western Area deposits of the Pumpkin Hollow Property as a project that mines and processes ore at a staged 37,000 stpd rate and later expands to a 70,000 stpd rate by open pit mining methods. This stand-alone project (separate to the Underground Project) is referred to as the Open Pit Project in this Report.

The Open Pit Project continues to be further evaluated through resource drilling and engineering and advanced study.

1.2.3 Technical Report

The Underground Project and Open Pit Project can be pursued as separate projects, as is the basis of current study and evaluation.

The stand-alone Underground Project is in construction, with production forecast to commence by the end 2019, while the Open Pit Project is in a study phase of development. Development options for the Property remain flexible on the timing of when, or if, to construct and commence the Open Pit Project.

Nevada Copper is constructing the Underground Project and can decide at any point in time in the future to commence the Open Pit Project. The only shared facilities for the Underground Project and Open Pit Project on the Property are the site access road and overall Property boundary.

This Report, therefore, discloses in summary form:

- A PFS on the stand-alone Underground Project
- A PFS on the stand-alone Open Pit Project

Several items in this report have two major subsections, one for each of the Underground Project and the Open Pit Project; for example, Item 16.0 - Mining Methods has major subsection Item 16.1 - Underground and subsection Item 16.2 – Open Pit. This permits the reader to have clarity as to each stand-alone project of the Pumpkin Hollow Project.

In addition, Item 22.0 includes a third major subsection where a ‘whole of property’ simple production and economic summary is presented after the stand-alone Projects, in order to present a clear view of the whole of property potential through the phased development of the stand-alone Underground and Open Pit Projects.

All QPs involved in preparing this Report either visited the Property or had reason not to visit the Property, as described in Item 2.0.

1.3 Reliance on Other Experts

Timothy M. Dyhr, Vice President, Environmental and External Relations for Nevada Copper provided expert advice on project regulatory requirements, permitting, environmental, social and socio-economic issues. Mr. Dyhr has been responsible for the permitting efforts since 2010 that have culminated in the receipt in 2015 of all key construction and operating permits for both the Underground and Open Pit Projects. Routine approvals, permits and licenses of lesser importance will be required in the future in the ordinary course.

1.4 Property Description & Location

The Pumpkin Hollow Property is located approximately seven miles southeast of Yerington, Nevada, in Lyon County. Yerington is an approximately 80-mile drive southeast of Reno. The Pumpkin Hollow Project is located in the north–south trending Mason Valley situated between the Singatse and Wassuk mountain ranges, as shown in Figure 4-1.

Nevada Copper controls approximately 17,500 acres of contiguous mineral rights near Yerington, Nevada, including approximately 10,700 acres of private land and leased patented claims comprising the Pumpkin Hollow Property. The Property contains two adjacent but unconnected copper, gold and silver deposits separated by approximately two miles. Since the Pumpkin Hollow Property was acquired by Nevada Copper in 2006, these deposits have been extensively drilled and are the subject of several previous engineering reports.

The eastern-most deposits (Eastern Area Deposits) are too deep for open pit mining, and modeling by previous engineering studies has presented them as being amenable to mining by underground methods. The western-most deposits (Western Area Deposits) are larger and shallower, and modeling by previous engineering studies has presented them as being amenable to mining by open pit methods.

The mineral and surface rights held or controlled by Nevada Copper consist of:

- Patented claims and fee land held under lease from RGGS Land & Minerals Ltd., L.P. (RGGS)
- Private surface and mineral rights acquired from the federal government in 2015
- Federal unpatented mining claims

As a result of the Yerington Land Conveyance in 2015, whereby Nevada Copper indirectly acquired federal lands surrounding the area of the Underground Project, all of the proposed facilities are contained entirely on the private lands owned and controlled by Nevada Copper and will not require approval by the federal Bureau of Land Management (BLM) pursuant to its Surface Regulations for Mining (43CFR3809).

Nevada Copper has received the Nevada state permits needed to construct and operate the Underground and Open Pit Projects, with some design changes expected to meet the design requirements in the current permits and regulations. No federal permits are required. The design changes proposed in this Report are considered “engineering design changes” (EDCs), or minor modifications, to the permits and are not a new permit or “major modification” that require a new application and public notice and review.

1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

Yerington, the Lyon County seat, is approximately seven miles northwest of the Property, and Reno is an 80 minute drive away. Local services can support a mining project as demonstrated by the closed Anaconda open pit mine nearby, which operated into the 1970s.

The area is accessible by a sealed state road network. A 120 kV power line exists east of the Property. A rail line runs north of the Property. All three infrastructure networks are proposed to be used to support the Underground Project and Open Pit Project development.

Water sources are adequate and will include groundwater pumped from the underground and open pit mines and surface runoff.

Infrastructure on the Property consists of initial mine development infrastructure including:

- A production-sized 1,900 ft deep 24 ft diameter concrete lined production shaft, production-sized headframe, over 600 ft of initial lateral mine development, hoist house with 12 ft diameter hoist, compressors, dewatering wells, diesel storage facility, explosives magazine, and monitoring wells

- Existing buildings including mine operations office, mine warehouse, mine workshop and mine dry
- Manufactured offices trailer complex, core storage buildings, a small ranch house, a local non-drinkable water source
- Minor roads/tracks
- Five existing rapid infiltration basins (RIBs), a lined pond and an irrigation area west of the shaft, some ponds, and four ponds for dewatering the current underground workings south-southwest of the shaft
- 25 kV incoming line and switchyard
- Laydown yard

A skilled workforce is available and abundant.

The climate is arid with hot summers and relatively mild winters. Nearby mining operations have no problem working year round.

1.6 History

Substantial exploration activity has been carried out on the Pumpkin Hollow Property land holdings and surrounding areas since the initial discovery by the U.S. Steel Corporation (USS) of high grade iron skarn mineralization and later copper mineralization. From 1960 to present, 1,224,253 ft has been drilled for 826 drill holes. This drilling has been undertaken by USS, Anaconda Corporation (Anaconda), Conoco Inc. (Conoco), Plexus Resources Inc. (Plexus), Cyprus Metals Exploration Corporation (Cyprus), International Taurus Resources Inc. (Taurus) and now by Nevada Copper.

Nevada Copper gained ownership of the Pumpkin Hollow Property in 2006.

In June 2006, a Mineral Resource estimate was disclosed for the Property. The 2006 Mineral Resource estimate was based on all drill hole and geological data collected through the year 1999.

Since October 2006, Nevada Copper has drilled over 600,000 ft of resource, hydrologic and geotechnical drill holes with the objective of advancing the potential mine development options. In addition, Nevada Copper initiated a program to assay and re-assay selected historic core and drill rejects for copper, gold, silver, and molybdenum. Traditionally, previous operators had not always assayed for gold, silver and molybdenum, and some core with visible chalcopyrite had not been assayed, even when within the limits of projected mining boundaries. Nevada Copper has completed several drill programs since 2006. The drilling has been considered successful in achieving its objectives of expanding the resource base and upgrading the mineral classifications.

Between 1960 and 1982, eight major geophysical surveys accentuating magnetic and electrical geophysical systems of various types were attempted on the claims, by USS, Anaconda and Conoco. Much of the data has been lost over time or is not available. An aeromagnetic survey was flown over the Property in December 1998.

1.7 Geological Setting & Mineralization

The Pumpkin Hollow Property area is located within the western Great Basin of the Basin and Range Province on the east side of the Sierra Nevada in Lyon County, Nevada. The east slope of the range is cut by a number of major north-trending normal faults delineating north-trending ranges which are connected to the main mass of the Sierra Nevada on their south ends but diverge from the range northward. The Singatse Range, which forms the western boundary of the Mason Valley, and the Wassuk Range, which forms its eastern boundary, reflect two block ranges of this type. The Property is located in the basin between these two ranges.

The Yerington district, which includes the Pumpkin Hollow Property, is located in the approximate west-central portion of Mason Valley and underlain by a sequence of Mesozoic meta-volcanic and sedimentary rocks that have been intruded and mineralized by the Jurassic-age Yerington batholith. The Mesozoic rocks were deeply eroded during Late Cretaceous and early Tertiary time and overlain by a thick sequence of Tertiary volcanic and sedimentary lithologies. All units have been tilted steeply to the west and displaced into numerous blocks by easterly dipping listric normal faults.

Granodiorite to diorite rocks belonging to the Jurassic Yerington Batholith intrude the limestones of the Triassic Mason Valley Formation and calcareous argillites and siliceous shales, siltstones and limestones of the Gardnerville Formation. Associated with this intrusive episode is the development of large areas of iron oxide-copper-gold (IOCG) mineralization, which is dominantly skarn with associated copper and magnetite mineralization with varying levels of gold and silver. The skarn occurs primarily in the middle to lower portion of the Gardnerville Formation and the upper part of the Mason Valley Formation, as well as within the intrusive granitoid itself.

1.8 Deposit Type

The northern area of mineralization in the Western Area is located 1,500 ft north of the South Deposit and is centered on a sub-horizontal, pipe-like, copper-rich, magnetite-poor skarn breccia body hosted by hornfels of the Gardnerville Formation (Northwest Deposit).

The South Deposit located in the Western Area, was the first discovery on the Pumpkin Hollow Property claims, and is a magnetite-chalcopyrite body closely associated with an intrusive contact of granodiorite into limestone of the Mason Valley Formation.

The Southeast Deposit, located 2,000 ft southeast of the South Deposit, is a 300 ft wide lens of chalcopyrite-magnetite-garnet-actinolite skarn developed within limestone of the Mason Valley Formation. The zone is unique for the Pumpkin Hollow Property due to its higher than average magnetite grades (locally up to 75%).

The East Deposit in the Eastern Area, located 7,000 ft east of the North Deposit, measures approximately 2,000 ft by 1,200 ft and consists of flat-lying to gently dipping, bedding-controlled, stacked, mineralized zones within the limestone of the Mason Valley Formation at depths of 1,400 to 2,200 ft.

The E2 Deposit in the Eastern Area is a steeply northwest-dipping lens of high grade copper-magnetite skarn breccia within the Mason Valley limestone, which lies on the hanging wall of an endoskarn sill. The chalcopyrite-magnetite mineralization follows the marble front, similar to the East Deposit. A major east-trending rotational fault appears to exist between the two deposits and results in a significant variation in the deposit orientation.

1.9 Exploration

Since being acquired by Nevada Copper, exploration at the Pumpkin Hollow Property has focused on resource drilling and development. In addition to the developments discussed in this Report, Nevada Copper will continue to advance the Pumpkin Hollow Property, including exploration drilling. Future exploration programs will focus on expanding mineralization in, and around, the known deposits as well as other targets within the Nevada Copper landholdings and potential within the district.

1.10 Drilling

From 1960 to 2018, previous operators and Nevada Copper drilled over 800 drill holes for approximately 1.2 million ft of drilling on the Property. Of that total, Nevada Copper has drilled approximately 49%.

Within the Eastern Area Deposits, a total of 9,728 ft of drilling was completed with 10 underground drill holes and one geotechnical hole within the East and E2 Deposits. The limited amount of 2015 drilling had no material effect on the existing underground mineral resource model's geometry and grades and as a result these holes are not included in the current Mineral Resource estimate for the Eastern Area Deposits (Underground Mineral Resource)

The Mineral Resource estimate for the Western Area (Open Pit Mineral Resource) included in this Report is based on the most recent drilling through July 2018.

1.11 Sample Preparation, Analysis & Security

Following multiple site visits and a diligent review of standard procedures, QPs formed the conclusion that Nevada Copper’s sample preparation, analysis and security protocols are adequate and meet generally accepted industry standards.

1.12 Data Verification

The historical drill data prior to 2006, including core and records, were kept very organized and well preserved by previous operators. Verification of the historical data was assessed by Nevada Copper and external experts and the results of this external work were validated by QPs in establishing an initial Mineral Resource estimate in June 2006.

Nevada Copper's current data collection procedures and analytical quality assurance / quality control (QA/QC) program have been reviewed and observed by QPs. The procedures in place at the Pumpkin Hollow Property meet current industry standards and requirements and in the opinion of the relevant QPs, are adequate for the purposes used in this Report. Nevada Copper staff are experienced and well versed in both the importance of procedures and the protocols to follow in order to ensure that the data being collected meets industry standards and is suitable for the preparation of a geologic resource model to support resources and reserves.

1.13 Mining Methods

1.13.1 Underground

The Underground Project is planned as a 5,000 stpd operation and has one mining area. The Eastern Area is planned to be mined by underground methods, specifically longhole stoping, with predominantly cemented paste fill (CPF) methods.

Access to the mine will be via a vertical shaft. Mining will be performed using the productive mechanized transverse longhole mining method, with CPF in the primary and some secondary stopes, and uncemented paste fill (UPF) or unconsolidated rock fill (URF) of remaining secondary stopes. While waste rock can be hoisted to the surface and disposed of on the waste rock stockpile, this is only planned during initial development, until the surface paste fill plant is commissioned. Once at steady-state production, all waste rock is planned to remain underground to be used as backfill for secondary stopes.

One production/service shaft and three ventilation/emergency egress shafts are included in the mine design. Stopes will be 100 ft high by 50 ft wide for the ES and E2 zones, and 75 ft high by 50 ft wide for the EN zone.

Mining will be carried out using longhole drilling and blasting, with ore and waste material mucked using load-haul-dumps (LHDs), direct to ore passes or to remuck bays situated for optimum materials handling. Ore material will be transported via haul trucks and/or ore passes to the coarse ore bins (COBs) for storage before being hoisted out of the mine. Haul trucks will be used to transport ore material from the remuck bays to the COBs, or to transport waste to the backfill levels. The majority of the underground mobile mining fleet will be battery powered. Primary crushing is located on the surface.

Uncrushed rock will be conveyed to skips and hoisted to the surface, then crushed and stockpiled, for either direct-feed to the process plant or stockpiling to the low grade stockpile.

For all stopes that will be backfilled using CPF or UPF, a bulkhead will be constructed at all access points and the stope will be filled with paste delivered by a piping network from the paste plant. The paste plant will be located on the surface and booster pumps will be used where necessary to transfer paste fill through the mine workings to the fill point.

1.13.2 Open Pit

The Open Pit Project is currently designed as a 37,000 stpd operation, expanding in Phase II to a 70,000 stpd operation, The Open Pit Project has two mining areas, the North Pit and the South Pit.

Open pit mining will commence mining with the North Pit, which possesses higher grade followed by the lower grade, South Pit. Four phases were designed for the North Pit and two phases were designed for the South Pit. Ore from the pits will be hauled to a primary crusher, or to the stockpile, located to the west of the pits. Waste will be hauled to the waste rock storage facility (WRSF), which was designed to accommodate waste from both the North Pit and South Pit. Once the North Pit is exhausted, South Pit waste is used to backfill the North Pit.

1.14 Recovery Methods

The grind specifications required to produce an acceptable concentrate grade at high overall copper recovery were established in the 2010 G&T program, with an optimal regrind target size determined to be a P80 of 28 µm for both underground and open pit ores.

1.14.1 Underground

The underground process plant has been designed to process 5,000 stpd of copper ore. The plant and the unit operations therein are designed to produce a marketable concentrate targeted at 26.0% Cu, or greater.

The proposed underground process plant is expected to achieve an estimated 92% Cu flotation recovery. Gold recovery is expected to be 78% and silver recovery is expected to be 70%. Concentrate moisture is expected to be <10%, and tailings cake moisture is expected to be <15%.

The plant will consist of a coarse ore storage facility, a SAG mill, a semi-autogenous ball mill crusher (SABC) comminution circuit, rougher flotation, regrind circuit and cleaner flotation, to liberate, recover and upgrade copper from underground ore. Flotation concentrate will be thickened, filtered and sent to a concentrate load-out stockpile for subsequent transport/shipping.

Dry stack tailings (DST), in conjunction with underground paste backfill, are the preferred means of final deposition, having substantially less water contained than tailings discharged directly from a concentrator. DST will be produced by thickening and filtering the final flotation tailings. The underground paste backfill portion of the tailings will be thickened, classified, filtered and combined with cement before being deposited in the underground mine workings.

Thickening and filtration of tailings allows better process water management and control. Process water will be recycled from the tailings and concentrate thickener overflows. Fresh water will generally be used only for pump gland service, mill lube cooling, SAG mill ring motor cooling, reagent preparation, and safety showers / eyewash stations.

1.14.2 Open Pit

The stand-alone process plant for the Open Pit Project is designed in two phases: Phase I and Phase II. Phase I is designed as a 37,000 stpd throughput concentrator, from run of mine (ROM) crushing through to filtered concentrate and tailings. Phase II expands the capacity to 70,000 stpd, which involves the addition of a coarse ore stockpile and process circuit of milling and flotation through to unfiltered concentrate and filtered tailings. The additional throughput of concentrate in Phase II will be filtered using Phase I's concentrate thickening and filtration equipment. The single process facility (SPF) and the unit operations therein are designed to produce a marketable concentrate targeted at 25.5% Cu or greater.

Based on historical test work on the North and South Deposits, estimated average copper recovery is 89.5% in the flotation circuit. The estimated gold and silver recoveries were 67.3% and 56.3% respectively, for the ore from both the North and South open pits. The estimated North Pit r recovery for copper was 90%, with the South Pit ore having estimated copper recovery of 88%.

The SPF will consist of a crushing station and overland conveyor, and the following for each phase: coarse ore stockpile and reclaim, a comminution (SAG mill, ball mill, pebble crushing) circuit and a flotation circuit (rougher, cleaner, cleaner-scavenger, regrind) circuit. These circuits are designed to liberate, recover and upgrade copper from the ROM ores. Flotation concentrate will be thickened, filtered and stored in concentrate containers for subsequent transport/shipping.

DST is the method of final deposition, which will have substantially less water contained than tailings discharged directly from a concentrator. DST will be produced by thickening and filtering the final flotation tailings.

Thickening and filtration of tailings allows for better process water management and control. Process water will be recycled from the tailings and concentrate thickener overflows. Fresh water will generally be used only for pump gland service, mill lube cooling, SAG mill ring motor cooling, reagent preparation and safety showers / eyewash stations.

The open pit process plant will consist of the following unit operations and facilities:

- A shared (Phase I and II) ROM pad, containing a truck dump area and primary gyratory crusher, designed for a 70,000 stpd throughput
- A single overland conveyor suitable for 70,000 stpd throughput, and a transfer station to split the crushed material into Phase I and Phase II streams. This transfer station will feed individual Phase I and Phase II coarse ore stockpiles via individual stacking conveyors
- A coarse ore stockpile reclaim system per phase
- A combined SAG/ball mill grinding circuit with hydrocyclones for classification per phase

- A SAG mill pebble crushing circuit per phase
- A rougher flotation circuit per phase
- A rougher concentrate regrinding circuit suitable for 70,000 stpd
- A 1st cleaner, 2nd cleaner and cleaner scavenger flotation circuit per phase
- A concentrate thickening and filtration circuit including a concentrate stockpile and dispatch area suitable for 70,000 stpd
- A tailings thickening and filtration circuit per phase

1.15 Mineral Resource (Underground)

The Underground Mineral Resource estimate was prepared by Tetra Tech and included in a technical report titled *'NI 43-101 Technical Report Integrated Feasibility Study Pumpkin Hollow Project Yerington, Nevada'* with an effective date for the Underground Mineral Resource of April 15, 2015.

The Underground Mineral Resource estimate was based on the results of all drilling up to the end of 2013. The 2015 and 2018 drilling has not been used to modify the current Mineral Resource estimate. Tetra Tech confirmed that there has been no material change in the current Mineral Resources estimate based on these drilling results. In addition, there has been no change in sampling protocols including drilling, sample preparation, analytical method, verification and security measures. On this basis, Tetra Tech has deemed that no revision to the current Mineral Resource estimate is required.

Mineral Resources are subdivided into classes of Measured Resources, Indicated Resources and Inferred Resources, with the level of confidence reducing with each class, respectively. Mineral Resources are reported as in situ tonnage and are not adjusted for mining losses or mining recovery. The Mineral Resources reported are inclusive of Mineral Reserves.

Table 1-1 details the Underground Mineral Resources for the Eastern Area. In addition to cutoff grade, the Underground Mineral Resources have been limited to the 0.5% Cu mineralized shell interpretation.

Table 1-1: Mineral Resource Underground Eastern Area

Category	Cutoff Grade %Cu	Tons (millions)	Grade %Cu	Contained Cu lb (millions)	Grade Au oz/st	Contained Au ozs (thousands)	Grade Ag oz/st	Contained Ag ozs (thousands)	Grade %Fe	Contained Fe Tons (millions)
Measured	0.75	12.1	1.60	389	0.006	74	0.127	1,541	18.7	2.3
Indicated	0.75	41.9	1.33	1,114	0.005	217	0.112	4,716	17.6	7.4
Measured + Indicated	0.75	54.1	1.39	1,503	0.005	291	0.116	6,257	17.8	9.6
Inferred	0.75	29.2	1.09	636	0.003	87	0.064	1,875	12.8	3.7

Notes: Includes East and E2 deposits.
Measured and Indicated Resources are stated as inclusive of reserves.
Columns may not total due to rounding.
Resources are constrained by a 0.5% Cu mineralized interpretation.
Effective date on Underground Mineral Resource is April 15, 2015.

The reader is cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.16 Mineral Resources (Open Pit)

The Open Pit Mineral Resources for the Western Area Deposits were prepared by Golder as part of this Technical Report. Geological modeling and subsequent Mineral Resource estimation was performed by under the supervision of the Golder QP in accordance with Golder internal modeling and Mineral Resource estimation guidelines and in accordance with industry best practices.

The Mineral Resources reported are inclusive of Mineral Reserves. Mineral Resources were based on a Lerch Grossman optimization using the parameters set forth in Table 1-2 and have been constrained to an optimized pit shell.

Table 1-2: Lerch Grossman Mineral Resource Optimization Parameters

Parameter	Unit	Value
Cu Price	$/lb	3.75
Waste Mining Cost	$/st	1.45
Ore Mining Cost	$/st	1.55
Incremental Mining Cost	$/st per 50 ft bench	0.03
Mining Recovery	%	100
Mining Dilution	%	0
Processing Cost	$/st	5.37
Cu Processing Recovery – North Pit	%	90
Cu Processing Recovery – South Pit	%	88
Cu Selling Cost	$/lb	0.55
Au Price ($0 selling cost)	$/toz	1,343
Ag Price ($0 selling cost)	$/toz	19.86

The Open Pit Mineral Resources are estimated to contain 134 million short tons (Mst) of Measured Mineral Resource, 419 Mst of Indicated Mineral Resource and 28 Mst of Inferred Mineral Resource at a cutoff grade of 0.12% Cu. See Table 1-3 and Table 1-4.

Table 1-3: Open Pit Measured & Indicated Mineral Resources

Confidence Category	Ore (Mst)	Average Ore Grades			Contained Metal		
		Cu (%)	Au (oz/st)	Ag (oz/st)	Cu (Mlbs)	Au (Koz)	Ag (Koz)
Measured Mineral Resources	134	0.561	0.002	0.064	1,508	255	8,593
Indicated Mineral Resources	419	0.417	0.001	0.051	3,492	623	21,185
Measured and Indicated Mineral Resources	**553**	**0.452**	**0.002**	**0.054**	**5,000**	**879**	**29,778**

Note: Effective date on Open Pit Mineral Resource is January 21, 2019.

Table 1-4: Open Pit Inferred Mineral Resources

Confidence Category	Ore (Mst)	Average Ore Grades			Contained Metal		
		Cu (%)	Au (oz/st)	Ag (oz/st)	Cu (Mlbs)	Au (Koz)	Ag (Koz)
Inferred Mineral Resources	28	0.358	0.001	0.040	197	37	1,088

The reader is cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.17 Mineral Reserve Estimates (Underground)

The estimation of proven and probable mineable reserves for the Eastern Area Deposits involved the application of several modifying factors to the Measured and Indicated Mineral Resource values as provided in the properties block models. The parameters included net smelter return (NSR) cutoff determination, stope design, external dilution and mining recovery.

The Underground Mineral Reserve base was limited to the Underground Mineral Resource of the Eastern Area deposits. After application of the modifying factors to the Underground Mineral Resource, the resulting estimated Proven and Probable Mineral Reserves are shown in Table 1-8 below.

An NSR cutoff value of $46/st ore was used, reflecting the estimated costs for mining, processing and general and administrative (G&A), based on a contractor-miner scenario until steady-state production is achieved, followed by an owner-miner scenario thereafter. The NSR cutoff value is not intended as a break-even value, rather, it is an elevated value intended to target higher grade material. Metal pricing assumptions for the Mineral Reserve estimate are $3.00/lb, $1,343/oz and $19.86/oz for copper, gold and silver, respectively. Mineable Shape Optimizer (MSO) was used to interrogate the resource block models to determine preliminary economic stope shapes with design considerations give to rock mechanics, mining method and equipment maneuvering capabilities.

The transverse longhole stoping method has been selected as optimal for all zones (East North [EN], East South [ES] and E2), based on safety, mining recovery and dilution, productivity and the ability to mine large spans given the ground conditions. Stopes will be extracted through a bottom-up sequence, reducing lead time and requirements for upfront development in most instances. In the E2 zone, there are some narrower parts of the orebody that have been identified as being favorable for longitudinal longhole stoping methods, since this will provide maximum efficiency in operating lateral development.

1.17.1 Initial Capital Costs (Underground)

The capital cost estimate (Capex) consists of direct costs, indirect costs (including Nevada Copper costs) and contingency. The initial Capex for the Underground Project is approximately $182.4 million, subject to qualifications, assumptions and exclusions, all of which are detailed in Item 21.0. The initial Capex is at a prefeasibility level with an accuracy of ±25%.

The initial capital costs estimate summary and distribution are shown in Table 1-5.

Table 1-5: Initial Capital Costs Summary

Item	$, millions
Direct Costs	
Underground mining	42.3
Process Plant (including Concentrate Handling)	59.9
Infrastructure and Tailings	49.9
Indirect Costs	
Infrastructure - EPCM Costs	7.0
Sales & Tax Use Tax on Purchased Equipment	Included in Directs
Construction Indirects	4.6
Owner's Costs	8.8
Spares and First Fills	0.6
Commissioning and Start-up	0.4
Total Indirect Costs	**21.7**
Total Direct and Indirect Costs	**173.4**
Contingency	**9.0**
Total Initial Capital	**182.4**

1.17.2 Sustaining Capital (Underground)

Sustaining capital costs over mine life total $110.6 million and includes replacement of, and additions to, underground mobile equipment; lease costs for the initial mining fleet; reclamation costs; and expenditures on the tailings storage facility. Table 1-6 shows the breakdown of the sustaining capital cost estimates. The sustaining Capex is at a prefeasibility level with an accuracy of ±25%.

Table 1-6: Life-of-Mine Total Sustaining Capital Expenditures

Area	$, millions
Underground Mine Development	67.7
Process Plant, Infrastructure and Tailings	32.3
Deferred Capital	3.5
Contingency	7.1
Total Sustaining Capital	**110.6**

1.17.3 Operating Costs (Underground)

The LOM operating costs for the Underground Project average $44.52 per ton milled. The first 1.5 years of costs are higher due to the use of a mining contractor. LOM site unit operating cash costs are as summarized in Table 1-7.

Table 1-7: Life of Mine Unit Operating Cost Summary

Area	LOM operating cost $/st-ore milled (Contractor Miner)	LOM operating cost $/st-ore milled (Owner Miner)
Mining	35.33	27.20
Processing	12.65	12.65
G&A	4.57	3.98
Total	52.55	43.83

1.17.4 Statement of Underground Mineral Reserve Estimate

Approximately 51% of the total Underground Measured and Indicated Mineral Resources were converted to a Mineral Reserve by the mine plan, due to the targeting of higher grade ore within the Deposits. The stated Proven and Probable Mineral Reserves estimate has been shown to be economic on the basis of reasonable cost assumptions and NSR values assigned to the resource model.

Table 1-8: Mineral Reserve Estimate (Underground)

Category	Tons	Cu	Au	Ag
	(millions)	%	oz/st	oz/st
Proven	7.4	1.85	0.007	0.144
Probable	16.5	1.47	0.006	0.138
Net Reserves	**23.9**	**1.59**	**0.006**	**0.139**

Note: Effective date on the Underground Mineral Reserve is September 15, 2017.

Dilution was estimated to be between 2.5% and 5.0% for primary stopes, and 10% for secondary stopes. An external dilution grade of 0.75% Cu was applied to primary stopes, and a range of 0.19% to 0.38% Cu dilution grade was applied to secondary stopes. These external dilution grades were assigned based on the underground mining method and geologic wireframe boundaries. A mining recovery rate ranging from 94.9% to 95.7% was then applied to the diluted stope shapes.

1.18 Mineral Reserve Estimates (Open Pit)

Golder was retained by Nevada Copper to complete the Open Pit PFS. This study includes mine plans (including open pit mine and dump design, production plans, mining equipment selection, and mine operating cost estimates) and determination of mine capital and operating cost estimates for the Open Pit Project. The open pit mining operations are located on the west half of the Property and include the mine rock storage facility (MRSF) and two open pits: the North Pit and the South Pit.

Economic constraints use a minimum cutoff grade, which is compared against each block within the model. If the block grade is above the designated cutoff, then the net block value before tax is calculated for the block using the assumed parameters and appropriate block tonnages estimated. If the block grade is below cutoff grade, the block is designated as waste and assigned a cost of mining for that block of material.

Golder developed an Open Pit Mineral Reserve Estimate based on the Open Pit Mineral Resource estimate discussed above. This included the scheduling of the extraction of the Open Pit Mineral Resources, applying recovery factors, plant recoveries and an economic analysis of the mine plan. Ultimate pits were designed based on Lerch Grossman pit optimization analysis and phased pit designs were developed to facilitate the development of the Life of Mine (LOM) Plan mine development and sequencing. Inputs for the Lerch Grossman optimization are provided in Table 1-9.

Table 1-9: Lerch Grossman Mineral Reserve Optimization Parameters

Parameter	Unit	Value
Waste Mining Cost[1]	$/st	1.45
Ore Mining Cost[1]	$/st	1.55
Incremental Mining Cost	$/st per 50 ft bench	0.03
Reference Elevation	ft	4,600
Mining Recovery	%	100
Mining Dilution	%	5
Processing Cost[2]	$/st	5.37
Cu Processing Recovery - North Pit	%	90
Cu Processing Recovery - South Pit	%	88
Au Processing Recovery	%	67
Ag Processing Recovery	%	56
Cu Price	$/lb	2.75
Cu Selling Cost[3]	$/lb	0.55
Au Price ($0 selling Cost)	$/oz	1,343
Ag Price ($0 selling Cost)	$/oz	19.86
Pit Slope Angle	degrees	Varied 41 to 55

Notes:

1. The average reference mining cost was based on a preliminary cost estimate. The additional cost associated to mining ore includes items such as grade control, contact definition, and material handling.
2. Includes processing, dry stack tailings, environmental expenses, and G&A.
3. Copper selling costs includes transport and royalties.

Golder ran an economic pit sensitivity analysis in Whittle using a base copper selling prices of $2.75/lb Cu by assuming revenue factors ranging from 0.58 to 1.1 times the selling price of copper to generate nested pit shells. Based on the assumed mill feed, the resultant nested pit shells were used to design potential pushbacks. Figure 16-8 provides a summary of the total waste and mill feed tonnages and resultant preliminary discounted pit value for each selling price of copper. A discount of 5% was used for the pit optimization.

The pit shell analysis shows that the highest cashflows possess a range of about 340 Mst to 430 Mst of mill feed with a strip ratio of approximately 2.7. Revenue factor (RF) 0.90 was the pit shell selected for the ultimate pit design.

A pit optimization was completed using Whittle and based on the metal prices, metallurgical recoveries, and concentrate off-take terms. A pit design was completed based on the revenue factor 0.90 Whittle pit shell, which corresponds to a $2.48/lb Cu selling price. Resource within the pit design is assumed ore at a break-even copper cutoff of 0.129% for the North Pit and 0.132% for the South Pit. These cutoffs take into account the variable process recovery. The modifying factors of 5% dilution and 98% mining recovery, were applied to the resource allocated as ore within the pit design. Only sulfide material classified as Measured and Indicated Resources are included in the Proven and Probable Reserve categories.

The Lerch Grossman pit selected was used to define the limits for the ultimate pit designs for the North and South Pits. The ultimate pit designs are shown in Figure 16-9.

The ultimate pits were split into logical mining phases for the purposes of developing the LOM Plan. Golder developed the mining sequence, equipment and manpower requirement, operating and capital costs for mining as part of the overall economic evolution of the open pit mining operation at the Pumpkin Hollow Project.

1.18.1 Initial Capital Costs (Open Pit)

The capital cost estimate for the Open Pit Project consists of initial capital costs for the open pit process facility, DST facility, mining equipment, capitalized stripping and infrastructure. A summary of the Open Pit Project initial capital costs is provided below as Table 1-10. The capital cost estimate is at a prefeasibility level with an accuracy of ±25%.

Table 1-10: Initial Capital Cost Summary

Description	Initial $, millions
Mining (including pre-strip)	128
Process (including tails filters)	427
Infrastructure	90
Dry Stack, Site Water, Env & Reclamation	7
Owner / G&A	20
Total	**672**

1.18.2 Sustaining Capital (Open Pit)

The capital cost estimate for the Open Pit Project consists of expansion and sustaining capital costs for the open pit process facility, DST facility, mining equipment, capitalized stripping and infrastructure. A summary of the expansionary and sustaining capital costs is provided below as Table 1-11.

Table 1-11: Expansion and Sustaining Capital Summary

Description	Expansion $, millions	Sustaining $, millions	Total $, millions
Mining	106	291	397
Process (including tails filters)	333	-	333
Infrastructure (including dry stack)	35	-	35
Dry Stack, Site Water, Env & Reclamation	-	120	120
G&A	-	-	-
Total	**473**	**410**	**884**

1.18.3 Operating Costs (Open Pit)

LOM operating costs for the Open Pit Project have an average of $11.51/st mill feed. LOM unit operating costs are as summarized in the Table 1-12.

Table 1-12: Operating Cost Summary

Description	$/st Mill Feed
Mining	5.71
Process (including tailings)	5.38
G&A	0.43
Total	11.51

1.18.4 Statement of Open Pit Mineral Reserves

Golder has estimated Mineral Reserves for the Western Area Deposits (Open Pit Mineral Reserves), as presented in Table 1-13.

Table 1-13: Mineral Reserve Estimate (Open Pit)

Confidence Category	Ore (Mst)	Average Ore Grades			Contained Metal		
		Cu (%)	Au (oz/st)	Ag (oz/st)	Cu (Mlbs)	Au (Koz)	Ag (Koz)
Proven Mineral Reserves (North)	75.4	0.65	0.002	0.070	983	151	5,302
Proven Mineral Reserves (South)	31.3	0.36	0.002	0.045	223	48	1,420
Proven Mineral Reserves (North + South)	106.6	0.57	0.002	0.063	1,206	199	6,722
Probable Mineral Reserves (North)	147.4	0.48	0.001	0.055	1,407	215	8,086
Probable Mineral Reserves (South)	131.7	0.37	0.002	0.049	977	203	6,458
Probable Mineral Reserves (North + South)	279.1	0.43	0.001	0.052	2,384	419	14,544
Proven and Probable Mineral Reserves	**385.7**	**0.47**	**0.002**	**0.055**	**3,590**	**617**	**21,266**

Note: Effective date on Open Pit Mineral Reserve is January 21, 2019.

Reserves are stated with 5% dilution and 98% mining recovery applied.

1.19 Market Studies & Contracts

1.19.1 Underground

There a number of possibilities for marketing the concentrates for the Pumpkin Hollow Project, including Asian, U.S. domestic and European smelters, the latter likely under a concentrates swap arrangement. For cash flow purposes, average concentrate transportation costs are estimated at $75 per wet metric tonne (wmt) based on product moved:

- Via the Port of Vancouver, Washington.
- To North American consumers.

There is an existing offtake contract covering 25.5% of the copper concentrates produced from the Underground Project.

1.19.2 Open Pit

The likely markets for the concentrates are in North America, as well as in Asia, specifically Japan, Korea and China. Rail and truck have been considered in transport options domestically and to reach a port of the West Coast of the U.S. for onward shipping.

There are no offtake contracts or agreements in place for the Open Pit Project concentrates.

1.20 Environmental Studies, Permitting & Social or Community Impact

1.20.1 Underground

1.20.1.1 Social or Community Impacts

The Underground Project occurs entirely within Lyon County, Nevada, which has historically had the highest unemployment rate in the state. The Underground Project is expected to bring about 430 direct jobs with additional indirect jobs to the area.

1.20.1.2 Approvals, Permits & Licenses

The Underground Project is fully permitted as part of the ongoing efforts for the Pumpkin Hollow Property development. The permit applications were structured to include:

- A stand-alone 6,500 stpd (maximum) underground mine and dedicated process facility
- A stand-alone 62,500 stpd (maximum) open pit mine with a different, dedicated process facility
- A combined 70,000 stpd (maximum) underground and open pit mine with a single process facility

The location of the process facility for the 6,500 stpd or 62,500 stpd cases is the same, but the throughput is different. In any case, the permits are for the “maximum throughput.” Any configuration with a lower throughput does not require a revised permit, as long as:

- The process is fundamentally the same (mine, crush, grind, float, filtered tailings, DST disposal facility).
- The environmental controls are the same for containment of process fluids and control of emissions from air emissions sources.

Most developments require changes during final design from the original permit. These changes are permit compliance items that require notification and submission of revised designs to the respective State of Nevada agencies. Items include any changes in location, configuration and/or size of environmental control facilities to ensure that the changes meet design requirements in the permits and regulations. These design changes are considered “engineering design changes” (EDCs) or minor modifications to the permit and are not a new permit or “major modification” that require a new application and public notice and review.

The Underground Project will be completed on 100% privately owned lands and as such, the development is under local and State of Nevada oversight. There is no other nexus under federal statutes and regulations that requires federal environmental permits or preparation of an environmental impact statement pursuant to the National Environmental Policy Act (NEPA). There are no endangered species located on or near the Property, no surface waters, no jurisdictional waters of the U.S. that require a permit, no designated wilderness near the Property, no Class I air quality designations, no critical habitat areas, no sage grouse (a species of concern in Nevada), and no wildlife migration zones that cause environmental constraints.

Archaeological surveys were performed on the private lands owned or controlled by Nevada Copper in 2011–2012. There are currently three prehistoric sites and two historic sites (a total of five sites) within the previously federal lands that were conveyed to Nevada Copper that are either recommended for eligibility on the national register of historic places (three sites) or require further evaluation (two sites). These sites are now administered by the Nevada State Historical Preservation Office (SHPO) pursuant to a Memorandum of Understanding among the SHPO, BLM, the City of Yerington and Nevada Copper, and will be evaluated and mitigated (data recovery, recordation and collection and recovery of artifacts [if necessary]) prior to any disturbance. In any event, none of these are within the area of disturbance of the Property. The Property area does not affect any Native American Reservation Lands or sacred sites.

Table 20-2 in this Report shows the status of Nevada Copper's mine permitting efforts at the Pumpkin Hollow Property to date.

1.20.1.3 Mine Closure

The area within the perimeter fence is approximately 1,200 acres. Of this area, a total of approximately 220 acres will be disturbed as part of mining operations for the Underground Project. A portion of this area will not be reclaimed—permanent water management diversion channels and selected infrastructure that will be retained for post-mining industrial use. Reclaimed areas will include the waste rock stockpile, low grade stockpile, DST facility, reclamation material stockpiles, infrastructure that will be removed at closure, and water management features that will be reclaimed at closure.

1.20.2 Open Pit

1.20.2.1 Social or Community Impacts

The Open Pit Project occurs entirely within Lyon County, Nevada, which has historically the highest unemployment rate in the state. The Open Pit Project is expected to bring approximately 400 direct jobs, with more during construction phase.

There have been no formal objections to the Open Pit Project from environmental groups or other non-governmental organizations.

1.20.2.2 Approvals, Permits & Licenses

The Open Pit Project has received its key construction and operating permits but will require routine approvals, permits and licenses for other components of the work. The permit applications were structured as outlined in Item 1.20.1.2 above.

Most developments require changes during final design and the Open Pit Project will require changes from the original permit. These changes are permit compliance items that require notification and submission of revised designs to the respective State of Nevada agencies. Items include any changes in location, configuration and/or size of environmental control facilities to ensure that the changes meet design requirements in the permits and regulations. These design changes are considered “engineering design changes” (EDCs) or minor modifications to the permit and are not a new permit or “major modification” that require a new application and public notice and review.

The Open Pit Project will be completed on 100% privately owned lands as a result of the Yerington Land Conveyance and as such, the development is under local and State of Nevada oversight. There is no other nexus under federal statutes and regulations that requires federal environmental permits or preparation of an environmental impact statement pursuant to the National Environmental Policy Act (NEPA). There are no endangered species located on or near the Property, no surface waters, no jurisdictional waters of the U.S. that require a permit, no designated wilderness near the Property, no Class I air quality designations, no critical habitat areas, no sage grouse (a species of concern in Nevada), and no wildlife migration zones that cause environmental constraints.

Archaeological surveys on the Property were performed, as outlined above, in Item 1.20.1.2.

1.20.2.3 Mine Closure

The area within the perimeter fence is approximately 6,700 acres. Of this area, a total of approximately 3,600 acres will be disturbed as part of mining operation. A portion of this area will not be reclaimed, including the North and South pits, permanent water management diversion channels, and select infrastructure that will be retained for post-mining industrial use. A total area of approximately 3,000 acres will be reclaimed, including the mine rock storage facilities, DST facility, reclamation material stockpiles, infrastructure that will be removed at closure, and water management features that will be reclaimed at closure.

1.21 Economic Analysis

1.21.1 Underground

Metal prices for the economic analyses employed the mean of analyst’s consensus prices for copper, gold and silver to 2022, and thereafter the prices were held constant. These metals prices are shown in Table 1-14.

Table 1-14: Metal Prices

Item	Unit	2019	2020	2021	2022+
Consensus Copper Prices	$/lb	2.83	3.05	3.14	3.20
Consensus Gold Prices	$/oz	1,276	1,285	1,284	1,325
Consensus Silver Prices	$/oz	18.77	19.40	19.53	20.01

Source: Consensus Economics Inc. - 2017.

The Consensus Economics Inc. copper price forecast of 2017 is still considered current and relevant for the purpose of this Report.

In addition to the stated price assumptions, the economics were also examined with alternate metals price scenarios, with copper prices lower and higher than current spot prices, as shown in Table 1-15. Gold and silver prices were held constant at the levels shown due to their low importance relative to copper. All prices were held constant.

The economics of the stand-alone Underground Project are summarized in Table 1-15 based upon the inputs disclosed in other sections of this Report.

Table 1-15: Comparison of Economic Analysis for Underground Project

Item	Units	Low Case	Consensus Case	High Case
Copper Price	$/lb	2.60	Consensus[2]	3.50
Gold Price	$/oz	1,300	Consensus[2]	1,300
Silver Price	$/oz	17.0	Consensus[2]	17.0
		$, millions	**$, millions**	**$, millions**
Net Smelter Revenue[1], after royalty	LOM	1,582	1,941	2,150
Operating Margin	LOM	518	876	1,085
Operating Margin	Avg/yr	40	67	83
Undiscounted Net Cash Flow	Pre-tax	224	582	791
NPV 0%	After-tax	212	496	658
NPV 5%	Pre-tax	108	356	510
NPV 5%	After-tax	100	301	421
IRR	Pre-tax (%)	13.4	27.2	36.8
IRR	After-tax (%)	12.8	25.2	33.6
Payback	After-tax (yr)	6.50	4.75	4.00

Notes:

1. Net revenues less smelter charges, concentrate transport and site operating costs.
2. Consensus prices, as shown in Table 1-14.

1.21.2 Open Pit

The stand-alone Open Pit PFS and the cost estimates and economics are prepared on a quarterly basis for the calendar years 1 to 4 after production commences and annually thereafter. Based upon design criteria presented in this Report, the level of accuracy of this estimate is considered ±25%.

The economics of the stand-alone Open Pit Project are summarized in Table 1-16 based upon the inputs disclosed in other sections of this Report, which includes the same long-term metal price assumptions as in the economic analysis for the Underground Project.

Table 1-16: Open Pit PFS Economic Analysis Summary

Item	Units	LOM	Avg/Year[1]
Production Summary			
Waste Mined	Kst	1,174,895	60,842
Tons Processed	Kst	385,693	20,300
Cu Grade	%	0.47	-
Cu-Equivalent Grade	%	0.50	-
Cu-Equivalent Grade (Yr 1-5)	%	0.65	-
Copper Recovered to Concentrate	Mlbs	3,207	-
Payable Cu Production	Mlbs	3,098	163
Payable Cu Production	Ktonnes	1,405	74
Copper Concentrate Production	Kdmt	5,704	316
Financial & Economic Indicators			
NSR (net of royalties)	$, millions	8,986	473
Operating Cost	$, millions	4,440	193
EBITDA	$, millions	4,546	239
C1 Cash Costs	$/lb-pay	1.73	-
AISC	$/lb-pay	2.03	-
		Pre-tax	**Post-tax**
NPV 5%	$, millions	1,482	1,203
NPV 7.5%	$, millions	1,042	829
IRR	%	23	21
Payback	yrs	4.5	8.1

Note:

1. Based on the PFS LOM annual plan.

1.21.3 Whole Property

A combined Underground Project and Open Pit Project scenario was prepared to provide an overview of the whole property economic analysis, although decisions to advance the stand-alone Underground and Open Pit Projects may be made at different times in a phased development approach.

For the purpose of this combined scenario, the timeline for the Underground Project is set with production commencing in at the end 2019 (noting the underground mine is in construction) and the Open Pit Project timeline assumes that construction starts in 2021 with production ramping up in 2023.

Economic input assumptions draw for the details provided throughout this study for each stand-alone Underground and Open Pit Project component of the Property. The results are based from a combination or production, revenue, costs and cashflows as in each stand-alone economic model. The "Combined NPVs" in the table below are the arithmetic sum of the individual case NPVs. However, note that the NPVs have differing start dates and will not match the NPV of the combined annual net cashflows.

A summary of the production and economic results of the whole of property analysis is shown in Table 1-17.

Table 1-17: Whole of Property Economic Analysis Summary

Parameter	Units	U/G PFS	O/P PFS	Combined: O/P & U/G	Avg/Year
Production Summary					
Waste Mined	Mtons	0	1,175	1,175	-
Tons Processed	Mtons	23.9	385.7	409.6	18.6
Production Years	Years	14	20	-	-
Cu Grade	%	1.56%	0.47%	0.53%	-
Cu-Equivalent Grade	%	1.73%	0.65%	0.71%	-
Payable Cu Production	Mlbs	671	3,098	3,768	164
Payable Cu Production	Ktonnes	304	1,405	1,709	74
Financial & Economic Indicators					
NSR (net of royalties)	$, millions	2,060	8,986	11,046	480
Operating Cost	$, millions	1,183	4,440	5,623	244
EBITDA	$, millions	877	4,546	5,423	246
C1 Cash Costs	$/lb-pay	1.81	1.73	1.74	-
AISC	$/lb-pay	2.26	2.03	2.06	-
Pre-tax					
NPV 5%[1]	$, millions	357	1,482	1,839	-
NPV 7.5%[1]	$, millions	278	1,042	1,320	-
IRR	%	27	23	24	-
Post-tax					
NPV 5%[1]	$, millions	301	1,203	1,504	-
NPV 7.5%[1]	$, millions	233	829	1,062	-
IRR	%	25	21	22	-

Notes:

1. Based on PFS LOM annual plan.

1.22 Adjacent Properties

Adjacent properties include a number of small mineral deposits that are within a few miles of the Property and have copper mineralization. Within the district and across the Mason Valley, there are several other mineral deposits/resources.

1.23 Other Relevant Data & Information

1.23.1 Underground Project Development

As of the effective date, the Underground Project is progressing on schedule in its construction phase. Underground works consisting of the production shaft and shaft stations, the ventilation shaft, and lateral development are being built and in progress. Surface works consisting of the processing plant, dry stack storage and all other surface facilities are also under construction. The earthworks are completed for the primary dry stack facilities and concrete foundations for the grinding and cyclone areas are well underway. The construction work is on schedule for the Underground Project, with production expected to commence in late 2019.

1.24 Interpretations & Conclusions

1.24.1 Underground

Sedgman concludes that the proposed development of the Underground Project is technically feasible and economically viable under the conditions described herein.

Item 25.2 outlines the detailed interpretations and conclusions of this Report for the Underground Project, including:

- Nevada Copper is not aware of any significant factors or risks that may affect access, title or the right or ability to perform work on the Property
- The Underground Mineral Reserve estimate has been prepared in accordance with industry standard methods. These estimates are based on proven mining methods, mining practices, and modeling techniques applied to the mineral resource block models for the East and E2 deposits
- Mine design and capital costs were developed to standard industry practice methods. Capital costs were typical of industry standards for the proposed scope
- Operating costs are typical of industry standards for projects of this size. Operating costs are highly dependent on labor and consumable costs and could fluctuate significantly with market conditions
- The Underground Project has been fully permitted. Proposed design changes to the existing permit are considered minor modifications and are not a new permit or “major modification.”
- There is no requirement for federal environmental permits or preparation of an environmental impact statement pursuant to the NEPA
- There are currently five archaeological sites within Nevada Copper development land. None of these sites are within the area of disturbance of the underground development. The mine development areas do not affect any Native American Reservation Lands or sacred sites

- Reclamation is anticipated to achieve approved post-mining land uses and meet the requirements of the reclamation permit to achieve full bond release based on current development understanding and assumptions
- Based on ongoing delineation drilling, continued optimization of the stoping sequence could improve the grade profile, in particular during the early years of production
- The design, estimating and execution planning undertaken for the process plant and surface infrastructure component of the Open Pit was completed by Sedgman, an Engineering, Procurement and Construction (EPC) Contractor, using working knowledge from its execution of the Underground Project

1.24.2 Open Pit

The study of the stand-alone Open Pit Project has been completed to a prefeasibility study level of accuracy. The Open Pit PFS showed that there exists a positive business case to further advance the Open Pit Project through engineering, geological work as well as further study.

There are several areas that could benefit from further examination to enhance and optimize the potential of the Open Pit Project. Additional drilling of areas with open mineralization has the potential to increase and convert mineral resources and mineral reserves. Further review of the timing of the expansion (acceleration or deferment) of production could further optimize life-of-mine economics; and depending on market (price) conditions, the open pit design could be expanded to potentially increase mineral reserves.

1.25 Recommendations

1.25.1 Underground

The study for the Underground Project has been performed to a prefeasibility study level of confidence. The Underground Project has been constructed to the point described in Item 1.23.1 and as further detailed in Item 24. Further construction is progressing on schedule.

Item 26.1 outlines additional technical recommendations to improve the level of confidence in the study of the Underground Project. The costs provided in Item 26.1 are summarized in Table 1-18.

The scopes for the costs include:

- Further infill and extensional exploration drilling, which could upgrade and/or grow the overall underground mineral resource base and improve model certainty
- Undertaking further detailed engineering and study works, as described in Item 26.1
- Undertaking a paste fill study
- Further investigating the use of international containers for transport of concentrate from the Property to overseas markets

- Undertaking further metallurgical test work on composites to continue to improve the understanding of ore properties early in the mine schedule, confirm Underground PFS assumptions, to assess optimized process conditions and allow a process guarantee to be developed
- Progressing DST, environmental, reclamation, geochemical and water management area recommendations, as listed in Item 26.1

Table 1-18: Underground Recommended Activities & Cost

Recommended Activity	Cost ($, thousands)
Resource Definition Drilling	Refer to Opex
Exploration and Condemnation Drilling	1,000
Underground Material Handling System Simulation	220
Underground Mining Alternatives	20
Underground Geotechnical	260
Optimized Open Pit Mine Planning	0
Supplemental Geotechnical Investigation	80
Supplemental Mine Planning	230
Additional Metallurgical Testing	180
Tailings, Civil Infrastructure and Geotechnical	300
Data Gathering for Reclamation	85
Geochemical Assessment	50
Water Management	400
Total	**2,825**

Note: These costs are included in Capex, Sustaining Capex and Opex described in Item 21.1.

1.25.2 Open Pit

Based on the positive results of the Open Pit PFS, Golder recommends that Nevada Copper progress the Open Pit Project to the Feasibility Study Level. This would include further refinement and optimization of the current study. As part of a Feasibility Study Golder recommends the following items which may further enhance the Project.

Item 26.2 outlines the technical recommendations with which to further develop the Open Pit Project. The cost estimates provided in Item 26.2 are summarized in Evaluate alternative modeling methodology to provide a more robust Mineral Resource estimate.

- Undertake an expansion and continuation of the condemnation drilling program in areas such as the potential WSRF and future site infrastructure
- Evaluate the usage of a sub blocked modelling approach to apply geologic dilution as an alternative to applying all dilution as part of the mining process.

- To better support the definition of the estimation domains of the different elements, evaluate the potential to develop an alteration and mineralization geological model
- Undertake infill drilling of Inferred material within the open pit design to potentially increase resources and reserves
- Undertake exploration drilling of open areas
- Continue to evaluate other potentially economic commodities, including iron, molybdenum and talc.
- Work with Nevada Energy to develop appropriate power design(s).
- Review the most appropriate time (accelerate or defer) expansion of the open pit production.
- Review the scale of the open pit to accommodate a larger final pit with more material depending on market (price) conditions.
- Undertake the next level of detailed assessment of concentrate handling options to refine design and cost options.
- Golder recommends advancing the Open Pit Project to a feasibility study.

The scope for the cost estimate (Table 1-19) includes:

- Evaluate alternative modeling methodology to provide a more robust Mineral Resource estimate.
- Undertake an expansion and continuation of the condemnation drilling program in areas such as the potential WSRF and future site infrastructure
- Evaluate the usage of a sub blocked modelling approach to apply geologic dilution as an alternative to applying all dilution as part of the mining process.
- To better support the definition of the estimation domains of the different elements, evaluate the potential to develop an alteration and mineralization geological model
- Undertake infill drilling of Inferred material within the open pit design to potentially increase resources and reserves
- Undertake exploration drilling of open areas
- Continue to evaluate other potentially economic commodities, including iron, molybdenum and talc.
- Work with Nevada Energy to develop appropriate power design(s).
- Review the most appropriate time (accelerate or defer) expansion of the open pit production.

- Review the scale of the open pit to accommodate a larger final pit with more material depending on market (price) conditions.
- Undertake the next level of detailed assessment of concentrate handling options to refine design and cost options.
- Golder recommends advancing the Open Pit Project to a feasibility study.

Table 1-19: Recommended Activities & Costs

Recommended Activity	Cost ($, thousands)
Resource Definition Drilling	4,000
Exploration & Condemnation Drilling	2,750
Additional Mining Studies	500
Additional Processing and Infrastructure Studies	450
Additional Metallurgical Testing	80
Tailings, Water, Environment and Reclamation	600
Total	**8,380**

Item 2.0 INTRODUCTION

2.1 Background Information

Several studies have been undertaken on the Pumpkin Hollow Property by Nevada Copper. A short commentary of these studies and sequence is presented below:

- In 2017, a stand-alone Underground Project PFS was completed, with an amended and restated version of the technical report being filed on SEDAR on January 9, 2018, which documents the option for developing a 5,000 stpd underground-only development, to allow a lower cost development scenario to be considered by Nevada Copper. The Underground Project has subsequently been approved and is in a process of construction. This Underground PFS remains current and is summarized in this Report.
- In 2019, a stand-alone Open Pit Project PFS was completed, and filed on SEDAR on April 16, 2019, which documents the stand-alone Open Pit Project initially at 37,000 stpd with a Phase II expansion to 70,000 stpd.

This Report includes a new PFS for a stand-alone Open Pit Project on the Property. In addition, the stand-alone Underground Project is also described in this Report and the study for the Underground Project is unchanged from the underground PFS in the 2017 Technical Report. The studies and this Report have been prepared for Nevada Copper.

2.2 Terms of Reference

The Scope of Work conducted by Golder, per the request of Nevada Copper, was to compile a Technical Report for the Pumpkin Hollow Property that includes the following:

- Stand-alone Underground Project: PFS for the Underground Project with production at 5,000 stpd. The study was completed on November 30, 2017, with an amended version of the Technical Report being filed on SEDAR on January 9, 2018. The results of the 2017 study, forms the basis for the study of the Underground Project summarized in this Technical Report. All information (other than the Underground Mineral Resource estimate and Underground Reserve estimate) relating to the stand-alone Underground Project has an effective date of January 21, 2019.
- Stand-alone Open Pit Project PFS f with production at 37,000 stpd with expansion during the life of mine to 70,000 stpd. The study was completed on January 21, 2019, with this Report being filed on SEDAR on April 16, 2019.

The intent of this Report is to summarize the stand-alone Projects for which Nevada Copper is currently considering development, or in the case of the Underground Project, has commenced construction.

This Report has been prepared in compliance with the standards prescribed by National Instrument 43-101 (NI 43-101) for Nevada Copper by Golder, Sedgman, Mining Plus, Tetra Tech, Inc., and Nevada Copper.

This Report summarizes a phased approach for the development of the Pumpkin Hollow Property by means of a stand-alone Underground Project and a stand-alone Open Pit Project. Both Projects have been studied to a prefeasibility level study of a range of options for the technical and economic viability of a mineral development that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions of the modifying factors and the evaluation of any other relevant factors, which are sufficient for QPs to estimate Mineral Resources and Mineral Reserves in accordance with industry best practices. A PFS is conducted at a lower confidence level than a feasibility study.

2.3 Sources of Information and Data

This Report is based on information collected by individual QPs and other expert contributors during site visits, discussions with Nevada Copper personnel, public information and additional information provided by Nevada Copper. The Report presents all material, scientific and technical information on the Pumpkin Hollow Project, as of the effective date.

Principal technical documents and files relating to the Pumpkin Hollow Project, used in the preparation of this Report, are listed in Item 27.0.

2.4 Units of Measure

The U.S. Units (Customary Units) have been used throughout this Report. All currency is in U.S. dollars ($) unless otherwise stated.

2.5 Qualifications and Responsibilities

The following individuals, by virtue of education, experience and professional association, are considered “Qualified Persons,” as defined in NI 43-101, and are members or good standing with appropriate professional institutions or associations. The QPs are solely responsible for their specific Report sections, as shown in Table 2-1.

Table 2-1: Qualified Person Report Item Responsibilities

Qualified Person	Company	Report Item(s) of Responsibility
Greg French	Nevada Copper Corp.	1.4, 1.5, 1.6, 4.0, 5.0, 6.0, 23.0
Bob McKnight	Nevada Copper Corp.	1.19, 1.21, 1.22, 1.23, 18.1.6, 18.1.8, 18.1.9, 18.2.6, 19.0, 21.1.10, 21.1.11, 21.2.14, 21.2.15, 22.0, 24.1, 26.1.3, 26.3.1
Edward Minnes	Golder Associates Ltd.	1.1, 1.2.2, 1.2.3, 1.3, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.13.2, 1.18, 1.24.2, 1.25.2, 2.0, 3.0, 15.2, 16.2, 21.2.1, 21.2.6, 21.2.7, 21.2.8, 21.2.9, 21.2.10, 21.2.11, 21.2.12, 21.2.13, 25.3.1, 26.2.2, 26.2.7
Maurice Mostert	Mining Plus Canada Consulting Ltd.	1.2.1, 1.13.1, 1.17, 1.24.1, 1.25.1, 15.1, 16.1, 18.1.10, 18.1.11, 21.1.1, 21.1.2, 21.1.3.1, 21.1.4, 21.1.5, 21.1.6, 21.1.7, 21.1.8, 25.2.1, 26.1.2, 26.1.5
Aleksandar Petrovic	Sedgman Canada Limited	1.14, 13.0, 17.0, 18.1.1, 18.1.2, 18.1.3, 18.1.4, 18.1.5, 18.1.7, 18.2.1, 18.2.2, 18.2.3, 18.2.4, 18.2.7, 21.1.3.2, 21.1.3.4, 21.1.3.7, 21.1.3.8, 21.1.3.9, 21.1.3.10, 21.1.3.11, 21.1.3.12, 21.1.3.13, 21.1.3.14, 21.1.3.15, 21.1.9, 21.2.2, 25.2.2, 25.2.3, 25.3.2, 25.3.3, 26.1.4 (excluding 26.1.4.5, 26.1.4.6, and 26.1.4.7), 26.2.3, 26.2.4, 26.2.5, 26.3.3
Vicki Scharnhorst	Tetra Tech Inc.	1.20, 18.1.13, 18.2.8.1, 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 20.7, 21.1.3.5, 21.1.3.16, 25.2.5, 25.3.5, 26.1.4.6, 26.2.6 (excluding 26.2.6.3), 26.3.5
Rex Bryan	Tetra Tech Inc.	1.15, 7.0, 8.0, 9.0, 10.0, 12.1, 12.2, 12.3, 12.4, 12.5, 12.6, 14.1, 26.1.1
Ronald Turner	Golder Associates Ltd.	1.16, 11.0, 12.0, 14.2, 25.1, 26.2.1
April Hussey	Tetra Tech Inc.	18.1.12, 20.8.1, 20.8.2, 25.2.6, 25.3.6
Keith Thompson	Tetra Tech Inc.	18.1.14 (excluding 18.1.14.5), 18.2.8 (excluding 18.2.8.1 and 18.2.8.6), 21.1.3.17, 21.2.3, 25.2.8, 25.3.8, 26.1.4.7
Chris Johns	Tetra Tech Inc.	18.2.5, 20.8.3, 20.8.4, 20.8.5, 21.1.3.3, 21.1.3.6, 21.2.5, 25.2.4, 25.3.4, 26.3.2
Dave Richers	Tetra Tech Inc.	18.2.9, 25.2.7, 25.3.7, 26.1.4.5, 26.2.6.3
Guy Roemer	Tetra Tech Inc.	18.1.14.5, 18.2.8.6, 21.2.4, 25.3.9, 26.3.4

2.6 Personal Inspection

Table 2-2 shows the site visit details for the QPs. Most QP site visits were led by Greg French, Vice President Exploration and Development of Nevada Copper.

Table 2-2: Site Visit Log

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Greg French	Nevada Copper	Works for Company, several visits.	Multiple	Works for Company, several visits.
Robert McKnight	Nevada Copper	Works for Company, several visits.	Multiple	Works for Company, several visits.
Edward Minnes	Golder	February 2, 2018	Alone	Mining
Maurice Mostert	Mining Plus Canada Inc.	-On site representative for Mining Plus. Ongoing time spent at property.	-	Project Familiarization, Underground tour, Surface Facilities Tour, Project Meetings
Aleksandar Petrovic	Sedgman	-	-	No Site Visit[1]
Vicki Scharnhorst	Tetra Tech	-	-	No Site Visit[1]
Rex Bryan	Tetra Tech	September 1, 2012 June 5-6, 2017	Greg French	One day in 2012 Two days in 2017
Ronald Turner	Golder	March 1, 2019	Greg French	Geology
April Hussey	Tetra Tech	September 1, 2011		Visited the Project Site
Keith Thompson	Tetra Tech	-	-	No Site Visit[1]
Chris Johns	Tetra Tech	-	-	No Site Visit[1]
Dave Richers	Tetra Tech	-	-	No Site Visit[1]
Guy Roemer	Tetra Tech	-	-	No Site Visit[1]

Note:

1. QPs for these subsections did not require personal inspection to complete their responsibilities.

2.7 Effective Date

The effective date of the mineral resource statement for the stand-alone Underground Project in this Report is April 15, 2015. There have been no material changes to the Underground Mineral Resource estimate since that date. The effective date for the mineral reserve estimate for the stand-alone Underground Project in this Report is September 15, 2017. There have been no material changes to the Underground Mineral Reserve estimate since that date.

The stand-alone Underground Project was studied to a PFS level in 2017 and summarized in the previously filed 2017 Technical Report. Nevada Copper made a formal construction decision for the Underground Project on August 28, 2018, and construction activities on the Underground Project have advanced since then. The current construction, mine plan, economic and other aspects have been reviewed against the 2017PFS and there are no material differences with respect to construction schedule, costs, mine plan, production, markets, assumptions and economic analysis. The 2017 PFS study forms the basis for Underground Project summarized in this Report. All other information relating to the stand-alone Underground Project (other than with respect to Underground Mineral Resource estimate and the Underground Mineral Reserve estimate) has an effective date of January 21, 2019.

The effective date of the mineral resource statement for the stand-alone Open Pit Project in this Technical Report is January 21, 2019. There have been no material changes to the underground Mineral Resources since that date. All other information relating to the stand-alone Open Pit Project also has an effective date of January 21, 2019.

Item 3.0 RELIANCE ON OTHER EXPERTS

In producing their parts of this report, the QPs specified in this Item relied upon on a report, opinion, or statement of another expert who is not a qualified person, or on information provided by the issuer, concerning legal, political, environmental, or tax matters relevant to the Technical Report.

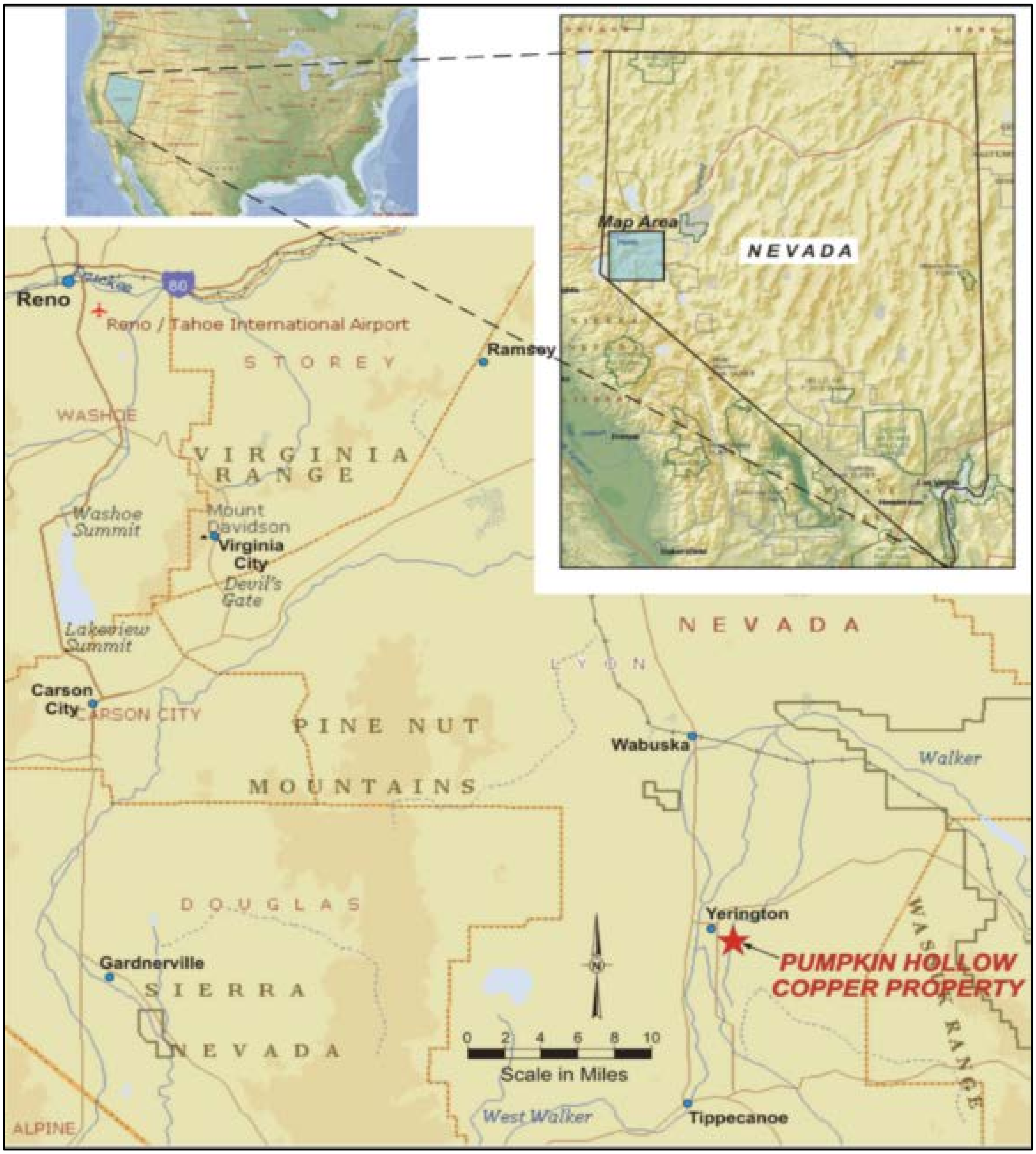
Timothy M. Dyhr, Vice President, Environmental and External Relations for Nevada Copper Company, was relied upon by the Tetra Tech QPs for non-independent knowledge of the Pumpkin Hollow Project's regulatory status, permitting, environmental and social and socio-economic issues, as disclosed in Item 20.0. This reliance is based upon Mr. Dyhr's professional qualifications and extensive permitting and environmental compliance experience in Nevada.

Item 4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The property is located approximately seven miles southeast of Yerington, Nevada, in Lyon County. Yerington is an approximate 80-mile drive (50 straight line direct miles) southeast of Reno, Nevada. The development is located in the north–south trending Mason Valley situated between the Singatse and Wassuk mountain ranges (Figure 4-1).

Figure 4-1: Property Location (2017 Technical Report)



4.2 Property

The eastern-most deposits are too deep for open pit mining, and modeling by previous engineering studies has presented them as being amenable to mining by underground methods. The western-most deposits are larger and shallower, and modeling by previous engineering studies has presented them as being amenable to mining by open pit methods.

The Underground and Open Pit Projects are located entirely within the mineral and surface rights held or controlled by Nevada Copper at the Property. These consist of:

- Patented claims and fee land held under lease from RGGS
- Private surface and mineral rights acquired from the federal government in 2015

In addition to and peripheral to the Projects, the company holds federal unpatented mining claims at the Pumpkin Hollow Property.

Approximately 10,059 acres of federal land were acquired by Nevada Copper in 2015, via Congressional legislation (the Yerington Land Conveyance), of which approximately 9,145 acres were deeded to Nevada Copper by the City of Yerington. A summary of the RGGS-leased private land, Nevada Copper–held surface and mineral rights, and federal claims held by Nevada Copper outside of the boundaries of the private lands is shown in Table 4-1. The claim blocks and land status are shown on a map (Figure 4-2).

The Yerington land conveyance boundary with the underground, open pit, and subsequent surface infrastructure proposed locations are depicted in Figure 4-3.

Except for a small amount of land held in Mineral County, the private lands are located entirely within the boundaries of the City of Yerington. All of the area of projected mine development are zoned industrial for mine development.

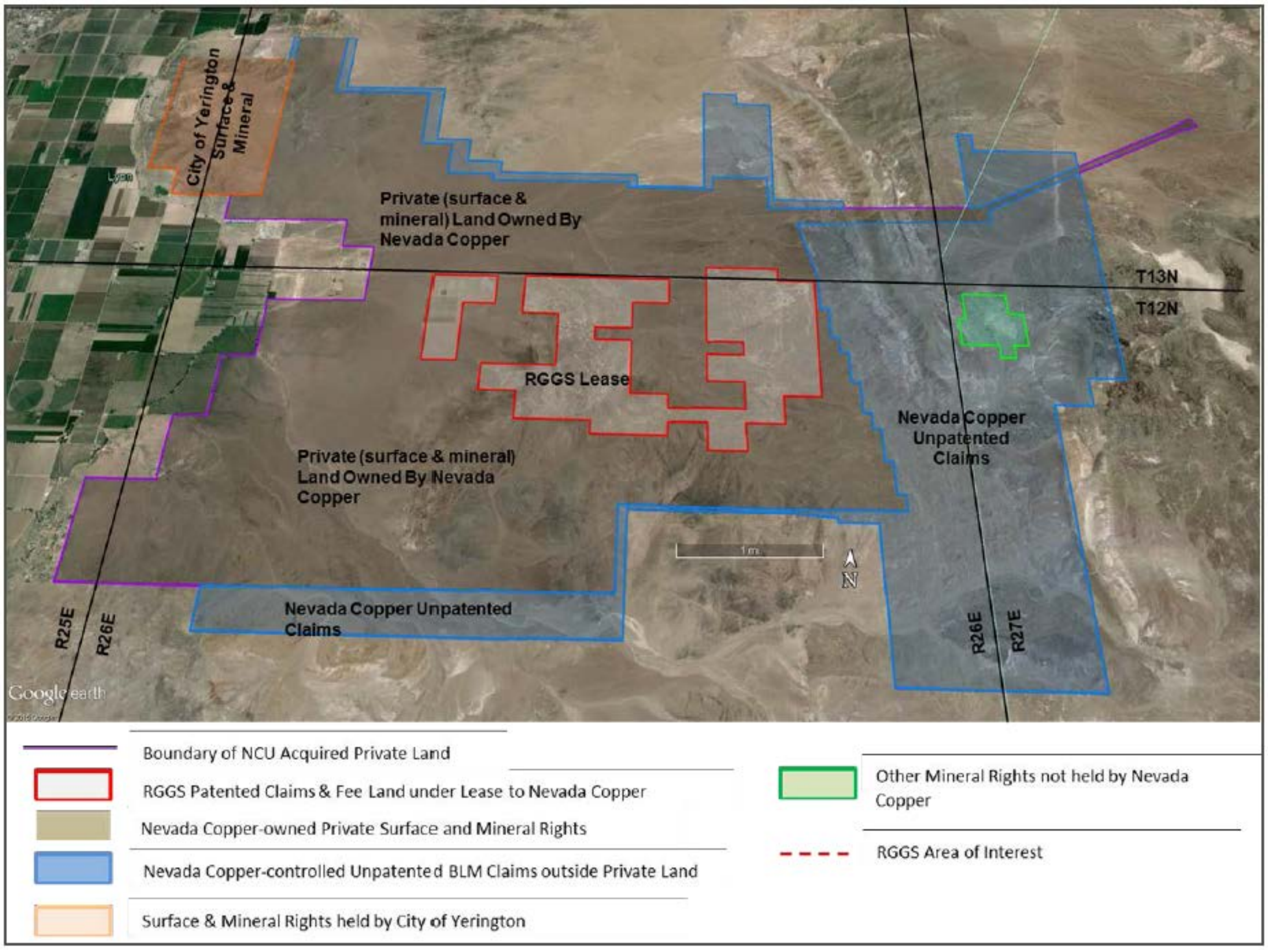
Table 4-1: Property Rights Controlled by Nevada Copper

Description	Mineral Rights Held by Nevada Copper			Surface Rights held by Nevada Copper		
	Acres	Sq. miles	Sq. km	Acres	Sq. miles	Sq. km
Nevada Copper Land acquired from BLM in Lyon County (Includes ~ 80 acres of land where common materials - sand and gravel - held by NDOT[1]	9,040.1	14.1	36.6	9,040.1	14.1	36.6
Nevada Copper Land acquired from BLM in Mineral County (surface & mineral)	105.3	0.2	0.4	105.3	0.2	0.4
Total Deeded to Nevada Copper	**9,145.4**	**14.3**	**37.0**	**9,145.4**	**14.3**	**37.0**
Private land currently held (RGGS Patented & Fee land)	1,539.6	2.4	6.2	1,537.8	2.4	6.2
Nevada Copper unpatented federal mining claims outside conveyance area[2]	6,830.0	10.7	27.6	0.0	0.0	0.0
Total Nevada Copper Controlled Lands	**17,513**	**27.4**	**70.9**	**10,683.2**	**16.7**	**43.2**

Note:

1. Nevada Department of Transportation (NDOT) has material sites on approximately 120 acres, and as pre-existing rights, are retained by NDOT, but the surface on 80 acres is now owned by Nevada Copper and the surface on 40 acres is owned by the City. Locatable minerals on all 120 acres are owned by Nevada Copper. The exact acres of these have not been surveyed, but are estimated based on unsurveyed typical (40 ac) aliquot subdivisions of the original cadastral survey.
2. Source: Summit Engineering drawing, “ANNEXMAP_24X36_20150908.”

Figure 4-2: Claim Block & Land Status



Note: Non-orthogonal, Google earth Image Nevada Copper 2018.

4.2.1 Federal Unpatented Mining Claims

The federal unpatented mining claims outside the Project areas consist of 316 PMK claims and 8 P claims.

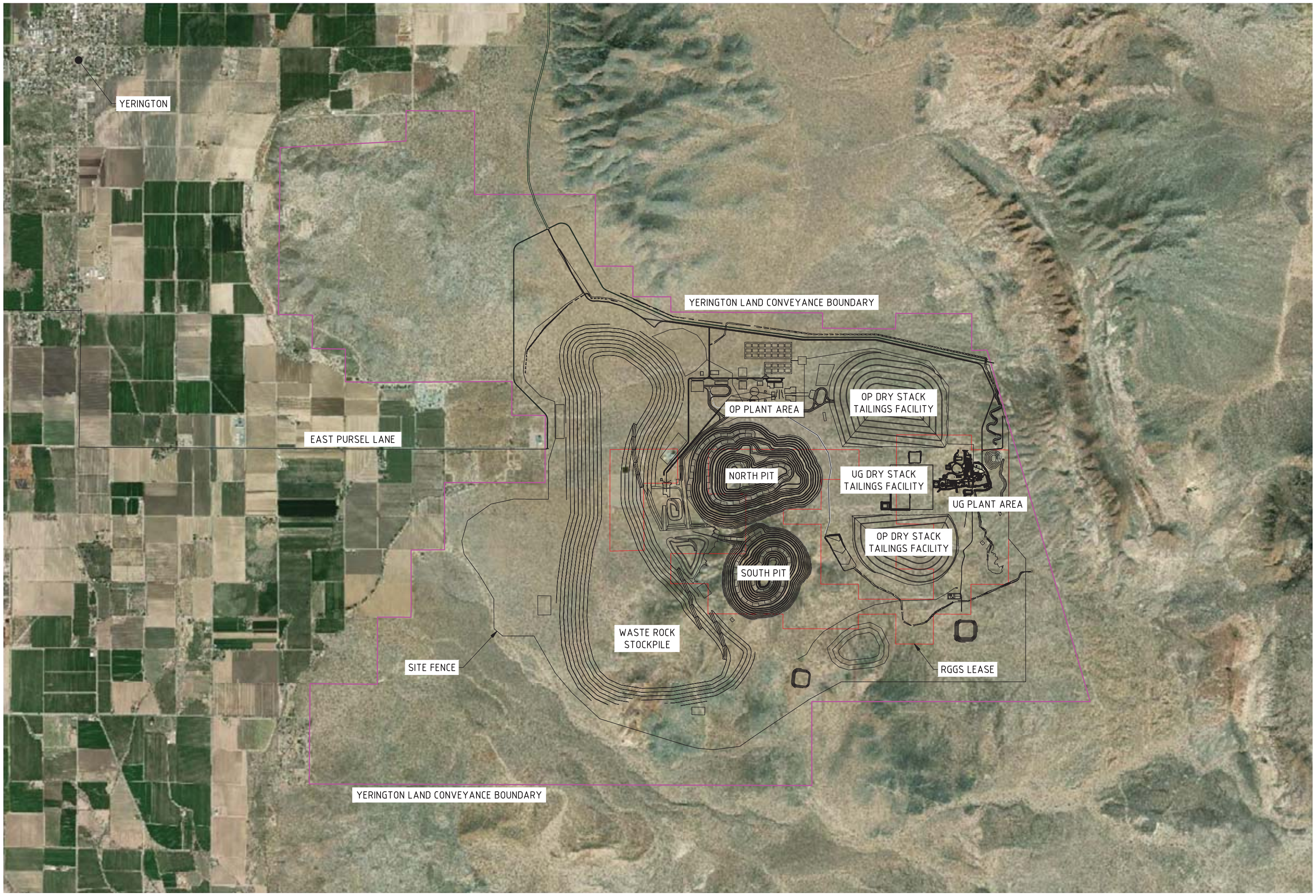


Figure 4-3: Yerington Land Conveyance Boundary with Proposed Site Layout (Sedgman, 2019)

DRAWING NO	TITLE	REV	DESCRIPTION	BY	DRG CHK	ENG CHK	DATE	APPROVED
		A	PRELIMINARY	MDC			25.03.19	

REFERENCE DRAWINGS | DRAWING REVISIONS

CLIENT NEVADA COPPER

NEVADA COPPER

DRAWN	MDC	25.03.19
CHECKED		
DESIGNED		
LEAD ENG		
APPROVED		
SCALE	AS NOTED	OR AS SHOWN
	DO NOT SCALE	ANSI D

SEDGMAN

PRELIMINARY
NOT FOR CONSTRUCTION

PROJECT PUMPKIN HOLLOW PROJECT

TITLE COPPER HANDLING AND PROCESSING PLANT SITE LOCATION DIAGRAM

PROJECT NO	DRAWING NO	REVISION
A608-P01	A608-D1-01-0003_01	A

4.2.2 Leased Patented & Fee Land

The tax roll for the Patented and Fee grounds ate the Pumpkin Hollow Project are depicted in Table 4-2.

Table 4-2: Patented & Fee Land

Patent 27-82-0003		**Patent 27-82-0005**		**Patent 27-82-0037**				**Patent 27-82-0004**	
Lyon Co. APN 001-662-02		Lyon Co. APN 001-662-04		Lyon Co. APN 001-662-03				Lyon Co. APN 001-662-05	
Tax Roll 4024215		Tax Roll 024217		Tax Roll 024216				Tax Roll 024218	
Patent Claim		Patent Claim		Patent Claim				Patent Claim	
Lyon	1	Lyon	6	Lyon	16	Lyon	88	Lyon	60
Lyon	2	Lyon	7	Lyon	17	Lyon	89	Lyon	62
Lyon	3	Lyon	8	Lyon	20	Lyon	90	Lyon	65
Lyon	4	Lyon	11	Lyon	21	Lyon	91	Lyon	66
Lyon	15	Lyon	12	Lyon	34	Lyon	92	Lyon	67
Lyon	18	Lyon	13	Lyon	35	Lyon	101	Lyon	79
Lyon	19	Lyon	24	Lyon	38	Lyon	102	Lyon	80
Lyon	93	Lyon	25	Lyon	52	Lyon	145	Lyon	82
RGGS Fee Land Patent # 1221146 61 E. Pursel Ln. (Nevada Copper offices) 159.570 acres Lyon Co. APN 001-662-01 Nevada Copper Fee Land 9145.64 acres total 9040.11 acres Lyon Co. APN 001-661-02 105.53 acres Mineral Co. APN 005-110-05		Lyon	26	Lyon	53	Lyon	146	Lyon	94
		Lyon	29	Lyon	56			Lyon	95
		Lyon	30	Lyon	57			Lyon	99
		Lyon	31	Lyon	70			Lyon	110
		Lyon	42	Lyon	71				
		Lyon	43	Lyon	72				
		Lyon	44	Lyon	73				
		Lyon	47	Lyon	74				
		Lyon	48	Lyon	75				
		Lyon	61	Lyon	85				
		Lyon	135	Lyon	86				
		Lyon	136	Lyon	87				

4.3 Other Contracts and Royalty Obligations

The portion of the concentrates derived from the Underground Project are subject to an offtake agreement. Under the contract, 25.5% of the concentrates are under a concentrates purchase agreement, initially with MF Investments.

Certain properties currently held by Nevada Copper are subject to a royalty equal to a total of 3% of the net smelter returns (the Majuba Royalty) owed to Majuba Mining Ltd. (Majuba), a Nevada corporation, as to a 50% interest, and Renegade Resources Corporation (Renegade), a Nevada corporation, as to a 50% interest, pursuant to a settlement agreement dated July 19, 2006, among NCI, Majuba and Renegade. The Majuba Royalty affects: (i) the P 32 through P 36, P 45, P 46, and P 51 unpatented mining claims held by Nevada Copper; and likely (ii) the Nevada Copper owned fee land, but only to the extent any of the now inactive P 1 through P 31, P 37 through P 44, and P 47 through P 50 unpatented mining claims previously held by Nevada Copper were previously situated on that ground.

The Stream Agreement, while not requiring delivery of gold and silver from the Underground Project, provides for delivery obligations on Nevada Copper of gold and silver referenced to production from the Underground Project. Please see Item 1.1 for a discussion of the Stream Agreement.

4.4 Land & Mineral Rights Held

RGGS, a limited partnership headquartered in Houston, Texas, is the title holder on all the patented claims and fee title land comprising the Pumpkin Hollow Property. 607792 B.C. Ltd. (BC), a private British Columbia company, acquired a lease option to a 100% interest in all the patented claims from RGGS in December 2005. The lease option contains an area of interest clause that makes any other Nevada Copper mineral claims outside the RGGS patented claims but within approximately 1 mile from the RGGS claim boundary subject to the terms of the lease but at a lower royalty rate. No mining is contemplated to occur within the area of interest.

On May 4, 2006, BC exercised its option to lease the property from RGGS. Nevada Copper acquired 100% of BC in August 2006. Nevada Copper carries out business at the Pumpkin Hollow Project through its 100% owned subsidiary, NCI, a Nevada corporation. The initial term of the lease was 10 years, and was renewed in 2016 by NCI for a further 10 year term. It is renewable for up to two more additional 10-year terms or a total of 40 years.

The Property is subject to an annual non-recoverable bonus payments totaling $175,000 over the first five years, which have been made. Commencing on the sixth anniversary and thereafter, $600,000 minimum advance royalty payments are due annually and have been made. To December 31, 2018, cumulative advance royalty payments of $4.2 million, creditable against any royalties payable, have been made.

A sliding scale royalty on copper applies to the NSR value from mine production from the patented ground leased from RGGS, as detailed in Table 4-3. The royalty rate on precious metals is 5%, regardless of metal price.

Table 4-3: Patented Land and Royalty Schedule

Commodity and Price	Royalty Payments (% of NSR)
Cu at <$1.00/lb	4.0
Cu at $1.00 to $2.00/lb	5.0
Cu at >$2.00/lb	6.0
Gold	5.0
Silver	5.0

RGGS is also entitled to a 2% NSR value royalty on any non-ferrous mineral production, and a $0.20 per ton royalty on any ferrous mineral production, that might be derived from lands held by Nevada Copper outside the leased patented claims, but within the area of interest, as defined in the lease with RGGS.

4.5 Environmental Liabilities

No environmental liabilities have been identified that will materially impede the advancement of the Pumpkin Hollow Project. Nevada Copper is responsible for surface disturbances associated with the advanced exploration activities including the areas around the existing shaft and headframe. These activities have been permitted and Nevada Copper has posted a cash bond as financial assurance to cover the costs of reclamation and revegetation.

4.6 Permitting

As a result of the Yerington Land Conveyance in 2015, all of the proposed Underground and Open Pit Project facilities are contained entirely on the private lands owned and controlled by Nevada Copper and will not require approval by the federal BLM pursuant to its Surface Regulations for Mining (43CFR3809). Other federal agencies that may require permits include the U.S. Army Corps of Engineers (USACE), the U.S. Environmental Protection Agency (EPA) and the U.S. Fish and Wildlife Service (USFWS). There is currently no nexus requiring permits from those agencies. There are no jurisdictional waters requiring a USACE permit, no surface water discharge or air permits (state delegated programs) and no endangered species that will involve USFWS.

Nevada Copper has received the all key state permits needed to construct and operate the underground mine. Within the Nevada Division of Environmental Protection (NDEP), two bureaus, the Bureau of Mining Regulation and Reclamation (BMRR) and the Bureau of Air Pollution Control (BAPC), issue key state permits. Within BMRR, the Regulation Branch issues Water Pollution Control Permits (WPCPs), the Reclamation Branch issues Reclamation Permits, and the Closure Branch reviews and approves mine closure plans. The Regulation Branch also issues permits for water discharges to the surface and groundwater mine developments, rather than the Bureau of Water Pollution Control (BWPC), which consolidates and streamlines the permitting process for mines into BMRR.

Local permits and approvals will be managed by the City of Yerington. The City has annexed all of the private land owned and controlled by Nevada Copper. As part of annexation, the City zoned the majority of the property M-1 Industrial and M-2 Special Industrial District (mining and processing).

Nevada Copper controls sufficient water rights to supply the Pumpkin Hollow Project.

4.7 Significant Factors & Risks

Nevada Copper is not aware of any significant factors or risks that may affect access, title, or the right, or ability to perform work on the Property. The surface rights to lands comprising the Property that are not privately held by Nevada Copper are leased to Nevada Copper by the BLM or private owners and Nevada Copper is not aware of any risks to access or title under these leases.

Item 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

Year-round access to the Pumpkin Hollow Property area is via U.S. Interstate 80 east from Reno, then south along Nevada State Highway 95 to Yerington. Paved and gravel roads lead directly to the Pumpkin Hollow Property from Yerington. Yerington, the Lyon County seat, is approximately seven miles northwest of the Property, and Reno is an 80-minute drive.

The area is serviced by a spur on the Union Pacific (UP) Railroad and a natural gas line to Yerington. The Fort Churchill electrical generating plant is approximately 12 miles north of the Pumpkin Hollow Property. A major power line from the Fort Churchill Power Plant is located east of the Property claims.

5.2 Climate & Physiography

The climate is arid with hot summers and relatively mild winters. Nearby mining operations have no problem working year round. The Pumpkin Hollow Property is located in a dry alluvial valley with low barren hills to the east. Topography is gentle, and sufficient flat or gently sloping land exists for placement of the facilities, tailings disposal area and waste rock.

Vegetation is sparse low brush with local grasses suitable for limited cattle grazing. The agricultural Mason Valley to the west contains numerous alfalfa and onion fields and grazing lands. These fields are watered by irrigation canals from the nearby east fork of the Walker River and from wells.

5.3 Local Resources

The township of Yerington (population 6,000) is located in Lyon County, which was the site of major open pit porphyry copper production by the Anaconda Copper Mining Company from 1953 until 1978 and of heap leach copper oxide operations by Arimetco International Inc. Mining infrastructure, including electrical power and railroad lines, remains intact and a skilled workforce is available and abundant in the local and surrounding area.

5.4 Infrastructure

Infrastructure at the Pumpkin Hollow Property consists of:

- A production-sized 1,900-ft deep, 24-ft diameter, concrete-line production shaft, production-sized headframe, over 600 ft of initial lateral mine development, house with 12-ft diameter hoist, compressors, dewatering wells, diesel storage facility, explosives magazine, and monitoring wells.
- Existing buildings including mine operations office, mine warehouse, mine workshop and mine dry.

- Manufactured offices trailer complex, core storage buildings, a small ranch house, a local non-drinkable water source.
- Minor roads/tracks.
- Five existing RIBs, a lined pond and an irrigation area west of the shaft, some ponds, and four ponds for dewatering the current underground workings south-southwest of the shaft.
- 25 kV incoming line and switchyard.
- Laydown yard.

The Property is located in a favorable area for natural resource development with significant resources to support the mining industry. Electricity and a county road to these buildings are available and remain in good condition. The City of Yerington is only seven miles away and provides good access to power, water and major roads. roads. In addition, local workforce and contractor support and general suppliers will come from the nearby cities of Reno and Carson City and the communities of Fernley and Fallon. A main railroad line is located approximately 10 miles north of the Pumpkin Hollow Property. Water supply is available from the City, if required. An oversized water line partially funded by Nevada Copper and with capacity allocated to Nevada Copper exists at the western boundary of the Pumpkin Hollow Property. High voltage 120 kV power can be secured with a connection from the Nevada Energy power lines north and east of the Property. Additional details on the planned underground development infrastructure, such as power supply, concentrate transportation, water and waste management, access roads, buildings, and tailings management are discussed in Item 18.0. Mine rock storage locations are discussed in Item 18.0.

5.5 Surface Rights

The Pumpkin Hollow Property is composed of patented claims and private lands (fee) under lease to Nevada Copper, unpatented U.S. mineral claims, and private land acquired by Nevada Copper in 2015 from the federal government. Nevada Copper has surface rights to leased patented claims and private land through lease agreement with owners and its own acquired private lands. Surface rights to unpatented U.S. mineral claims are provided through leases with the BLM. Nevada Copper has sufficient surface rights (including relevant permits) for all planned mining operations at the Property, and as discussed in Item 4.7 above, is not aware of any risks to such surface rights and rights of access.

Item 6.0 HISTORY

6.1 Exploration History

The deposit at the Pumpkin Hollow Property was discovered by USS in 1960. USS sold the property along with other assets to RGGS in 2002. Nevada Copper leased the Pumpkin Hollow Property from RGGS in early 2006 and has controlled the Property since.

Nevada Copper gained ownership of the Pumpkin Hollow Property in 2006.

In June 2006, a Mineral Resource estimate, prepared in accordance with NI 43-101, was disclosed for the Property. The 2006 Mineral Resource estimate was based on all drill hole and geological data collected through the year 1999.

Since October 2006, Nevada Copper has drilled over 600,000 ft of resource, hydrologic and geotechnical drill holes with the objective of advancing the potential mine development options. In addition, Nevada Copper initiated a program to assay and re-assay selected historic core and drill rejects for copper, gold, silver, and molybdenum. Traditionally, previous operators had not always assayed for gold, silver and molybdenum, and some core with visible chalcopyrite had not been assayed, even when within the limits of projected mining boundaries. Nevada Copper has completed several drill programs since 2006. The drilling has been considered successful in achieving its objectives of expanding the resource base and upgrading the mineral classifications.

Substantial exploration activity has been carried out on the Pumpkin Hollow Property and surrounding areas since the initial USS discovery. Prior to Nevada Copper, approximately 424 drill holes had been completed, containing approximately 594,652 ft by five major firms. Although numerous geophysical and geochemical techniques have been attempted over time, the close association of copper mineralization with magnetite has highlighted magnetic exploration programs as the favored method and, due to depth of mineralization, drilling remains the only test. Initially, the Property deposits were evaluated for their iron content by USS and later for their large bulk mineable copper potential. Only recently has any attempt been made to evaluate the Deposits as lower tonnage, but higher grade, underground copper operations with substantial precious metal by-products.

The following paragraphs describe the pertinent exploration history of the Pumpkin Hollow Property and summarize the results.

6.1.1 U.S. Steel Corporation (1960–1974)

In 1960, USS performed an extensive aerial regional aeromagnetic survey in west-central Nevada in search of iron deposits to supply its Geneva iron pellet plant located in Orem, Utah. The Pumpkin Hollow Property was included in the survey, which showed that the magnetite skarn mineralization gives rise to large and intense magnetic anomalies. Initial drilling in 1960 located the South Deposit, a magnetite-rich, low grade copper deposit. In its continuing effort to locate large open pit mineable iron deposits, USS conducted detailed helicopter and ground magnetic surveys with follow-up drilling of anomalies from the mid-1960s to the early 1970s.

USS completed approximately 282 rotary and core drill holes (392,135 ft) on the Property during this exploration phase, with drill hole depths averaging 1,390 ft and some attaining depths greater than 4,000 ft. These drill programs led to the discovery of the North, Northwest, Southeast, East and E2 deposits. The costs of developing and operating the potentially large open pit mineable South Deposit were greater than the costs of a similar deposit under USS control in Wyoming (the Atlantic City Deposit) that eventually was developed to supply the Geneva plant, and USS ceased its exploration activity at the Property.

6.1.2 The Anaconda Corporation (1974–1977)

Anaconda acquired an option on the Pumpkin Hollow Property in 1974 from USS and drilled an additional 96 drill holes (143,905 ft) on the claims emphasizing copper exploration. These holes were completed in 1977. The company's intent was to locate additional feed sources for its Yerington operation to the west, which was being depleted of its higher grade reserves.

Anaconda focused on the North Deposit, a potentially low copper grade open pit mineable or bulk mineable copper-magnetite deposit primarily contained within hornfels fine-grained calcareous sedimentary rocks of the Gardnerville Formation and through this exploration activity, subsequently discovered the Northwest Deposit, the chalcopyrite-rich, magnetite poor northwesterly continuation of the North Deposit. The company also completed several drill holes on the Southeast, East and E2 deposits.

Anaconda's target was an open pit mineable or underground bulk mineable resource, which proved to be elusive. This was also an unstable period for Anaconda as its acquisition by Atlantic Richfield was in progress and Anaconda subsequently relinquished its option.

6.1.3 Conoco, Inc. (1981–1982)

Conoco was the first to attempt evaluation of the higher grade but lower tonnage potential of the area. Conoco optioned the Pumpkin Hollow Property for one year between 1981 and 1982 and concentrated its efforts on the East and E2 deposits, where it completed 13 drill holes for 27,106 ft, focusing on the outlining of high grade Mineral Resources in the East Deposit and delineating the E2 Deposit.

In the East Deposit, drill holes C-6, C-7 and C-12 confirmed the presence of thick accumulations of high grade copper mineralization along a marble front adjacent to the granodiorite intrusive contact. In the E2 Deposit, Conoco attempted to test both the depth and strike continuation.

6.1.4 Plexus Resources, Inc. (1984–1989)

Plexus optioned the east half of the Pumpkin Hollow Property in 1984. The company performed little more than assessment requirements during its tenure and completed two drill holes in the E2 Deposit, one in 1985 and one in 1987. The drill holes completed by Plexus totaled 3,006 ft.

Its initial drill hole, PLXS851, tested the E2 Deposit between two prior Conoco intersections (C-1 and C-4) and verified continuity between these intersections.

Plexus's second drill hole was an attempt to verify the strike continuity of the deposit a slight distance to the south of PLXS851. Drill hole PLX87-N appears to have penetrated Tertiary rocks structurally emplaced and displacing the up-dip continuation of the mineralized deposit.

6.1.5 Cyprus Metals Exploration Corporation (1989–1998)

Cyprus's exploration efforts initially focused on the open pit copper potential of the North, South, and Southeast deposits and completed some limited drilling on the East Deposit. Its drilling activity was focused on the south side of the North Deposit and southwest extension of the South Deposit. Cyprus drilled 23 drill holes for a total of 20,986 ft in all zones.

6.1.5.1 Exploration 1989–1990

Cyprus acquired the property in 1989 and for two years performed little more than assessment requirements to retain its option with USS. Drill holes CLP-1 (1989) and CL-301 (1990) were completed on the South Deposit. Drill hole CLP-1 was completed to 508 ft with the initial 210 ft as a rotary drill hole. The drill hole was meant to test an induced polarization anomaly detected by Geoterrex in 1971. Altered granodiorite, hornfels and locally pyritic skarn were observed but proved barren. Drill hole CL-301, a core drill hole to 420 ft, encountered hornfels and magnetite skarn below a shallow listric fault. The magnetite skarn returned a low copper values but high iron values over a significant core length.

6.1.5.2 Exploration 1991

An initial drilling program consisting of four drill holes totaling 6,680 ft, and based on geologic interpretations from prior operators, yielded mixed results. A total of three drill holes plus the extension of an earlier drill hole were completed on the East and Northwest deposits. The deep East Deposit was tested by drill holes E91-1, E91-2 and MN-18. Drill hole E91-1 (2,230 ft) was successful in intersecting economically significant copper grades over material widths plus significant precious metal values within a limestone skarn breccia of the Mason Valley Formation. This intersection extended known mineralization an additional 250 ft to the southeast.

Cyprus's second attempt, E91-2, was rotary drilled to 1,111 ft, where it was cased and abandoned. It remains available for deepening. The drill hole had been an attempt to check for mineralization in an area where two prior attempts by Conoco had failed due to excessive drifting of the drill holes.

An attempt was made to deepen a previously completed USS drill hole. Drill hole MN-18 was re-entered and cleaned to 1,225 ft. Core drilling then continued to a depth of 1,602 ft where, unfortunately, the drill hole was lost prior to attaining the desired depth.

The Northwest Deposit was tested by Cyprus by one angle drill hole, NW91-1. The drill hole was an attempt to intersect the down-dip continuation of a wide, high grade copper zone penetrated by Anaconda in drill hole USS-44. The Cyprus drill hole encountered significant copper grades at a distance of 155 ft down-dip from the Anaconda drill hole, which suggests the zone may be faulted off to the northwest.

6.1.5.3 Exploration 1992

In 1992, Cyprus re-logged all pertinent drill holes and developed new geologic models based on a re-interpretation of data from the Northwest, East and E2 deposits. Cyprus’s recalculation of the mineralized intercepts of the North Deposit resulted in a new in-house open pit resource, but this is not considered current. A second drilling phase commenced in 1992 to confirm and test for the presence of additional shallow mineralization in the North Deposit.

Five drill holes (N92-1 to N92-5) were completed for a total of 4,526 ft and managed to upgrade previously determined Inferred Resources to the Indicated category; however, the intersections remained of low tenor.

6.1.5.4 Exploration 1993

Five additional core drill holes (N93-1 to N93-5) were completed in 1993 on the North Deposit, for a total of 3,422 ft. The program tested the continuation of the shallow mineralization defined by past workers in a southwesterly direction. The program proved disappointing in that the complicated faulting patterns appear to have substantial vertical displacements and that shallow mineralization does not appear to continue in this direction.

Cyprus's final exploration attempt at the Property consisted of 12 reverse circulation drill holes (S93-6 to S93-17) completed in 1993 on the South Deposit, for a total of 5,425 ft as a test of the ground peripheral to the South Deposit magnetite deposit. Three of the drill holes returned grades and intercepts of note (all below 2% Cu) in the southwest extension area.

6.1.5.5 Geophysical Exploration

Geophysical surveys conducted by Cyprus included an induced polarization program over portions of the North Deposit in 1993 utilizing a 400 ft dipole spacing. Due to the short dipole spacing, results from the survey proved less than useful; however, past surveys by Geoterrex and Kennecott, with much wider dipole spacing, indicated the presence of intense and widespread responses at depth.

The contrast between barren Tertiary volcanic rocks overlying pyritic hornfels, skarn and marble should allow for mapping of blind sulfide accumulation. Results suggest the presence of sulfides at increasing depth to the west and south of the North Deposit. One anomaly was tested by drilling and resulted in the intersection of barren pyritic (2% to 10%) hornfels under a shallow alluvial cover.

6.1.5.6 Geochemical Exploration

Due to the alluvial cover and the presence of thick accumulations of Tertiary rocks overlying the mineralized units, geochemical prospecting in the area has been limited. A geochemical survey, consisting of approximately 500 soil samples, was carried out by Cyprus west of the South Deposit in 1993. Results were subdued; however, one anomaly composed of +200 parts per million (ppm) copper, lead and zinc (Ridge

Prospect) was located coincident with an induced polarization anomaly outlined in a survey conducted by Geoterrex in 1971. The area of interest is localized within pre-mineralization Triassic sediments.

In conjunction with the soil sampling survey, an experimental soil gas survey by the U.S. Geological Survey (USGS) was commissioned by Cyprus. Two experimental traverses were completed over the North and South Deposits. One line transected the width of the North Deposit while the South Deposit was tested along its length. Readings were taken every 200 ft along these lines.

Positive hydrogen (H_2) and carbon dioxide (CO_2) anomalies were present over both mineralized areas and negative oxygen (O_2) anomalies occurred over the central portion of the North Deposit. Antithetical CO_2 and O_2 anomalies were noted in the area of the soil anomalies at Ridge Prospect. This feature has been interpreted to reflect oxidizing sulfides at depth.

6.1.6 International Taurus Resources Inc. (1998–2000)

In September 1998, International Taurus Resources Inc. (Taurus) signed an exploration agreement with Cyprus to explore the copper-magnetite skarn property. USS held the underlying royalty on copper and any other product from the Property.

Following due diligence assay sampling and several evaluation reports, Taurus completed a 398 line kilometer aeromagnetic survey in 1998. Extensive drilling over the previous 25 years had tested the most prominent magnetic anomalies on the property. However, several magnetic anomalies, located near the eastern Property boundary, could represent smaller, untested skarn deposits.

In early 1999, eight drill holes totaling 7,513 ft successfully tested for continuity of the high grade chalcopyrite mineralization, centered on skarn breccia in the Northwest Deposit (now combined with the North Deposit).

During the initial stages of a prefeasibility investigation of the Northwest Deposit, SRK Consulting examined the Mineral Resource and the potential ground conditions for an underground mining scenario. Drill data from Taurus's drill program and Cyprus relogs of earlier drill holes indicated the high grade potential that exists within the Northwest Deposit. It is not indicative of the total potential of the Northwest Deposit because it only concentrated on potentially underground mineable material.

6.1.6.1 1998 Program

Prior to signing the exploration agreement, Taurus conducted due diligence sampling and commissioned an evaluation report and a scoping study.

Taurus commissioned Laxey Mining Services to review the economic potential of the high grade, underground, mineralization in the Northwest, East and E2 deposits and to prepare an estimate of capital and operating costs for an underground mine-surface mill complex. Laxey Mining Services developed a report that included a capital and operating cost estimate for a 3,400 stpd underground mine.

6.1.6.2 1999 Exploration Program

Since the Northwest Deposit's shallower depth and thinner Quaternary overburden will make both drill testing and possible future development considerably cheaper than in the East or E2 deposits, Taurus elected to begin drill testing there first. The February to mid-April drill program was designed to demonstrate continuity in the high grade chalcopyrite mineralization.

Eight reverse circulation pre-collars, totaling 4,037 ft, were followed by six HX-sized core tails for an additional 3,476 ft. All drill holes were gyroscopically surveyed on completion of drilling. Prior to demobilizing the drill rig, core drill holes were filled with either a cement grout or granular bentonite for their entire length. Steel caps were welded on the surface casing of Drill holes 99-4 and 99-5, which were not completed with core tails.

The silicate skarn and skarn breccia are limited on the northwest by a steeply dipping zone of faults and post mineral dikes up to 100 ft wide. The skarn and surrounding hornfels are also affected by splays of the listric faults, which separate these units from the overlying post-mineral Tertiary tuff and welded tuff.

The infill core drilling by Taurus successfully intersected wide intervals of chalcopyrite mineralization centered on the skarn breccia body confirming the existing high grade mineralized core. The drilling also encountered broken ground in the hanging wall over the northern portion of the mineralization and numerous zones of retrograde talc alteration within the mineralization.

6.2 Geophysical Exploration

The original discovery of the copper-magnetite deposits on the Property came as a direct result of a regional airborne magnetic survey conducted by USS in late 1959 to 1960. The discovery Drill hole L-1 was collared on a classic magnetic high caused by the South Deposit. Between 1960 and 1982, eight major geophysical surveys accentuating magnetic and electrical geophysical systems of various types were attempted on the claims by USS, Anaconda and Conoco. Although much of the data have been lost over time or are not available, it is known that each type of survey produced some anomalous features. The most encompassing surveys, other than magnetic, appear to be the 1964 MacPhar Geophysics and the Geoterrex Limited induced polarization programs, which delineate sulfide mineralization in areas removed from magnetite deposition.

6.2.1 1998 Aeromagnetic Survey

A 398 line km, fixed-wing, high resolution aeromagnetic survey was flown over the property by High-Sense Geophysics in December 1998. The survey was designed to provide more resolution over the main magnetic anomalies than the 1960 USS survey and to investigate several smaller magnetic anomalies along the eastern property boundary. Stratagex Geophysical Consulting reviewed the High-Sense data and estimated depths and widths for most of the magnetic sources. Calculated source depths are generally in good agreement with drill indicated depths to significant magnetite skarn intersections. Significant differences between calculated and drilled depths to magnetite skarn could result from a substantial thickness of overlying, weakly magnetic hornfels. The weaker magnetic sources, located immediately east of the main, drill-tested skarn bodies, are interpreted to be generally shallow, 10 m to 150 m (32 ft to 492 ft) below surface, with weaker magnetic susceptibilities than the main skarn anomalies. These weaker magnetic anomalies might represent the upper portions of the main skarn deposits offset along flat faults.

Although it is beyond the scope of this Report to detail all the geophysical surveys that have been conducted on the property, the varied magnetic, gravity, and induced polarization surveys conducted over the property constantly display anomalous values over the North and South deposits but are less productive over the remainder of the area. However, the most productive geophysical tools have been the use of magnetic and induced polarization surveys.

The close spatial association of chalcopyrite with magnetite skarns renders the magnetic geophysical method the most useful in detailing prospective areas of copper accumulation, while the association of induced polarization anomalies with known copper deposits on the flanks of magnetite skarns indicates that both methods utilized together may maximize results, especially in areas adjacent to the chalcopyrite-enriched marble front.

Item 7.0 GEOLOGICAL SETTING & MINERALIZATION

7.1 General Geologic Setting

The Pumpkin Hollow Copper Property is located within the western Great Basin of the Basin and Range Province on the east side of the Sierra Nevada in Lyon County, Nevada. The east slope of the Sierra Nevada is complex and irregular and the physiographic and structural break between the Sierra Nevada and the Basin and Range is gradational.

The east slope of the range is cut by a number of major north-trending normal faults delineating north-trending ranges that are connected to the main mass of the Sierra Nevada on their south ends but diverge from the range northward. The Singatse Range, which forms the western boundary of the Mason Valley, and the Wassuk Range, which forms its eastern boundary, reflect two block ranges of this type. The Property is located in the basin between these two ranges.

The Singatse Range is a prominent north-trending range approximately 25 miles in length flanked to the east by the Mason Valley. It is primarily a west-tilted fault block consisting of pre-Tertiary granitic and metamorphic sedimentary, volcanic, and volcanoclastic rocks unconformably capped by several thousand ft of westerly dipping Tertiary rhyolitic tuffs, andesites, sedimentary rocks, and basalts. The eastern front of the range is an irregular normal fault scarp with indicated movements of 1,500 ft down-throw on the east.

The Wassuk Range to the east is similarly asymmetrical in section to the Singatse Range as it displays a steep eastern slope and gradual western slope into Mason Valley. Thick sequences of Tertiary rhyolite tuffs east of the property on the western slope of the range unconformably overlay Triassic granodiorites and dip moderately 30° to the west, reflecting a large west-tilted block.

The Yerington District, which includes the property, is located in the approximate west-central portion of Mason Valley and underlain by a sequence of Mesozoic meta-volcanic and sedimentary rocks that have been intruded and mineralized by the Jurassic-age Yerington batholith. The Mesozoic rocks were deeply eroded during late Cretaceous and early Tertiary time and overlain by a thick sequence of Tertiary volcanic and sedimentary lithologies. All units have been tilted steeply to the west and are displaced into numerous blocks by easterly dipping listric normal faults.

The geology of the Yerington area incorporates portions of the Singatse Range in the west, the Property in the Mason Valley Basin, and the extreme western portion of the Wassuk Range to the east. The stratigraphic relationships between Mesozoic volcanic and sedimentary units of immediate economic importance in the Yerington area are shown in the stratigraphic column included as Figure 7-1. Figure 7-2 depicts the general geologic map of the Property area.

Figure 7-1: General Stratigraphic Column of the Pumpkin Hollow Area (2017 Technical Report)

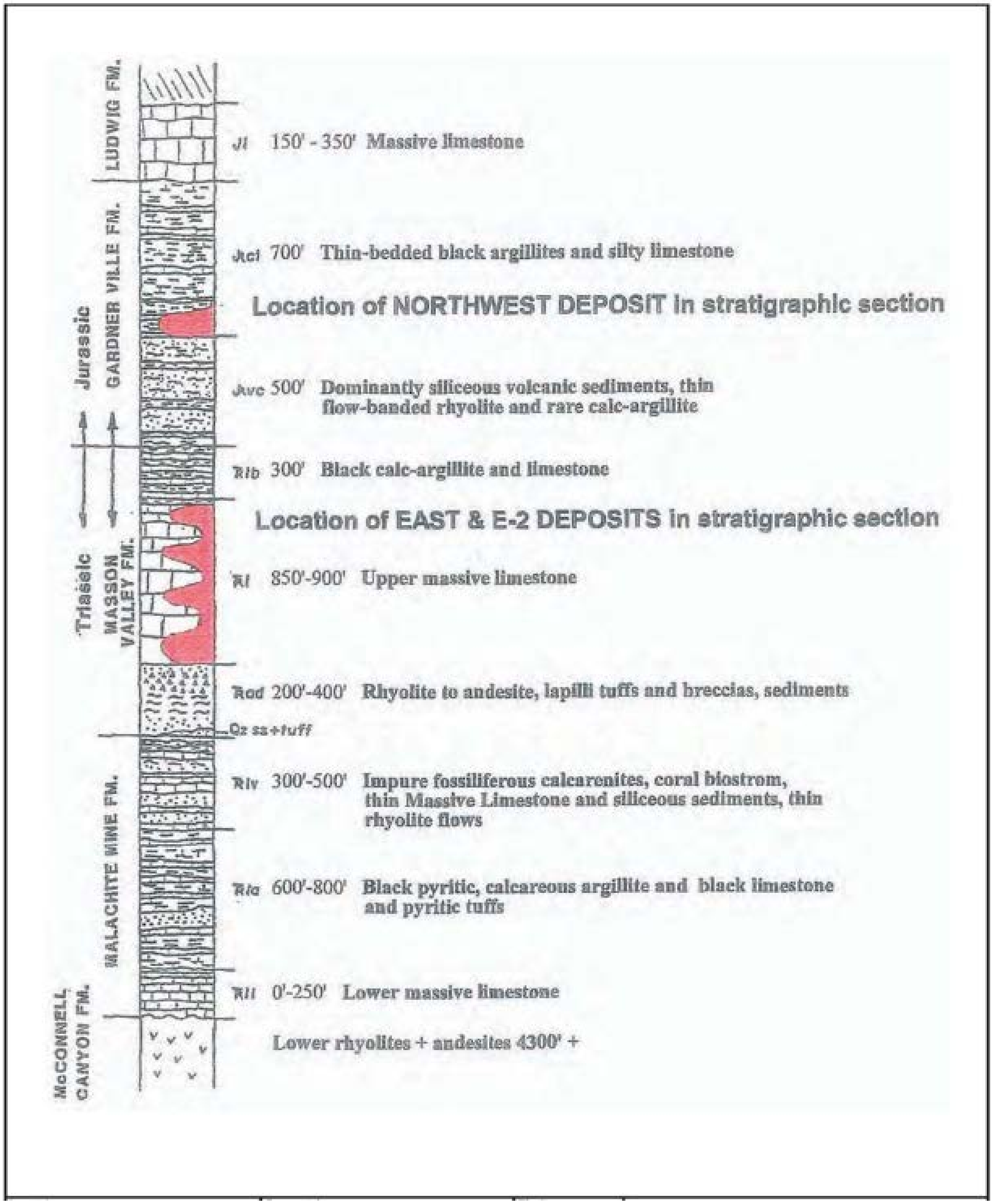
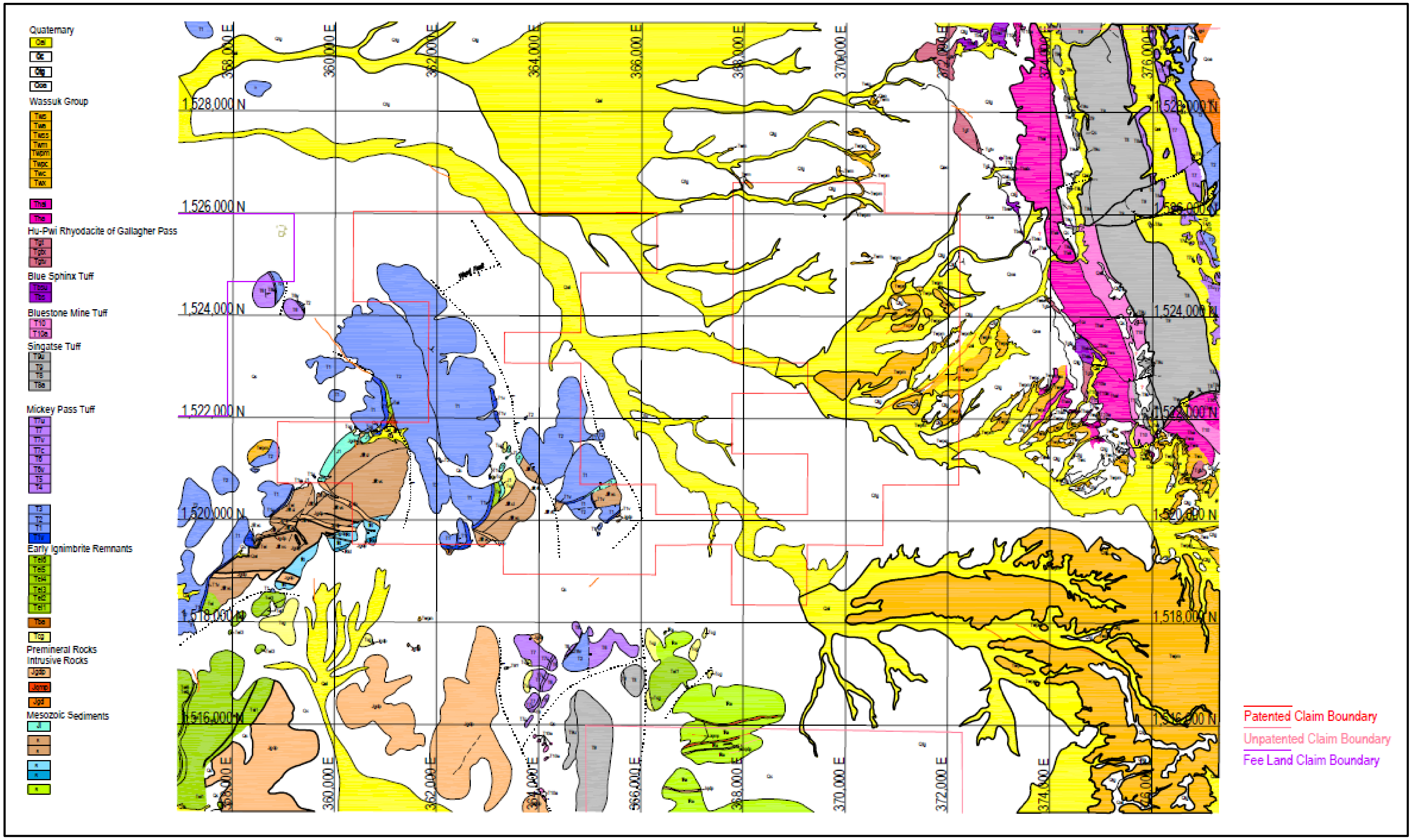


Figure 7-2: General Geologic Map of the Development Area (grid in ft & geographic north)



The oldest rocks exposed in the Yerington area and surrounding region of west-central Nevada are a sequence of metamorphosed volcanic and sedimentary rocks associated with volcanic arc formation in effect at the western margin of the North American continent during early Mesozoic time. From oldest to youngest, rock units consist of a Triassic andesitic to rhyolitic volcanic sequence (McConnell Canyon Formation), overlain by a sequence of interbedded fine clastic sediments, carbonates, tuffaceous sediments and tuffs known as the Malachite Mine Formation. This sequence of volcanic rocks is overlain by a limestone and calcareous argillite unit (Mason Valley Formation), in turn overlain by Upper Triassic to Lower Jurassic siltstones, argillites, and silty limestone of the Gardnerville Formation. Finally, the Ludwig Formation, consisting primarily of limestone, gypsum and quartzites, overlies the Gardnerville Formation.

This sequence of rocks is overlain by Middle Jurassic volcanic rocks (andesites of the Artesia Lake Formation and latites of the Fulstone Spring Formation). The sedimentary-volcanic package, which totals approximately 10,000 ft in thickness, was folded and metamorphosed during emplacement of Middle Jurassic and Cretaceous-age granitoid intrusives.

Mesozoic plutonic rocks comprise 80% of the exposed pre-Tertiary rocks in the Yerington District and the area is considered part of the Sierra Nevada Batholith. An episode of igneous activity in the Middle Jurassic resulted in the emplacement of several large batholiths of granodioritic to quartz monzonitic composition. The Yerington batholith, emplaced at 169 million years (Ma), hosts three porphyry deposits (Yerington Mine, Ann Mason, and The Bear-MacArthur-Lagomarsino) and is associated with several magnetite and copper skarns (Minnesota, Ludwig, Castings Copper, Douglas Hill, Bluestone, Mason Valley, Western Nevada Mine, and the deposits).

The Yerington Batholith is a composite calc-alkaline intrusive complex, covering approximately nine by nine miles and displays a compositional range from hornblende-quartz monzodiorite through hornblende-biotite quartz monzonite to late-stage hornblende-biotite granite. The youngest intrusions, closely associated with porphyry copper mineralization, comprise strongly porphyritic granite dikes. The porphyry dikes occur in three separate swarms within the batholith, each localized over individual cupolas within a deeper granite pluton. The dike swarms strike northwest and have steep dips controlled by regional fracture patterns.

The Shamrock Batholith, emplaced at 165 Ma, truncates the Yerington Batholith to the south and is barren of any significant copper mineralization or related skarns. Late Cretaceous plutons present in the Wassuk Range are similarly barren. These observations strongly indicate the Yerington Batholith is the only productive pluton in the district.

All Mesozoic rocks were deeply eroded during late Cretaceous and early Tertiary times and then unconformably overlain by a thick succession of Tertiary volcanic and sedimentary rocks. The entire package was then tilted steeply westwards and structurally fragmented into numerous blocks through easterly dipping normal faulting.

7.2 Property Geology

The Property is located on the eastern edge of the Quaternary gravel covered Mason Valley. The Property is elevated from the surrounding farmland (>400 ft) exposing bedrock that primarily consists of a Tertiary (Oligocene) tuffaceous volcanic assemblage. The volcanic package consists of quartz-latite ash flows and crystal tuffs of the Mickey Pass Tuff unit overlain by the Singatse Tuff Formation, which includes lithic-rich tuffs. The Singatse Tuff is conformably overlain by units of the Bluestone Mine Tuff Formation composed of unwelded rhyolitic tuffs and ash flows. The volcanic package strikes to the north-northwest with steep westerly dips.

The southwestern portion of the Pumpkin Hollow Property is dominated by Mesozoic units of early Jurassic (possible Gardnerville Formation) composed of fine-grained felsitic siltstones and tuffs overlain by thin bedded carbonaceous, calcareous argillites, tuffs and limestones. These units are conformably overlain by blue-gray laminated limestones of the Ludwig Formation. Locally medium blue, carbonaceous skarns of the late Triassic Mason Valley Formation are present and contain small copper showings. Small-scale pitting was carried out on these showings in the past. The general trend of the pre-Tertiary rocks is to the northeast with steep dips to the northwest.

East–west striking porphyritic quartz monzonite dikes cut across the pre-Tertiary sedimentary package. These rock types of Jurassic age are synchronous with late-stage phases of the Yerington Batholith.

Most of the geological relationships in the Mesozoic are based on drill hole data. Drill-derived data reveal the preponderance of variably inclined limestones of the Mason Valley Formation and calcareous sediments of the Gardnerville Formation of Triassic-Jurassic age intruded by Jurassic granodiorite and diorite. The pre-Tertiary setting in the area is suggestive of a large, centrally located intrusive body flanked in semi-circular fashion by altered and mineralized portions of its host, the Mason Valley and Gardnerville formations.

The Mesozoic-Tertiary contact is a low-angle structural one that appears to displace the Mesozoic stratigraphy to the east. Later structural rotation to the west of the entire geological column (Mesozoic and Tertiary) has occurred along stacked, parallel, normal faults displaying displacements up to several thousand feet.

Tertiary faulting is significant to conducting exploration of the Pumpkin Hollow Property, as low angle normal faults have juxtaposed barren Tertiary and pre-Tertiary rocks above the known mineralized Deposits.

7.3 Mineralization

Granodiorite to diorite rocks belonging to the Jurassic Yerington Batholith intrude the limestones of the Triassic Mason Valley Formation and calcareous argillites and siliceous shales, siltstones and limestones of the Gardnerville Formation. Associated with this intrusive episode is the development of large areas of IOCG mineralization, which is dominantly skarn with associated copper and magnetite mineralization with varying levels of gold and silver. The skarn occurs primarily in the middle to lower portion of the Gardnerville Formation and the upper part of the Mason Valley Formation, as well as within the intrusive granitoid itself.

The geological environment reflects a classic copper skarn in one of its type localities (Basin and Range) where deposition is associated with I-type, magnetite series, calc-alkaline quartz diorite granodioritic stocks of hypabyssal character, proximity to stock contacts, assemblages of andradite garnet with diopside pyroxene, magnetite with hematite and moderate to high sulfide content consisting of pyrite, chalcopyrite, minor tennantite, and sphalerite. The large copper skarns are generally associated with altered and mineralized porphyry copper stocks and display extensive retrograde alterations.

Sulfides and iron oxides occur as disseminations, as massive streaks and in veins in skarn, and as massive replacements of marble at the skarn front. As skarns are commonly zoned, with massive garnetite near the pluton, increasing pyroxene and finally idocrase near the marble contact, sulfide mineralogy and metal ratios may also be systematically zoned relative to the pluton. Pyrite, chalcopyrite and magnetite are most abundant near the pluton with increasing chalcopyrite and finally bornite near the marble contact, with decreasing magnetite reflecting an outward decrease in total iron.

Examples of this group include some of the world's largest skarn deposits, such as the porphyry copper–related skarns of southwestern North America, which commonly contain 50 to 500 Mst of open pit copper ore in skarn and calc-silicate hornfels.

Initial contact metamorphism altered Gardnerville sediments to diopside-garnet, calc-hornfels and siliceous hornfels, and converted the limestone of the Mason Valley Formation to calcite ± dolomite ± tremolite marble. The intrusive rock is commonly altered to diopside-plagioclase endoskarn. Later metasomatism formed diopside-garnet-magnetite ± sulfide replacement zones.

Hydrothermal fluid movement and alteration were enhanced by fracturing and brecciation along and near diorite contacts, within the diorite, within beds of limestone and along fractures at a high angle to bedding in hornfels. Higher copper values are generally on the fringe of magnetite zones near a mineralized limestone contact (marble front) or within bodies associated with late stage retrograde actinolite-epidote-garnet-magnetite skarn.

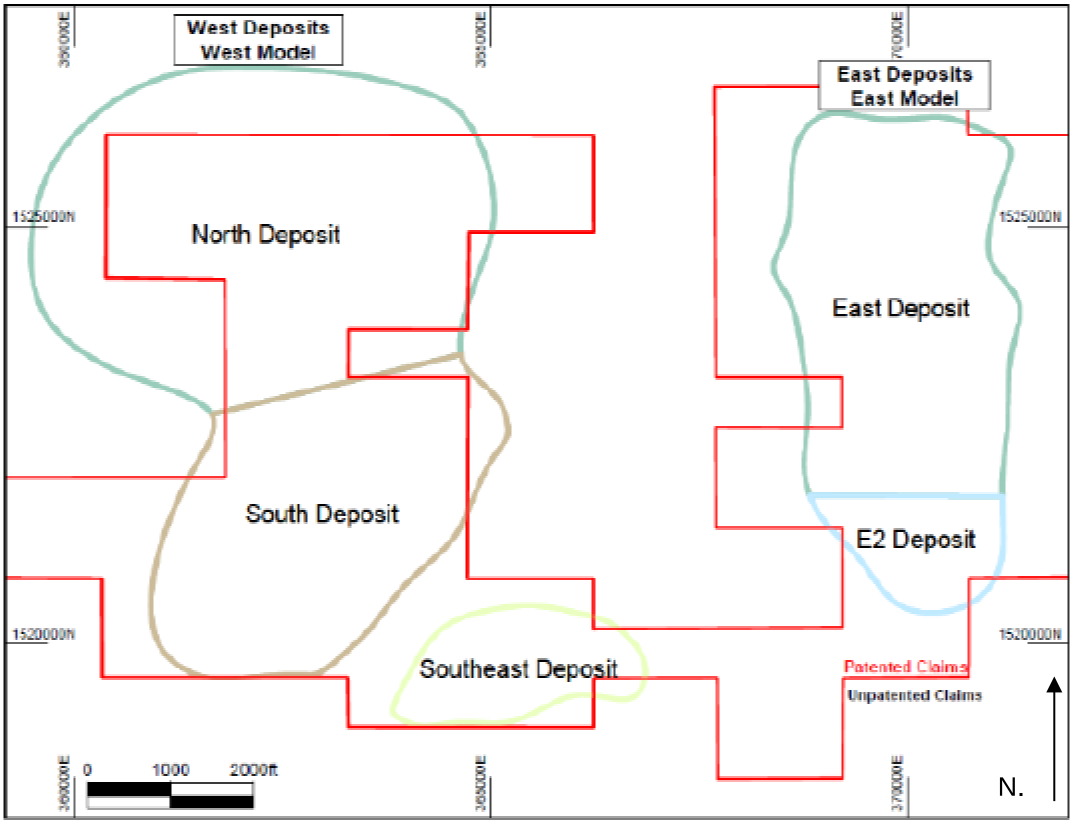
Retrograde skarn development was accompanied by deposition of pyrite, pyrrhotite and chalcopyrite, and may also be synchronous with the intrusion of altered and weakly mineralized andesite and quartz-monzonite porphyry dikes. There is a general overall zonation to the mineralization, with magnetite decreasing and copper increasing in relative content away from the intrusive. These two end member zones are commonly referred to as proximal and distal zones.

All deposits are believed to be directly associated with the intrusive granodioritic mass. The deposits are shown in Figure 7-3 and are known as the North, South, Southeast, East and E2 Deposits. There is a strong possibility that all Deposits were originally part of a single unit peripheral to and within the intrusive but have since been displaced and separated to their present locales by a complex post-mineralization tectonic history.

In general, each of the Deposits displays an early pro-grade mineralogy consisting of diopside-garnet-calcite-dolomite-tremolite and plagioclase, which represents the initial metamorphism followed by diopside-garnet-magnetite ± sulfide replacement along fractures. Retrograde alteration (actinolite-epidote-garnet-magnetite-calcite-dolomite) was accompanied by deposition of pyrite, pyrrhotite, and chalcopyrite.

All deposits are "blind" and are known strictly from drilling results. The uppermost oxidized portions of the North and South deposits lie within 100 ft of the surface but the bulk of the mineralization lies 300 to 400 ft below barren hanging wall rock. The top of the Southeast Deposit lies beneath 300 to 500 ft of barren granodiorite porphyry. The East Deposit is located beneath 1,500 ft of Tertiary Volcanics (TV), Quaternary gravels and barren hornfels units while the E2 Deposit is covered by 1,000 ft of TV and Quaternary gravels. The discoveries were made by drill evaluation of magnetic anomalies associated with large magnetite-rich bodies. The areal extent of the Deposits is shown in Figure 7-3. The South and Southeast Deposits represent areas of large magnetite deposition within the Mason Valley FM associated copper mineralization of lower tenor, whereas the Northwest, East, and E2 Deposits, although less explored, contain localized zones of much higher grade copper mineralization. The North Deposit is a low to moderate grade copper deposit with weak magnetite content and is primarily within hornfels and silicate skarns of the Gardnerville Formation. Figure 7-1 above shows the relationship of the Deposits within the local stratigraphy.

Figure 7-3: Deposit Locations



Source: Nevada Copper, 2019.

Item 8.0 DEPOSIT TYPES

8.1 Underground (Eastern Area)

8.1.1 East Deposit

The East Deposit, located 7,000 ft east of the North Deposit, measures approximately 2,000 ft by 1,200 ft and consists of flat-lying to gently dipping, bedding-controlled, stacked skarn-IOCG mineralized zones within the limestone of the Mason Valley Formation at depths of 1,400 to 2,200 ft. The East Deposit is defined by drill holes spaced approximately 175 to 250 ft apart. Figure 8-1 shows the geologic model in a typical cross section and Figure 8-2 shows the mineralized interpretation.

Higher grade copper occurs mainly in dolomitized limestones and skarn-chalcopyrite-magnetite-pyrite breccias immediately adjacent to diorite or diorite endoskarn. Much of the high magnetite rock that grounds the East Deposit is within diorite endoskarn. A good amount of continuity is evident in the East Deposit with mineralization possibly continuing and thickening to the west. The area between the East Deposit and the North Deposit still requires detailed exploration to accurately determine the lateral extent of each Deposit.

The top of the East Deposit appears to have been displaced to the east by the flat faults that separate the gently dipping limestones of the Mason Valley Formation from the overlying steeply dipping Gardnerville Formation.

The Mason Valley limestone member is locally overlain along a low-angle normal fault by the Gardnerville Formation, composed of barren hornfels and skarn with numerous diorite sills. These formations are, in turn, overlain along another low-angle normal fault by Tertiary volcanic and sedimentary rocks.

8.1.2 E2 Deposit

The E2 Deposit is a steeply northwest-dipping lens of high grade copper-magnetite skarn-IOCG breccia within the Mason Valley limestone, which lies on the hanging wall of an endoskarn sill. The lens has been explored along approximately 1,200 ft of strike length, is 40 to 120 ft thick, and is locally continuous for at least 1,600 ft down-dip. Figure 8-3 shows the geologic model in a typical cross section and Figure 8-4 shows the mineralized interpretation.

The chalcopyrite-magnetite mineralization follows the marble front, similar to the East Deposit. A major east-trending rotational fault appears to exist between the two deposits and results in a significant variation in the deposit orientation. Analysis of the drill hole intersections suggests that the E2 Deposit may, in fact, be a series of steeply dipping, plunging shoots, as much as 400 ft wide and 3,000 ft long on dip. The main portion of the mineralization starts approximately 800 ft below the present surface and extends 2,400 ft below the surface.

8.2 Open Pit (Western Area)

8.2.1 North Deposit

The North Deposit of mineralization is located 1,500 ft north of the South Deposit and is centered on a sub-horizontal, pipe-like, copper-rich, magnetite-poor skarn and skarn breccia body hosted by hornfels of the Gardnerville Formation (Northwest Deposit). Figure 8-5 shows the geologic model in a typical cross section and Figure 8-6 shows the mineralized interpretation. The higher grade mineralization in the skarn breccia grades into a zone of lower copper grade mineralization hosted by granodiorite endoskarn and marble (North Deposit).

The combined Deposit has a 3,500 ft strike length, with true widths of 200 to 900 ft, and a down-dip extent of 1,500 ft and remains open in several directions. Retrograde alteration and sulfide deposition in the North Deposit was focused along fractures at angles to bedding and overprints earlier prograde alteration. Retrograde alteration and mineralization along fractures diminishes away from the breccia core. Chalcopyrite tends to be the last sulfide phase deposited.

The top portion of the deposit is truncated by a zone of south-dipping listric faults. The faulting has emplaced post-mineral Tertiary volcanic and sedimentary rocks over the deposit.

8.2.2 South Deposit

The South Deposit, the first discovery on the Property, is a skarn-IOCG magnetite-chalcopyrite body closely associated with an intrusive contact of granodiorite into limestones of the Mason Valley Formation. The deposit is roughly tabular, measuring 3,600 ft along strike, 1,800 ft down-dip, trends northeast, and dips steeply to the northwest. The mineralized section, as defined by USS, can be divided into two zones: a footwall or proximal zone consisting of a 400 to over 1,000 ft thick relatively high grade magnetite with low copper content and a hanging wall or distal zone consisting of a 100 to 800 ft thick low grade iron but with higher copper content. Figure 8-7 shows the geologic model in a typical cross section and Figure 8-8 shows the mineralized interpretation.

The top of the South Deposit varies from 35 ft from the surface on the southwest side to 450 ft on the northeast side. The current surface is defined by a low-angle normal fault that places unaltered Tertiary volcanic rocks and conglomerate over the South Deposit.

8.2.3 Southeast Deposit

The Southeast Deposit, located 2,000 ft southeast of the South Deposit, is a 300 ft wide skarn-IOCG lens of chalcopyrite-magnetite-garnet-actinolite developed within limestones of the Mason Valley Formation. The zone is unique for the Pumpkin Hollow Property due to its elevated magnetite grades (locally up to 75%). The zone of mineralization strikes northeast and dips steeply northwest with a strike length of 1,500 ft. Magnetite-garnet-epidote skarn hosts the mineralization with chalcopyrite increasing toward the footwall marble front. The Southeast Deposit has a 600 ft vertical extent and is fault bounded, above and below, by low-angle normal faults.

Geologic similarities between the Southeast and South Deposits strongly suggest that the Southeast Deposit may be the upper portion of the South Deposit which has been displaced 2,000 ft to the southeast along the lower listric fault. Following the 2008 drilling program, sufficient geologic and mineralization information was developed to generate both a geologic and resource model. Figure 8-9 shows the geologic model in a typical cross section and Figure 8-10 shows the mineralized interpretation.

Cross section locations are shown in Figure 8-11.

Figure 8-1: East Deposit – Cross Section 53,200N (Rock Types) (2017 Technical Report)

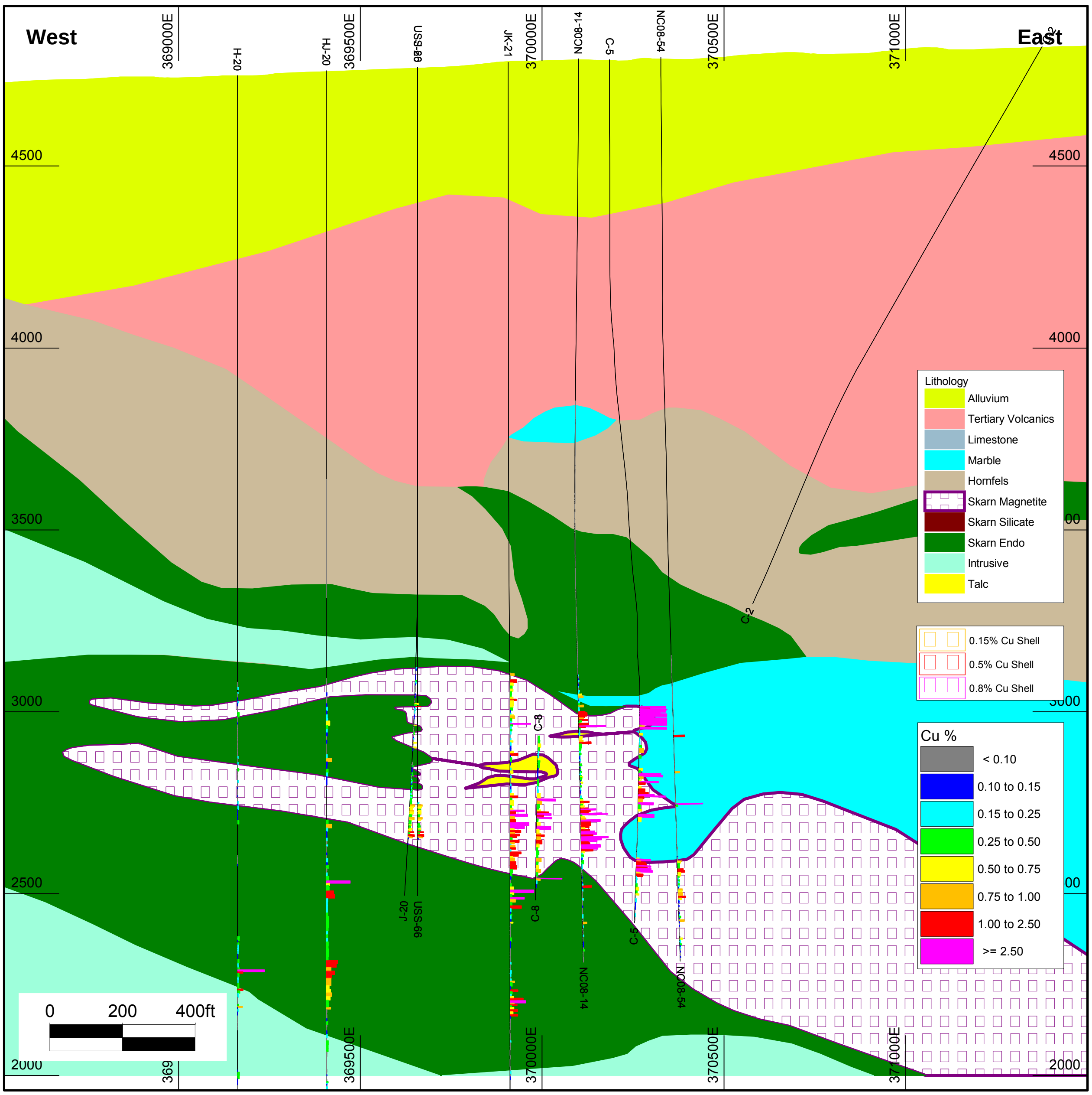


Figure 8-2: East Deposit – Cross Section 53,200N (Mineral Zones) (2017 Technical Report)

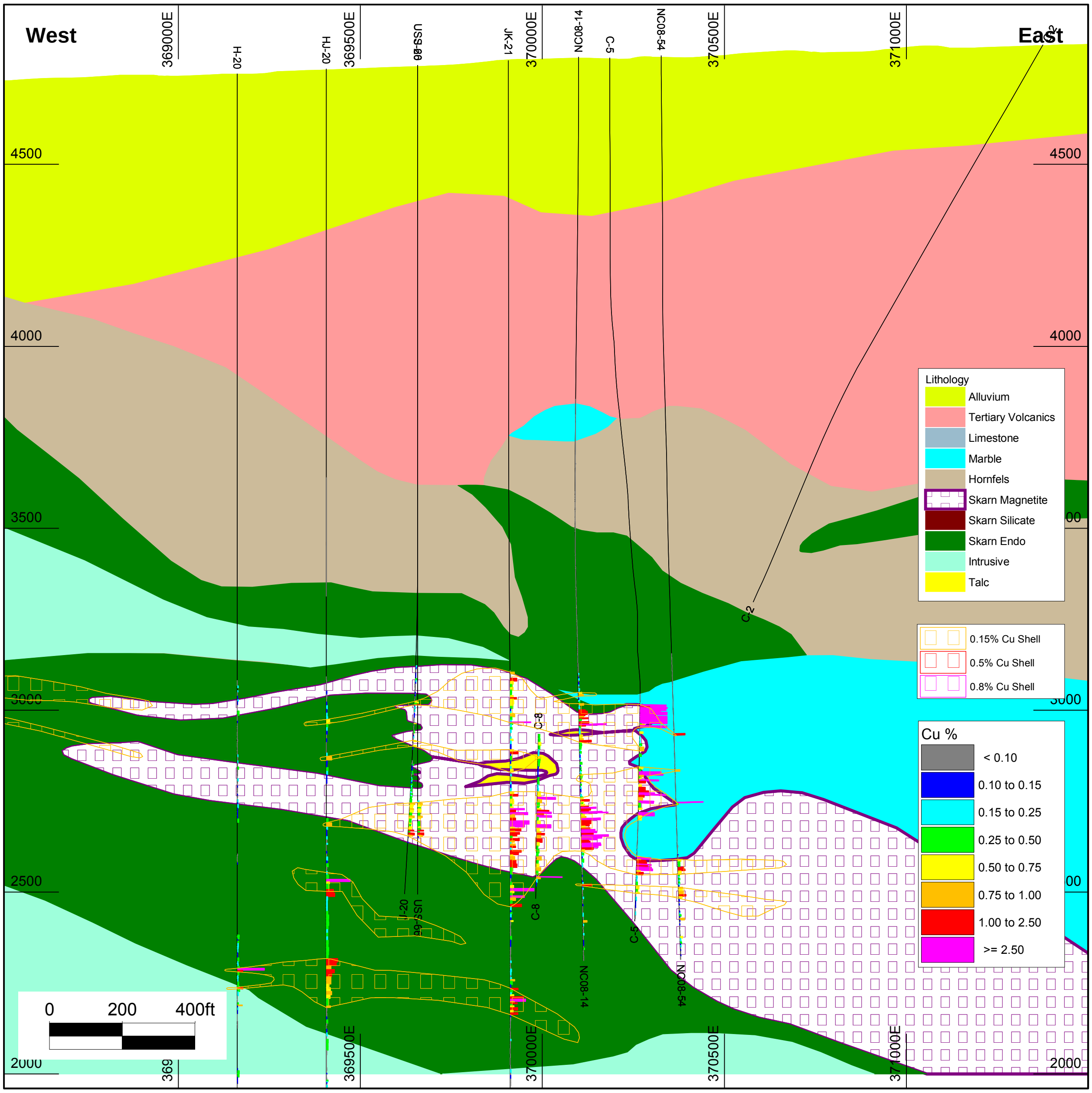


Figure 8-3: E2 Deposit – Cross Section 17 (Rock Types) (2017 Technical Report)

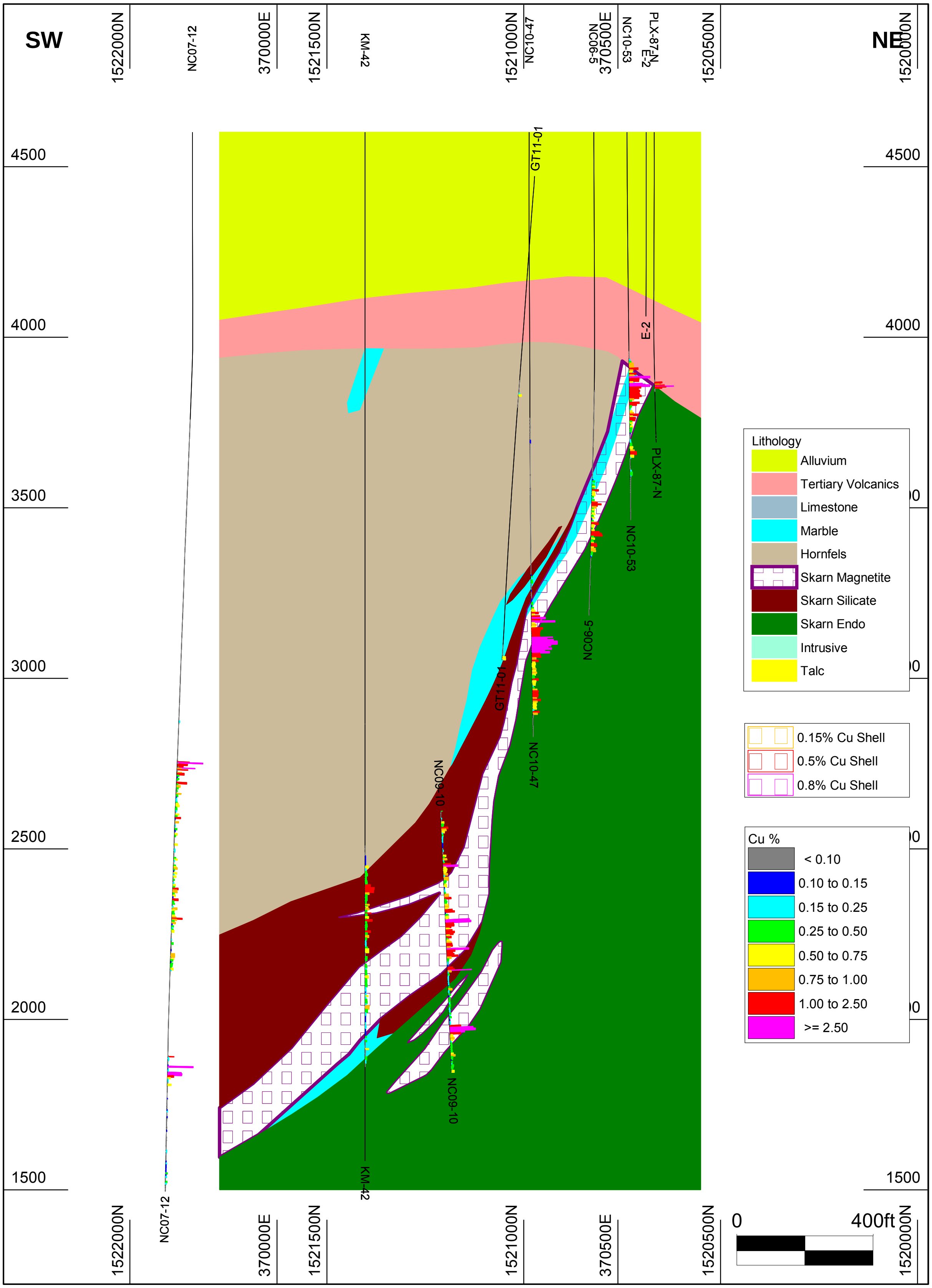


Figure 8-4: E2 Deposit - Cross Section 17 (Mineral Zones) (2017 Technical Report)

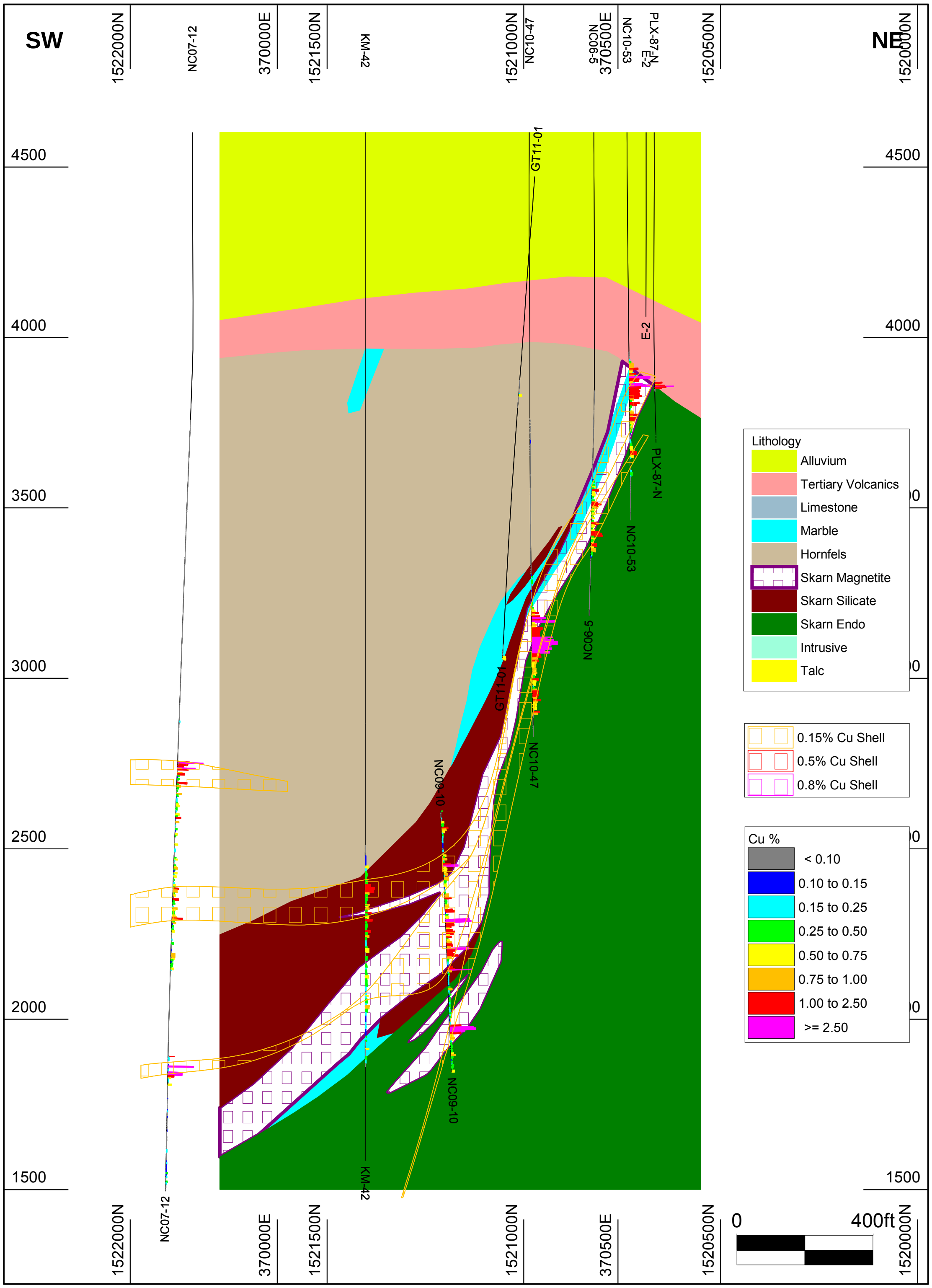


Figure 8-5: North Deposit – Cross Section 361,220E (Rock Types) (2017, Technical Report)

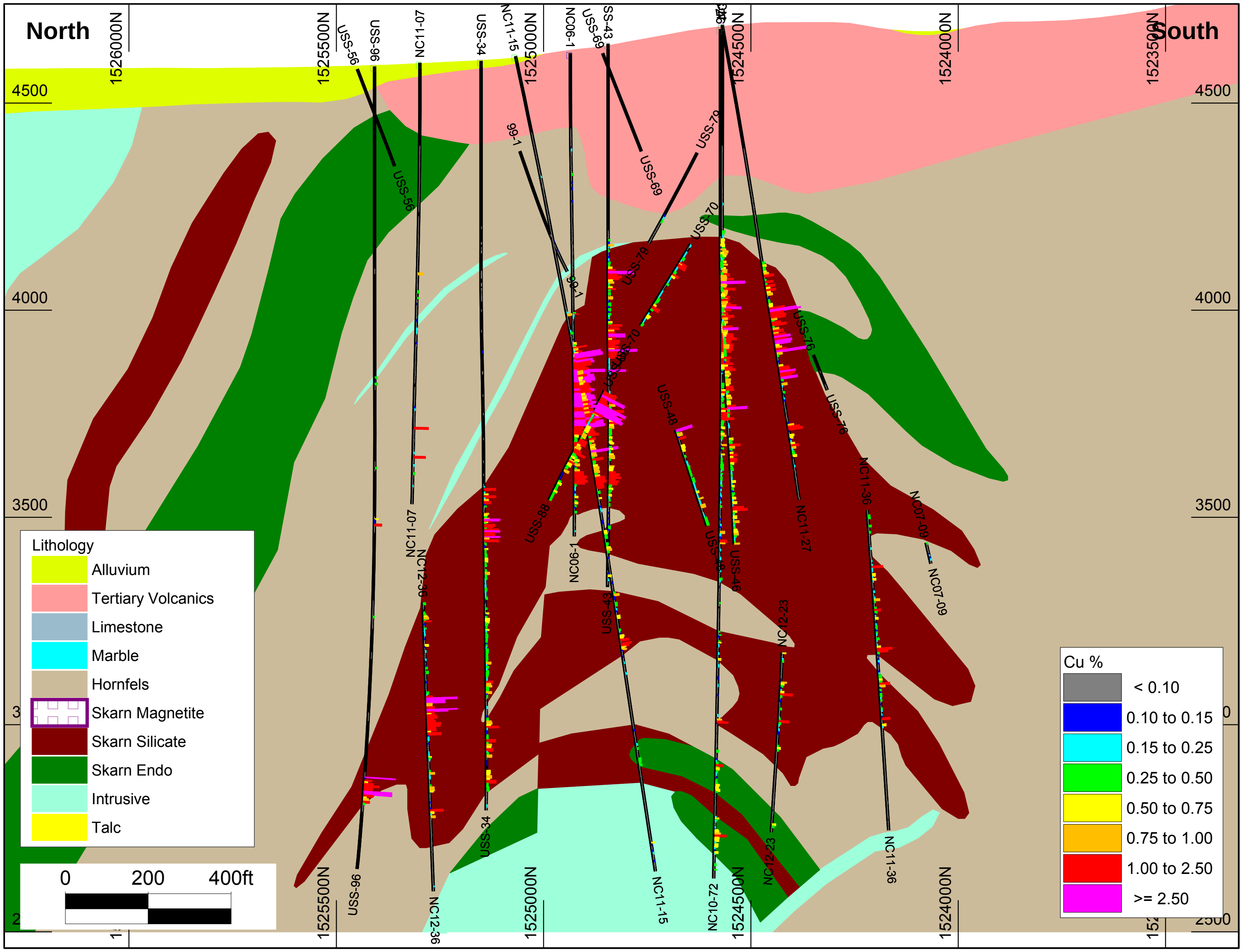


Figure 8-6: North Deposit – Cross Section 361,220E (Mineral Zones) (2017 Technical Report)

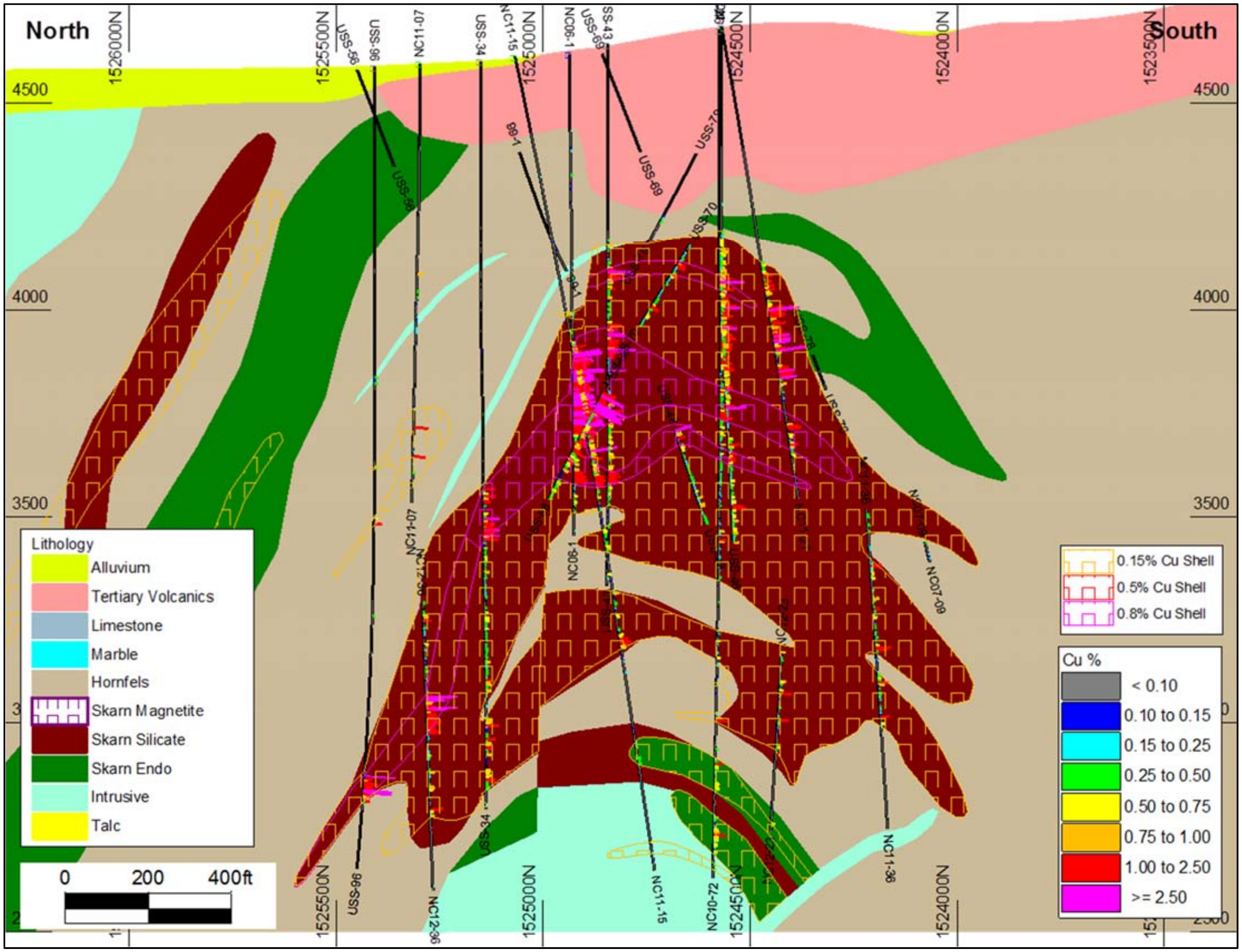


Figure 8-7: South Deposit – Cross Section 1,850 E NE (Rock Types) (2017 Technical Report)

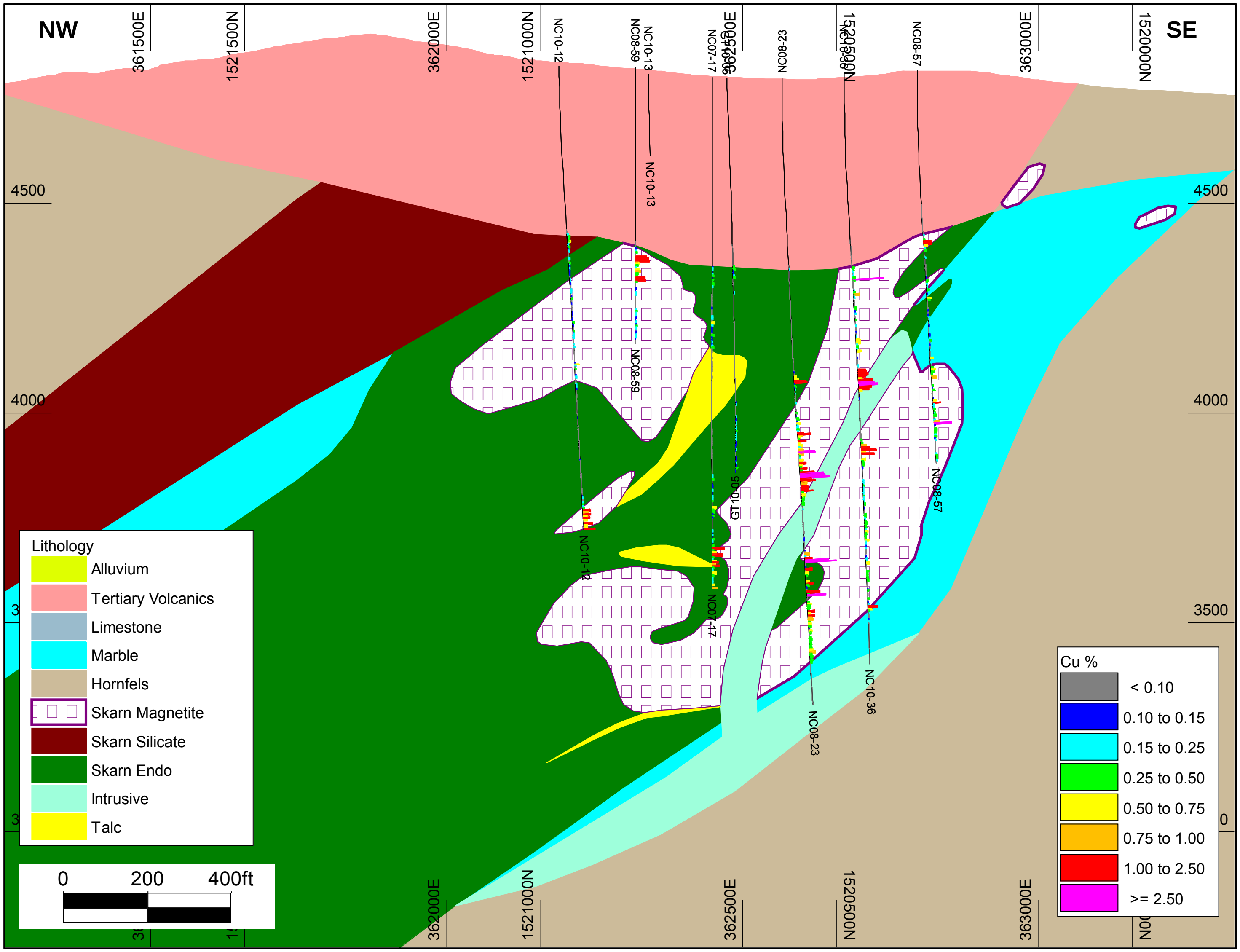


Figure 8-8: South Deposit – Cross Section 1,850 E NE (Mineral Zones) (2017 Technical Report)

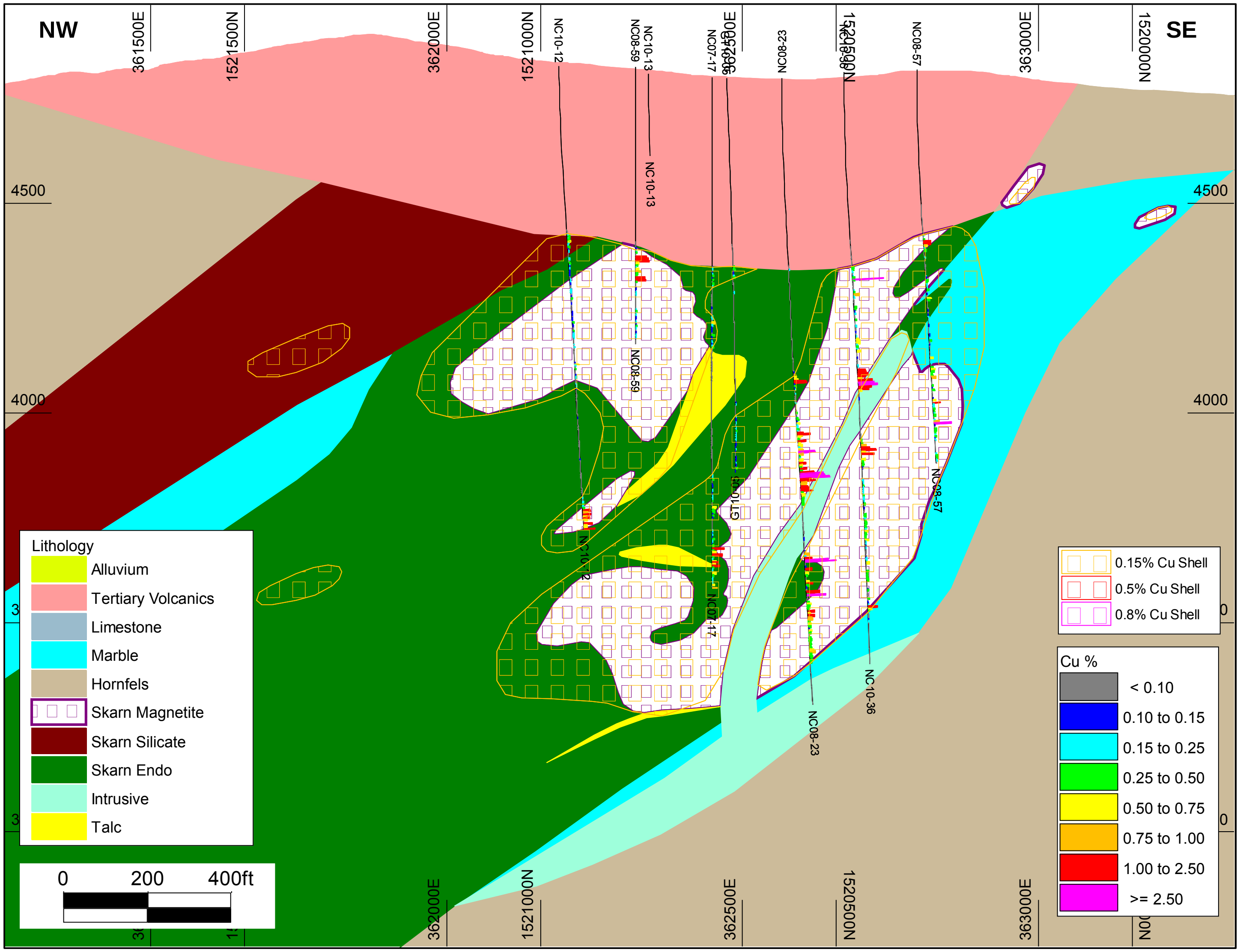


Figure 8-9: Southeast Deposit – Cross Section 1200 E NE (Rock Types) (2017 Technical Report)

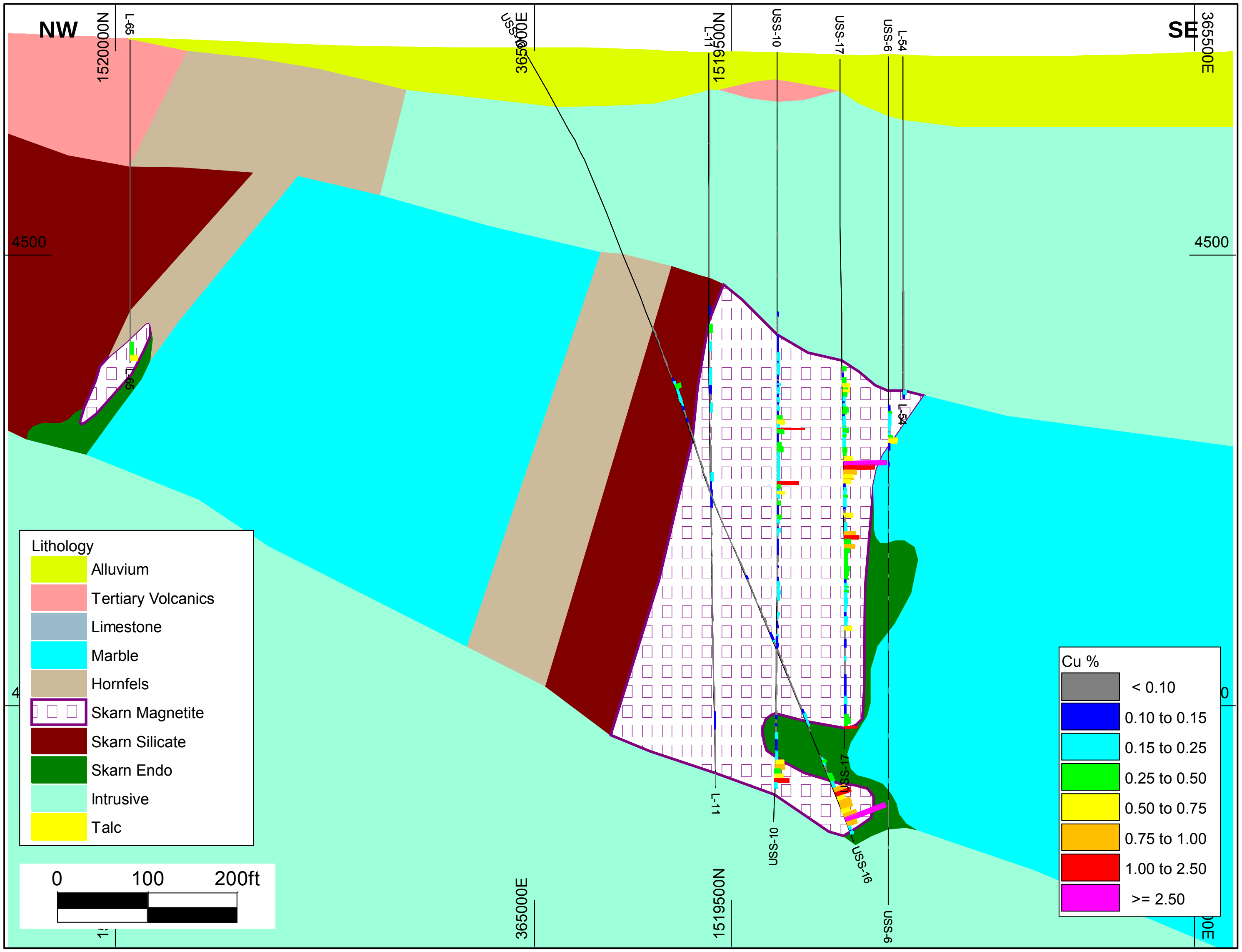


Figure 8-10: Southeast Deposit - Cross Section 1200 E NE (Mineral Zones) (2017 Technical Report)

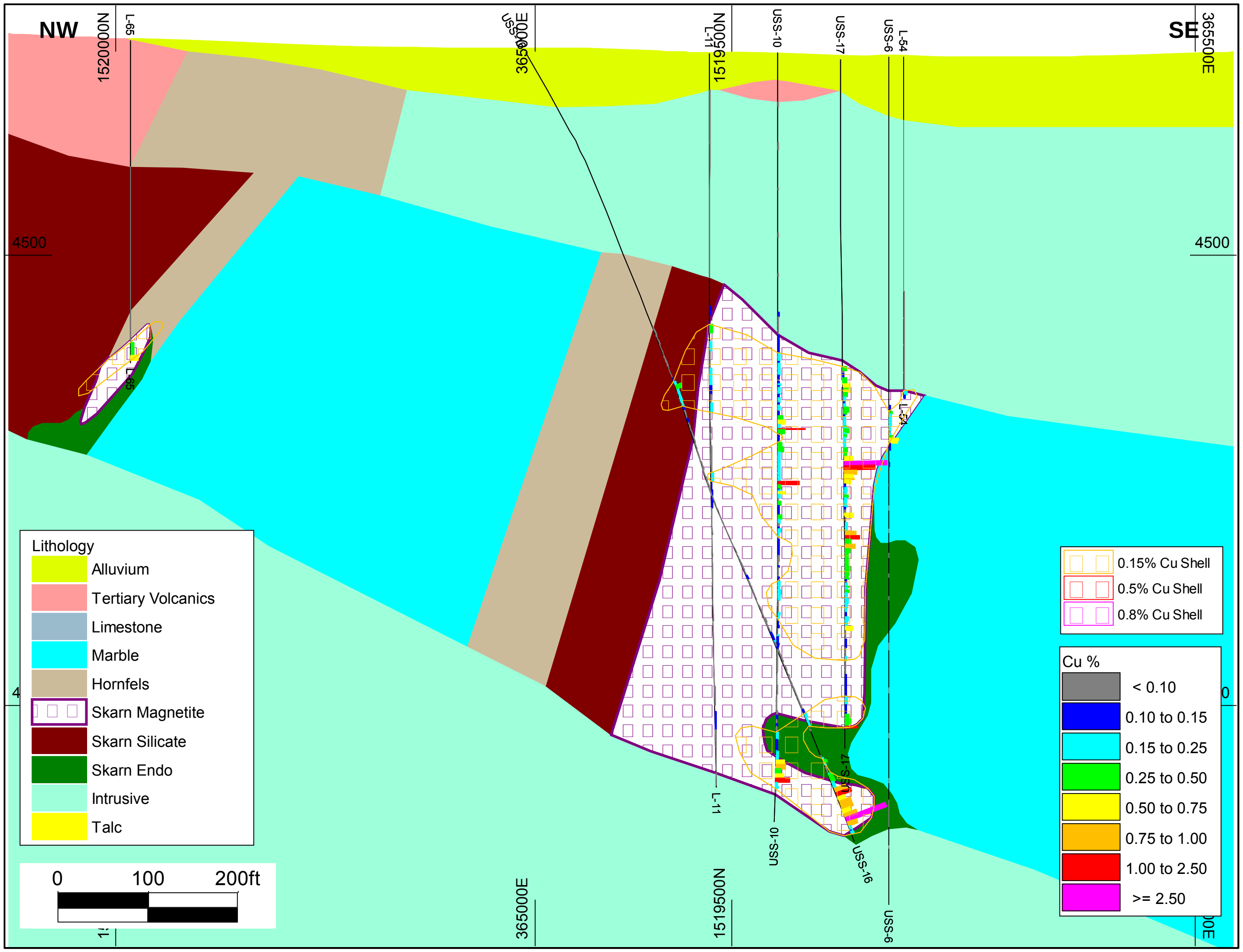
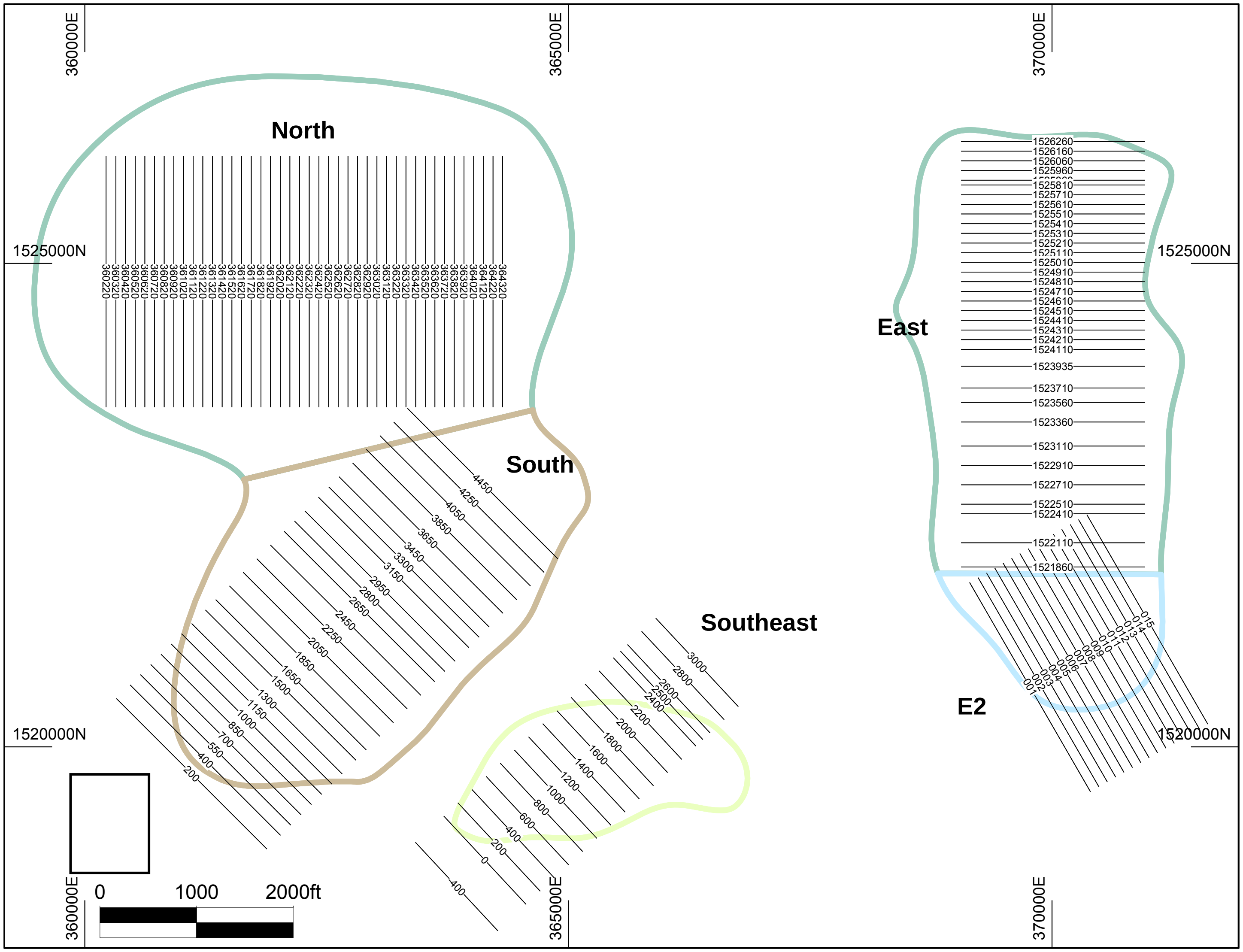


Figure 8-11: Geologic Cross Section Index Map (2017 Technical Report)



Item 9.0 EXPLORATION

The Property has been explored by major mining companies since the discovery of iron and copper mineralization by US Steel in the 1960s. Much of the exploration of the property was done by previous operators and summaries are located in Item 6.0. Nevada Copper's exploration began in 2006 when it acquired the Property. Exploration by Nevada Copper has focused on resource expansion through drilling and limited additional surface mapping or non-drilling sampling has been conducted. The following describes the geological, sampling and geophysical exploration completed on the Pumpkin Hollow Property by Nevada Copper:

- Exploration focused on review of past geological interpretation and modeling. Geology was standardized and has been used as a guide for modeling since 2006.
- Drill spacing is 150 ft or less in resource areas.
- There are a total of 54,615 Cu samples in the exploration database.
- The extent of drilling is shown in Figure 10-1 in Item 10.0.
- Nevada Copper completed a detailed ground magnetic survey that was performed on the Eastern Area of the Property in 2009.
- Because the orebodies do not outcrop, selected samples from core were taken for alteration and mineralogical information. The drill samples were representative and taken at nominal 5-foot intervals. Detailed description of the sampling is found in Item 11.0 and Item 12.0.
- Surface mapping and sampling on the Property was very limited. Sampling consisted of representative and grab rock chips.
- There are no known biases in the exploration data.
- The exploration data is sufficient and appropriate for the development of geologic resource models.

The future exploration programs will focus on expanding mineralization in and around the known deposits as well as other targets within the claim block. The areas targeted for future exploration include:

- Exploration around the existing deposits consists of drilling and refining the geological and geochemical model.
- Geophysical targets need further evaluation.
- The Eastern exploration targets are located approximately one mile east of the East and E2 deposits. Detailed ground magnetic surveys have been completed and surface mapping and sampling are ongoing. Skarn altered limestone with visible oxide copper mineralization crop out and are coincident with weak magnetic and IP anomalies.

Other targets within the Property could result in the discovery of additional resources and reserves; however, these targets will require additional geologic mapping, data compilation and review.

Item 10.0 DRILLING

Based on review of the drilling data reviewed and observations during the site visit, it is Tetra Tech's opinion that Nevada Copper and the previous operators' drilling data collection methods meets recognized and accepted industry best practices and the drilling data is suitable for the development of the geologic resource model. The drilling and drill sampling has been carried out by reputable companies using recognized practices and procedures. There are no known inconsistencies or inaccuracies in drilling collection methods, recovery factors or in the exploration database that would materially affect the geologic resource model.

From 1960 to 2018, previous operators and Nevada Copper drilled over 800 drill holes for approximately 1.2 million ft of drilling on the Pumpkin Hollow Property. Since acquiring the Property in 2006, Nevada Copper has performed approximately 578,000 ft of drilling, representing approximately 49% of the total drilling on the Property to date.

Table 10-1 shows a summary of historical drilling at the Pumpkin Hollow Property.

Table 10-1: Brief History of Drilling at the Pumpkin Hollow Property

Company	Period	No. of Drill Holes	Total Ft Drilled
United States Steel Corp.	1960–1974	282	392,135
Anaconda Corp.	1975–1977	96	143,905
Conoco Inc.	1981–1982	13	27,106
Plexus Resources	1985–1987	2	3,006
Cyprus Metals Exploration	1989–1993	23	20,986
International Taurus Resources	1998–1999	8	7,513
Nevada Copper	2006–2007	39	64,904
Nevada Copper	2008	66	84,640
Nevada Copper	2009	12	21,236
Nevada Copper	2010	77	134,583
Nevada Copper	2011	54	100,183
Nevada Copper	2012	37	56,989
Nevada Copper	2013	12	22,097
Nevada Copper	2015	42	54,571
Nevada Copper	2018	29	39,106
Outside of Resource Models	1960–Present	34	51,292
Total	1960–Present	826	**1,224,253**

The drill holes of previous operators and Nevada Copper at the Pumpkin Hollow Property are usually pre-collared through un-mineralized rock with rotary reverse circulation drill rigs. They were followed up with NC to HQ core tails in the mineralized zone. Occasionally, shallow mineralization (less than 500 ft) drilling is completed using only a reverse circulation rig. This makes up less than 1% of the mineralized sample intervals. Due to the competency of the rock, core recoveries were usually greater than 95%. The drill holes were surveyed using a gyro as the magnetite content of the of the rock types will have a negative effect on standard camera surveys.

The pre-collar drill holes were drilled to a pre-determined depth and casing was set. The samples were marked by geologists at 10 foot intervals. Sampling was completed under the supervision of geologists. Samples were split using standard wet splitter. Geological logging samples were also collected in standard plastic chip trays or chip boards. Samples were picked up by company personnel and delivered to the secure sample facility on the property. Core boxes were transported by company personnel to the secure sample facility (core sheds) on the Pumpkin Hollow Property for geotechnical and geological logging and sampling.

Since the preparation of the April 2015 resource statement for the Underground Mineral Resource, drilling was completed in 2015 and 2018. Most of the drilling (57 holes) focused on the Western Area. In addition to confirming mineral continuity, grade and the geometry, new mineralization was intersected.

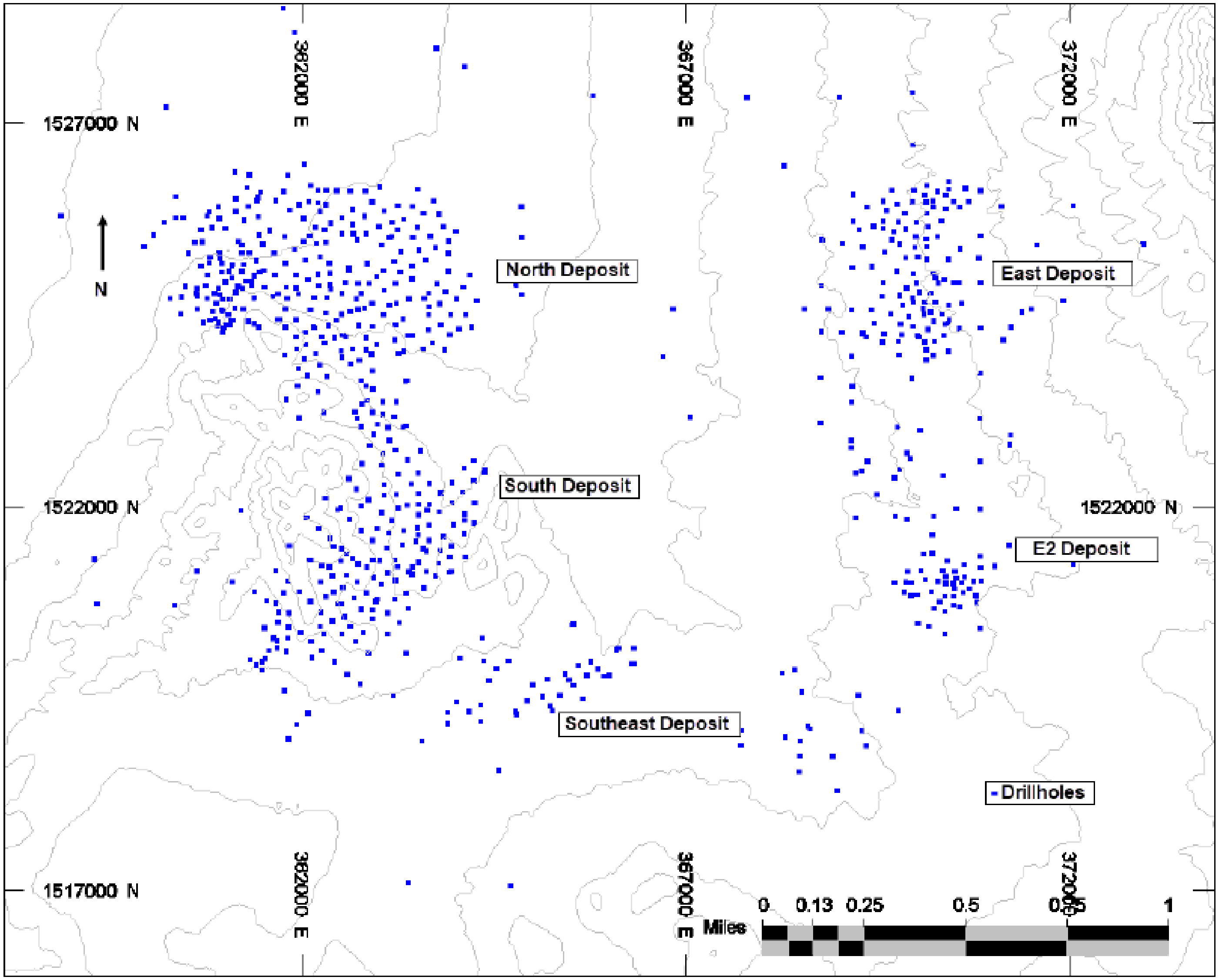
Within the Eastern Area, a total of 9,728 ft of drilling was completed with 10 underground drill holes and 1 geotechnical hole within the East and E2 deposits. The limited amount of 2015 drilling had no material effect on the existing mineral resource model’s geometry and grades. These holes are not included in the current Mineral Resource estimate, leaving the statement of April 15, 2015, unchanged.

The open areas in the Western Area would benefit from additional drilling. Figure 10-1 is a plan map of the holes drill on the Property.

The Mineral Resource estimate for the Western Area included in this Report is based on the most recent through July 2018.

Representative drill cross sections for the individual Deposits can be found in Item 8.0.

Figure 10-1: Drill Hole Location Map (Golder, 2019)



Item 11.0 SAMPLE PREPARATION, ANALYSES & SECURITY

Based on review of the data and a review of the previously performed analyses and observations during the site visit, it is Golder's opinion that Nevada Copper and the previous operators' sample preparation, analysis, and security protocols are appropriate and meet widely accepted industry standards, methods, and definitions for defining Mineral Resources.

11.1 Sample Preparation

Core samples were marked by Nevada Copper geologists prior to delivery to the analytical laboratory. The sawed core splits were placed into sample bags for drying and processing. For the previous drilling the core samples were marked and split on site. The bagged core and reverse circulation samples were delivered to the analytical lab. The core intervals sampled by Nevada Copper varied from one to six ft runs and one to ten foot runs for previous operators were based on geology. The reverse circulation and rotary samples were either five or ten foot intervals. The following is a summary of the sample preparation procedures:

- For Nevada Copper, the samples were sent to American Assay Laboratories (AAL) Samples weighing 10 to 15 lb are dried in high-air volume, temperature- controlled ±5°, gas-fired dying ovens. Bagged and tray samples are dried at 105°C. The dried samples are then jaw crushed to <6 mesh and weighed, then roll crushed so that >80% is less than 10 mesh. Samples are Jones riffle split and a two-lb. sample split is pulverized in a ring mill to >90% at <150 mesh. The sample is then placed in a labeled pulp packet.
- The previous operators were major mining companies and used both independent and internal laboratories. Review of the historical documentation and verification check assays indicated that they used industry standard sample preparation procedures during that time.

11.2 Sample Analysis

All assaying and whole rock geochemistry done by Nevada Copper is processed at AAL in Sparks, Nevada. AAL is ISO/IEC 17025 certified for the methods used in assaying samples and has successfully completed Canadian Certified Reference Materials Project proficiency testing. Samples are delivered from the core logging facility to AAL by AAL personnel. A QA/QC assay protocol has been implemented by Nevada Copper whereby blanks and standards are inserted into the sampling stream for every 20 to 30 samples.

The AAL sample procedures are as follows:

- Fire Assay: A 30 gram sample is weighed and mixed with ~130 grams of flux. The sample is fused/coupled and parted. The solution is then read on an ICP-AES. Repeat analysis is performed on sample results >0.2 ppm, and gravimetric analysis is performed on samples with results >10 ppm.

- Geochemical: A 0.5 gram sample is weighed into beakers. A three-acid mix is added to the sample and digested. The sample is normalized to volume and analyzed by ICP-AES.
- Duplicates are sent in and analyzed in the identical manner, as described above.
- Check assay pulps, core and rejects (~ every 30, higher frequency in mineralized zones) are submitted to BSI-Inspectorate, Sparks, Nevada, and/or Chemex Labs, Sparks, Nevada, for analysis. Duplicate core samples were also sent to these labs. Both labs are ISO 9002 certified.
- Ore Grade Fe (Magnetite): A two gram sample is weighed into tubes. Hydrochloric acid is added to the sample and digested. The sample is normalized to volume and analyzed by ICP-AES.

The blanks and standards were obtained from independent labs. The blanks are composed of barren quartz sand purchased from Shea Clarke Smith of Nevada. The field standards are prepared from material sourced on the Pumpkin Hollow Property because of difficulty obtaining a suitable standard with high copper and iron content. The standards are prepared at Inspectorate America Labs in Reno Nevada using ore sourced from the Pumpkin Hollow Property. Material is dried, crushed to -10 mesh and then ground to -200 mesh. The entire sample is blended in a “V” blender for 24 hours. Ten 100 gram aliquots are sent for assay at Inspectorate. Once sample pass assay testing, 60 to 100 gram splits are created and put into marked pulp envelopes for use as standards.

A subset of samples (about 4.5%) were sent for re-assay at AAL while a subset of these (about 2% to 5%) were periodically sent to a second laboratory, Inspectorate was used for the 2006 to 2012 campaigns. Golder reviewed the check assays from the 2006 to 2012 drilling campaign using the Half Absolute Relative Difference (HARD) method. HARD is a parameter used to determine the precision of a population. It is produced by dividing half the absolute difference between two values by the means of the two values. In general, if 90% of the population had a less than 10% HARD, it is considered acceptable. For copper, the checked assays show an acceptable precision, where 90% of the samples had a HARD below 15% and no bias. For gold only 60 pairs were analyzed and 90% of the samples had a HARD value below 50%. The silver checks assays had 90% of the samples with a HARD value below 40%.

Nevada Copper has used a suite of 30 standard/blanks samples as part of the QA program, covering a range of copper, gold, silver, and iron concentration from 2006 to 2013 drill campaigns. Most of the standards have duplicate gold assays run at the laboratory (that is, two different pulp samples). Tetra Tech developed an evaluation process for copper, gold and silver by comparing the mean laboratory results plus/minus two times the standard deviation. Ninety-five percent of the gold standards plotted well within two times the standard deviation, while 97% of the silver, copper and iron fell within that range. Golder concurs with the findings and methodology developed by Tetra Tech and applied those to the 2018 Open Pit Mineral Resource estimation.

Golder reviewed the fine blanks inserted by Nevada Copper from 2006 to 2013. The copper blank samples show 22.4% result above the 20 ppm copper tolerance limit. The gold and silver blank samples also has samples above the tolerance limits, 59% for gold and 22% for silver. However, most of the samples above the tolerance limits are close to the detection limits whereby higher variability is observed.

In the 2018 campaign check assays were performed by Bureau Veritas and were reviewed by Golder using the HARD method. For copper, the check assays show a good precision, where 90% of the samples have a HARD value below 5%. For gold, the check assays exhibit low precision, where 90% of the samples have a HARD value below 40% with a mean HARD value of 22.2%. Gold check assays are of relatively low precision, however, only 31 pairs of samples were able to analyze as most of the samples are below the detection limits. Golder also reviewed any potential sample bias and found that silver and gold samples show no apparent bias, however the copper samples show a negative bias.

Golder reviewed the fine blanks through graphical analyses. The copper blank samples show only one result above the 20 ppm copper tolerance limit, and no samples outside the tolerance limits for silver and gold. In general, the 2018 campaign shows good results, with no evidence of contamination

The AAL primary laboratory inserted samples for their internal QA/QC, including standards, blanks and duplicates, which Golder reviewed. Analysis of the standards shows reasonable results with acceptable errors of precision and no obvious bias for the copper, silver and gold values. Graphical analyses of the blanks show only local values above the tolerance limit, but no evidence of contamination. The copper duplicates show good precision with 90% of the samples having a HARD value below 5%, the silver and gold duplicates show low precision with 90% of the samples having a HARD value below 35%.

11.3 Security

Drill core and reverse circulation samples are under the control of either Nevada Copper or AAL personnel once the samples are picked up from the drill rigs. Nevada Copper personnel collect the samples from the drill rig and deliver them to the secure core logging facility located at the Property. There is 24-hour supervision at the Property. Following geological logging, samples are picked up by AAL personnel and delivered to the secure AAL facility in Sparks, Nevada. Upon completion of the analytical work, samples are returned to the Property by AAL personnel and are placed in the core storage building located at the Property.

Item 12.0 DATA VERIFICATION

Golder reviewed Nevada Copper's data collection procedures and analytical QA/QC program for the 2018 drilling program. It is Golder's opinion that the procedures in place meet current industry standards and requirements and are adequate for the purpose of preparing the respective prefeasibility studies. Nevada Copper staff are well versed in both the importance of procedures and the protocols to follow in order to ensure that the data being collected meet industry standards. Additional data verification was carried on previous campaigns and historical work. The historical work on the property has been carried out by reputable companies and there does not appear to be any reason to question the validity of the information.

Golder reviewed the data collection and verification processes and conclusions disclosed in the 2017 Technical Report (2017 Technical Report). Golder found the processes and conclusions provided in the 2017 Technical Report met industry standards and are consistent with the requirements for use in preparing a Mineral Resource estimate. No limitations on the verification procedures applied by the QP were identified.

Other data, such as geotechnical and hydrological data, was collected and analysis was performed. This data was reviewed by the previous QP and found to meet industry standards and fit for the purposes of preparing a prefeasibility study. There have been no material changes in the application of this information and it is the current QP's opinion that this information remains valid and that no further work was required to verify this data.

12.1 Drill Core & Geologic Logs

Tetra Tech has reviewed the procedures for core handling and the geological logging. The core storage facilities were visited and found to be clean and well maintained. Individual drill holes were easily located and verified.

Sample Box intervals were marked with permanent marker and aluminum tags along the side of the core. The geologist markings with permanent marker and metal tags on the core were checked as well as the core received back from the lab. No inconsistences were found.

The geological logs are marked with lines that correspond to the beginning and ending sample intervals in the assay column. Tetra Tech has reviewed the geotechnical, geologic logs, previous geological logs and geological re-logs and is satisfied that the logs represent the geotechnical, geological and mineralogical conditions. Detailed photos of the core and geologic log can be found in the 2017 Technical Report.

12.2 Topography

It is Tetra Tech's opinion that the current topographic map is accurate and fairly represents the topography of the area. In addition, it is suitable for the development of the geologic models, resource estimates, and mine planning.

12.3 Analytical Verification

Nevada Copper is following a QA/QC program of inserted standards and blanks and periodic re-assay of core (duplicates) at their primary lab and also at secondary labs (check assays). The primary analytical laboratory, AAL, is ISO 17025 certified. First pass quality control uses international standards and blanks. AAL includes three standards and one blank per batch of 43 assays and requires recovery of 90 to 110% of the recorded value for sample results to be valid. It is required that the blank be less than twice the detection limit for low level results to be valid. A second pass quality control is a duplicate run of samples called controls (8 per 43 assays). The reproducibility of the controls is specific. The third pass quality control repeats any unusual results. This includes low results in high value areas or high results in low value areas.

12.3.1 Quality Control

Tetra Tech undertook a review of the QA/QC results for sampling done in 2018, which included evaluation of the results of field standards, blanks and duplicates. When Nevada Copper receives analytical results from the lab, the field standards, blanks and duplicate results are entered into a separate data set, that is, the results are not commingled in the main database and are not included in the resource estimate.

12.3.2 Field Standards, Blanks, and Duplicates

Tetra Tech reviewed the protocols for the insertion of standards, blanks and duplicates. Nevada Copper protocol calls for insertion of field standards and blanks randomly inserted for every 100th sample. The blanks are composed of barren quartz sand purchased from Shea Clarke Smith of Nevada and CDN Laboratories of Canada. The standards are prepared at Inspectorate America Labs using ore sourced from the Property. No inconsistences were found in the blanks, standards or duplicates.

12.4 Independent Analyses

The Property has a drill hole database for the Underground and Open Pit Projects composed of drill core, photographs of the drill core, assay certificates and results, and geologic logs. Preservation of the drill core and associated hardcopies of the data have been maintained by the originators of the property data and the subsequent companies that have looked at the Property. All data is readily available for inspection and verification. The Tetra Tech geologist completed “spot” checks of four core drill holes selected at random during a site visit. This was followed by a detailed review of the complete QA/QC data, including geologic logs, check assays and assay certificates. No significant discrepancies were found with the existing drill hole geologic logs, and Tetra Tech is satisfied that the geologic logging, as provided for the development of the 3D geologic models, fairly represents both the geologic and mineralogic conditions. The Golder geologist reviewed the process and is of the opinion that it meets industry standards.

“Normal” types of errors inherent in this size (i.e., mislabeled intervals, number transpositions, and so forth) were noted in the databases and associated data. No major issues were identified. Golder found no significant discrepancies with the existing drill hole geologic logs, and the QP is satisfied that the geologic logging, as provided for the development of the 3D geologic models, adequately represents both the geologic conditions.

After final drill hole results for the Nevada Copper drilling have been received from AAL, rejects are selected by geologists for periodic independent laboratory verification. Check assay rejects are submitted to Inspectorate, Sparks, Nevada, and/or Chemex Labs, Sparks, Nevada, for analysis. Both labs are ISO 9001:2008 certified, and Chemex is ISO 17205 certified as well. Check assays by Nevada Copper are performed periodically in tandem with the field duplicates. Approximately 5% of samples sent for duplicate sampling were sent to the second lab for check assay. Some samples were sent to Inspectorate (Sparks) and others were sent to Chemex (Reno). The following results were captured:

- For gold, the average relative percent difference between the duplicate and the primary value (originally assayed at AAL) was -6% for Inspectorate and 1% for Chemex labs. However, when plotted on a scatterplot, and both labs were consistent. When compared with the primary value, the overall correlation coefficient was greater than 0.9 for both labs. As is usual, the greatest amount of scatter is seen at values near the detection limit.
- For silver, the overall correlation coefficient was also greater than 0.9 for results from both laboratories plotted against the primary value. However, values from Chemex showed a slight bias toward higher values at higher primary values. The bias is small, with an average relative percent difference of 6%. It is also noted that AAL prepared the samples at that time with a two-acid digestion, where the check labs both used a three-acid preparation, which may impact results.
- Copper showed excellent correlation with correlation coefficients at >0.99 for both laboratories with both having an average 6% higher value than the primary.

12.5 Survey Data

Field checking a selected number of drill holes was completed. The drill hole markers and GPS coordinates were documented and the downhole survey data were also reviewed. No discrepancies were found.

12.6 Data Verification

Tetra Tech conducted a review of the 2010 Resource and observed data entry errors of the ppm values in the oz/st data spaces for gold and silver values of four drill holes. It was observed that oz/st values were truncated rather than properly rounded in the database.

Following the review, the oz/st values were verified/corrected. This achieved a reduction in gold ounces for the Eastern Area Deposits, but did not affect the copper grade and resource tonnage. Additionally, the truncated oz/st values were replaced by the original values prior to the re-evaluation of the current resource update. This resulted in a marginal increase in reported gold ounces for the Western Area Deposits, which did not affect the copper grade and resource tonnage.

Review of the assay data for errors should continue as any errors could have an influence on model grade. Lithological coding should also be reviewed for potential errors as errors will have an effect on model density and metallurgical characteristics.

It is Tetra Tech's opinion that the data verification for the Pumpkin Hollow Project, as described in Item 12.1 through Item 12.6, meets industry standards and is consistent with requirements and best practices for use in preparing a Mineral Resource estimate.

Item 13.0 MINERAL PROCESSING & METALLURGICAL TESTING

This item presents a summary of mineral processing and metallurgical test work associated with both the stand-alone Underground Project and the stand-alone Open Pit Project.

The stand-alone Underground Project is based on a mill feed of 5,000 stpd throughput. Mill feed material for the Underground Project will be sourced from the Eastern Area Deposits in the Underground Pine.

The stand-alone Open Pit Project is based on a mill feed of 37,000 stpd with expansion to 70,000 stpd throughput. Mill feed material will be sourced from the North and South Pits in the Western Area.

Previous process engineering and metallurgical test work undertaken by Tetra Tech has been reviewed and updated by Sedgman.

13.1 Test Work Program History

USS conducted the drilling and directed the test work surrounding the deposit as early as 1967. The bulk of this test work was focused on recovery of magnetite addressing issues regarding the copper content of the magnetite products. In 1996, Midland Research Center performed additional magnetite testing involving the integration of flotation. In 2007, Hazen Research of Denver, CO. conducted scoping-level test work examining the responses of both open pit and underground material to comminution. By-product testing again produced unacceptable levels of copper and sulfur. In 2008, additional magnetite testing was completed by University of Minnesota. None of the magnetite test work above was included in this study. In 2010, G&T Metallurgical Services of Kamloops, BC performed a significant body of test work on four types of material: North and South (both considered part of the Western Area), and East and E2 (both considered part of the Eastern Area). This test work included single pass rougher and cleaner flotation, the results of which demonstrated that all four deposits are amenable to flotation processing. The relevant Eastern Area results are used in this Report.

Hazen performed additional single pass flotation testing in 2011 on open pit and underground material. Concurrent with this, Pocock Industrial, Inc. (Pocock) performed considerable test work regarding the solid-liquid separation characteristics of open pit and underground material. The results of relevant Eastern Area test work are used in this Report.

In 2012, Hazen performed additional flotation test work including alternative reagent scoping, blended deposit flotation, and locked cycle flotation of underground material. The results of the underground locked cycle tests were used for characterizing the expected concentrate.

In 2015, Dawson conducted SAG Mill Comminution (SMC) and Bond work index testing on samples from the open pit and underground deposits. Dawson completed flotation test work on samples of specific rock types from both Western Area and Eastern Area Deposits, although additional work is required to characterize the metallurgical response by ore zones.

In late 2017/early 2018, the SGS laboratory (SGS) in Vancouver performed flotation test work on the underground ore composite samples to address all deficiencies of the historical test work, most notably the lack of conclusive locked cycle test work. In addition to flotation test work, dewatering test work at SGS as well as test work done in the equipment vendor labs was also conducted to adequately size solids/liquid separation equipment. All above mentioned test work findings are contained in the internal reports that are currently being used by the project execution team.

In January 2019, as a part of the Open Pit PFS test work campaign, comminution test work was performed at the SGS laboratory in Vancouver on eleven variability samples from the North and South Deposits to complement existing test work data, most notably the results from the DML 2015 test work campaign.

13.2 Historical Test Work Results

13.2.1 Comminution Testing

Three distinct test work programs evaluated the comminution parameters of the material. Metso performed the first series of test work in 2007, yielding the Bond ball mill work index (BBMWi) and Bond abrasion index (Ai) results shown in Table 13-1.

Table 13-1: 2007 Resource Development Inc. / Metso Comminution Testing

Sample	BBMWi (kWh/t)	BBMWi (kWh/st)	Ai (g)
North Open Pit	12.59	11.42	0.2064
NW High Grade area of North	13.59	12.33	0.3238
East/E2	12.47	11.31	0.2627

While these tests were performed to professional standards, the results are used as a comparison for the 2010 G&T and 2011 Hazen testing, which indicated generally harder material and examined a greater variety of mineralized material. The G&T testing consisted of only BBMWi and Ai testing, while the Hazen test regime evaluated these in addition to the JKTech parameters. The results of these test regimes are shown below in Table 13-2 and Table 13-3.

Table 13-2: 2010 G&T Comminution Testing

Sample	BBMWi (kWh/t)	BBMWi (kWh/st)	Ai (g)
North	14.44	13.10	0.2257
South Starter	11.57	10.50	0.0536
South	14.39	13.05	0.1488
E2	**15.00**	**13.61**	**0.2687**
East	11.98	10.87	0.1470

Table 13-3: 2011 Hazen Comminution Testing

Data by	Sample	BBMWi (kWh/t)	BBMWi (kWh/st)	Ai (g)	A	b	A x b	ta
JKTech	North Hornfels	15.27	13.85	0.3929	63.3	0.64	40.5	0.33
CSS	North Skarn	13.80	12.54	0.2801	51.1	1.17	59.8	0.93
JKTech	South Endoskarn	15.30	13.89	0.1568	53.7	2.00	107.9	0.80
CSS	South Magnetite	12.50	11.40	0.0695	50.6	2.68	136.1	2.56
JKTech	E2	14.80	13.51	0.4281	67.4	0.68	45.8	0.31
CSS	East	12.70	11.53	0.2130	54.0	1.05	56.7	0.96

BBMWi results indicate that the open pit ores are somewhat harder than the underground ores. Ai results show the underground ores to be slightly more abrasive than the open pit ores.

In addition to these two test regimes, another set of comminution test work was performed in 2015 at the Dawson laboratory in Salt Lake City, Utah. The results from this test program are shown below in Table 13-4.

Table 13-4: 2015 Dawson Comminution Testing

Data by	Sample	Ore Type	BBMWi (kWh/t)	BBMWi (kWh/st)	A	b	A x b	ta
Dawson	Hornfels	Open Pit	13.4	12.1	51.8	1.23	63.7	0.54
Dawson	Skarn # 1 Upper	Open Pit	14.2	12.9	57.4	0.86	49.4	0.40
Dawson	Skarn # 2 Lower	Open Pit	18.7	17.0	68.2	0.56	38.2	0.34
Dawson	Skarn # 3 Upper	Open Pit	15.1	13.7	61.6	0.71	43.7	0.36
Dawson	Skarn # 4 Lower	Open Pit	16.8	15.2	90.1	0.67	60.4	0.26
Dawson	Endoskarn	Open Pit	16.7	15.1	54.3	0.61	33.1	0.28
Dawson	Mag Zone 1	Open Pit	15.0	13.6	62.4	0.70	43.7	0.29
Dawson	Mag Zone 2	Open Pit	14.5	13.1	69.0	0.67	46.2	0.30
Dawson	Mag Zone 3	Open Pit	12.1	10.9	56.3	1.20	67.6	0.45
Dawson	Mag Zone 4	Open Pit	15.7	14.3	71.1	0.48	34.1	0.23
Dawson	Skarn Zone 5	Open Pit	14.9	13.6	75.9	0.48	36.4	0.31
Dawson	E2 Magnetite	Underground	14.6	13.3	79.4	0.46	36.5	0.25
Dawson	E2 Skarn	Underground	15.8	14.3	84.0	0.35	29.4	0.23
Dawson	East-North Endoskarn	Underground	12.9	11.7	63.2	0.62	39.2	0.31
Dawson	East-North Magnetite	Underground	10.2	9.2	70.2	0.73	51.2	0.35
Dawson	East-South Magnetite 1	Underground	11.4	10.3	54.4	1.88	102.3	0.74
Dawson	East-South Magnetite 2	Underground	11.1	10.0	56.0	1.96	109.8	0.87
Dawson	East-South Endoskarn	Underground	13.9	12.6	65.4	0.60	39.2	0.31

The above results relevant to the Eastern Area Deposits were used by Sedgman to model and design the grinding circuit. The values were assigned to lithology units and scheduled across the mine life on a monthly basis. Any values above design parameters would be stockpiled beside the low grade ore and blended later in the mine life to maintain mill feed within specification. Refer to Item 17.0 for the underground process plant design criteria.

13.2.2 Rougher Flotation Testing

Rougher flotation test work was performed on ores from both open pit and underground materials to examine the effects of grind size, retention time, reagent suite and reagent addition levels on copper recovery. The main results from this test work indicated that:

- Finer grinding improved recovery for all ores more than changes to any of the other parameters
- The bulk of the copper in the feed to the rougher flotation cells was recovered into the concentrate relatively quickly

Test work results for the Eastern Area Deposits indicated that a laboratory flotation time of 8 minutes, scaled up to 24 minutes, would be an appropriate rougher flotation time for the ores with a grind size of P_{80} 100 µm. It is believed that these flotation conditions should produce copper rougher recoveries of approximately 92%.

13.2.2.1 Rougher Flotation of Mineralized Material

Initial metallurgical flotation testing for the North, East and E2 Deposits was performed in 2007 by Hazen. This test work was detailed in reports by Hazen, as well as Resource Development Inc. Additional testing was conducted later by G&T in 2010 on the North, South, East and E2 Deposits. Data from the test work performed by G&T was supplied to Nevada Copper. Additional test work programs were conducted by Hazen in 2011 and 2013 and by Dawson in 2015.

The initial test work performed in 2007 was on two composites prepared from the open pit North Deposit identified as North and North High Grade, as well as composites from the East and E2 Deposits. First phase testing was performed to determine grind-recovery relationships associated with rougher flotation. Copper recovery increased as grind size decreased. Results from both the East and E2 composite samples showed recoveries ranging from 83.9% copper recovery at the coarsest grind to 97.1% copper recovery at the finest grind. Flotation times of 12 minutes were used in the tests, although it was noted that the concentrate reached 90% copper recovery in as little as four minutes.

Based on these results, as well as later tests by G&T in 2010, a flotation time of eight minutes was selected as appropriate for rougher flotation. A scale-up factor of three was applied to the test rougher flotation residence time, resulting in an ultimate proposed rougher flotation time of 24 minutes.

Eight additional tests were performed by G&T in 2010 aimed at optimizing the grind size parameters.

Hazen conducted a further 31 flotation tests, including 1 locked cycle test, on East, E2 or East/E2 composites. Dawson conducted a further 21 rougher flotation tests in 2015, with a flotation time of 16 minutes.

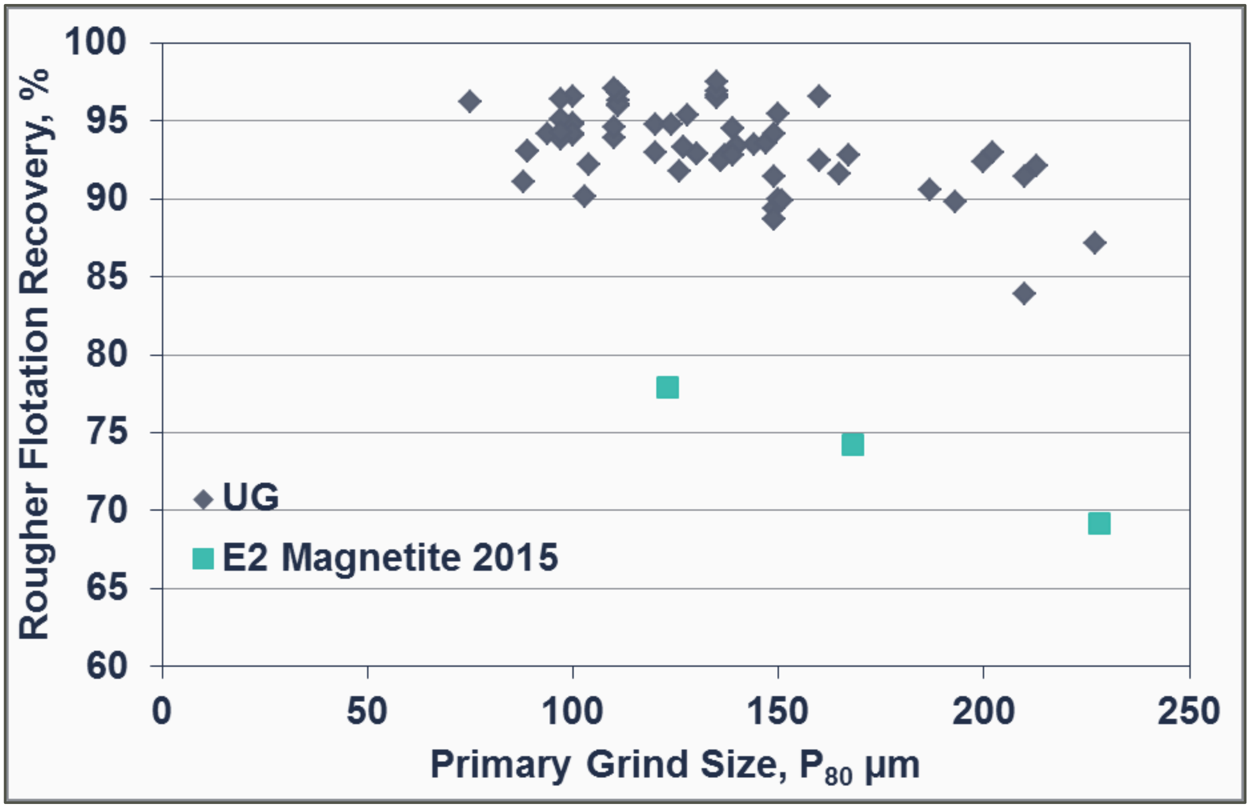
Ultimately, results from these tests were used for development of the design criteria and process flowsheets.

The first tests performed were standard one-product rougher flotation tests. These tests, along with results from later tests, showed that primary grind size had the greatest impact on rougher flotation recovery. Overall rougher flotation copper recovery achieved during these tests ranged from 83.9% to 93.0%, average 90.0%, at the coarser end (P_{80} 187 µm to 227 µm) and from 91.1% to 97.1%, average 94.6%, at the finer end (P_{80} 75 µm to 111 µm). These results excluded three tests on E2 magnetite ore that gave rougher recoveries of 69.2% to 77.9% at grind size P_{80} of 228 µm to 123 µm. Four tests on ore mapping samples (East/E2 Deposits) also gave poor rougher flotation recoveries, ranging from 67.0% to 80.1%. As these tests were not repeated it is not possible to determine whether or not the flotation conditions were optimized for these samples. The grind conditions for the mapping tests were 149 µm primary grind and 44 µm regrind. These conditions were not optimized.

Test work results indicated that ores from the Eastern Area Deposits generally responded favorably to rougher flotation. The grind-recovery results for the underground ore are depicted in Figure 13-1 and shows that the rougher flotation recovery generally increased with the decrease of the particle grind size. The lower than expected rougher flotation recoveries for the E2 magnetite sample may have been due to a higher collector addition requirement for this ore.

Prior studies on the underground project stated that further test work would be required to determine the reason for the poor flotation response. To complete this test work, the bulk flotation, batch cleaner and locked cycle flotation tests with revised, and optimized conditions were repeated in December 2017 at SGS in Vancouver, British Columbia. These updated test results were then carried into the detailed design of the underground processing facilities.

Figure 13-1: Primary Grind Size versus Rougher Flotation Recovery



Source: Tetra Tech, 2017 Technical Report.

A primary grind to P_{80} 100 µm, with a cleaner regrind to P_{80} 28 µm, in the proposed process plant is expected to achieve an estimated 92% copper flotation recovery for the underground ores.

The rougher flotation concentrate grades showed a general trend of a slight increase in average grade with decreasing grind size, i.e., the average grade at the coarser end was 11.0% Cu and increased to 13.6% Cu at the finer end. The E2 magnetite samples achieved an average rougher concentrate grade of 18.8% Cu, while the four East/E2 mapping samples had an average rougher concentrate grade of 16.6% Cu. The higher rougher concentrates for these samples tend to suggest that the flotation conditions were not optimized for these samples.

A review of the above test work indicated that a rougher flotation time of 10 minutes with a scale-up factor of 2.5, giving a plant rougher flotation time of 25 minutes should be adequate to achieve a rougher copper recovery of approximately 92%.

13.2.3 Cleaner Flotation Testing

Open-circuit and locked cycle cleaner flotation tests were performed by G&T in 2010 and Hazen in 2012 and 2013 to evaluate regrind size, cleaner flotation retention time, pH, reagent choice and reagent addition levels. Results indicate that regrinding to a nominal P_{80} 28 µm will be required to achieve a saleable concentrate grade targeting 26.0% Cu. A finer regrind size produced higher average final concentrate grades, i.e. 27.6% Cu compared to 24.9% Cu, at a slightly higher overall copper recovery, i.e., 88.8% compared to 87.2%.

A single locked cycle test on an East/E2 composite was conducted by Hazen in 2012 and achieved a final concentrate grade of 27.4% Cu at a copper recovery of 85.1%. It is recommended that this test should be repeated at a finer grind and regrind sizes, i.e. 100 µm and 28 µm compared to the 149 µm and 44 µm used in the test work. An analysis of the results of this test also indicated that the test had not stabilized when it was terminated.

Some tests have shown that higher pH has a positive impact on both copper grade and recovery. Results of these tests were used to size the regrind mill and flotation cells in the plant, as well as to determine the appropriate reagent additions, including lime to control flotation pH. Concentrate analysis values encountered in each test regime for payable and impurity metals are listed below in Table 13-5.

Table 13-5: Projected Concentrate Analyses for the East and E2 Underground Deposits

Item	East Deposit	E2 Deposit
Copper (Cu) %	22 to 24	28 to 30
Gold (Au) g/tonne	1.2 to 4.7 2.6 avg	0.7 to 9.7 4.5 avg
Silver (Ag) g/tonne	16 to 87 50 avg	12 to 145 87 avg
Iron (Fe) %	25	31
Sulfur (S) %	28	32
Acid Insol %	16.2	0.9
Calcium (Ca) %	0.23	0.18
Magnesium (Mg) %	0.13	0.04
Mercury (Hg) ppm	18	<0.1
Antimony (Sb) ppm	< 2	< 2
Arsenic (As) ppm	26	<5
Lead (Pb) ppm	1,188	1,657
Zinc (Zn) ppm	618	1,429
Selenium (Se) ppm	< 1	< 1
Molybdenum (Mo) ppm	35	4
Cobalt (Co) ppm	31	153
Nickel (Ni) ppm	141	114
Manganese (Mn) ppm	19	30
Bismuth (Bi) ppm	< 2	< 2
Cadmium (Cd) ppm	< 0.5	< 0.5
Fluoride ppm	491	18
Chloride ppm	< 5	< 5

The concentrate analysis from the 2010 G&T test work forms the basis for the expected impurity profile, however, these results were obtained in single-pass flotation testing of the material. The concentrate profile obtained in the 2012 Hazen testing came from a locked cycle test of the East Deposit material, which demonstrated the grades obtained in single-pass testing for payable metals could also be achieved with recirculating conditions. Prior studies on the Underground Project recommended that further detailed concentrate analyses should be conducted. This was done in December 2017 at SGS in Vancouver, British Columbia, when the locked cycle tests with revised and optimized conditions were conducted, and detailed concentrate analysis was obtained.

As a result of these underground ore tests, a nominal regrind size of P_{80} 28 µm was selected for use on the rougher flotation concentrate from the comingled materials. The pH level of the cleaner flotation circuit will be maintained at 11.0 to improve recovery. The proposed retention times in the plant for first cleaner flotation, first cleaner scavenger and second cleaner flotation will be 21, 30 and 15 minutes, respectively.

Test work indicates that the comminution and flotation circuitry proposed will provide an overall copper recovery of 92% with a targeted concentrate grade of 26.0% Cu. Estimated overall recoveries for copper, gold and silver are listed in Table 13-6.

Table 13-6: Overall Metal Flotation Recoveries

Recovery by Metal	Value
Cu Recovery (%)	92.0
Au Recovery (%)	78.0
Ag Recovery (%)	70.0

13.2.3.1 Cleaner Flotation of Underground Mineralized Material

Tests designed to consider regrind and cleaner flotation of the combined East and E2 ores were performed during the second stage of the 2007 Hazen test work. These tests included open-circuit cleaner flotation with two stages of cleaning and a single cleaner scavenger. Concentrate grade of the second cleaner ranged from 25.5% Cu to 30.3% Cu, with an overall copper recovery of between 73.0% and 86.6%. It was suggested that the wide range of grades and recoveries could possibly be due to presence of talc, which was observed during the tests. All cleaner flotation tests were performed at a pH of 11.0 to 11.5 to suppress the pyrite.

A third phase of the 2007 Hazen flotation test work employed locked cycle tests to determine the flotation response to recirculating loads. The test was performed as a 12-cycle test with a regrind P_{80} of 325 mesh (44 µm). Tailings from the second cleaner as well as the cleaner scavenger concentrate were screened at 400 mesh (38 µm), with the plus 400 mesh (38 µm) material reporting back to the regrind feed. Average copper recovery realized, after equilibrium conditions were met, was 85.1% with a corresponding concentrate grade of 27.4% Cu.

A second set of test work was performed by G&T in 2010 on composite samples from the East and E2 deposits. Primary grind size, regrind size, reagent dosage, float time and pH were varied in these tests in an effort to determine optimal flotation parameters. Two additional tests were performed with the optimized parameters derived during the earlier tests. These tests were ultimately used to derive the underground project design criteria for development of the underground process flow sheets.

A second set of one-product cleaner flotation tests performed by G&T. These tests varied regrind sizes (both before and after the first cleaner), pH and reagent dosage. Overall copper recoveries for these tests ranged from 75.5% to 94.0%, achieving a final concentrate grade between 33.3% Cu and 18.9% Cu. The inversely proportional recoveries and concentrate grades were highly dependent on regrind size with the maximum recovery occurring at coarser grinds and maximum grades occurring at finer grinds. This indicates that the optimal regrind size will be as coarse as possible while still achieving a targeted concentrate grade of 26.0% Cu.

Reagents and flotation times chosen for these tests performed well, although there was no evidence that an increase in flotation times beyond 12 minutes would significantly increase recovery.

The test work conducted by Hazen in 2011 to 2013 on the East1 and E2 composites included a single locked cycle test that achieved a final concentrate grade of 25.7% Cu at a copper recovery of 95.2% for cycles G to I. For this locked cycle test the primary grind was a P_{80} of 82 µm and a regrind size was a P_{80} of 43 µm. The rougher flotation time was 8 minutes, the cleaner 1 flotation time was 4 minutes, the cleaner 1 scavenger flotation time was 3 minutes and the cleaner 2 flotation time was 4 minutes. These times would be equivalent to 20 minutes, 12 minutes, 9 minutes and 12 minutes residence times in the full-scale plant. Caution must be applied when using these results as the calculated head grade was 2.72% Cu, which is significantly higher than the projected mill feed for the proposed plant. Prior studies on the underground project recommended that additional locked cycle flotation tests, using optimized conditions, should be conducted on composite samples that represent the most likely mill feed for Year 1, Year 2 and Years 3 to 5.

To address this, locked cycle tests with revised and optimized conditions were repeated in December 2017 at SGS in Vancouver British Columbia. These updated tests were then carried into the detailed design of the underground processing facilities.

Based on the aforementioned test work, the laboratory flotation times, scale-up factors and full-scale residence times selected for the proposed underground plant are given in Table 13-7.

Table 13-7: Laboratory and Plant Flotation Residence Times

Flotation Circuit	Laboratory Flotation Time (minutes)	Scale-Up Factor	Plant Residence Time (minutes)
Rougher	10.0	2.5	25.0
Cleaner 1	7.0	3.0	21.0
Cleaner 1 Scavenger	10.0	3.0	30.0
Cleaner 2	5.0	3.0	15.0

G&T test work data showed that increased collector dosage during rougher flotation had a positive effect on cleaner concentrate grade and a negligible although potentially positive effect on rougher recovery. An increase in cleaner flotation frother addition results in markedly higher overall recovery with a subsequent decrease in concentrate grade. Given the overall high concentrate grades achieved, there is little support for increasing reagent dosage above a minimum level that adequately facilitates flotation. In this case, a minimum reagent addition of approximately 11 g/st ore for Aerophine 3418A and 15 g/st ore for methyl isobutyl carbinol (MIBC) to rougher flotation was shown to produce greater than 90% rougher copper recovery.

Cleaner flotation circuit addition of 6 g/st ore for Aerophine 3418A and 20 g/st ore for MIBC with the lime addition required to raise the pH to target levels culminated in an estimated overall recovery of 89.3% and a copper concentrate grade of 25.5% Cu. The use of lime to alter pH and suppress pyrite had inconclusive results. Further testing is planned to optimize the reagent suite of collector and frother, as well as lime addition, for the East and E2 deposits. Prior studies on the underground project recommended test work to find an alternative frother as a result of the health, safety and environmental concerns associated with MIBC.

To address this, an alternative and non-flammable W31 frother was tested in December of 2017 at the SGS lab in Vancouver.

Prior studies on the underground project recommended to test alternative collectors, which was also done during the above mentioned test work in 2017, when a combination of 3418A and A3477 collectors was used.

13.2.4 Dewatering Test Work

13.2.4.1 Concentrate Thickening

The cleaner concentrate achieves the targeted copper grade but requires dewatering prior to shipping. Concentrate thickening will be used to produce slurry with a higher solids density, which in turn will be further dewatered by filtration. Kynch settling tests were performed to evaluate the settling behavior of the cleaner concentrate and determine thickener unit area requirements as well as the necessary flocculant addition. Filtration tests examined what kind of filters can be used and required filtration time, as well as the extent to which the feed slurry can be dewatered.

Copper concentrate was easily thickened and the addition of flocculant improved settling behavior. Tests also demonstrated the amenability of thickened concentrate to filtration. The process plant should be able to provide copper concentrate product with only 8% w/w moisture using pressure filtration. Test work by Bilfinger indicated that a final concentrate moisture of approximately 9% w/w moisture content should be achievable. In the absence of definitive concentrate test work, a conservative 10% w/w moisture content for the final copper concentrate filter cake was selected for design. Additional confirmatory final concentrate filtration test work should be conducted once sufficient amount of concentrate has been generated from future test work programs detailed in Item 26.0.

Initial settling tests on the East/E2 materials were performed in 2007 by Hazen Research. These tests revealed that a concentrate thickener underflow density of 60% w/w solids was achievable without the use of flocculant. Unit areas derived from these tests span a range of values, with the smallest being 0.14 ft^2/stpd to as high as 1.13 ft^2/stpd concentrate. In general, the addition of flocculant, up to 13 g/st concentrate, results in significantly smaller unit areas as well as better-defined solid-liquid interface. The tests also suggest that higher feed solids densities have a negative impact on settling behavior. Table 13-8 shows the concentrate settling results for the samples tested.

Table 13-8: Hazen 2007 Concentrate Settling Behavior

Description	East/E2	
Unit Area (ft2/stpd)	0.67	0.24
Settling Rate(ft/hr)	3.7	51.0
Feed % Solids (w/w)	33.7	13.5
Final Pulp Density (% w/w solids)	57.0	55.0
Dose (g/st)	3.3	9.9
Flocculant	Hychem NF-301	

Due to the limited amount of cleaner concentrate available, these tests were performed on a smaller scale than would otherwise be used. As such, the results of this test work are indicative that the concentrate can be successfully thickened, but could not be used for equipment sizing purposes.

A second series of thickening tests was run in March of 2008 by Hazen Research, targeting concentrate thickener underflow density of 60% for both Deposits. The average unit area from these two tests, which in the test work corresponds to a unit area of 0.21 ft^2/stpd, resulted in the scaled-up (safety factor of 10) unit area of 2.10 ft^2/stpd for the process plant. This has resulted in a concentrate thickener diameter of 34 ft with a flocculant addition of 5 g/t.

Additional confirmatory concentrate thickening tests were conducted at the SGS laboratory in early 2018 in order to confirm concentrate thickener sizing.

13.2.4.2 Concentrate Filtration

Test work was performed in November 2007 to examine the parameters associated with dewatering the thickened concentrate utilizing a bench scale vacuum filtration unit. The feed slurries had solids contents ranging from 62% w/w to 66% w/w, as well as associated flocculant contents. Results ranged between 75% w/w and 83% w/w solids content and no solids were observed in the filtrate.

While pressure filters were not specifically tested with the concentrate material, they have been used successfully for similar applications. Test work conducted on similar applications suggests that pressure filters will work well at filtering the concentrate from the Underground and Open Pit Projects. Final concentrate moisture of 8% w/w or less should be achievable using pressure filtration.

Additional filtration test work performed by Bilfinger in 2014 indicated that a final concentrate moisture content of approximately 9% w/w should be achievable. In the absence of definitive test work data, a moisture content of 10% w/w was selected as the design point for the concentrate filtration plant. It is recommended that additional pressure filtration testing should be performed when sufficient final copper concentrate becomes available for the tests proposed in Item 26.0.

13.2.4.3 Tailings Thickening

Tailings discharges from the flotation circuit will require significant dewatering. Thickened tailings will be either pumped to the DST pressure filters or to the paste backfill plant for the preparation of paste used as backfill in the underground mine. Kynch settling tests were performed to determine the settling behavior of the flotation tailings.

A substantial amount of test work was performed by numerous laboratories to examine the settling behavior of tailings material. Initial test work was performed by Hazen in November 2007 utilizing combined rougher and scavenger flotation tailings from the North, North High Grade and East composite blends. Unit area requirements were shown to vary from 0.11 ft2/stpd to 3.9 ft2/stpd for high and no flocculant additions, respectively. Table 13-9 is a summary of the settling behavior as determined during the 2007 Hazen tailings test work program.

Table 13-9: Hazen 2007 Tailings Settling Behavior

Description	North			North HG			East/E2		
Unit Area (ft2/stpd)	3.9	0.21	0.12	2.81	0.15	0.11	3.0	0.15	0.13
Settling Rate (ft/hr)	0.84	29.5	51.8	1.1	61	91	1.2	77	91
Feed (% w/w solids)	23.7	27.6	26.7	25.3	25.1	24.9	24.5	23.8	23.7
Final Pulp Density (% w/w solids)	46	59	59	50	63	62	57	64	63
Flocculant	**Hychem NF-301**								
Dose (g/st)	0	24	51	0	17	25	0	14	32

A second set of settling tests were performed by Outotec in April 2010 to examine the suitability of the tailings for use in a high rate thickener, as well as the effects of flocculant addition. Initial tests revealed a maximum settling rate of 0.709 ft/hr for un-flocculated tailings at a feed density of 15% w/w solids. This is well below the accepted minimum settling rate required for producing clear overflow in a high rate thickener, indicating flocculant addition is needed. The tests also showed that the settling rate improves with increasing feed solids density. Results when adding flocculant at a 15% w/w solids feed density resulted in significantly higher settling rates ranging from 127.56 to 204.10 ft/hr, thus indicating that high rate thickening is viable with significant flocculant addition. Table 13-10 shows the settling rates of 2010 and 2011 test results by Outotec and Pocock.

Table 13-10: Tailings Settling Rates

Description	Outotec - 2010	Pocock – 2011 - Unscaled
	East South	**Underground**
Settling Rate (ft/hr)	204.1	141.7
Flocculant	Cytec A-130	Hychem AF-303
Dose (g/st)	27.5	16.5

Additional settling test work was performed by Pocock in April 2011 examining the tailings response under both static and dynamic thickening conditions with various solid feed densities. The tests samples were previously processed by Hazen and subsequently categorized as underground final tails. The static "conventional" thickening test results were used to determine the unit area requirements ultimately used in the process design criteria. Caution must be applied to the use of these results as the underground ore tailings supplied to Pocock had a P_{80} of 150 µm compared to the current design P_{80} of 100 µm. No information was available for the particle sizing for the sample supplied to Outotec.

Static thickening tests indicate ideal flocculant (Hychem AF 303) addition of 16.5g/st of feed solids for underground tailings. All unit thickening area results included a scale-up factor of 1.25 and all rise rates included a scale-up factor of 0.5. Unscaled rise rates were lower in general but comparable to those listed by Outotec. The differences in choice and dosage of flocculant are likely responsible for the differences in these tests.

Thickener unit areas ranged from 0.410 ft^2/stpd to 0.869 ft^2/stpd for the underground samples. Results from static thickening tests also indicate that the unit area will be at peak values in the range of 20% w/w solids to 25% w/w feed solids for underground material. A unit area of 0.474 ft^2/stpd was determined to be appropriate for the thickening of material from the East Deposit. As a result, the tailings thickener diameter of 59 ft was sized based on the throughput rates used in the design criteria.

13.2.4.4 Filtration

Filtration testing revealed that pressure filtration will be the most viable means of dewatering the thickener underflow to the maximum 15% w/w moisture content required for discharge as DST.

Tailings filtration tests were performed by Pocock to determine the best method of dewatering the thickener underflow for discharge DST. The underground tailings produced lower filtration rates than the 61.4 lbs/ft^2.hr (300 kg/m^2.hr) considered to be the lower limit for economic use of belt filtration, thus recommending the use of pressure filtration over vacuum filtration.

Pressure filtration tests revealed that filter cake moisture content could be reduced below 15% w/w moisture, as required for DST.

Additional pressure filtration test work was performed by Larox in April 2010. Average calculated filtration rates for the East and E2 rougher tails were between 302.3 and 373.7 kg/m^2.hr, with average cake moisture ranging from 10.0% w/w moisture to 13.9% w/w moisture. These tests produced similar final cake moistures as compared to the previous test work. Wet cake bulk densities were comparable between these tests and previous Pocock tests. Reported feed densities were significantly lower than those used in the Pocock tests, thus the resulting filtration rates for both are not entirely comparable. However, the results of the Larox test work support the conclusion of the Pocock that pressure filtration is of the tailings is a viable option.

Additional filtration test work was performed by Bilfinger in 2014. This test work examines the filtration characteristics of the Pumpkin Hollow underground ore at P_{80} of 100 µm and 150 µm. The test work has confirmed that the equipment sizing used in previous studies was correct.

13.2.5 Historical Test Work Review

The test work review has been completed on the data provided by Nevada Copper in the technical report entitled "*Integrated Feasibility Study NI 43-101 Technical Report on the Pumpkin Hollow Copper Open Pit and Underground Ore Project"* prepared for Nevada Copper Corp. by Tetra Tech in July 2015.

In addition to this review, results from the 2015 Dawson metallurgical test work campaign, which focused on variability comminution test work, were also reviewed.

This metallurgical test work review has been preliminary in nature and will need to be reviewed in further detail during the further phases of study. Sedgman was able to draw some pertinent conclusions from this review:

- In Sedgman's opinion, the historical test samples collected for the purpose of metallurgical testwork meet industry standards for representative samples for the various deposits, styles of mineralogy, lithology and the mineral deposit as a whole.
- Final copper concentrates were sufficiently free of the deleterious elements that can cause a significant effect on potential economic extraction.
- Based on the available comminution test work data, a single stage crushing followed by SAG milling, with pebble crushing, and secondary grinding in a ball mill would be a suitable processing solution
- 2015 Dawson comminution test work has revealed that ore from South Pit Endoskarn zone 1 is hard from the SAG milling perspective (A x b ore hardness value of 33.1)
- 2015 Dawson comminution test work has also revealed that ore from the same deposit was moderately hard from the ball milling perspective (BBMWi value of 16.7 kWh/t or 15.2 kWh/st)
- Rougher flotation kinetics were in general fast with acceptable copper recoveries being achieved
- High pH in the cleaner stages of flotation tended to be beneficial regarding concentrate copper grade
- While the majority of the cleaner test work involved three stages of flotation, the results indicate that for a full-scale plant, two stages of cleaner flotation should be adequate to achieve acceptable copper concentrate grades and recoveries.
- Based on the batch flotation test work final concentrates, grades of greater than 25.5% Cu at a copper recovery of greater than 89.3% should be achievable from the North and South deposit ores that were tested.

However, the locked cycle test work performed by Hazen in 2007 has shown that by selectively mining ore from the North Deposit, the test work achieved expected grades of 31.4% Cu and 90.9% Cu recovery (Table 13-11).

Table 13-11: Test Work Recoveries

Open Pit Locked Cycle Test work results	% Cu in Cleaner Conc.	Cu Recovery, %
North Composite sample	26.5	87.6
North High Grade Composite sample	31.4	90.9
Open Pit Tetra Tech report design values	25.5	89.3

Based on Table 13-11, the North Pit has overall higher recovery due to the blending of the high grade material in the North Deposit providing an average of 90% recovery. The South Deposit has lower average recoveries of 88%.

Limited detailed analysis of the final copper concentrates by Hazen indicated that the concentrates were relatively free from potential penalty elements such as mercury and arsenic.

A full analysis of gold and silver flotation responses was not possible due to these assays only being available for some tests. Gold and silver contents of the final copper concentrates will be in the payable range, but dependent upon the ore source being treated.

13.3 Open Pit 2018 Prefeasibility Study Test Work

During the Open Pit PFS, the plan was to conduct test work on the following areas to complement historical test work, while enabling new test work to be aligned with the new study mine plan and planned mining sequence of mining the North and South pits independently and later in the LOM together:

- Sulfide ore:
    - Composite samples: comminution test work to confirm ore properties such as hardness and its effect on the mill throughput and performance done on the samples that accurately represent first five years of the mill production
    - Composite samples: flotation test work to confirm metal grades and recoveries done on the samples that accurately represent the first five years of the mill production
    - Composite samples: liquid/solid separation test work to help size thickening and filtration equipment
    - Variability samples: comminution and flotation test work to establish ore characteristics and metal grades and recoveries beyond the first five years of the mill production
- Oxide ore:
    - Acid leaching test work to assess amenability of this option for treating oxide and transitioning material, which is to be stockpiled as waste

The PFS test work commenced in late November 2018 as the lithology of the mine plan was established during the Open Pit PFS work. Variability comminution and oxide leaching test work was not completed in time to be reported in the Open Pit PFS. The balance of the test work is ongoing and the findings will be included in the subsequent study updates.

13.3.1 Sulfide Ore Variability Test work

To better understand the ore hardness and compare its results with the historical test work data, 11 variability samples from the North and South pits were selected and shipped to SGS for the purpose of SMC and BBMWi comminution test work. Results from the test work are tabulated below (Table 13-12, Table 13-13).

Table 13-12: 2018 SAG Mill Comminution Variability Test Work Results

Sample ID	Hole ID	Deposit	Location depth	RX Type	A	b	A x b	Hardness Percentile	t_a	SCSE (kWh/t)	Hardness Percentile	DWI (kWh/m^3)	M_a (kWh/t)	M_h (kWh/t)	M_c (kWh/t)	Relative Density
Var-1	NC08-16	South	Upper	Endoscarn	95.4	0.37	35.3	71	0.27	11.72	86	9.5	20.6	16.3	8.4	3.37
Var-2	NC10-28	South	Upper	Endoscarn	87.6	0.37	32.4	78	0.24	12.24	91	10.6	21.9	17.8	9.2	3.47
Var-3	NC18-02	North	Upper	Hornfels	90.3	0.36	32.5	78	0.29	11.45	83	8.8	22.6	17.6	9.1	2.90
Var-4	NC18-02	North	Upper	Silicate Skarn	86.1	0.48	41.3	58	0.33	10.76	74	7.9	18.4	14.0	7.3	3.26
Var-5	NC18-03	North	Upper	Silicate Skarn	100.0	0.33	33.0	77	0.30	11.10	79	8.4	22.4	17.3	9.0	2.81
Var-6	NC15-18	North	Lower	Silicate Skarn	65.7	1.12	73.6	20	0.64	7.94	23	4.0	11.8	7.8	4.1	2.97
Var-7	NC12-26	South	Lower	Mag Skarn	59.8	0.77	46.0	49	0.32	9.88	58	8.2	16.4	12.6	6.5	3.75
Var-8	NC18-05	North	Upper	Silicate Skarn	71.9	0.55	39.5	62	0.36	10.26	65	7.2	19.5	14.6	7.6	2.85
Average					82.1	0.54	44.6	62	0.34	10.67	70	8.1	19.2	14.8	7.7	3.17

Results from the 2019 PFS variability SMC comminution test work conducted at SGS on the South Pit samples are similar to the results from 2015 Dawson test work campaign.

Ore from the South Pit Endoskarn zone is considered hard (A x b values in the 30 to 38 range) from the SAG milling perspective, as the average A x b value from the two samples is 33.9. For comparison, A x b values from the Dawson 2015 comminution test work were 33.1.

However, SGS comminution test work done on the Hornfels type deposit from the North Upper zone has revealed that this ore is potentially significantly harder from the SAG milling perspective than what was reported by Dawson in 2015. The A x b value from this sample was 32.5, which is much lower than the value of 63.7 originally reported in 2015. The latest results would characterize this type of ore as hard, compared to previous characterization of moderately soft from the SAG milling perspective.

For the sizing of the SAG mills in the current design phase, Sedgman has assumed a LOM 75th percentile A x b value of 38 that takes in account hard Endoskarn ore and potentially hard ore from the Hornfels deposit, as well as other softer ores from the South and North deposits, respectively. The latest test work results from SGS indicate that 75th percentile A x b value is somewhat lower at 34, indicating more presence of the harder ore in the early years of mine life than initially thought based on 2015 test work.

SGS test work results were obtained on January 15, 2019, after the PFS SAG Mill equipment sizing had been completed. While the comminution variability test results indicate a modest increase in hardness from a SAG mill perspective compared with the 2015 test work, Sedgman has assessed that the PFS SAG mill sizing should still be sufficient to meet the target throughput, provided that there is a reasonable ability to blend some of the harder ores with softer ores. Therefore, the SAG mill equipment size determined in the

Open Pit PFS remains unchanged. However, given the shift in distribution to include some harder ores, there is some risk that SAG mill throughput could be restricted below the 37,000 stpd design throughput target in Phase I of the Open Pit Project. This risk should be assessed and mitigated in the Phase II, for example by a more detailed look at the mine plan and blending options, and the potential to use more intensive blasting to limit the SAG mill feed size.

It is recommended in the next project phase to closely match more variability samples with the existing mine plan and perform comminution testing to further assess ore characteristics/hardness and its effect on the mill throughput, particle grind size and metal recoveries.

Table 13-13: Bond Ball Mill Work Index Test Work Results – 2018

Sample ID	Hole ID	Deposit	Location depth	RX Type	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile	Category
Var-1	NC08-16	South	Upper	Endoscarn	120	2,602	99	1.39	15.2	60	Medium
Var-2	NC10-28	South	Upper	Endoscarn	120	2,660	99	1.30	16.1	70	Moderately Hard
Var-3	NC18-02	North	Upper	Hornfels	120	2,439	94	1.18	17.1	77	Hard
Var-4	NC18-02	North	Upper	Silicate Skarn	120	2,414	96	1.54	13.9	45	Medium
Var-5	NC18-03	North	Upper	Silicate Skarn	120	2,485	95	1.29	15.9	68	Moderately Hard
Var-6	NC15-18	North	Lower	Silicate Skarn	120	2,386	95	1.31	15.8	67	Moderately Hard
Var-7	NC12-26	South	Lower	Mag Skarn	120	2,444	96	1.65	13.1	36	Moderately Soft
Var-8	NC18-05	North	Upper	Silicate Skarn	120	2,497	95	1.27	16.1	70	Moderately Hard
Average					120	2,491	96	1.4	15.4	61	

These results categorize the Pumpkin Hollow open pit ore as medium-hard from the ball mill perspective. Sedgman has accounted for this hardness by using BBMWi 75th percentile value of 14.3 kWh/st or 15.8 kWh/t (Dawson 2015 test work) for sizing of the ball milling circuit. The latest SGS test work have shown that 75^{th} percentile BBMWi value is virtually unchanged at 14.4 kWh/st or 15.9 kWh/t, validating the current PFS-stage ball mill design.

13.3.2 Oxide Ore Leaching Test work

Currently, copper oxide bearing mineralization and transitioning material is being treated as waste and the intent is to stockpile it separately for possible future processing. At the request of Nevada Copper, viability of the oxide ore leaching option has been investigated at the high level, as there may be up to 10 Mst of oxide material in the waste stripping rock.

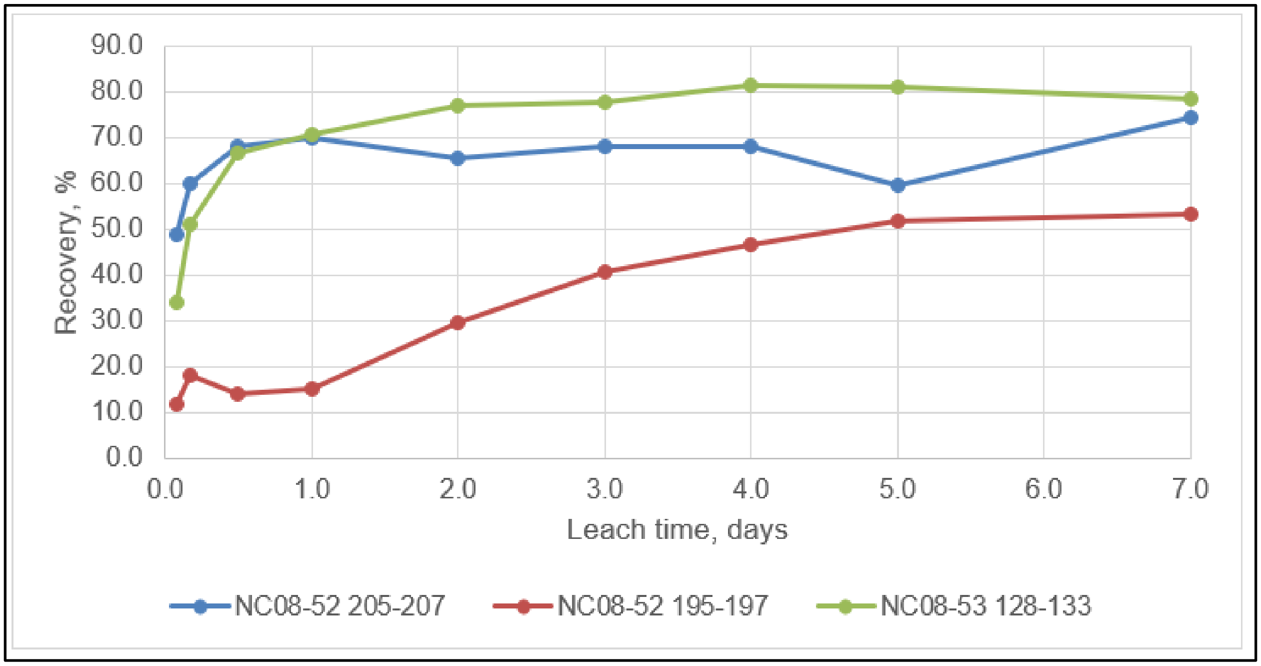
Three samples with the highest copper grade and proportion as a copper oxide were selected by Nevada Copper and shipped to SGS testing facility in Burnaby, British Columbia. Sample head assays are provided in Table 13-14.

Table 13-14: Oxide Ore Sample Head Assays

Sample ID	Cu, g/t	S, %	CuO, %	CaO, %	Fe2O3, %	MgO, %	Cu Total,%	Cu as Ox, %
NC08-52 205-207	2810	0.03	0.206	4.3	17.7	6.86	0.281	0.16
NC08-52 195-197	2650	0.18	0.171	19	20	3.24	0.265	0.14
NC08-53 128-133	4500	<0.01	0.421	29.3	23.4	1.69	0.45	0.34

Results of the all three individual bottle roll leach tests are presented in Figure 13-2.

Figure 13-2: Copper Extraction Rates from Three Oxide Samples



Source: Sedgman, 2019.

The overall highest copper recovery was for the NC08-53 sample, which had the highest copper grade and highest ratio of copper oxide in the feed. After seven days, the copper extraction rate was around 80%, however, it came with a high acid consumption of 54 kg/st of mineralized material (considering that similar copper recoveries can be achieved with the acid consumption of 18 kg/st of mineralized material)

The copper extraction from the other two samples was low, which is not surprising as they both had significantly lower copper grade and proportion as copper oxide. Acid consumption was also high at 55 kg/st and 138 kg/st of mineralized material, which can be attributed to higher presence of carbonates in these two samples.

Based on these results, it was concluded that the oxide leaching process option is unlikely to provide an economic benefit to the Open Pit Project, given the relatively small volume of potential oxide material. The viability of this option will mostly depend on the source of affordable sulfuric acid and favorable market copper price.

The PFS test work commenced in late November 2018, with the first lot of results dated January 2019 used in the Open Pit PFS. Outstanding results from February 2019 were not included in this Report, such as:

- Composite samples comminution and flotation
- Variability samples comminution and flotation
- Liquid/solid separation test work

These results, along with the results from the vendor-specific test work, such as specific energy regrind requirement and filtration and settling test work, will be included in further study of the stand-alone Open Pit Project, as outlined in Item 26.2.7.

Item 14.0 MINERAL RESOURCE ESTIMATES

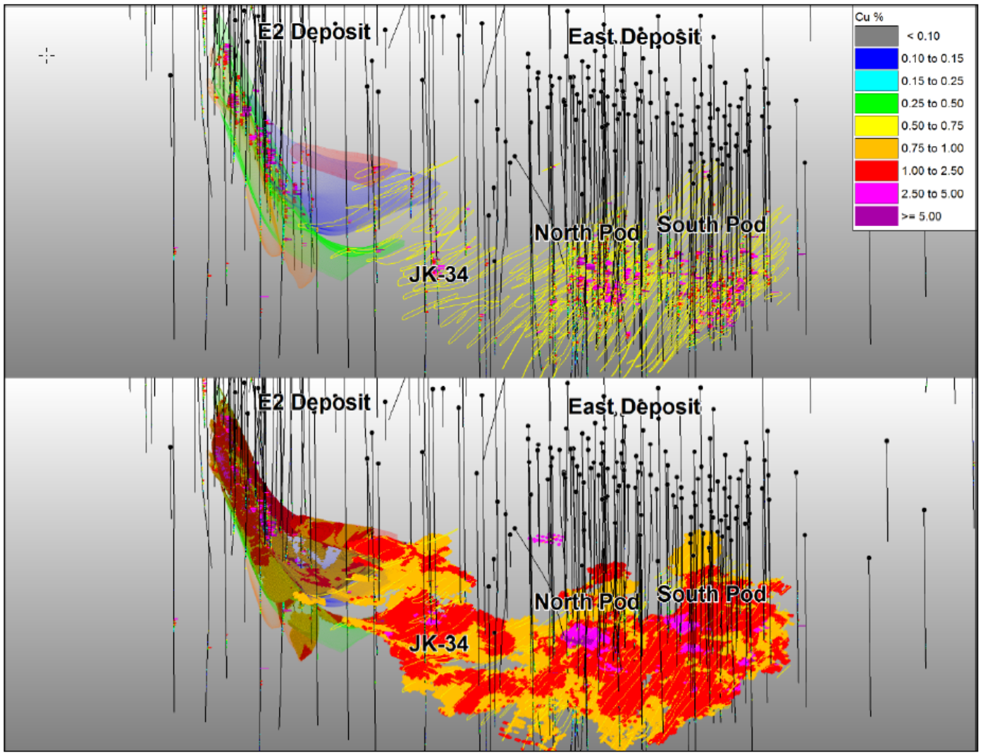
14.1 Underground (Eastern Area) Mineral Resource Estimation

The East Deposit model had been updated based on the addition of seven drill holes, completed at the end of 2011, and the E2 Deposit model has been updated based on the addition of two drill holes also completed at the end of 2011. The nine additional drill holes total 19,324 ft. The models also include additional improvements and considerations as result of evolving knowledge of the deposits and third party input and review. The estimation is similar to previous estimates with different methods used to construct the mineral zone models and other minor changes.

The East and E2 Deposits have been modeled with two separate block models due to data size constraints given the potential size of a combined block model. The East and E2 Deposits have distinctly different orientations. The mineralization of the East Deposit dips gently to the south-southeast where the E2 Deposit dips steeply north-northwest. Drilling between the Deposits in the JK-34 zone, has demonstrated the two deposits form the limbs of a single complex synclinal structure. At present, the drilling in the JK-34 zone is insufficient to completely unite the two deposits with comparable grade and thickness as in the richest areas, but the East and E2 Deposits can be considered a continuous mineralized system.

Figure 14-1 shows the East and E2 model drill holes, mineralized interpretation and all blocks greater than 0.75% Cu, both inside and outside of the mineralized interpretation. The viewing orientation is looking southwest from above.

Figure 14-1: East Deposits Mineralized Interpretation and Blocks 3D View (2017 Technical Report)



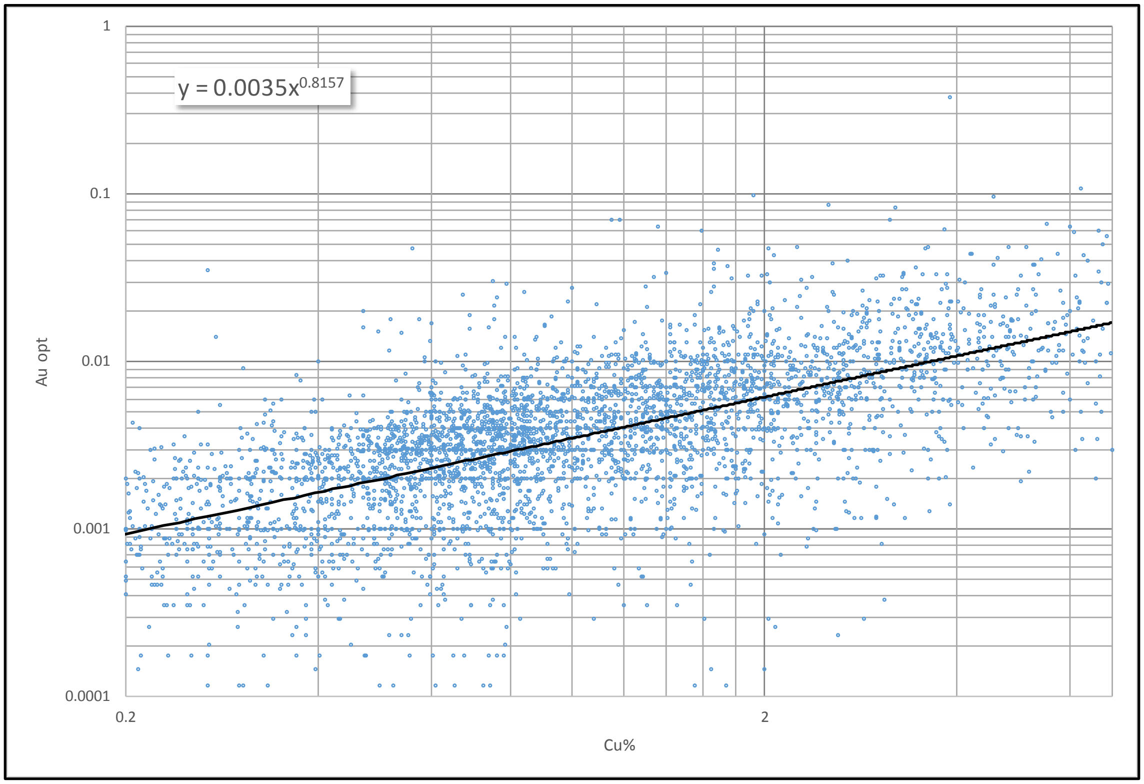
14.1.1 Input Data

The East and E2 models include analytical data intervals from 210 drill holes, 111 of which have been interpreted to intersect the 0.5% Cu grade shell, totaling 19,201 ft.

The drill hole database contains analytical data were locally copper, gold, silver, iron, molybdenum and sulfur was not available due to differences in the manner of the historic drilling program. Additional assaying of the pulps/rejects might be beneficial to increase the gold and silver data.

For missing intervals of copper, silver, iron, molybdenum and sulfur, detection limit values were inserted. For missing intervals of gold outside of the 0.5% Cu grade shell, detection limit values were inserted; for missing intervals of gold inside the grade shell, a regression equation based on the relationship of copper and gold was used to infill missing gold values, accounting for approximately 12% of gold intervals used for estimation within the grade shell. The regression equation used is Au oz/st = 0.0035*(Cu%) 0.8157. Figure 14-2 shows a scatter plot of existing copper and gold sample pairs used as the basis of the regression formula.

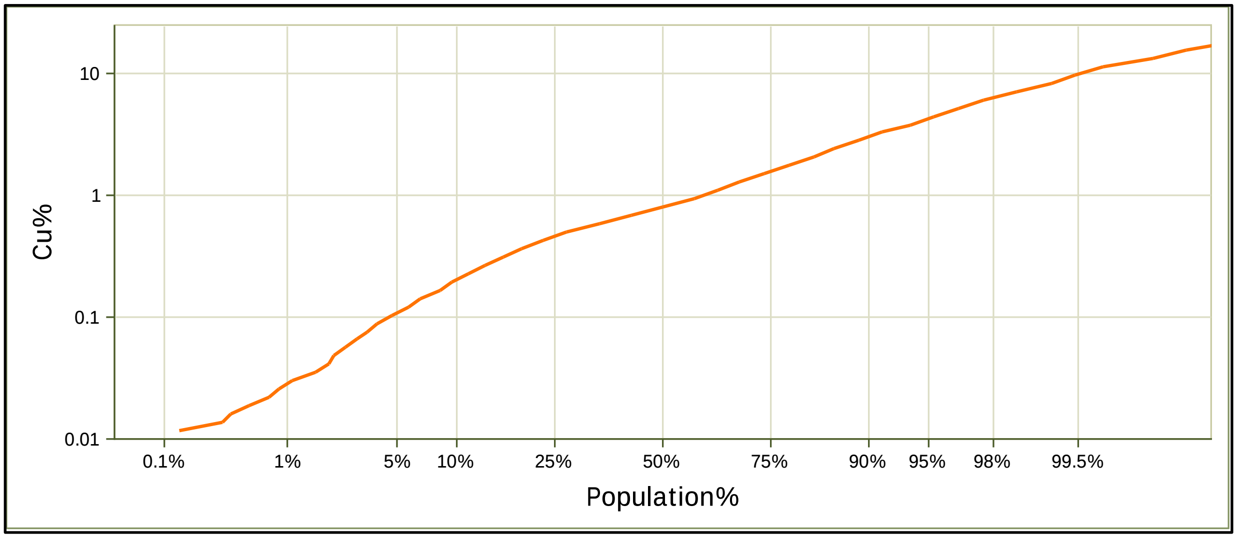
Figure 14-2: Scatterplot Basis of Au Regression (2017 Technical Report)



14.1.2 Grade Capping

Inspection of the probability and cumulative frequency plots suggest no capping of grades is required. Figure 14-3 shows the high grade end of the raw copper grade population within the grade shell is sufficiently linear when plotted as a log normal probability plot and does not have an erratic tail, confirming capping is not required.

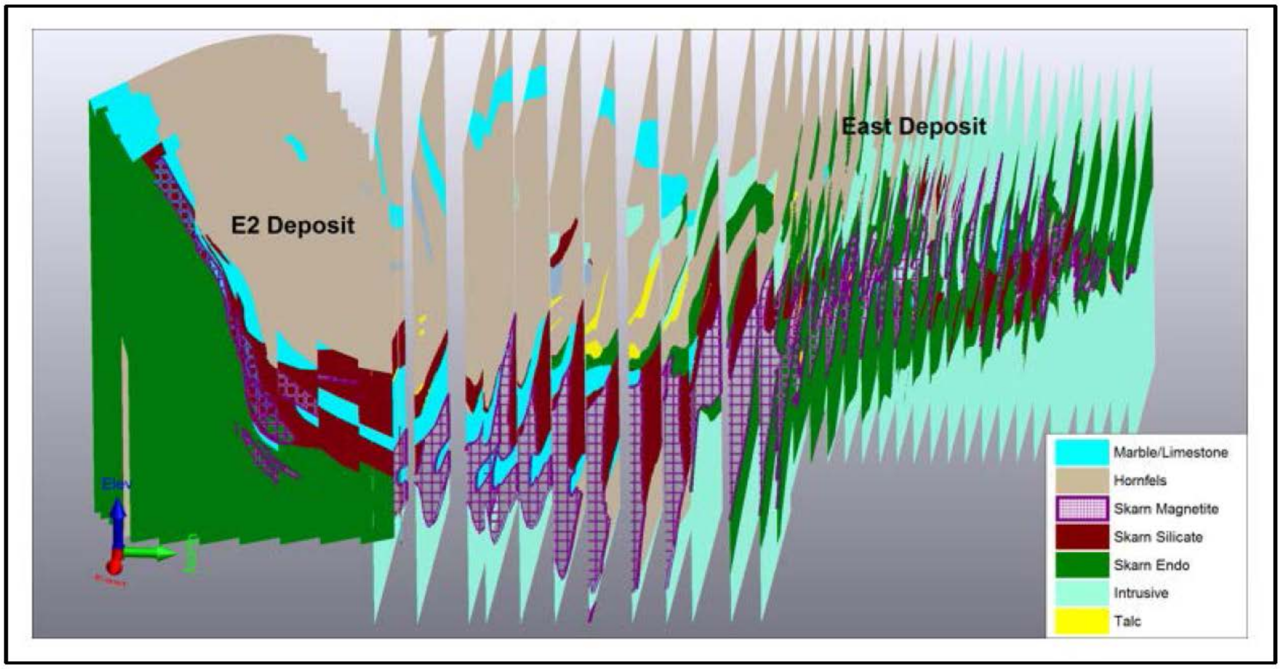
Figure 14-3: Log Normal Probability Plot Cu (2017 Technical Report)



14.1.3 Geologic & Mineralization Modeling

Block lithology codes were derived from paper cross section interpretations generated by Nevada Copper site geologist and then digitized and registered. Cross section locations are shown in Figure 14-4. Figure 14-4 shows a 3D “melon slice” section of the cross section interpretations looking west from above. Post-mineral alluvium and TV have been omitted from the figure. Lithology types interpreted for the Eastern Area are the same as used in the Western Area and listed in Table 14-16.

Figure 14-4: Lithologic Sections 3D Section Slice (2017 Technical Report)

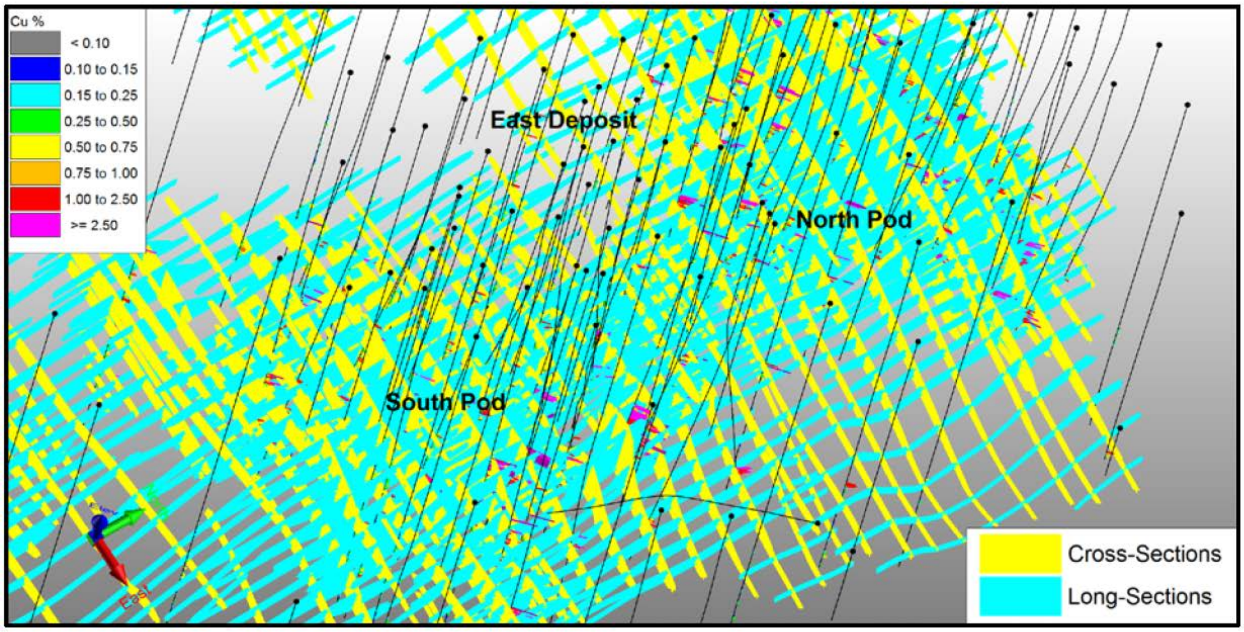


The 0.5% Cu grade shell, or mineral zone, for the East and the E2 Deposits were constructed using different methods because of differences in the shape of the mineralization between the two bodies. In the East Deposit, the mineralized shape is complex, typical of a contact skarn, having rapidly changing grade and shape over short distances near the contact with the country rock. The E2 Deposit is more tightly constrained and “vein like” in the most enriched areas.

For the East model, constructing a mineralized zone using a set of cross sections drawn at one orientation and connected by wireframing was attempted at first but proved too cumbersome. The resulting shape using wireframing reflected the shortcomings of the wireframing approach more than an accurate representation of mineralized body.

Instead, interpretation began with cross sections constructed on the same section northing as the lithology followed by additional mineralized sections constructed longitudinally (long section). Long-section spacing was reduced where additional detail was required. The cross and long sections were then reviewed against each other. Figure 14-5 shows the grade shell interpretation sections and drill hole traces looking northwest from above. The yellow lines are cross sections and the blue lines are long sections.

Figure 14-5: East Deposit Mineral (Grade Shell) Cross and Long Sections 3D (2017 Technical Report)



With the long and cross section interpretation approach, no wireframe or triangulation solid was created. The interpretations were converted to volumes or blocks by inverse distance weighting to the power of two (IDW2) using a "0" or "1" indicator. A fence of pseudo-drill holes was generated on each of the cross and long section lines. Intervals within the interpretation were coded as "1" and intervals outside the interpretation were coded as "0." Next, line orientations of the cross and long section segments were extracted from the lines. The long section line orientations represented local dip and the orientation of the cross sections represented roll. An IDW2 estimation of "1"s and "0"s was made with the local orientation of the search ellipse being guided from the section interpretation line orientations. Blocks with an indicator value of 0.5 and greater were coded into a blank block model as "in mineral," blocks with a code less than 0.5 were coded as outside. The resulting volume of the blocks selected as in mineral was checked against the extruded volume of the sections. Table 14-1 compares the extruded volumes to the resulting in mineral blocks.

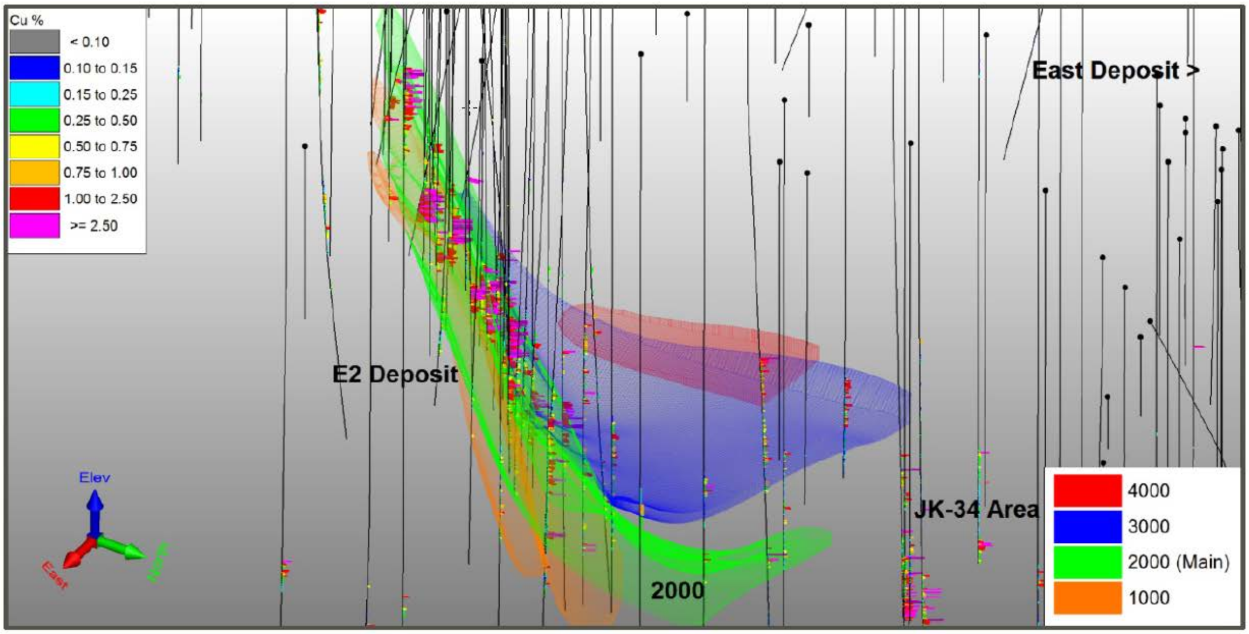
Table 14-1: Indicator Block Volume Check

Items	Extruded Cross Sections	Extruded Long Sections	Extruded Sections Average	Blocks
Volume (million ft^3)	1,025	1,011	1,018	1,000

Downhole sample intervals were assigned as inside or outside of the mineralized shape based on flagging from the cross and long section line outlines on section given their corresponding section window.

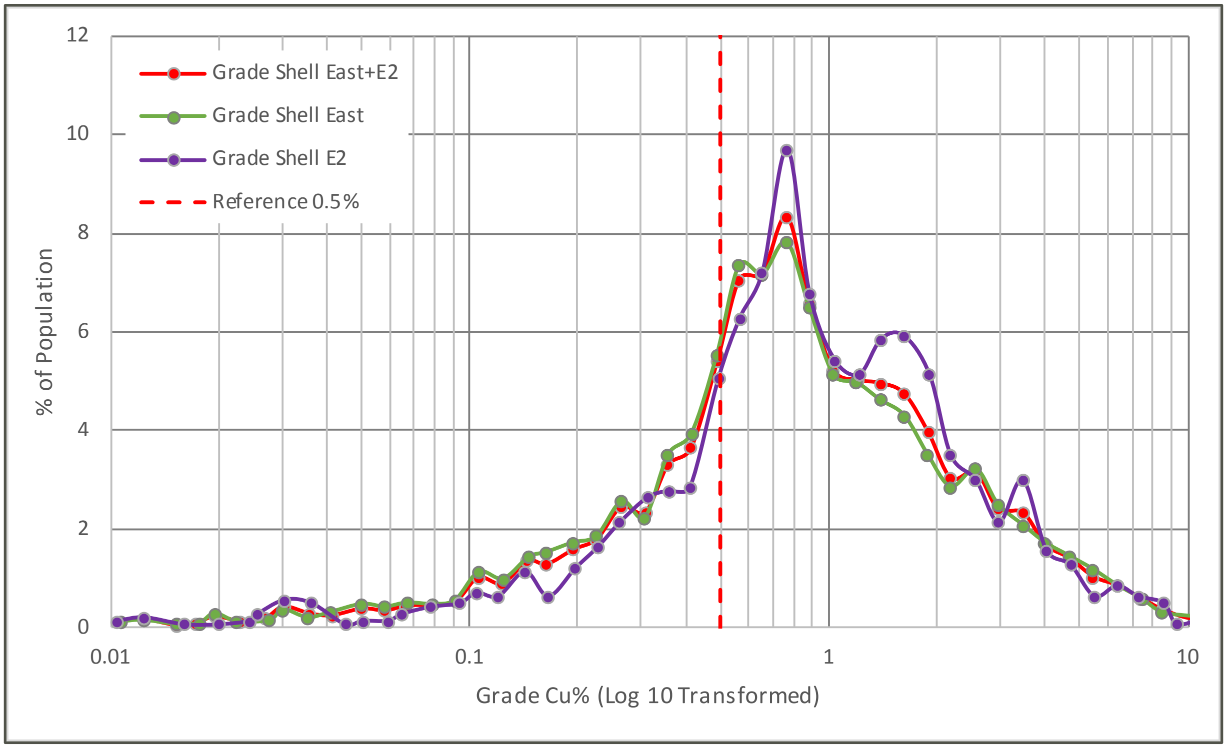
Mineral zone interpretation for the E2 Deposit was completed using implicit surface modeling. Four individual grade shell domains were modeled. Exit and entry points were generated for each of the drill holes and each of the domains. The exit and entry points along with guiding points were used to generate top and bottom surfaces. The top and bottom surfaces were then clipped and combined to make domain solids. Downhole sample intervals were assigned based on their location in respect to the domain solids. Figure 14-6 shows the grade shell domains as transparent solids and drill hole traces looking southwest from above. The solids have been named from bottom to top with codes 1000, 2000, 3000 and 4000. The names do not have a particular significance, and are a continuation of the four digit numeric code used in the West and East models. The primary resource domain is coded 2000.

Figure 14-6: E2 Deposit Mineral Solids 3D (2017 Technical Report)



As in the Western model, the interpreted grade shell domains are not grade contour boundaries but rather interpreted copper enrichment zones and include internal waste intervals. The population distributions of the two Deposits and the Eastern Area combined are shown in Figure 14-7.

Figure 14-7: Interpreted Mineralized Zone Grade Populations (2017 Technical Report)



14.1.4 Compositing

Raw sample intervals have been composited from a model length of 5 ft to a composite length of 10 ft. Raw assay interval centroids were coded by the grade shells, or section interpretations, and the composite intervals initiate at the boundary of the zone based on the coded raw assay intervals, resulting composites less than 5 ft were discarded for inadequate support. Composites were then coded with extruded lithology solids.

14.1.5 Variography & Search

Log-variography, pair-wise relative variography, indicator variography and correlograms were reviewed, and used to guide lognormal variography. Variograms were also reviewed and compared within the different deposits, rock types and mineral zones until a unified interpretation was developed for each element. Variograms were generated and reviewed for copper, iron, gold, silver, sulfur and molybdenum in multiple directions to establish directional variogram models and to guide search ellipse anisotropy.

Figure 14-8 to Figure 14-11 show copper and gold experimental omni-directional log variography for the East and E2 Deposits. Table 14-2 and Table 14-3 detail the omni-directional log variogram models used to estimate each element for the East and E2 Deposits. Following experimental variography, it was concluded secondary elements could be estimated with the ultimate search ranges and anisotropy established by copper although the actual modeled variograms are not the same.

Table 14-2: Log Variogram Omni-Directional Models East Deposit

Element	Nugget	Partial Sill C1	Range (ft) C1	Model C1	Partial Sill C2	Model C2	Final Range (ft)
Cu	0.5	0.16	70	Spherical	0.4	Spherical	300
Au	0.32	0.39	140	Spherical	0.2	Spherical	250
Ag	0.32	1.17	345	Spherical	-	-	345
Mo	0.7	0.9	250	Spherical	-	-	252
S	0.1	0.3	380	Spherical	-	-	380
Fe	0.1	3.4	300	Spherical	-	-	300

Table 14-3: Log Variogram Omni-Directional Models E2 Deposit

Element	Nugget	Partial Sill C1	Range (ft) C1	Model C1	Partial Sill C2	Model C2	Final Range (ft)
Cu	0.26	0.55	93	Spherical	0.27	Spherical	208
Au	0.5	0.6	100	Spherical	0.9	Spherical	180
Ag	0.25	1.8	140	Spherical	-	-	140
Mo	0.67	0.9	270	Spherical	-	-	270
S	0.06	0.06	130	Spherical	0.05	Spherical	300
Fe	0.1	1.4	300	Spherical	-	-	300

Figure 14-8: East Model Pair-Wise Relative Variography for Cu (2017 Technical Report)

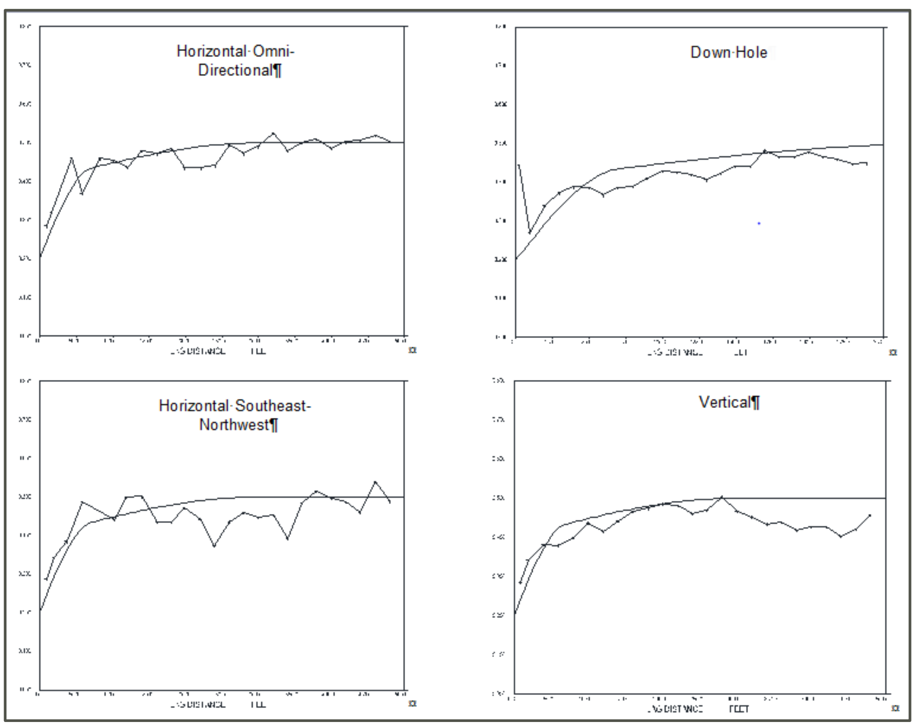


Figure 14-9: East Model Log-Normal Variography for Au (2017 Technical Report)

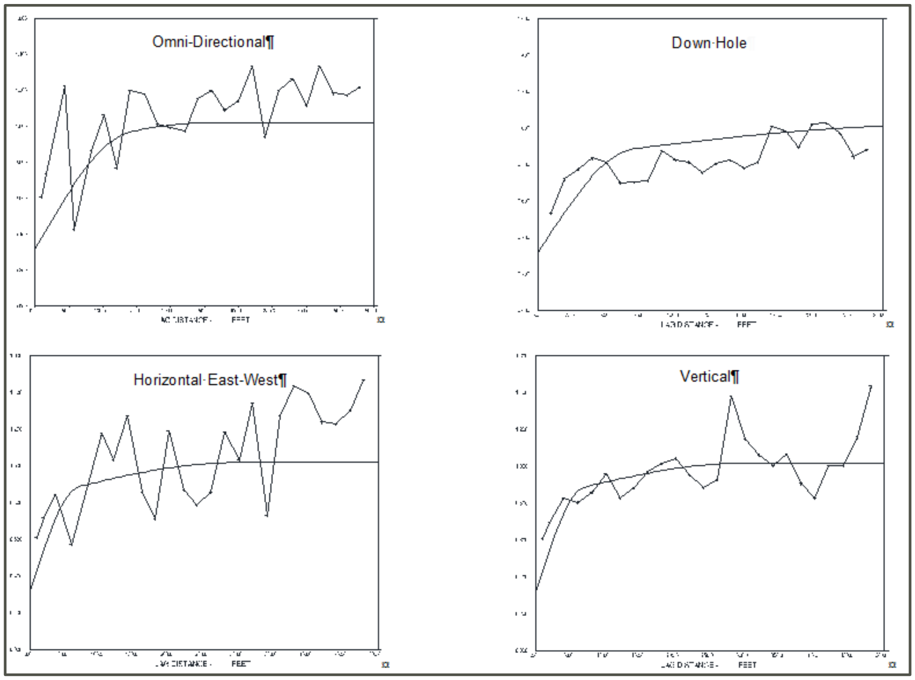


Figure 14-10: E2 Model Log-Normal Variography for Cu (2017 Technical Report)

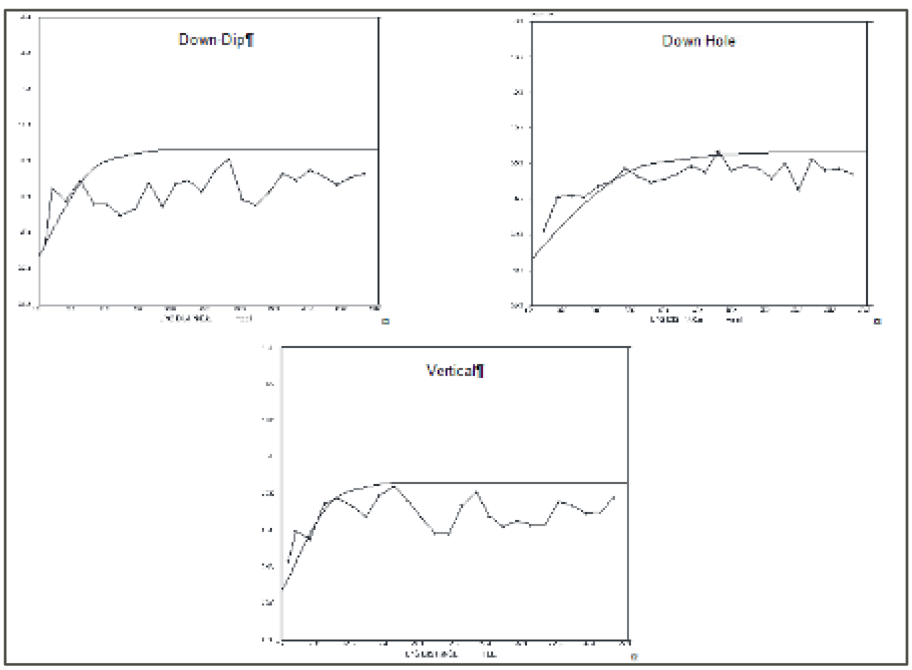
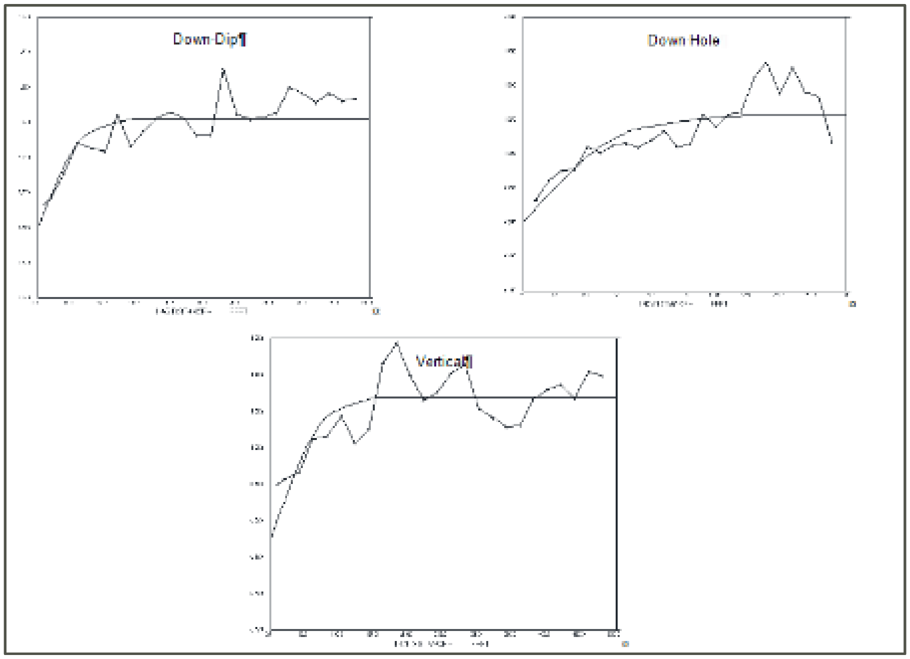


Figure 14-11: E2 Model Log-Normal Variography for Au (2017 Technical Report)



Search orientation was adjusted based on the best fit local orientation of the deposit. Blocks were not confined to sourcing composites from other search regions but were confined to mineral zone domain coded with 1000, 2000, 3000, 4000 and 5000.

14.1.6 Estimation Methods & Parameters

Resources were estimated with MicroMine™ from 10 ft composites with multiple pass ordinary kriging. Block model setup parameters for the East and E2 block models are detailed in Table 14-4.

Table 14-4: East and E2 Block Model Setup Parameters

Model	Origin X (Corner)	Origin Y (Corner)	Origin Z (Corner)	Blocks X	Blocks Y
East	368,162.5	1,521,722.5	1,190	134	196
E2	368,887.5	1,520,097.5	1,190	98	66

Model	Blocks Z	Block Size X	Block Size Y	Block Size Z	Min Sub-size X	Min Sub-size Y	Min Sub-size Z
East	372	25	25	10	25	25	10
E2	372	25	25	10	5	5	5

Block and composite rock codes are the basis of the estimation domains. Each block was assigned a lithologic code and a mineral zone code.

Block copper, gold, silver, molybdenum and sulfur grades were estimated in multiple passes separately within the following domains:

- Each of copper grade shell domains 1000, 2000, 3000, 4000 and 5000 with any lithologic code

- Each of the lithologic codes outside the mineral zone: 40, 50, 60, 70, 80, 90, 100

Given that the mineralization is skarn related and gradational, blocks and composites within each grade shells were treated as a continuous copper domain regardless of lithologic designation.

East and E2 model copper grades along with gold, silver, molybdenum and sulfur were estimated with the parameters shown in Table 14-5 and Table 14-6. The last pass was sized to fill in the interpretation, assuming that blocks within the interpretation are of sufficient confidence to warrant estimation. Pass parameters in the East and E2 models go from smallest (highest confidence) to largest, with each subsequent pass only estimating blocks remaining as not estimated from the previous pass.

Table 14-5: Pass Parameters and Search Criteria East Model

Pass	Domain Code	Max Search 1st	Anisotropy 1:2:3	Points (Max)	Points Per/DH Max	Points Min	Max DH	Min DH	Preliminary Classification	Classification Upgrade Conditions
9	5000	250	1:0.75:0.3	20	3	2	7	2	Indicated	Measured if: DH>=3 and average distance <=130, and closest distance <=125 and KE<=0.55
2	5000	500	1:0.75:0.5	20	2	1	7	1	Inferred	Indicated if: DH>=2 and closet distances <=275 and KE <=0.75
1 (IDW²)	40-100	500	1:0.75:0.25	-	2	1	-	1	Inferred	Indicated if: within dilution bounding shape and closest distance <=100 and DH>=3

Table 14-6: Pass Parameters and Search Criteria E2 Model

Pass	Domain Codes	Max Search 1st	Anisotropy 1:2:3	Points (Max)	Points Per/DH Max	Points Min	Max DH	Min DH	Preliminary Classification	Classification Upgrade Conditions
1	2000	250	1:0.75:0.1	20	3	2	7	2	Indicated	Measured if: DH ≥ 3 and average distance ≤ 140, and closest distances ≤ 90 and KE ≤ 0.95
2	2000	500	1:0.75:0.6	20	2	1	7	1	Inferred	Indicated if: DH ≥ 2 and closest distances ≤ 150 and average distance ≤ 300 and Northing ≤ 1,521,500
3	2000	1000	1:0.75:0.75	20	2	1	7	1	Inferred	NA
1	1000	250	1:0.75:0.1	20	3	2	7	2	Indicated	Measured if: DH ≥ 3 and average distance ≤ 140, and closest distances ≤ 90 and KE ≤ 0.95
2	1000	500	1:0.75:0.6	20	2	1	7	1	Inferred	Indicated if: DH ≥ 2 and closest distances ≤ 150 and average distance ≤ 300 and Northing ≤1,521,500
3	1000	1000	1:0.75:0.75	20	2	1	7	1	Inferred	NA
1	3000	250	1:0.75:0.1	20	3	2	7	2	Indicated	Measured if: DH ≥ 3 and average distance ≤ 140, and closest distances ≤ 90 and KE ≤ 0.95
2	3000	500	1:0.75:0.6	20	2	1	7	1	Inferred	Indicated if: DH ≥ 2 and closest distances ≤ 150 and average distance ≤ 300 and Northing ≤1,521,500
3	3000	1000	1:0.75:0.75	20	2	1	7	1	Inferred	NA
1	4000	250	1:0.75:0.1	20	3	2	7	2	Indicated	Measured if: DH ≥ 3 and average distance ≤ 140, and closest distances ≤ 90 and KE ≤ 0.95
2	4000	1500	1:0.75:0.6	20	2	1	7	1	Inferred	NA
1 (IDW2)	40-100	500	1:0.75:0.25	-	2	1	-	1	Inferred	Indicated if: Within dilution bounding shape and closest distance ≤ 100 and DH ≥ 3

14.1.7 Resource Classification

Each estimated block has been assigned measured, indicated or inferred classification for its contained mineral resources. The CIM 2014 defines mineral resources as:

Mineral Resource

- A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.
- The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Inferred Mineral Resource

- An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.
- An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Mineral Resource:

- An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.
- Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Measured Mineral Resource:

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

- Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

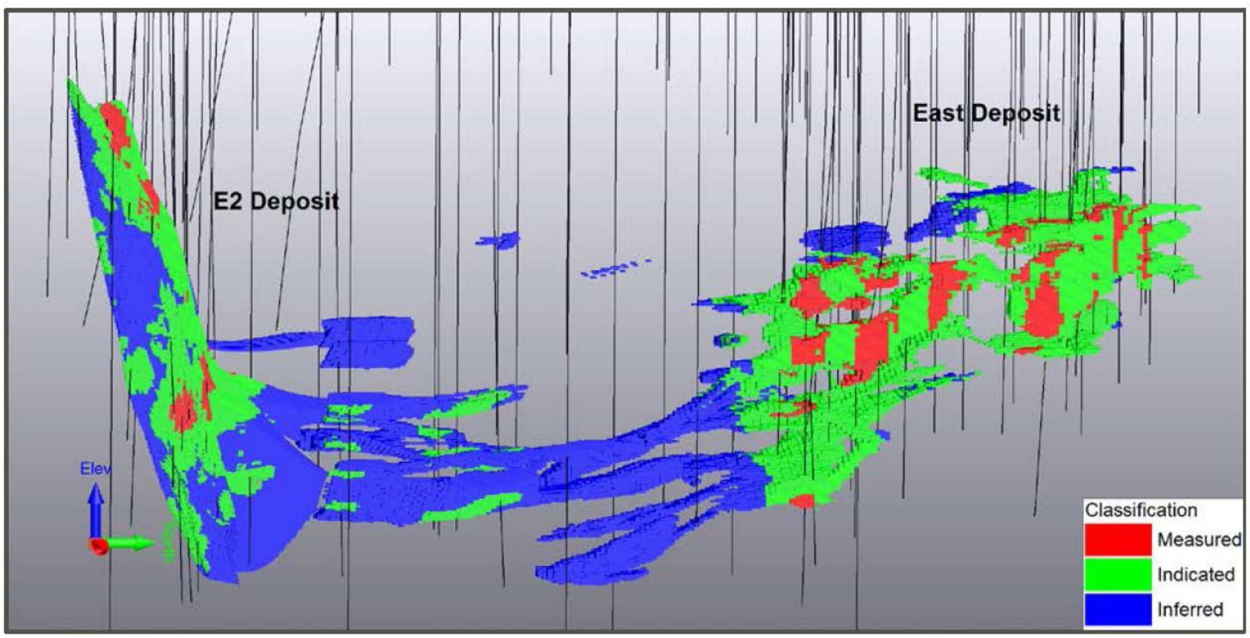
Modifying Factors:

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

Classification was accomplished by a combination of nested passes with increasing search ranges, composite sample selection criteria, and kriging error assessment. Table 14-5 and Table 14-6, above, show the pass criteria and resulting classification.

Figure 14-12 shows the resulting block classifications within the grade shells regardless of Cu% grade, in a 3D "melon slice" section.

Figure 14-12: Block Model Classification 3D Section Slice (2017 Technical Report)



In the East and E2 model, iron was not independently classified.

14.1.8 Density Determination

Density is assigned based on rock type, with the exception of blocks designated as silicate and magnetite skarn, which are regressed based on estimated iron grade. Tonnage factors were assigned to each block according to Figure 14-13. For blocks assigned silicate and magnetite skarn rock type and that met minimum iron grade, thresholds were determined to have a tonnage factor given the regression formulas shown in Figure 14-14.

14.1.9 Dilution

Block dilution has not been included in the mineral resource statement and is accounted for in the subsequent section regarding mining and reserves. Blocks have been coded as inside the grade shell or outside if the block centroid is located inside or outside of the shell. Grades have not been partial block factored for purposes of resource estimation.

14.1.10 Cutoff Grade & Reasonable Prospects for Economic Extraction

Cutoff grade has been approximated at 0.75% Cu prior to completion of cost optimization and has been applied for purposes of resource statement only. Detailed cutoff grade assumptions are provided in subsequent sections of this Report. To further approximate reasonable extraction, reporting of resources has been constrained to within the mineralized domains, with the exception of waste amongst the mineral zone interpretation that was confined with a waste solid for purposes of internal waste inclusion for the mine plan. Constrained resources are not reserves and do not have demonstrated economic viability.

14.1.11 Resource Statement for Eastern Area Deposits

The grade tonnage relationship for Measured and Indicated Resources of the Eastern Area is shown in Table 14-7, and the relationship for Eastern Area Inferred Resources is shown in Table 14-8 and Table 14-9.

Table 14-7: Mineral Resource Eastern Underground Area (E and E2)

Category	Cutoff Grade %Cu	Tons (millions)	Grade %Cu	Contained Cu lb (millions)	Grade Au oz/st	Contained Au ozs (thousands)	Grade Ag oz/st	Contained Ag ozs (thousands)	Grade %Fe	Contained Fe Tons (millions)
Measured	0.75	12.1	1.60	389	0.006	74	0.127	1,541	18.7	2.3
Indicated	0.75	41.9	1.33	1,114	0.005	217	0.112	4,716	17.6	7.4
Measured + Indicated	0.75	54.1	1.39	1,503	0.005	291	0.116	6,257	17.8	9.6
Inferred	0.75	29.2	1.09	636	0.003	87	0.064	1,875	12.8	3.7

Notes:

Includes East and E2 deposits.
Measured and Indicated Resources are stated as inclusive of Reserves.
Columns may not total due to rounding.
Effective date on Underground Mineral Resource is April 15, 2015.

Table 14-8: Mineral Resource Underground East Deposit (Excludes E2)

Category	Cutoff Grade %Cu	Tons (million)	Grade %Cu	Contained Cu lb (millions)	Grade Au oz/st	Contained Au ozs (thousands)	Grade Ag oz/st	Contained Ag ozs (thousands)	Grade %Fe	Contained Fe Tons (millions)
Measured	0.75	10.9	1.57	342	0.005	59	0.114	1,242	18.3	2.0
Indicated	0.75	33.6	1.31	880	0.005	157	0.099	3,321	17.3	5.8
Measured + Indicated	0.75	44.5	1.37	1,222	0.005	217	0.102	4,563	17.6	7.8
Inferred	0.75	21.9	1.10	484	0.003	63	0.052	1,131	12.9	2.8

Notes:

Measured and Indicated Resources are stated as inclusive of Reserves.
Columns may not total due to rounding.
Effective date on Underground Mineral Resource is April 15, 2015.

The reader is cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-9: Mineral Resource Underground E2 Deposit (Excludes E)

Category	Cutoff Grade %Cu	Tons (million)	Grade %Cu	Contained Cu lb (million)	Grade Au oz/st	Contained Au ozs (thousand)	Grade Ag oz/st	Contained Ag ozs (thousand)	Grade %Fe	Contained Fe Tons (million)
Measured	0.75	1.2	1.89	47	0.012	14	0.242	299	22.6	0.3
Indicated	0.75	8.3	1.40	234	0.007	60	0.167	1,395	18.6	1.5
Measured + Indicated	0.75	9.6	1.47	281	0.008	74	0.177	1,694	19.1	1.8
Inferred	0.75	7.2	1.05	152	0.003	23	0.103	744	12.5	0.9

Notes:

Measured and Indicated Resources are stated as inclusive of Reserves.
Columns may not total due to rounding.
Effective date on Underground Mineral Resource is April 15, 2015.

Tetra Tech is unaware of any known environmental, permitting, legal, title, taxation, mining, metallurgical, infrastructure, socio-economic, marketing and political factors other than those discussed that could materially affect the Underground Mineral Resource. The Underground Project is fully located on privately owned or leased lands and there are no known legal or title issues affecting the Property. The Underground Project has all material permits and Nevada Copper is not aware of any known socio-economic factors that could impact the Underground Project.

Increases in items such as mining cost, processing cost and selling cost or a decrease in the copper price would result in decrease to the Underground Mineral Resources. An increase in the copper price, or decreases in items such as mining cost, processing cost and selling cost would result in an increase to the Underground Mineral Resources.

Figure 14-13: Grade Tonnage Curve East Area – Measured and Indicated Resources (2017 Technical Report)

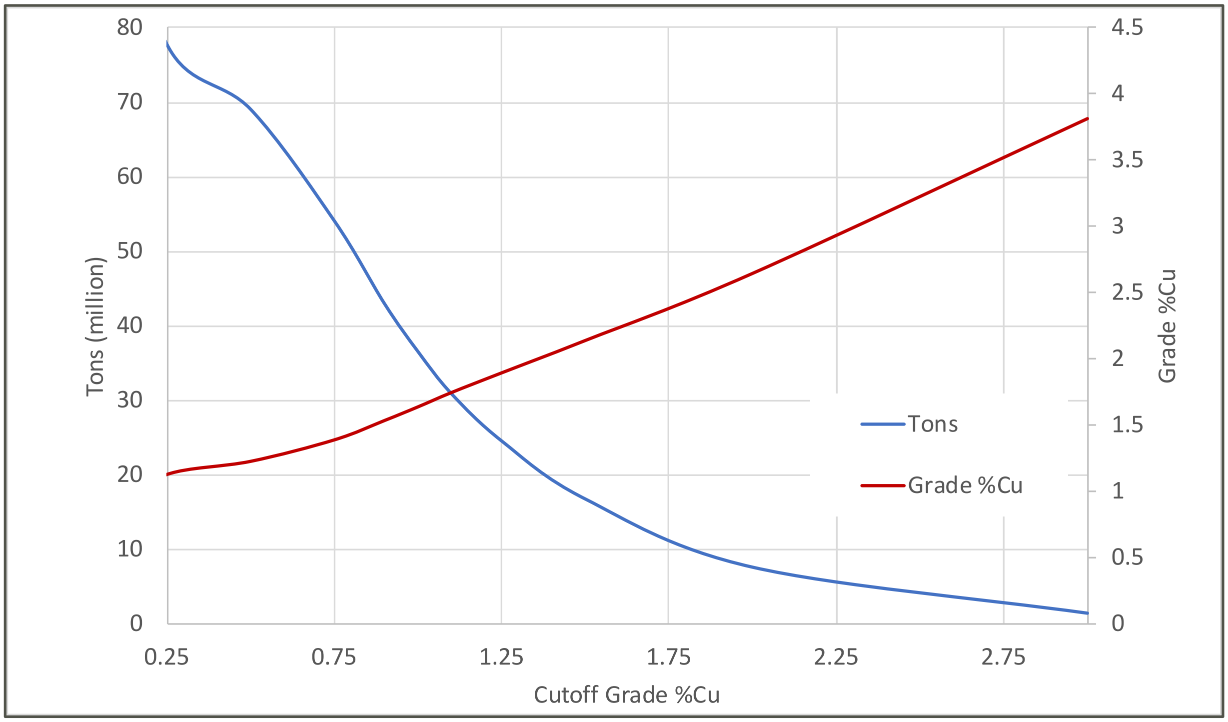
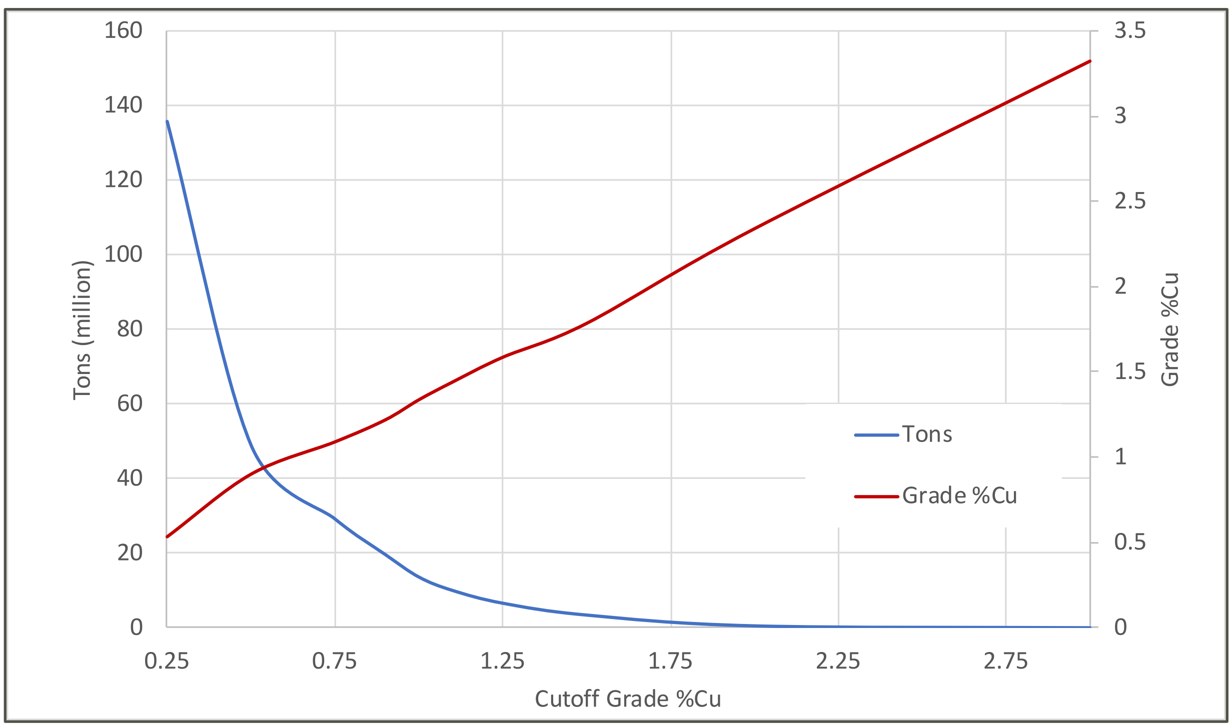


Figure 14-14: Grade Tonnage Curve East Area – Inferred Resources (2017 Technical Report)



14.1.12 Verification

Estimations were verified through statistical and visual review. Statistical review was completed with analysis of copper grade populations for assays, composites and blocks within the mineralized shell, block search ranges by classification, as well as copper grade swath plots. Figure 14-15 shows the population progression of copper for the East model and Figure 14-16 for the E2 model, demonstrating the moderation of grade from assays to composites and to blocks.

Figure 14-15: Histograms of Assays, Composites & Blocks Cu% Grade East Deposit (2017 Technical Report)

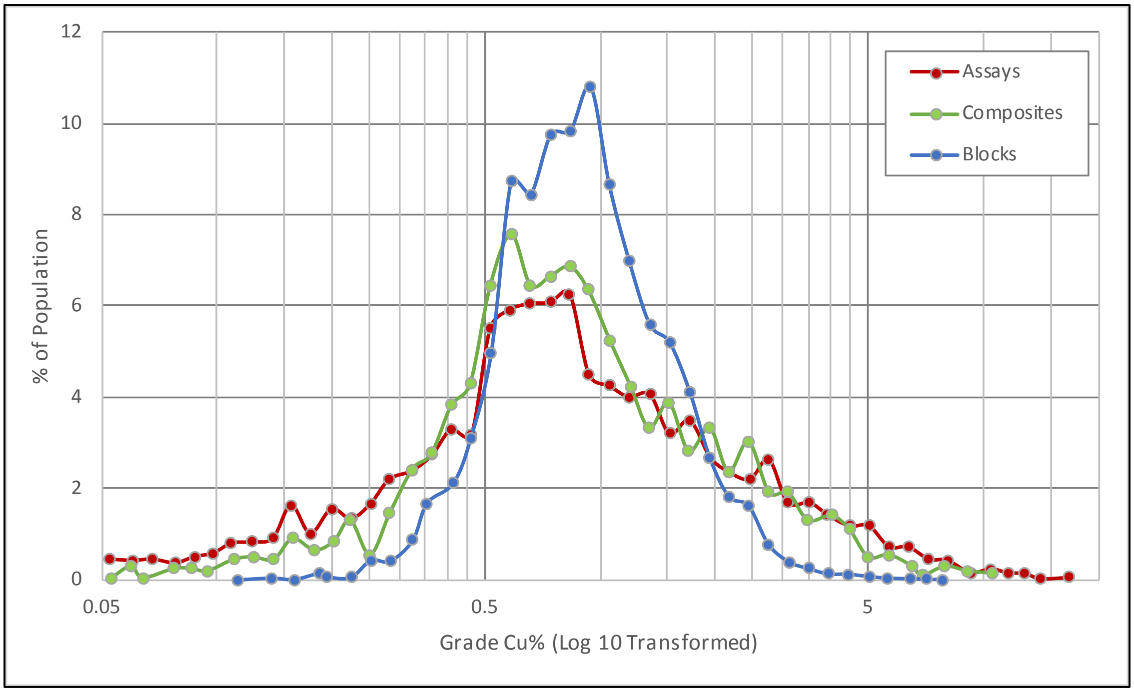
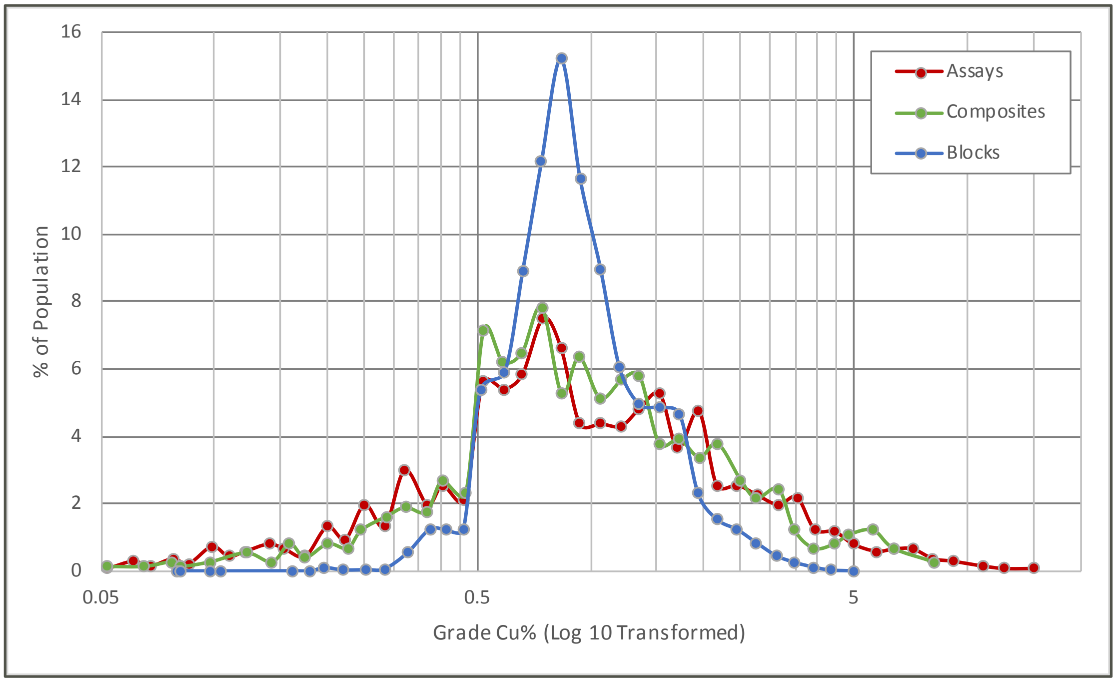


Figure 14-16: Histograms of Assays, Composites & Blocks Cu% Grade E2 Deposit (2017 Technical Report)



Classified blocks above the 0.75% Cu cutoff grade within the grade shells were reviewed for their proximity to the nearest composite sample in Figure 14-17 and Figure 14-18 block populations for the East and E2 Deposits grouped as Measured, Indicated and Inferred Resources.

Figure 14-17: Nearest Composite Sample to Blocks East Model (2017 Technical Report)

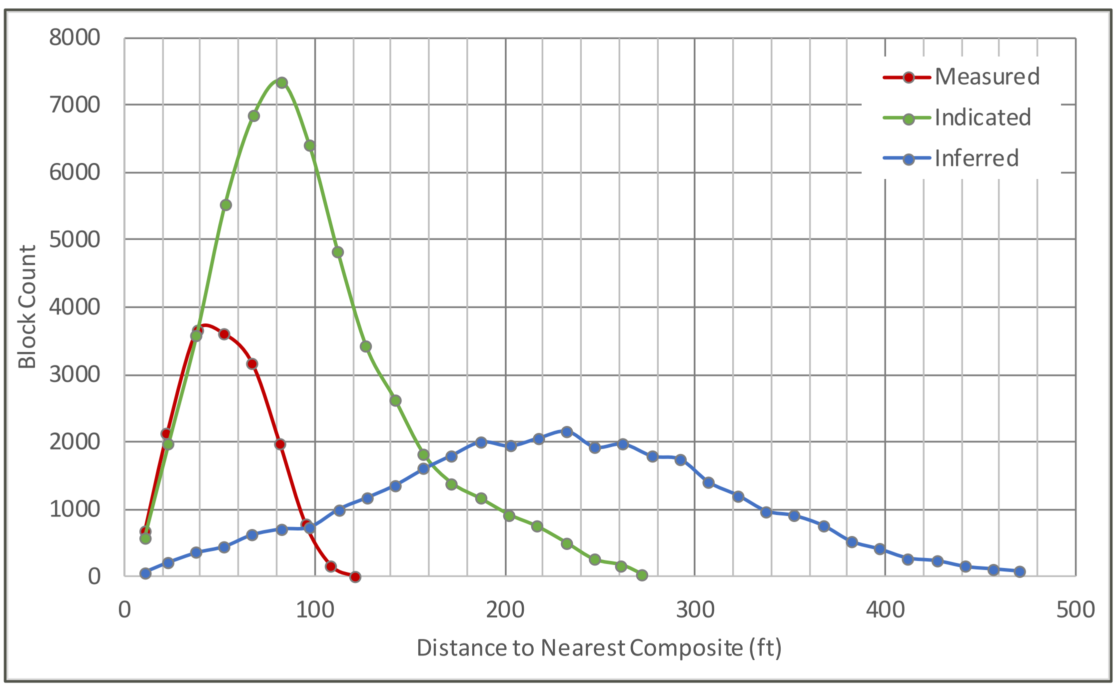


Figure 14-18: Nearest Composite Sample to Blocks E2 Model (2017 Technical Report)

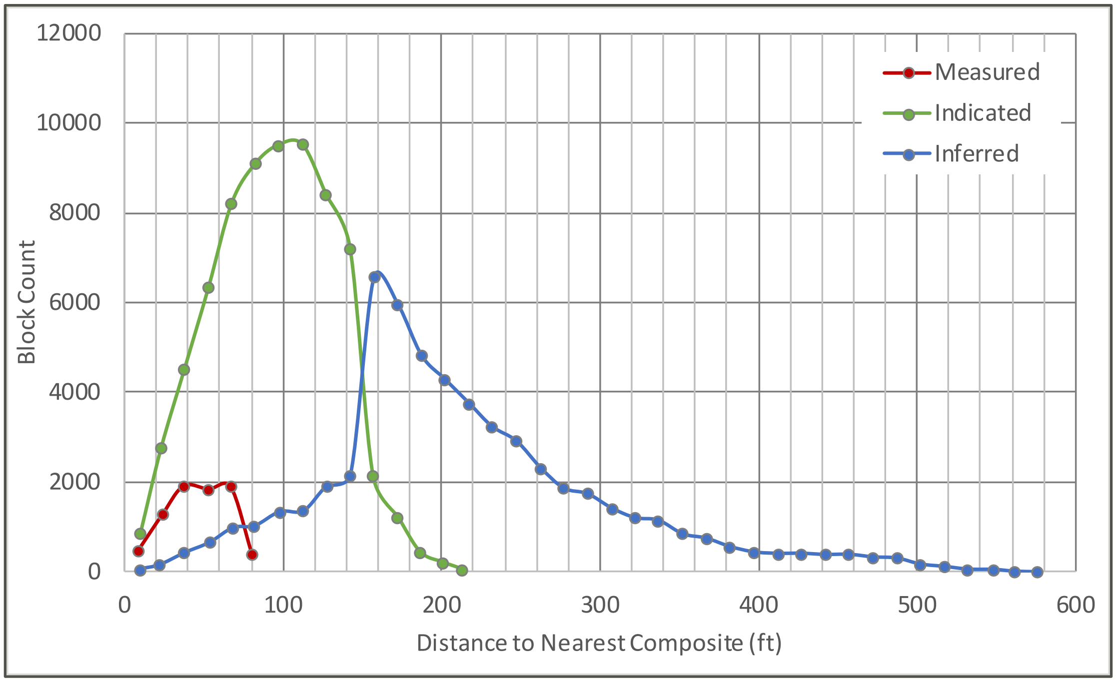
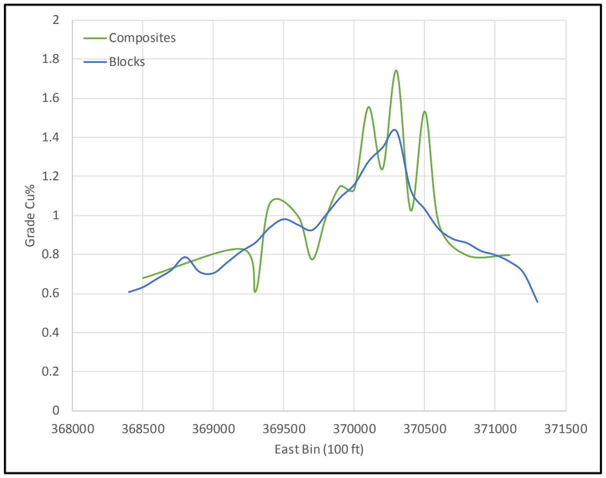


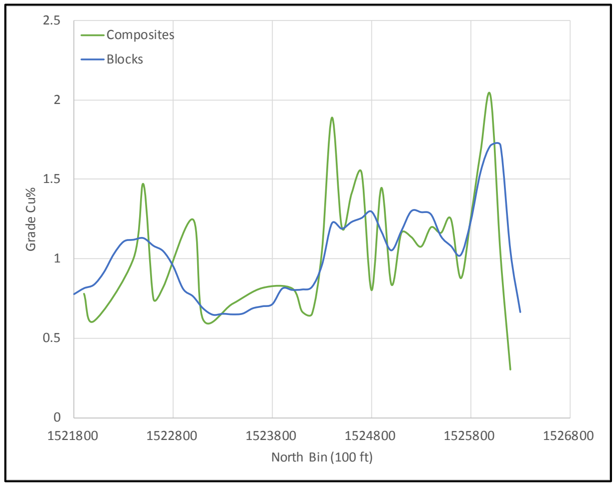
Figure 14-19 shows swath plots of the East and E2 block model and composites and demonstrates that the average grade of the blocks closely resembles the composite average grades.

The E2 swath plots suggests grade is being moderated more than in the East or West model. Investigations concluded the oblique drilling orientation through the highest grade portion of the deposit was responsible for the moderation of grade. The main zone of the E2 Deposit exhibits the highest grades halfway between the hanging wall and footwall of the zone with a lower grade bias at the boundary. The drilling is primarily vertical and often intersects the mineralization near the dip angle. Because of this, blocks between holes are influenced more strongly by the first and last composite through the zone, which are often lower grade. Resource upside could exist in improving grade modeling by implementing hanging wall and footwall controls or estimating in space relative to the mineralization centerline.

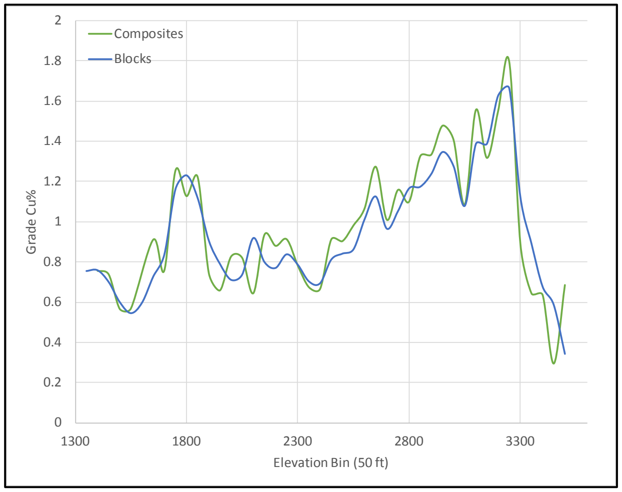
Figure 14-19: Swath Plots East and E2 Models Cu% (2017 Technical Report)



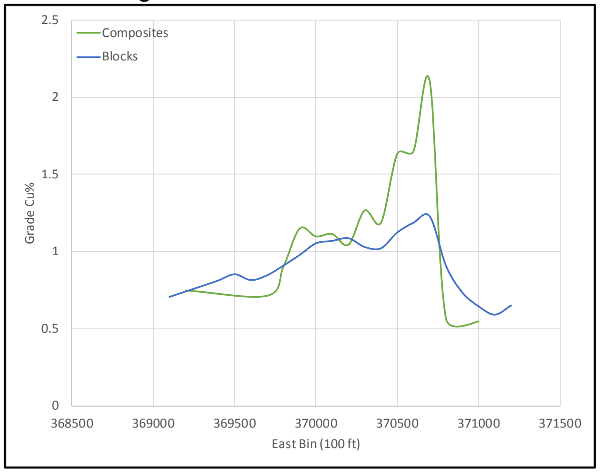
East Easting



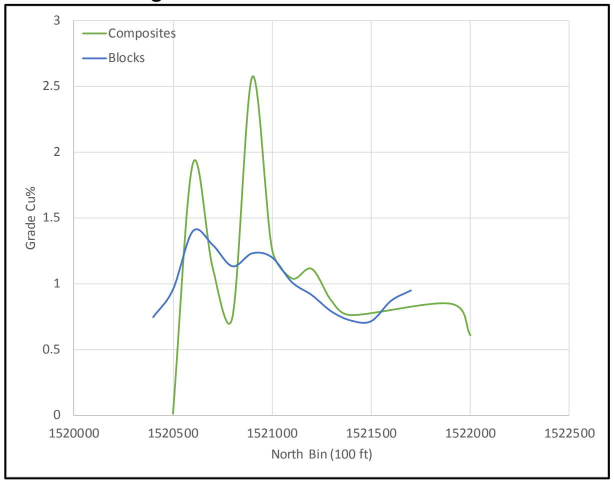
East Northing



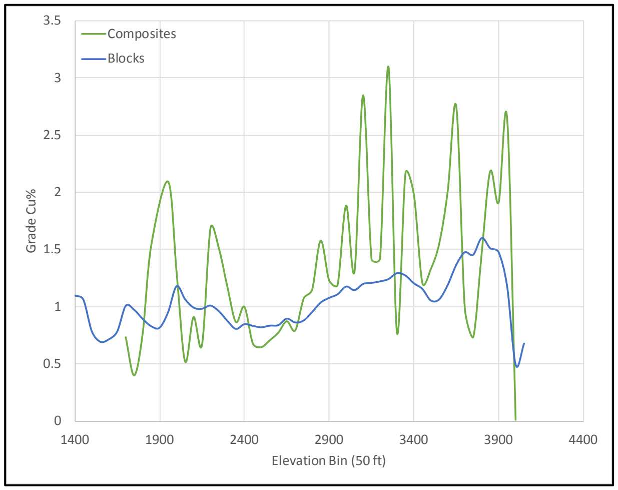
East Elevation



E2 Easting



E2 Northing



E2 Elevation

Visual inspection of the model was completed digitally on screen in 3D and in cross section. Large format paper cross sections, level plans and supporting digital data were provided to Nevada Copper's site geologist for inspection. Figure 14-20 and Figure 14-21 show example cross sections of the East and E2 deposits and the estimated block grades.

Figure 14-20: East Deposit – Cross Section 1524510 (Block Model) (2017 Technical Report)

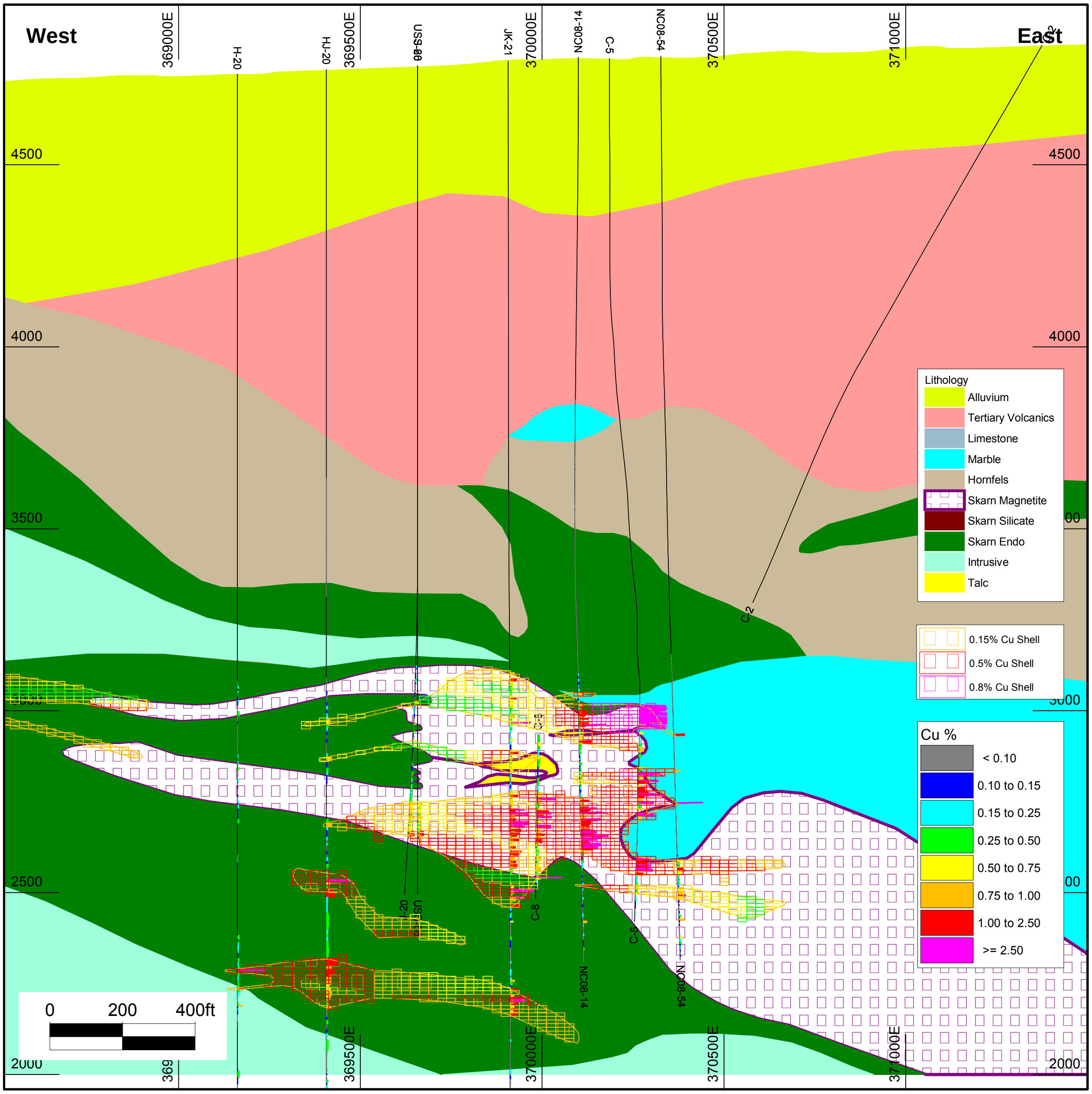
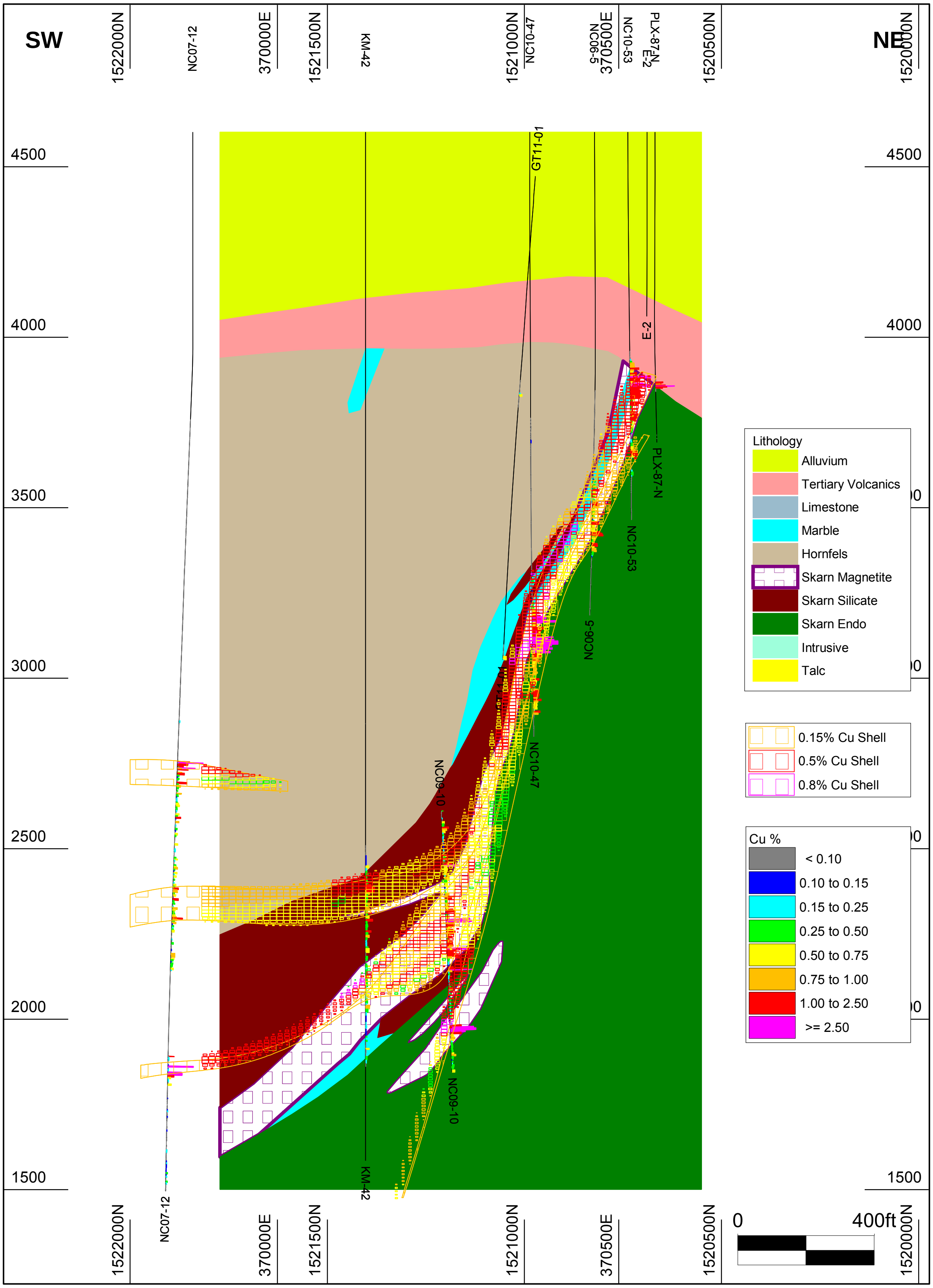


Figure 14-21: E2 Deposit – Cross Section 008 (Block Model) (2017 Technical Report)



14.1.13 Relevant Factors

The above Item describes the approaches used to estimate Underground Mineral Resources given current data availability. There are minor intercepts where sampling or recovery factors could impact the accuracy and reliability of the drill results in isolated areas. These intercepts are not anticipated to materially affect the mineral resource. As development drilling is conducted and data analyzed, modeling methods may be altered to better represent the mineralization.

14.2 Open Pit (Western Area) Estimation

14.2.1 Geological Model

The geological model of the Open Pit Project considers two main models:

- Lithology
- Grade shells

The lithology model was developed from the drill hole data using Leapfrog geological modeling software.

The Nevada Copper drill hole database included 826 drill holes. A total of 616 drill holes were used for the Open Pit resource model.

The Property lithology plays an important role in the geologic model to define density, particularly in areas with high iron content. However, mineralization in the Pumpkin Hollow Project mineral deposit is not controlled by lithology. The drill hole data show that the Deposit has two distinct populations: a high-grade core surrounded by lower grade material. Golder constructed 0.15% and 0.80% total copper shells to model these populations. As the orebody is highly irregular especially in the north area, seven trends were used to guide the grade shells. The grade shells are designed to isolate the high-grade samples to be used for interpolation of the high grade zones and the low grade samples to be used to in the low grade zones. The 0.15% total copper boundary is designed to isolate samples outside the mineralized zone from influencing the blocks within the ore zone.

14.2.1.1 Backflagging

Backflagging is a method of assessing the effectiveness of how the grade shells segregated the high grade and low-grade samples. Approximately 82% of the samples backflagged within the 0.8% total copper shell had grades above 0.8% total copper. Approximately 87% of the samples in the 0.15% total copper grade shell were in the 0.15 to 0.8% total copper range and approximately 22% of the samples were either above 0.8% total copper or below 0.15%. The backflagging results show a high level of consistency between the composited total copper grades and the interpreted grade shells.

14.2.1.2 Lithology Model

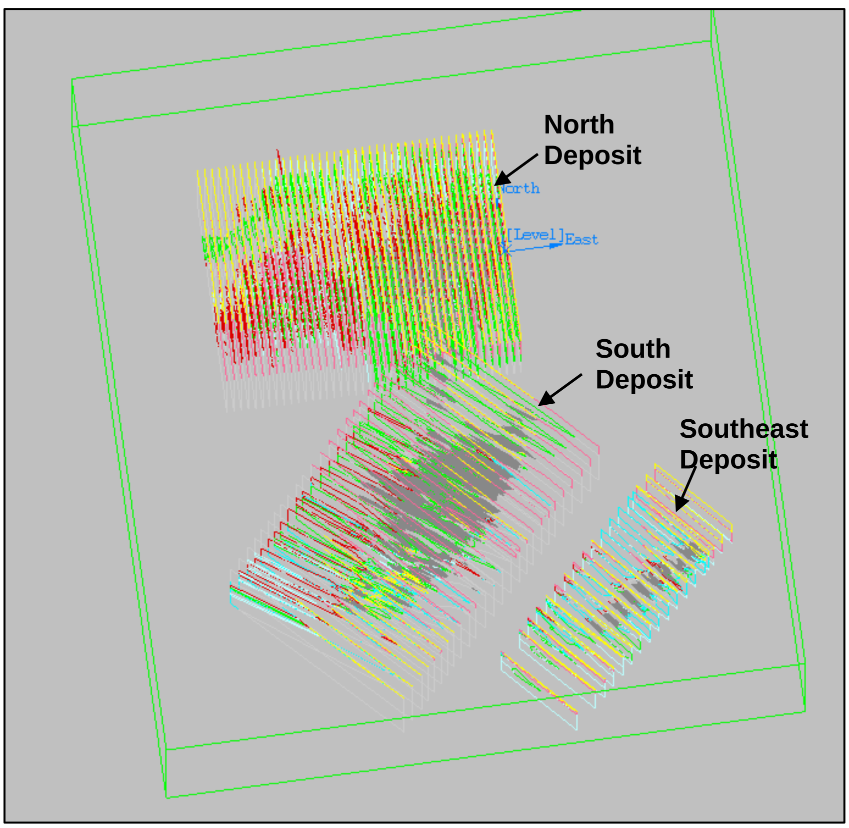
The Pumpkin Hollow Property geological model includes a lithology variable that is coded using both computerized methods and input from the geologist. The 2015 lithology model has 10 lithological units

(Table 14-10) and was created by Nevada Copper geologists by interpreting from cross sections that have an east–west orientation and a 100 m spacing in the North Deposit and cross sections with northwest–southeast orientation that are between 100 to 200 m apart in the South and Southeast deposits (Figure 14-22).

Table 14-10: 2015 Lithology Codes

Code	Description
10	Alluvium
20	Tertiary volcanic
30	Limestone
40	Marble/Limestone
50	Hornfels
60	Magnetic Skarn
70	Silicate Skarn
80	Endoskarn
90	Intrusive
100	Talc

Figure 14-22: 2015 Lithological Model Interpreted Sections (Golder, 2019)



For the 2018 lithology modeling, Leapfrog Geo 4.2.1 was used to construct the models for the North, South and South East Deposits. The Leapfrog software provides a 3D solid that better defines the transition between sections.

Model Preparation

For the construction of each of the models, the following was considered:

- Database tables in csv format: collar, survey, lithology, and assay.
- Database validation for each one of the variables.
- Display of information in Leapfrog Geo.
- Cross Sections interpretation of the previous models (2D).
- Topography.
- Block model.

Model Construction

The 2018 lithology model was built using implicit modeling for all geological units. Implicit modeling refers to the creation of wireframes defined by mathematical functions, geology and stratigraphic relationship. Creation of the final wireframe for a geological unit required several solid regenerations with the addition of polylines to further control the lithologic boundaries. The 2015 sectional interpretations provided input from geologists familiar with the Property and were used as required when adding polylines to further guide the Leapfrog software.

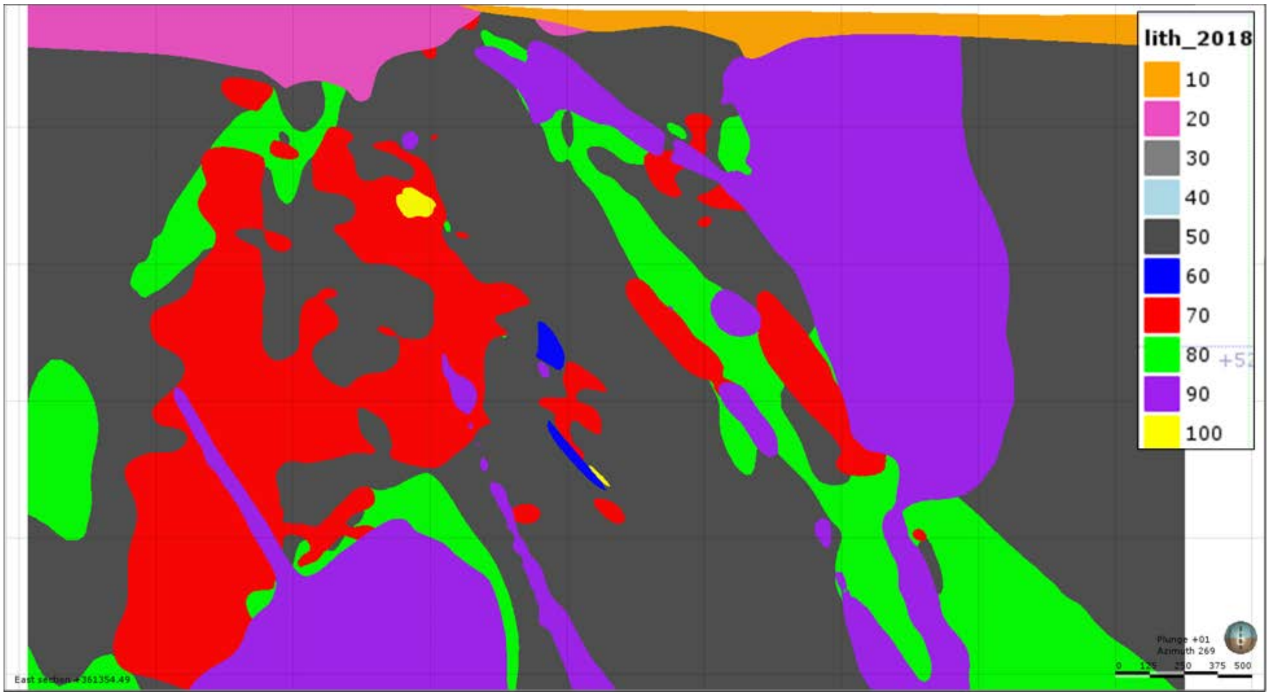
The lithological units present in the 2018 lithology model and their respective codes are presented in Table 14-11.

Figure 14-23 shows a typical section of the lithology model in the North Deposit.

Table 14-11: 2018 Lithological Units in Database

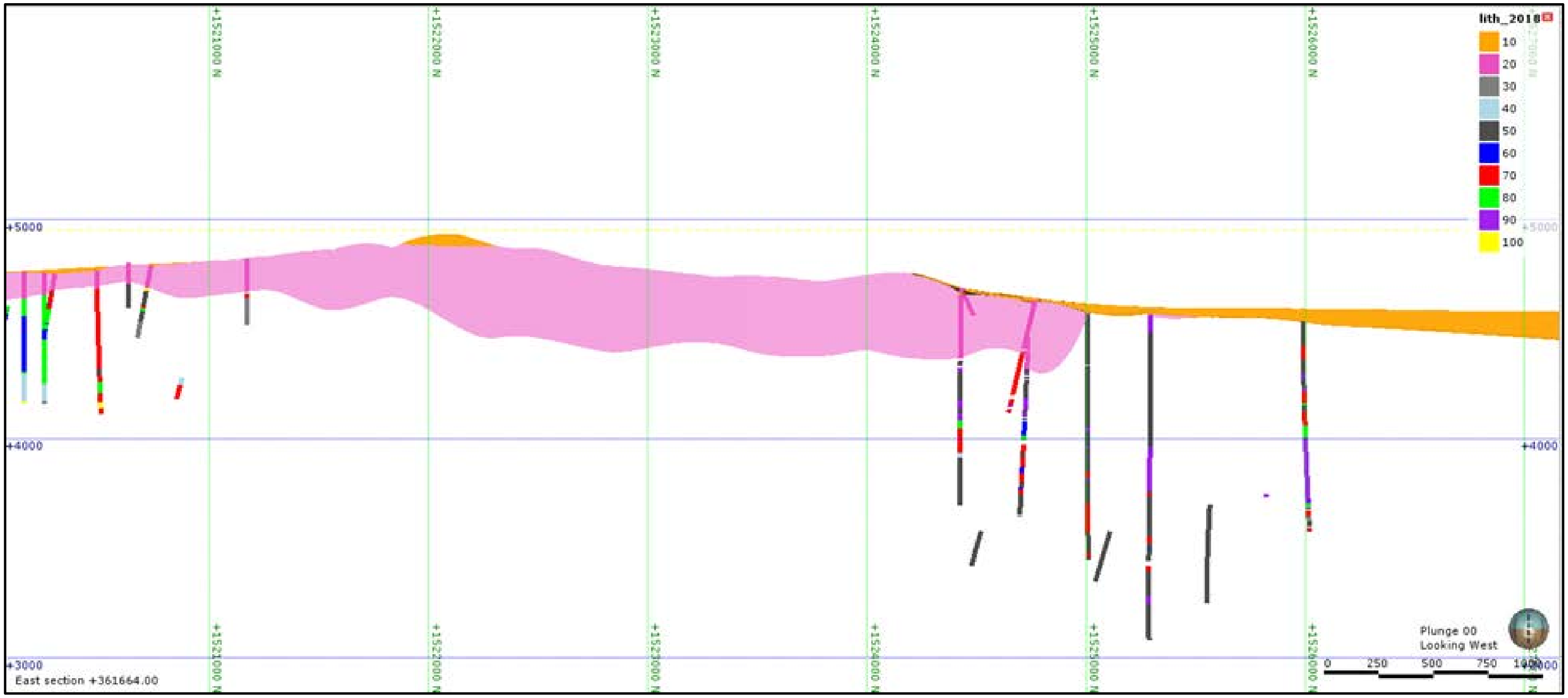
Codes	Description	Model Units	Feet	% Total
10	Alluvium	10	929.5	0.44%
20	Tertiary rocks	20	56,514.8	26.45%
30	Limestone	30	5,433.4	2.54%
40	Marble	40	10,934.3	5.12%
50	Hornfels	50	8,459.3	3.96%
60	Magnetic Skarn	60	48,250.9	22.58%
70	Silicate Skarn	70	23,539.4	11.02%
71	Fault	Ignore	1,698.8	0.80%
72	Clay zone		50.0	0.02%
73	Breccia		427.3	0.20%
74	Calcite vein		265.5	0.12%
75	Quartz vein		6.7	0.00%
80	Endoskarn	80	44,868.6	21.00%
90	Intrusive	90	9,232.3	4.32%
100	Talc	100	2,914.7	1.36%
152	Magnetite skarn breccia	60	120.0	0.06%

Figure 14-23: Section N361354 Showing the Lithology in North Deposit (Golder, 2019)



The Deposits are buried and have been cut off or covered by the post mineral alluvium (unit 10) and tertiary rocks (unit 20). Figure 14-24 shows host lithology and the sub horizontal geometry of the post mineral alluvium and tertiary rocks units in North, South and South East Deposits.

Figure 14-24: Section N361664 - Sub horizontal Character of Alluvium and Tertiary Rocks (Golder, 2019)



14.2.1.3 Lithology Validation

Statistical analysis was carried out to compare the correlation between the original data (drill holes logs) and the interpreted data (lithological model). In order validate the lithology model, drill holes composites were backflagged with the wireframes solid. Composites outside the block model's volume and without information (code – 99) were excluded from the analysis, which is necessary to understand the relationship of coincidence, between wireframes compared against the composited database (Table 14-12).

Table 14-12: Backflagging Example

Units	Values	Wireframe		
		1	2	3
1	feet	n/a	1,440	n/a
	% coincidence	n/a	6%	n/a
	% pureness or contamination2	n/a	4%	n/a
2	feet1	188	34,193	412
	% coincidence1	1%	98%	1%
	% pureness or contamination2	1%	96%	25%
3	feet	n/a	23	n/a
	% coincidence	n/a	100%	n/a
	% pureness or contamination2	n/a	0%	n/a

Note:

Coincidence: is the amount of feet of drilling inside a given lithologic wireframe coded to that same lithologic unit
Contamination or Pureness: the amount of feet of drilling inside a given lithologic wireframe coded to a different lithologic unit.

In general, results show good agreement between the lithology data and the solids created. No inconsistencies were observed that would result in a material impact on the estimation.

In the North Deposit, the results indicate a good agreement of the block model for the main units as seen in Table 14-13. The Talc unit had less data and was irregular, which resulted in lower representativeness than expected. The pureness percentages are acceptable for all the units (except Talc) that are near or above 90%, which means that the solids contain mainly composites of their own unit, and low contamination with other units is present.

Table 14-13: North Deposit Backflagging

Lithologic Unit	Values	10	20	50	60	70	80	90	100	Total
10	ft	25,266	722	240	10	0	51	240		26,528
	% coincidence	91%	5%	2%	-	-	2%	-	-	100%
	% pureness	100%	1%	-	-	-	-	1%	-	6%
20	ft	-	62,054	486	26	130	60	40	30	62,824
	% coincidence	-	96%	3%	-	1%	1%	-	-	100%
	% pureness	-	95%		-	-	-	-	2%	15%
30	ft	-	-	4,756	-	1,212	182	1,244	-	7,394
	% coincidence	-	-	51%	-	31%	5%	13%	-	100%
	% pureness	-	-	3%	-	1%	0%	3%	-	2%
40	ft	-	-	750	-	365	451	65	-	1,630
	% coincidence	-	-	48%	-	25%	23%	4%	-	100%
	% pureness	-	-	1%	-	-	1%	-	-	-
50	ft	45	683	119,294	30	1,174	507	621	17	122,371
	% coincidence	-	-	97%	-	1%	1%	1%	0%	100%
	% pureness	-	1%	87%	-	1%	1%	1%	1%	28%
60	ft	-	5	295	14,778	549	406	542	140	16,715
	% coincidence	-	-	2%	87%	4%	3%	2%	1%	100%
	% pureness	-	-		89%	1%	1%	1%	8%	4%
70	ft	-	11	2,452	779	80,225	516	369	79	84,430
	% coincidence	-	0%	3%	1%	94%	1%	1%	-	100%
	% pureness	-	0%	2%	5%	90%	1%	1%	4%	20%
80	ft	5	15	818	705	465	47,497	828	1	50,335
	% coincidence	-	0%	2%	1%	1%	93%	2%	-	100%
	% pureness	-	0%	1%	4%	1%	90%	2%	-	12%
90	ft	10	795	2,753	210	1,831	1,814	38,336	18	45,767
	% coincidence	-	-	8%	1%	6%	6%	79%	-	100%
	% pureness		1%	2%	1%	2%	3%	90%	1%	11%
100	ft	-	-	390	-	758	164	186	1,448	2,945
	% coincidence	-	-	13%	-	26%	7%	7%	47%	100%
	% pureness	-	-	-	-	1%	-	-	79%	1%
Ignore	ft	-	1,197	4,183	36	2,537	1,261	138	98	9,469
	% coincidence	0%	7%	44%	1%	30%	15%	2%	1%	100%
	% pureness	0%	2%	3%	0%	3%	2%	0%	5%	2%
Total	ft	25,346	65,481	136,416	16,573	89,246	52,909	42,607	1,830	430,407

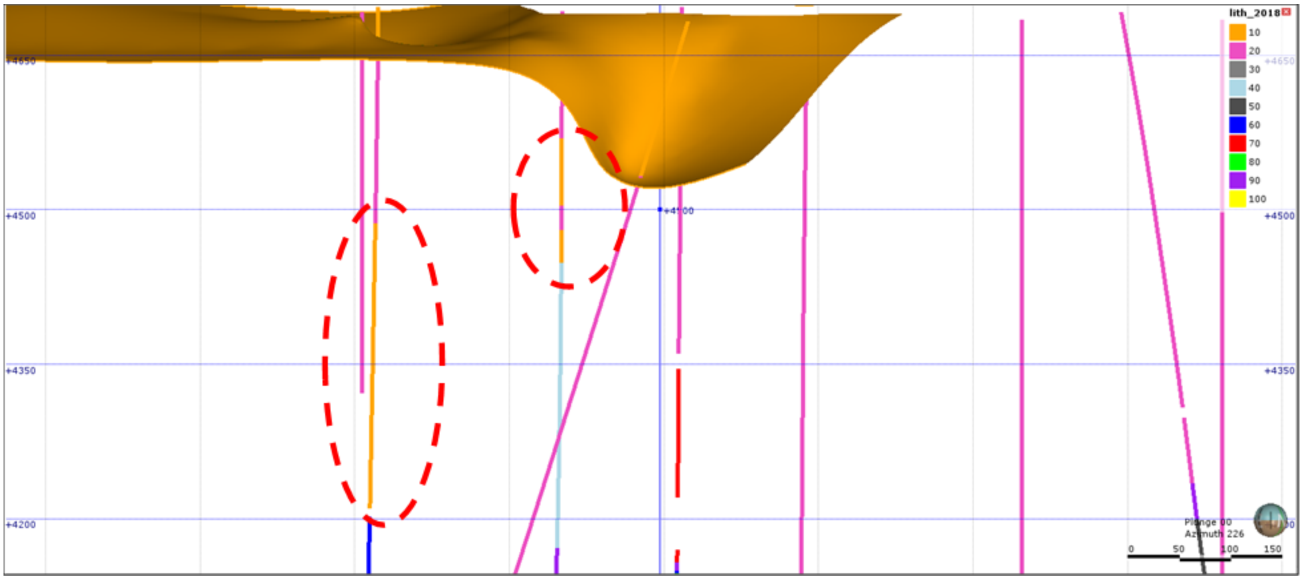
In the South Deposit, the results of the validation between the drill hole data and the model shows no obvious anomalies and acceptable consistency between the drill hole data and the geological modeling for the lithological wireframes as shown in Table 14-14. In general, the values of coincidence are near 90%, with the exceptions of units for Alluvium (10) and Intrusive (90).

Table 14-14: South and Southeast Deposits Backflagging

Lithologic Unit	Values	10	20	40	50	60	70	80	90	100	Total
10	ft	3,830	2,114	154.1	92	16.5	200	41	90	-	6,537
	% coincidence	53%	36%	4%	3%	2%	1%	2%	1%	-	100%
	% pureness	93%	4%	1%	1%	-	1%	-	1%	-	3%
20	ft	201.6	55,254	305	94	10	112.5	58	267	-	56,302
	% coincidence	-	95%	1%	1%	-	1%	-	1%	-	100%
	% pureness	5%	92%	2%	1%	-	-	-	2%	-	23%
30	ft	80	555	4,498	2,039	35.1	2,451	10	22	-	9,691
	% coincidence	-	1%	56%	12%	1%	29%	-	1%	-	100%
	% pureness	2%	1%	28%	17%	-	8%	-	-	-	4%
40	ft	-	6	10,513	5	593	30	113	13	32	11,305
	% coincidence	-	-	88%	-	9%	1%	2%	-	1%	100%
	% pureness	-	-	65%	-	1%	-	-	-	1%	5%
50	ft	-	459	55	8,623	39	63	132	12	4	9,387
	% coincidence	-	2%	1%	93%	1%	1%	2%	-	-	100%
	% pureness	-	1%	-	73%	-	-	-	-	-	4%
60	ft	-	5	219	69	50,411	203	639	107	61	51,712
	% coincidence	-	-	1%	-	96%	1%	2%	-	-	100%
	% pureness	-	-	1%	1%	94%	1%	1%	1%	2%	21%
70	ft	-	24	30	402	222	25,405	472	85	28	26,667
	% coincidence	-	-	-	2%	1%	94%	2%	-	-	100%
	% pureness	-	-	-	3%	-	86%	1%	1%	1%	11%
80	ft	-	-	201	158	926	468	45,659	290	44	47,745
	% coincidence	-	-	1%	-	2%	1%	94%	1%	-	100%
	% pureness	-	-	1%	1%	2%	2%	94%	2%	1%	20%
90	ft	-	1,135	53	47	475	169	856	14,361	12	17,107
	% coincidence	-	2%	1%	1%	6%	3%	13%	74%	-	100%
	% pureness	-	2%	-	-	1%	1%	2%	92%	-	7%
100	ft	-	-	45	7	146	20	183	36	2,750	3,187
	% coincidence	-	-	2%	-	5%	1%	8%	2%	83%	100%
	% pureness	-	-	-	-	-	-	-	-	93%	1%
Ignore	ft	-	321	217	266	921	446	446	307	27	2,950
	% coincidence	-	6%	7%	9%	36%	14%	18%	9%	1%	100%
	% pureness	-	1%	1%	2%	2%	2%	1%	2%	1%	1%
Total	ft	4,111	59,873	16,290	11,801	53,794	29,568	48,607	15,589	2,957	242,590
Total % coincidence	% coincidence	0%	2%	4%	5%	28%	14%	37%	7%	3%	100%

In the case of Alluvium (lithologic unit 10)the correspondence is 53%, which is due to some composites of alluvium that are logged very deep and away of the bottom surface of the unit as can be seen in Figure 14-25, these composites were not included in the Alluvium unit in order to maintain continuity and respect the 2015 geological interpretation. The Intrusive units have a correspondence of 74%, 18% of the composites are inside the Endoskarn unit and mainly have poor continuity and length.

Figure 14-25: Example of Alluvium Logged at Depth (Golder, 2019)



Note: Section N1523134 shows an example of drill holes with alluvium, which is found in an area typically logged as a Tertiary unit.
Red dashed lines indicate 2015 geological interpretation of alluvium.
Golder reviewed and found that it did not materially affect the 2018 geological interpretation.

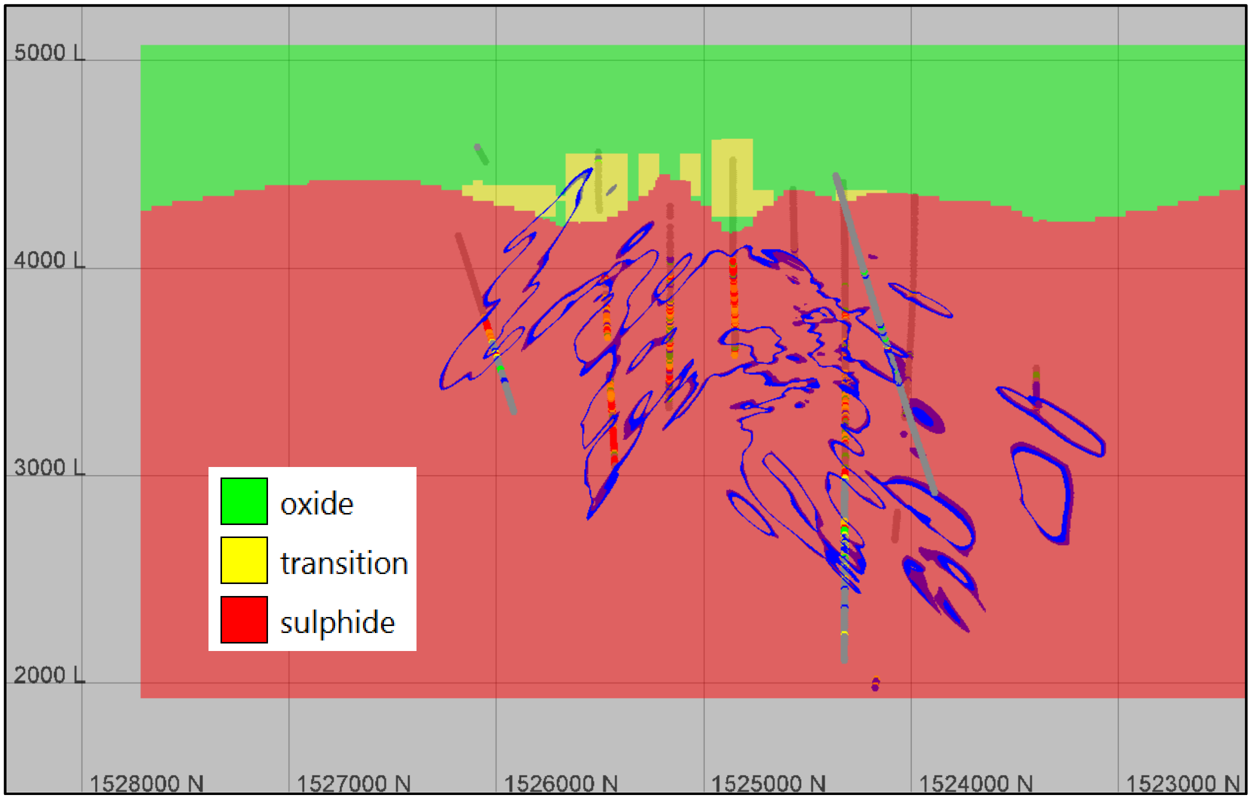
14.2.1.4 Oxide/Sulfide Model

The Oxide/Sulfide model was not updated and accordingly the 2015 Oxide/Sulfide model was used in the 2018 estimation. Table 14-15 shows a description of each unit code. Golder compared the surface to the drill holes and found that the surface well represented the drill holes.

Table 14-15: Description of Oxide/Sulfide Codes

Code	Description
1	Oxide
2	Transition
3	Sulfide

Figure 14-26: Cross Section Showing the Oxide/Sulfide Model, 0.15% Cu Grade Shell in Blue (Golder, 2019)



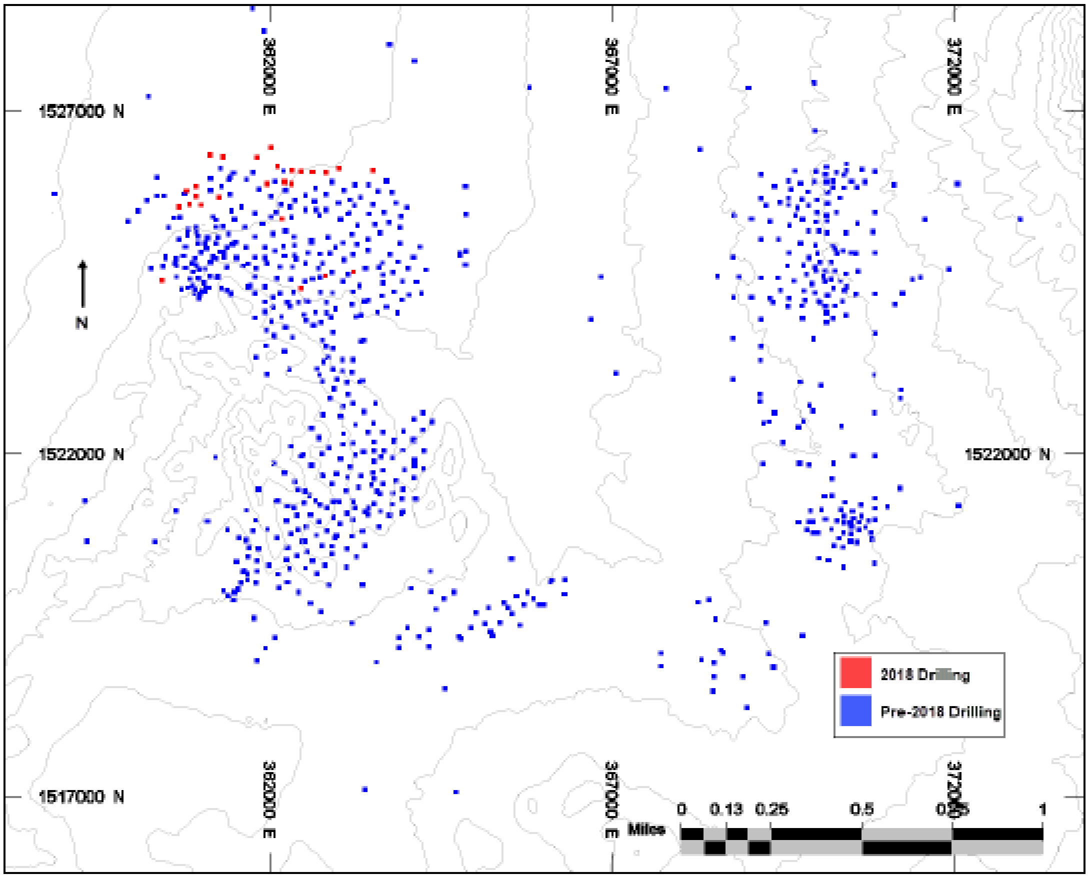
Due to metallurgical and process considerations, material lying in the oxide and transition zones was defined as waste material. Consequently, blocks with an oxidation state value of 1 or 2 were set to a default value even if they are within a grade shell envelope. In the same way, samples from zones 1 or 2 were not used to estimate blocks in the sulfide zone.

Database Description

A total of 616 holes are within the Open Pit Mineral Resource model.

Figure 14-27 shows the location of the 29 additional drill holes, which were drilled during the 2018 drill campaign. The 2018 drill holes, shown in red, are focused on the north end of the North Pit.

Figure 14-27: Drill Hole Distribution Showing the Last Campaign Collar in Red (Golder, 2019)



14.2.2 Resource Estimation Methodology

14.2.2.1 Exploratory Data Analyses

The objective of exploratory data analysis (EDA) is to investigate similarities or differences between populations of grades and to determine possible groupings or separations by different criteria such as geological attributes. The exploratory data analysis also seeks to detect the existence of drifts that may affect the result of the estimation.

The statistical suitability of the estimation units was reviewed through the implementation of statistical and geostatistical tools. Basic statistics, dispersion diagrams of standard deviations versus average grades and accumulated probabilistic graphs were carried out. All statistical analyzes were developed using the composite database.

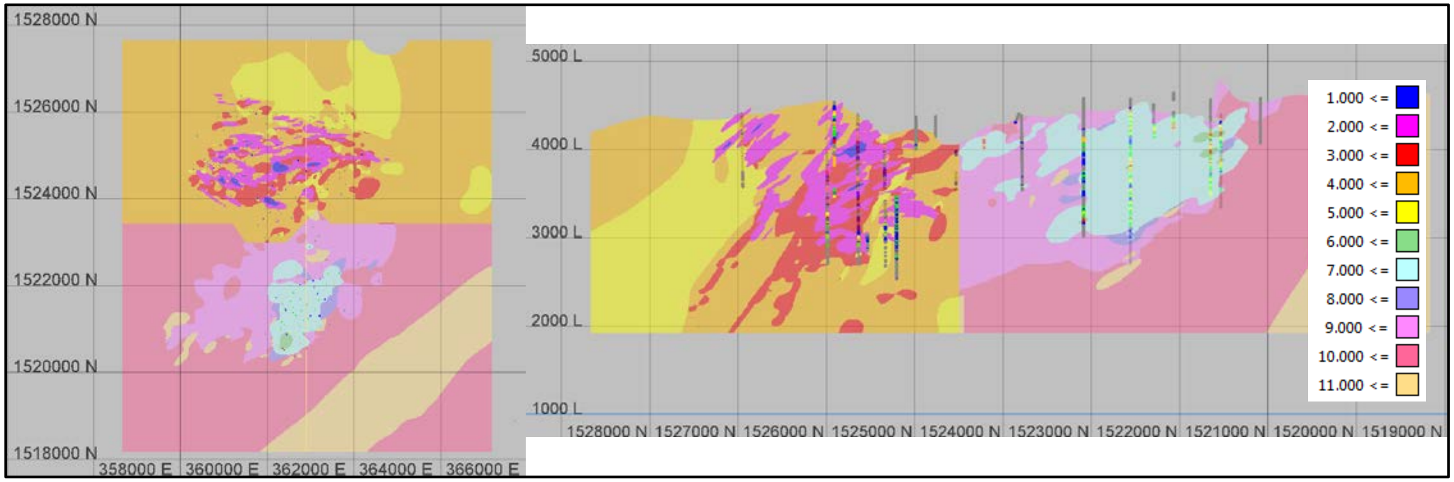
Estimation Unit Definition

The 2018 Open Pit Mineral Resource model used a combination of grade shells and lithology to define the estimation units (EU) of each variable, these domains are mainly based on the grade shells, and outside defined by lithology. Table 14-16 shows the EU definition and the Figure 14-28 and Figure 14-29 show their spatial distribution, the same EUs were used for the estimation of gold and silver.

Table 14-16: Estimation Unit definition for Total Copper, Gold and Silver

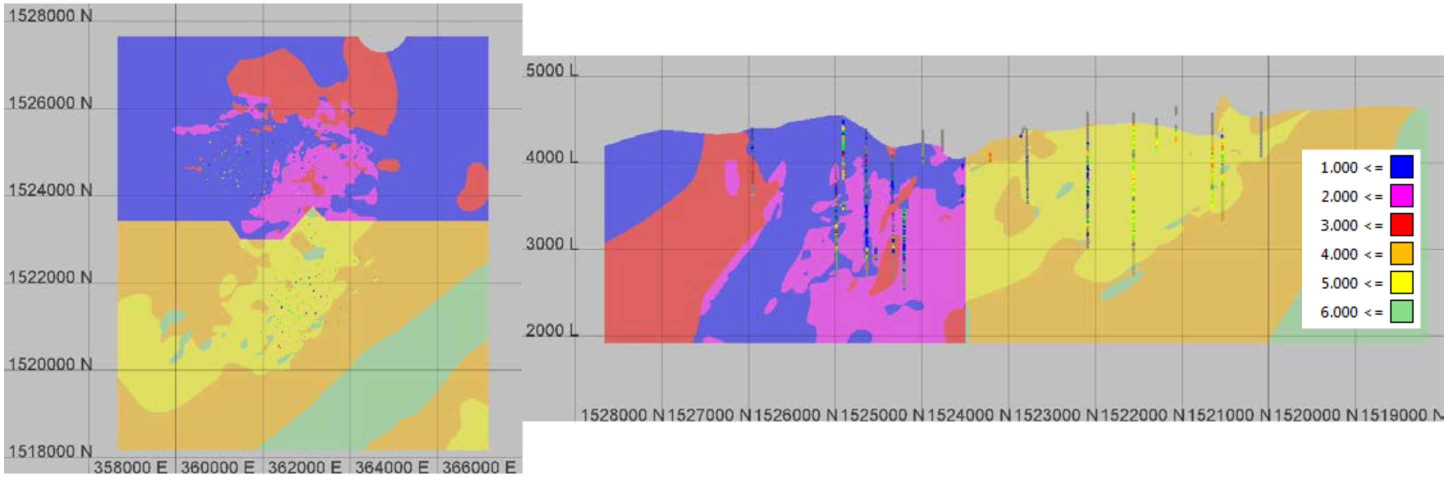
Estimation unit	Lithology	Grade Shell	Deposit
1	All	0.80% total copper	North
2	All	0.15% total copper	North
3	60, 70 and 100	None	North
4	40, 50 and 80	None	North
5	90	None	North
6	All	0.80% total copper	South
7	All	0.15% total copper	South
8	60	None	South
9	70 and 80	None	South
10	40, 50 and 100	None	South
11	90	None	South

Figure 14-28: Spatial Distribution of the Estimation Units (Golder, 2019)



Note: Plan View 3440 (left) & Section 362900 (right).

Figure 14-29: Spatial Distribution of Mo Estimation Units (Golder, 2019)



Note: Plan View 3440 (left) & Section 362900 (right).

In the case of Mo and Fe, the EU were defined using only the lithology, as shown in Table 14-17 and Table 14-18.

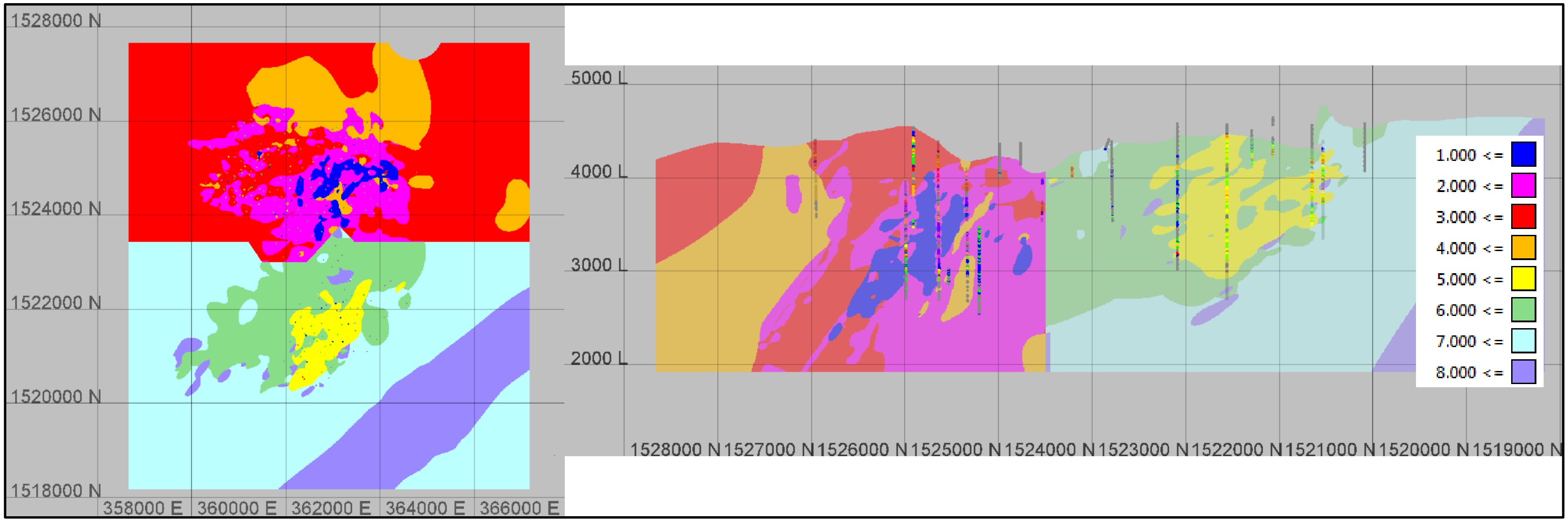
Table 14-17: Estimation Unit Definition for Mo

Estimation unit	Lithology	Grade Shell	Deposit
1	50, 70 and 100	None	North
2	40, 60 and 80	None	North
3	90	None	North
4	50, 70 and 100	None	South
5	40, 60 and 80	None	South
6	90	None	South

Table 14-18: Estimation Unit Definition for Fe

Estimation Unit	Lithology	Grade Shell	Deposit
1	60	None	North
2	40, 70, 80 and 100	None	North
3	50	None	North
4	90	None	North
5	60	None	South
6	70 and 80	None	South
7	40, 50 and 100	None	South
8	90	None	South

Figure 14-30: Spatial Distribution of Fe Estimation Units (Golder, 2019)



Note: Plan View 3440 (left) & Section 362900 (right).

Lithologies 10, 20 and 30 were not part of the estimation process and appear with a default value (0.00001) on the block model for all variables.

Missing Values Treatment

As in previous models, gold and silver missing values were assigned using a regression with total copper. It is recognized that the correlation is weak; however, the presence of gold with the copper is clear. As the gold contribution to value is relatively small, and with no other clear method of including the gold contribution, the regression method was considered as the most appropriate approach. Figure 14-31 and Figure 14-32 show scatter plots of total copper and gold and the regression formula used in the North and South Deposits. The silver estimation showed a somewhat stronger correlation to copper than did the gold estimation.

Figure 14-31: Scatter Plot Total Copper versus Gold (Golder, 2019)

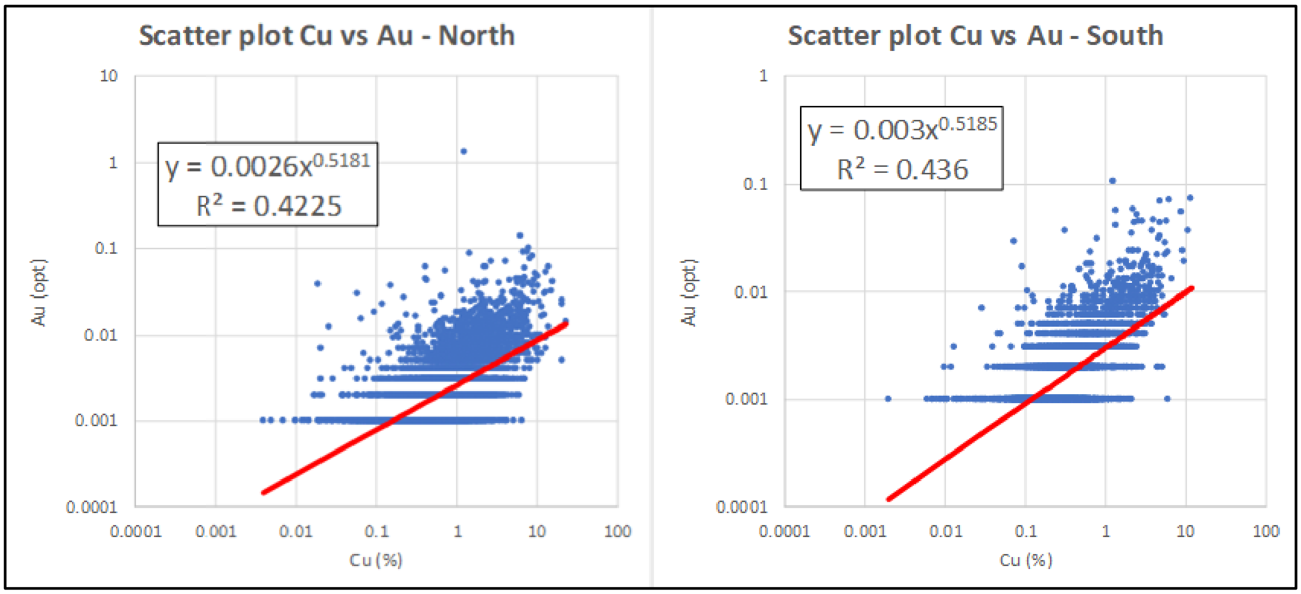
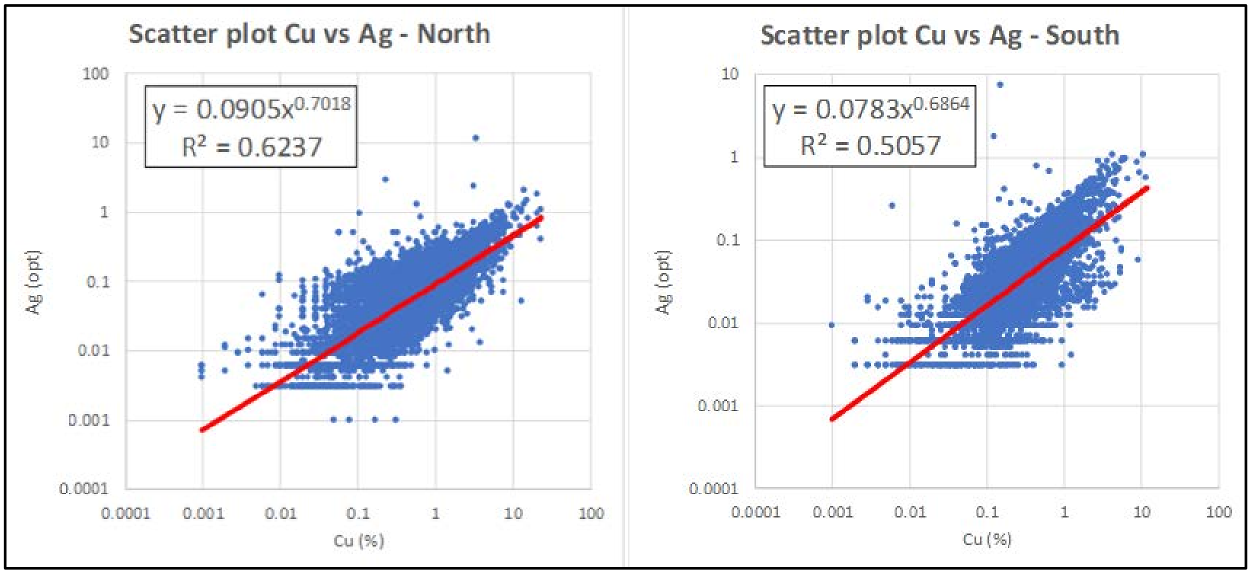


Figure 14-32: Scatter Plot Total Copper versus Silver (Golder, 2019)



Regression between total copper and molybdenum showed poor results. In this case, a default value was used to populate drill hole data.

In an effort to make a more robust gold and silver estimation it is recommended that selected stored samples to be re-assayed for gold and silver.

Composite Statistics

Sample centroids were flagged with grade shells and 10 ft composites were created starting at the boundary of each grade shell. Lithology was flagged in the resulting composites from the block model to create the EU.

Table 14-19, Table 14-20, Table 14-21, Table 14-22 and Table 14-23 show the basic statistics of total copper, silver, gold, molybdenum and iron by EU respectively. The statistics provide basic information on the differences of the EU units and are used as a check to ensure there are sufficient samples in each unit for grade interpolation.

Table 14-19: Composite Statistics by Estimation Unit – Total Copper

EU Total Copper	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev	CV
Total	54,615	0.0001	23.4500	0.1833	0.0501	0.0100	0.1827	0.4308	2.3497
1	1,537	0.0001	23.4500	1.7240	1.3600	1.0500	1.9500	1.3380	0.7761
2	9,041	0.0001	5.6400	0.3873	0.3060	0.1980	0.4918	0.3177	0.8204
3	5,724	0.0001	1.4224	0.0540	0.0377	0.0145	0.0801	0.0581	1.0776
4	15,802	0.0001	1.1160	0.0357	0.0190	0.0054	0.0500	0.0479	1.3425
5	4,151	0.0001	0.5800	0.0175	0.0067	0.0001	0.0166	0.0365	2.0850
6	208	0.1680	7.9800	1.8843	1.5880	1.1597	2.2780	1.1191	0.5939
7	5,855	0.0015	7.7660	0.3377	0.2500	0.1800	0.3913	0.3091	0.9154
8	2,095	0.0001	0.2710	0.0895	0.0900	0.0600	0.1200	0.0415	0.4639
9	6,154	0.0001	0.5755	0.0464	0.0300	0.0100	0.0719	0.0512	1.1051
10	2,826	0.0001	0.5120	0.0203	0.0100	0.0021	0.0200	0.0347	1.7117
11	1,222	0.0001	0.4066	0.0137	0.0073	0.0012	0.0100	0.0264	1.9328

Table 14-20: Composite Statistics by Estimation Unit - Gold

EU Au	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev	CV
Total	54,767	0.00001	0.28600	0.00067	0.00020	0.00001	0.00070	0.00208	3.10011
1	1,542	0.00001	0.05440	0.00469	0.00330	0.00233	0.00510	0.00509	1.08683
2	9,141	0.00001	0.28600	0.00138	0.00109	0.00065	0.00163	0.00328	2.37705
3	5,726	0.00001	0.01698	0.00027	0.00015	0.00001	0.00035	0.00055	2.05266
4	15,831	0.00001	0.10302	0.00017	0.00001	0.00001	0.00020	0.00097	5.79175
5	4,154	0.00001	0.00517	0.00009	0.00001	0.00001	0.00012	0.00020	-9999999
6	208	0.00038	0.03430	0.00622	0.00464	0.00334	0.00724	0.00525	0.84368
7	5,861	0.00001	0.05314	0.00145	0.00108	0.00057	0.00173	0.00201	1.38658
8	2,096	0.00001	0.01467	0.00036	0.00026	0.00012	0.00047	0.00050	1.40948
9	6,160	0.00001	0.00496	0.00021	0.00012	0.00001	0.00026	0.00032	1.54950
10	2,826	0.00001	0.04658	0.00013	0.00001	0.00001	0.00015	0.00091	7.00535
11	1,222	0.00001	0.00747	0.00012	0.00001	0.00001	0.00015	0.00032	2.63253

Table 14-21: Composite Statistics by Estimation Unit - Silver

EU Ag	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev	CV
Total	54,767	0.0010	5.7086	0.0236	0.0076	0.0010	0.0277	0.0566	2.3936
1	1,542	0.0010	1.2688	0.1481	0.1287	0.0846	0.1800	0.1078	0.7280
2	9,141	0.0010	5.7086	0.0516	0.0400	0.0248	0.0627	0.0790	1.5309
3	5,726	0.0010	0.3862	0.0098	0.0058	0.0010	0.0117	0.0149	1.5285
4	15,831	0.0010	0.5542	0.0090	0.0026	0.0010	0.0088	0.0231	2.5623
5	4,154	0.0010	0.1319	0.0035	0.0010	0.0010	0.0026	0.0082	2.3078
6	208	0.0081	0.7948	0.1661	0.1163	0.0633	0.2431	0.1464	0.8818
7	5,861	0.0010	3.7156	0.0419	0.0300	0.0192	0.0490	0.0644	1.5372
8	2,096	0.0010	0.2000	0.0157	0.0134	0.0070	0.0204	0.0138	0.8798
9	6,160	0.0010	1.4613	0.0077	0.0049	0.0010	0.0099	0.0212	2.7604
10	2,826	0.0010	3.4198	0.0059	0.0010	0.0010	0.0058	0.0649	10.9202
11	1,222	0.0010	0.0603	0.0032	0.0010	0.0010	0.0034	0.0050	1.5647

Table 14-22: Composite Statistics by Estimation Unit - Molybdenum

EU Mo	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev	CV
Total	54,751	0.00001	0.65165	0.00208	0.00050	0.00003	0.00225	0.00533	2.56146
1	24,378	0.00001	0.65165	0.00364	0.00180	0.00035	0.00475	0.00728	1.99894
2	7,542	0.00001	0.08760	0.00118	0.00030	0.00006	0.00100	0.00341	2.88505
3	4,463	0.00001	0.03380	0.00055	0.00010	0.00001	0.00035	0.00175	3.16624
4	3,645	0.00001	0.05629	0.00129	0.00045	0.00001	0.00145	0.00269	2.08373
5	13,419	0.00001	0.03392	0.00064	0.00020	0.00001	0.00066	0.00143	2.23699
6	1,304	0.00001	0.01392	0.00043	0.00002	0.00001	0.00040	0.00096	2.24316

Table 14-23: Composite Statistics by Estimation Unit - Iron

EU Fe	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev	CV
Total	54,751	0.01	64.10	7.7316	3.8809	2.0102	8.2729	10.6684	1.3798
1	1,769	0.01	59.90	21.8776	21.7000	13.5489	30.5563	12.3996	0.5668
2	15,646	0.01	59.00	5.2569	3.9950	2.5950	6.3400	5.2208	0.9931
3	14,505	0.01	47.70	2.5241	2.6300	0.0100	3.8500	2.5066	0.9930
4	4,463	0.01	55.65	2.5451	2.1900	0.0100	3.3530	3.9340	1.5457
5	6,016	0.01	64.10	29.1415	28.2253	18.1000	39.2000	13.8759	0.4762
6	7,970	0.01	60.30	7.8390	6.6353	3.9250	10.3950	6.1498	0.7845
7	3,078	0.01	56.00	3.8481	2.1000	0.2521	5.3000	5.5666	1.4466
8	1,304	0.01	56.80	3.6464	2.8965	0.9638	4.2000	4.6784	1.2830

Figure 14-33, Figure 14-34, Figure 14-35, Figure 14-36 and Figure 14-37 shows the probabilistic distribution and the mean versus standard deviation graph for total copper, silver, gold, molybdenum and iron by EU. In general, the graphs show an adequate differentiation in the mean grade for the different EUs and a low coefficient of variation, which indicates acceptable values of variability.

Figure 14-33: Probability Distribution by Estimation Unit – Copper (Golder, 2019)

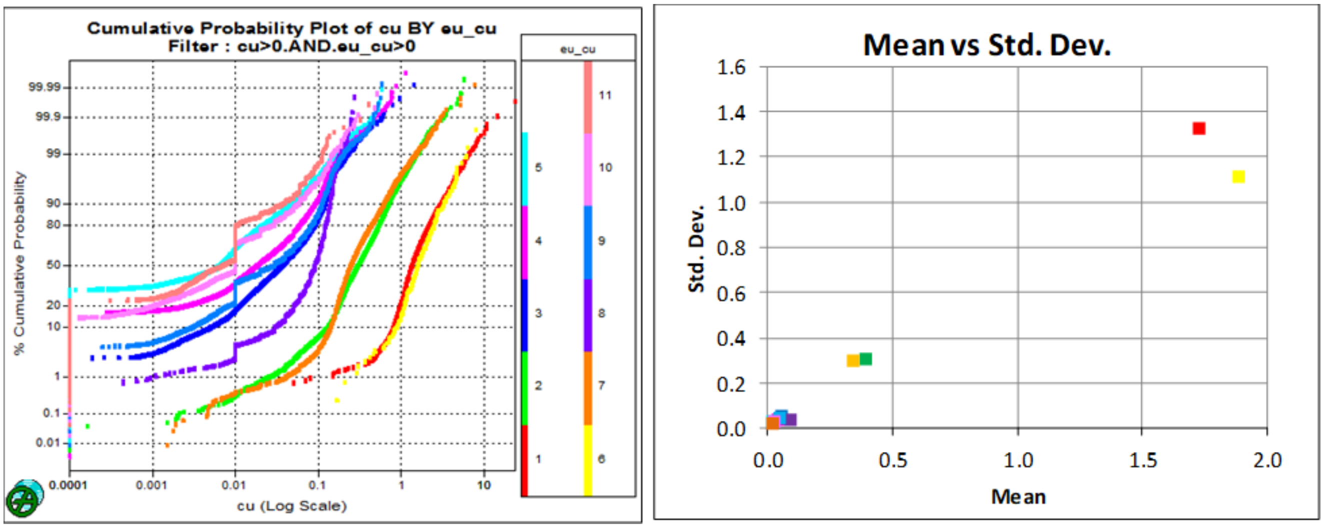


Figure 14-34: Probability Distribution by Estimation Unit – Gold (Golder, 2019)

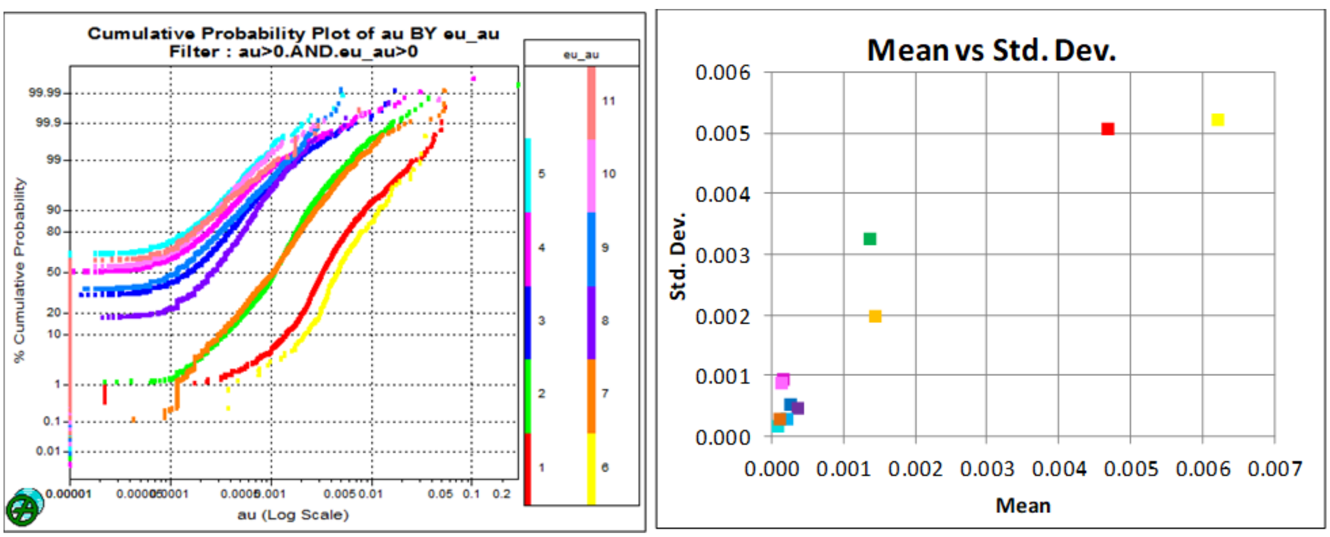


Figure 14-35: Probability Distribution by Estimation Unit – Silver (Golder, 2019

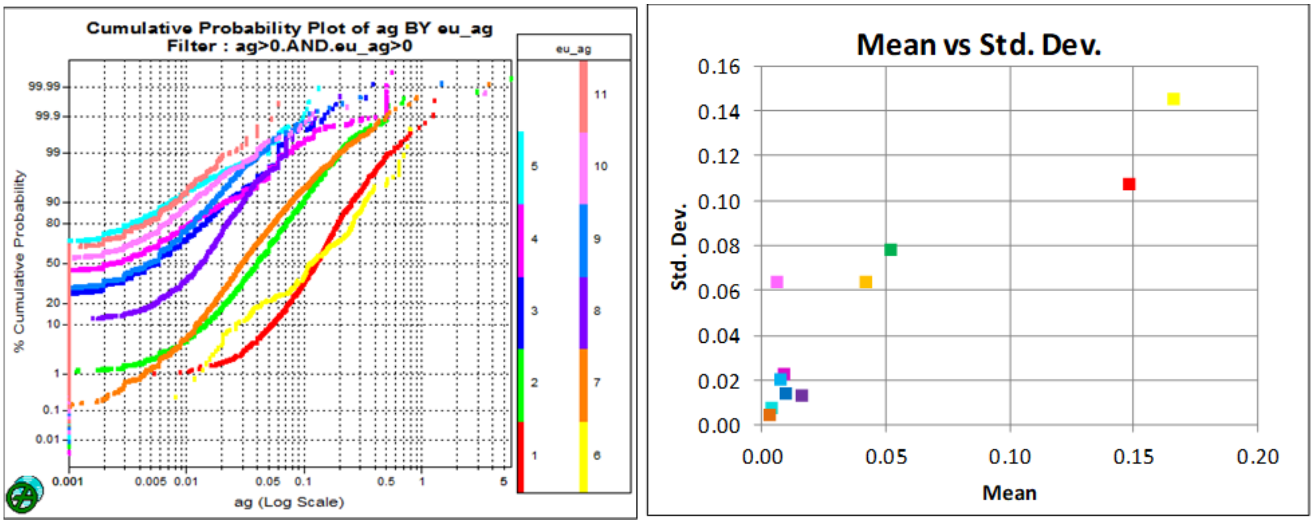


Figure 14-36: Probability Distribution by Estimation Unit – Molybdenum (Golder, 2019)

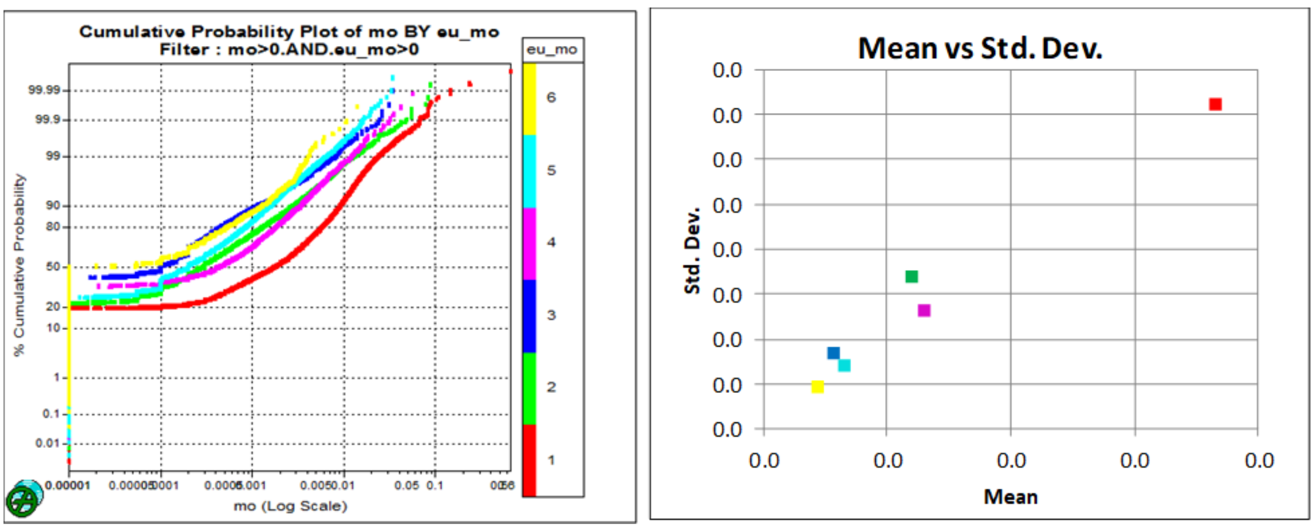
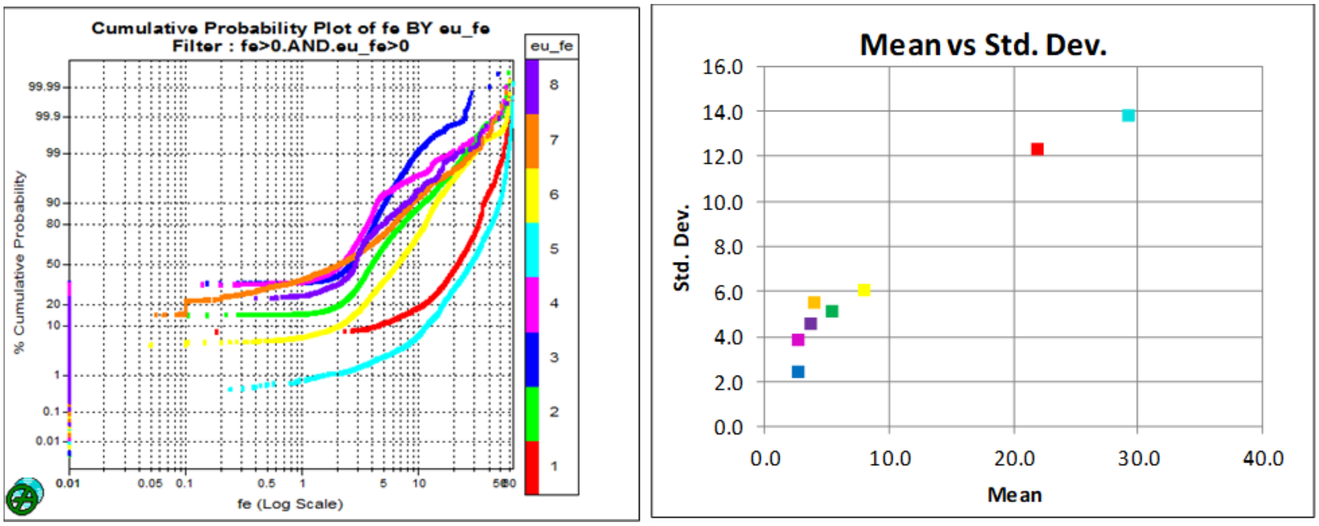


Figure 14-37: Probability Distribution by Estimation Unit – Iron (Golder, 2019)



Outlier Treatment

The definition and control of anomalous samples in the different populations is a practice accepted by the industry and is necessary to avoid possible overestimation of grade and/or tonnage.

The anomalous values were defined from probabilistic distribution curves, Figure 14-38 to Figure 14-41 show the probabilistic distribution and the graph of relative differences for EU 1, 2, 6 and 7 of total copper, where the thresholds were defined depending on population breaks or loss of continuity. The same approach was applied to the other variables, being Au, Ag, Fe and Mo.

Figure 14-38: Probability Distribution and Mean vs Standard Deviation, EU 1 – Total Copper (Golder, 2019)

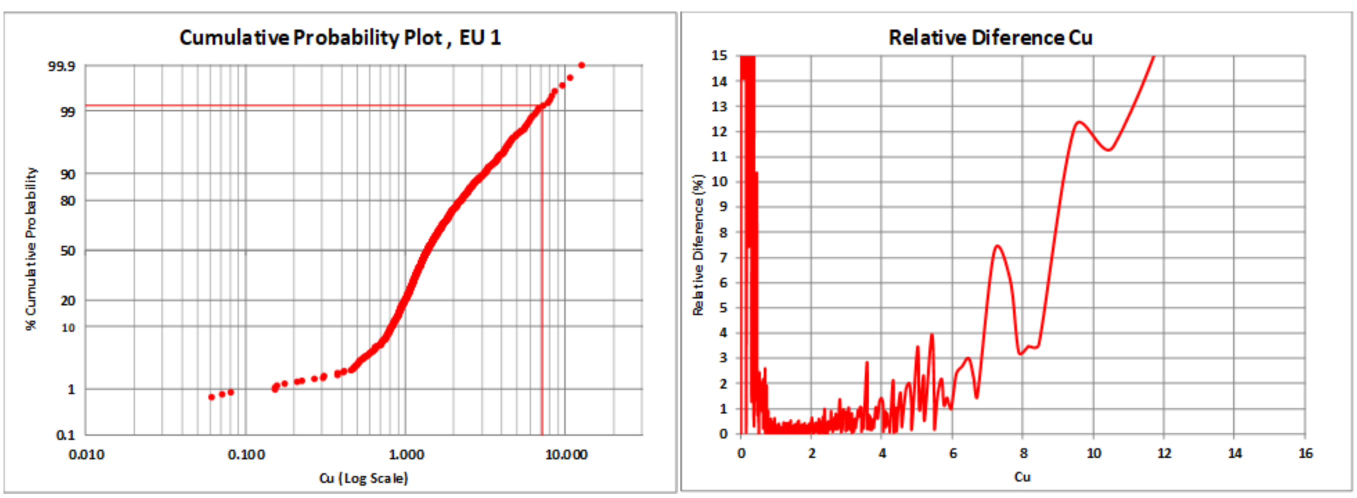


Figure 14-39: Probability Distribution and Mean vs Standard Deviation, EU 2 – Total Copper (Golder, 2019)

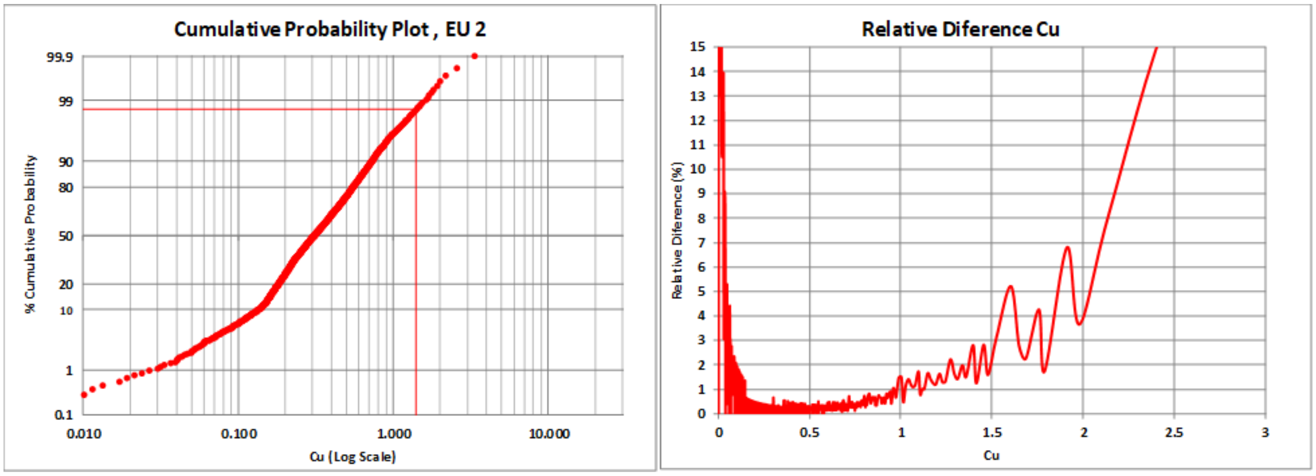


Figure 14-40: Probability Distribution & Mean vs Standard Deviation, EU 6 – Total Copper (Golder, 2019)

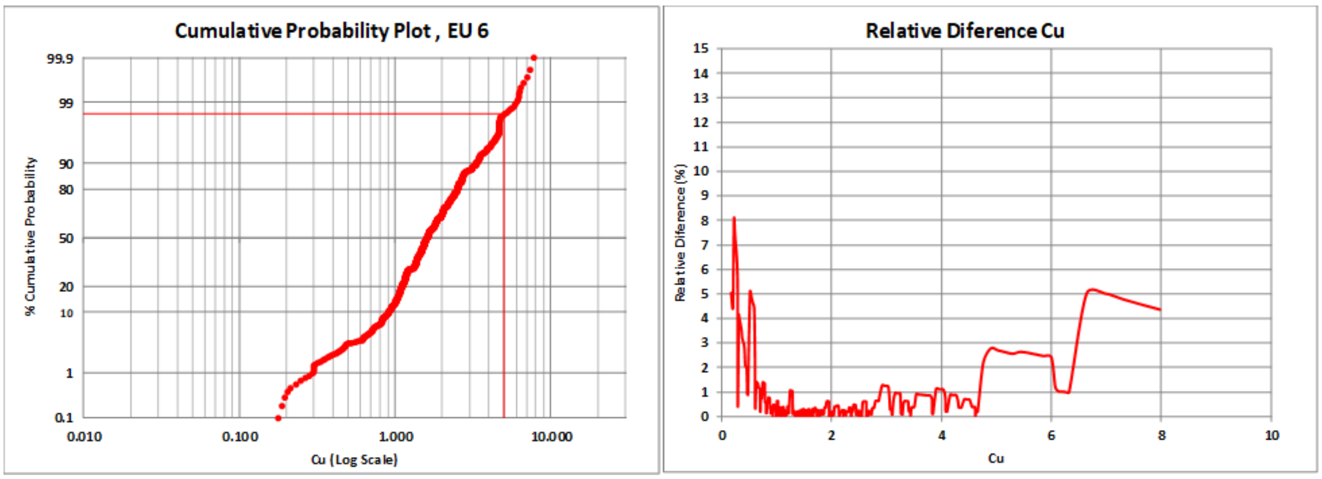


Figure 14-41: Probability distribution & Mean vs Standard Deviation, EU 7 – Total Copper (Golder, 2019)

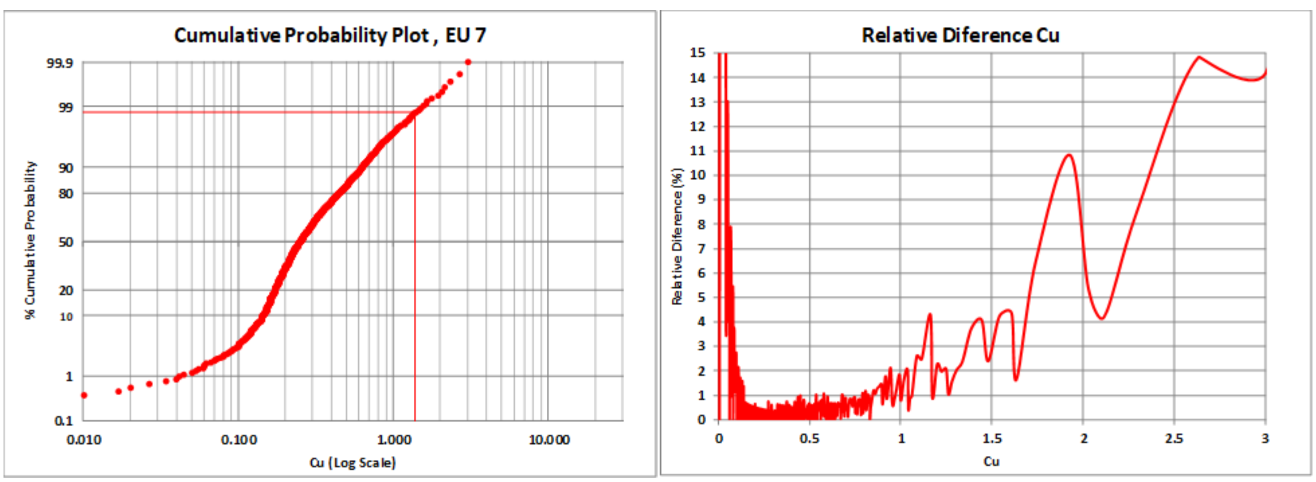


Table 14-24 shows the treatment of anomalous values for each variable and EU. The influence of anomalous high values was limited by using High Yield Restriction. High Yield Restriction consists of using the composite with its real value but in a restricted radius. For example, in EU 6 a threshold of 6.0% of total copper was defined. So, if a composite has 8.0% total copper (>6.0%) it will be used in the estimation but with a reduced radius of influence of 200 ft (4 blocks of influence in each direction).

Table 14-24: High Yield Restriction Values for Each Variable & Estimation Unit

EU	Cu	Au	Ag	Mo	Fe
1	9.00	0.04	0.70	0.050	-
2	1.4	-	0.50	0.025	35.0
3	0.50	0.01	0.10	0.010	20.0
4	0.50	0.01	0.10	0.020	35.0
5	0.40	-	0.05	0.020	-
6	6.00	0.03	0.70	0.004	40.0
7	1.4	0.03	0.50		30.0
8	0.30	0.002	0.10		20.0
9	0.30	0.002	0.10		
10	0.20	0.002	0.10		
11	0.10	0.002	0.04		

Contact Analysis

To determine the type of contacts (soft, transitional or hard) between the different EUs, a contact profile analysis was performed. The contact analysis is a mathematical method to define the behavior of the grade across the contact between EUs. During the grade estimation is important to consider the type of contact between EU because the possibility of sharing samples and improvement of the estimation near the contact.

The Contact Profile takes samples from one EU and pairs it with samples from other EUs based on the separation distance. The pairs are constructed over an increasing separation distance. For each distance, the average grade of the first domain is plotted against the average grade calculated with the second domain. These points are in the chart with the distance in the X-axis and the average grade in the vertical axis.

The contact analysis graphs show the composite average grades in relation to distance to the contact. The blue and green continuous lines correspond to the average of total copper in each of the EUs in relation to the distance to the contact. Negative distances represent the distance to the contact from inside domain "A" and positive distances represent distances to the contact from inside domain "B." The black dashed line corresponds to the number of pairs used in the calculation of the average grade estimating. Some examples of the type of contacts are shown in Figure 14-42, Figure 14-43 and Figure 14-44.

Figure 14-42: Contact Profile between EU 2 & 3, Total Copper (Golder, 2019)

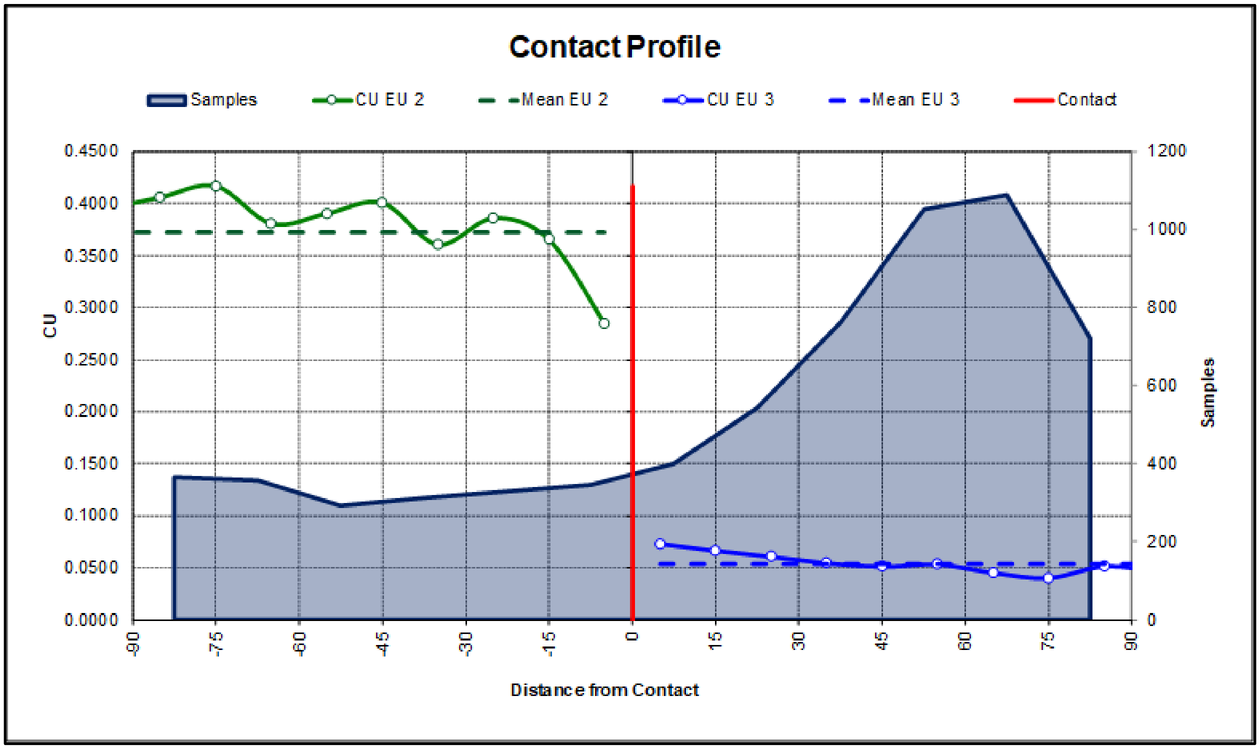


Figure 14-43: Contact Profile between EU 2 & 4, Total Copper (Golder, 2019)

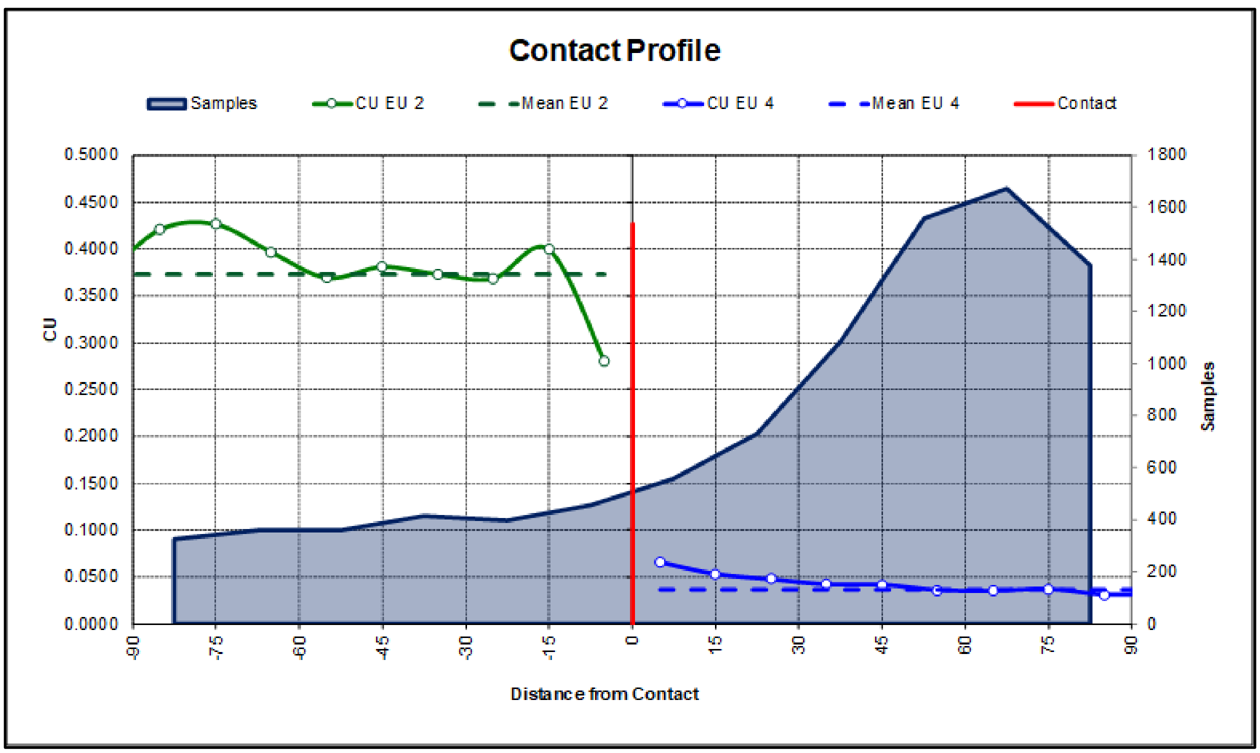
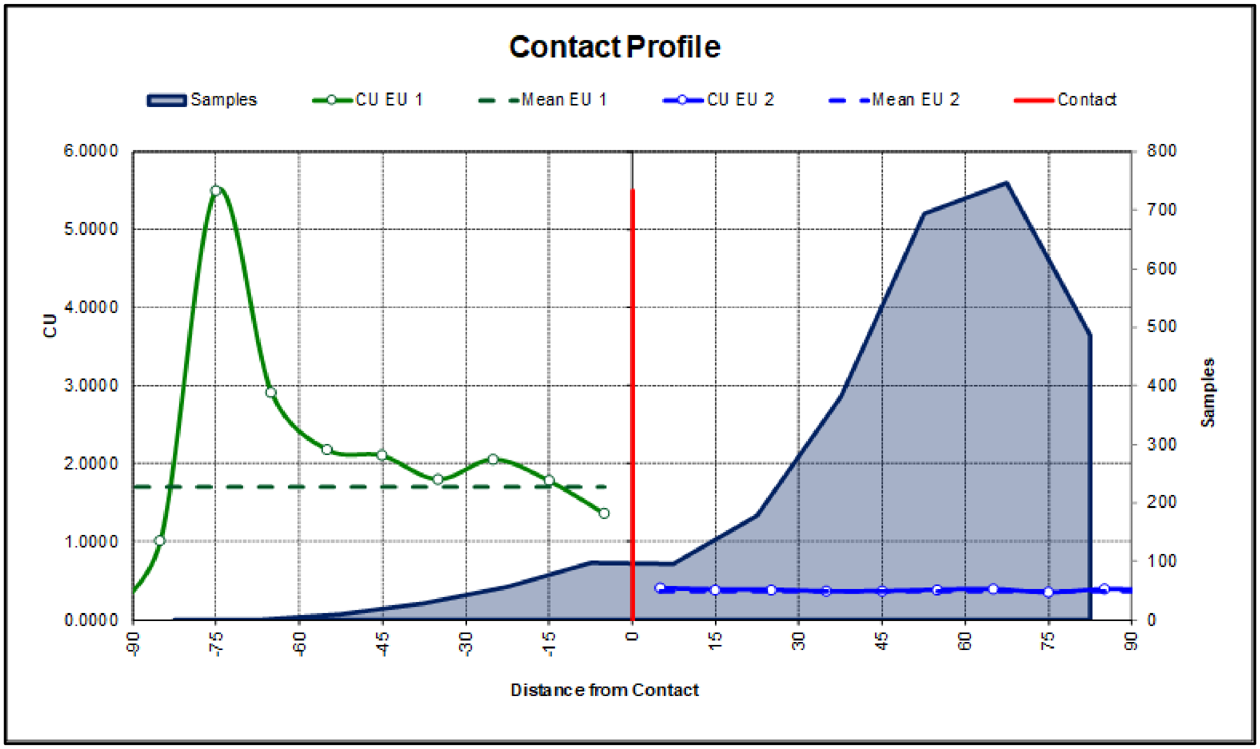


Figure 14-44: Contact Profile between EU 1 & 2, Total Copper (Golder, 2019)



The results show there is some transitional contact. However, Golder elected to use hard boundaries between all EUs, which is an acceptable method and keeping the same methodology used in previous models.

Drift Analyses

The objective of the drift analysis is identifying the presence of trends in grades along specified coordinates for each EU.

For each EU, east, north and elevation directions were analyzed. The analysis was done by calculating the mean grade of the composites in adjacent panels of 100 by 100 by 50 ft for every defined direction. From the analysis, the presence of drift (a gradually and consistently increase or decrease in mean grade in the samples in any direction) and the presence of entropy (abrupt changes in mean grade without clear control between neighboring panels) will be identified, the intensity of the drift will be defined as strong or low according to the intensity of the effect observed in the graphs.

Table 14-25, Table 14-26, Table 14-27, Table 14-28 and Table 14-29 present a summary of the drift analysis for each variable and EU. For EUs for which a mineralization drift has been identified, the mineralization drift should be considered in the estimation process, particularly in the definition and treatment of outlier values, the study of the spatial correlation (variography) and the estimation plan orientation.

The interpretations used in the drift analysis are described below:

- Abrupt local changes of grade: that do not obey to any trend. These areas will be identified by a **blue circle** in dotted line.
- Boundary conditions: correspond to a grade increase throughout the coordinate in conditions of less information (boundaries). These areas will be identified by a **red circle** in dotted line.
- High grade center: some EU have a high grade core, with a grade decreases towards its boundaries. These areas will be identified by a **red rectangle** in dotted line.

Table 14-25: Drift Summary – Total Copper

EU	Direction X	Direction Y	Direction Z	Observation
1	No	No	Yes	High grade center and boundary conditions
2	Yes	Yes	Yes	Boundary conditions
3	No	No	No	No observation
4	No	No	Yes	Boundary conditions
5	No	No	No	No observation
6	Yes	No	Yes	High grade center and boundary conditions
7	Yes	Yes	Yes	High grade center and boundary conditions
8	No	No	No	No observation
9	No	No	No	No observation
10	No	No	No	No observation
11	No	No	No	No observation

Table 14-26: Drift Summary – Au

EU	Direction X	Direction Y	Direction Z	Observation
1	Yes	Yes	Yes	High grade center and boundary conditions
2	Yes	Yes	Yes	High grade peaks and boundary conditions
3	No	No	No	No observation
4	No	No	No	No observation
5	No	No	No	No observation
6	No	No	No	No observation
7	Yes	Yes	Yes	High grade peaks and boundary conditions
8	Yes	Yes	No	High grade peaks and boundary conditions
9	No	No	No	No observation
10	Yes	Yes	No	High grade peaks
11	Yes	No	No	High grade peaks

Table 14-27: Drift Summary – Ag

EU	Direction X	Direction Y	Direction Z	Observation
1	No	Yes	No	High grade peaks
2	Yes	Yes	Yes	Boundary conditions
3	No	No	No	No observation
4	No	No	No	No observation
5	Yes	No	No	High grade peaks
6	No	No	No	No observation
7	Yes	No	No	High grade peaks
8	Yes	Yes	No	High grade peaks
9	Yes	Yes	No	High grade peaks and boundary conditions
10	Yes	Yes	Yes	High grade peaks
11	No	No	Yes	High grade peaks

Table 14-28: Drift Summary – Mo

EU	Direction X	Direction Y	Direction Z	Observation
1	Yes	Yes	Yes	Boundary conditions
2	Yes	Yes	Yes	Boundary conditions
3	Yes	Yes	Yes	High grade peaks
4	Yes	Yes	No	High grade peaks
5	Yes	Yes	No	High grade peaks
6	Yes	Yes	No	High grade peaks

Table 14-29: Drift Summary – Fe

EU	Direction X	Direction Y	Direction Z	Observation
1	No	No	No	No observation
2	Yes	No	Yes	Boundary conditions
3	Yes	Yes	Yes	High grade peaks
4	No	No	Yes	Boundary conditions
5	Yes	No	Yes	Boundary conditions
6	Yes	No	No	High grade peaks
7	No	Yes	No	High grade peaks
8	Yes	Yes	Yes	High grade peaks

The estimation process proposed for each EU, will consider the presence and type of drift identified in this analysis. Drifts may be decreased by the interpolation process of Ordinary Kriging.

Figure 14-45 shows the drift analysis for EU 1 Cu, it is observed slight trend in the Z axis. For X and Y axis, copper does not show trend except in the Z direction where the grade decreases downward. In addition to this, there are some boundary conditions, in which the grade increases due to presence of high grade samples in the boundary zones.

Figure 14-45: Drift analysis EU 1 – Total Copper (Golder, 2019)

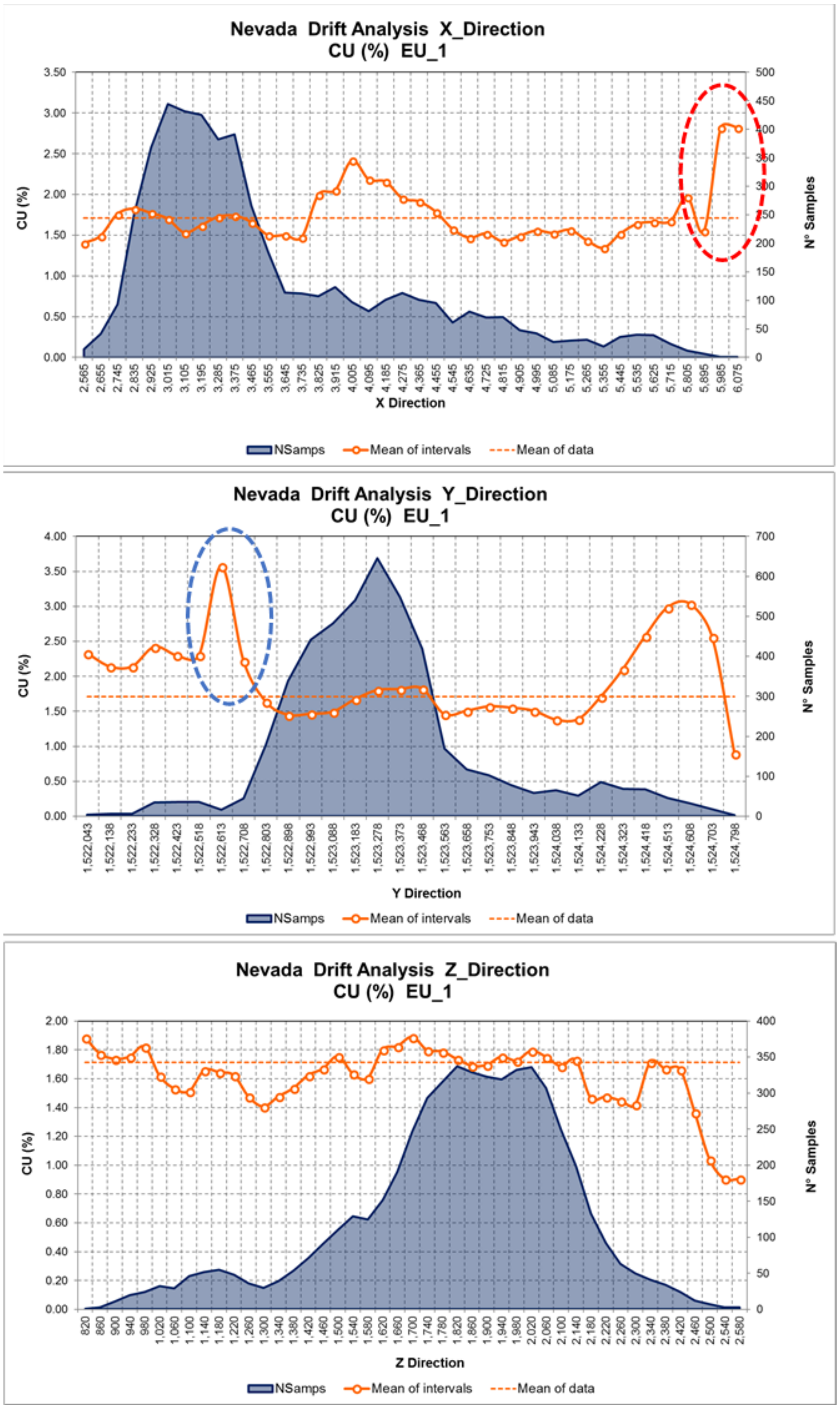
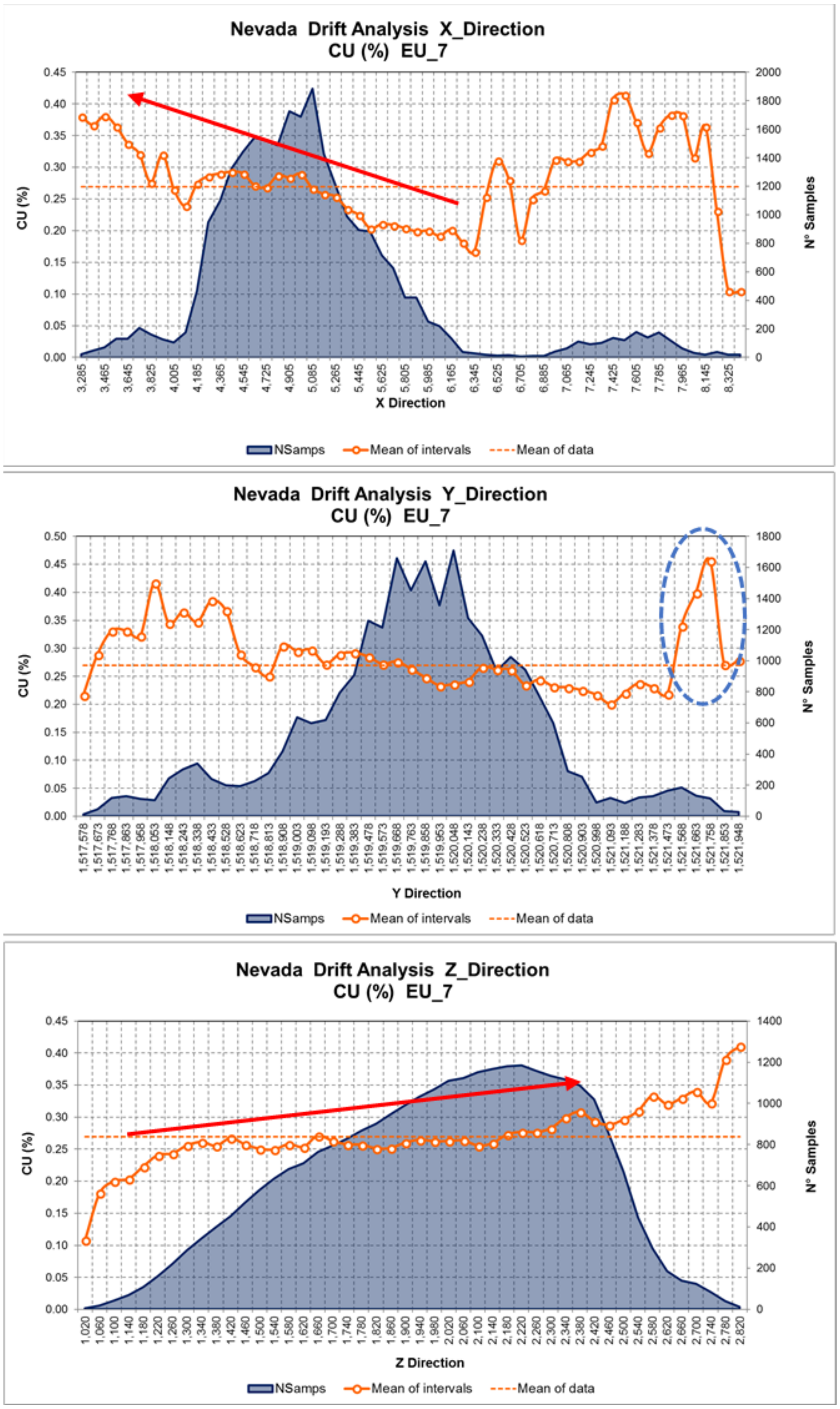


Figure 14-46 shows the drift analysis for EU 3 Cu. In general, there are not erratic grades of copper; however, there are some boundary conditions in zones with large amounts of samples.

Figure 14-46: Drift analysis EU 7 – Total Copper (Golder, 2019)



The last example is showed in Figure 14-46, the drift analysis for EU 7 shows that there are some erratic copper values.

Spatial Correlation & Variography

To define the spatial continuity for an EU, it is necessary to calculate and model experimental variograms. All variograms were calculated and modeled with Golder software: OBO, V11.05®. Directional and Down the Hole (DTH) variograms were calculated for using the 10 ft composites database.

Golder performed the variography process as follows:

- Calculate of the variograms maps through OBO software, V11.05®
- Calculate of the experimental variograms through OBO software, V11.05®
- Calculate DTH variograms to determine the nugget effect
- Model experimental variograms in the principal directions

Table 14-30, Table 14-31, Table 14-32, Table 14-33 and Table 14-34 present a summary of the variograms models defined for EU for total copper, gold, silver, molybdenum and iron, respectively.

Table 14-30: Summary of Correlograms Models for Total Copper

Variable	EU	Direction	Nugget	First Structure			Second Structure		
				Sill	Type	Range (ft)	Sill	Type	Range (ft)
Total Copper	1	Omni	0.40	0.40	Spherical	30	0.20	Spherical	180
Total Copper	2	Omni	0.40	0.40	Spherical	20	0.20	Spherical	150
Total Copper	3	Omni	0.40	0.30	Spherical	60	0.30	Spherical	550
Total Copper	4	Omni	0.40	0.35	Spherical	40	0.25	Spherical	450
Total Copper	5	Omni	0.40	0.35	Spherical	30	0.25	Spherical	300
Total Copper	6	Omni	0.40	0.60	Spherical	110			
Total Copper	7	Omni	0.50	0.2	Spherical	80	0.3	Spherical	300
Total Copper	8	Omni	0.30	0.40	Spherical	70	0.30	Spherical	450
Total Copper	9	Omni	0.60	0.40	Spherical	350			
Total Copper	10	Omni	0.40	0.40	Spherical	70	0.2	Spherical	350
Total Copper	11	Omni	0.40	0.40	Spherical	70	0.2	Spherical	350

Table 14-31: Summary of Correlograms Models for Gold

Variable	EU	Direction	Nugget	First Structure			Second Structure		
				Sill.	Type	Range	Sill	Type	Range
Au	1	Omni	0.50	0.20	Spherical	50	0.30	Spherical	150
Au	2	Omni	0.40	0.50	Spherical	20	0.10	Spherical	200
Au	3	Omni	0.75	0.25	Spherical	300			
Au	4	Omni	0.90	0.10	Spherical	200			
Au	5	Omni	0.60	0.40	Spherical	350			
Au	6	Omni	0.50	0.30	Spherical	90	0.20	Spherical	250
Au	7	Omni	0.50	0.20	Spherical	70	0.30	Spherical	250
Au	8	Omni	0.90	0.10	Spherical	50			
Au	9	Omni	0.70	0.30	Spherical	100			
Au	10	Omni	0.40	0.30	Spherical	50	0.30	Spherical	200
Au	11	Omni	0.40	0.30	Spherical	50	0.30	Spherical	200

Table 14-32: Summary of Correlograms Models for Silver

Variable	EU	Direction	Nugget	First Structure			Second Structure		
				Sill.	Type	Range	Sill	Type	Range
Ag	1	Omni	0.40	0.30	Spherical	30	0.30	Spherical	120
Ag	2	Omni	0.40	0.30	Spherical	40	0.30	Spherical	200
Ag	3	Omni	0.40	0.30	Spherical	30	0.30	Spherical	200
Ag	4	Omni	0.15	0.85	Spherical	200			
Ag	5	Omni	0.40	0.30	Spherical	50	0.30	Spherical	300
Ag	6	Omni	0.10	0.20	Spherical	100	0.70	Spherical	400
Ag	7	Omni	0.40	0.50	Spherical	20	0.10	Spherical	120
Ag	8	Omni	0.30	0.40	Spherical	40	0.30	Spherical	200
Ag	9	Omni	0.40	0.60	Spherical	250			
Ag	10	Omni	0.60	0.25	Spherical	50	0.15	Spherical	350
Ag	11	Omni	0.60	0.25	Spherical	50	0.15	Spherical	350

Table 14-33: Summary of Correlograms Models for Molybdenum

Variable	EU	Direction	Nugget	First Structure			Second Structure		
				Sill.	Type	Range	Sill	Type	Range
Mo	1	Omni	0.40	0.30	Spherical	30	0.30	Spherical	400
Mo	2	Omni	0.20	0.40	Spherical	50	0.40	Spherical	300
Mo	3	Omni	0.20	0.40	Spherical	30	0.40	Spherical	150
Mo	4	Omni	0.20	0.40	Spherical	60	0.40	Spherical	400
Mo	5	Omni	0.30	0.40	Spherical	100	0.30	Spherical	600
Mo	6	Omni	0.20	0.80	Spherical	400			

Table 14-34: Summary of Correlograms Models for Iron

Variable	EU	Direction	Nugget	First Structure			Second Structure		
				Sill.	Type	Range	Sill	Type	Range
Fe	1	Omni	0.20	0.40	Spherical	300	0.40	Spherical	500
Fe	2	Omni	0.20	0.40	Spherical	200	0.40	Spherical	700
Fe	3	Omni	0.20	0.70	Spherical	200	0.10	Spherical	400
Fe	4	Omni	0.20	0.30	Spherical	150	0.50	Spherical	400
Fe	5	Omni	0.20	0.40	Spherical	120	0.40	Spherical	450
Fe	6	Omni	0.20	0.50	Spherical	250	0.30	Spherical	500
Fe	7	Omni	0.20	0.30	Spherical	100	0.50	Spherical	450
Fe	8	Omni	0.20	0.80	Spherical	300			

Figure 14-47 and Figure 14-48 provide examples of modeled 3D and DTH variograms of total copper for EU 1 and EU 5, respectively.

Figure 14-47: Correlogram Copper, EU 1 (Golder, 2019)

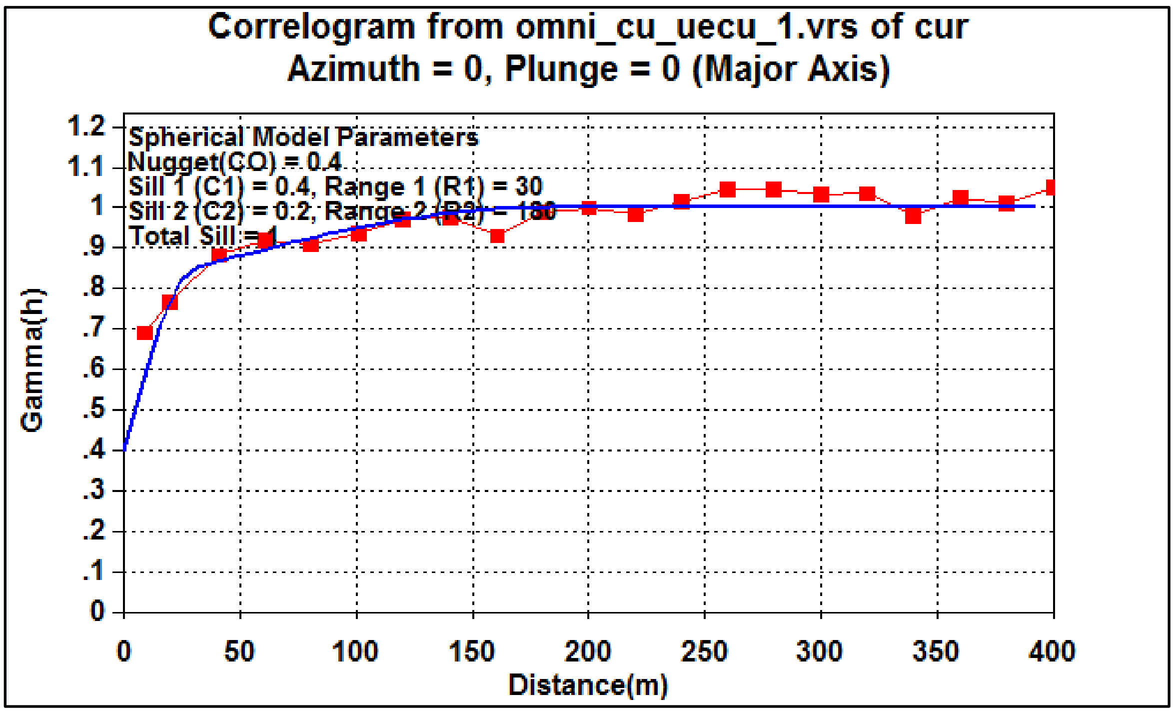
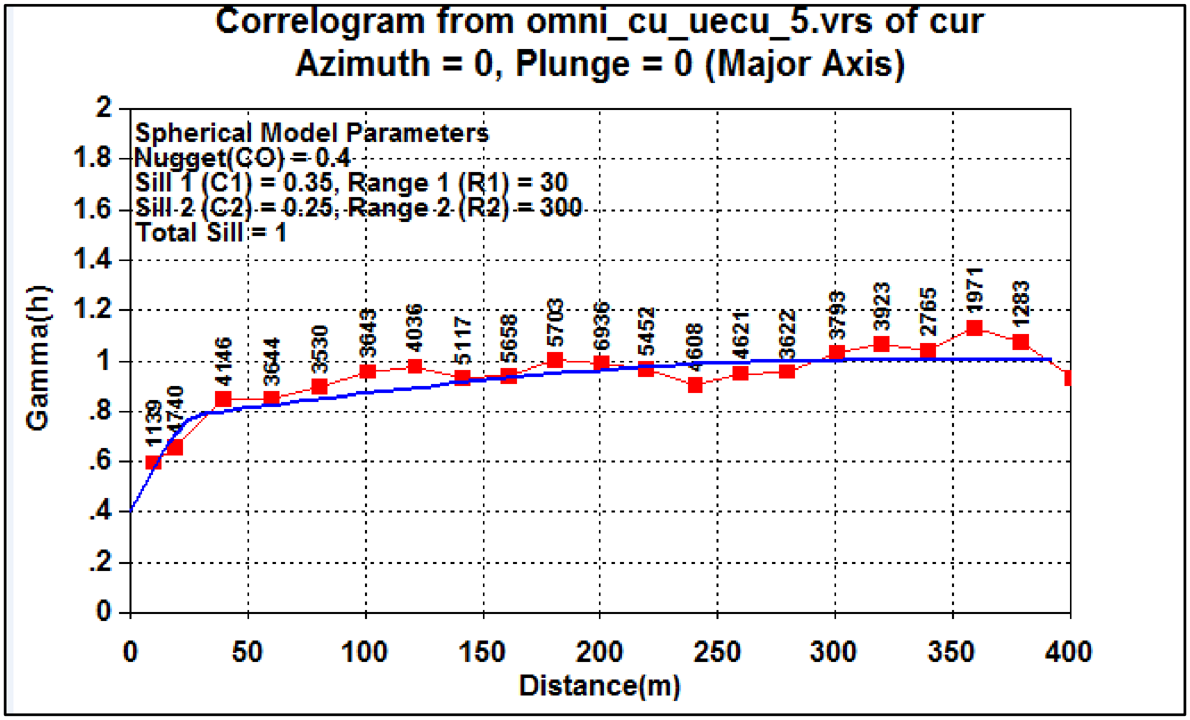


Figure 14-48: Correlogram Copper, EU 5 (Golder, 2019)



14.2.2.2 <u>Block Model Estimate</u>

Block Model Definition

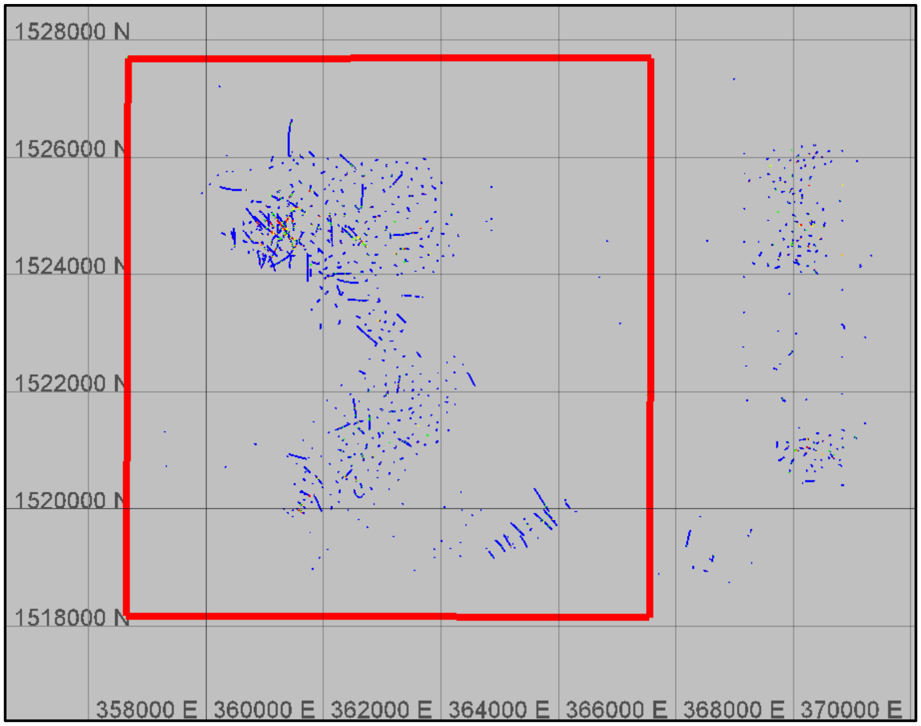
The limits of the block model are show in Table 14-35. Figure 14-49 shows the drill hole distribution that supports the model dimensions. The drill data to the east is related to the Underground Project and is not part of the Open Pit Project model.

It is recommended that a subcell block model approach should be evaluated to include both geological and mining dilution as opposed to the current method which applies all dilution as part of the mining process.

Table 14-35: Block Model Definition

Orientation	**Azimuth**	**Dip**	**Plunge**
	90°	0°	0°
Origin (ft)	**East**	**North**	**Elevation**
	358,665	1,518,165	1,925
Extension (ft)	8,900	9,550	3,150
Parent block size (ft)	50	50	25

Figure 14-49: Block Model Extensions (red line) (Golder, 2019)



Samples Selection Criteria

The estimation was performed using Ordinary Kriging implemented in three nested passes for each EU. The strategy of search and selection of samples is a crucial factor in the estimation process because it has a direct impact on the level of quality and smoothing of the estimate. The proper selection of the kriging parameters will apply the proper smoothing of the exploration data such that it accurately predicts the tons and grade on both a local and global basis.

A set of main schemes that defined different search radius, sample selection strategies, use of octants, outlier control and contact analysis were implemented and analyzed in order to select the appropriate estimation method for each EU and variable.

The kriging plan considered the following criteria and restrictions:

- Discretization block size of 4 ft x 4 ft x 2 ft.
- For the control of outliers, restriction of high grade by High Yield Restriction was made for all the estimations.
- Hard contacts were used in all estimations.
- Composites with a length less than 5.0 ft are not considered in the estimation.
- No octant restrictions were used for any EU.

The sample selection strategy was defined based on the density of information present in the deposit and the variography models defined for each EU.

Table 14-36 summarizes the search radius and general parameters implemented and the selection of samples in each EU and estimation pass.

Table 14-36: Estimation Plans for all Variables

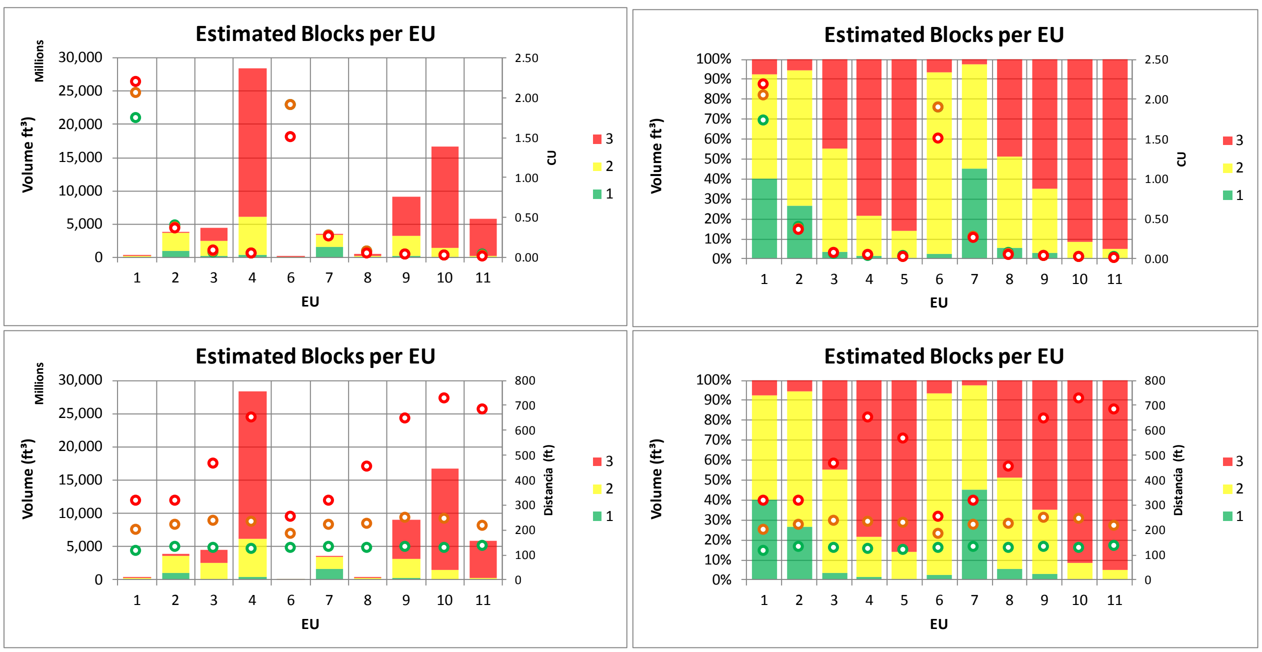
EU	Pass	Type	Axis			Samples		Samples per Drill Hole
			Major	Semi	Minor	Min.	Max.	
All	1	OK	200	200	70	9	16	3
	2	OK	400	400	140	6	16	3
	3	OK	1200	1200	500	2	16	3

Estimation Results

Estimation of all variables was completed using Ordinary Kriging with three estimation passes.

Figure 14-50 shows the estimated blocks percentage and mean grade by passes for each EU.

Figure 14-50: Estimated Blocks Percentage & Mean Grade by Passed for EU – Cu (Golder, 2019)



Analysis of the Samples Selection Schema

An evaluation was performed to determine how many samples were selected for various EUs. The objective of this analysis is to evaluate the implementation of the Kriging estimation plan, i.e. how the estimation plan is considering the number of samples and number of drill holes in every estimation pass. This analysis also allows the definition of the spatial coverage of every block determining whether the estimation was performed by an interpolation or extrapolation process.

Figure 14-51 and Figure 14-52 show the number of samples and drill holes by estimation pass for EU 2 and 6 respectively. The first pass uses between 9 and 12 samples, which implies a more local estimation, while second and third passes use a higher number of samples with a maximum of 16 samples to avoid grade extrapolation. Most of the blocks have been estimated with at least three drill holes, which is considered adequate to allow an acceptable grade interpolation.

Figure 14-51: Number of Samples & Drill Holes per Estimation Pass, EU 2 – Total Copper (Golder, 2019)

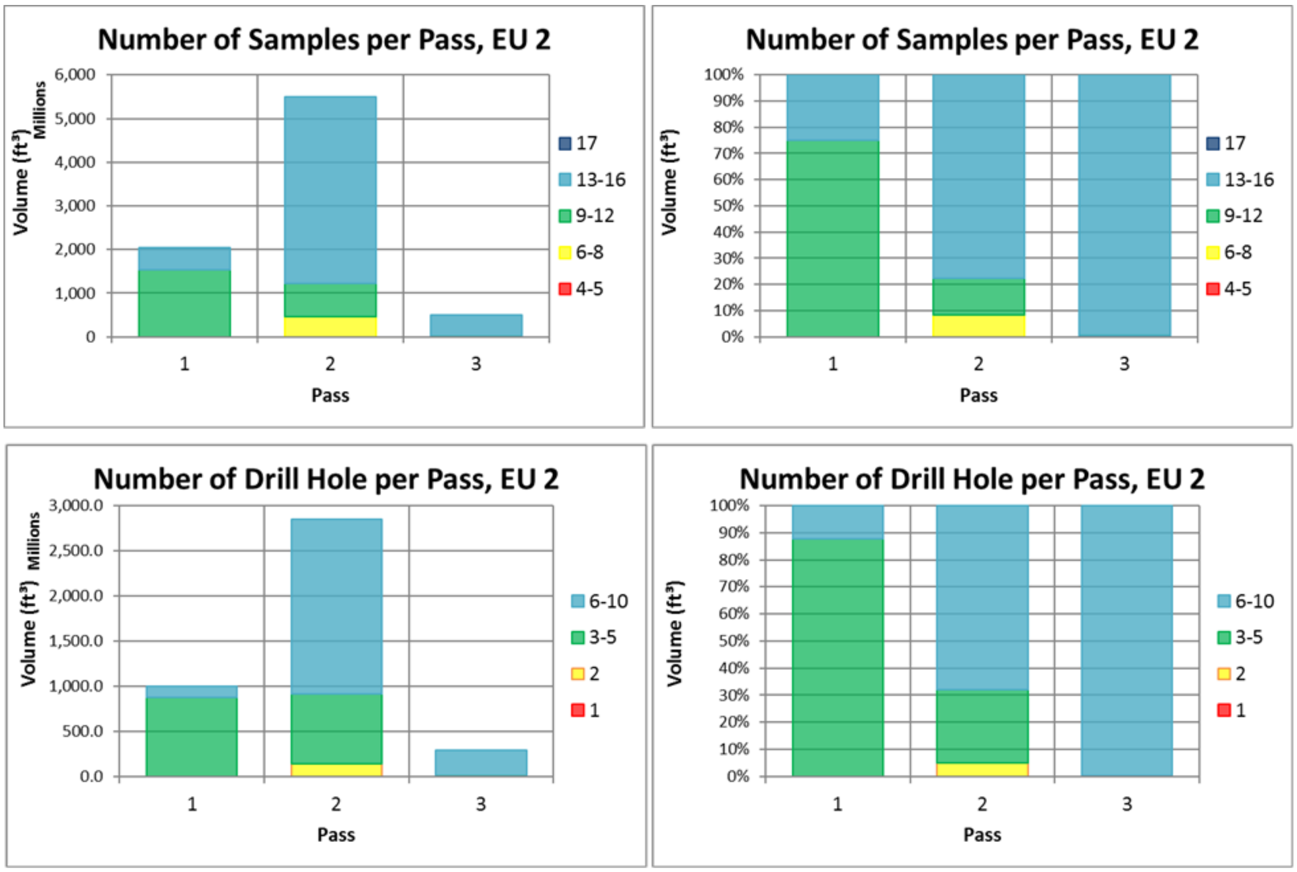
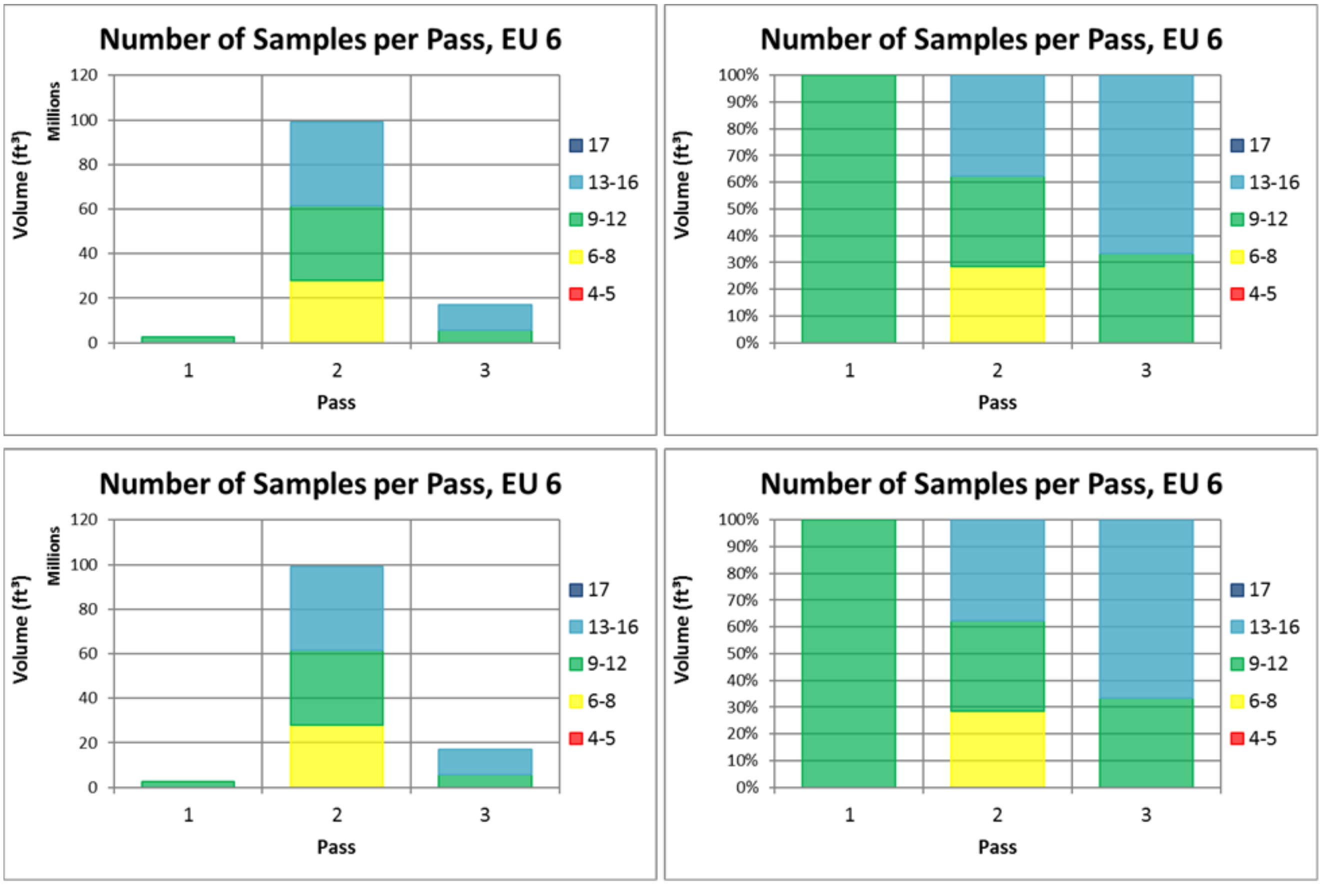


Figure 14-52: Number of Samples & Drill Holes per Estimation Pass, EU 6 – Total Copper (Golder, 2019)



14.2.2.3 Block Model Validation

An independent block model validation was completed to evaluate the performance of the block model to honor the input data. The validation was performed only on the estimated blocks and using the composite database. The result of the grade estimation process for the Underground Mineral Resource model was validated using the following checks:

- Comparison of basic statistics between declustered composite and estimated block grades to validate the global means reproduction
- Swath plots to validate local means reproduction and smoothing degree
- Visual validation of estimated grades versus composites grades

Statistical Comparison

The validation through statistical comparison is considered an accepted practice to verify the ability of the block model to honor the exploration data. Global statistics of the average grades of the composites could be influenced by different factors such as sample density, clustering, and the influence of high grades that have been restricted in the estimation plan.

Global means of estimated blocks were compared to composites declustered means using a Nearest Neighbor weight. Weight for near neighbor was obtained from the neighborhood methodology and considering search radii similar to the search radii used in the estimation process.

The results could be interpreted as follows:

- Average differences around 0% indicates the blocks, on average, represent the composite data
- Average differences lower than 0% (negative), show that the blocks have a lower grade than the composites
- Average differences higher than 0% (positive), show that the blocks have a higher grade than the composites
- Average differences lower than ±10% are considered adequate
- Average differences higher than ±10% should be reviewed

The result of the validation indicates that is no evident bias in the global average. Table 14-37, Table 14-38 and Table 14-39 summarizes the differences in percentage between the estimated blocks with the declustered composites for total copper, gold, and silver, respectively.

The results show the mean is generally within the 10% target and considered adequate. Some EUs present a relative difference greater than 10% but usually is in low grade domains or very small in volume.

Table 14-37: Statistics Comparison by EU for Cu

EU	# Data		Minimum		Maximum		Mean			Std. Dev.	
	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	%Diff	Comp.	Blocks
1	1541	64401	0.0001	0.7904	23.450	5.975	1.706	1.908	11.8%	1.34	0.71
2	9318	646951	0.0001	0.0001	5.249	1.637	0.373	0.373	0.0%	0.32	0.11
3	5,798	323364	0.0001	0.0001	1.422	0.556	0.054	0.062	14.6%	0.06	0.036
4	15,697	563515	0.0001	0.0001	1.514	0.431	0.036	0.039	8.2%	0.05	0.029
5	4,125	96,398	0.0001	0.0001	0.620	0.200	0.018	0.021	18.0%	0.04	0.020
6	215	10,273	0.0600	0.9622	7.980	3.347	1.824	1.892	3.8%	1.15	0.331
7	8,188	399,217	0.0001	0.0242	7.766	2.200	0.271	0.268	-1.0%	0.28	0.116
8	940	41,558	0.0001	0.0110	0.250	0.155	0.058	0.061	6.1%	0.03	0.015
9	5,132	255,964	0.0001	0.0006	0.559	0.197	0.032	0.029	-9.4%	0.04	0.018
10	2,700	125,563	0.0001	0.0002	0.221	0.125	0.016	0.018	14.4%	0.02	0.016
11	1,177	31,165	0.0001	0.0004	0.391	0.135	0.011	0.012	3.5%	0.02	0.012

Table 14-38: Statistics Comparison by EU for Au

EU	# Data		Minimum		Maximum		Mean			Std. Dev.	
	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	%Diff	Comp.	Blocks
1	1541	61376	0.00002	0.00118	0.054	0.025	0.0047	0.0047	0.4%	0.0051	0.0020
2	9318	645779	0.00001	0.00001	0.286	0.041	0.0013	0.0013	-0.6%	0.0033	0.0009
3	5,798	387688	0.00001	0.00001	0.017	0.003	0.0003	0.0003	-6.9%	0.0006	0.0002
4	15,697	563552	0.00001	0.00001	0.103	0.012	0.0002	0.0002	3.2%	0.0010	0.0002
5	4,125	96,403	0.00001	0.00001	0.005	0.001	0.0001	0.0001	47.9%	0.0002	0.0001
6	215	10,273	0.00038	0.00139	0.034	0.017	0.0060	0.0065	7.0%	0.0053	0.0018
7	8,188	399,217	0.00001	0.00015	0.053	0.023	0.0012	0.0011	-3.1%	0.0017	0.0007
8	940	41,051	0.00001	0.00004	0.005	0.001	0.0003	0.0003	-5.6%	0.0004	0.0001
9	5,132	255,938	0.00001	0.00001	0.005	0.001	0.0002	0.0002	-6.7%	0.0003	0.0001
10	2,700	125,563	0.00001	0.00001	0.047	0.001	0.0001	0.0001	-2.2%	0.0009	0.0001
11	1,177	31,284	0.00001	0.00001	0.007	0.001	0.0001	0.0001	6.6%	0.0003	0.0001

Table 14-39: Statistics comparison by EU for Ag

EU	# Data		Minimum		Maximum		Mean			Std. Dev.	
	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	Comp.	Blocks	%Diff	Comp.	Blocks
1	1541	61376	0.001	0.040	1.269	0.398	0.148	0.151	1.4%	0.108	0.042
2	9318	645779	0.001	0.001	5.709	0.991	0.051	0.049	-2.8%	0.079	0.026
3	5,798	323704	0.001	0.001	0.386	0.121	0.010	0.011	16.0%	0.015	0.008
4	15,697	563454	0.001	0.000	0.554	0.402	0.009	0.009	-1.8%	0.023	0.011
5	4,125	96,312	0.001	0.001	0.132	0.043	0.004	0.004	1.8%	0.008	0.003
6	215	10,273	0.008	0.049	0.795	0.587	0.161	0.217	34.5%	0.146	0.086
7	8,188	399,217	0.001	0.004	3.716	0.513	0.035	0.034	-2.5%	0.056	0.022
8	940	54,913	0.001	0.001	0.125	0.075	0.013	0.012	-9.9%	0.015	0.010
9	5,132	255,808	0.001	0.001	1.461	0.376	0.006	0.006	-9.7%	0.022	0.007
10	2,700	125,556	0.001	0.001	3.420	0.655	0.006	0.005	-2.5%	0.066	0.021
11	1,177	31,217	0.001	0.001	0.060	0.014	0.003	0.004	19.0%	0.005	0.003

Figure 14-53, Figure 14-54 and Figure 14-55, show for each EU a comparison between the sample grade mean (orange circle) and the block model grade (blue point) for total copper, gold and silver, respectively. Each column is referring to the number of composites (orange) and estimated blocks (blue). In general, is observed a good consistency for each EU, and no excessive bias in the global mean. The differences in the mean grades for the blocks are under 10%, which is considered adequate for the reproduction of the global average by EU with the exception for some EU relating to molybdenum due to variable values being too small.

Figure 14-53: Chart Global Statistics Comparison – Copper (Golder, 2019)

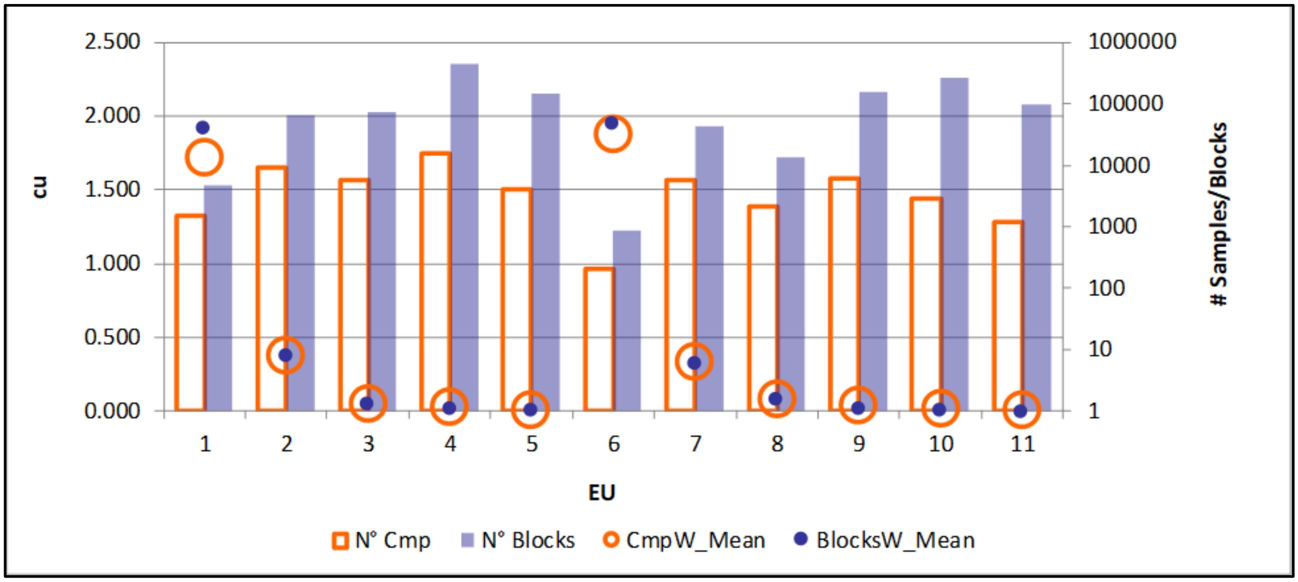


Figure 14-54: Chart Global Statistics Comparison – Gold (Golder, 2019)

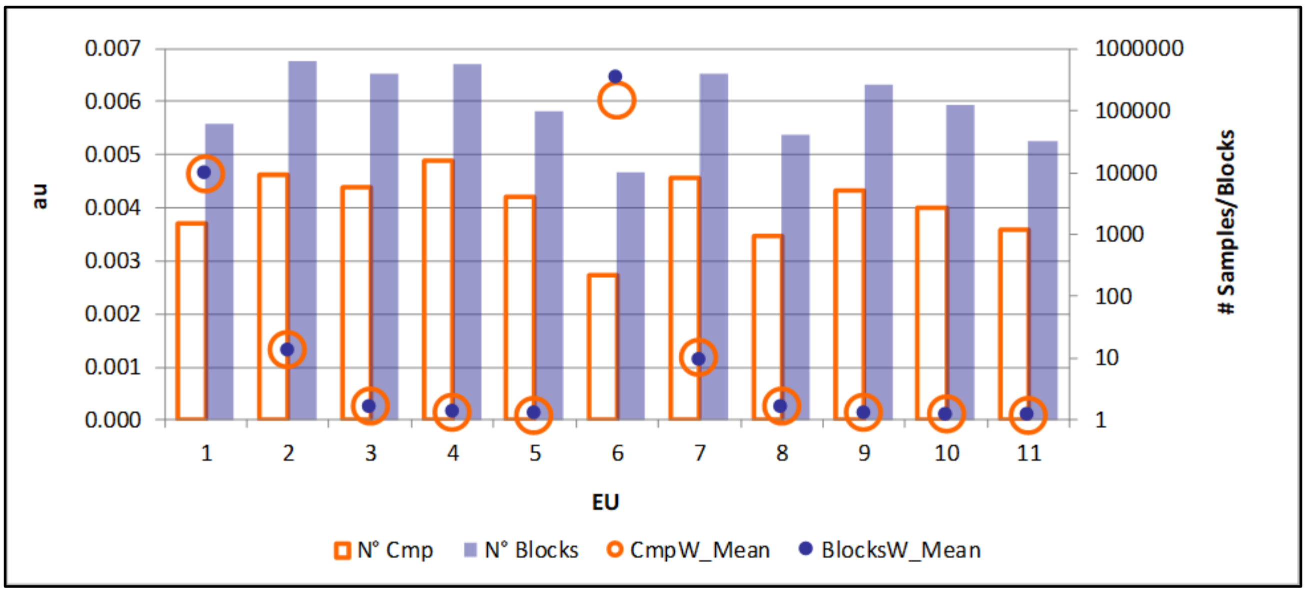
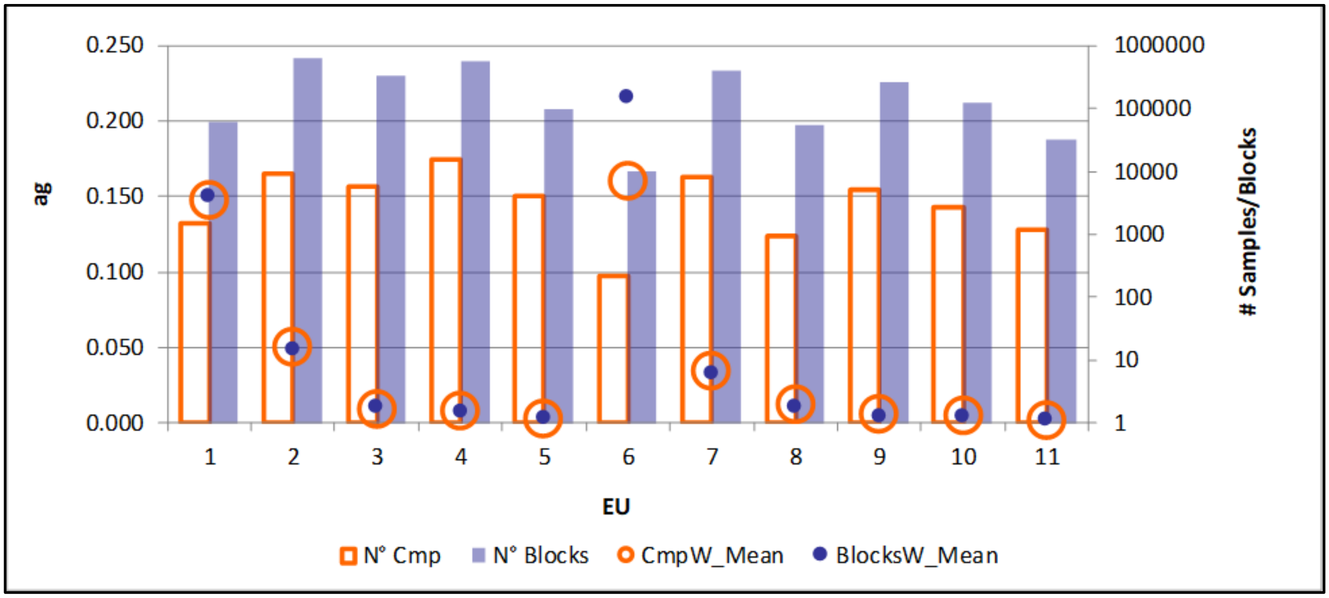


Figure 14-55: Chart Global Statistics Comparison – Silver (Golder, 2019)



Swath Plots

The EUs should have a close relationship with the drill hole composite data used for estimation. The swath plots are useful for assessing average grade conformance, and to detect any obvious interpolation issues. The process involved averaging both the blocks and samples grades within panels of 200 ft E by 200 ft N by 100 ft RL. The result of the validation is also presented as scatter and Q-Q plots, allowing an assessment of the smoothing degree. In the two plots samples distribution should be close to the 45° line, meaningful deviations indicate potential smoothing and/or bias product of the estimation.

The criteria to define the bias on swath plots are the following:

- Respect to bias presence:
  - Positive difference: the block model estimation regularly presents a greater mean grade than the composites.
  - Negative difference: the block model estimation regularly presents a lower mean grade than the composites.
  - No bias: the block model estimation regularly presents a similar mean grade than the composites.
- Respect to smoothing degree:
  - Non-adequate: EUs present higher smoothing as expected. The stationary of the EU, drifts and variography analysis in conjunction with the samples strategy, should be reviewed for these EUs.

- o Regular: the estimation of the block model trends towards a specific direction follow the composites, though bias on the composites is observed.
- o Adequate: the estimation of the block model trends towards east, north and elevation directions follow the composites, and no bias on the composites is observed.

Table 14-40, Table 14-41 and Table 14-42 show a summary of swath plot observations for total copper, gold and silver, respectively. Validation through swath plots was made only for the first three passes. These results shows that the estimated block model reasonably follows the composite trend. As an example, Figure 14-56 shows the results of this analysis for EU 1 for total copper.

Table 14-40: Summary of Swath Plot Observations – Copper

Domain	Bias	Smoothing Degree	Global Trend Reproduction	Local Trend Reproduction
1	No difference	Regular	Suitable	Regular
2	Slight Negative difference	Suitable	Suitable	Suitable
3	No difference	Suitable	Suitable	Suitable
4	No difference	Suitable	Suitable	Suitable
5	No difference	Suitable	Suitable	Suitable
6	Slight Positive difference	Regular	Suitable	Regular
7	No difference	Suitable	Suitable	Regular
8	No difference	Suitable	Suitable	Suitable
9	No difference	Suitable	Suitable	Suitable
10	No difference	Suitable	Suitable	Regular
11	Slight Positive difference	Suitable	Suitable	Regular

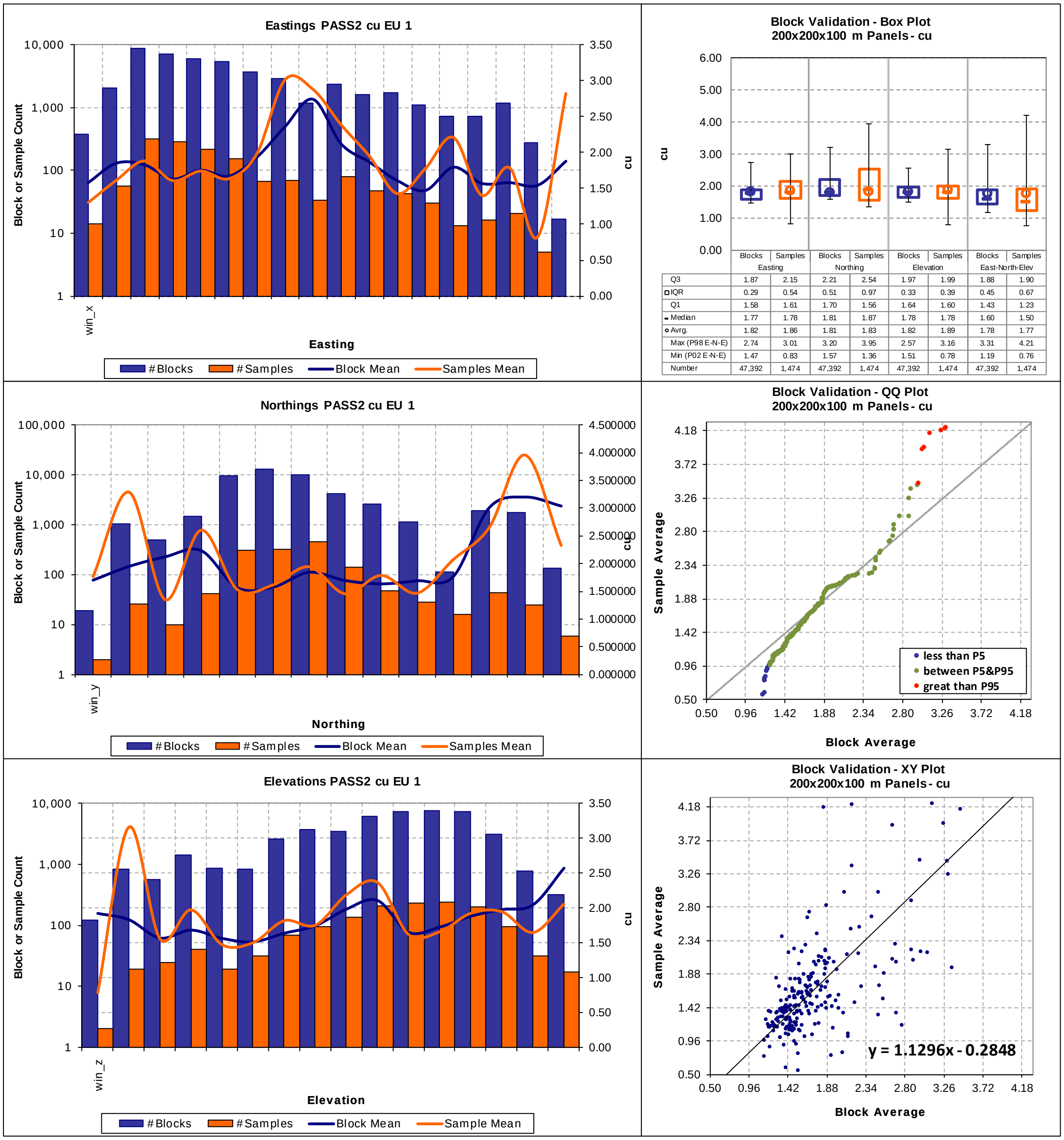
Table 14-41: Summary of Swath Plot Observations – Gold

Domain	Bias	Smoothing Degree	Global Trend Reproduction	Local Trend Reproduction
1	No difference	Regular	Suitable	Regular
2	Slight Negative difference	Suitable	Suitable	Suitable
3	Slight Positive difference	Suitable	Suitable	Regular
4	No difference	Suitable	Suitable	Regular
5	No difference	Suitable	Suitable	Suitable
6	No difference	Regular	Regular	Regular
7	No difference	Suitable	Suitable	Regular
8	No difference	Regular	Regular	Regular
9	No difference	Regular	Suitable	Regular
10	Slight Positive difference	Regular	Regular	Regular
11	No difference	Regular	Suitable	Regular

Table 14-42: Summary of Swath Plot Observations – Silver

Domain	Bias	Smoothing degree	Global trend reproduction	Local trend reproduction
1	No difference	Regular	Suitable	Regular
2	Slight Negative difference	Suitable	Suitable	Suitable
3	Slight Positive difference	Suitable	Suitable	Suitable
4	No difference	Suitable	Suitable	Suitable
5	Slight Positive difference	Suitable	Suitable	Suitable
6	Slight Positive difference	Regular	Suitable	Suitable
7	No difference	Suitable	Suitable	Regular
8	No difference	Suitable	Suitable	Suitable
9	No difference	Suitable	Suitable	Suitable
10	No difference	Suitable	Suitable	Suitable
11	Slight Positive difference	Regular	Suitable	Regular

Figure 14-56: Swath Plot EU 1 – Copper (Golder, 2019)



	Easting		Northing		Elevation		East-North-Elev	
	Blocks	Samples	Blocks	Samples	Blocks	Samples	Blocks	Samples
Q3	1.87	2.15	2.21	2.54	1.97	1.99	1.88	1.90
IQR	0.29	0.54	0.51	0.97	0.33	0.39	0.45	0.67
Q1	1.58	1.61	1.70	1.56	1.64	1.60	1.43	1.23
Median	1.77	1.78	1.81	1.87	1.78	1.78	1.60	1.50
Avrg.	1.82	1.86	1.81	1.83	1.82	1.89	1.78	1.77
Max (P98 E-N-E)	2.74	3.01	3.20	3.95	2.57	3.16	3.31	4.21
Min (P02 E-N-E)	1.47	0.83	1.57	1.36	1.51	0.78	1.19	0.76
Number	47,392	1,474	47,392	1,474	47,392	1,474	47,392	1,474

Visual Validation

Visual validation was performed in sections along each coordinate axis. Estimates and composites were compared using the same color scheme to identify visually if problems of negative or positive difference occurred.

In general, the visual validation for total copper estimates indicates that the composite grades are adequately represented by the block model. High grades zones are adequately represented, high grade samples are adequately controlled, validating the outliers treatment applied. Smoothing levels increase in deeper parts of the deposit due to the reduction in the number of composites available. However, the results show an acceptable level of smoothing. Figure 14-57, Figure 14-58 and Figure 14-59 show examples of visual validation for the total copper estimated model in plan view, east section and north section, respectively.

Figure 14-57: Visual Validation – Total Copper Section 362,920 East (Golder, 2019)

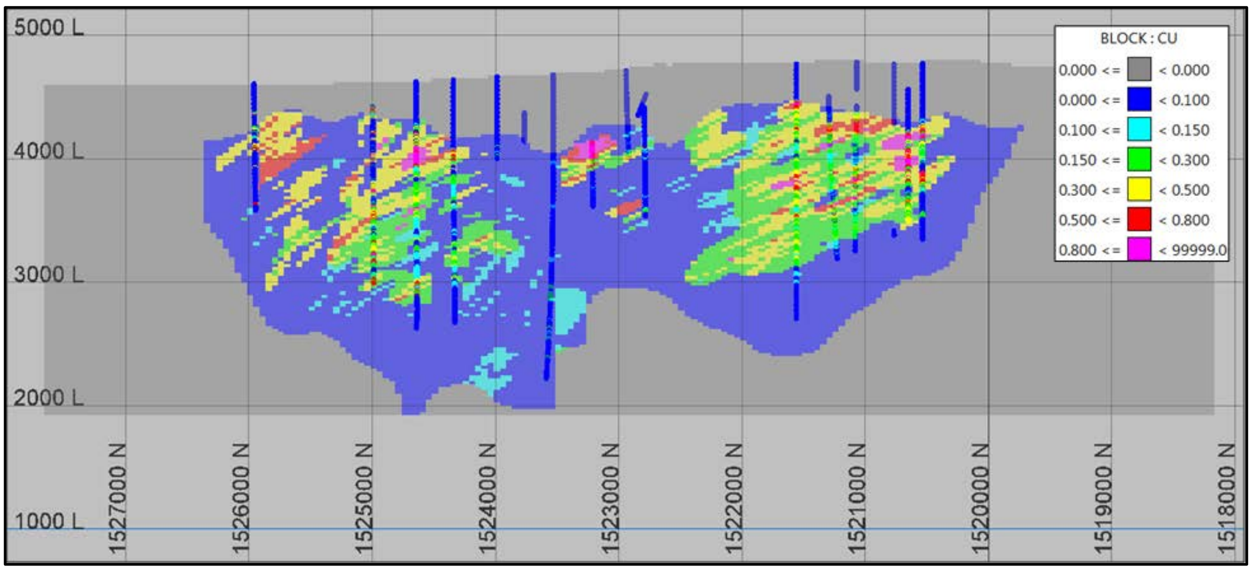


Figure 14-58: Visual Validation – Total Copper, Section 363,280 East (Golder, 2019)

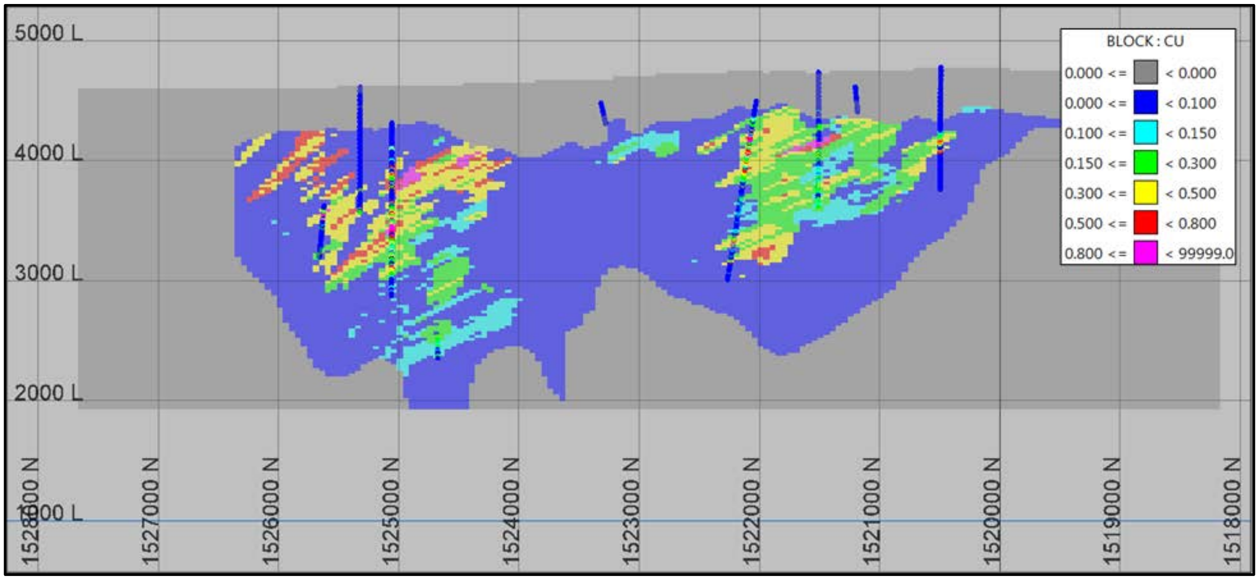
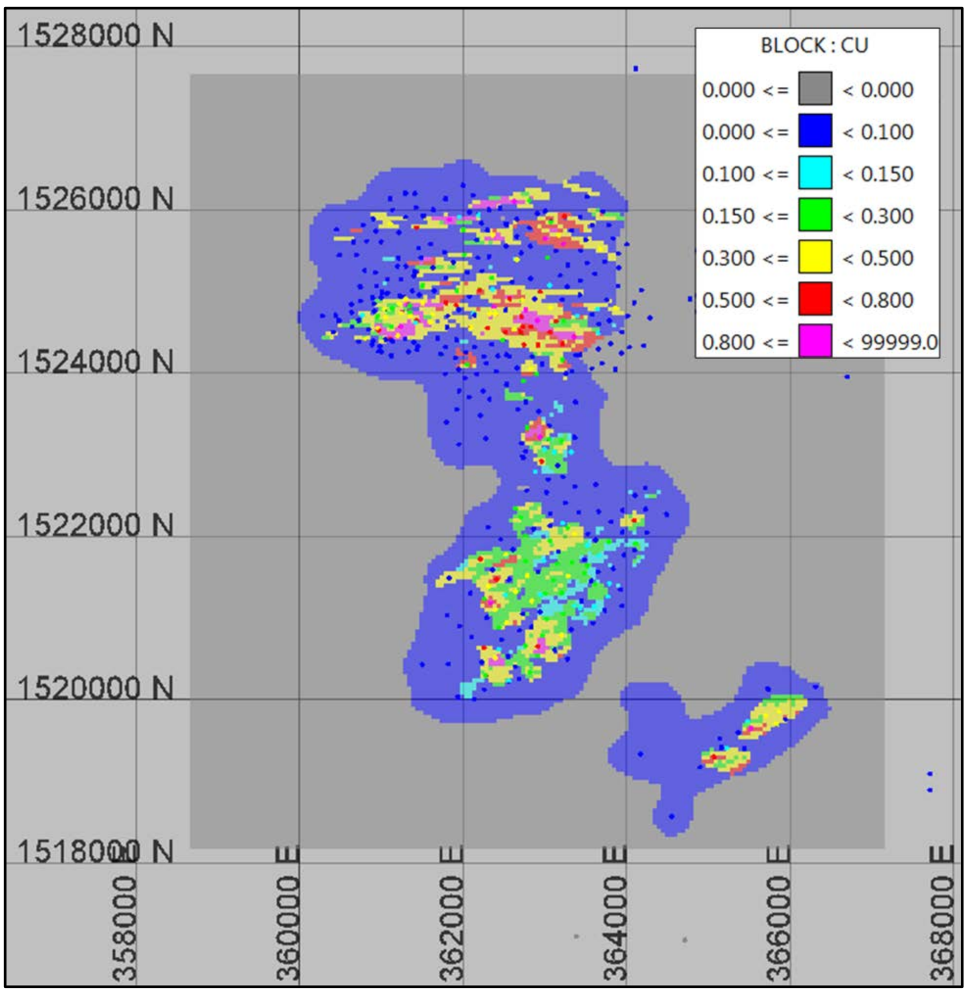


Figure 14-59: Visual Validation – Total Copper, Section 4000 ft (Golder, 2019)



14.2.3 Density

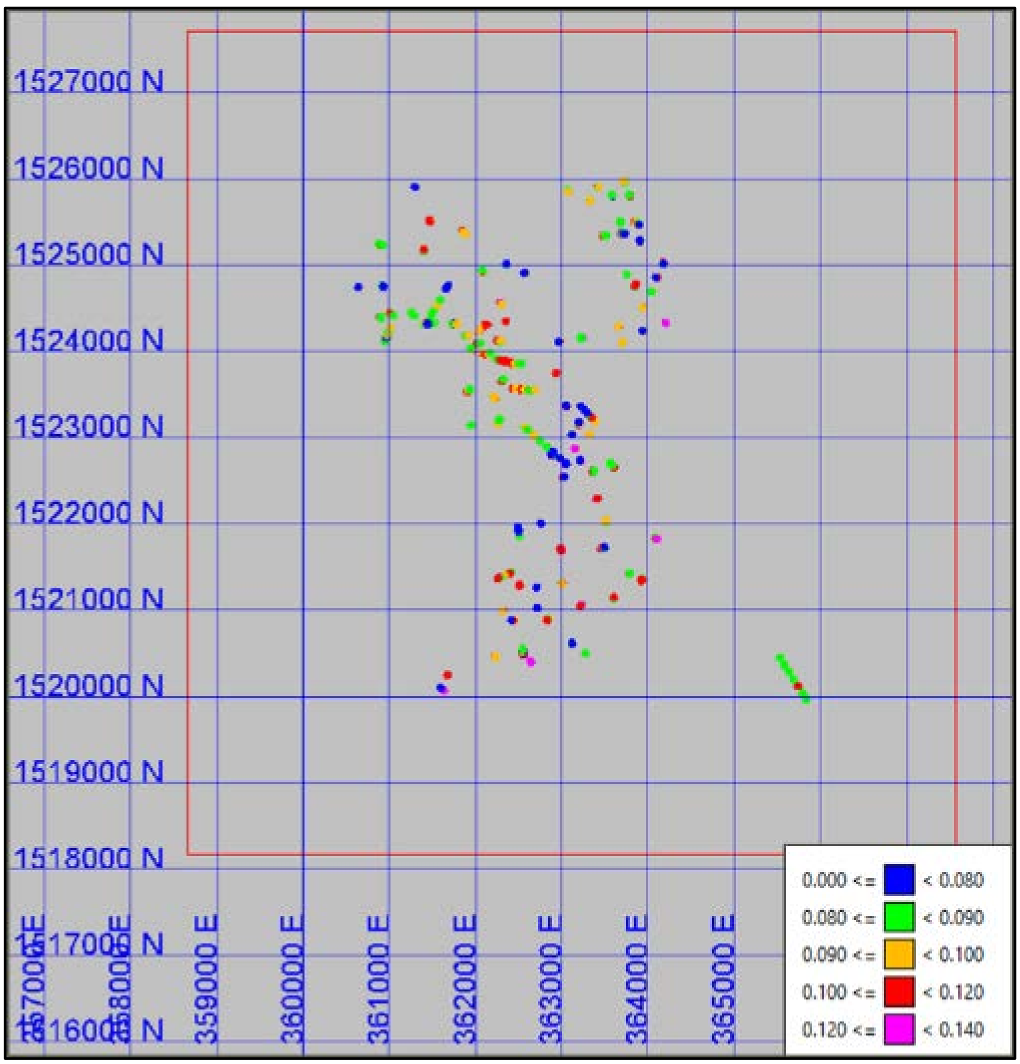
The drill hole database includes updated data on density for the 2018 Resource Model. Table 14-43 provides a summary of the data used in the density estimation process. The database has 125 drill holes and a global mean density value of 0.095 t/ft³. Additional density samples outside of the model extents were not used in the estimation. Drill holes were composited in Vulcan® using the "straight" method using, which composites at the original drill hole sample. The geological codes have been back flagged from the block model.

Figure 14-60 shows the spatial distribution of density points of observation.

Table 14-43: Estimation Database – Density

Samples	Average	Drill Holes
920	0.095	125

Figure 14-60: Spatial Distribution of Density Data (Golder, 2019)



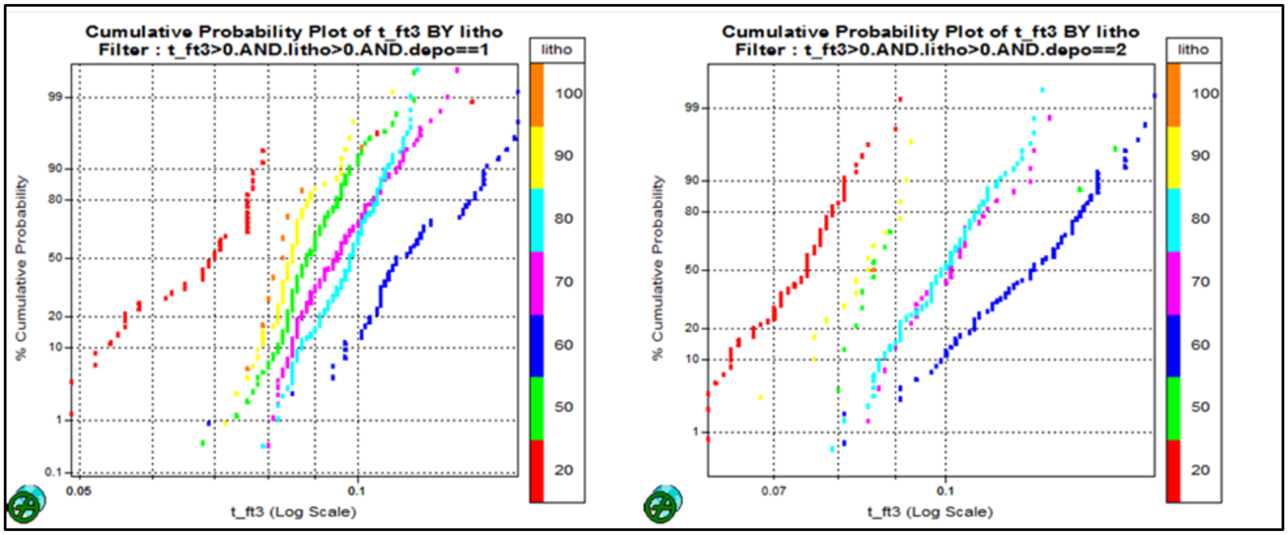
14.2.3.1 Exploration Data Analysis

The exploration density data analysis conducted was similar to that described in Item 14.2.2.1.

Estimation Unit Definition

To determine the geologic drivers controlling the density, cumulative probability plots for, lithology, and grade shell were generated. Figure 14-61 shows the database differentiated by lithology and grade shell. For density, the main control is determined by lithology.

Figure 14-61: Cumulative Probability Plot for Density by, Lithology, & Grade Shell – Density (Golder, 2019)



The definition of density estimation units is summarized in Table 14-44. Basic statistics are provided in Table 14-45 and Table 14-46 for the North Deposit and the South Deposit, respectively

Table 14-44: Estimation Units Definition – Density

EU	Lithology
10	Gravel & Younger Alluvium
20	Tertiary Volcanics – rhyolite tuffs, andesite dikes
40	Marbles
50	Hornfels – includes argillites, siltstones
60	Magnetite Skarn
70	Silicate Skarn
80	Endoskarn – Strongly altered granodiorite
90	Intrusive – granodiorite, diorite, quartz monzonite porphyry, granodiorite porphyry
100	Talc

Table 14-45: Basic Statistics per Estimation Unit, Density, North Deposit

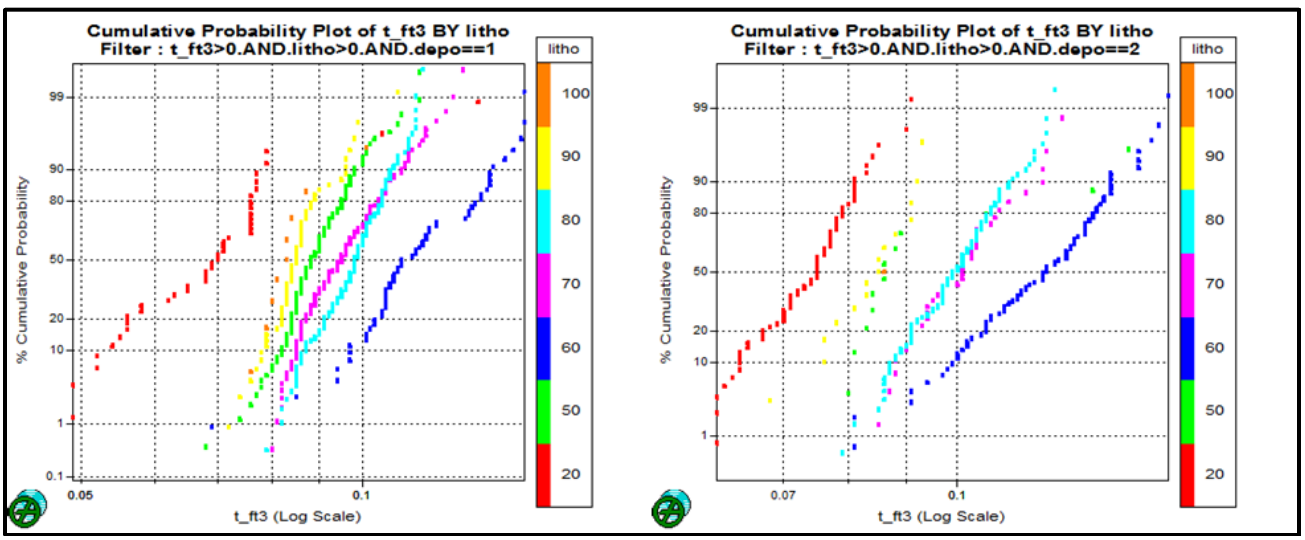
Litho	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev.	CV
Total	592	0.0490	0.1490	0.0940	0.0930	0.0850	0.1010	0.0143	0.1518
20	41	0.0490	0.1330	0.0696	0.0700	0.0580	0.0760	0.0147	0.2113
50	130	0.0680	0.1150	0.0895	0.0880	0.0850	0.0940	0.0074	0.0831
60	59	0.0690	0.1490	0.1154	0.1100	0.1060	0.1300	0.0166	0.1438
70	144	0.0800	0.1280	0.0961	0.0950	0.0880	0.1020	0.0099	0.1026
80	150	0.0790	0.1160	0.0981	0.0980	0.0930	0.1040	0.0077	0.0782
90	59	0.0720	0.1090	0.0856	0.0850	0.0823	0.0870	0.0064	0.0747
100	9	0.0760	0.1010	0.0838	0.0830	0.0798	0.0848	0.0068	0.0809

Table 14-46: Basic Statistics per Estimation Unit, Density, South Deposit

Litho	Samples	Min.	Max.	Mean	Median	Q1	Q3	Std. Dev.	CV
Total	328	0.0610	0.1540	0.0973	0.0970	0.0825	0.1090	0.0194	0.1993
20	68	0.0610	0.0910	0.0738	0.0750	0.0700	0.0780	0.0067	0.0912
40	19	0.0770	0.0900	0.0831	0.0830	0.0800	0.0858	0.0036	0.0431
50	12	0.0800	0.1420	0.0951	0.0860	0.0835	0.0975	0.0199	0.2093
60	79	0.0810	0.1540	0.1192	0.1200	0.1070	0.1300	0.0158	0.1323
70	34	0.0850	0.1240	0.1012	0.1010	0.0940	0.1070	0.0100	0.0992
80	100	0.0790	0.1220	0.0998	0.0995	0.0925	0.1050	0.0091	0.0910
90	15	0.0680	0.0930	0.0837	0.0850	0.0788	0.0903	0.0068	0.0813
100	1	0.0860	0.0860	0.0860	0.0860	0.0860	0.0860	0.0000	0.0000

Figure 14-62 shows the cumulative probability plot for density EUs.

Figure 14-62: Cumulative Probability Plot, Estimation Units – Density (Golder, 2019)



Outlier Treatment

No control was implemented for the treatment of anomalous samples. Density distribution by EU does not show clear discontinuities or changes on the slope (Figure 14-62).

Contact Analysis

The low amount of data by EU does not allow for perform a contact analysis. Due to this, hard boundaries were considered between all EUs.

Spatial Correlation & Variography

The low amount of data by EU does not allow for variogram modeling for any EU. Due to this, an inverse distance squared estimation method was employed.

14.2.3.2 Block Model Estimate

The estimation of density for the Open Pit Project has been conducted using inverse distance squared in two nested passes for all EUs. Non-estimated blocks have been assigned to the mean by EU and deposit (Table 14-45 and Table 14-46).

Estimation Plan

The sample selection strategy for density considered a generic estimation plan for all EUs (Table 14-47).

Table 14-47: Density Estimation Plan

Pass	Type	Bearing	Plunge	Dip	Axis			Samples		Max. Samples per Drill Hole
					Major	Semi Major	Minor	Min.	Max.	
1	ID²	0	0	0	500	500	500	6	12	3
2	ID²	0	0	0	1000	1000	100	8	16	4

14.2.3.3 Block Model Validation

The set of implemented validations is similar to that described in Item 14.2.2.3.

Swath Plots

Definitions, calculation parameters, and assessment criteria used in the validation with swath plots are the same as those detailed above.

Table 14-48 provides the summary of swath plots for density EUs. The validation has been carried out considering the block estimation up to the second pass. In general, results show that estimates reasonably follow the trends observed in the deposit’s density variability at a local level. Figure 14-63 and Figure 14-64 show the result of this analysis for EUs 60 and 70, respectively.

Table 14-48: Summary of Swath Plots per Estimation Unit – Density

Domain	Bias	Smoothing	Global Trend Reproduction	Local Trend Reproduction
20	None	Suitable	Suitable	Suitable
50	None	Suitable	Suitable	Suitable
60	None	Suitable	Suitable	Suitable
70	None	Suitable	Suitable	Suitable
80	None	Suitable	Suitable	Suitable

Figure 14-63: Swath Plots for EU 60 – Density (Golder, 2019)

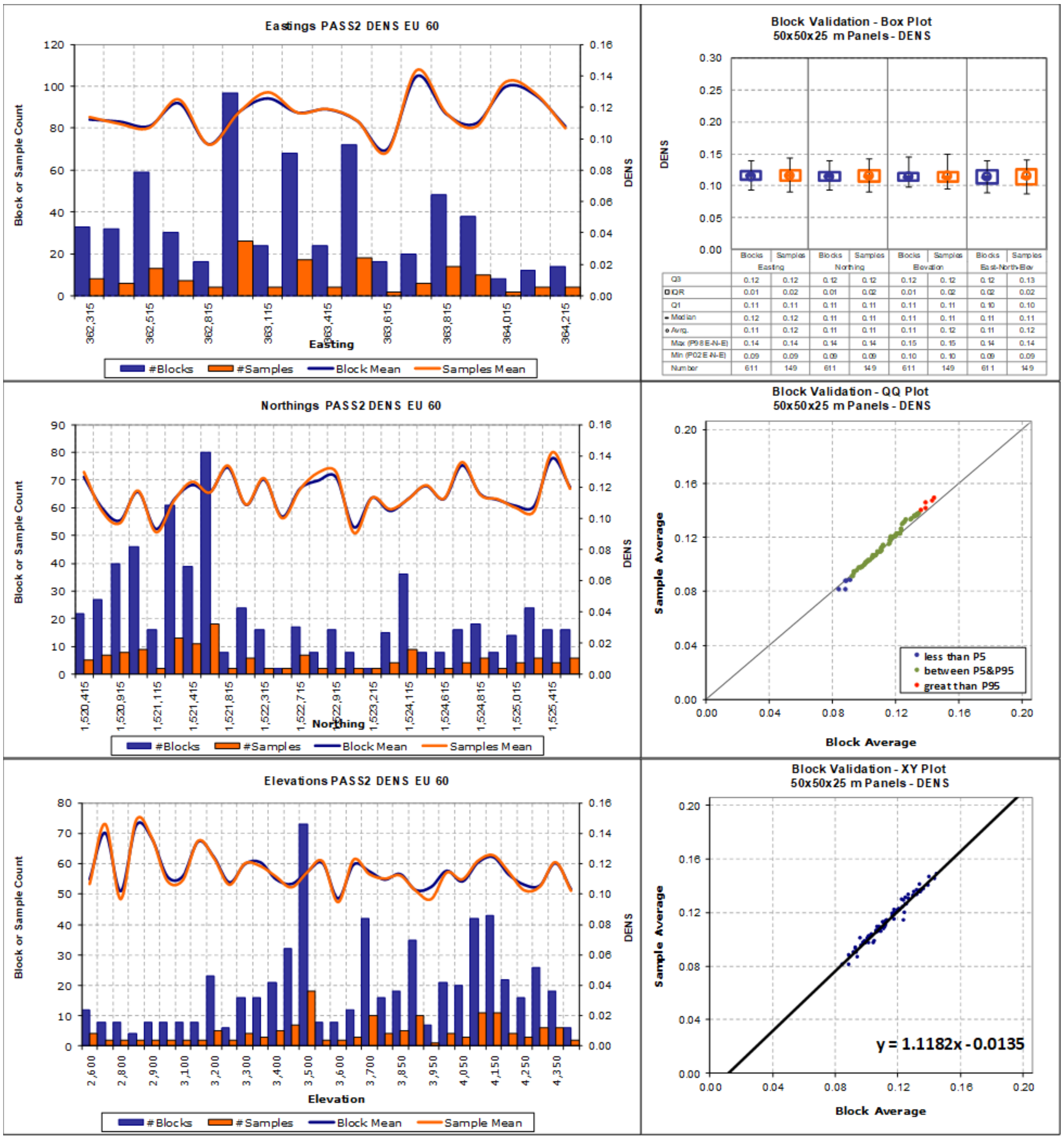
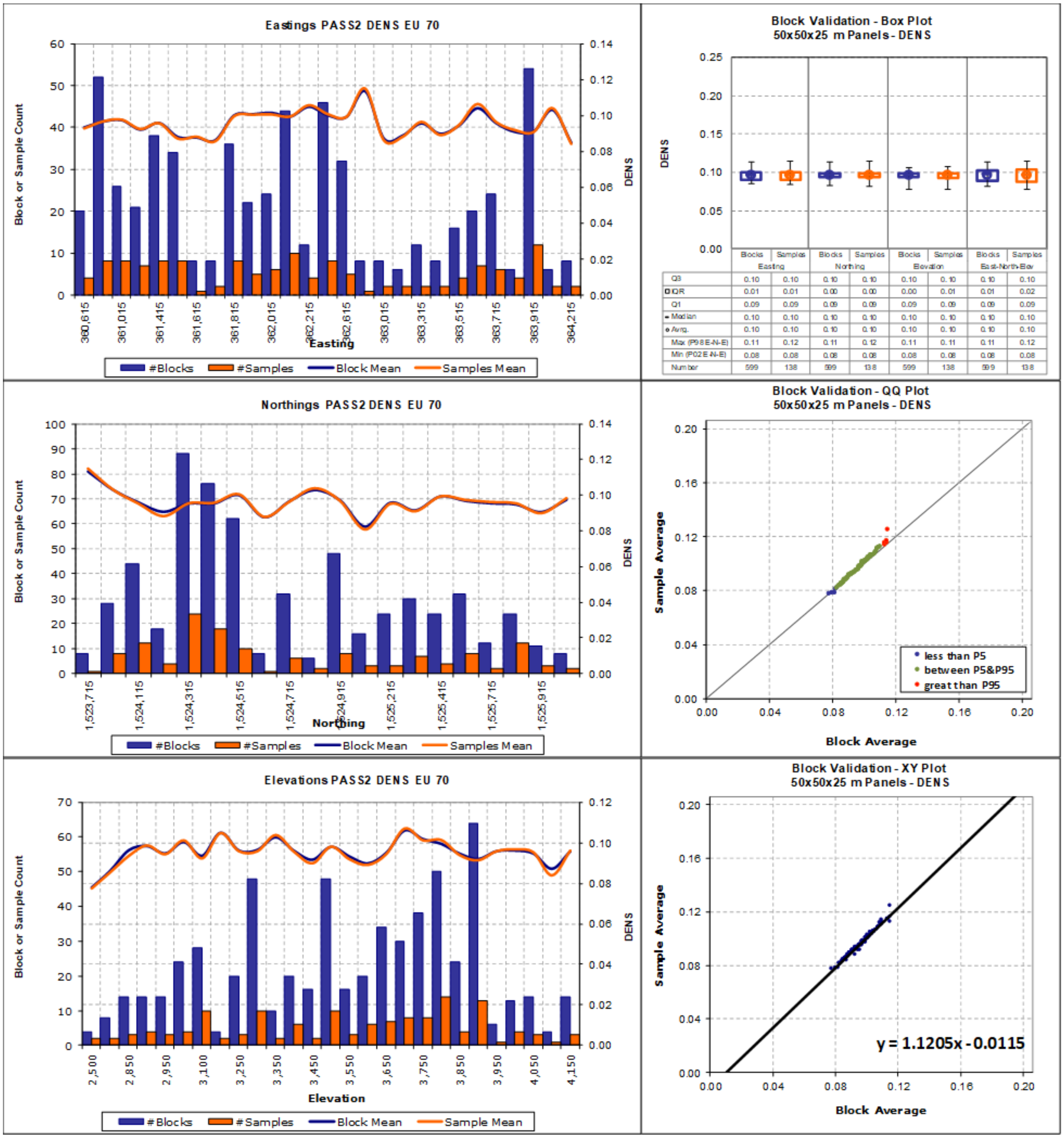


Figure 14-64: Swath Plot for EU 70 – Density (Golder, 2019)



	Blocks	Samples	Blocks	Samples	Blocks	Samples	Blocks	Samples
	Easting		Northing		Elevation		East-North-Elev	
Q3	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
IQR	0.01	0.01	0.00	0.00	0.00	0.01	0.01	0.02
Q1	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09
Median	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Avrg.	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Max (P98 E-N-E)	0.11	0.12	0.11	0.12	0.11	0.11	0.11	0.12
Min (P02 E-N-E)	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08
Number	599	138	599	138	599	138	599	138

Visual Analysis

To visually validate the density estimation, Golder conducted a review of a set of cross sectional and plan views. The validation shows a suitable representation of samples in blocks. Locally, the blocks match the estimation composites both in cross section and plant views.

In general, there is agreement between composite data and block model data for densities.

Figure 14-65 and Figure 14-66 show cross section and plan views for the density model.

Figure 14-65: Visual Analysis for Density – Section 362980 ± 40 (Golder, 2019)

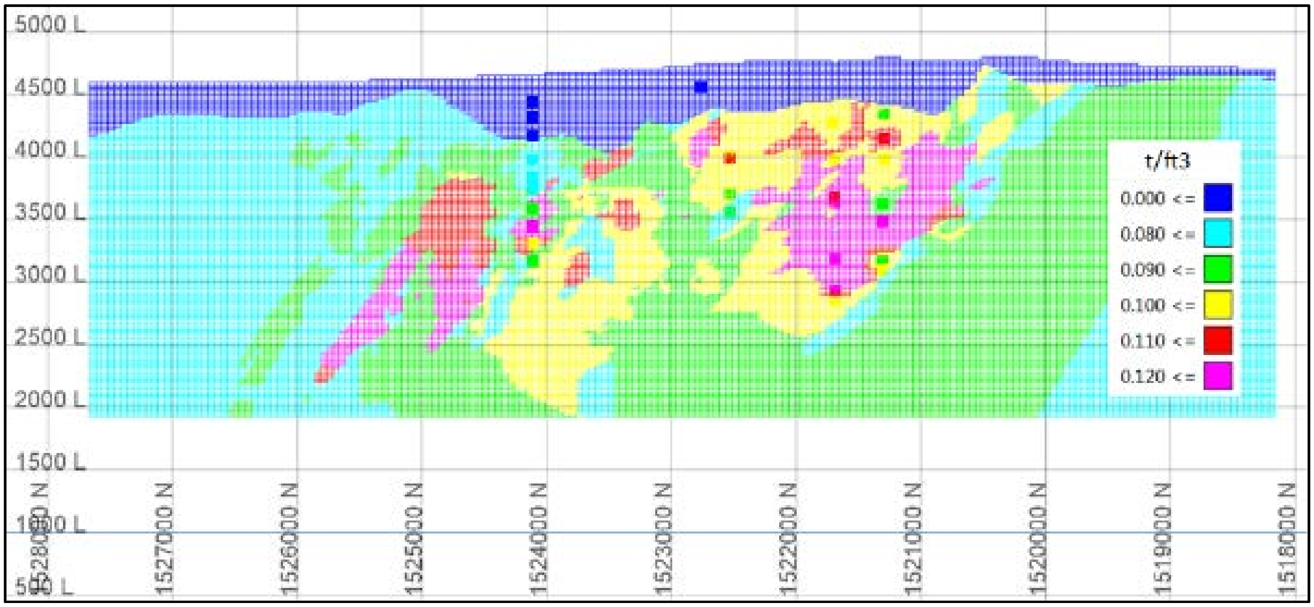
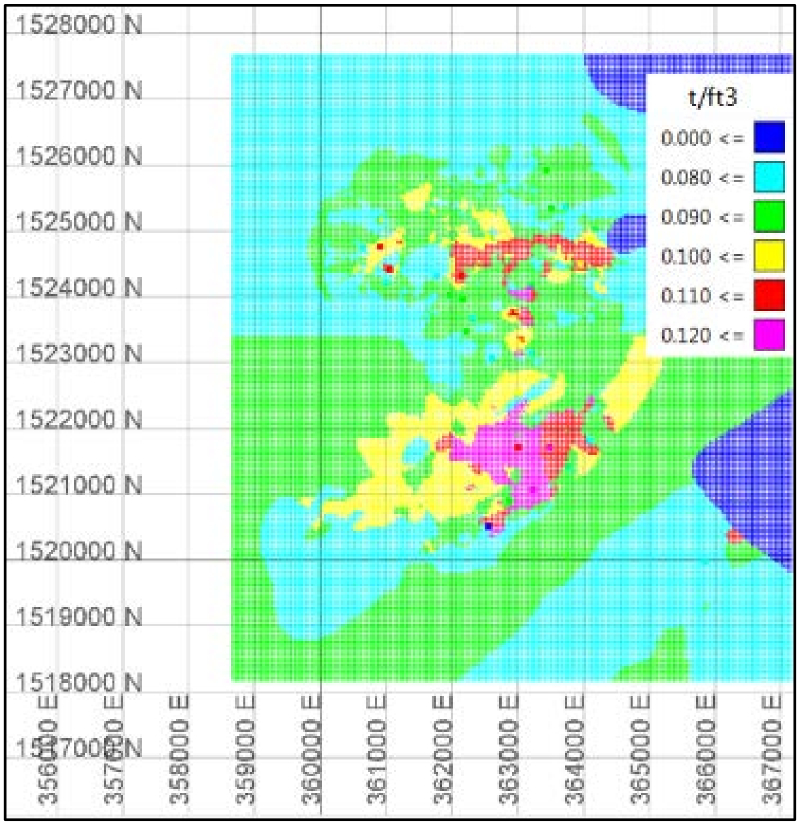


Figure 14-66: Visual Analysis for Density – Plan View at 3,680 level ± 40 (Golder, 2019)

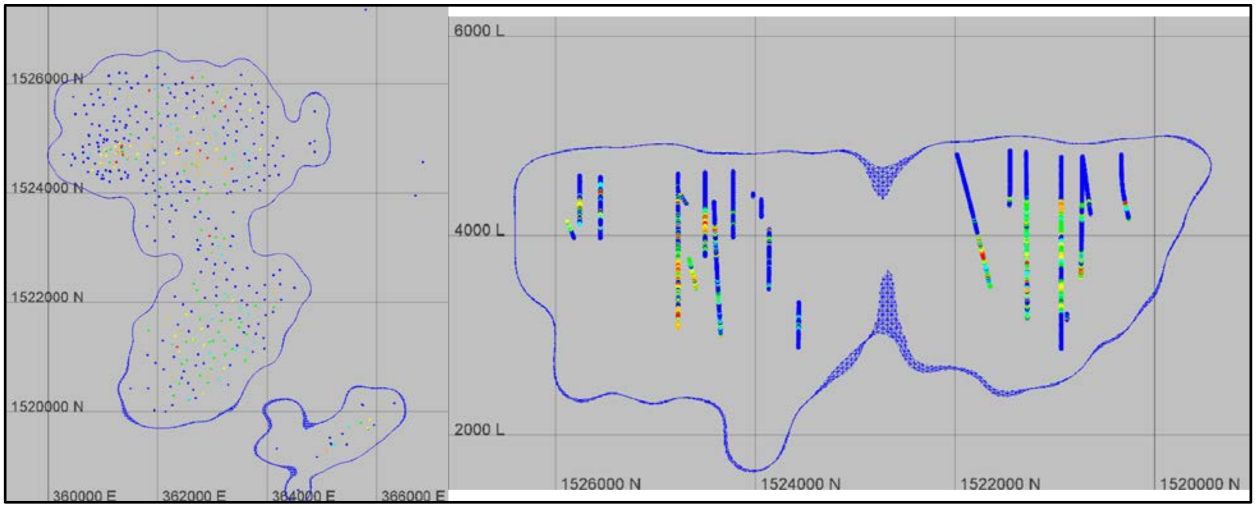


14.2.4 Post Processes

As previously mentioned in Items 14.2.1.2 and 14.2.1.4 some variables of the block model were set to a default value. The cases where defaults were applied as follows:

- Blocks lying in the oxide and transition zones (oxstate 1 or 2)
- Blocks lying in lithologies 10, 20 and 30 (even if they are inside grade shells)
- Blocks lying outside the limit of geological information, this triangulation was generated to avoid extrapolation of grade too far past drill holes (Figure 14-67)

Figure 14-67: Limit of Geological Information Triangulation (Golder, 2019)



Note: Elevation 4,000 ft (left) and section 362500 E (right).

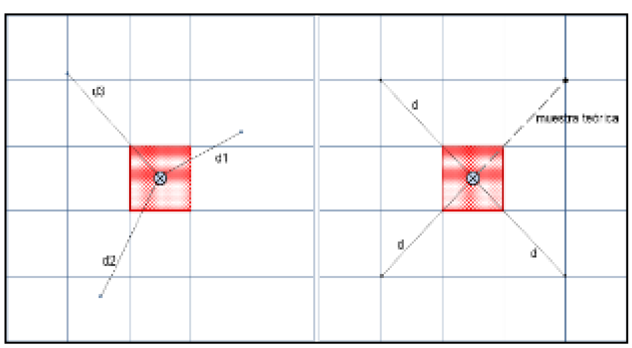
The sub cell block model was re blocked to the original block size (50 x 50 x 25 ft). Density estimation and classification procedures were performed in the re-blocked model.

14.2.5 Resource Classification

The previous resource model for the Open Pit Mineral Resource used a combination of kriging passes and kriging error to define block classification. Golder modified this approach trying to decrease the range of passes in order to not allow blocks to faraway of the drill holes be classified as indicated or inferred. Considering that the maximum variograms ranges for high grade domains about the 150-200 ft, the limit for the indicated resources was set to 260 ft.

An equivalent grid calculation was performed to determine the level of information in the neighbor of each block using the scheme described in Figure 14-68. This method used the distance to the closest three drill holes to calculate the "equivalent grid" surrounding the block.

Figure 14-68: Equivalent Grid Calculation Scheme (Golder, 2019)



$$Equivalent\ distance = \frac{(d_1 + d_2 + d_3)}{3}$$
$$Equivalent\ grid = \sqrt{2} * Equivalent\ distance$$

Samples with -9 and -7 code were not considered in the calculation of the equivalent grid so blocks near these samples have higher grid values and therefore lower classification. Re-assaying available pulps/rejects would improve the classification in these localized intervals.

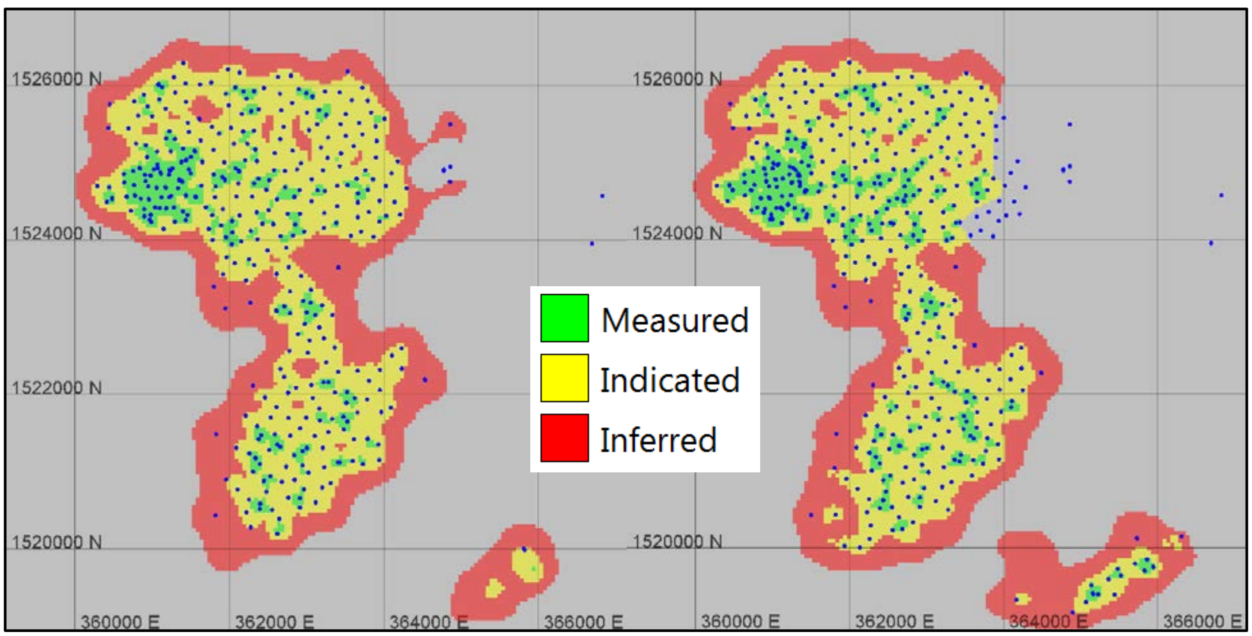
Thresholds for the theoretical grids have been defined according to Table 14-49.

Table 14-49: Classification Parameters

Classification	Grid (ft)
Measured	0-150
Indicated	150-260
Inferred	>260

Figure 14-69 shows a visual representation of the resource classification results.

Figure 14-69: Results of the Resource Classification, Plan View 3700 (left) &4000 (right) (Golder, 2019)



In order to eliminate "spotted dogs" that are common in this kind of classification method a smoothing process was carried out, the routine keep the same proportions of each class in the final result. Several smoothing windows were tried using a 5×5×5 blocks window.

Figure 14-69 also shows that there is opportunity to increase Resources in areas of open drilling. The figure also shows that there is limited drilling outside of the Mineral Resource area and it would be prudent to include a condemnation program for the WRSF and infrastructure area in any proposed exploration program.

The most significant change is in the indicated resources that are more restricted in the new model. In the previous model blocks with an equivalent grid up to 300 ft. can be classified as indicated, in the new model, that distance was reduced 260 ft.

14.2.6 Resource Statement Western Area Deposits

Mineral Resources are subdivided into classes of Measured, Indicated and Inferred, with the level of confidence reducing with each class respectively. Mineral Resources are reported as in situ tonnage and are not adjusted for mining losses or mining recovery. The Mineral Resources reported are inclusive of those reported in Mineral Reserves. Mineral Resources were based on a Lerch Grossman optimization using the parameters set forth in Table 14-50.

Table 14-50: Lerch Grossman Mineral Resource Optimization Parameters

Parameter	Unit	Value
Cu Price	$/lb	3.75
Waste Mining Cost	$/st	1.45
Ore Mining Cost	$/st	1.55
Incremental Mining Cost	$/st per 50 ft bench	0.03
Mining Recovery	%	100
Mining Dilution	%	0
Processing Cost	$/st	5.37
Cu Processing Recovery – North Pit	%	90
Cu Processing Recovery – South Pit	%	88
Cu Selling Cost	$/lb	0.55
Au Price ($0 selling cost)	$/toz	1,343
Ag Price ($0 selling cost)	$/toz	19.86

The Open Pit Resources are estimated to contain 134 million short tons (Mst) of Measured Mineral Resource, 419 Mst of Indicated Mineral Resource and 28 Mst of Inferred Mineral Resource at a cutoff grade of 0.12% Cu.

Table 14-51 shows the Mineral Resource reported at the 0.12% cutoff grade. The Mineral Resource Estimate in Table 14-51 could be materially affected by the copper selling price.

Table 14-51: Resource Inventory, Inside the Open Pit Optimization Shell

North Deposit		Average Grade			Contained Metal		
Classification	**Mst**	**Cu %**	**Au (oz/t)**	**Ag (oz/t)**	**Cu (Mlbs)**	**Au (Koz)**	**Ag (Koz)**
Measured (M)	90.6	0.66	0.002	0.072	1,195	188	6,481
Indicated (I)	218.5	0.47	0.002	0.056	2,040	330	12,141
M+I	309.0	0.52	0.002	0.060	3,235	518	18,622
Inferred	16.0	0.39	0.001	0.047	126	22	759
South Deposit							
Classification	**Mst**	**Cu %**	**Au (oz/t)**	**Ag (oz/t)**	**Cu (Mlbs)**	**Au (Koz)**	**Ag (Koz)**
Measured (M)	43.8	0.36	0.002	0.048	313	67	2,112
Indicated (I)	200.1	0.36	0.001	0.045	1,452	293	9,044
M+I	243.9	0.36	0.001	0.046	1,765	360	11,156
Inferred	11.5	0.31	0.001	0.029	71	14	329
North + South							
Classification	**Mst**	**Cu %**	**Au (oz/t)**	**Ag (oz/t)**	**Cu (Mlbs)**	**Au (Koz)**	**Ag (Koz)**
Measured (M)	134.3	0.56	0.002	0.064	1,508	255	8,593
Indicated (I)	418.6	0.42	0.001	0.051	3,492	623	21,185
M+I	552.9	0.45	0.002	0.054	5,000	879	29,778
Inferred	27.5	0.36	0.001	0.040	197	37	1,088

Note: The Mineral Resource above is defined using a 0.12% Cu cutoff grade with the pit shell created using the parameters described in Table 14-50. The Mineral Resources are reported inclusive of the Mineral Reserves. Mineral Resources do not include dilution and 100% mining recovery. The tons are reported on a dry basis.
Effective date on Open Pit Mineral Resource is January 21, 2019.

The reader is cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-50 outlines inputs into the Lerch Grossman optimization that was used to limit Mineral Resources. In regard to the Mineral Resources, Golder is unaware of any material effects to the Mineral Resource potentially caused by known environmental, permitting, legal, title, taxation, mining, metallurgical, infrastructure, socio-economic, marketing and political factors. The Open Pit Project is fully located on privately owned or leased lands and there are no known legal or title issues affecting the Property. The Open Pit Project has all material permits and Nevada Copper is not aware of any known socio-economic factors that could impact the Open Pit Project or the Open Pit Mineral Resource. Increases in items such as mining cost, processing cost and selling cost or a decrease in the copper price would result in decrease to the Open Pit Mineral Resources. An increase in the copper price, or decreases in items such as mining cost, processing cost and selling cost would result in an increase to the Open Pit Mineral Resources.

Item 15.0 MINERAL RESERVE ESTIMATES

The Mineral Reserves have been prepared and estimated for each of the Underground Project and the Open Pit Project.

15.1 Underground Mineral Reserves

The Mineral Reserve estimate for the Underground Project was prepared in compliance with industry accepted best practices. The QP for the Mineral Reserve estimates was Mr. Maurice Mostert FSAIMM.

The Mineral Reserve estimate for the Underground Project is based on the original Mineral Resource block models provided by Nevada Copper for the East (encompassing East North and East South zones) and E2 deposits. The files were provided in comma separated value (.csv) format “BMEast.csv” and “BME2.csv” and were imported and checked in Deswik CAD. Both the East and E2 models have similar attributed fields and properties. While both models have the same parent block size, the E2 model has sub-block material in increments of 5 ft (to a minimum of 5 ft). A detailed table of model attributes can be found in Item 16.1.5.

The Mineral Resource block model used for the Mineral Reserve estimate for Underground Project has been confirmed by Tetra Tech as being the same Underground block model, as described in Item 14.1.

Mining Plus created mining block models (160902_east_md_nsr_final_extra02.dm and e2bm_nsr_cl12.dm) for the East and E2 deposits to include calculated fields for Copper Equivalence (CuEq) and NSR as outlined below. The formulae for CuEq and NSR are further discussed in Item 16.0. The additional fields reflect the byproduct values of gold and silver within the copper concentrate, based on recoveries, metal prices, payable terms, moisture content, treatment and refining costs and transportation costs, as presented below.

Copper equivalence has been calculated using the following formula:

$$CuEq\% = \left(\frac{100 \times NSR}{CuMetRec \times CuPybl \times 2000 \times CuPrice}\right) - \left(\frac{TC \times \left(\frac{CuMetRec}{ConGr}\right) \times \left(\frac{mt}{st}\right)}{1 - ConMoist\%}\right) - (CuRC \times CuMetRec \times 2000) - \left(\frac{\frac{FC \times CuMetRec}{ConGr}}{1 - ConMoist\%}\right)$$

Copper equivalence can also be calculated using the equation below, based on the gold and silver grades:

$$CuEq = Cu + Au \times 20.25041 + Ag \times 0.26875$$

The NSR has been calculated based on gross revenues from copper and precious metals, costs for treatment, refining, transportation, assay, insurance and marketing. NSR can be calculated as per the equation below (units are dollars per short ton):

$$NSR = (Cu\% \times CuMetRec \times CuPybl \times 2000 \times CuPrice) + (Au \times AuMetRec \times AuPybl \times AuPrice) + (Ag \times AgMetRec \times AgPybl \times AgPrice) - \left(\frac{TC \times \left(Cu\% \times \frac{CuMetRec}{ConGr} \right) \times \left(\frac{mt}{st} \right)}{1 - ConMoist\%} \right) - (CuRC \times Cu\% \times CuMetRec \times 2000) - \left(\frac{FC \times Cu\% \times CuMetRec}{\frac{ConGr}{1 - ConMoist\%}} \right)$$

The key modifying factors used in the estimation of Mineral Reserves for the Underground Project are shown in Table 15-1 and Table 15-2. Other factors (including environmental, hydrological, marketing, etc.) are detailed separately in this report, and the modifying factors presented below are a function of these factors.

Table 15-1: Key Modifying Factors for the Underground

Commodity	Price		Metallurgical Recoveries	Percentage Payables
	Unit	$	Percent	Percent
Cu	($/lb)	3.00	92	96
Au	($/oz)	1,343	78	90
Ag	($/oz)	19.86	70	90

Table 15-2: Net Smelter Return Concentrate Economic Parameters

Category	Unit	Value
Moisture Content	**%**	**10**
Copper Concentrate	**%**	**26**
Treatment Costs	**$/dmt Conc**	**75.00**
Refining Costs	**$/lb Cu Conc**	**0.075**
Transportation Costs	**$/mt Conc**	**90.00**

The percent payable factors are based on dry material in concentrate.

Refining costs are stated in dollars per pound of copper contained in concentrate. Refining costs for precious metals are assumed to be negligible and have not been considered.

Assay, insurance and marketing costs are assumed to be negligible and have not been considered.

Penalties for deleterious elements in concentrate are considered to be immaterial and have not been considered.

For the purpose of estimating Mineral Reserves, Inferred Mineral Resource blocks in all models were assigned zero grades (Please see Item 16.1.5).

Mining Plus assessed NSR cutoff values to reflect the estimated costs for mining, processing and G&A, based on a contractor-miner scenario until steady-state production is achieved, followed by an owner-miner scenario thereafter. The NSR cutoff value is not a break-even value; rather, it is an elevated value intended to target higher grade material. The result of this assessment was using a NSR cutoff value approximately equivalent to the average total Opex for the contractor-miner and owner-miner scenarios of $46/st. A summary of the estimated mining, processing and G&A costs for both the contractor-miner and owner-miner scenarios that were used for this assessment are presented in Table 15-3.

Table 15-3: Estimated Unit Opex Cost Assumptions

Description	Contractor	Owner-Miner
	Unit Opex Costs ($/st milled)	Unit Opex Costs ($/st milled)
Total UG Mine	35.33	27.20
Total Surface (incl Mill)	12.66	12.66
Geological Drilling	0.81	0.81
Mine G&A	2.54	2.54
Surface G&A	0.63	0.63
TOTAL OPEX	52.55	43.83

Mine Stope Optimizer (MSO) was initially conducted on the mining block models, reflecting the NSR cutoff value. The results generated from MSO were utilized for generating detailed and refined crosscut and stope designs within Deswik software.

The Mineral Reserves (Table 15-6) reflect the transverse mining method, primary and secondary stope sequence, along with use of CPF, UPF and URF within the respective stope sequencing. Dilution and mine loss percentage estimates have been applied in the Mineral Reserves. While dilution has been applied to waste development, it was not applied to ore to avoid a double count in overlapping development and stope boundaries. Table 15-4 and Table 15-5 summarize dilution and mine loss percentages applied as modifying factors to the Mineral Resource. Further detail on the relevant mining aspects used as modifying factors to the Mineral Resource are provided in Item 14.1.

This Report did not identify any other mining, metallurgical, infrastructure, permitting or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production. The Underground Project is fully located on privately owned or leased lands and there are no known legal or title issues affecting the Property. The Underground Project has all material permits and Nevada Copper is not aware of any known socio-economic factors that could impact the Underground Project.

It is noted that substantial changes in copper price or mining cost could affect the Mineral Reserves. However, it should be noted that the Project remains positive with a price decrease of 30%, or with a cost increase of 30%, which is based on the sensitivity analysis depicted in Figure 22-1 and Figure 22-2.

Table 15-4: Stope Dilution

Zone	Primary Stope Dilution		Secondary Stope Dilution	
	Grade % Copper	Dilution %	Grade % Copper	Dilution %
East South	0.75	3.10	0.24	10
East North	0.75	2.50	0.19	10
E2	0.75	5.00	0.38	10

Table 15-5: Summary of Mining Recovery for Stopes & Development

Parameter	Stope Losses (%)		Development Loss (%)
	East North Zone	East South and E2 Zones	
Pillars	1.9	1.9	
Drill & Blast	2.1	1.6	
Mucking	1.4	1.0	1.8
Total Recovery	**94.9**	**95.7**	**98.2**

Table 15-6: Mineral Reserve Estimates

Category	Tons	Cu	Au	Ag	NSR
	Million tons	%	oz/st	oz/st	$/st
E2					
Proven	1.1	1.92	0.012	0.245	79.28
Probable	5.0	1.58	0.009	0.197	64.30
E2 Reserves	6.1	1.65	0.009	0.206	67.04
East Deposit					
Proven	6.3	1.83	0.006	0.126	68.74
Probable	11.5	1.43	0.005	0.112	54.00
East Reserves	17.8	1.57	0.005	0.117	59.18
Total Reserves					
Proven	7.4	1.85	0.007	0.144	70.33
Probable	16.5	1.47	0.006	0.138	57.10
Total Reserves	23.9	1.59	0.006	0.139	61.18

Note: Effective date on Underground Mineral Reserves is September 15, 2017.

There are no material differences to the 2017 PFS with respect to the construction schedule, costs, mine plan, production, markets, assumptions and economic analysis and no material updates to the 2017 PFS are currently required.

15.2 Open Pit Mineral Reserves

The Mineral Reserve estimate for the Open Pit Project was prepared in compliance with industry accepted best practices. The QP for the Mineral Reserve estimates was Mr. Edward Minnes, P.E.

A pit optimization was completed using Whittle and based on the metal prices, metallurgical recoveries and concentrate off-take terms. A pit design was completed based on the revenue factor 0.90 Whittle pit shell, which corresponds to a $2.48/lb Cu (approximate) selling price. Measured and Indicated (sulfide) Resource within the pit design is classified as ore at a break-even copper cutoff of 0.129% for the North Pit and 0.132% for the South Pit. These cutoffs take into account the variable process recovery. The modifying factors, of 5% dilution and 98% mining recovery, were applied to the resource allocated as ore within the pit design. Only sulfide material classified as Measured and Indicated are included in the Proven and Probable categories.

The following are the input parameters that were used to convert the Open Pit Mineral Resource into the Open Pit Mineral Reserves:

- The Mineral Reserve estimates were prepared using generally accepted industry standards.
- Mineral Reserves are the economic portion of the Measured and Indicated Mineral Resources. Not all Mineral Resources have been converted to Mineral Reserves. Mineral Reserve estimates are influenced by several factors including deposit type, deposit shape and mining methods.
- Inferred Mineral Resource was considered waste for the Open Pit Reserve estimate.
- The cutoff 0.129% Cu for the North Pit and 0.132% Cu for the South Pit is based on the copper processing recoveries (90% for the North Pit, 88% for the South Pit) and costs.
- The value of the metal within the pit design was calculated using a long-term price forecast for copper ($2.75/lb), gold ($1,343/toz), and silver ($19.86/toz). The process recoveries used to calculate value include the copper recoveries described above gold 67%, and silver 56%
- A selling cost of $0.55/lb was applied to the Cu in concentrate to account for NSR. No selling costs were applied to Au or Ag. NSR and CuEq calculations are provided in Item 15.1.
- The pit designs targeted only Measured and Indicated Mineral Reserves. Inferred Mineral Resources within the pit design was included as waste with zero grade.
- Pit design geotechnical parameters were established by a geotechnical assessment performed by Golder in 2012.
- Dilution was assumed 5% based on various block model analyses.
- Mining recovery of 98% was applied assuming diligent drilling, blasting and surveying.
- Mineral Reserves are effective January 21, 2019.
- The Mineral Reserves are included in the Mineral Resource Estimates presented in Table 14-51.
- The Open Pit Reserves are summarized in Table 15-7.

Table 15-7: Mineral Reserve Summary

Confidence Category	Ore (Mt)	Average Ore Grades			Contained Metal		
		Cu (%)	Au (oz/t)	Ag (oz/t)	Cu (Mlbs)	Au (Koz)	Ag (Koz)
Proven Mineral Reserves (North)	75.4	0.65	0.002	0.070	983	151	5,302
Proven Mineral Reserves (South)	31.3	0.36	0.002	0.045	223	48	1,420
Proven Mineral Reserves (North + South)	106.6	0.57	0.002	0.063	1,206	199	6,722
Probable Mineral Reserves (North)	147.4	0.48	0.001	0.055	1,407	215	8,086
Probable Mineral Reserves (South)	131.7	0.37	0.002	0.049	977	203	6,458
Probable Mineral Reserves (North + South)	279.1	0.43	0.001	0.052	2,384	419	14,544
Proven and Probable Mineral Reserves	**385.7**	**0.47**	**0.002**	**0.055**	**3,590**	**617**	**21,266**

Note: Effective date on Open Pit Mineral Reserves is January 21, 2019.

The extent to which the mineral reserve estimates could be materially affected by mining, metallurgical, infrastructure, permitting, and other relevant factors that are different than the factors used in the PFS and described in this report is shown by the sensitivity analysis in Item 22.0 Except for commodities prices, all other relevant factors including mining, metallurgical, infrastructure, and permitting factors related to the Project and described in this report are factors affecting estimated Project costs and are reflected in the PFS cost estimates that are summarized in this report. If for any reason any of these Project cost factors are changed such that the Project capital or operating cost estimates change materially, then the mineral reserve estimates stated in this report could be materially affected. As shown on Figure 22-1, IRR Sensitivity and Figure 22-2, NPV Sensitivity, even if the cost factors are changed such that total operating or capital cost estimates for the Project are increased by 30%, the after-tax Project IRR and NPV remain positive, and therefore the mineral reserve estimates may remain unaffected. As of the effective date, there are no known Project cost factors that are materially different from the factors used in the PFS and summarized in this Report to the extent that the mineral reserve estimates would be materially affected.

Projected Project revenues depend upon forecast commodity prices used for the PFS and described in this report. As shown on the same Figures in Item 22.0, if the forecast price of copper used in the PFS over the study period declines by 25% or more, the Project is no longer economic and the mineral reserve estimates will be materially and adversely affected. In this case, if the estimated price of copper used in the PFS over the study period declines by 25% or more, the Project NPV and IRR become negative, and the Mineral Reserve estimates would be materially and adversely affected. In this case, the extent to which the Mineral Reserve estimates could be affected is estimated to be about a 40 to 50% reduction. This impact is based on the pit shell analysis described in this Item. However, it is also important to note that, based on Nevada Copper's economic model, and even at a Cu price of $2.25/lb, this PFS mine plan delivers an average LoM operating cashflow (net revenue - operating cost) of $94 Mpa and a positive free cashflow (net revenue - operating cost, capital cost, working capital and tax) is maintained at an average of $44 Mpa. Infrastructure and permitting factors are not anticipated to materially affect the Mineral Reserve estimate. The enhancements to the drilling database used for the Open Pit Project, outlined in Item 26.0, are not anticipated to materially impact the Mineral Reserve estimate in any way.

Item 16.0 MINING METHODS

16.1 Underground

16.1.1 Mining Design Criteria

The block model used as the basis of design has assigned densities based on lithology and iron content. The assumed swell factor is based on typical values for the blasting of the rock types present at the Property. Densities and swell factor used for the Underground Project PFS are shown in Table 16-1 and Table 16-2, respectively.

Table 16-1: In Situ Rock Properties

Description	Tonnage Factor (ft3/st)	Dry Density (st/yd3)
East North (EN)	9.11	2.96
East South (ES)	9.44	2.86
E2	9.08	2.97
Waste	10.02	2.70

Table 16-2: Ore & Waste Loose Densities & Swell Factors

Description	Value
Ore Loose Density	2.25 st/yd3
Waste Loose Density	2.08 st/yd3
Swell Factor	30%

16.1.2 Mining Method Selection

The mining method for the Underground Project has been selected based on qualitative and quantitative assessment techniques. The transverse longhole stoping method has been selected as optimal for all zones (EN, ES and E2), based on safety, mining recovery and dilution, productivity and the ability to mine large spans given the ground conditions. Stopes will be extracted through a bottom-up sequence, reducing lead time and requirements for upfront development in most instances. In the E2 zone, there are some narrower parts of the orebody that have been identified as being favorable for longitudinal longhole stoping methods, since this will provide maximum efficiency in operating lateral development.

Additional small flat-lying mining areas close to the E2 connector drift (west of the ES Deposit) and below the connector drift were considered as to whether they could be efficiently recovered using alternative mining methods. However, these areas were deemed uneconomic to mine due to substantial development requirements, low mining recovery and relatively low cash flow generation.

16.1.3 Geotechnical Design

16.1.3.1 Geotechnical Data Review and Database Update

Mining Plus reviewed the existing geotechnical database, which is composed of diamond drill hole data, including those collared from underground. Geotechnical data obtained from these drill holes includes intact rock properties, joint characterization and oriented core. This information, along with previous technical reports and a site visit completed in April 2017, has been used to assess ground conditions and generate mine design parameters for the Underground Project.

16.1.3.2 Geotechnical Domains

In order to understand the ground conditions at the Pumpkin Hollow Project, geotechnical domains were identified. Preliminary geotechnical parameters were initially assessed by major lithology type (“simplified lithology”), where it was found that ground conditions were similar across many lithologies, including those that host the majority of the deposits. Given that mineralization is hosted in multiple lithologies, geotechnical domains were identified for each Deposit (E2, EN and ES) based on location for mining:

- Hanging wall (HW)
- Ore
- Footwall (FW)
- Waste

The HW and FW domains represent a 100 ft step-out from ore boundaries, using the 3D shapes developed by Mining Plus as a guide.

16.1.3.3 Rock Mass Assessment

Rock Quality Designation

An assessment of the rock mass and geotechnical domains was completed by considering the Rock Quality Designation (RQD) (after Deere 1964). RQD values and descriptions are presented in Table 16-3, while RQD data by deposit and geotechnical domain are presented in Table 16-4, Table 16-5 and Table 16-6. It can be seen that Very Poor RQD values are more prevalent in the HW domains of the EN and ES deposits. In order to limit over-break (and dilution), it may be necessary in some cases to install ground support in stope backs (such as >12 ft Super Swellex or fully encapsulated cable bolts) formed in HW material (i.e., upper-most stopes). This is estimated to affect <10% and <5% of upper-most stopes in the EN and ES deposits respectively.

Table 16-3: Rock Quality Designation Values and Descriptions

RQD (Description)	RQD Value (%)
Very Poor	0 to 25
Poor	25 to 50
Fair	50 to 75
Good	75 to 90
Excellent	90 to 100

Source: Deere 1964.

Table 16-4: Summary of Rock Quality Designation Values (E2 Deposit by Geotechnical Domain)

Domain		HW	Ore	FW	Waste
25%	Rec. Core (ft)	10	35	24	414
	%	0.8%	4.1%	1.6%	15.8%
25-50%	Rec. Core (ft)	156	155	74	444
	%	12.4%	18.5%	4.8%	17.0%
50-75%	Rec. Core (ft)	341	191	243	659
	%	27.3%	22.7%	15.8%	25.2%
75-90%	Rec. Core (ft)	291	257	432	416
	%	23.3%	30.6%	28.1%	15.9%
90-100%	Rec. Core (ft)	452	202	763	684
	%	36.2%	24.1%	49.7%	26.1%

Table 16-5: Summary of Rock Quality Designation Values (EN Deposit by Geotechnical Domain)

Domain		HW	Ore	FW	Waste
25%	Rec. Core (ft)	542	849	609	1,431
	%	30.4%	12.9%	17.3%	16.4%
25-50%	Rec. Core (ft)	409	1,036	441	1,535
	%	23.0%	15.8%	12.5%	17.6%
50-75%	Rec. Core (ft)	453	1,598	1,080	1,944
	%	25.4%	24.3%	30.7%	22.3%
75-90%	Rec. Core (ft)	267	1,505	637	1,775
	%	15.0%	22.9%	18.1%	20.4%
90-100%	Rec. Core (ft)	111	1,586	754	2,021
	%	6.2%	24.1%	21.4%	23.2%

Table 16-6: Summary of Rock Quality Designation Values (ES Deposit by Geotechnical Domain)

Domain		HW	Ore	FW	Waste
25%	Rec. Core (ft)	140	590	252	2,642
	%	13.3%	15.8%	8.4%	21.0%
25-50%	Rec. Core (ft)	160	512	337	2,520
	%	15.2%	13.7%	11.2%	20.1%
50-75%	Rec. Core (ft)	302	735	627	3,280
	%	28.9%	19.7%	20.9%	26.1%
75-90%	Rec. Core (ft)	261	745	581	2,321
	%	25.0%	20.0%	19.4%	18.5%
90-100%	Rec. Core (ft)	185	1,142	1,198	1,798
	%	17.6%	30.7%	40.0%	14.3%

Based on observations of core during the site visit and discussion with Nevada Copper personnel, the talc and marble simplified lithologies can, in some limited cases, feature Very Poor RQD values (i.e., approaching 0%). An example of variability of conditions over a short distance is shown in Figure 16-1. Where poor RQD values are encountered, this will likely pose localized challenging mining conditions. In the case of lateral and vertical development, this could require heavier and/or deeper ground support. Stoping areas would need to be considered on a case-by-case basis and may involve reduction of stope sizes and additional ground support such cable bolts and/or fibercrete. These additional ground support costs have been estimated and included in overall operations costs.

Figure 16-1: Example of Variability of Marble over Short Distance (Drill hole NC15-10, 393-442 ft)

Note: Where recovered, whole core has been cut for assay sampling. Source: 2017 Technical Report.

Rock Mass Classification

Rock mass classification was conducted using the Norwegian Geotechnical Institute's tunneling quality index (the Q-system), as proposed by Barton et al. (1974), where Q is obtained from the following relationship, using RQD (as described above), joint set number (Jn), joint roughness (Jr), joint alteration (Ja), joint water factor (Jw) and the stress reduction factor (SRF):

$$Q = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot \frac{J_w}{SRF}$$

Q values have been estimated using the methods outlined by Barton and Grimstad (1994), whereby summarized values for each input are considered. These values were derived by interrogating a database composed of geotechnical data primarily sourced from drill hole core. These data are understood to have been collected by representatives from Nevada Copper and Golder. In order to better understand the statistical variability and character within each data set, basic descriptive statistics and histograms were generated for each input value and within each geotechnical domain. This information was used to identify representative values for each Q input value. In the case of Jw and SRF, site experience, assumed far-field stress conditions and typical depth of mining were applied, along with engineering judgment to derive input values. The Q value estimates range from very poor to good.

16.1.3.4 Stability Analysis

Stability analysis of all planned excavation types (LOM, development, production, and vertical) were undertaken using empirical approaches typically applied at a prefeasibility level technical study, along with kinematic assessments using UnWedge software (Rocscience 2017).

Stope Stability

Stope stability assessment for the E2, EN, and ES deposits was completed using the empirical Modified Stability Graph Method (after Mathews et al. 1981; Potvin 1988; Trueman and Mawdesley 2003), as described in Hutchinson and Diederichs (1996). It should be noted that the method is approximate only and early stoping should be carefully monitored and designs adjusted in response to actual performance.

In this method, the stability number N' is calculated from the following expression, using the modified Tunneling Quality Index (Q') (after Barton et al. 1974); rock stress factor (A); joint orientation adjustment factor (B) and the gravity adjustment factor (C):

$$N' = Q' \times A \times B \times C$$

N' values have been plotted against the hydraulic radius (HR) of the stope face being assessed. The HR is calculated by dividing the area of a stope face by the perimeter of that face. Analyses considered the maximum HR value for stope side walls, end walls and backs for each deposit, as permitted by the N' values.

The stope stability analysis followed a design criterion whereby data points plotted either within the “stable” zone or on the “stable”/“transition” divide. The recommended stope design parameters are presented in Table 16-7.

Table 16-7: Stope Design Parameters by Deposit

Deposit	Source	Sub-level Spacing (ft)	Transverse Width (ft)	Length (ft)
E2	FS (2015)	100	50	25-100
	PFS (2017)	100	50	75
ES	FS (2015)	100	50	75
	PFS (2017)	100	50	75
EN	FS (2015)	75	50	75
	PFS (2017)	75	50	75

Stope Overbreak

An assessment of stope overbreak has been conducted by calculating the Equivalent Linear Overbreak / Slough number, as first described by Pakalnis et al. (1996). The assessment indicates that HR and N’ value of stope walls and backs plot in the “blast damage only” to “minor sloughing” zones (after Pakalnis et al. 1996). Based on industry experience, this suggests overbreak on walls and backs of around <1.5 ft (<0.5 m) (after Oddie and Pascoe 2005).

Pillar Stability

An assessment of rib pillar stability for the sublevel open stoping mining method was completed in accordance with Diederichs et al. (2002), as outlined in Brady and Brown (2006). This approach considers the ratio of far-field stress and intact rock strength. Based on available data, the stability of the planned pillar plots well within the “intact rock mass” area (i.e., little to no pillar damage is forecast).

Given rock mass quality, planned depth of mining and assumed in situ stress conditions of the Underground Project, relatively favorable mining conditions are likely (but not assured) to prevail with regard to excavation interaction.

Based on the available data, it is suggested that minimum development standoff distances of 25 m and 30 m from production stopes and large excavations be assumed for mine design in the E2 and East (North and South) Deposits, respectively. This parameter may need to be refined after actual mining experience is gained at the Underground Project.

Further empirical pillar stability analysis could be undertaken in a future feasibility study to further refine the PFS work. The current assessment indicates that a material negative impact on the mineral reserve is unlikely.

Raisebore Stability

An assessment of raisebore risk was completed using available data, an assumed diameter of 12 ft (3.66 m) and a probability of failure (PoF) of 5%. In order to assess face and wall stability during and after raiseboring, the McCracken and Stacey (1989) raisebore stability method was used. This required the application of weathering and joint orientation adjustment factors to Q values. However, it should be noted that due to the lack of site-specific data, a detailed assessment was not performed and the results should be considered as indicative only. A detailed and reliable McCracken and Stacey (1989) analysis can only be completed using data from a pilot hole drilled along (or very close to) the planned raisebore axis.

Results suggest that some lithologies may be challenging to successfully create a 12 ft (3.66 m) diameter raisebore shaft. However, these results should not be considered definitive for the reasons outlined above.

Creating moderate-sized vertical excavations (including raisebore diameters greater than around 6.5 ft (2 m) through the upper ~100 ft (30 m) of weathered surface material (often soil) can be difficult and carries a high risk of failure. However, discussions with Nevada Copper personnel indicate that this was not the experience while sinking the production shaft at the Pumpkin Hollow Project. Nonetheless, problems were encountered while sinking the shaft through the volcanic lithology / regional (sub-horizontal) fault zone, where “squeezing,” time-dependent rock mass behavior was observed. Attempting to raisebore through such material is likely to be challenging. Alternative options may include, but are not limited to:

- Sinking shaft through adverse, near-surface ground conditions, followed by raising the remainder (majority) of the shaft

- Raiseboring a small diameter shaft, say 3 ft (~1 m) then strip and line the “pilot” raise to a larger diameter

Based on discussion with site personnel, it is understood that high-content garnet skarn has been encountered at the Underground Project, which has resulted in excessive drill bit wear. This implies that such material is likely to have high tensile strength values. Industry experience has shown that attempting to raisebore through high-tensile strength material can be problematic. Therefore, flagging and appropriate laboratory testing (such as tensile strength tests) should be undertaken in the event that high-content garnet skarn is identified in drill core from pilot holes.

In cases where raiseboring is absolutely required to be situated in a garnet rich skarn, the process will take longer and scheduling impacts will need to be considered.

Dependent on timing for drilling pilot holes along (or very close to) the planned raisebore axis for the proposed ventilation raises, a detailed and reliable McCracken and Stacey analysis may be completed and incorporated into a future feasibility study.

Kinematic Analysis

A kinematic assessment of excavations has been competed using UnWedge software (Rocscience 2017). This assessment was based on available data (including joint character and orientation). A design criteria of a Factor of Safety (FoS) of 1.5 was assumed. Results indicate wedge volumes to be moderate (<11 st) and can be held in place with moderate ground support (discussed below), featuring FoS values that meet or (in most cases exceed) the design criteria.

16.1.3.5 Ground Support Selection

Preliminary ground support requirements have been estimated using the updated rock reinforcement design chart (Grimstad and Barton 1993), which uses the Q-system. The Q-system rock reinforcement design chart relates the rock quality, excavation span and service life to support requirements. The method converts the width (span) of the excavation to an equivalent dimension, which takes into account the function of the excavation.

Preliminary empirical ground support results from this assessment suggest ground support requirements will either be Category 1 (“no ground support required”) and Category 4 (systematic bolting and unreinforced shotcrete). Based on modern industry practices, such ground conditions that plot in the “no ground support required” category will require moderate ground support installations.

Based on the estimated rock mass classification Q values, three ground support classes have been developed for lateral development drifts at the Underground Project. These classes have been selected so that site staff will be easily able to assess the Q values and assign the relevant ground support class. In order to confirm and refine assumptions applied in these preliminary designs, geotechnical mapping and estimation of Q values should commence once development begins.

16.1.4 Mine Hydrogeology

The most recent data relating to the mine hydrogeology was prepared in 2012 technical report by Tetra Tech entitled "*Technical Report - Underground Only Alternative for the Pumpkin Hollow Copper Project*" with effective date of December 12, 2012.

The mine inflow estimates, prepared by Tetra Tech at the time, were based upon available empirical data related to environmental aspects, seasonal effects and the thickness of the alluvial aquifer.

The most significant concern for larger expected inflows will be for the Eastern Area Deposits, reaching up to 1,800 gpm. The inflow direction for the eastern portion of the Property is toward the north and the west, whereas other portions of the Property are towards the north. The inflow will be greater during shaft construction and will lessen as the mine life progresses. The largest drawdown expected is about 10 ft. Therefore, this was used by Tetra Tech for the estimates. It is not expected that the drawdown will exceed this, as the estimate is already conservative. The extent of the drawdown will also not exceed 0.3 miles distance surrounding the mine.

Additional aquifer testing is required as well as the addition of empirical data moving into development. It is predicted that for the purpose of this study, passive collection will be required. Sumps will be placed strategically underground and water will be pumped out of the mine via the Main Shaft in the East zone and via the E2 ventilation raise in the E2 zone.

16.1.5 Resource Model

The original resource block models supplied for the Underground Project PFS are divided to represent the East and the E2 deposits. The resource block models entitled "BMEast.csv" and "BME2.csv" were used for compilation of the mineral reserves.

The majority of the model attributes for both the East and E2 Deposits are comparable. The main difference between the models is the block size. The East model contains one parent cell size whereas the E2 model contains the same parent cells with sub blocks in increments of 5 ft (to a minimum of 5 ft).

The block models were imported into Deswik CAD software, as provided originally, with no alteration that would affect the Global Mineral Resource results.

The original models had an "NSR" and "CU_EQ" attribute, with values coded for all Resource Classifications. The Inferred material was also coded with values for these attributes. In reference to the CIM Definition Standards:

> "An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Prefeasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines."

Therefore, Inferred Mineral Resource blocks in both imported models were assigned zero grades for the purpose of estimating Mineral Reserves.

The block models have been validated using Deswik software with model checks performed in order to ensure the validity and integrity of the model and the transfer process. The global unconstrained resource was compared to the results presented by Tetra Tech (see Item 14.1). All differences in resources ranged from 0% to 0.06%. Anything under 1% of difference is considered negligible.

16.1.5.1 Block Model Coordinate System

The original block models for the East (North and South) and E2 deposits were provided in UTM NAD27 coordinate system. No model rotation was applied. The Easting (x) and Northing (y) coordinates are in U.S. Nevada State Plain coordinates and the Elevation (z) coordinate is presented in ft above sea level.

The model origins for the East and E2 models are as follows:

- East:368,162.5 ft (x) by 1,521,722.5 ft (y) by 1,190 ft (z)
- E2: 368,887.5 ft (x) by 1,520,097.5 ft (y) by 1,190 ft (z)

As shown in Item 16.1.9, the parent block size for both models is 25 ft (x) by 25 ft (y) by 10 ft (z), whereas E2 also is sub-blocked to 5 ft increments.

16.1.5.2 Model Densities

The densities coded into the model (ft^3/st) follow a regression curve, which is explained in Item 14.1.8. The models (E2 and East) have not been changed since this time.

16.1.5.3 Model Recoveries

The model recoveries coded into the model (and used for the NSR analysis) were based on Sedgman's recommendation.

The 2017 PFS NSR calculations and coding in the model (as discussed in Item 16.1.7) were based on the recoveries and concentrate information shown below in Table 16-8. The batch flotation test work yields ore recoveries of at least 92% for concentrate grades greater than 26% Cu.

Table 16-8: Metallurgical Recoveries

Item	Value	Unit
Concentrate Grade (Dry material in concentrate)	26	%
Concentrate Moisture Percent	10	%
Process Flotation Recovery (Cu)	92	%
Au Metallurgical Recovery	78	%
Ag Metallurgical Recovery	70	%

16.1.6 Cutoff Grade Calculation

In order to account for gold and silver by-product values, a calculated NSR cutoff, based on mining, processing and G&A costs, will be utilized to evaluate minable material. Table 16-9 lists the Opex costs used to calculate the NSR cutoff value.

Table 16-9: Unit Opex Cost Assumptions

Description	Contractor	Owner-Miner
	Opex Costs - Initial Production ($/st milled)	Opex Costs - Initial Production ($/st milled)
Total UG Mine	35.33	27.20
Total Surface (incl Mill)	12.66	12.66
Geological Drilling	0.81	0.81
Mine G&A	3.01	2.54
Surface G&A	0.75	0.63
Total Opex	52.55	43.83

Based on a contractor-miner scenario until steady-state production is achieved, followed by an owner-miner scenario thereafter, the economic cutoff NSR value equivalent to the total Opex for such scenario is approximately $46.00/st milled.

Copper equivalent economic cutoff grades have been calculated for stope production and for development, as per Table 16-10 and Table 16-11.

Table 16-10: Economic Cutoff Production CuEq Grade

Category	Item	Unit	Value PFS
Production Cost	Production (per ton mined)	$/st mined	46.00
	Cu Cutoff Grade	% Cu	0.93
	Precious Metals Revenue (CuEq)	%	9.29
	Cu Equivalent Cutoff Grade	**% Cu**	**1.02**
Process Recovery	Cu Flotation Recovery	%	92.0
Dilution	Dilution Loss	%	96.1
	Payable lbs. Cu per ton mined	lbs. Cu/st mined	18.0
	Payable for 1 lbs. of Cu Shipped/Smelted	$/lb	2.56
	Ton Value at the cutoff grade	$/st	46.00
	Profit	$st value-prod Costs	0.00

Table 16-11: Economic Cutoff Development CuEq Grade

Category	Item	Unit	Value PFS
Development Only Cost	Production (per ton mined)	\$/st mined	18.04
	Cu Cutoff Grade	% Cu	0.37
	Precious Metals Revenue (CuEq)	%	8.00
	Cu Equivalent Cutoff Grade	**% Cu**	**0.40**
Process Recovery	Cu Flotation Recovery	%	92.0
Dilution	Dilution Loss	%	96.1
	Payable lbs. Cu per ton mined	lbs. Cu/st mined	7.1
	Payable for 1 lbs. of Cu Shipped/Smelted	\$/lb	2.56
	Ton Value at the cutoff grade	\$/st	18.04
	Profit	\$ ton value-prod Costs	0.00

16.1.7 Net Smelter Return & Copper Equivalency

The percent payables are based on dry material in concentrate in Table 16-12.The NSR is calculated based on the copper concentrate grade of 26% (Table 16-13).

Table 16-12: Percentage Payable Metal

Metal	Percentage Payable (%)
Copper	96.5
Gold	90
Silver	90

Table 16-13: Net Smelter Return Concentrate Economic Parameters

Category	Unit	Value
Moisture Content	%	10
Copper Concentrate	%	26
Treatment Costs	\$/dry short ton Conc	68.04
Refining Costs	\$/lb Cu Conc	0.075
Transportation Costs	\$/mt Conc	90.00

Refining costs are stated in dollars per pound of copper contained in concentrate. Refining costs for precious metals are assumed to be negligible and have not been considered.

Assay, insurance and marketing costs are assumed to be negligible and have not been considered.

Penalties for deleterious elements in concentrate are considered to be immaterial and have not been considered.

The NSR has been calculated based on gross revenues from copper and precious metals, costs for treatment, refining, transportation, assay, insurance and marketing. The equation below was derived, as follows (units are dollars per short ton):

$$NSR = (Cu\% \times CuMetRec \times CuPybl \times 2000 \times CuPrice) + (Au \times AuMetRec \times AuPybl \times AuPrice) + (Ag \times AgMetRec \times AgPybl \times AgPrice) - \left(\frac{TC \times \left(Cu\% \times \frac{CuMetRec}{ConGr}\right) \times \left(\frac{mt}{st}\right)}{1 - ConMoist\%}\right) - (CuRC \times Cu\% \times CuMetRec \times 2000) - \left(\frac{\frac{FC \times Cu\% \times CuMetRec}{ConGr}}{1 - ConMoist\%}\right)$$

Copper equivalence has been calculated using the equation below:

$$CuEq\% = \left(\frac{100 \times NSR}{CuMetRec \times CuPybl \times 2000 \times CuPrice}\right) - \left(\frac{TC \times \left(\frac{CuMetRec}{ConGr}\right) \times \left(\frac{mt}{st}\right)}{1 - ConMoist\%}\right) - (CuRC \times CuMetRec \times 2000) - \left(\frac{\frac{FC \times CuMetRec}{ConGr}}{1 - ConMoist\%}\right)$$

Copper equivalence can also be calculated using the equation below, based on the gold and silver grades.

$$CuEq = Cu + Au \times 20.25041 + Ag \times 0.26875$$

16.1.8 Mineable Shape Optimization Analysis

Mining Plus undertook an MSO analysis using the imported block models derived from resource models (see Item 16.1.5). The MSO input models (ensbm_nsr2_cl12.dm and e2bm_nsr_cl12) calculates the copper equivalency and NSR values reflecting gold and silver by-products and updated metal prices, along with processing recoveries and charges, as described in Item 16.1.7.

The MSO software was used to interrogate these resource block models to determine preliminary economic stope shapes based on preliminary estimated costs. For MSO, the E2 and East models were analyzed separately.

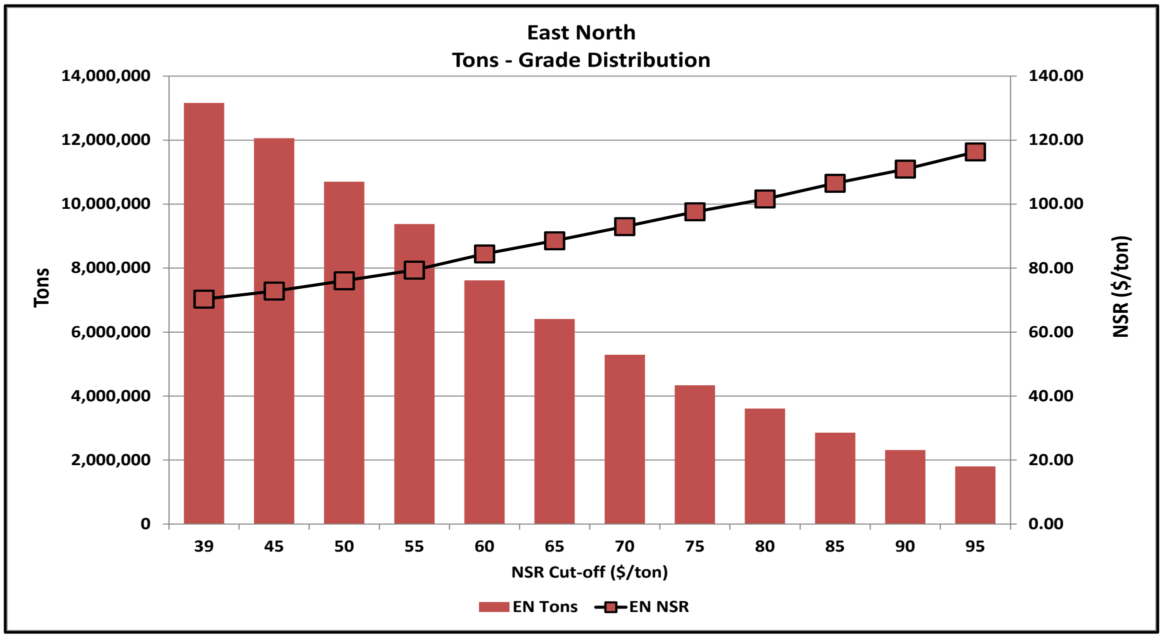
A cleaning process was applied to the MSO runs to eliminate any discontinuous shapes. Tabulated results are summarized in Table 16-14 and the grade ton curve for the EN model is illustrated in Figure 16-2. It should be noted that the application of gradients to the development will remove some minable inventory in the following MSO results.

Table 16-14: Summary of Mineable Shape Optimization Results

Category	$39/st NSR Cutoff		$65/st NSR Cutoff		$80/st NSR Cutoff		$90/st NSR Cutoff	
	Tons	Avg NSR	Tons	Avg NSR	Tons	Avg NSR	Tons	Avg NSR
ES	8,313,423	71.83	4,384,439	88.69	2,261,072	104.61	1,111,387	125.95
EN	13,163,396	70.30	6,406,812	88.53	3,609,790	101.59	2,310,539	110.92
E2	7,964,313	79.16	5,013,115	125.77	3,182,179	198.13	2,666,664	236.43
Total	**29,441,132**	**73.13**	**15,804,366**	**100.39**	**9,053,041**	**136.28**	**6,088,590**	**168.64**

Note: Tonnages from MSO runs are not final mine design tonnages. Subsequent to running MSO, stope shapes were refined to account for updated cost information and to optimize resource recovery.

Figure 16-2: East North Grade Tons Distribution (2017 Technical Report)



16.1.9 Mine Design

16.1.9.1 <u>Mine Access</u>

The main access to the underground mine will be serviced by the East Shaft, which will include a cage, skip, services and ventilation compartments. The E2 zone will be accessed through the E2 connector drive connecting to the COB from the ES zone. Spiral decline ramps and level accesses will provide access to mining levels within the East and E2 deposits. Figure 16-3 illustrates an isometric view of the mine accesses mentioned.

Development Design

Figure 16-4 depicts an isometric view, looking to the southeast, of the Main Shaft area.

Figure 16-3: Mine 3D View of the E2, ES, & EN Zones (Viewed from Southeast) (2017 Technical Report)

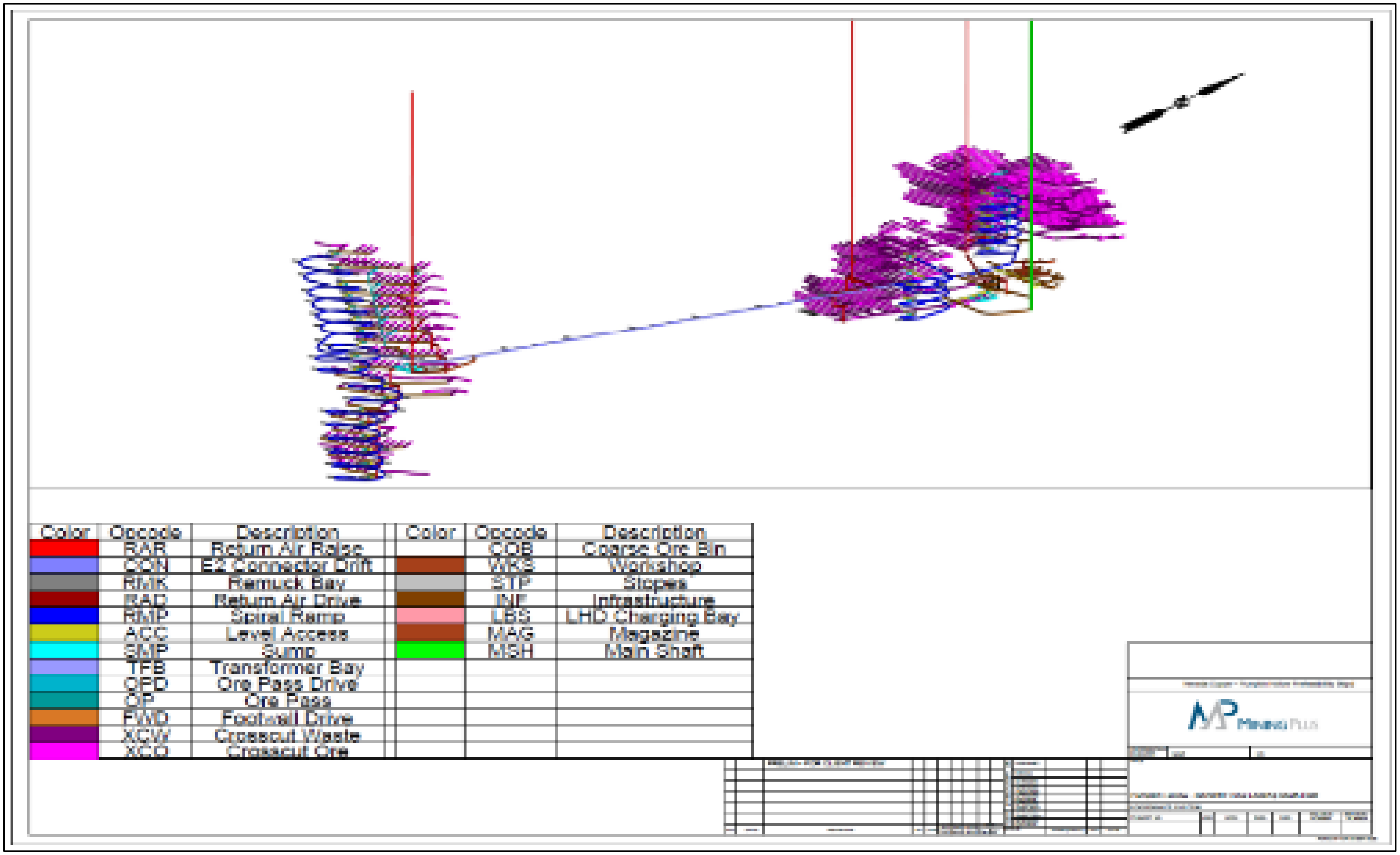
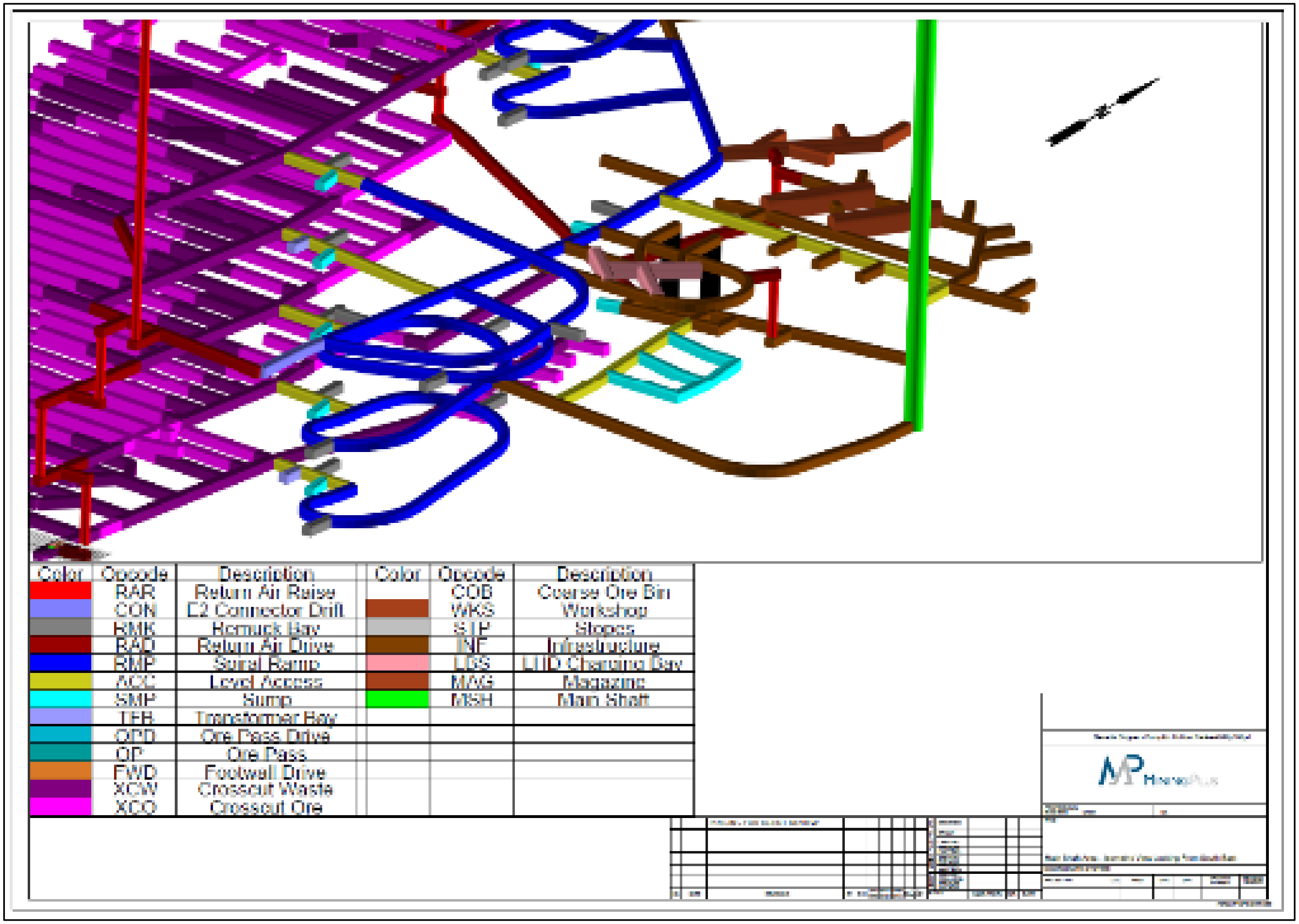


Figure 16-4: Main Shaft Area – Isometric View (Viewed from Southeast) (2017 Technical Report)



E2 Connector Drift

Access to the E2 zone is gained through a 16.0 ft by 16.0 ft drive connecting the E2 truck chute loading point at 1740 Level to the COBs at 1990 Level in the Main Shaft Area. The E2 connector drive is designed at 6% gradient to with remuck bays located every 500 ft. A connector drift will service ore mined in the E2 zone, connecting both the sacrificial and the ultimate ore pass to the EN COB.

E2 Connector Drift Material Transport Optimization

A study was done to determine the most cost efficient method to transport ore from E2 to the COB. The results showed that truck haulage was the most cost efficient and would allow for maximum flexibility in the scheduling of ore out of the underground mine.

Once truck haulage was chosen, the optimum gradient for the transport drift needed to be determined, as the battery-operated trucks would regenerate power traveling downhill loaded towards the COB.

A key metric that came out of discussions with OEMs and battery manufacturers was that the battery-operated trucks would not show significant energy regeneration to the battery at gradients less than 5%. This metric also drove the ore extraction level designs, as the 5% rule applies to LHD and haul truck battery-operated machines.

Upon reviewing the design and discussing the machine's regenerative capabilities with OEMs, a gradient of 6% was chosen for the design. Using 4% gradient as a base case, moving to a 6% gradient results in a small increase in costs of 1.9%, but will enable significant energy regeneration.

Spiral Declines & Level Access

Spiral declines will be used primarily for vehicle access and transporting supplies. Specific mining levels will utilize the spiral decline for hauling ore; however, ore passes will be the primary method of moving ore between levels. Declines are designed at 11.25% to 13.5% gradient with turning radiuses of 75 ft, or greater. The level access and decline intersect will have a 40 ft long straights on both sides with a lower gradient of 5% to ensure the safety of vehicles exiting from the access.

Ore Passes

Ore passes are applied at 80° incline with ore finger passes inclined at 50° to allow LHDs to directly dump into the ore pass.

Two ore passes are utilized for the EN zone spanning from 1920 Level to 1550 Level. Material from the remaining 1920 to 2000 Levels will be trucked up the spiral decline. The ore pass located in the northwest portion of the deposit will be sacrificial in order to minimize hauling distance for extracting higher valued material during the initial portion of the mine life. The ore pass will be used as a slot raise once mining enters the area where it is located. The second ore pass will be permanent and service the remainder of material throughout the mine life. It will be located northeast of the spiral declining ramp. The permanent ore pass links up to the main EN access drift at the 1990 Level where mine trucks will transport material to the COBs in the Main Shaft Area. The sacrificial ore pass ends on the 1920 Level and its material will be rehandled to the permanent ore pass or loaded directly onto trucks as needed.

The ES zone does not utilize ore passes and will truck material through spiraling declines to COBs on the 1990 Level.

The E2 zone will consist of an ore pass servicing 1140 to 1740 Levels with the remainder of material moved by truck up the spiral decline for 1840 Level and below. Material at 1040 Level will be trucked down to the 1140 Level to be placed in the ore pass there.

Battery Charging & Exchange Stations

Two multiple-battery charging and exchange stations are located on the 1990 Level to accommodate battery-operated trucks and LHDs. Each station consists of a T-drift, which allows the vehicle to park in a position where the depleted battery can be switched out and replenished with a fresh battery using an overhead crane. The crane will also be utilized to lift fully charged batteries and place in the cool down area.

Single-battery exchange stations will be placed for E2 to enable machines working in the area to optimize utilization that would otherwise be taking up traveling to the main change-out station.

Fueling Bays

Diesel fuel will be used for some support equipment for the development. Fuel bays are located near the workshop on the 1990 Level.

Workshops

Workshops are located in the 1990 Level within the Main Shaft Area for servicing mine equipment.

Refuge Chambers

The refuge stations will be placed at every entry access in the Pumpkin Hollow mine (Main Shaft, ES Vent Raise, EN Vent Raise and the E2 Vent Raise). Stations will be moved between levels in order to be closer to working areas as the mine progresses. Each refuge station will fit 16 people at a time and will meet all required Mine Safety and Health Administration (MSHA) regulations. The refuge stations will be air-locked and contain enough food and water for 16 people, as well as radio communication to surface, lighting, seating and first aid containers.

Explosives Magazine

Explosives magazines were included as required to safety store explosive materials.

Other items included in the design are concrete bunks, transfer pump, a jib-crane, an electric hoist, a trolley, jib monitor, access signs, services and lighting. There will also be sufficient parking room.

Major Pump Stations & Sumps

There will be four major pump stations in the Underground Project. Two will be associated with the ES and EN zones and be located adjacent to the Main Shaft at the 2150 and 960 Levels. The other two will be in the E2 zone located at the 1790 and 2640 Levels.

The main pump stations will generally consist of two development headings. The first heading will be declining and will receive water inflows from level sumps or other gravity flows and be used to decant the solids out of the water. The second heading will have a concrete box installed in it with an outflow to the pumps, which will pump the decanted water to surface.

Level sumps will be located on each development level, usually at or near the level access. There will be declining headings to allow for solids to decant out. Sumps on levels above pump stations will have boreholes drilled that will allow water to gravity flow down to the subsequent level sump until it reaches a pumping station. Sumps below pumps stations will have submersible pumps installed that will pump water up to the subsequent levels (up to 100 ft) until it discharges at a pump station.

All decanting areas in sumps will be designed to allow mucking out with an LHD unit.

Capital Vertical Development Design

East Shaft

Access to the underground mine will be serviced by the production shaft measuring 24 ft in diameter extending approximately 2,100 ft deep. The shaft will include a cage, skip, services and ventilation compartments. The production shaft will function as labor, equipment and materials delivery to the mine, hoisting of ore and waste and supply of services, including electricity, compressed air, mine dewatering, fresh water and fresh air. During initial mine development, the main production shaft will also contain ducting to carry exhaust air from the mine.

Ventilation Raises

A total of three vent raises are planned and will be located in EN, ES and E2. Each raise will be 12 ft in diameter (3.6 m) and excavated by pilot and ream raisebore. They will be supported with bolts, screen and fibercrete (where required to allow for utilization as an emergency egress via a hoist system). Paste fill lines will also be installed within these raises.

Two of the ventilation raises (EN and E2) that connect to surface will be equipped with hoist systems for emergency egress. Ventilation raises that connect levels will have MSHA- and State of Nevada-compliant ladderways between levels to provide emergency egress from these levels.

Operating Development Design

Development gradients are applied to footwall drives and crosscuts within each mining level. This will allow for LHDs to take advantage of regeneration opportunities while traveling loaded downhill to the ore pass. The gradients range between 0% and 7.5% but are typically at 5%. A representative operating development sectional and plan view can be seen in Figure 16-5 and Figure 16-6, respectively.

The majority of tunnel profiles for operating development consisted of 13.5 ft by 13.5 ft.

Figure 16-5 : EN Zone Sectional View 1 252 425N (2017 Technical Report)

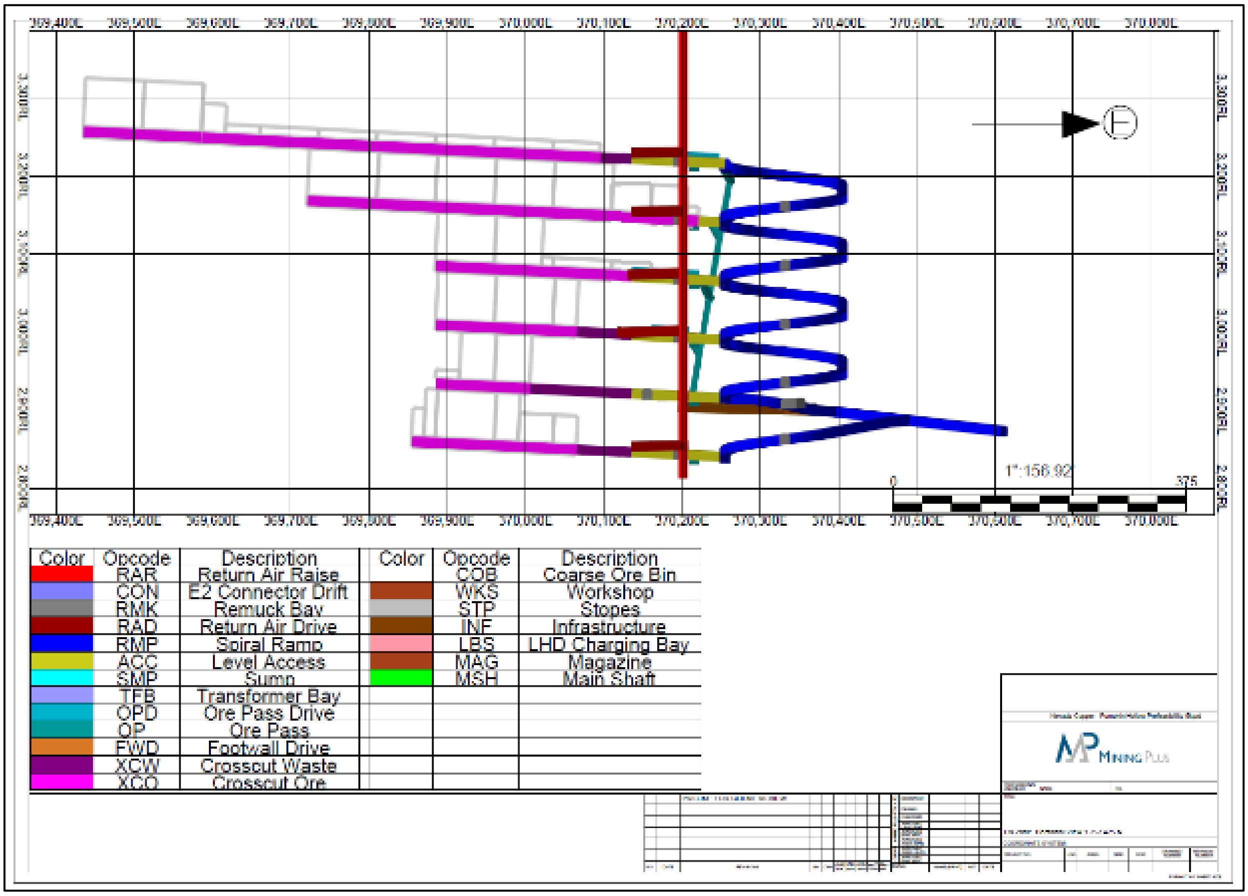
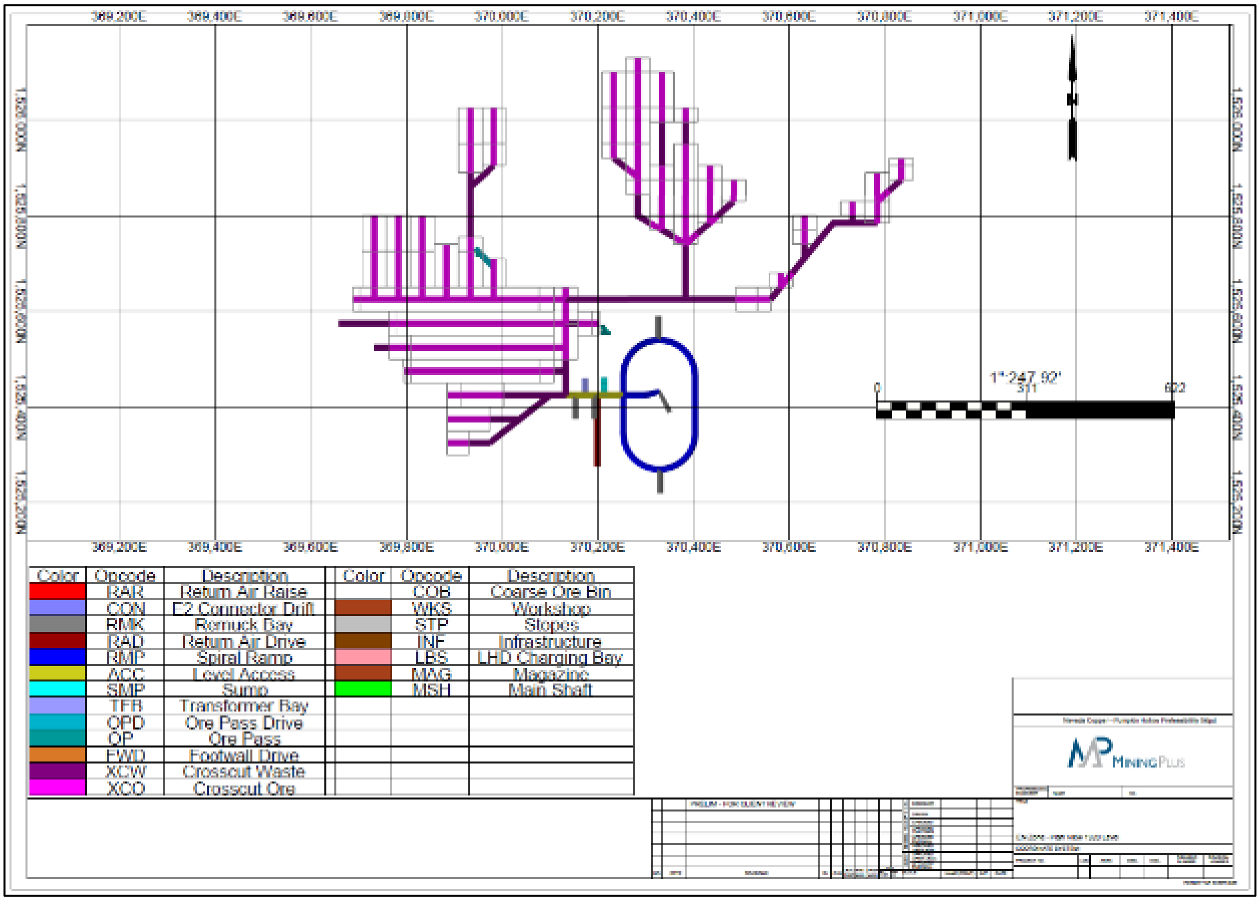


Figure 16-6: EN Zone – Plan View 1920 Level (2017 Technical Report)



Development Summary

A summary of capital and operating development by zone and development type is provided in Table 16-15.

Table 16-15: Total Foot Development by Zone

	Main Infrastructure Area	EN	ES	E2	Total
Capital Development					
Lateral	8,136	6,823	6,025	37,002	57,987
Vertical	2,316	2,782	2,150	3,857	11,106
Operating Development					
Lateral	0	59,493	31,561	26,760	117,814

A summary of operating development is provided in Table 16-16.

Table 16-16: Operating Lateral Development

Operating Development	Ft
Digout1	56,873
Development	117,814

Notes:

1. Digouts are development where CPF is mined and ground supported through again in order to access behind the previously mined stope.

16.1.9.2 Production Design

The overall mining method for the underground mine is described in Item 16.1.2.

Additional small flat-lying mining areas close to the E2 connector drift (west of the ES deposit) and below the E2 connector drift were evaluated to whether they could be efficiently recovered using alternative mining methods. However, these areas were deemed uneconomic to mine due to substantial development requirements, low recovery rates and relatively low cash flow generation.

East North stopes are designed using 75 ft level intervals, 50 ft widths and 75 ft lengths. Stopes mined have a vertical span of 450 ft from zone levels of 2000 to 1550. ES stopes consist of 100 ft level intervals by 50 ft widths and 75 ft lengths. Production spans 400 ft between 1850 and 2250 Levels in the ES and EN zones. The E2 zone will have stopes designed at 100 ft level intervals by 50 ft lengths and variable width. Stopes mined span 1,000 ft vertically between 1040 and 2140 Levels in the E2 zone. In order to maximize on material extraction for the East Deposit, sub-shapes from MSO were captured from the bottom up in incremental stope heights of 25 ft.

Table 16-17 summarizes key design parameters for the production design.

Table 16-17: Pumpkin Hollow Key Design Stope Parameters

Zone	Depth		Stope Height (ft)	Stope Width (ft)	Stope Length (ft)
EN	1,425	2,185	75	50	75
ES	1,680	2,660	100	50	75
E2	835	3,005	100	50	75

Stope Blast Ring Design

Stope blast ring designs were created for 100 ft high stopes in the E2 and ES zones and 75 ft high stopes in the EN zones. Each stope will have a drop raise (8 ft by 8 ft) that will be placed to allow for rock expansion. The first three rings will be spaced 6.5 ft apart to accommodate for the drop raise with the remaining rings evenly spaced at 8 ft intervals. Ring designs will consist of 4-inch diameter blast holes with ring and toe spacing of 8 ft.

16.1.9.3 Mine Dilution & Ore Recovery

Development Dilution

Based on RQD (Deere 1964), Tunnel Quality Index and typical blasting practices, it is estimated that the average back and wall overbreak in development will be 0.5 ft, resulting in an average waste development heading dilution of 11.4%. In general, waste development percent dilution will be calculated, as per the equation below:

$$Percent\ Dilution = \frac{ParentWidth + 2 * 0.5) * (ParentHeight + 0.5)}{ParentWidth * ParentHeight}$$

Ore development will not have dilution applied as this may result in double counting of ore (overlapping of development and stope boundaries). Given the relatively massive nature of the ore body, it is expected that ore development dilution will have the same grade as the development activity.

Stope Mining Dilution

The expected dilution from stope and paste fill walls is 1.5 ft. Given the nature of the ore body, the host rock adjacent to the stopes is assumed to have a grade of 75% of the cutoff grade (or approximately 0.75% Cu), while paste fill is assumed to have a grade of 0% Cu. The average number of host rock stope walls for each stope, and paste fill walls for secondary stopes, were considered in estimating dilution and grade for primary and secondary stopes, as summarized in Table 16-18.

Table 16-18: Average External (Host Rock) Stope Walls by Zone

Zone	Primary Stope Dilution		Secondary Stope Dilution		Development (13.5 ft by 13.5 ft)	
	Grade %	Percent %	Grade %	Percent %	Grade %	Percent %
ES	0.75	3.10	0.24	10	0.24	10
EN	0.75	2.50	0.19	10	0.19	10
E2	0.75	5.00	0.38	10	0.38	10

Mining Recovery

Mining recovery has been calculated for development and stopes based on quantitative estimates (considering stope geometries and experience) of losses associated with the following three factors:

- Pillars: preventing diagonal overlap between secondary stopes

- Drill and blast: losses at the upper limits of typical stope blasted rings
- Mucking
- Development: Unrecovered broken ore on the floor of development
- Stopes: Unrecovered broken ore remaining in the stope corners adjacent to the stope drawpoint

Table 16-19 summarizes mining recoveries applied in this Underground PFS.

Table 16-19: Summary of Mining Recovery for Stopes & Development

Parameter	Stope Losses (%)		Development Loss (%)
	East North Zone	East South and E2 Zones	
Pillars	1.9	1.9	n/a
Drill & Blast	2.1	1.6	n/a
Mucking	1.4	1.0	1.8
Total Recovery	**94.9**	**95.7**	**98.2**

16.1.10 Production Sequence & Scheduling

The Underground Project is scheduled assuming continuous operations using two 12-hour shifts per day. Operational time is calculated at 8.4 hours per shift after breaks, travel time and other delays are considered.

The Underground Project is scheduled over 15 years including pre-production development. Full ore production utilizing the mining methods described in Item 16.2 is expected to be reached by the fourth quarter of Year 2. Stope production from E2 is scheduled to begin in Year 4. Estimated annual production from Year 3 to Year 15 is 1.8 Mst with an average Copper Equivalent head grade of 1.75%.

The production sequence has been developed to allow for the initial targeting and extraction of higher grade ore, while still allowing for high overall recoveries over the life of the underground mine.

The overall goal of the mine sequence and schedule was to maximize the ore grade processed by the plant during the first few years of production in order to reduce the capital payback period.

16.1.11 Preproduction Development Schedule

The Main Shaft will restart sinking at the beginning of Year 1. Mine infrastructure (i.e., workshops, the shaft loadout area, ramps, level access and charging bays) will be established during this year, as well as the sinking of the EN ventilation raise. Waste rock and small quantities of ore material will be mined from lateral development in Year 1. Ore material will come from crosscut development in the ES and EN zones and will be stockpiled on the surface for the remainder of the year.

At the end of Year 1, a temporary hoist will be installed on the EN ventilation shaft. This hoist is planned to operate for months 1 to 4 of Year 2, when the Main Shaft is being commissioned for production hoisting. Manpower and supplies will enter and exit, and ore and waste material will be removed, using the temporary hoist. Once the Main Shaft is fully commissioned, the temporary hoist will be relocated to the Main Shaft and will be installed as the permanent cage hoist. This scenario was studied in a trade-off and found to be favorable as it allows the mine to reach production three months sooner than if the mine was shut down during Main Shaft commissioning.

At the end of Year 2, the underground mine will be producing ore at 5,000 stpd in a steady-state from the ES and EN Deposits. Starting in Year 4, the E2 Deposit will begin production, mining upwards from the E2 connector drift access. The lower portions of E2 will come online in two stages, the first stage starting in Year 10 and the second stage starting in Year 12.

16.1.12 Production Sequencing

16.1.12.1 East South & East North Stope Production Sequence

The ES and EN stope production sequence can be generally described as bottom-up, primary-secondary stoping in a “checker-board” extraction sequence with in-ore footwall drifts consisting of tertiary stoping. In the sequence, primary and secondary stopes alternate on the same crosscut and alternate on placement between adjacent crosscuts.

Primary stopes in the ES and EN are extracted from hanging wall to footwall, skipping over a secondary stope. After a primary stope is extracted, the secondary stope directly behind it is extracted before moving onto the next primary stope in that crosscut.

Primary stopes will be extracted in panels of about four to six crosscuts going from north-to-south, west-to-east or east-to-west dependent on the area. The first crosscut of a panel must be extracted before the final crosscut of an adjacent panel can be completed.

Secondary stopes are accessed by drifting back through the cemented fill of the primary stopes. Secondary stopes will also be extracted from hanging wall to footwall, with secondary stopes closer to the hanging wall requiring extraction before secondary stopes in adjacent crosscut and in-line secondary stopes closer to the footwall can be extracted.

Extrapolating the sequence bottom up from level-to-level, an entire crosscut’s stopes must be extracted before moving up to a level above. As primary and secondary stopes are completely exhausted on a level, the tertiary stopes on the level below will be taken.

In order to achieve the desired high grade production strategy, exceptions to the mining sequence in ES and EN were made. In particular, the 1950 Level in ES will be mined first, even though it is not the lowest level of the deposit. Also, selected high grade stopes will be extracted early in the mine life that do not adhere strictly to the “checkerboard” sequence described above.

16.1.12.2 E2 Stope Production Sequence

The E2 deposit will be mined in a bottom-up primary-secondary stoping with an in-waste footwall drive. Veins thinner than 15 m in width will be mined as longitudinal stopes. These longitudinal stopes are typically at the eastern and western extremities of the deposit. There is a smaller vein in the footwall that will be mined longitudinally.

The sequence starts at the 1840 Level and moves upwards targeting higher grade stopes. As stopes are exhausted on the 1840 Level and above in elevation, lower parts of the mine are opened up for mining in two stages. The first stage opens up 2140, 2040 and 1940 Levels. The second stage opens up 2740, 2640, 2540, 2440, 2340 and 2240 Levels.

16.1.13 Mine Strategy

The business strategy applied to the Underground Project was to isolate and prioritize high grade ($90/st or higher NSR value) stoping areas in ES and EN for initial production with minimal development, and afterwards add additional development to access lower grade stopes once mining of high grade stopes was complete. In many cases, the later additional development includes digout development through CPF placed in high grade stopes mined early in the mine life.

In particular, the 1950 Level in ES was identified and developed in the schedule as the best level to reach early-access high grade ore and have secondary stopes available for depositing development waste while hoisting capacity is being established.

When full production from high grade stoping areas in ES and EN is nearly exhausted, the mine plan brings in production from E2 above the 1840 Level.

16.1.14 Mining Manpower

Mining salaried personnel estimates were developed to provide adequate supervision and technical support for the daily operation of the mine. There are 269 required personnel for the Underground Project.

Contractor personnel estimates were developed for the pre-production operations (shaft sinking, off-shaft development, lateral and vertical development). There are 231 required contractor personnel for the Underground Project.

16.1.15 Mobile Equipment

As far as possible, mobile equipment will be battery operated. Diesel equipment would only be considered where a BEV alternative is not available in the current marketplace, or where BEV equipment has been deemed uneconomical or operationally unviable. This decision was based on two general criteria:

- Lowering of ventilation costs
- Confidence that battery operated trucks and scoops that are available in the marketplace today would be able to meet production targets

The Underground Project will initially use the battery-energized LHDs and haul trucks as its primary movers. These have been in operation at underground mines in Northern Ontario and have performance, availability and utilization metrics. While several equipment manufacturers intend to produce a larger 14-tonne LHD in the near term, the specifications for this equipment are not available and its performance has not been proven. As such, those machines will not be included or considered in the LOM mobile equipment fleet in the Underground PFS.

When considering the sizing of excavations in some areas of the mine, future models of larger BEVs were considered. Several manufacturers intend to produce not only larger LHDs but also a matching larger haulage trucks. Comparable diesel units of larger LHDs and haul trucks would require different minimum dimensions in access drifts and working areas due to ventilation requirements. When evaluating transport drift sizing, it was determined that there is a higher probability that a larger LHD would become available in the near future and utilized in operations instead of a larger truck. Therefore, the drift profile of 13.5 ft by 13.5 ft was sized to accommodate for larger scoops.

LHDs, haulage trucks, scissor lifts, jumbo drills, bolters and longhole drills have been quoted and included in the PFS study as battery operated. All other equipment will be diesel operated, as this was proven to be most economic.

During the initial off-shaft development, it is expected that some diesel LHDs, drills, bolters and haulage trucks will be used until proper battery change-out infrastructure is in place.

A summary of equipment, capacity and key dimensions and a schedule of maximum equipment numbers by year is shown in Table 16-20.

Table 16-20: Battery Equipment Key Metrics & Schedule

Battery Equipment	Power Source	Capacity	Project Year Dimensions (in, L x W x H)	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Loaders	Battery	7 tons	396 x 93 x 96	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Trucks	Battery	22 tons	356 x 90 x 97	0	5	6	7	7	7	7	7	7	8	8	8	8	8	3
Face Drill Rigs	Battery	259.2 x 403.2 in	553.1 x 100.4 x 125.2	0	4	5	8	9	9	9	9	9	9	9	9	9	9	4
Production Drills	Battery	334.6 x 224.4 in 335.4 x 220.5 in	410.2 x 87.0 x 122.0 422.0 x 87.0 x 123.6	2	3	4	4	3	2	2	2	2	3	3	2	2	2	1
Bolters	Battery	8.2 x 11 yd	553.1 x 100.4 x 125.2	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Scissor Lift	Battery	7 ft 6 in x 12 ft deck	317 x 91 x 101	2	3	4	5	5	4	4	4	4	4	4	4	4	4	1
Rock Breaker	Diesel		364 x 72 x 99	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fork Lift	Diesel	2000 lb	133 x 65 x 78	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Grader	Diesel		342 x 144 x 123	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer	Diesel		427 x 93 x 97	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Transmixer	Diesel		317 x 93 x 108	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Personnel Carriers	Diesel		260 x 80 x 80	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Duty Vehicles	Diesel		260 x 80 x 80	0	2	4	4	4	4	4	4	4	4	4	4	4	4	4
Service Trucks	Diesel		323 x 93 x 90	0	2	3	4	4	4	5	5	5	5	5	5	5	5	5
ANFO Loader	Diesel		359 x 78 x 90	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2

16.1.15.1 Mine Equipment Productivities

Battery-operated equipment has been shown to perform with the same productivity as the diesel versions of the same machine models and at times outperform them. When modeling the productivity of the equipment, known values for diesel equipment were used to calculate travel speeds, loading times and dumping times. The equipment manufactures agreed that these assumptions are acceptable based on BEV performance in the field.

The following tables show the average loading and dumping times for the LHDs and the haulage trucks, along with their assumed bucket capacities.

Equipment suppliers have provided information on the primary fleets utilized for the Underground PFS. In Table 16-21 and Table 16-22, the base information for calculating equipment performance is presented. Results where compared to industry standards and industry benchmarked with similarly sized machines and are considered acceptable.

Table 16-21: Productivity for Loaders & Haul Trucks

Equipment	t/h	t/d
BEV 7-ton Loader	63.7	1,069
BEV 20-ton Haul Truck	55.7	936

Table 16-22: Productivity for Bolters, Longhole & Jumbo Drills

Equipment	ft/hr	ft/d
Bolter	0.57	10
Longhole Drills	53.68	902
Jumbo Drills	1.28	22

BEV development and production drilling equipment are more readily available than prime movers due to the reduced requirement for tramming time under motive force, compared with their time connected to the mine power distribution system while working at the face. Productivities from Table 16-21 and Table 16-22 were utilized to calculate equipment drill requirements for the mine plan.

During the first two years of the mine life, critical rate productivities were utilized in order to minimize equipment requirements to alleviate initial capital costs and due to limited space within the underground mine.

The rest of the equipment was considered non-critical path and though productivities were estimated for some of the equipment, it was not found to have an effect on the outcome of the mine plan.

16.1.15.2 Compatibility of Equipment

Compatibility of equipment sizes has been confirmed with the smallest tunnel profile 13.5 ft by 13.5 ft. Both tramming (7-ton BEV loader) and haulage (20-ton BEV haulage truck) equipment have respective lateral clearances of 6.0 ft and 5.5 ft and height clearances of 2.5 ft and 1.3 ft. These clearances are within regulatory requirements for mobile equipment in a travel way with personnel.

Selected production drills, jumbos and bolting equipment have lateral clearances of 5.0 ft, 6.2 ft and 6.2 ft respectively.

16.1.16 Mine Operations

The Underground Project will be a mechanized underground mine using primarily battery-operated equipment. A primary-secondary longhole stoping production method with paste fill will be used for ore extraction. The shaft accessed mine will operate 24 hours a day on two shifts of 12 hours each.

The Underground Project will use LHD scoops to load the blasted rock from the development face or the stope drawpoint. Material will either be loaded into ore passes or onto haulage trucks, before being transported to its final destination. Once the underground mine is in full production, waste rock will cease being skipped to surface and be dumped back into secondary stopes as fill.

Backfill will consist of CPF for primary stopes and UPF or URF for secondary stopes.

Lateral development and transport will be primarily in 13.5 ft by 13.5 ft drifts. These will be developed primarily using twin boom jumbos.

Stopes will be drilled using battery-operated longhole drills. The majority of the stopes (>90%) will be drilled at full length from above; however, a minority will be drilled upwards at lesser heights from below.

Compressed air, electrical power, mine water and ventilation tubing will be installed in the transportation drifts, on the levels and into the crosscuts where needed.

16.1.16.1 Truck Loading versus Ore Pass

To evaluate the need for ore passes, each level of each Eastern Area deposit was analyzed and given a cycle time for loading trucks on the level, hauling to the associated COB, loading the ore passes with material and rehandling the material to the associated COB.

The lower levels for both scenarios considered the requirement for trucking directly to the COBs. For the ore pass scenario, rehandling costs were considered to bring material to the required COBs. Material utilizing the ore pass will incur rehandling costs associated with trucking this material to the COB. The lower levels of E2 and EN that are not serviced by the ore pass will require material to be trucked directly to the COBs.

Ultimately, the use of ore passes in EN and E2 presented higher productivities when compared with truck loading. As well, the overall costs incurred are predicted to be less for the ore passes. In ES, all material will be truck loaded, as the location of the deposit relative to the COBs favors this approach from an overall cost perspective.

16.1.16.2 Drilling & Blasting

The Pumpkin Hollow underground mine will have drilling and blasting operations for the following activities:

- Shaft sinking and off-shaft development
- Lateral development

- Vertical drop raise development
- Longhole stoping

Drilling and blasting equipment and operations are outlined in the following subsections by activity.

Shaft Sinking & Off-Shaft Development

Shaft sinking will be performed by an experienced mining contractor whom has executed this operation previously. The shaft at the Underground Project has already been sunk to the 1910 Level (to a depth of 1,910 ft). The excavation for the remainder of the shaft will be performed using conventional blind sinking (drill and blast) techniques. A hydraulic shaft jumbo will drill the shaft bottom. Blasting will utilize emulsion explosives with non-electric blasting caps. The shaft will be sunk an additional 237 ft to a final depth of 2,147 ft.

Off-shaft development drilling will be done by an experienced mining contractor, using jacklegs and manually loading ANFO and emulsion cartridges. In most cases, advances will be taken in two benches in a “split face” advance, where the upper face of the tunnel is drilled, blasted and mucked out and the next blast advances the lower face of tunnel is then advanced. All development headings that come off of the shaft will have minimum profile of 16 ft by 16 ft.

Lateral Development

The lateral development scheduled at the Underground Project will primarily be drilled using twin boom battery powered and tethered jumbo drills. Blast holes will be loaded with emulsion cartridges or ANFO. The lifters and knees at the base of the design will be loaded with emulsion cartridges as the mine is expected to have some moisture. Holes that do not present a risk of contamination from moisture will be loaded with ANFO from a diesel powered ANFO loader or a hand held loading mechanism.

Vertical Development

Vertical development scheduled for drilling and blasting operations at the Underground Project will consist of 8 ft by 8 ft drop raises and ore pass dump point fingers. Ventilation raises to surface are planned to be excavated using raisebores.

Drilling will be done by a battery powered tethered longhole drill. ANFO will be loaded using a diesel powered ANFO loader. The vertical development is not expected to require emulsion.

Longhole Stoping – Production Drilling & Blasting

The production stopes at Underground Project will be drilled using battery powered tethered longhole drills. Blast holes will be 4 inches in diameter and rings will be spaced 8 ft apart. Stopes are 50 ft wide and 75 ft long and vary from 75 to 100 ft high.

Blast holes will be loaded with ANFO and if water is present, emulsion. As all production stopes will have an access developed underneath to allow for the slot raise to be developed and mucked out, it was estimated that emulsion requirements would be minimal. Therefore, total emulsion content in the longhole production blast holes was estimated at 20%.

16.1.16.3 Off-Shaft Development Plan

Off-shaft development will occur on three levels:

- 2036 Level: The first main level will be where all manpower and supplies will enter the mine. Located adjacent to the workshop.
- 2060 Level: The skip loadout system will be installed on this level.
- 2140 Level: Shaft bottom. This access will be used to remove debris and other accumulated materials from the bottom of the shaft and will serve as an emergency water storage area should the Main Shaft pumping station fail during peak water inflows.

Off-shaft development advances will be taken in two benches in a "split face" advance where the upper portion of the tunnel is drilled, blasted and mucked out using slushers or other semi-mechanized equipment such as small backhoes. Once the upper half of the tunnel has been blasted and ground supported, a lower portion of the tunnel will be blasted, mucked out and ground support installed. The tunnel will maintain the split face advance, where the upper portion of the tunnel will always be one round ahead of the lower portion of the tunnel while it is being developed using jacklegs.

16.1.17 Shaft Sinking Operations

The first activity in the mine schedule is to sink the Main Shaft the remaining 237 ft to its final depth of 2,147 ft. The Main Shaft is currently at a depth of 1,910 ft. Ground support has been installed to the full depth of the shaft, and a concrete lining has been installed to a depth of 1,888 ft. A 14 ft diameter double drum hoist was installed and will be used for the construction of the Main Shaft. Upon completion of the shaft sinking, the sinking hoist serves as the production hoist.

The excavation for the remainder of the shaft will be performed using conventional drill and blast techniques. A hydraulic shaft jumbo will drill the shaft bottom. Blasting will utilize emulsion explosives with non-electric blasting caps. The blasted material will be loaded into muck buckets using an excavator and then hoisted to the surface. Temporary ground support will be 8 ft friction bolts installed in a 5 ft by 5 ft pattern and a 12-inch concrete lining will be installed in conjunction with shaft sinking. The average shaft sinking rate is expected to be 5 ft per day, allowing for breaks due to off-shaft development and unplanned delays such as weather, power outages etc.

A probe hole will be drilled in the shaft bottom at regular intervals to detect the presence of ground water. It will be kept a minimum of 20 ft below the excavated shaft bottom. Drilling and grouting will be completed for zones of high water inflows. Two pumping station will be used to pump water discharge to the surface; the first at 960 ft depth and the second at the ultimate shaft bottom. The 960 Level pump station will be upgraded and maintained for production. The second pump station near the shaft bottom will serve the entire Eastern area.

16.1.18 Mine Production Schedule

The Underground Project mine development and production schedule is summarized in Table 16-23.

Table 16-23: Mine Development & Production Schedule

Project Year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Name	Units	Total															
Cap Lat Devt	Ft	57,819	13,876	4,635	11,469	8,921	2,749	48	112	201	-	5,620	-	8,562	1,625	-	-
Op Lat Devt	Ft	174,687	3,946	23,801	18,352	14,727	14,438	11,979	9,193	13,455	11,607	10,548	9,829	8,740	9,591	11,973	2,507
Lat Devt	Ft	232,506	17,822	28,436	29,821	23,648	17,187	12,028	9,305	13,656	11,607	16,169	9,829	17,302	11,216	11,973	2,507
Vert Devt	Ft	9,363	2,827	1,767	1,545	1,465	305	236	106	200	-	313	-	502	98	-	-
Devt Ore Tons	Million st	1.93	0.07	0.31	0.25	0.12	0.16	0.14	0.12	0.15	0.15	0.1	0.12	0.1	0.11	0.04	0
Stope Ore Tons	Million st	21.98	-	0.99	1.59	1.68	1.68	1.68	1.71	1.68	1.65	1.75	1.72	1.72	1.71	1.8	0.61
Feed Tons – East South	Million st	6.6	0.02	0.94	0.59	0.21	0.02	0.08	0.28	0.36	0.79	1.03	0.81	0.5	0.43	0.36	0.17
Cu	%	1.63	1.34	2.02	1.49	2.61	2.04	1.26	1.44	1.56	1.55	1.72	1.58	1.49	1.4	1.19	1.19
Au	Oz/st	0.004	0.004	0.005	0.004	0.005	0.005	0.004	0.005	0.004	0.004	0.005	0.004	0.004	0.004	0.003	0.003
Ag	Oz/st	0.087	0.057	0.111	0.076	0.121	0.082	0.055	0.072	0.085	0.079	0.1	0.087	0.077	0.068	0.076	0.045
Feed Tons – East North	Million tons	11.21	0.05	0.35	1.24	1.2	1.02	0.89	0.87	1.24	0.88	0.71	0.64	0.91	0.72	0.44	0.04
Cu	%	1.54	0.88	1.66	1.62	1.84	1.74	1.63	1.58	1.42	1.47	1.45	1.39	1.34	1.36	1.21	1.28
Au	Oz/st	0.006	0.004	0.006	0.006	0.007	0.007	0.006	0.006	0.006	0.006	0.006	0.006	0.005	0.006	0.005	0.005
Ag	Oz/st	0.135	0.125	0.141	0.14	0.145	0.15	0.145	0.151	0.123	0.127	0.124	0.112	0.127	0.138	0.109	0.14
Feed Tons – E2	Million st	6.1	-	-	0	0.38	0.8	0.84	0.69	0.23	0.13	0.11	0.39	0.4	0.67	1.04	0.41
Cu	%	1.64	-	-	2.07	1.85	1.81	1.93	1.99	1.98	1.69	1.67	1.45	1.76	1.26	1.3	1.32
Au	Oz/st	0.009	-	-	0.018	0.011	0.011	0.013	0.014	0.014	0.008	0.008	0.008	0.009	0.006	0.006	0.006
Ag	Oz/st	0.205	-	-	0.288	0.271	0.233	0.246	0.239	0.245	0.242	0.236	0.201	0.255	0.15	0.132	0.135
Feed Tons – Total	Million st	23.91	0.07	1.29	1.84	1.8	1.84	1.82	1.84	1.83	1.8	1.85	1.84	1.81	1.83	1.84	0.62
Cu	%	1.59	1	1.92	1.58	1.93	1.77	1.75	1.71	1.52	1.52	1.61	1.49	1.47	1.34	1.26	1.28
Au	Oz/st	0.006	0.004	0.005	0.006	0.007	0.009	0.009	0.009	0.006	0.006	0.005	0.006	0.006	0.005	0.005	0.005
Ag	Oz/st	0.139	0.107	0.119	0.119	0.169	0.186	0.188	0.172	0.13	0.115	0.118	0.119	0.142	0.126	0.116	0.111
Cemented Paste Fill Volume	yd3	5	-	0.27	0.36	0.44	0.31	0.41	0.2	0.35	0.37	0.45	0.32	0.32	0.52	0.48	0.18
Uncemented Paste Fill / Rock Fill Volu	yd3	3.86	-	0.08	0.31	0.26	0.35	0.24	0.44	0.34	0.25	0.23	0.36	0.41	0.24	0.26	0.08

16.1.19 Mine Services

16.1.19.1 Compressed Air

The underground mine will have a main compressor house on surface located near the Main Shaft. Compressed air will enter the mine via 6 inch piping down the Main Shaft. Compressed air will be distributed using six-inch pipe on all the main transport travel ways and ramps throughout the mine. On footwall drifts and crosscuts, four-inch piping will be used to distribute compressed air to the required areas.

16.1.19.2 Dewatering

There will be four major pump stations in the underground mine.

The first pump stations to be developed will be associated with the Main Shaft. One will be on the lowest 2150 Level, off of the bottom of the Main Shaft, which will pump water up the shaft to the other station will be at the 960 Level. From the 960 Level water will be pumped up the shaft to surface. Both stations will be able to pump the maximum flow of 1,800 gpm that is predicted for the Underground Project.

The other two pump stations will be located in the E2 zone. The first pump station will pump all water to surface from the 1790 Level via the E2 ventilation raise. The final pump station will be located on the 2640 Level of E2 and will pump water up to the main E2 pump station on the 1790 Level via boreholes. Both pump stations are designed to handle maximum inflows of 1,300 gpm as projected in previous studies for the E2 zone.

Submersible pumps will be used during mine development and to pump water from lower level sumps to pump stations. Upper level sumps will allow for gravity drainage of water collected on the level to the sump on the subsequent level below via boreholes.

16.1.20 Backfill

Primary stopes in the EN, ES and E2 zones will be backfilled using CPF. CPF is expected to comprise 72% mill tailings, 7% cement and 21% water.

Secondary stopes in all zones will preferentially use URF from lateral development. Where digouts will have to be mined back through secondary stopes in order to access lower grade stopes, secondary stopes will need to be filled with CPF. All other secondary stopes will be backfilled with UPF. Uncemented paste fill is expected to comprise 73% mill tailings and 27% water.

For all stopes that will be backfilled using CPF or UPF, a bulkhead will be constructed at all access points once the stope has been mined out. The stope will then be filled with paste delivered by a piping network from the paste fill plant. The paste plant will be located on the surface and booster pumps will be used where necessary to transfer paste fill through the mine workings to the fill point.

There is expected to be sufficient paste fill available to provide CPF and UPF for all primary and secondary stopes, except prior to commissioning of the process and paste fill plants. Primary and secondary stopes mined prior to commissioning the process plant will preferentially be backfilled temporarily using URF. During this period, there will not be sufficient waste rock fill available from development to backfill all stope voids, and some stopes will need to remain unfilled until the paste fill plant has been commissioned. Stopes to be temporarily backfilled with URF during this period will be prioritized based on ground conditions and mining sequence. Once the paste fill plant has been commissioned, all stopes that were temporarily backfilled with URF will be progressively emptied of URF and re-backfilled with UPF or CPF as appropriate.

16.1.20.1 Digouts

In the mining sequence, it will be required at times to mine back through the CPF placed in the primary and secondary stopes in order to recover the secondary stopes that are behind them in the sequence. Any stope that is identified as having this requirement for a digout, will have a specific quality and depth of CPF placed in it to allow for an unconfined compressive strength of 1 MPa. This will allow operations to mine back through the CPF and extract the economic material on the far side of the backfilled stope. Digout footage has been accounted for, along with associated ground support costs that are required to perform this activity.

16.1.21 Mine Ventilation

The dilution of diesel fumes is the primary factor for ventilation requirements and a significant advantage of battery equipment is that it does not emit fumes and thus requires less air volume than diesel equipment. With a primary fleet of battery equipment, the ventilation of the Underground Project was primarily influenced by the removal of blasting fumes and the dissipation of heat generated by the battery powered equipment. The diesel auxiliary equipment’s impact on ventilation is minimal due to low utilization. The mine strata are shallow, cool and non-gassy and will not require additional ventilation.

There currently are no regulations in the United States on ventilation requirements in which battery-operated equipment is in use. Data in general are also limited for the performance and ventilation requirements of battery equipment in mines. A study by Halim and Kerai (2013) confirmed the heat from a battery-powered LHD was equal to its power consumption. This study involved simulation of the heat distribution within theoretical mines utilizing BEVs. The study resulted in unit values to be applied for use of BEVs as shown in Table 16-24. The unit values were used for Pumpkin Hollow’s ventilation design. It appears this is a conservative assumption.

At this time there is insufficient information available regarding geothermal gradient in this area to consider the impact of elevated rock mass or water inflow temperatures as factors in ventilation design. The underground mine is assumed to be a shallow, cool mine, therefore requiring 0.025 m^3/s per kilowatt of power. This is significantly less than the approximate 0.06 $m^3/s.kW$ for a diesel-powered machine. The overall risk of an elevated geothermal gradient in this area that would significantly alter the ventilation requirements is considered to be low due to the use of conservative assumptions in the ventilation approach used in the Underground PFS.

Table 16-24: Summary of Estimated Unit Ventilation Requirements for Electric Vehicles

Mine	Thermal Condition	Airflow Requirements	Note
Deep Mine	Cool	0.04 m3/s.kW	1.5 MW surface refrigeration plant
Deep Mine	Hot	0.04 m3/s.kW	5 MW surface refrigeration plant
Shallow Mine	Cool	0.025 m3/s.kW	No surface refrigeration plant required
Shallow Mine	Hot	0.037 m3/s.kW	1 MW surface refrigeration plant

Source: Halim and Kerai 2013.

The required air quantity to clear the blast fumes was calculated to be 160,000 cfm at the main exhaust for the level being blasted. This allows blast fumes to be cleared within 60 minutes. The primary ventilation fans will be equipped with variable speed drives to increase fan rpm and air volume during this time. This will be the largest amount of air required in a production or development heading and was modeled in VentSim to size appropriate fans and primary ventilation devices.

Additionally, air quantity requirements for equipment and infrastructure were calculated. The overall mine air volume required for operation is estimated to be 503,000 cfm including a 15% leakage factor.

It is expected that during initial development (prior to development of a return ventilation raise), fresh air will be delivered to the underground mine via the production shaft. During this period, fresh air will need to be delivered through ductwork installed within the production shaft with exhaust air traveling back up the production shaft to surface.

As mining progresses, the main production shaft and the ES ventilation raise will serve as intake airways. The EN and E2 ventilation raises will serve as exhaust airways.

Main fans for the mine will be surface vane axial fans installed at the three ventilation raises. E2 requires a more powerful fan due to higher resistance from the E2 connector drift and the length intake air must travel. All of the fans have variable speed drives to enable ramp up and down of air volume without having to change the fan configuration. The air volume for the ventilation raises/fans vary greatly depending on the number of mining units in each district.

Secondary ventilation will be provided by booster and auxiliary fans. Vent ducting will be used for ventilating single openings.

16.1.22 Operations Management

An experienced mining contractor is being utilized at the Underground Project to carry out pre-production operations (shaft sinking, off-shaft development, lateral and vertical development), with the contractor to provide management and supervision at and below the Mine Superintendent level. The mining contractor will also perform production activities until steady-state production has been achieved (within the third quarter of Year 2). Mobile mining equipment (with the exception of some equipment having long lead times or that is only required for a short duration) will be supplied by the Owner through lease financing and mobile equipment maintenance will be provided by the OEM through a maintenance and repair contract

(MARC) style agreement. Long lead time equipment or short-term equipment will be supplied and maintained by the mining contractor. The mining contractor will source the majority of its workforce from the Yerington area.

Following steady-state production, mining operations will transition over to be managed directly by Nevada Copper. The workforce employed by the mining contractor will be transferred over to Nevada Copper as part of this transition. Nevada Copper will manage mining operations for the remainder of the mine life.

Technical services will be supplied to the Underground Project throughout the mine life by a mine engineering consultant.

This structure is expected to offer the following benefits:

- Experienced mine management, supervision and other workforce from the onset of mine development
- Training of the local workforce on an ongoing and progressive basis to transition to an operation fully staffed by a workforce from the Yerington area
- Established management and operations systems, including Health and Safety, Development and Production Planning, and Performance Management
- Best practices operations and resulting performance from development and production crews
- Maximized equipment availability and utilization to minimize mobile fleet requirements

16.2 Open Pit

Golder was retained by Nevada Copper to develop a PFS for the potential development of a 37,000 stpd expanding in a second phase to a 70,000 stpd open pit operation. The following sub-items summarize the assessment of the stand-alone Open Pit Project.

16.2.1 Mining Block Model

The Open Pit Mineral Resource model, described in Item 14.0 of this Report, was modified to allow sensitivity scenarios to be analyzed in Whittle. Golder created a four-digit numeric Whittle block code designed to group blocks with the following, similar, properties:

- Lithology (litho): overburden/ Wassuk (10) = 1, tertiary volcanics (20) = 2, limestone and other rocks (30-90) = 4, talc (100) = 5)
- Oxide flag (oxstate: oxide = 1, transition zone = 2, sulfide = 3)
- Deposit flag (depo: north = 1, south = 2)
- Classification (class: Measured = 1, Indicated = 2, Inferred = 3, unclassified = 0)

As an example, a Whittle block code is 4321, which represents a block that has no overburden, or tertiary volcanic material, resides within the sulfide zone, and resides within the south deposit, and is classified as a Measured Resource.

Only blocks with lithology code of 30 and above, oxide flag = 3, and classification 1 and 2 were considered potential mill feed for the Open Pit PFS. All other blocks were considered to be waste.

The mining model was exported and titled, “190204.dat.”

The topographic file used for the Open Pit PFS is titled “topo_westmodels.00t,” and was provided by Nevada Copper in February 2018. The topographic file was used to visually inspect the block model extents, as well as serving as a reference for pit and WRSF designs.

16.2.2 Geotechnical

The pit slope parameters used in the preparation of the open pit mine are based on the Golder technical memorandum titled *Revised Feasibility Level Pit Slope Recommendation, Pumpkin Hollow, Lyon County, Nevada* (Golder 2012). These parameters are summarized in Table 16-25.

Table 16-25: Inter-ramp Designs for North & South Pits Assuming 50 ft High Production Benches

Geotechnical Unit	Dip Azimuth of Pit Sector (degrees)	Operating Practices	Bench Configuration & Height (feet)	Minimum Catch Bench Width (feet)	Achieved Bench Face Angle (degrees)	Design Inter-Ramp Slope Angle (degrees)
Indurated Alluvium (fully drained)	All	No blasting within 50 feet of final slope; 28-foot catch bench of top of bedrock	Single Bench 1 x 50 feet 50 feet between catch benches	25	65	46
Oxidized Tertiary Volcanic Rocks (top 120 feet)						
All Sectors Tertiary Volcanic Rock - top 120 feet	All	Trim Blasting	Single Bench 1 x 50 feet 50 feet between benches	25	65	46
Poor Quality Bedrock (RQD < 40)						
All Sectors All Rock Types	All	Trim Blasting	Single Bench 1 x 50 feet 50 feet between benches	25	65	46
Fair to Good Quality Bedrock (RQD > 40)						
Mesozoic Rock	010 to 155	Trim Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	63	49
Mesozoic Rock	155 to 230	Pre-Split Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	70	55
Mesozoic Rock	230 to 270	Pre-Split Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	67	52
Mesozoic Rock	270 to 010	Pre-Split Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	70	55
Tertiary Volcanic Rocks below 120 feet depth	North Pit: 40° to 080° South Pit: 110° to 160° (Clockwise)	Trim Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	63	49
Tertiary Volcanic Rocks below 120 feet depth	North Pit: 215° to 350° South Pit: 190° to 320° (Clockwise)	Trim Blasting	Double Bench 2 x 50 feet 100 feet between catch benches	35	70	55
Alluvium						
Non-indurated Alluvium (fully drained)	All	No Blasting	Top 50-foot Bench and Bench on top of Bedrock	25	1(H):1(V)	-

Note: Overall slope angles will be reduced from the recommended inter-ramp angle by incorporation of haul roads, or 80 ft wide geotechnical benches at vertical intervals no greater than 600 ft.
Source: Golder 2012.

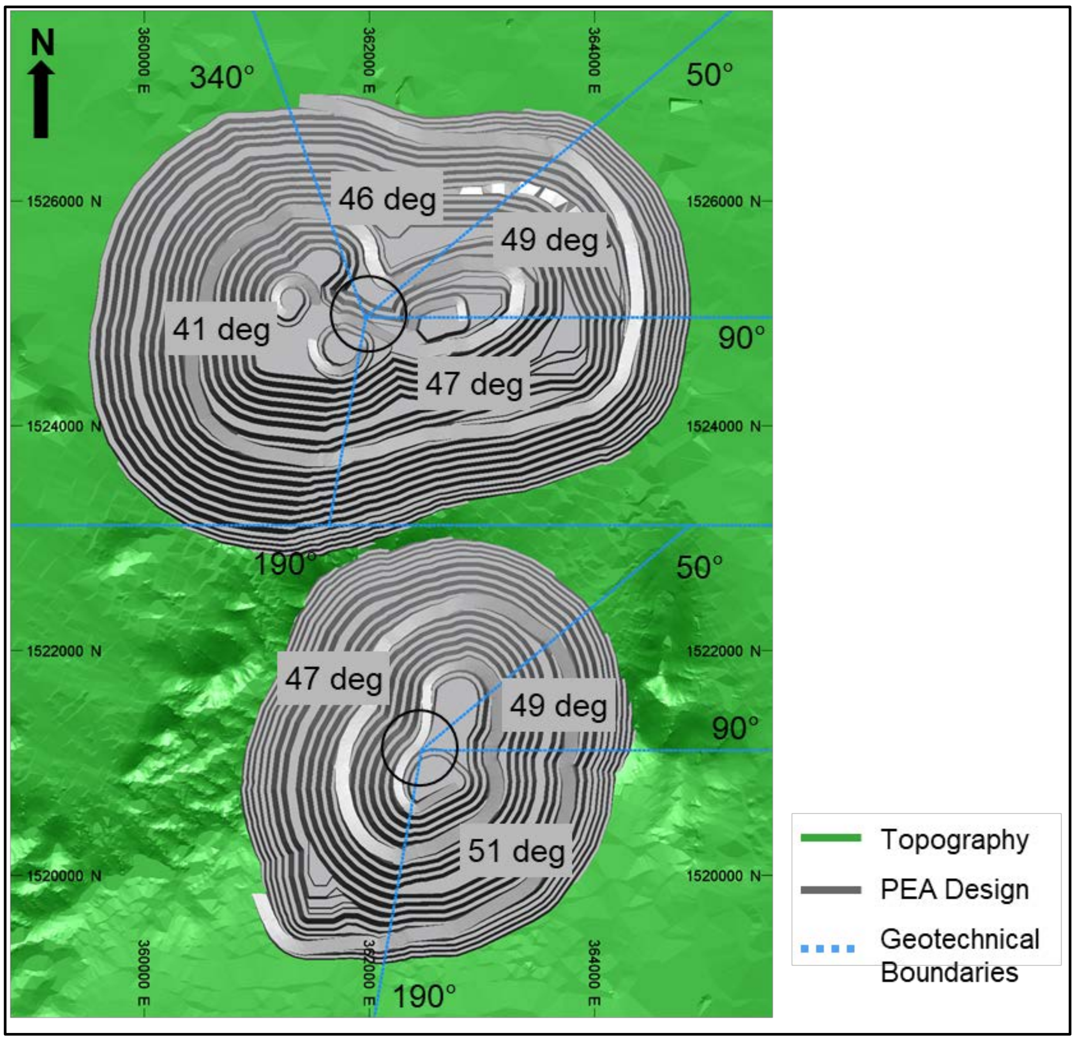
16.2.3 Pit Optimization

Using Whittle, an optimized pit shell analysis was performed. Whittle uses the Lerch Grossman algorithm (along with input parameters and constraints) to assign a value to each block within a block model, and to produce the most cost effective and profitable pit shells for assumed commodity prices. The block model described in Item 16.2.1 was imported into Whittle.

16.2.3.1 <u>Geotechnical Slope Parameters</u>

Slope parameters were based on the simplified geotechnical regions from Item 16.2.2. The updated slope angles are shown in Figure 16-7. Figure 16-7 shows the general overall slope angles used in Whittle, which were selected based on the 2012 geotechnical recommendations and reviewing the PEA design. Overall slope angles were reduced to account for proposed haul roads. Golder recommends a review of the wall angles, as pit economics are sensitive to the pit slopes and even a few degrees change in a sector can have a large impact on the overall strip ratio (SR).

Figure 16-7: Overall Simplified Slope Angles for Whittle Optimization (Golder, 2019)



16.2.3.2 Economic Parameters

Economic parameters used in the pit optimization were based on benchmarked against other properties. These economic parameters are shown in Table 16-26 through Table 16-28. The sales cost include all royalties, taxes, penalties, transport costs and so forth.

Table 16-26: Mining Parameters

Parameter	Unit	Value
Waste Mining Cost1	$/st	1.45
Ore Mining Cost1	$/st	1.55
Incremental Mining Cost	$/st/50 ft bench	0.03
Reference Elevation	ft	4600
Mining Recovery2	%	100
Mining Dilution2	%	5

Notes:

1. The average reference mining cost was based on a preliminary cost estimate. The additional cost associated to mining ore includes items such as grade control, contact definition, and material handling.
2. The block model described in Item 16.2.1 included no dilution or mining recovery. The dilution and recovery used in the scheduling is described in Item 16.2.8.

The waste mining cost of $1.45/st at surface was based industry benchmarking. For the initial years (assuming a production requirement of around 100 Mst with mining costs increasing at depth), the overall cost range is still within the lower benchmarking data.

Processing costs and metal recoveries (Table 16-27) were provided by Sedgman.

Table 16-27: Processing Parameters

Parameter	Unit	Value
Processing Cost	$/st	5.37
Cu Processing Recovery North Pit	%	90
Cu Processing Recovery South Pit	%	88
Au Processing Recovery	%	67
Ag Processing Recovery	%	56

Note: Includes processing, dry stack tailings, environmental expenses, and general & administrative.

Table 16-28: Selling Parameters

Parameter	Selling Price	Selling Cost
Copper	2.75 $/lb	0.55 $/lb
Gold	1343 $/toz	0 $/toz
Silver	19.86 $/toz	0 $/toz

Note: Copper selling costs includes transport and royalties.

The selling prices for copper, gold and silver, as shown in Table 16-28, were provided by Nevada Copper.

These did not include iron, moly and talc and there are opportunities to increase Resources and project economics.

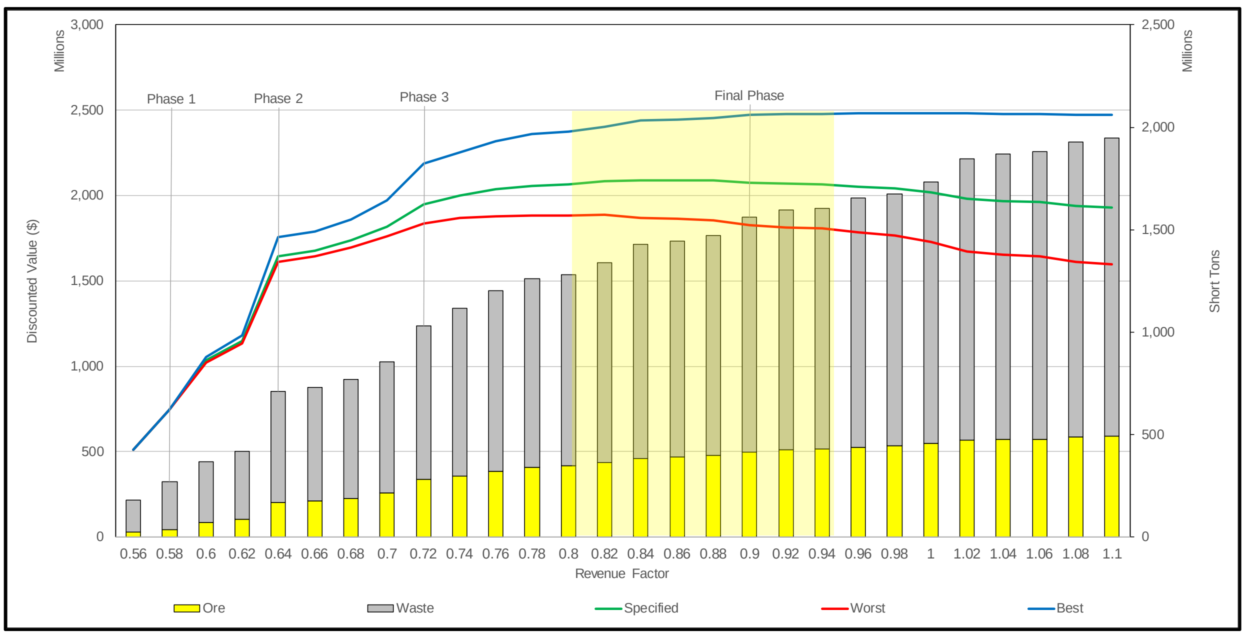
16.2.4 Constraints

The Lerch Grossman analysis was constrained to the North and South deposits. No other constraints, such as mine license boundaries, were applied. There are no property constraints which could affect the economics of the Project.

16.2.5 Whittle Results

Golder ran an economic pit sensitivity analysis in Whittle using a base copper selling prices of $2.75/lb Cu by assuming revenue factors ranging from 0.58 to 1.1 times the selling price of copper to generate nested pit shells. Based on the assumed mill feed, the resultant nested pit shells were used to design potential pushbacks. Figure 16-8 provides a summary of the total waste and mill feed tonnages and resultant preliminary discounted pit value for each selling price of copper. The blue line in the graph (best case) represents a potential value in the event that each nested pit shell is mined to completion before the next shell is mined. The red line (worst case) represents a potential value in the event that each pit shell is mined bench by bench. The green line (specified case) represents the pit shells selected as pushbacks, which will be mined prior to the next pushback. The translucent yellow rectangle shows a range of revenue factors that possess similar cashflows. A discount of 5% was used for the pit optimization.

Figure 16-8: Whittle Results for $2.75/lb Cu (Golder, 2019)



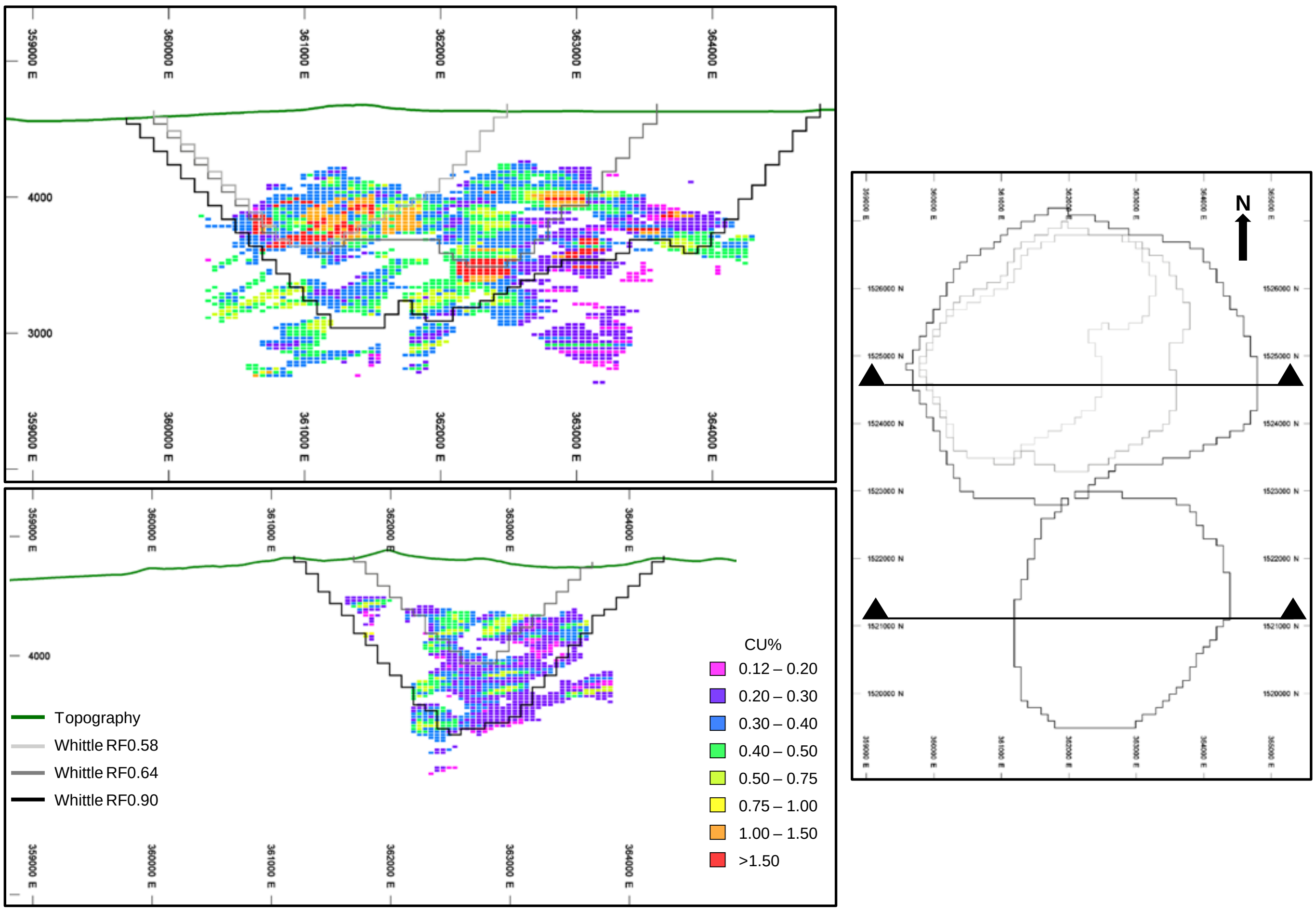
The pit shell analysis shows that the highest cashflows possess a range of about 340 Mst to 430 Mst of mill feed with a SR of approximately 2.7. Revenue Factor (RF) 0.90 was the pit shell selected for the ultimate pit design.

16.2.6 Pit Shell Selection

As discussed in Item 16.2.5, the ultimate pit shell selected for this study had a RF of 0.90 for a copper selling price of $2.75/lb. The RF 0.58 shell was selected as a starter pit, because it possessed approximately one year of potential mill feed. The RF 0.64 pit shell was selected for an additional phase in order to assist in scheduling. Details on the phase designs are described in Item 16.2.7.

The pit shells discussed in Item 16.2.5 are shown in Figure 16-9.

Figure 16-9: Whittle Pit Shells Plan View & Cross Section (Golder, 2019)



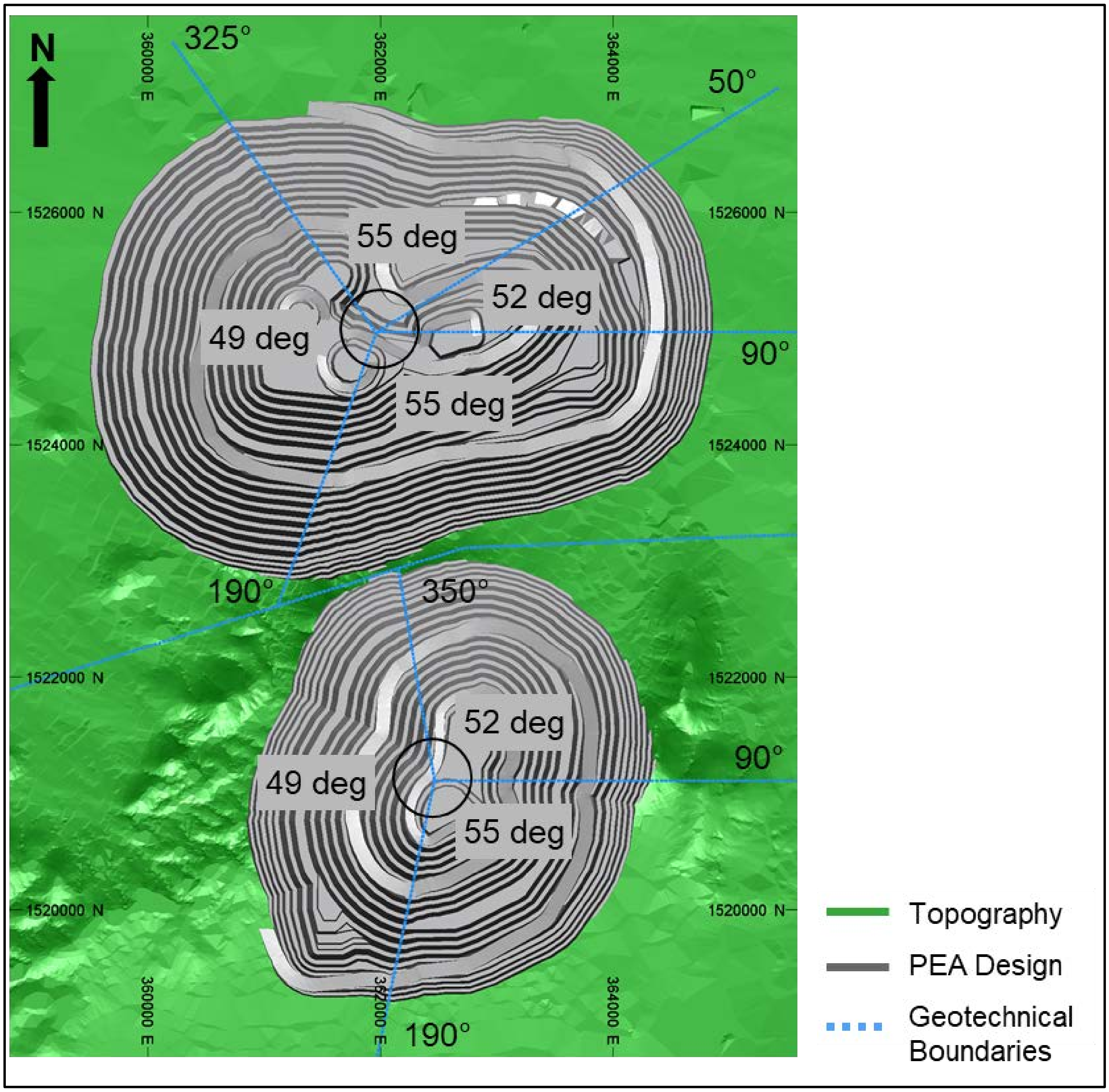
16.2.7 Pit Design

The pit phases and ultimate pit were designed based upon the results of the Whittle pit optimization described in Item 16.2.3 using the geotechnical parameters described in Item 16.2.7.1 and the haul road design described in Item 16.2.7.2.

16.2.7.1 Geotechnical Parameters

The simplified geotechnical parameters were used for the Open Pit PFS pit design. Within 120 ft of the overburden/tertiary volcanic contact, or within the overburden lithology, the pit was designed, using 100 ft benches with an inter-ramp angle of 46° and 25 ft wide catch benches. In all other areas, 100 ft double benches (50 ft by 2 ft benches) were used with 35 ft catch benches. Inter-ramp angles are shown in Figure 16-10. The parameters used for the Open Pit PFS design were simplified from Golder's 2012 recommendations. The overall slope angles of both the design and recommended parameters match and therefore have no economic impact.

Figure 16-10: Geotechnical Zones for Pit Design (Golder, 2019)



16.2.7.2 Haul Road Design Parameters

The majority of the haul road was designed to be double-lane, although single-lane road was used to complete each pit. As seen in Figure 16-11, the double-lane sections of the haul ramp were designed to accommodate three and a half times the width of a 320 st class haul truck with additional clearance for a berm and ditch. Single-lane sections were designed to accommodate two times the width of the haul trucks (Figure 16-12). A 6 ft berm (approx.) is required on the outside of the ramps for safe operation, which is equivalent to the radius of a 320 st class truck tire. A 3 ft wide ditch was also included on the inside of the haul ramp to allow for drainage of surface water. The total width of the double-lane ramp was calculated to be 119 ft, and the total width of the single-lane ramp was calculated to be 76 ft.

Figure 16-11: Double-Lane Design for 320 st Class Haul Truck (Golder, 2019)

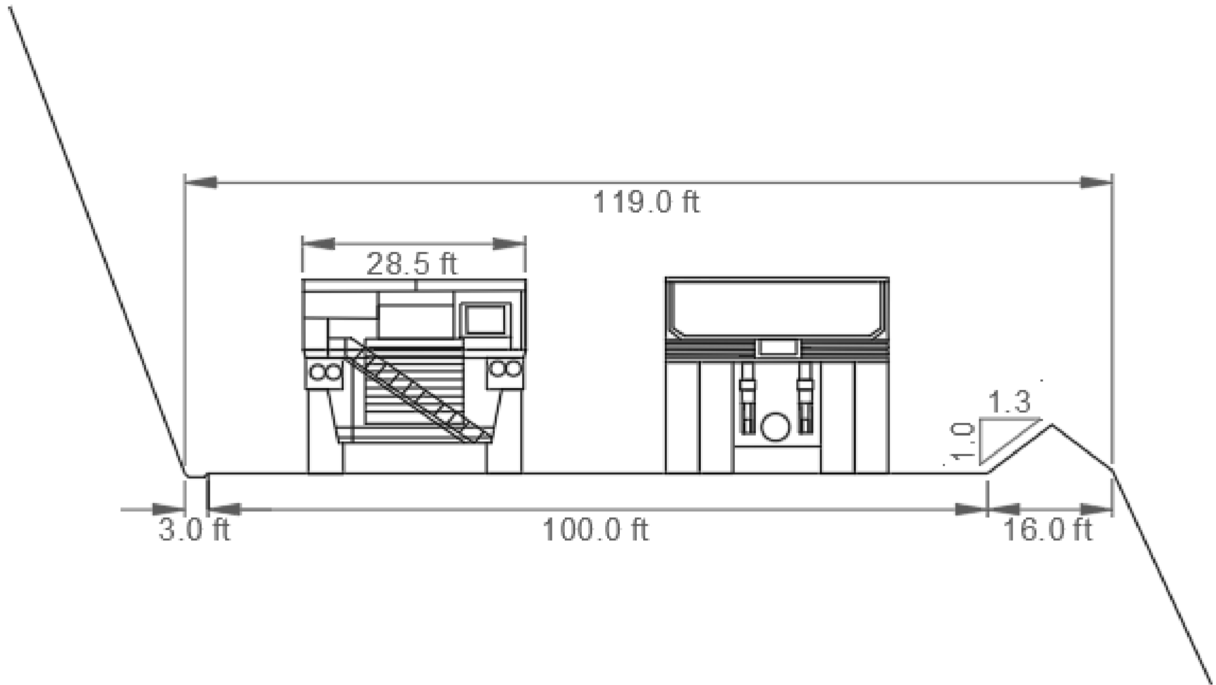
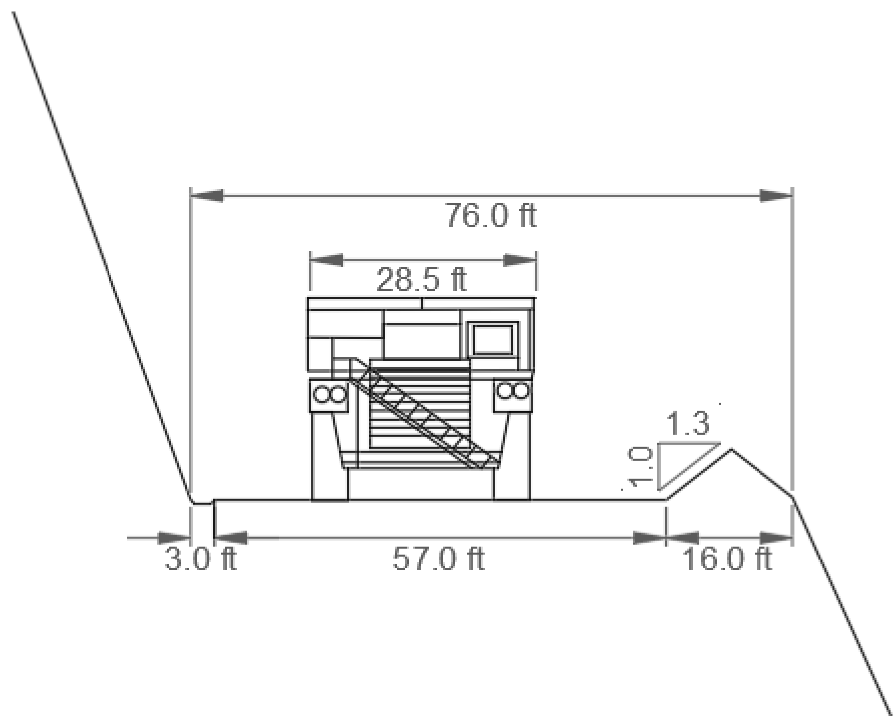


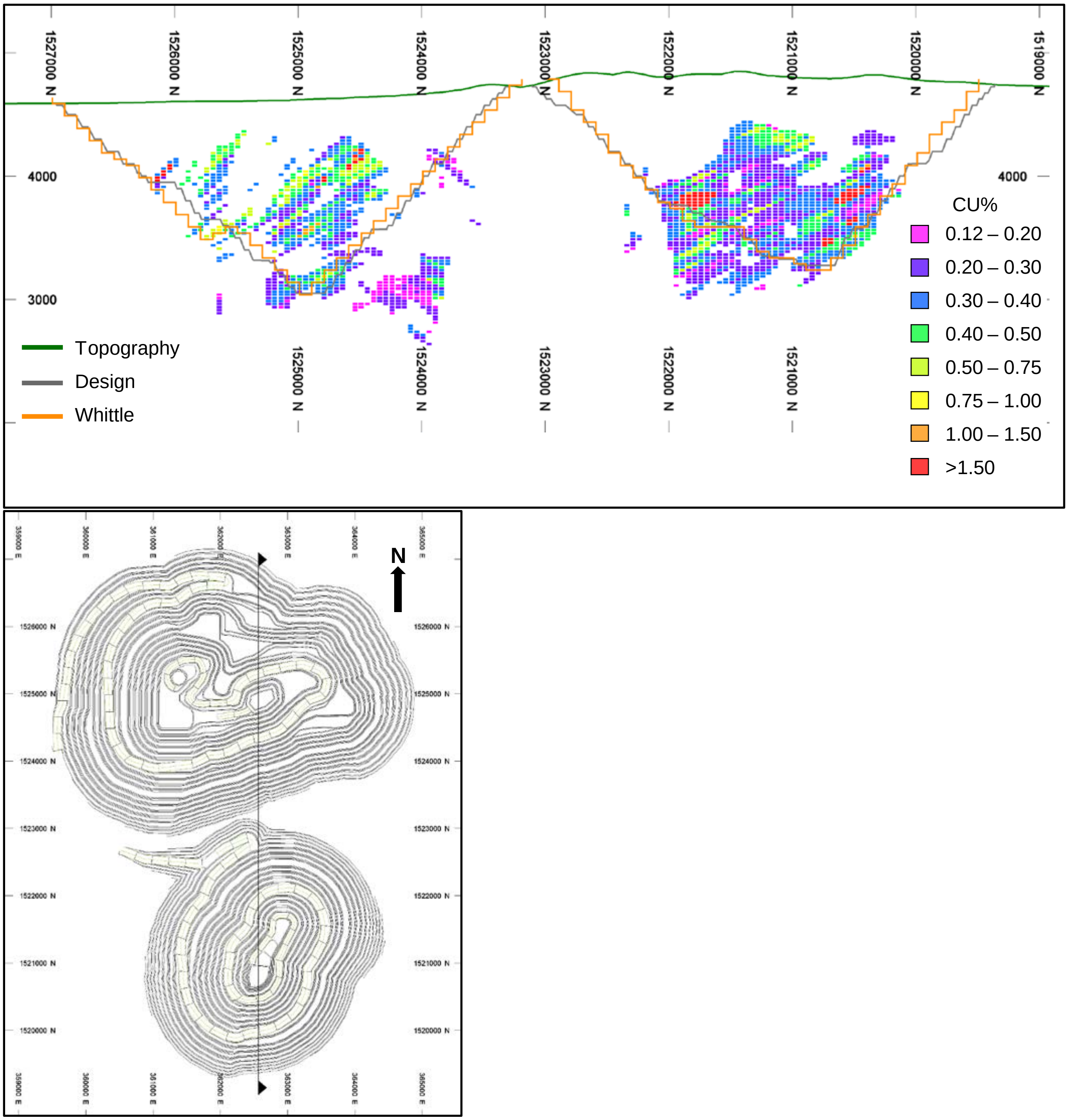
Figure 16-12: Single-Lane Design for 320 st Class Haul Truck (Golder, 2019)



16.2.7.3 Ultimate Pit Design

The ultimate pit was designed using the Whittle RF 0.90 shell generated from a copper selling price of $2.75/lb. Sections comparing the pit design and the RF 0.90 Whittle shell are shown in Figure 16-13.

Figure 16-13: Ultimate Pit Sections (Golder, 2019)



Note: The cross section on the left depicts how closely the designs follow the selected Whittle pit shell (RF 0.90).

16.2.8 Mine Dilution & Recovery

To estimate the amount of potential mine dilution that may be mined as ore within the ultimate pit design, Golder compared the geological sub-blocked model with the undiluted, 50 ft by 50 ft by 25 ft, regularized block model. The analysis showed a 5% (approx.) difference in metal between the two models. Therefore, an average 5% dilution modifying factor was applied to the PFS ore within the ultimate pit shell. The selective mining unit was 50 ft by 50 ft by 25 ft. The bucket size of the shovels selected for this Study was 64 cubic yards (yd^3); however, when the swell and fill factors are applied, the effective bucket size is reduced to 40 yd^3 (approx.). It is Golder's opinion that the block size and modeling methods adequately address mining recovery; and therefore, applied a 98% mining recovery to account for the small operational losses, such as incorrect dig limits, blast movements, and misallocation of ore. The application of a high mining recovery adjustment assumes well-controlled drilling, blasting, and surveying.

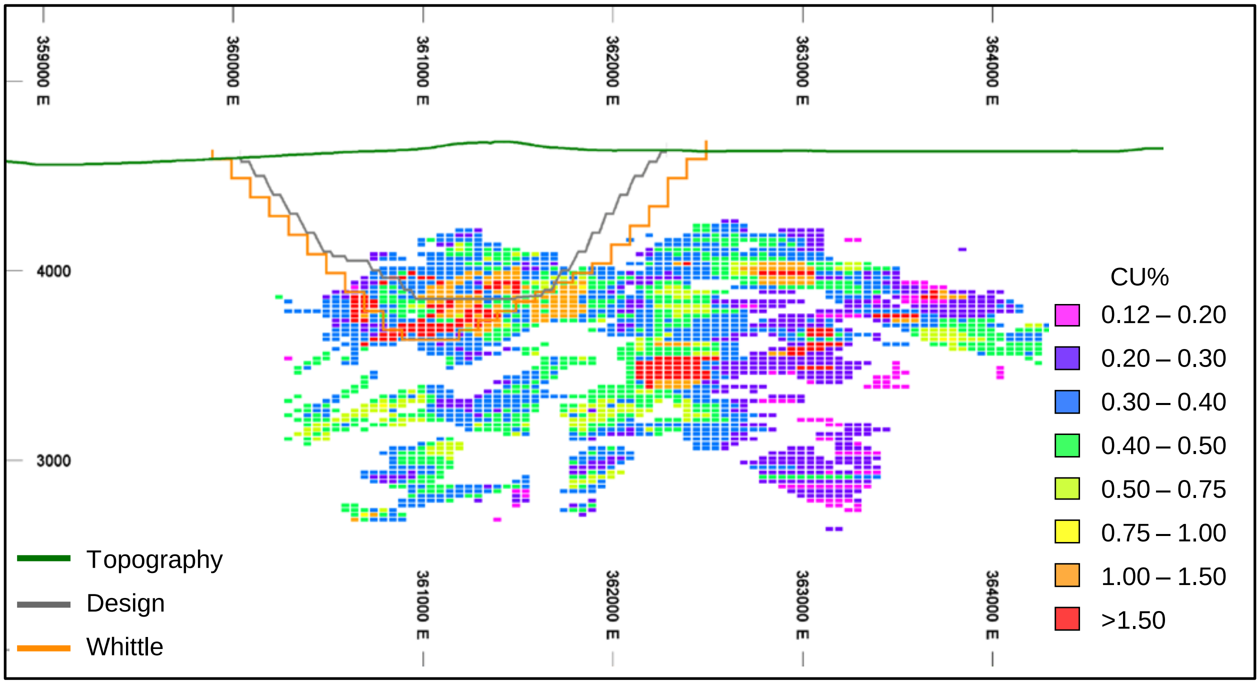
16.2.9 Pit Phases

The North Pit was designed using four phases and the South Pit was designed in two phases. Three of the North Pit phases followed the Whittle pit shells described in Item 16.2.5. To improve scheduling the first few years of production, an additional starter pit was designed within the RF0.58.

16.2.9.1 North Pit Phase 1

Figure 16-14 depicts the North Pit Phase 1 design and the Whittle RF0.58.

Figure 16-14: North Pit Phase 1 (plan view & 1524715 N section) (Golder, 2019)



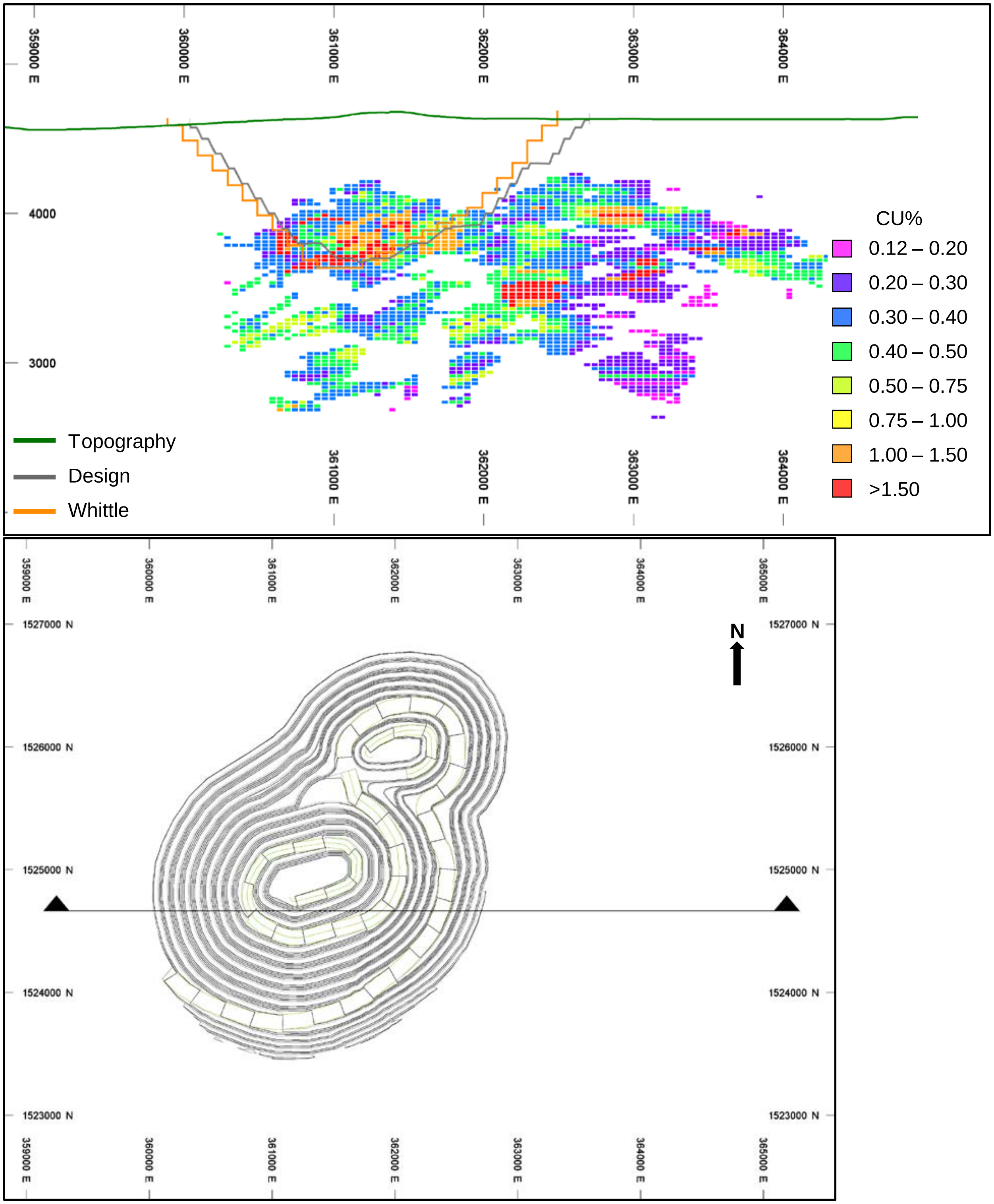
The Phase 1 pit design targeted 12 months (approx.) of ore within the Whittle RF 0.58 for a base selling price of $2.75/lb Cu, which represents an economic pit for a copper selling price of $1.60/lb Cu (approx.). Most of the ramp in Phase 1 will be suitable for two-way traffic, although the bottom four benches were designed to accommodate single-lane traffic to maximize recovery of mill feed material. The bottom of the pit narrows to 200 ft (approx.), which is suitable for hydraulic shovels, but may add operational complexity for electric rope shovels.

16.2.9.2 North Pit Phase 2

Figure 16-15 depicts the North Pit Phase 2. For the Open Pit PFS, Golder targeted a 200 ft mining width constraint between pushbacks, however, some areas employed a minimum width constraint of 130 ft.

The Phase 2 pit was designed based on the Whittle RF 0.58 pit shell for a base selling price of $2.75/lb Cu, which represents an economic pit at $1.60/lb Cu (approx.). While most of the ramp in Phase 2 will be suitable for two-way traffic, the bottom four benches were designed to accommodate single-lane traffic. The bottom of the Phase 2 pit narrows to 190 ft (approx.).

Figure 16-15: North Pit Phase 2 (plan view & 1524715 N section) (Golder, 2019)

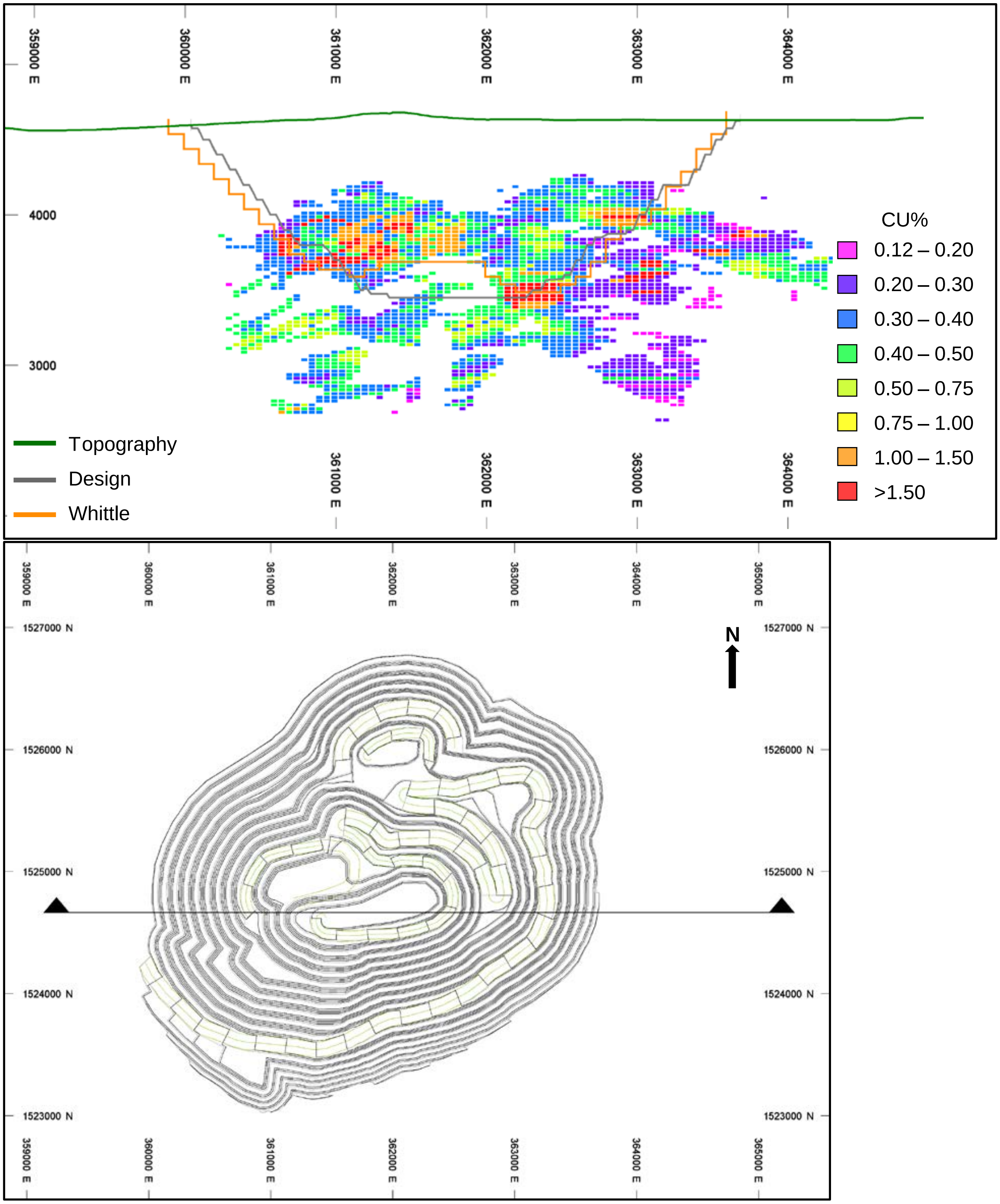


16.2.9.3 North Pit Phase 3

Figure 16-16 depicts the North Pit Phase 3.

Phase 3 of the North Pit was designed based on the Whittle shells with RF 0.64 for a base selling price of $2.75/lb Cu, which is equivalent to an economic pit at $1.76/lb Cu (approx.). Most of the ramp in Phase 3 will be suitable for two-way traffic, with the bottom four benches narrowing to single lane to maximize recovery of mill feed material. The bottom of the pit narrows to a minimum of 180 ft; while this is acceptable for hydraulic shovels, it may be operationally challenging for electric shovels.

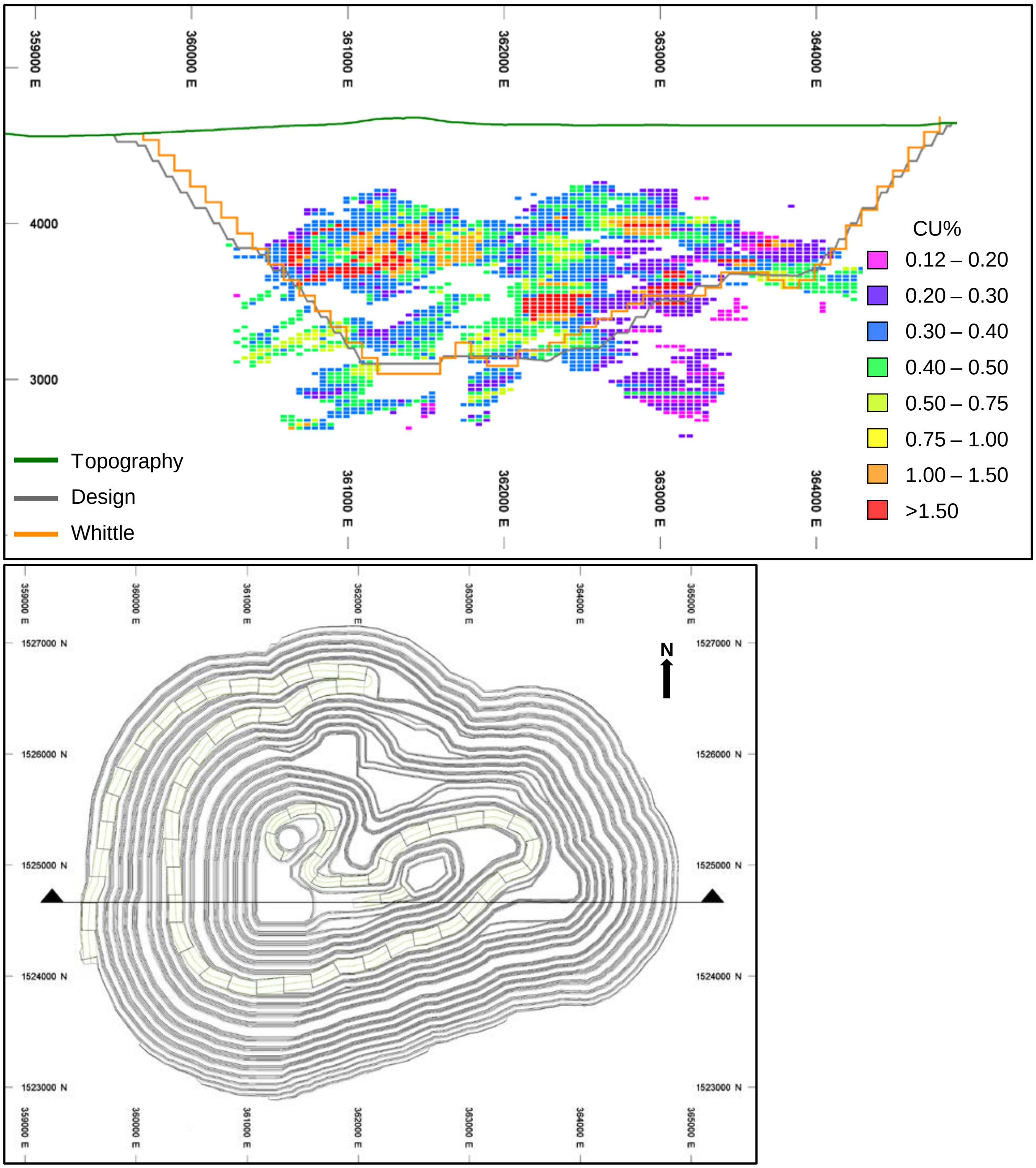
Figure 16-16: North Pit Phase 3 (plan view & 1524715 N section) (Golder, 2019)



16.2.9.4 North Pit Phase 4

Phase 4 of the North Pit was designed based on the Whittle RF 0.90 pit shell for a base selling price of \$2.75/lb Cu, which represents an economic pit at \$2.48/lb Cu (approx.). While most of the ramp in Phase 4 will be suitable for two-way traffic, the bottom four benches were designed to accommodate single-lane traffic. The bottom of the Phase 4 pit narrows to 200 ft (approx.). The Phase 4 pit is shown in Figure 16-17.

Figure 16-17: North Pit Phase 4 (plan view & 1524715 N section) (Golder, 2019)

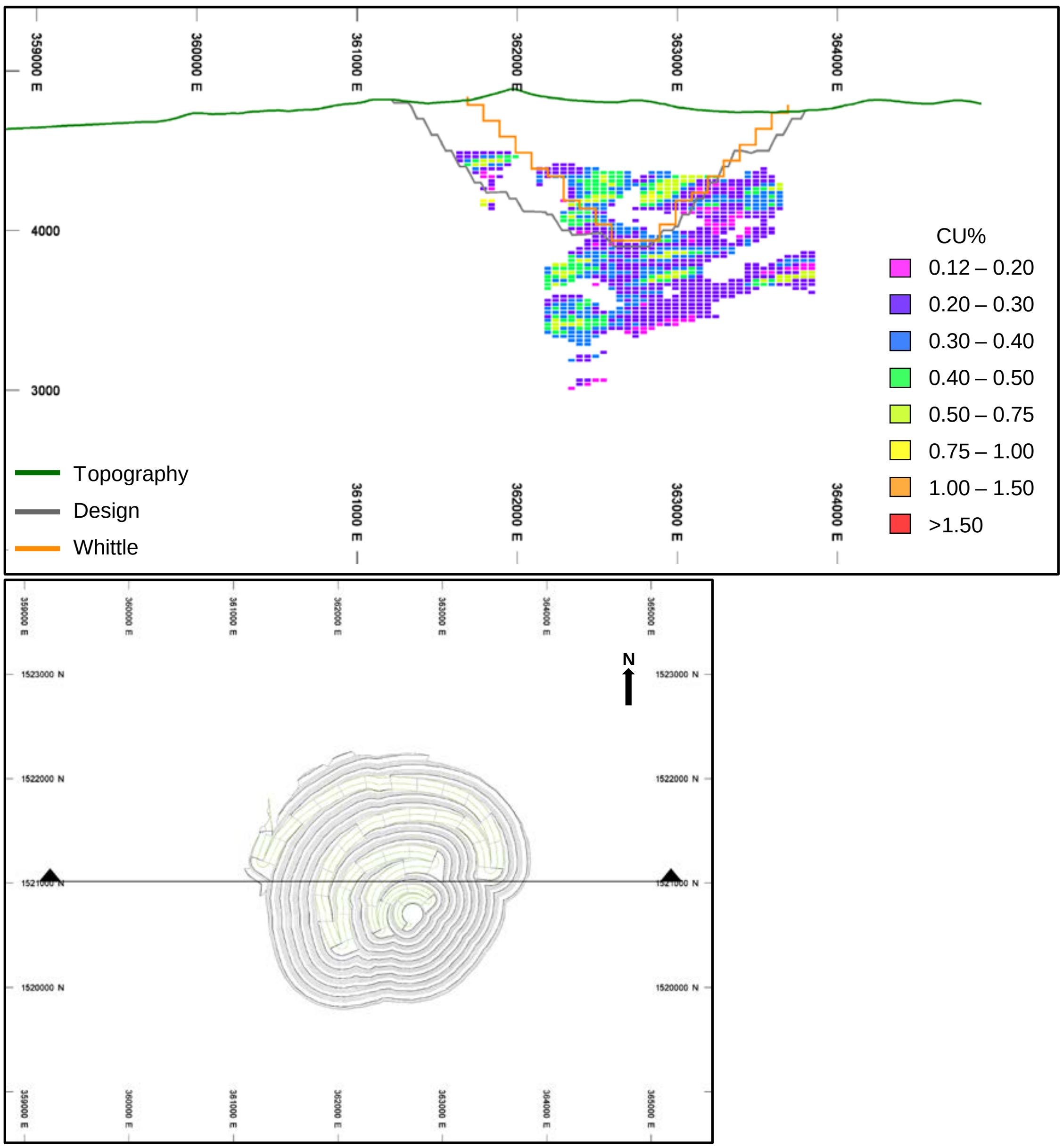


16.2.9.5 South Pit – Phase 1

The South Pit for Phase 1 is shown in Figure 16-18.

The South Pit Phase 1 was designed based on the Whittle RF 0.64 for a base selling price of $2.75/lb Cu, which represents an economic pit at $1.76/lb Cu (approx.). Most of the ramp in Phase 1 will be suitable for two-way traffic, though the bottom four benches were designed to accommodate single-lane traffic to maximize recovery of mill feed material. The bottom of the South Pit narrows to 200 ft (approx.), which is suitable for hydraulic shovels, but may present a challenge for electric rope shovels and the required infrastructure to provide power.

Figure 16-18: South Pit Phase 1 (plan view & 1521015 N section) (Golder, 2019)

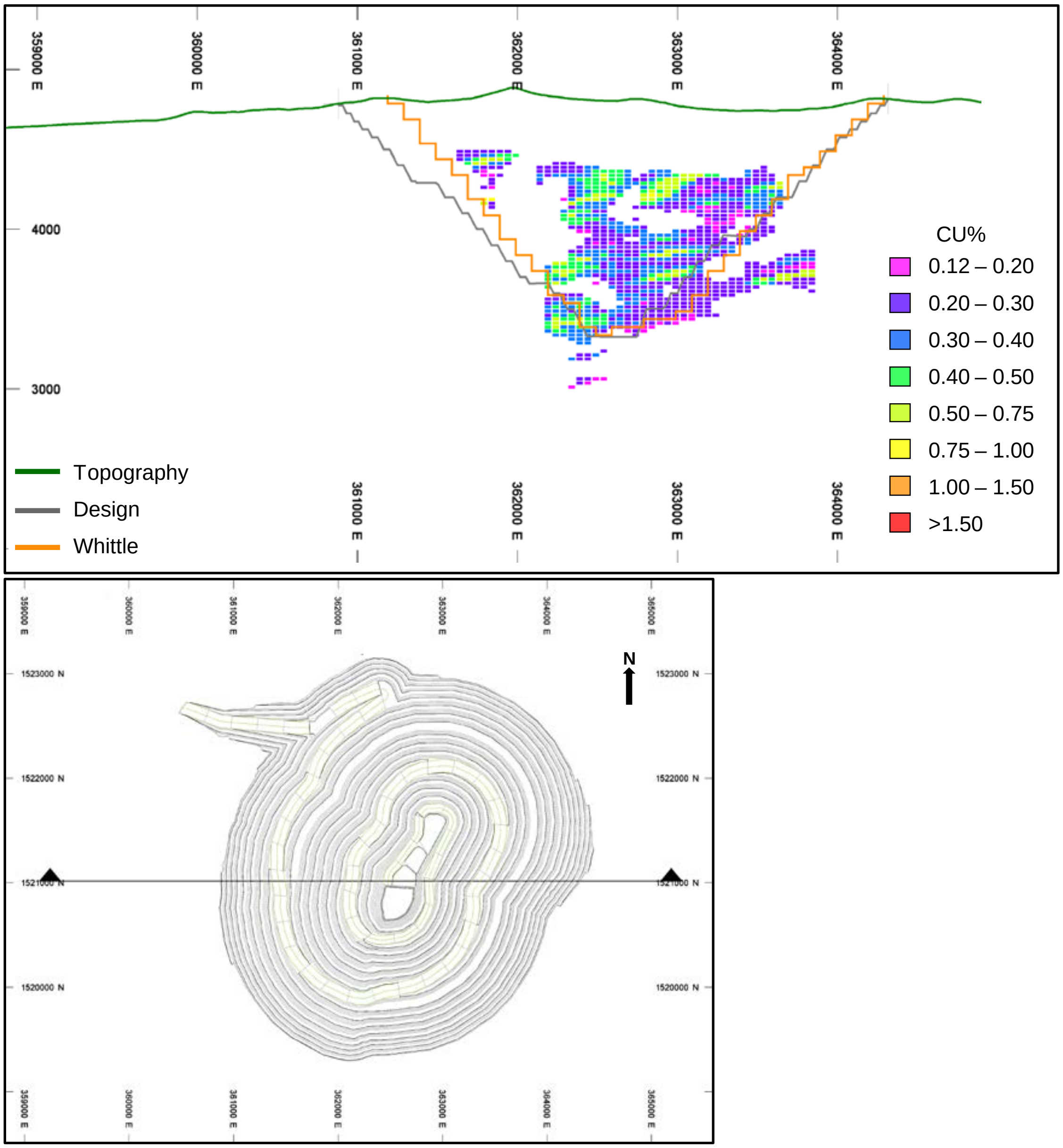


16.2.9.6 South Pit – Phase 2

The South Pit Phase 2 is also constructed as a single design, as shown in Figure 16-18.

The South Pit Phase 2 was designed based on the Whittle RF 0.90 for a base selling price of $2.75/lb Cu, which represents an economic pit at $2.48/lb Cu (approx.).

Figure 16-19: South Pit Phase 2 (plan view & 1521015 N section) (Golder, 2019)



16.2.10 Cutoff Grade Calculation & Mineral Reserves by Pit

Table 16-29 and Table 16-30 summarizes the Mineral Reserves by phase and by pit. Measured and Indicated Resource blocks within the pit designs above an economic cutoff are reported with the modifying factors outlined elsewhere in this Report applied.

Cutoff grade, which is defined as the lowest grade of material that is economic to process, is calculated using the following equation:

$$COG = \frac{\text{Processing Cost * Dilution}}{\text{Processing Recovery * (Selling Price - Selling Cost)}}$$

$$COG = \frac{\$5.37/\text{st} * 1.05}{0.90 * (\$3.00/\text{lb} - \$0.55/\text{lb})} * \frac{1\ \text{st}}{2000\ \text{lb}}$$

The cutoff grade, as shown above, applies to the North Pit and equals 0.129% Cu; however, the South Pit mill feed has an 88% processing recovery, and would therefore have a slightly higher cutoff grade (i.e., 0.132% Cu).

Table 16-29: Mineral Reserve by Phase – North Pit

Phase	Waste (Mst)	Ore (Mst)1	Total (Mst)	SR	Cu (%)	Au (oz/st)	Ag (oz/st)
1	106.5	9.3	115.7	11.5	0.71	0.002	0.079
2	105.0	21.2	126.1	5.0	0.89	0.002	0.084
3	132.5	56.0	188.5	2.4	0.46	0.001	0.055
4	434.0	136.3	570.3	3.2	0.50	0.002	0.057
Total	777.9	222.8	1000.7	3.5	0.54	0.002	0.060

Note:

1. The Mineral Reserve includes those Measured and Indicated Mineral Resources within the design above a copper cutoff of 0.129%. The Mineral Reserve does not include Inferred Mineral Resource material, because, according to the Canadian Institute of Mining, Metallurgy and Petroleum definition, this resource cannot be considered economic. Inferred Resource within the design is considered waste.

Table 16-30: Mineral Reserve by Phase – South Pit

Phase	Waste (Mst)	Ore (Mst)1	Total (Mst)	SR	Cu (%)	Au (oz/st)	Ag (oz/st)
1	134.9	37.5	172.5	3.6	0.41	0.002	0.053
2	262.0	125.4	387.4	2.1	0.35	0.002	0.047
Total	397.0	162.9	559.9	2.4	0.37	0.002	0.048

Note:

1. The Mineral Reserve includes those Measured and Indicated Mineral Resources within the design above a Cu cutoff of 0.132%. The Mineral Reserve does not include Inferred Mineral Resource material, because, according to the Canadian Institute of Mining, Metallurgy and Petroleum definition, this resource cannot be considered economic. Inferred Resource within the design is considered waste.

A comparison of the total contained resource between the designed pit and the Whittle RF 0.90 pit shell is provided in Table 16-31. The higher SR in the design is due to the required minimum mining widths and ramp geometry, which is not considered in Whittle. The difference is considered reasonable for this level of study.

Table 16-31: Whittle & Pit Design Comparison

Item	Waste (Mst)	Ore (Mst)1	Total (Mst)	SR	Ore Cu (%)	Au (oz/st)	Ag (oz/st)
Whittle2	1155.2	406.4	1561.6	2.8	0.47	0.002	0.056
Design	1174.9	385.7	1560.6	3.0	0.47	0.002	0.055
Difference	2%	-5%	0%	7%	0%	0%	-1%

Notes:

1) Reserves above 0.0129% Cu (North Pit) and 0.132% Cu (South Pit) are contained within the Whittle shell and within the design. The Whittle figures do not correspond directly to the numbers reported in Item 16.2.5 due to the higher cutoff grade being used for the schedule and updated modifying factors applied. Figures include 5% dilution and 98% mining recovery.
2) Whittle Shell is a RF 0.90 pit with a base $2.75/lb Cu price, as discussed in Item 16.2.6.

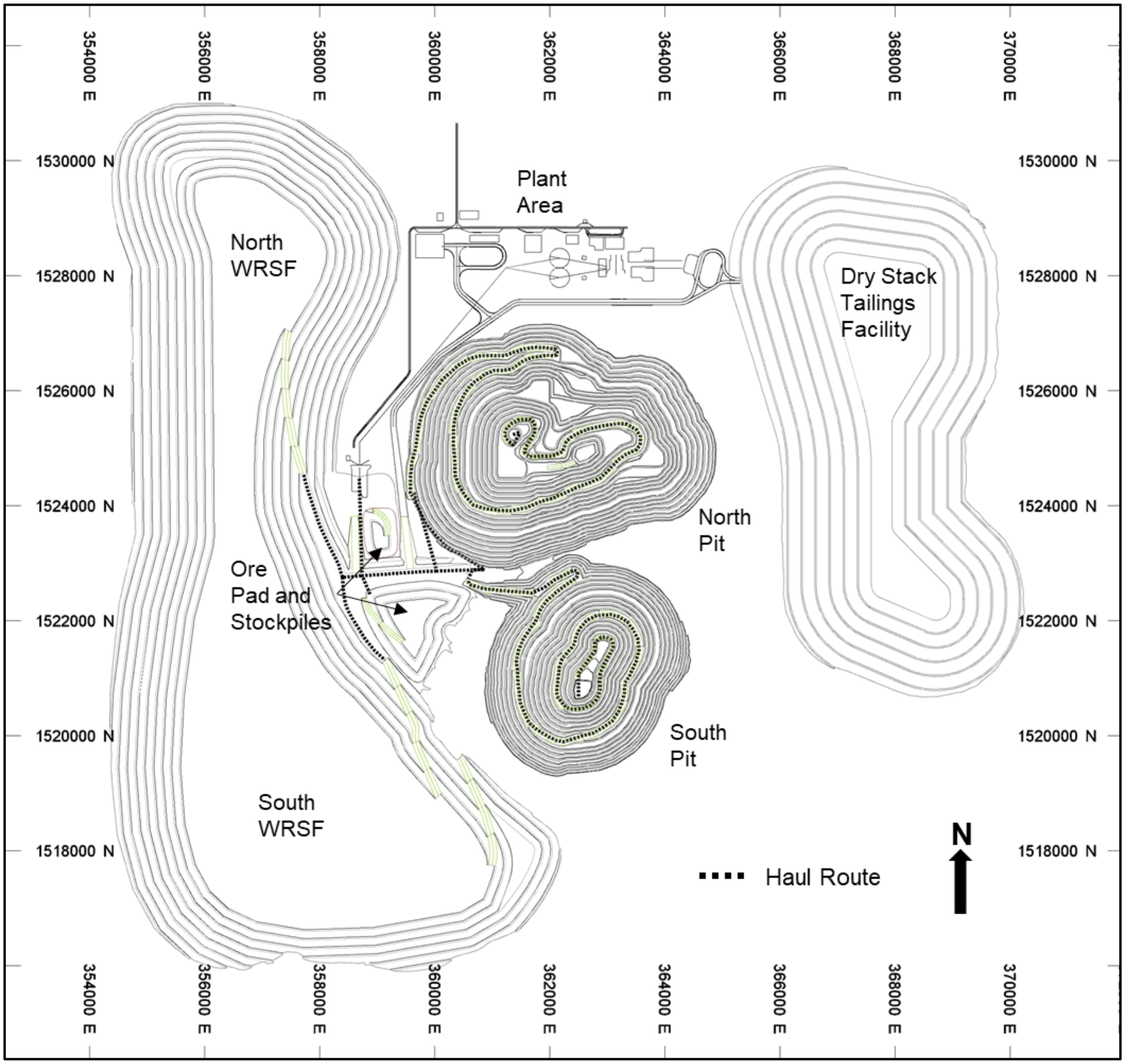
16.2.11 Haulage

Haulage requirements were calculated using MineSight's Haulage module, MSHaulage. MSHaulage calculates point-to-point haul times, employing user-defined assumptions and the software's suite of information, such as manufacturer rimpull curves. A maximum speed of 25 miles per hour (mph) was applied site wide, with a rolling resistance of 3%, on all ramps and within the pits, and a rolling resistance of 5% on the WRSF.

16.2.11.1 Haulage Network

The haulage network was designed to allow each block to be hauled from its location in the pit phase to the primary ore crusher, the ore stockpile, or the WRSF. MineSight Scheduler Optimizer (MSSO), a sub-module of MineSight, was used to calculate the cycle time from each block (in the pit) to each potential dumping location. To calculate reserves on each cut, the phase designs were sliced into 400 ft by 400 ft by 50 ft solids in MineSight and loaded to MineSight Planner (MS Planner), another sub-module of MineSight. The WRSF was sliced into 1,600 ft by 1,600 ft by 50 ft height solids and loaded into MSSO. Figure 16-20 depicts the haulage network for the North and South Pits.

Figure 16-20: Haulage Network for North and South Pits (Golder, 2019)



As shown in Figure 16-20, the haulage network includes a path from each pit to the WRSF, to the ore stockpile, and to the primary crusher. Haulage was optimized by allowing MSSO to maximize the shortest haul routes. In-pit backfilling was utilized in the later years once the North pit has been completed.

The location of the WRSF will be further confirmed with additional condemnation drilling.

16.2.12 Schedule

Golder generated the production schedule using MSSO. MSSO uses IBM's™ CPLEX Optimizer to generate a schedule based on user-defined constraints and objectives. MSSO uses the reserves cut in MS Planner and the calculated cycle times, as discussed in Item 16.2.11.1, to generate highest cashflow schedules. The selectivity in the MSSO schedule is more rigid than a schedule implementing a bench-by-bench methodology. To maximize the mill feed head grade, stockpiling was considered. Stockpiling obtains higher copper grades during the first 5 years of production. Further, stockpiling is required to achieve a mill expansion in Year 8 of the mining schedule. A 10 Mst (approx.) capacity stockpile was designed for this Study. A “First In – First Out” method was used in the reclaiming of the stockpile.

Additional parameters used in MSSO include the following:

- Mill feed cap of 1.05% Cu
- Shovel productivities based on the following:
  - Mechanical availability of 85%, use of availability of 91%, and operator efficiency of 84%, which support the following:
    - Electric shovel (bucket size 64 yd3) with a maximum of 37 Mstpa
    - Hydraulic shovel (bucket size 47 yd3) with a maximum of 27 Mstpa
    - Loader (bucket size 30 yd3) with a maximum of 13 Mstpa
    - Total loading capacity of 105 Mstpa (approx.) for the first few years, and decreasing in the later years along with decreasing availability
- Truck productivity information to include the following:
  - Payload of 320 st
  - Queue, load, spot, and dump time of four minutes (approx.)
- The vertical advance of eight benches per year and per phase was used with the exception of the pre-strip operations
- Mining was scheduled quarterly for the first five years and annually thereafter

MSSO optimizes the schedule based on the above constraints, targeting the highest NPV sequence of mining. Thus, the software targets the North Pit first, and stockpiles ore during years 2, 3, and 4 to feed the mill to a maximum head grade of 1.05% Cu. Sedgman provided this processing constraint.

Mineral resource above 0.129% Cu from the North Pit and above 0.132% Cu from the South Pit, are either selected to move directly to the primary crusher, or be stockpiled. MSSO suggests the best destination for each material in each period to maximize the NPV.

The schedule was designed to minimize waste pre-strip and maximize head grade feed. As seen in Figure 16-21, the mine ramp up is planned to start in the first quarter of Year 2 and be at full capacity by the start of Year 3. This schedule requires approximately 180 Mst of pre-stripping before a constant mill feed can be established.

Figure 16-21: Ore and Waste Tons by Mining Year by Pit (annual) (Golder, 2019)

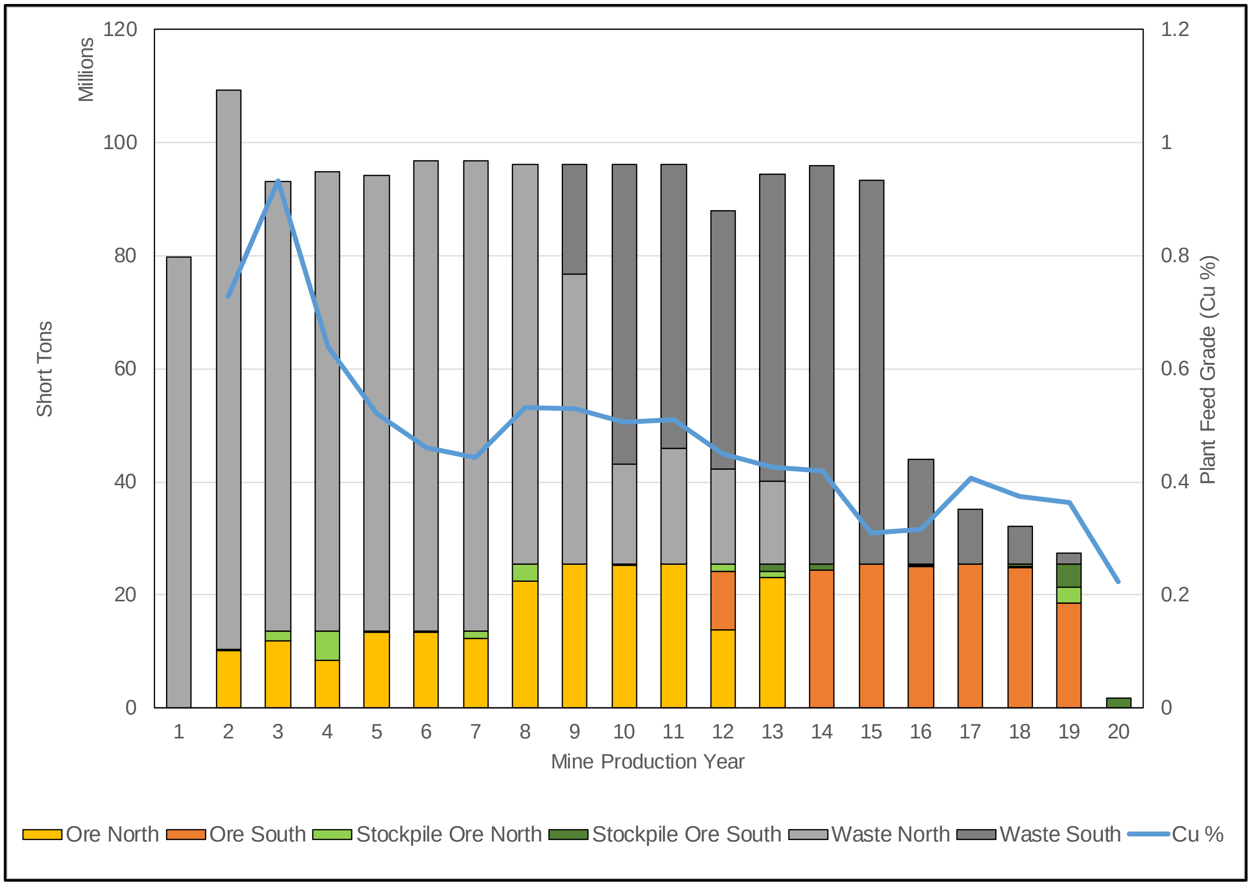
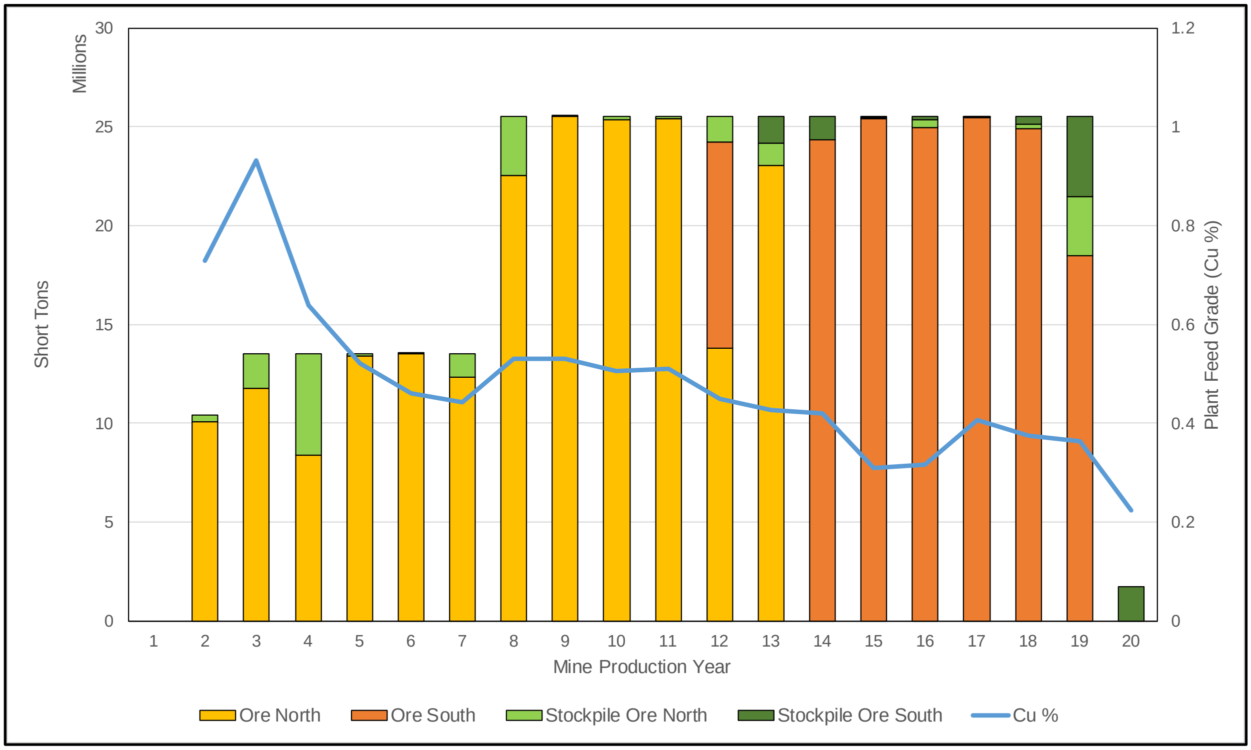


Figure 16-22 and Figure 16-23 show the material produced by pit by year (quarterly periods were combined into an annual schedule).

Figure 16-22: Ore Tons Sent to the Primary Crusher by Year by Pit (Golder, 2019)



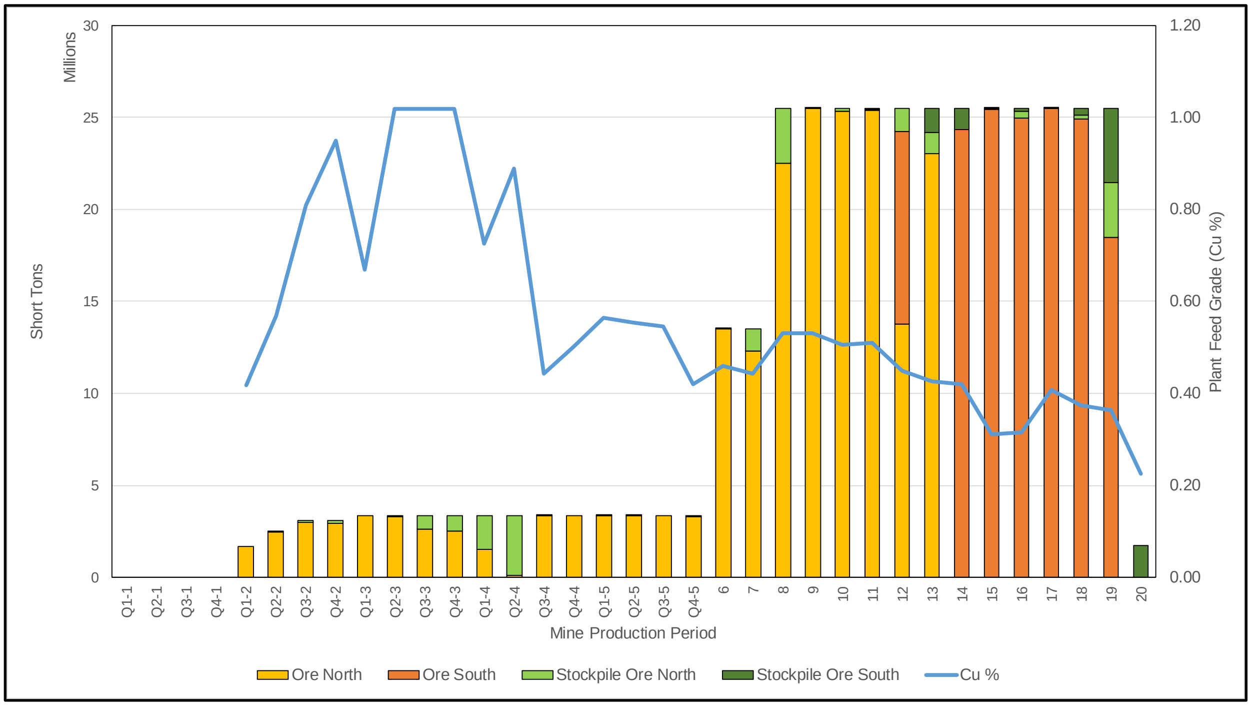
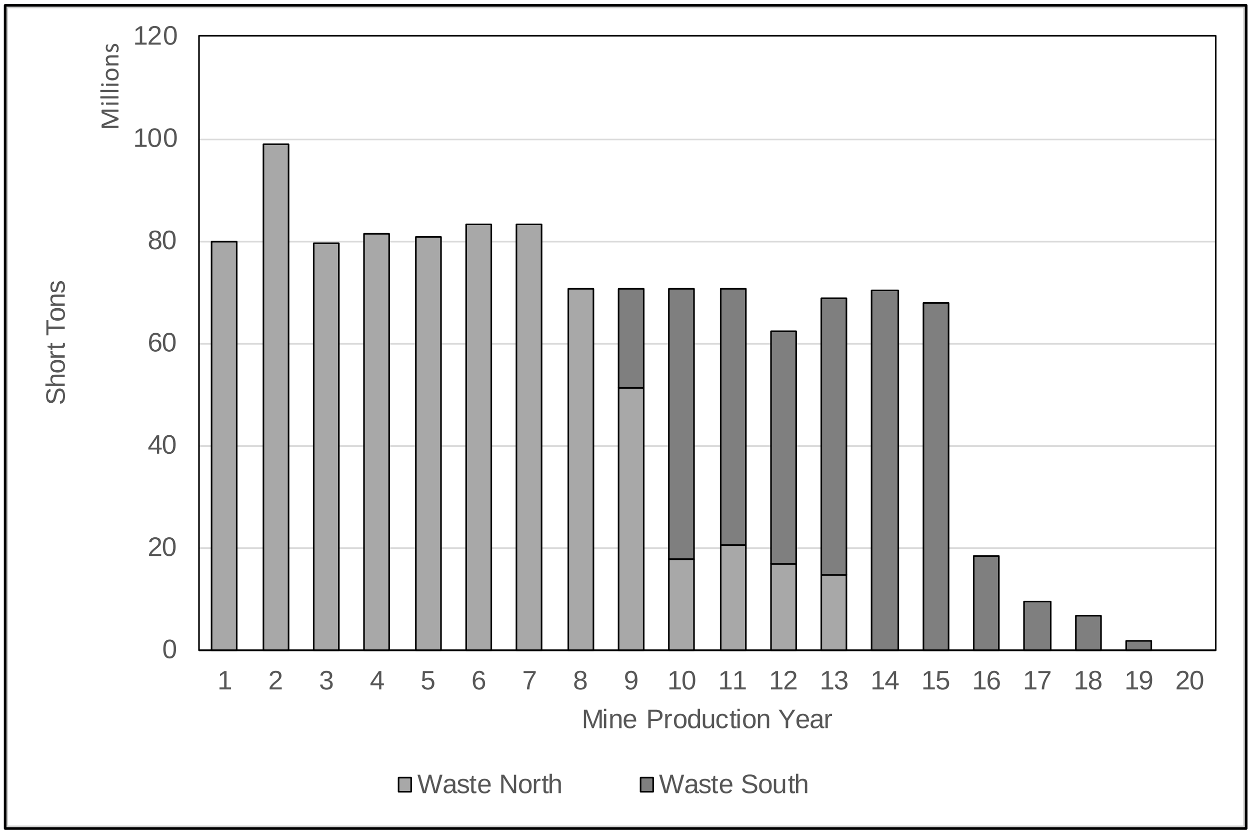
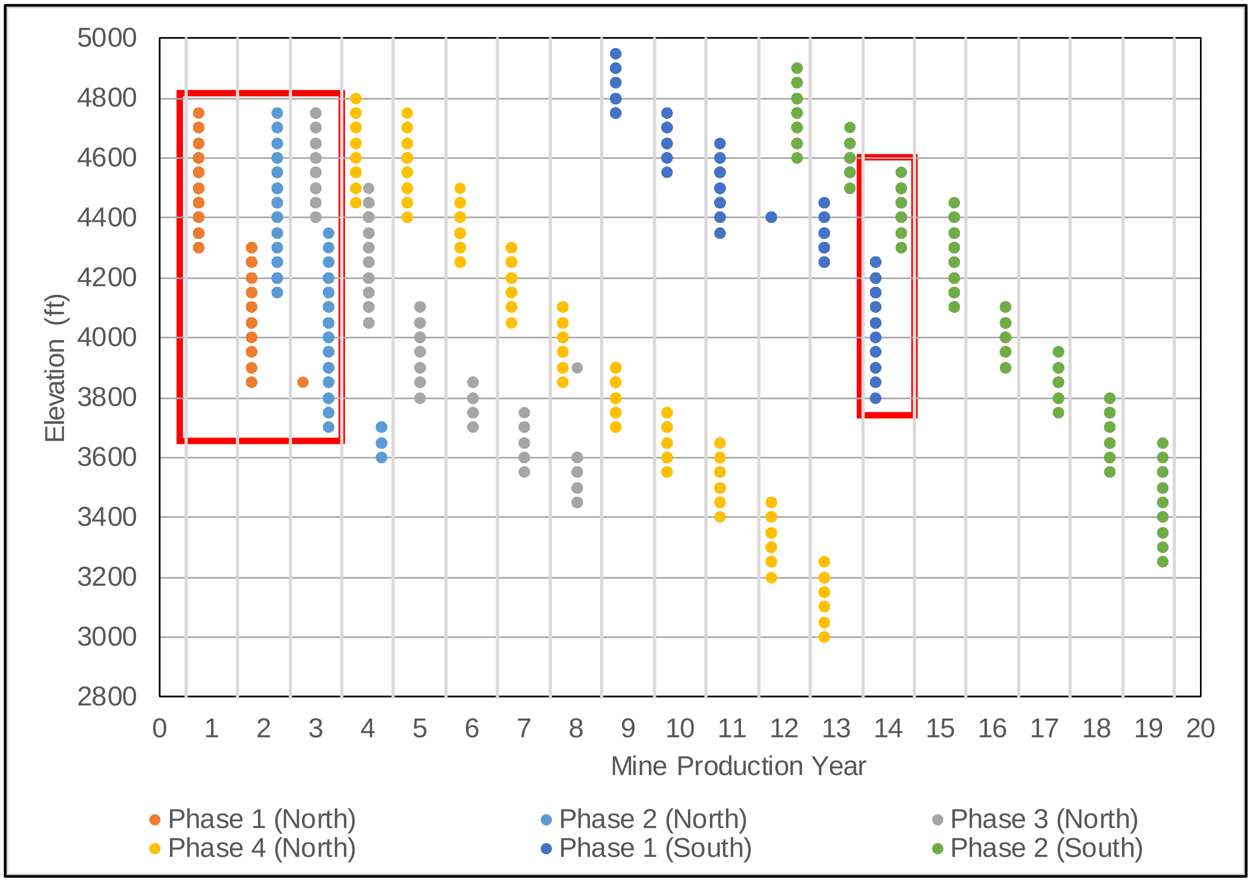


Figure 16-23: Waste Tons by Year by Pit (Golder, 2019)



For each of the pit phases, Figure 16-24 depicts the vertical advance rate by year. At the start of mining, an aggressive mining rate is required to advance the mill feed material as soon as practical. As an example, in the initial scheduled production years, the North Pit Phase 1 requires more than eight benches from a single phase to be mined (in a single year). A portion of these benches are low tonnage and in a ridge area that will require the establishment of a full working face. Golder recommends querying a contractor, one with smaller mining gear, to assist in the pre-development of these areas. In Figure 16-24, the years with aggressive mining rates are outlined in red.

Figure 16-24: Elevation of Mining by Phase by Year (Golder, 2019)



Note: Each of the various colored points denote a single mining bench. The groups outlined in red indicate periods where an aggressive (over 8 benches of vertical advance combined) mining rate was required to produce the schedule.

Item 17.0 RECOVERY METHODS

17.1 Underground

The Pumpkin Hollow Copper Underground Ore Concentrator has been designed to process 5,000 stpd of copper ore supplied from the East and E2 Deposits. The concentrator and the unit operations therein have been designed to produce a marketable concentrate targeted at 26.0% Cu while achieving overall copper recovery of 92%.

The copper recovery circuit will consist of a primary jaw crusher, crushed ore stockpile, SAG/ball mill comminution circuit, rougher flotation circuit, and a regrind/cleaner flotation to further liberate, recover and upgrade the copper from the ROM ores. Flotation concentrate will be thickened, filtered and stored in a concentrate loadout concrete bunker for subsequent truck loadout via front end loader.

DST have been progressed as the means of final deposition, to be consistent with the current permitting strategy. DST will be produced by thickening and filtering the final flotation tailings. A portion of the tailings will be used consistently as paste backfill but on an intermittent basis to meet mine schedule demand for backfill. Stacking of dry tailings by both a grasshopper/stacker system and trucking methods was assessed in this study and trucking was adopted as the preferred placement method.

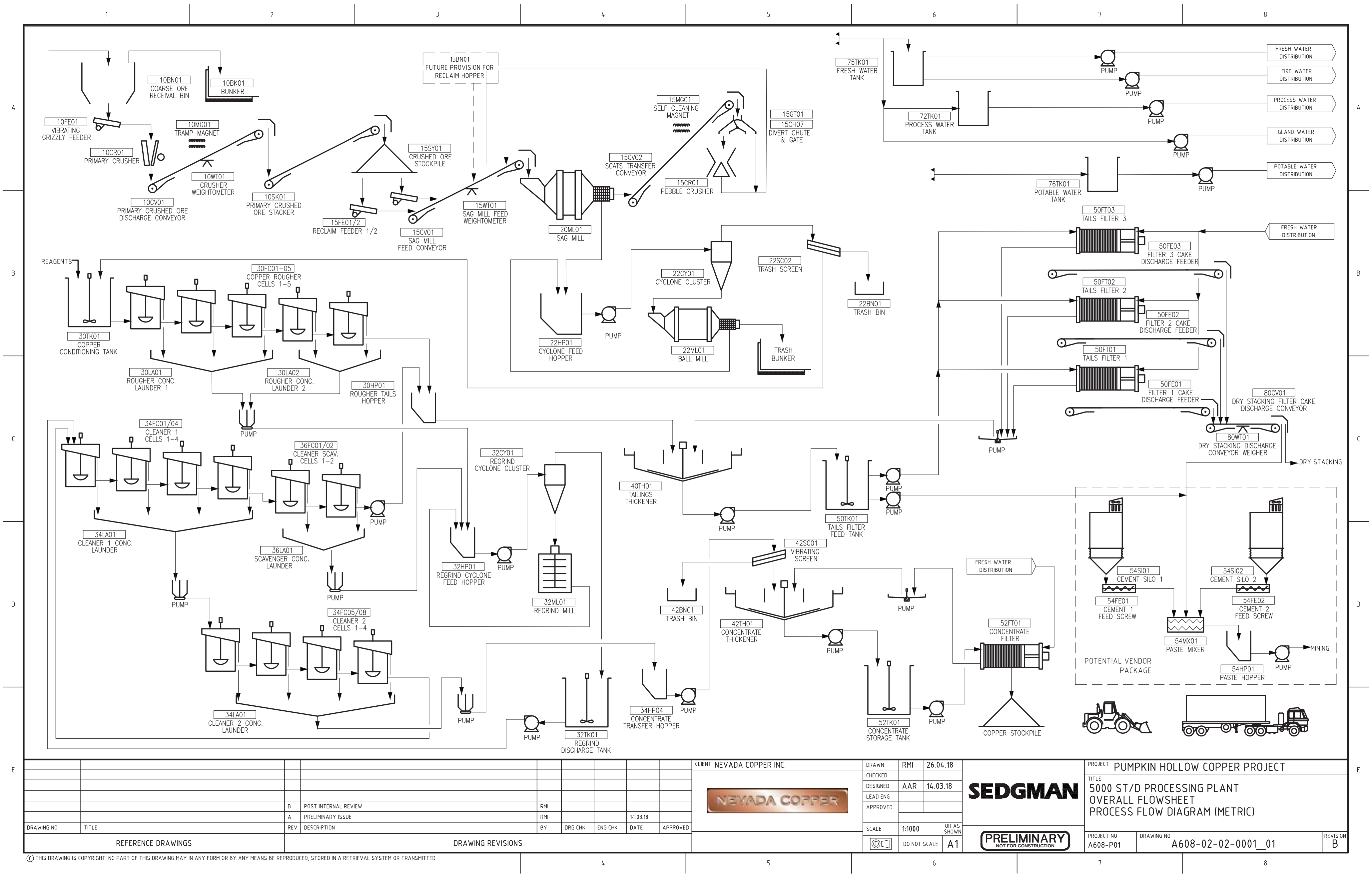
Thickening and filtration of tailings will allow for better process water management and control. Process water will be recycled from the tailings and concentrate thickener overflows. Fresh water will generally be used only for pump gland service, mill lube cooling, reagent preparation and safety showers/eyewash stations.

The process plant will consist of the following unit operations and facilities:

- A primary crushing jaw crusher
- A coarse ore stockpile and reclaim system
- A combined SAG/ball mill grinding circuit incorporating cyclones for classification
- A pebble crushing circuit
- A rougher flotation circuit
- A rougher concentrate regrinding circuit
- A first cleaner, cleaner-scavenger and second cleaner flotation circuit
- Concentrate thickening and filtration circuit, including a concentrate stockpiling and loadout area
- Tailings thickening and filtration circuit
- Tailings disposal at a dry stack storage facility
- A paste-backfill tailings processing facility to be used on a regular but intermittent basis for underground workings

A block flow diagram is presented in Figure 17-1.

Figure 17-1: Process Flow Sheet (Sedgman, 2018)



17.1.1 Major Design Criteria

The underground concentrator is designed to process 5,000 stpd, equivalent to 1,825,000 stpa with a 92% availability. The major criteria used in the design are outlined in Table 17-1.

Table 17-1: Major Design Criteria

Criteria	Unit	Value
Operating Year	d	365
Grinding and Flotation Availability	%	92
Milling & Flotation Process Rate	stph	228
SAG Mill Feed Size, 80% Passing	µm	117,000
SAG Mill Product Size, 80% Passing	µm	1,500
Pebble Crusher, Circulating Load	%	25
Ball Mill Grind Product Size, 80% Passing	µm	100
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/st	12.88
Bond Abrasion Index (Ai)	g	0.24
Regrind Size, 80% Passing	µm	28

The grinding mills were sized based on JKTech SMC test work for SAG mill sizing and Bond ball mill work index (BBMWi) data for ball mill sizing. Both sets of data were provided by Nevada Copper. The regrind mills were sized using the conventional BBMWi equation for ball mills and using the standard tower mill to ball mill efficiency factor as contributed by the vendors.

The flotation cells were sized based on the flotation times used during the laboratory test work in conjunction with typical flotation cell design parameters. Flotation cell sizes were adjusted with scale-up factors from the laboratory test work including an aeration factor, the froth lip loading and froth carrying capacity.

Concentrate streams downstream of the regrind cyclone feed have been sized for head grades of 1.80% Cu. Tailings streams downstream of the cyclone cluster have been sized for head grades of 1.48% Cu.

The process flowsheet does not involve the depression of pyrite in the rougher flotation circuit. If present in a great enough quantity, pyrite can impact flotation performance, effecting both copper grade and recovery. The pyrite to copper sulfide ratio has been mapped in the reserve, as it varies over the mine life.

Resolution of this pyrite to copper sulfide ratio is limited to parameters inherent in the resource model and mine planning/scheduling, as described in Item 14.1 and Item 16.1, respectively. There may be pockets of ore where the pyrite to copper sulfide ratio is higher, but ratios calculated on a monthly basis do not appear to be a concern for the development case option.

17.1.2 Process Plant Design

17.1.2.1 Operating Schedule and Availability

The underground process plant will be designed to operate on two 12-hour shifts per day, 365 days per year.

The SAG/ball mill comminution and flotation circuit availabilities are expected to be 92% or 336 operating days per year. This will allow sufficient downtime for scheduled and unscheduled maintenance of the process plant equipment.

Major scheduled maintenance commonly requires five consecutive days and occurs twice a year (10 days total). The remaining 19 days per year reflect a combination of minor scheduled maintenance and unscheduled maintenance.

17.1.3 Plant Description

17.1.3.1 Underground Ore Transportation

Ore from underground operations will be transported by two 14.5 st capacity skips to the coarse ore receival bin located on the surface. The receival bin will control the throughput of the primary crusher station.

17.1.3.2 Primary Crushing

The primary jaw crusher located in the primary crusher station will be fed from the coarse ore receival bin via a vibrating feeder. This equipment will be located above ground, and discharge via conveyor to the crushed ore stockpile. The jaw crusher will reduce ROM ore to 80% passing size (P_{80}) of 125 mm (4.9 in).

17.1.3.3 Coarse Ore Stockpiles & Reclaim

The coarse ore stockpile will be a production surge facility allowing for steady feed to be drawn to the grinding circuit.

Major equipment and facilities in this area includes:

- Coarse ore stockpile with 2,200 st live capacity. However, the feed to the coarse ore stockpile will be via a radial stacker. When primary crushing of plant feed is operating, the ore can be selectively placed within the live reclaim volume. Thereafter, dozer push of dead stockpile may be required, depending on when hoisting is scheduled within each shift. The dead stockpile capacity will be over 20,000 st and situated within 150 ft of the live reclaim areas.
- Reclaim vibrating feeders.
- Conveyor belts, metal detectors and belt rip detectors.
- Belt scale.
- Dust collection system.

Ore from the crushed ore stockpile will be reclaimed at a controlled feed rate using vibrating feeders onto a reclaim conveyor to supply the mills. A belt scale will be used to measure the feed and its output will be used to adjust the rate at which the vibrating feeders operate.

The coarse ore stockpile and reclaim area will be equipped with a dust collection system to control fugitive dust that can be generated during conveyor loading and the transportation of the ore.

17.1.3.4 Grinding & Classification

The grinding circuit will consist of a SAG/ball mill combination circuit. It will be a two-stage grinding process operated in a closed circuit with a pebble crusher and classifying cyclones. Grinding will be conducted as a wet process.

The grinding circuit will consist of:

- SAG mill feed conveyor
- Pebble crusher feed conveyor
- Conveyor belt weigh scale
- SAG mill, 21.0 ft diameter by 11.1 ft long effective grinding length (EGL), 3,200 hp
- Ball mill, 14.5 ft diameter by 23.2 ft long, 3,200 hp
- Pebble/Cone crusher
- SAG mill trommel screen
- Pair of cyclone feed slurry pumps
- Cyclone cluster
- Mass flow meter and nuclear density gauge
- Particle size analyzer system

Note all equipment sizing has been based on the process design criteria and could be changed during the tender and evaluation process, should additional information become available.

Ore from the coarse ore stockpile will be reclaimed and conveyed to the SAG mill. Water will be added to the SAG mill feed for wet grinding of the ore. The SAG mill will generally operate at 75% of critical speed, but will be fitted with a variable frequency drive to allow variation in rotational speed, to enable the mill to cope with some variations in ore characteristics.

The SAG mill discharge will be equipped with 2.5 inch by 2.75 inch pebble ports to remove critical size material. Oversize material removed at the SAG mill discharge will be conveyed via transfer conveyor to the pebble crusher. A pebble crusher will crush the pebbles to a P_{80} of 0.47 inches. The crushed material will be returned to the conveyor belt feeding the SAG mill for further grinding. The SAG mill discharge trommel screen underflow will be discharged into the cyclone feed hopper.

The ball mill and SAG mill will both discharge into the common cyclone feed hopper from which slurry will be pumped to a cyclone cluster. Cyclones will be operated in a manner that will ensure targeted particle size of the cyclone overflow stream (P_{80} of 100 µm) will be achieved. Circulating load to the ball mill will be 300% with cyclone underflow returning to the ball mill for further grinding.

Discharge from the ball mill / cyclone circuit will be the feed to the copper flotation circuit. The ball mill will operate at approximately 75% of critical speed and dilution water will be added to the grinding circuit as required.

Cyclone overflow will feed the rougher flotation conditioning tank at the head of the flotation process. Pulp density of the rougher flotation feed slurry will be maintained at approximately 35% w/w solids.

Provisions will be made for addition of lime to the rougher flotation conditioning tank for adjustment of slurry pH prior to flotation, if required.

Grinding media will periodically be added to the SAG and ball mills to maintain ball load and corresponding grinding efficiency. Steel balls will be added to each mill using a manual ball charging system.

17.1.3.5 Flotation & Regrind Circuits

Milled pulp will be processed in the rougher flotation circuit to recover the targeted minerals. Regrinding of the rougher concentrate followed by the cleaner flotation will be used to further upgrade concentrate to a saleable grade. Tank style flotation cells will be used throughout the flotation circuit.

The copper flotation circuit will include the following equipment:

- Flotation reagent addition facilities
- Rougher flotation tank cells, one bank of 5 × 1,413 ft³ each
- Regrind tower mill feed distribution box
- One concentrate regrind tower mill, 937 hp
- Regrind cyclone feed hopper and classification cyclone cluster
- First cleaner flotation tank cells – 4 × 706 ft³ each
- First cleaner-scavenger flotation tank cells – 2 × 706 ft³ each
- Second cleaner flotation tank cells – 4 × 353 ft³ each
- Hoppers, standpipes, slurry and concentrate pumps
- Particle-size analyzer for the regrind product and sampling system

Cyclone overflow from the grinding circuit will feed the flotation circuit via gravity flow. The slurry will be monitored for P_{80} particle size, and flotation feed samples will be taken periodically for process control and metallurgical accounting.

Cyclone overflow from the ball mill will discharge into the rougher flotation conditioning tank. Rougher flotation will consist of one bank of five rougher flotation cells operating at a design dry solids feed rate of 228 stph. Flotation reagents will be stage added at various points in the circuit as defined through testing. Flotation reagents to be added will include copper flotation collectors Aerophine 3418A and Aerophine 3477, frother MIBC and sodium metabisulfite, a pyrite depressant. Provisions will be made for supplementary reagent addition to the cleaner stages of the flotation circuit.

The copper minerals will be selectively floated into a rougher concentrate away from the unwanted or gangue minerals present in the slurry. Approximately 12% of the plant feed mass will report as rougher concentrate to the regrind circuit. Rougher tailings will be sampled automatically prior to discharge into the final tailings hopper for process control and metallurgical accounting purposes. The tailings hopper will also receive the cleaner scavenger tailings stream. The combined stream will constitute the final plant tailings and will be disposed of by using either DST disposal method or paste disposal.

Regrinding of the rougher concentrate before cleaner flotation will be incorporated to further liberate fine copper mineral grains from gangue constituents and enhance copper concentrate grade. For this purpose, a vertical stirred media tower mill will be used. A single stage of regrinding plus, two stages of cleaner flotation, and a stage of cleaner scavenger flotation have been selected to produce a final copper concentrate of acceptable grade and recovery.

Rougher concentrate enters the regrind circuit where it will be combined with cleaner scavenger flotation concentrate at the regrind cyclone feed hopper. The regrind circuit cyclone cluster classify the finely ground flotation concentrate into a cyclone overflow product with the grind size P_{80} of 28 µm and a cyclone underflow stream. The cyclone underflow will feed the regrind mill, which will discharge the finely ground material into the regrind discharge tank.

The regrind mill discharge will be combined with the regrind cyclone overflow to make up the feed for the cleaner flotation cells. Tailings from the first cleaner stage will report to the first cleaner scavenger flotation stage. Tailings from the first cleaner scavenger flotation stage will be pumped to the rougher tailings hopper. The cleaner scavengers concentrate will be pumped back to the regrind cyclone feed hopper.

First cleaner concentrate will be pumped to the second cleaner flotation stage. The second cleaner concentrate will be the final copper concentrate with a targeted grade of 26.0% Cu. Copper concentrate will be pumped to the concentrate thickener for dewatering before filtration.

Concentrate thickener overflow water will be reused in the grinding and flotation circuit as process water, providing this does not have a deleterious effect on the flotation of the copper minerals.

17.1.3.6 Concentrate Handling

Cleaner flotation concentrate will be thickened, filtered and stored prior to shipment. The concentrate handling circuit will have the following equipment:

- Concentrate thickener of 36 ft diameter and overflow standpipe
- Process water tank (common for all thickeners, and referenced in the process water section)
- Concentrate thickener underflow slurry pumps and filter feed tank (storage tank with agitator)
- Concentrate filter press feed pumps and filter press
- Filter press washing and filtrate handling equipment
- Dewatered concentrate storage and dispatch facility

Copper concentrate produced in the second cleaner flotation stage will be pumped to the concentrate thickener feed well. Flocculant will be added to the thickener feed to aid the settling process. Thickened concentrate will be pumped to the concentrate filter feed tank at an approximate density of 60% w/w solids. The concentrate filter will be vertical plate and frame filter press. Since filtration will be a batch process, the concentrate thickener and filter feed/storage tank will act as surge capacity for the filtration operation. The filter press will dewater concentrate producing a final concentrate with a moisture content of approximately 10% w/w. Filtrate will be returned to the concentrate thickener. The filter cake will be discharged directly onto the concentrate stockpile, from where it will be regularly be loaded into trucks for dispatch for sale.

Concentrate thickener overflow will gravitate to the process water tank for distribution within the crushing, milling and flotation circuits.

17.1.3.7 Tailings Handling

Final tailings will be thickened, filtered and dry stacked in the tailings dry stack facility, or diverted to the paste backfill plant. When used as a component of paste backfill, final tailings will be directed to the paste backfill plant. Thickened tailings will be the main paste constituent used in conjunction with cement and potentially other binders.

17.1.3.8 Dry Stack Tailings

The following process equipment will be required in the tailings handling area:

- Tailings thickener at 59 ft diameter and overflow standpipe
- Process water tank (common for all thickeners, and referenced in the process water section)
- Tailings thickener underflow slurry pumps and filter feed tanks
- Tailings filter press feed pumps and tree tailings vertical plate and frame filter presses

- Filter press washing and filtrate handling equipment including belt feeders and transfer conveyors

Rougher flotation tailings, together with the first cleaner scavenger tailings, will constitute the final plant tailings, which will report to the tailings cyclones for classification coarse/fine. Cyclone overflow (fine tails) will be thickened in the tailings thickener, and underflow from this thickener will report to a filter feed tank and 1 x plate and frame filter dedicated to filtering fine tails.

Cyclone underflow (coarse tails) will report to a filter feed tank and 2 x plate and frame filters dedicated to filtering coarse tails.

The filter cake discharge from these tails filters then depends whether the paste plant is operational or not as follows:

- Paste plant feed off: Filter cake from the tails filters report to the dry stacking conveyor system to be stacked on a stockpile and dry stacked by a mobile equipment fleet
- Paste plant feed on: Filter cake from the fine tails filter reports to dry stack system, and the cake from the coarse tails filters (x2) discharges on a conveyor belt leading to the paste plant (mixer)

Final plant tailings will be thickened in the tailings thickener to an underflow density of 55% w/w solids. Flocculant will be added to facilitate the settling of the solids and to aid in supernatant clarity. The tailings thickener has been sized to handle the entire quantity of flotation plant tailings, ensuring continued operation even when the paste facility is offline or not in use.

The thickened tailings will be pumped to agitated tailings filter feed tanks. Since filtration by filter press is a batch process, the tailings thickener and filter feed tanks will act as surge capacity before the filtration operation. The filter press will dewater the tailings to produce a relatively dry cake with a moisture content target of 15% w/w or less. Filtrate from the filter presses will be returned to the tailings thickener.

17.1.3.9 Paste Plant

When the underground mine demands paste backfill for stopes, the coarse tails filter cake can be diverted from DST to the paste plant.

The paste plant will be provided with a blend of deslimed flotation tailings and flotation tailings. Blending a coarse cut of deslime cyclone underflow with finer flotation tails will provide the greatest flexibility to regulate and control grading to the paste plant. Cyclone underflow and a blend of flotation tailings will be used to provide the necessary quantity of fines for paste production.

The following process equipment will be required in the paste backfill area:

- Paste plant feed conveyor (metering conveyor – carrying coarse tails filter cake)
- 520 yd^3 binder silo and feeder system capable of delivering approximately 8 stph
- Paste mixer, pump and discharge hopper

The paste backfill plant has been designed to process 145 yd^3/hr of feed tails. The portion of tailings intended for use in the paste backfill will be mixed with binder and water to produce a final paste mixture.

The paste fill requirements will meet the requirements of the mining schedule developed as part of this PFS study, apart from three months late in the mine life where the mine paste fill demand exceeds available paste fill. This is proposed to be addressed in the feasibility study, but is considered manageable

17.1.3.10 Reagent Handling & Storage

Various chemical reagents, including collectors, frother and flocculant, will be added to the process slurry streams to facilitate the recovery of the copper minerals during the flotation process as well as to aid in solids/liquid separation process. Preparation of the various reagents will require:

- A depressant (sodium metabisulfite) bulk handling system including totes, holding tanks and metering pumps
- A flocculant preparation facility
- A hydrated lime preparation and distribution facility
- Eye-wash stations, safety showers and relevant safety equipment

Reagents will first be added to the flotation circuit to modify the mineral particle surfaces and enhance the floatability of the valuable minerals so they can be separated from the gangue minerals. Fresh water will be used in the preparation of the reagents supplied in bulk powder/solids form and reagent solutions that require dilution prior to addition to the process. The reagent solutions will be added at various addition points, such as conditioning tank, rougher, cleaner, cleaner scavenger flotation circuits, and thickeners using metering pumps.

Aerophine 3418A and Aerophine 3477 will be the preferred collectors and will arrive at the plant as a neat liquid in the 35 ft^3 totes.

The preferred frothing agent will be MIBC, which will also be supplied as a neat liquid.

Sodium metabisulfite will be delivered to site in bulk bags and prepared in the depressant mixing system to produce a 20% mixing strength solution. It will be used as a pyrite depressant in the cleaner flotation cells.

Hydrated lime will be delivered in bulk tanks and off-loaded pneumatically into a silo. Lime slurry will be prepared in a hydrated lime mixing system at a slurry concentration of 20% w/w solids.

Flocculant will be prepared in a flocculant mix system to produce a dilute solution with a 0.25% w/w solution strength. Grinding media will be added to the mills as required using manual ball charging systems.

Estimated grinding media consumption is based on the Bond abrasion index (Ai) equation, using the average Ai of the deposits and estimated equipment power consumption.

To ensure spill containment, the reagent preparation and storage facility will be located within a contained area with references to secondary containment description in Item 18.1.

17.1.3.11 Assay & Metallurgical Laboratory

An on-site assay and metallurgical laboratory will be provided that will consist of basic analytical equipment required for operational requirements of the processing facility. Detailed analysis for samples from the mine, concentrator and environmental compliance monitoring, which require specialized equipment, will be performed by local third-party laboratories.

During site construction and mine development, all assays will be performed by the outside laboratories.

17.1.3.12 Water Supply

The concentrator and paste plant have common water supply facilities. At each of these locations, fresh water and process water will be provided to support the operation. This carries forward the assumption that fresh water supply will come from on-site groundwater and will be pumped from dewatering wells to the fresh water tank. Total fresh water demand for the process is estimated to be 326 gpm (74 m^3/hr).

17.1.3.13 Fresh Water Supply System

Fresh/fire water will be supplied from the fresh water tank for the plant wide distribution.

Fresh water supply will be used primarily for the following:

- Fire water for emergency use
- Cooling water for mill motors and mill lubrication systems
- Gland service for slurry pumps
- Reagent preparation water
- Potable water treatment plant feed
- Make-up water for the main process facility
- Filter cloth wash water
- Particle size analyzer

The fresh/fire water tank will be configured ensure that tank always holds the minimum fire water volume requirements.

The potable water from the fresh water source will be treated and stored in a potable water storage tank prior to delivery to various service points.

17.1.3.14 Process Water Supply System

Process water generated in the flotation circuit as thickener overflows will be reused in the main plant's process circuit via the process water tank. Reclaimed water derived from the concentrate and tails thickener overflows will be gravity fed directly to the main process water tank for distribution around the process plant.

17.1.3.15 Air Supply

Separate air service systems will supply air to the following areas:

- Low-pressure air for flotation cells will be provided by air blowers.
- High-pressure air supply for operation of the concentrate and tailings filter presses will be provided by dedicated air compressors.
- Plant and instrument air for distribution throughout the plant will be provided from the dedicated plant air compressors.

17.1.3.16 Online Sample Analysis

Process control will rely on a particle size and slurry on stream analyzer located in the flotation plant area. The analyzer will be fed from multiple samplers located within the mill and flotation area and will perform analysis on the particle size, copper, iron, sulfur content and other specific metallurgical parameters from the various process streams. Samples will be taken at a frequency sufficient to ensure real time circuit control and material balance.

17.1.4 Process Manpower

Process plant salaried personnel estimates were developed to provide adequate supervision and technical support for the daily operation of the processing facility. 53 personnel for the facility are estimated to be required, as detailed in Table 17-2.

Table 17-2: Process Plant Salaried Manpower

Description	No. per Crew	No. of Crews	Total
Operations Superintendent	1	1	1
Production General Supervisor	1	1	1
Processing Clerk	1	1	1
Foreman - Shift	1	4	4
Control Room Operator	1	4	4
Crushing Operator	1	4	4
Milling Operator	1	4	4
Flotation Operator	1	4	4
Filtration Operator	1	4	4
Reagents/Services	1	2	2
Concentrate Loading	1	2	2
Paste Plant Operator	1	2	2
Clean-Up / Day Crew	4	1	4
Chemist	1	1	1
Senior Assayer	1	2	2
Laboratory Sampler/Assayer	3	2	6
Senior Metallurgist	1	1	1
Metallurgist	1	2	2
Maintenance General Supervisor	1	1	1
Maintenance Planner	1	1	1
Maintenance Supervisor	1	1	1
Maintenance Clerk	1	1	1
TOTAL			**53**

Note: The following operations have been priced as contract operations and the manning is not included in the above table:

1. Dry stacking of tails, which will consist of 6 operators on day shift, 7 days per week.
2. Concentrate trucking, which will consist of 1 Operations Supervisor, 2 Transload operators and 2 truck drivers operating day shift, 5 days per week.

17.1.5 Process Plant Control

17.1.5.1 Overview

A programmable logic controller (PLC) with a supervisory control and data acquisition (SCADA) system will provide equipment interlocking, process monitoring, and process control functions with a supervisory control. The PLC/SCADA system will generate production reports and provide real time data and malfunction analysis, as well as logging of all process upsets. All process alarms and events will be logged by the PLC/SCADA.

Operator interface to the PLC/SCADA will be via computer-based operator workstations located in the process plant. Control room will be staffed by trained personnel 24 hours per day. The operator workstations will be capable of monitoring the entire plant site process operations, viewing alarms and controlling equipment within the plant.

Intelligent-type motor control centers (MCCs) will be located in electrical rooms throughout the plant. A serial interface to the PLC will facilitate the MCC’s remote operation and monitoring.

For site-wide infrastructure (telephone, internet, security, fire alarm and control system), a fiber optic backbone will be installed throughout the Property.

17.1.5.2 Control Philosophy

The control objective of the primary crushing area will be to provide a crushed product to the crushed ore stockpile prior to grinding and flotation. A single PC workstation will be installed in the primary crushing area to monitor all crushing operations. Control and monitoring functions will include:

- Equipment power draw, bearing temperatures and lubrication system status
- Vendors’ instrumentation packages

The control objective of the coarse ore storage and reclaim area will be to provide a crushed ore delivery buffer and a consistent blended SAG mill feed.

To control and monitor all concentrator processes and ancillary operations, three PC workstations will be installed in the process building central control room. The PC workstations will control and monitor all parts of the processing facilities:

- An automatic sampling system will collect samples from various process streams for online analysis and daily metallurgical balance.
- Particle size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits. The particle-size analyzers described earlier will provide main inputs to the control system.
- An online stream analyzer will be used to monitor the performance of the flotation process to optimize concentrate grade and metal recoveries.

- Closed circuit television cameras will be installed at various locations throughout the plant, including the primary crushing facility, stockpile reclaim tunnel, SAG and ball mill grinding area, flotation area, regrind area, concentrate handling building and the tailings handling facilities. The cameras will be monitored from the plant control rooms.

17.2 Open Pit

17.2.1 Major Design Criteria

The Phase I processing facility has been designed to process 37,000 stpd, equivalent to 13.5 Mstpa. The expanded Phase II processing facility has been designed to process an additional 33,000 stpd, equivalent to an additional 12.0 Mstpa. Upon completion of Phase I and Phase II, the overall Open Pit Project will be designed to process 70,000 stpd, equivalent to 25.5 Mstpa. The major criteria used in the design are outlined in Table 17-3.

Table 17-3: Major Design Criteria

Criteria	Unit	Phase I Value	Phase II Value	Overall Phase I & II Value
Operating Year	d	365	365	365
Grinding and Flotation Availability	%	92	92	92
Milling & Flotation Process Rate	stph	1,688	1506	3194
SAG Mill Feed Size, 80% Passing	µm	150,000	150,000	150,000
Ball Mill Circuit Product Size, 80% Passing	µm	150	150	150
Ball Mill Circulating Load	%	250	250	250
Bond Ball Mill Work Index	kWh/t	15.8	15.8	15.8
Bond Ai	g	0.32	0.32	0.32
Concentrate Regrind Size, 80% Passing	µm	28	28	28

The grinding mills were sized using the JK SimMet simulation program and the BBMWi data for ball mills was provided by Nevada Copper. The regrind mills were sized using the conventional BBMWi equation for ball mills, and using the standard tower mill to ball mill efficiency factor as contributed by the vendors.

The flotation cells were sized based on the bench flotation times used during the laboratory test work. Typical flotation cell design parameters have been used in the design of the flotation circuit. Flotation cell sizes were adjusted for scale-up factors from laboratory scale test work including an aeration factor.

17.2.1.1 Introduction

The Phase I processing facilities have been designed to process 37,000 stpd of copper ore. The processing facilities unit operations therein are designed to produce a marketable concentrate targeted at 25.5% Cu, or greater at a copper recovery of greater than 89.3%.

The Phase II processing facilities have been designed to process and additional 33,000 stpd of copper ore. Phase II process plant and associated facilities have been designed to largely replicate the Phase I design downstream of and including the coarse ore stockpile, while minimally extending shared facilities and services. The total process daily production rate will be 70,000 stpd ore when both the Phase I and Phase II plants are in operation. Both Phase I and Phase II are designed to share the same crushing plant, overland conveyor, concentrate area, analyzer and certain infrastructure. Phase II will add an additional stockpile and reclaim tunnel, which will feed the Phase II processing facilities, which will operate mostly independently from the Phase I processing facilities. Phase I and Phase II have independent processing facilities and tailings filter plants.

The Phase I processing facility will consist of a coarse ore storage facility, a SABC (SAG mill, ball mill and pebble crusher) comminution circuit, rougher flotation, concentrate regrind and cleaner flotation circuit to liberate, recover and upgrade copper from the ROM ores. The Phase II processing facility will have the same unit of operations, although with minor modifications for a reduced throughput. Flotation concentrate will be thickened, filtered, and sent to a concentrate load out area for subsequent shipping.

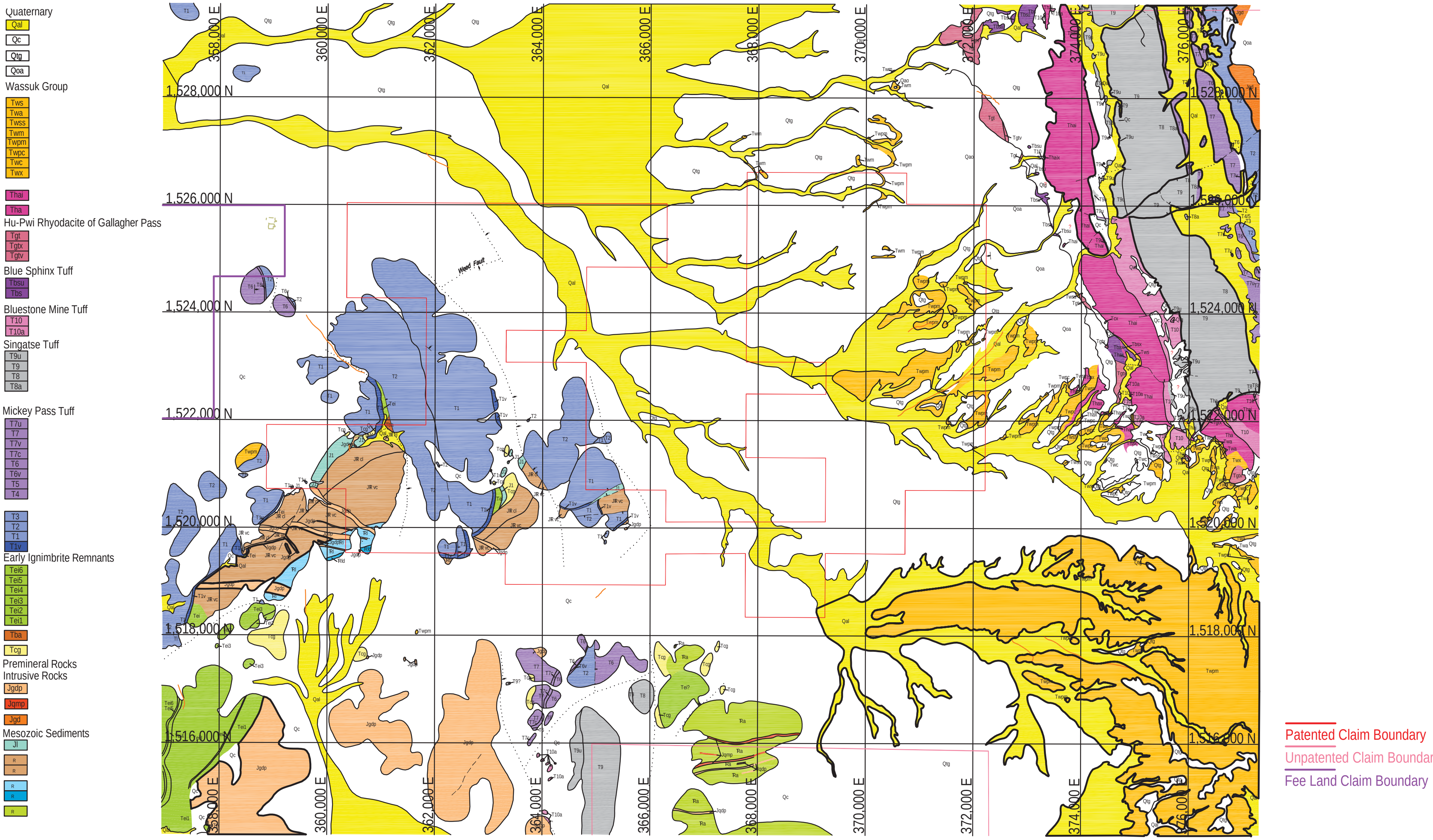
DST is the planned means of final tailings deposition, having substantially less water contained than tailings discharged directly from a concentrator. DST will be produced by a way of thickening and filtering the final flotation tailings.

Thickening and filtration of tailings allows for better process water management and control. Process water from the thickener overflow streams will be recycled back to the concentrator. Fresh water will generally be used only for pump gland service, mill lube cooling, SAG mill ring motor cooling, reagent preparation and safety showers/eyewash stations.

The Phase I process plant will consist of the following unit operations and facilities:

- ROM receival area, including primary crusher
- Overland conveyor
- Coarse ore receiving and storage area from the open pit mine(s)
- A coarse ore stockpile reclaim system
- A combined SAG/ball mill grinding circuit incorporating cyclones for classification
- A SAG mill pebble crushing circuit
- A rougher flotation circuit
- A rougher concentrate regrinding circuit
- A first cleaner, second cleaner and cleaner scavenger flotation circuit
- A concentrate thickening and filtration circuit including a concentrate stockpile and dispatch area
- Tailings thickening and filtration circuits
- Tailings disposal at a DST facility

A process flowsheet is presented in Figure 17-2.



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	Project: Pumpkin Hollow Project	Project Number: 114-311399
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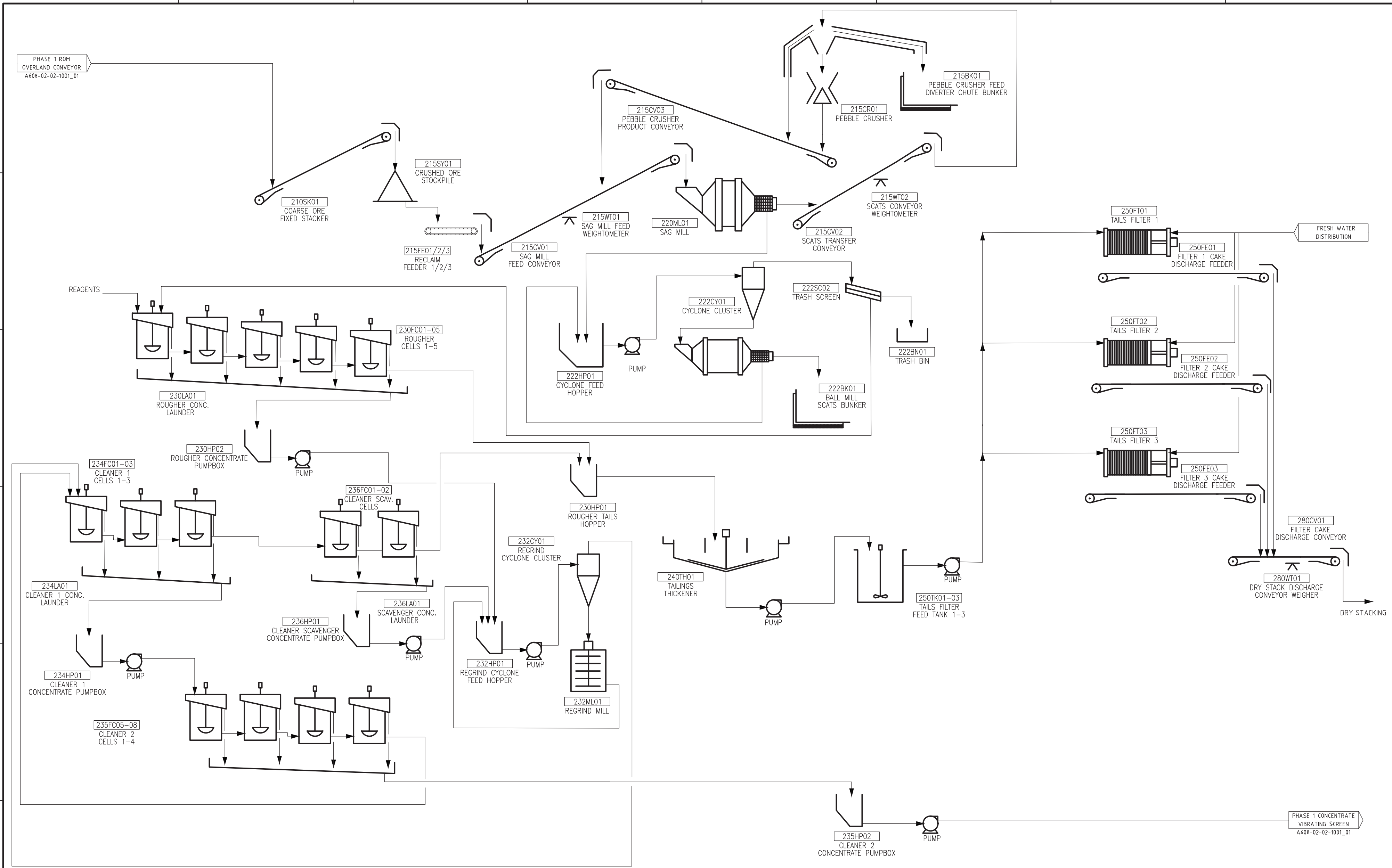
Figure 7-2: General Geologic Map of the Development Area (grid in ft & geographic north) - (2017 Technical Report)

The Phase II process plant will consist of the following additional unit operations and facilities:

- Overland conveyor modifications
- Coarse ore stockpile stacker
- A coarse ore stockpile reclaim system
- A combined SAG/ball mill grinding circuit incorporating cyclones for classification
- A SAG mill pebble crushing circuit
- A rougher flotation circuit
- A rougher concentrate regrinding circuit
- A first cleaner, second cleaner and cleaner scavenger flotation circuit
- Tailings thickening and filtration circuits
- Tailings disposal at a DST facility

A process flowsheet is presented in Figure 17-3.

Figure 17-3: Phase II Process Flowsheet (Sedgman, 2019)



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DRAWING NO	TITLE	REV	DESCRIPTION	BY	DRG CHK	ENG CHK	DATE	APPROVED
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TITLE 33,000 ST/D PROCESSING PLANT OVERALL FLOWSHEET PROCESS FLOW DIAGRAM (METRIC)

CLIENT DRAWING NO

SCALE 1:1000 OR AS SHOWN

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17.2.2 Process Plant Design

The Phase I and Phase II process plants are designed to operate as almost entirely independent plants. Both Phase I and Phase II will incorporate the following independent facilities:

- Stockpile, including stockpile reclaim
- Grinding and pebble crushing
- Flotation and concentrate regrind
- Tailings thickening, filtration and DST
- Process, fresh and potable water
- Low-pressure, high-pressure instrument air
- Reagent facilities

17.2.2.1 Operating Schedule & Availability

The process plant has been designed to operate on the basis of two 12-hour shifts per day, 365 days per year.

The SAG/ball mill comminution and flotation circuit availabilities are expected to be 91.3% or 333 operating days per year. This will allow sufficient downtime for scheduled and unscheduled maintenance of process plant equipment.

Major scheduled maintenance commonly requires 5 consecutive days and is planned to occur twice a year (10 days total). The remaining 22 operating days per year, allocated to maintenance, reflect a combination of minor scheduled maintenance and unscheduled maintenance.

17.2.3 Plant Description

17.2.3.1 Ore Transportation

Ore transportation from the North and South Pits to the ROM dump pocket is by haul trucks as described in Item 16.2.11.

17.2.3.2 Primary Crushing

The combined Phase I and II primary crushing of surface ore from the North and South Pits will be performed at the pit crest, with operation of the single crusher being handled by mining personnel. Crushed ore will be conveyed via a single overland conveyor to the Phase I coarse ore stockpile initially, and to both the Phase I and Phase II coarse ore stockpiles subsequently. The crushing plant has been designed to process 70,000 stpd, and will not need to be expanded during Phase II.

17.2.3.3 Coarse Ore Stockpiles & Reclaim

Each phase will have its own coarse ore stockpile, which will provide production surge capacity allowing steady feed to be directed to the Phase I and II the grinding circuits. The primary crusher will reduce ROM ore to a P_{80} of 150 mm.

The major equipment and facilities in this area include:

- Coarse ore stockpile, nominally 21,440 st live capacity (i.e., 14 hours live capacity)
- Reclaim apron feeders
- Dust collection system

Ore from the Phase I crushed ore stockpile will be reclaimed at a rate of 1,688 stph, and from the Phase II crushed ore stockpile at a rate of 1,506 stph. This will be under controlled feed rate conditions using apron feeders. These feeders will discharge the reclaimed ore onto a conveyor belts feeding the Phase I and Phase II SAG mills. Belt scales will control the feed to the Phase I and Phase II SAG mill(s) by controlling the rate at which the apron feeders operate.

Each coarse ore stockpile and reclaim area will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the crushed rock.

17.2.3.4 Grinding & Classification

Each grinding circuit will be configured with a SAG mill and ball mill. Each grinding circuit will be a two-stage operation with the SAG mill in closed circuit with a pebble crusher and the ball mill in closed circuit with a classifying cyclone. The SAG mills will be equipped with pebble ports to remove pebbles coarser than 2.55 inches. Grinding will be conducted as a wet process at a nominal feed rate of 1,642 stph of material.

Each grinding circuit will include:

- SAG mill feed conveyor
- Pebble crusher feed conveyor
- Conveyor belt weigh scales, metal detector and tramp metal magnet
- Phase 1 - SAG mill (36 ft x 18 ft) 15MW, ball mill (25 ft x37 ft) 15 MW
- Phase 2 - SAG mill (36 ft x 16 ft) 14MW, ball mill (24 ft x37 ft) 14 MW
- Pebble crusher (2 of units 73 inch), 600 kW
- SAG mill discharge trommel
- Set of cyclone feed slurry pumps
- Cyclone cluster (13 units of 30 inch)

- Mass flow meter, nuclear density gauge and particle size analyzer/sampling system

Crushed ore reclaimed from the stockpiles will be fed at a controlled rate to the SAG mills. Water will be added to the SAG mill feeds for wet grinding of the ore. The SAG mills will generally operate at 78% of theoretical critical speed.

The SAG mill discharges will be equipped with 2.55-inch pebble ports to remove critical size material. Each phase’s oversize material removed at the SAG mill discharge screens will be conveyed via transfer conveyors to that phase’s pebble crusher. The cone crushers will crush the pebbles to a P_{80} range of 0.25 to 0.75 inches. The crushed material will be returned to each phase’s conveyor belt feeding the SAG mills for further grinding. The SAG mill discharge screen underflows will be discharged into each phase’s cyclone feed pump box.

The ball mills subsequent to the SAG mills will operate in closed-circuit with classification hydrocyclones mounted in a cluster for each phase. The product from the ball mills will be discharged into each phase’s cyclone feed pump box where it is combined with the SAG mill discharge prior to being pumped to each phase’s primary grinding cyclopac. Classification size for the cyclone overflows will be a P_{80} of 150 µm; circulating load to the ball mills will be targeted at 250%, with the cyclone underflow returning to each phase’s ball mill for further grinding.

The fresh feed to the Phase I ball mill will total 1,688 stph, and 1,506 stph for the Phase II ball mill. These feeds and will also constitute the feed rate to that phase’s copper flotation circuit. The ball mills will operate at 75.0% of theoretical critical speed. Dilution water will be added to the grinding circuits as required.

Cyclone overflow from the classification circuits will discharge into the feed of that phase’s rougher copper flotation circuit. The pulp density of each phase’s cyclone overflow slurry will be approximately 35.1% solids.

Provision will be made for the addition of lime to the SAG mills and the cyclone feed pump boxes for the adjustment of slurry pH in the grinding circuit prior to flotation.

Grinding media will regularly be added to the SAG and ball mills to maintain charge level and grinding efficiency. Steel balls will be added to the SAG mill using an automatic ball charging system.

17.2.3.5 Flotation & Regrind Circuits

Milled pulp will be processed using rougher flotation to recover the targeted minerals. Regrinding of rougher concentrate, followed by the cleaner flotation stage, will be used to further upgrade the rougher copper concentrate into a high-grade copper concentrate. Tank style flotation cells will be used throughout each phase’s flotation circuit.

Each phase's copper flotation circuit will include the following equipment:

- Flotation reagent addition facilities
- Bank of five rougher flotation tank cells (5 of 300 m^3 cells)
- Regrind tower mill feed distribution box
- One concentrate regrind tower mill (3,000 kW unit)
- Regrind cyclone feed pump box and classification cyclone cluster
- Bank of five first cleaner flotation tank cells for Phase I (5 of 100 m^3 cells) and Phase II (3 of 100 m^3 cells)
- Bank of two first cleaner scavenger flotation tank cells Phase I (2 of 50 m^3 cells) and Phase II (2 of 50 m^3 cells)
- Two banks of four second cleaner flotation tank cells Phase I (8 of 50 m^3 cells) and Phase II (4 of 50 m^3 cells)
- Pump boxes, standpipes and concentrate pumps
- Elemental composition analyzer (shared for Phase I and II)

Each phase will operate independently and the following will occur:

- The cyclone overflow from each grinding circuit will feed the flotation circuit by gravity flow from the ball mill grinding circuit cyclone cluster. Flotation feed samples will be taken periodically for process control and metallurgical accounting.
- Cyclone overflow from the ball mill will discharge into the feed end of the rougher flotation line. Rougher flotation will consist of one line of rougher flotation cells operating at a design solids total feed rate of 1,688 stph for Phase I (1,506 stph for Phase II). Flotation reagents (collectors and frother) type and dosing rates will be as per metallurgical test work results. Provision will be made for supplementary reagent addition to the cleaner stages of the flotation circuit.
- The copper minerals will be selectively floated into a rougher concentrate away from the other minerals-gangue components present in the slurry. The rougher and cleaner scavenger tailings will be sampled automatically prior to discharge into the final tailings pump box for process control and metallurgical accounting purposes. This combined stream will constitute the final tailings leaving the plant.
- Regrinding and upgrading, via cleaner flotation, will be incorporated to more fully liberate the fine grains of copper minerals from the gangue constituents and enhance copper concentrate grade. Two stages of cleaner flotation as well as a stage of cleaner scavenger flotation, operated

in closed circuit with a single stage of regrinding, is the selected method by which to produce a final copper concentrate of acceptable grade and recovery.

- Rougher concentrate enters the cleaner flotation circuit where it will be combined with regrind mill circuit coarse discharge inside the regrind cyclone feed pump box. The regrind circuit cyclone cluster will then separate reground flotation concentrate into a fine cyclone overflow product (targeted P_{80} of 28 µm) and a coarse cyclone underflow product. The regrind mill will be a single vertical stirred tower mill. The regrind mill will discharge finely milled material back into the regrind cyclone feed pump box. This will be combined with rougher flotation concentrate, the first cleaner scavenger concentrate and the second cleaner tailings, constituting the feed for classification by the cyclones.
- The regrind cyclone overflow will become feed to the first cleaner flotation stage. Tailings from the first cleaner stage will report directly to the first cleaner scavenger flotation stage. Tailings from the first cleaner scavenger flotation stage will report to the final tailings pump box. The first cleaner scavenger concentrate will report to the regrind cyclone feed pump box for re-classification.
- The first cleaner concentrate will feed the second cleaner flotation stage. The second cleaner concentrate will be the final unfiltered copper concentrate with a design copper concentrate grade of 25.5%.
- Copper concentrate from both Phase I and Phase II will feed directly to the single copper concentrate thickener for dewatering. This thickener is designed for the full 70,000 stpd capacity. The concentrate thickener and tailings thickener overflow will be collected in the process water tank for recycling within the Phase I and Phase II mill circuits. A portion of the overflow will be used within the Phase I and Phase II filtration areas as a filter wash water.

17.2.3.6 Concentrate Handling

Final copper concentrate will be thickened, filtered and stored prior to shipment. The concentrate handling circuit will be capable of handling concentrate generated by the full 70,000 stpd plant, and include the following equipment:

- One concentrate thickener (30 m unit) sized for both phases' throughput and an overflow standpipe
- One concentrate thickener overflow pump and one underflow slurry pump (with standby pumps to be installed in the future)
- One concentrate filter feed tank and one filter press feed pump (with another standby pump to be installed in the future)
- One concentrate plate and frame filter press (1 of 2 x 2 m plate x 80 chamber units) sized for both phases' throughput
- Filter press washing and filtrate handling equipment
- Dewatered concentrate storage and dispatch facility

The copper concentrate from both the Phase I and Phase II flotation circuits will be pumped from the second cleaner flotation stage to the shared concentrate thickener feed well. Flocculant will be added to the thickener feed to aid the settling process. Thickened concentrate (thickener underflow) will be pumped to the concentrate filter feed tank at a density of approximately 55% solids. The concentrate filter feed tank will be agitated and the concentrate filter will be a plate and frame pressure filter unit. Since filtration will be a batch process, the concentrate filter feed tank will also act as a surge tank for the filtration operation. The filter will dewater the concentrate producing a final concentrate with a moisture content of approximately 10%. Filtrate will be returned to the concentrate thickener. Filter press solids will be discharged directly onto the concentrate stockpile. Dewatered concentrate will be loaded into fully enclosed containers by front end loader, which will be stored outside the building. These containers will be loaded into trucks periodically for dispatch off the Property.

17.2.3.7 Tailings Handling

Final tailings from the processing facility will be thickened, filtered and dry stacked in separate Phase I and Phase II tailings buildings.

In each phase, the separate tailings handling area includes the following process equipment:

- One tailings thickener (Phase I is a 54 m unit) and (Phase II is a 51 m unit) and overflow standpipes
- One tailings area process water tank (referenced in the process water section)
- Tailings thickener overflow pumps; underflow slurry pumps

- Tailings filter feed stock tanks (with agitators)
- Four tailings plate and frame filter presses Phase I is (4 of 2 x 4 m plate x 120 chamber units) and Phase II is (3 of 2 x 4 m plate x 120 chamber units), with two tailings filter feed pump per filter. One of the filter presses for Phase I is included as standby equipment
- Filter press washing and filtrate handling equipment
- One filter press belt feeder per filter
- One filter building discharge conveyor

For each phase, the rougher flotation tailings together with the first cleaner scavenger tailings will be the final plant tailings. These streams will be pumped to the corresponding filtration buildings where they will be thickened and filtered, producing a filter cake as a part of the tailings handling process. Once filtered, the tailings will be primarily conveyed to an elevated bin, which will be designed to gravity-feed the haul trucks. This bin will also have the option to discharge into an emergency stockpile during prolonged downtime event, in order to mitigate solidification of solids in the bin. Additionally, a filter cake stockpile can also be fed from the tailings filter, which will allow the elevated bin to be bypassed entirely.

The final plant tailings will initially be thickened in the tailings thickener to an underflow density of 55% solids. Flocculant will be used to facilitate the settling of the solids and to aid in supernatant clarity.

Thickened tailings will be pumped to the tailings filter feed tanks using thickener underflow slurry pumps. The tailings filter feed tanks will be agitated tanks. Tailings filtration will be done in multiple filter press units. Since filtration will be a batch process, the tailings filter feed tanks will also act as surge tanks for the filtration operation. There will be four filter presses for Phase I (three filter presses for Phase II), and each filter press will dewater the tailings to produce a “dry” cake with a moisture content of about 15%. The filtrate will be returned to the corresponding tailings thickener. The filter press solids will be discharged onto belt feeders, which in turn feed the corresponding transfer conveyor, which will feed the shared Phase I and Phase II dry-stack pad feed conveyor system.

Thickening and filtration of the tailings will facilitate the recovery of process water required for reuse in the plant prior to final deposition of the plant tailings. Reclaim process water will be recovered as overflow from the tailings thickeners as well as overflow from the concentrate thickener for reuse in the plant as general process water. Some overflow from the tailings thickeners will be diverted back to the corresponding thickener to dilute the overall thickener feed to 26.9% solids by weight, or less. In addition, some of the water bound for the general process water facility will be diverted back to the filtration buildings for use as cloth wash water for the tailings filter presses, as needed.

17.2.3.8 Reagent Handling & Storage

Various chemical reagents will be added to the process slurry streams to facilitate the recovery of the copper minerals during the flotation process. Preparation of the various reagents will require the following per phase:

- A bulk handling system.
- Mix and holding tanks.
- Metering pumps.
- A flocculant preparation facility.
- A lime slaking and distribution facility.
- Eye-wash stations, safety showers and other applicable safety equipment.

Various chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the copper concentrate product. Fresh water will be used in the preparation of the various reagents that will be supplied in powder/solids form, or as solutions that require dilution prior to addition to the slurry. These reagent solutions will be added at the addition points of the various flotation circuits and streams using metering pumps.

The collector reagents will arrive at the plant in dry solid bulk bags. Bulk reagent handling systems (including mixing and holding tanks, reagent transfer and dosing pumps) will be used to add collector to the process. The frother reagent will be delivered as a neat liquid and will be added to the process without dilution as needed.

For each phase, flocculant will be prepared in that phase's corresponding flocculant mix system to produce a dilute solution with a 0.25 wt. % solution strength. This solution will be further diluted in the corresponding thickener feed well. One flocculant mixing facility will be required per phase, which will service both the concentrate area and tailings thickening.

Lime, as hydrated lime, will be delivered in bulk and will be off-loaded pneumatically into each phase's silo. Lime slurry will then be mixed as a 20% w/w concentration slurry. This lime slurry will be pumped to the points of addition using a closed loop system. Discharge valves on the closed loop will be controlled by pH monitors that will regulate the amount of lime added.

Grinding media will be added to the SAG, ball, and regrind mills as required. Mill charging will be conducted initially through use of manual systems for the SAG, ball and regrind mills, with automatic ball charging systems for the ball mill being install post-startup in order to minimize initial capital costs at startup.

To ensure spill containment, the reagent preparation and storage facilities will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bunded area in order to limit spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire, safety protection, emergency shower and eye wash stations, and Material Safety Data Sheet stations will be provided at the facility.

Each reagent line and addition point will be labeled in accordance with MSHA standards. Operational personnel will receive MSHA training, along with additional training for the safe handling and use of the reagents.

17.2.3.9 Assay & Metallurgical Laboratory

Depending on availability and cost-effectiveness, local sub-contractors (e.g., in Reno, Nevada) will provide assay laboratory services. These sub-contractors will be equipped with the necessary analytical instruments to provide routine analyses for the mine, the concentrator and the environmental monitoring departments.

The most important of these instruments include:

- Atomic absorption spectrophotometer
- X-ray fluorescence spectrometer

The metallurgical laboratory will undertake necessary routine and discretionary test work to monitor metallurgical performance and, more importantly, to monitor and improve process unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, pulp drying, pH meters, and glassware and ancillary laboratory equipment necessary to conduct such test work.

Minimal personnel at site will be required, in order to prepare samples for assay analysis by others.

17.2.3.10 Water Supply

The main plant and the tailings facility for both Phase I and II will each have individual raw, fresh (potable), and process water distribution systems. Fresh water supply will come from the city of Yerington and will be transported through a main pipeline to a central potable water stand pipe.

17.2.3.11 Raw Water Supply System

Raw water will be supplied from raw water well(s) via the raw water tank.

Raw water will be used primarily for the following:

- Process water top-up
- Fire water for emergency use
- Wash down and dust suppression

The raw water tank will be always hold at least 60,000 gallons (227 m^3) of water as a reserve for fire water use only.

17.2.3.12 Fresh Water Supply System

Fresh (potable) water will be supplied to each area from the fresh water pipeline from Yerington, via the potable water tank.

Fresh water will be used primarily for the following:

- Cooling water for mill motors and mill lubrication systems
- Gland service for slurry pumps
- Reagent preparation water
- Potable water supply, including safety showers

The potable water from the fresh water source will be not be required to be treated prior to delivery to various service points, as the supply from Yerington is to be potable.

17.2.3.13 Process Water Supply System

Process water generated in the flotation circuits as concentrate thickener and the Phase I and II tailings thickeners overflow will be reused in the concentrator and filter process circuits via the Phase I and Phase II main water process water tanks.

17.2.3.14 Air Supply

Phase I and Phase II will each have multiple local air service systems will supply air to the following areas:

- Low-pressure air for flotation cells will be provided by air blowers.
- High-pressure drying air and pressing air for the concentrate filter press operation will be provided by dedicated air compressors and receiver tanks.
- High-pressure drying air and pressing air for tailings filter press operations will be provided by dedicated air compressors and receiver tanks.
- Air compressors are also supplied for general plant distribution.
- Instrument air will be prepared from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.2.3.15 Online Sample Analysis

Process control will rely on an on-stream analyzer(s) located within the Open Pit plant, which will be shared between Phase I and Phase II. Analysis for specific metallurgical parameters in the various flotation circuit streams is required for plant optimization. A sufficient number of samples will be taken so that the circuit can be balanced by analytical results and calculations, as required. Samples specifically taken for metallurgical accounting purposes will include rougher flotation circuit feed, final flotation tailings and final concentrate. These samples will be collected on a shift-basis and will be analyzed in the subcontractor's laboratory. In addition, on-stream particle size monitor (to be installed at after the plant is operating to minimize Open Pit Project capital costs) will determine the P_{80} particle size of the primary cyclone overflow and the regrind circuit products to ensure the quality of grind achieved.

17.2.4 Process Manpower

Process plant salaried personnel estimates were developed to provide adequate supervision and technical support for the daily operation of the processing facility. 77 required personnel for the processing facility are estimated, as detailed in Table 17-4.

Table 17-4: Process Plant Salaried Manpower

Description	No. per Crew	No. of Crews	Total
Operations Superintendent	1	1	1
Production General Supervisor	1	1	1
Processing Clerk	1	1	1
Foreman - Shift	1	4	4
Control Room Operator	1	4	4
Crushing Operator	1	4	4
Milling Operator	1	4	4
Flotation Operator	1	4	4
Filtration Operator	1	4	4
Reagents/Services	1	2	2
Concentrate Loading	1	2	2
Clean-Up / Day Crew	8	1	8
Chemist	1	1	1
Senior Assayer	1	2	2
Laboratory Sampler/Assayer	3	2	6
Senior Metallurgist	1	1	1
Metallurgist	1	2	2
Maintenance General Supervisor	1	1	1
Maintenance Planner	1	1	1
Maintenance Supervisor	2	1	2
Tradespersons (Mech/Elec/Instr)	3	2	6
Trade's Assistants	3	1	3

Description	No. per Crew	No. of Crews	Total
Crane Operator	1	1	1
Lead Rigger	1	1	1
Maintenance Clerk	1	1	1
Total			77

Note: The operations listed below were priced as contract operations. The manning below is not included in the above table.

1. Dry Stacking of tails, which will consist of 6 operators on day shift, 7 days per week.
2. Concentrate trucking, which will consist of 1 Operations Supervisor, 2 Transload operators and 2 truck drivers operating day shift, 5 days per week.

17.2.5 Process Plant Control

A combined Phase I and II distributed control system (DCS) will be utilized to provide equipment interlocking, process monitoring and control functions, supervisory control and communication with an expert control system. The DCS will generate production reports and provide real time data and malfunction analysis as well as a log of process upsets. Process alarms and events will be also logged by the DCS.

Operator interface to the DCS will be via PC-based operator workstations located in the following area control rooms:

- Gyratory crusher
- Phase I and II process plants
- Phase I and II filter plants

The plant control rooms will be staffed by trained personnel 24 hours per day.

Operator workstations will be capable of monitoring the entire plant site process operations and be capable of viewing alarms and controlling equipment within the plant. Supervisory workstations will be provided in the process staff offices.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area and wired to each respective area local field instrument junction boxes. Signal trunk cables will connect the field instrument junction boxes to DCS input/output (I/O) cabinets.

Intelligent-type MCCs will be located in electrical rooms throughout the plant. A serial interface to the DCS will facilitate the MCC’s remote operation and monitoring.

For site-wide infrastructure (telephone, internet, security, fire alarm and control system), a fiber optic backbone shall be installed throughout the plant site.

17.2.5.1 Primary Crushing Control System

The control objective of the primary gyratory crushing area will be to provide a crushed product to the coarse stockpile(s) prior to grinding and flotation.

A PC workstation will be installed in the shared Phase I and Phase II primary crushing site to monitor crushing and conveying operations onto the coarse stockpile. Control and monitoring functions will include:

- Plugged chute detection at transfer points.
- Zero speed switches, side travel switches and emergency pull cords.
- Weightometers on selected conveyors to monitor feed rates and quantities.
- Equipment power draw, bearing temperatures and lubrication system status.
- Vendors' instrumentation packages.

The control objective of the coarse ore storage and reclaim area will be to provide a crushed ore delivery buffer and a consistent SAG mill feed.

17.2.5.2 Concentrator

To monitor and control concentrator processes and ancillary operations, three PC workstations will be installed in the process building central control room for both Phase I and Phase II.

The PC workstations will monitor and control the following at a minimum:

- Conveyors (zero speed switches, side travel switches, emergency pull cords and plugged chute detection)
- Grinding mills (mill speed, bearing temperatures, bearing pressures, lubrication systems, clutches, motors and feed rates)
- Cone crusher (speed, bearing temperatures, lubrication systems, motors and feed rates)
- Particle size monitors (for grinding optimization)
- Pump boxes, tanks, and bin levels
- Variable speed pumps
- Cyclone feed density controls
- Thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- Flotation cells (level controls, reagent addition, and airflow rates)
- Regrind mills
- Samplers and x-ray analyzer (for flotation optimization)
- Concentrate pressure filters and load out

- Reagent handling, storage level and distribution systems
- Tailings pressure filters (via DST)
- Water storage, reclamation, and distribution, including tank level automatic control (via radio linked remote I/O)
- Air compressors
- Fuel storage (via radio linked remote I/O)

An automatic sampling system will collect samples from various product streams for online analysis and daily metallurgical balance.

Particle size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits.

A metallurgical online analyzer will be used to monitor the performance of the flotation process to optimize concentrate grade and metal recoveries.

17.2.5.3 Remote Monitoring

Closed circuit television cameras will be installed at various locations throughout the plant, including the primary crushing facilities, the stockpile conveyors' discharge, the stockpile reclaim tunnels, the SAG and ball mill grinding areas, the flotation areas, the regrind areas, the concentrate handling areas and the tailings handling facilities. The cameras will be monitored from the plant control rooms.

Item 18.0 PROJECT INFRASTRUCTURE

18.1 Underground

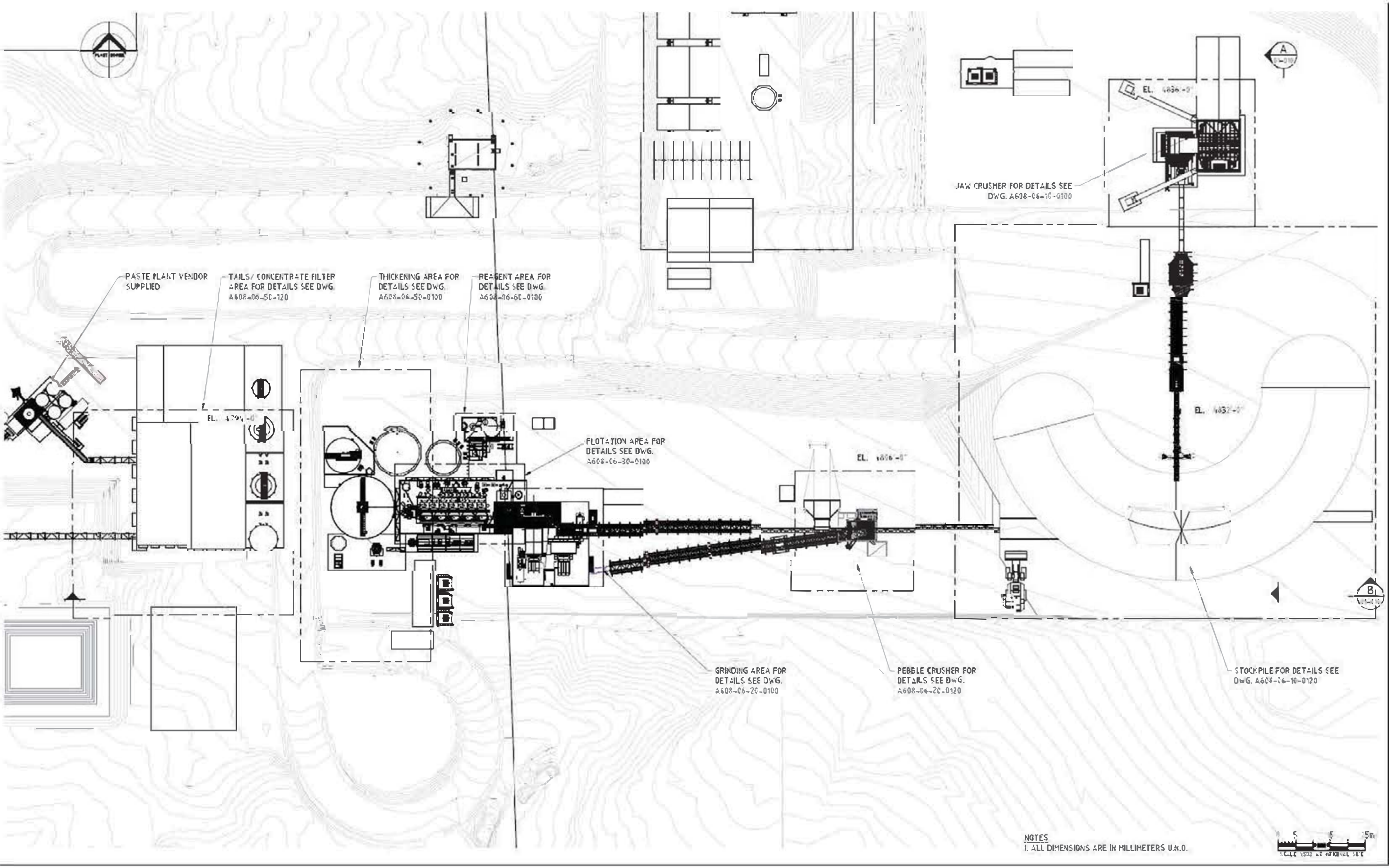
18.1.1 Site Layout & Preparation

This Item provides an overview of the site infrastructure, power supply and distribution, tailings management and hydraulic designs for the Underground Project.

Figure 18-1 shows the general site layout of the underground development. Key aspects of the layout design include:

- Minimization of movement of bulk materials (low grade ore, waste ore and tailings).
- Minimization of interaction between light vehicles (including delivery trucks) and heavy vehicles (moving low grade ore and/or dry stacked tailings).
- Minimization of interaction between pedestrians and vehicles.

Figure 18-1: Underground Project Layout (Sedgman, 2018)



18.1.2 Access Roads

Access to the Property is by a sealed public road network to E Pursel Lane to within 3.5 miles of the proposed mine gate. The sealed portion of E Pursel Lane is a state road. The unsealed portion of E Pursel Lane is a county road for approximately 0.2 miles and then a private road to the proposed mine gate. This existing portion of unsealed road will:

- Be upgraded by placing a base course and two-coat seal for 0.5 miles further to minimize dust impact on the closest resident. The design is per Lyon County Roadway and no topsoil removal is required.
- Be re-graded for the remaining 3.0 miles and left unsealed to the mine gate.
- Require an upgrade of the Little Pumpkin Hollow drainage crossing. This crossing (designed to Lyons County design criteria) will pass the 25-year, 24-hour storm event through a proposed triple 3 ft high by 7 ft wide box culvert. Flows that exceed this storm event will overtop the access road, and may limit site access for short durations. Flows that exceed this storm event will flow over the access road, and may limit site access for short durations. Erosion will be controlled by using rock protection.

The maintenance of the public network will be continued by the relevant authority. The maintenance of the unsealed portion of E Pursel Lane will be carried out as an extension to the maintenance of the internal roads.

18.1.3 Internal Roads & Earthworks

All internal roads, carparks and hardstands will be unsealed. All roads and earthworks were designed based on the topography provided by Nevada Copper. Civil earthworks and grading of all building and facility surface areas will be completed before construction.

Signage will be placed to meet the design requirements; this includes regulatory, preventative and informative signage. Road surfacing will use local materials. The mine property, substations and powder magazines will be fenced. Several gates and cattle guards will be installed and site drainage will be constructed.

Where there is regular traffic, dust will be suppressed by spraying water on the unsealed surfaces with a water truck. During operations, a site maintenance program has been designed to ensure that the roads and other infrastructure remain in their optimal condition.

Costs have been included for power line maintenance, fence and cattle guard maintenance and building and other infrastructure maintenance supplies.

18.1.4 Buildings & Facilities

The following support facilities have been designed and estimated for the Underground Project:

- A visitor and workforce parking area have been provided north of the administration area. Operational parking has been provided south of the administration area, as well as adjacent facilities such as the laboratory and workshops
- An administration complex near the gate, which will be used as a combined:
  - General administration
  - Process facility office
  - First aid room
  - Gatehouse: The administration complex has been located adjacent to the mine gate to avert the need for a gatehouse/site security building. This function will be provided at office reception.

This complex will have a floor area of 144 ft by 60 ft. It is intended to form this complex by relocating the 60 ft by 36 ft and 60 by 60 ft site offices and installing a new 60 ft by 48 ft building. This complex will consist of three separate buildings separated by short elevated walkways connecting the following locations:

- A process plant dry located near the administration complex
- Process plant workshop/store
- Concentrate storage shed, spanning from the concentrate filter station over the truck scale, allowing concentrate storage and loading in a sheltered area
- Truck scale located in the concentrate storage shed
- Sewage treatment plant is located downhill from all the facilities
- Potable water treatment plant with a potable water storage tank
- Fuel facility as a self-bunded tank. Note that lube storage will be provided in a bunded area at each workshop
- Truck wash bay for concentrate trucks departing the Property

It has been assumed DST will be placed by, and concentrate haulage will be done by, contractors with off-site workshops.

The following infrastructure currently exists on the Property and is intended to be kept in situ:

- Mine operations office, a 60 ft by 24 ft pre-engineered building
- The mine warehouse, a 55 ft long by 36 ft wide pre-engineered shed on a cast in situ concrete slab. This warehouse has external racking along both long sides of the shed

- The mine workshop, consisting of 40 ft sea containers (two levels) each side of a 26 ft wide cast in situ concrete slab, which is covered by a steel clad arch structure spanning between the second level containers. A 20 ft long sea container is placed across gap between containers at the western end
- The mine dry, a 40 ft by 36 ft complex consisting of a pre-engineered building either side of a cast in-situ concrete slab, which is covered by a steel clad arch structure spanning between the buildings. The dry has showers, toilets, basins and lockers
- The hoist house, which will be expanded to accommodate the installation of the service and Mary Anne hoists
- Explosive storage compound located 2,000 ft south of the shaft. This compound will be left for other purposes and a new compound built further south to provide the necessary exclusion zone to surface infrastructure

18.1.5 Waste & Water Management

A sewage treatment plant, meeting City of Yerington standards, will be supplied by a suitably qualified contractor and installed downhill from the mining and processing areas. Sewage will be collected at all main working areas and will gravitate to the treatment plant. The plant size will suitable to accommodate 200 equivalent people (EP).

No water treatment/retention ponds will be required for treated sewage (effluent) disposal. Treated effluent will be pumped to the tailings thickener for disposal as part of paste backfill or dry stacked tailings.

An on-site disposal system with an adsorption area of 3,333 ft^2 has been installed on the Property. This system will be decommissioned for the Underground Project

A policy will be established to minimize usage and maximize recycling of domestic wastes such as paper, aluminum, glass and plastics, through the provision of receptacles throughout the mine site and offices, combined with instructions to all personnel (including cleaning staff) in the use of these facilities. Collection will be undertaken regularly, with separated materials transferred to a secure, central storage facility on site for consolidation and onward transfer to re-processors. In general, dangerous waste will be collected and stored briefly at the point of generation, before being transferred to the central storage facility. Those materials that can be rendered inert will be treated at the site and transported to a suitable nearby landfill.

Fresh water supply is sourced from dewatering wells. Early in the mine life, dewatering well WW-01 will provide this demand, as it has a proven yield well above the required demands. Fresh water will be stored in a fresh water tank for use on site. A future deep bore has been allowed in the mining costs, should the dewatering well WW-01 draw down early in the mine life.

The fire water systems have been designed with guidelines and criteria that focus on the fire loss control by automatic fire protection, manual fire suppression, life safety considerations and control of miscellaneous hazards.

Potable water will be sourced from well WW-01 after treatment through a reverse osmosis treatment plant.

18.1.6 Transportation & Shipping

18.1.6.1 Introduction

This scope included assessing the required infrastructure and associated capital and operating costs.

The Underground PFS considered:

- Trucking from the mine site to an internal market, a transload facility, or a West Coast terminal
- Railing of the concentrate from the transload facility to West Coast terminals
- Ocean transportation from the West Coast terminals to the potential market in East Asia
- The estimated total transportation costs per short ton from mine to market

Annual volumes of copper concentrate will be approximately 114,117 wst per year, at full production of the Underground Project, as currently proposed.

Nevada Copper reviewed a variety of methods to transport the concentrate from Yerington, Nevada, to the final customer, including:

- Transport via dump trucks to U.S. smelters or ports
- Rail transport to U.S. smelters and ports
- Ocean container transport direct from mine to ports in Asia

After careful review of shipment options, Nevada Copper determined that rail transport from a transloading facility close to Yerington, Nevada, to export ports and U.S. smelters will provide Nevada Copper with the most stable, reliable and cost efficient form of transportation.

Nevada Copper reviewed three potential destinations for the Pumpkin Hollow concentrate:

- Export to Asia, through West Coast terminals:
  - Vancouver, Washington
  - Oakland, or Stockton, California
- Magna, Utah (the Kennecott Utah Copper smelter owned by Rio Tinto)
- Hayden, Arizona (the Hayden smelter owned by Grupo Mexico)

18.1.6.2 Trucking from Site

Concentrate will be stockpiled under the concentrate filter within the concentrate storage shed. A front end loader will then load 42 st trucks. Each truck will be parked on a truck scale also located in the concentrate storage shed. Once loaded, the trucks will exit the mine site via a wheel wash.

For transport of concentrate to an internal market and West Coast terminals, the concentrate will then be trucked to a proposed transload facility at an existing railway line. The distance to the closest railway line is about 11 miles to the north of the mine site at Wabuska, Nevada, where UP Railroad has a branch line.

18.1.6.3 Transload Facility

For the bulk handling option:

- A new transload facility would need to be constructed at Wabuska along the UP line.
- Trucks would deliver the concentrate to the facility, where it would be unloaded, temporarily stored in a 4,400 st under cover stockpile, and re-loaded into railcars by a front end loader for transport to an internal market or West Coast terminal by 110 st gondola railcars. UP has advised rail cars are limited to 268,000 lbs including tare weight. This study has assumed loads range between 195,000 to 198,000 lbs (97.5 to 99.0 st).
- A preliminary capital cost to develop facilities to receive, store and load out copper concentrate from Nevada Copper has been completed and amortized into the trucking rate
- Sufficient railway tracks exists at the Wabuska transload facility to hold the required railcar fleet, assuming bi-weekly deliveries, as discussed with UP.

The cost to truck and transload the wet concentrate has been costed at $16.41/st of wet concentrate.

18.1.6.4 Rail Transport

At full production from the Underground Project, Nevada Copper will produce on average approximately 2,195 wst of material weekly. Nevada Copper estimates a requirement of approximately 70 railcars to manage a continuous flow of material. Final cars required will depend on the ultimate destination of the concentrate. The production rates of concentrate are not large enough to provide for unit trains from the mine to the final destination, and hence Nevada Copper views a long-term contractual arrangement with UP as the most efficient method of managing the product flows vis-à-vis railcar leasing.

The rates below are based on advice from Concord after discussions with the UP Industrial Products group and have been used in this study based on the following assumptions:

- A Yerington-area siding being utilized to load railcars.
- Concentrates moving in railroad-owned open-top gondolas. If moved in shipper owned/leased cars, a discount could be expected in these rates, but has not been assumed in the Underground PFS.

- Final rates would be negotiated into a long-term contract with UP. Multi-year rail contracts include an annual increase. Typical annual increase rate would be between 3% and 4%.

Rail freights to the above destinations are as below:

- \$25.20/st Yerington – Magna, Utah (Approx. 552 rail miles)
- \$44.10/st Yerington – Vancouver, Washington (Approx. 857 rail miles)
- \$26.10/st Yerington – Oakland or Stockton, California (Approx. 323 rail miles)
- \$50.40/st Yerington – Hayden, Arizona (Approx. 1,149 rail miles)

In addition to base rail rates quoted, additional charges can be incurred when booking rail freight. As outlined below:

- Rail fuel surcharge: UP cars are subject to a mileage-based fuel surcharge, which is adjusted monthly. Its fuel calculation is based on the monthly Department of Energy On-Highway Diesel Fuel Price (U.S. average). For every \$0.05 increase above \$2.30 per gallon, UP will assess a \$0.01 per mile surcharge. Assuming a fuel price of \$2.50 per gallon, fuel surcharge to Vancouver, Washington, is \$34.28 per railcar or \$0.34/st.
- Covers for railcars: Concentrates move routinely in the southwest in open gondolas, due to favorable dry climate. Cars moving outside of that region will need covers to keep moisture limit down on the concentrates. Additionally, any cars moving into any U.S. West Coast port (Vancouver/Stockton/Oakland/ and so forth) will require covers due to environmental regulations to prevent dusting of concentrate while the material travels from mine to port. Cars moving in/out, loaded/empty, will all require covers. Ecofab covers are proposed, which can be leased for \$200 per cover per month (plus repositioning cost of equipment upon initiation and termination of lease period. For this study, we assume a cost of \$1.81/st.

Before railcars leave the loading facility, they must be entered into the UP system with a loaded weight. The railroad will not be responsible for weights, nor will they provide weighing. In-transit scales exist only to verify that railcar weights are within limits for the rail line as well as confirming railcar loads are in-balance, but are not proposed for contract payment purposes.

Railcar weights will be established using front end loaders with built-in scales. Each bucket of material is weighed as it is picked up, then dumped into the railcar.

Although the Underground PFS assumes Nevada Copper directly contracting with UP, Nevada Copper will consider railcar leasing as a long-term method to secure open-top gondolas for the development.

Preliminary indications of the destinations that were reviewed would indicate a maximum need of about 25 cars for bi-weekly delivery. There are several variables that will affect this:

- The actual quantity moving to each destination at any one time. Example: Stockton is much closer than Hayden; therefore, for the same tonnage, Stockton would require less cars as they will perform round trips more quickly.
- Tonnage loading into each car. See below for more details on railcar loading.

Pumpkin Hollow concentrate would not move in unit trains, due to weekly production rates. A unit trains consist of approximately 70 railcars, which would entail storage of over three weeks' worth of production prior to shipment. As such, railcars would steadily move in and out of the Wabuska transload facility two or three times per week, depending upon railroad schedule and actual volume moving out of the loading facility.

Without guarantee until a contract is signed, UP's Industrial Products group can provide a fleet of UP-owned gondolas, which should be adequate for the Underground Project. Concord reviewed the potential lease terms for railcars and based on a three-year lease, current gondola lease rates are $400 per car, per month. Some key items in a rail car lease include:

- Railcars have a maximum number of miles allowed under the lease. For example, a leased car might have a maximum of 40,000 miles (empty and loaded) per calendar year allowed, before a charge of $0.05 per mile for each additional mile is charged. No allowance has been added given this is less than the expected mileage per railcar.
- The leasing company seeks to provide equipment from nearest location to where the lease commences, but typically the freight expense to relocate cars to loading facility on initiation and termination of the lease is for lessee's account will be charged.
- Basic maintenance items are for car lessor's account. When a railcar goes "bad order" and is shifted into a railroad maintenance facility, all repairs are invoiced to car lessor. Repair charges outside of "basic maintenance" scope will be rebilled to lessee.
- Lessee must maintain commercial general liability insurance to amount specified by lessor.

Once material arrives at a port destination, the concentrate will be unloaded out of the railcar and into either an ocean-going vessel or a lined container. Concentrate moving via Oakland would be loaded into lined containers, while material destined for the ports of Stockton, or Vancouver, would be loaded into a bulk vessel. Any concentrate moving to Kennecott or Grupo Mexico's Hayden smelter would not incur a handling charge, as those expenses are borne by the smelter.

18.1.6.5 Port of Vancouver, Washington

Kinder Morgan has been handling U.S.-produced concentrate via its Vancouver, Washington, terminal for several years, and consequently represents the best option for the establishment of rates to East Asia for the Pumpkin Hollow material. Kinder Morgan's terminal is fully permitted, and currently handles a baseload of material from KGHM's Robinson mine, as well as other smaller mines.

Based on discussions with Kinder Morgan, Concord has estimated $25.88/tonne ($23.49/st) to handle the Pumpkin Hollow concentrate for export. This rate includes the following services:

- Unloading railcars and moving concentrate to berth for loading ($23.68/tonne).
- Storage during accumulation of concentrate included in the above rate.
- Environmental fees assessed by the Port of Vancouver ($2.21/tonne).

These costs include conveyor cleaning and warehouse cleaning.

18.1.6.6 Port of Oakland

Oakland has only recently begun to handle concentrate, and only has the capability to load lined ocean containers at the time of this Report.

Container loading charges in Oakland have been assessed at $46.80/tonne ($42.47/st) by Concord after discussions with Central Valley Ag (www.cv-ag.com), which is currently loading containers for another copper mine in Nevada. That rate includes the following services:

- Receiving of open top gondola railcars.
- Railcar unloading and cleaning.
- Ocean container loading including plastic lining and dunnage.
- Drayage to Oakland terminal.
- Documentation and inventory recording.

The rates assume containers arrive in Oakland evenly spread and are returned immediately to the container yard for export (no storage).

While Central Valley Ag, the operator at Oakland, has received clearance to handle copper concentrate, there remains some doubt as to the longevity of that clearance. Nevada Copper are not assuming use of this terminal in this study.

18.1.6.7 Port of Stockton

Alternatively, Central Valley Ag is currently implementing a plan to load concentrate via Stockton, California, but currently permitting is not in place for that facility.

After discussions with Central Valley Ag, Concord has estimated $40.50/tonne ($36.75/st) to handle the Pumpkin Hollow material. Central Valley Ag is currently in the process of obtaining approvals and permitting to bulk load concentrates out of Stockton. Initial timelines target readiness around Q4 2017 although an exact date has not been confirmed and it is expected that this timeline may slip.

The estimated rate includes of the following services:

- Receiving of open top gondola railcars
- Railcar unloading and cleaning
- Transfer to storage facility
- Bulk loading (once required loading minimum is reached – approximately 5,000 to 10,000/wmt)
- Inventory management and warehouse documentation

The rate is also subject to change once the permitting and bulk handling infrastructure are in place.

Central Valley Ag believes it should receive required permits within a 12- to 18-month timeframe, but this is not guaranteed. Nevada Copper is not assuming use of this terminal in this analysis.

18.1.6.8 Ocean Freight

Bulk freights have shown stability over the last two to three years, however, they could be subject to heightened volatility if charter rates were to increase significantly. If the decision is made to export material through the port of Vancouver or Stockton, Nevada Copper could explore a long-term contract to eliminate any volatility risks around the below rates (Table 18-1).

Table 18-1: Ocean Freight Rates

Destination	Ex Stockton or Vancouver ($ per wet short ton of concentrate)
Busan	22.87
Tokyo	22.87
Tuticorin	36.75
Shanghai	24.50
LYG	24.50

These rates reflect current market conditions for shipment of 10,000 wmt during Q2 2017. An average of $24.50/st of wet concentrate has been assumed for the Underground PFS.

18.1.6.9 Summary

Table 18-2 below provides a summary of freight rates.

Table 18-2: Freight Rate Summary

Destination	Freight Rate - Rail and Shipping ($/st wet concentrate)
Magna, UT	27.36
Hayden, AZ	52.56
Oakland, CA	95.23
Stockton, CA	89.51
Vancouver, WA	94.25

It is proposed to adopt $68.06/st wet concentrate, or $75.00/wmt for railing and shipping as an average of different markets, as described in Item 19.0

Given the concentrate will be tarped in trucks and covered in the gondolas, no loss of concentrate is assumed in the freight from the Pumpkin Hollow Project to market.

18.1.7 Power Supply, Substations & Main Distribution Lines

A field assessment and desktop study were executed by Power Engineers, Inc. for the power supply to the Pumpkin Hollow property from the NV Energy electrical grid. In addition to the field assessment and desktop study, Power Engineers provided cost estimates for the temporary power supply for the shaft sinking. The substation to the north of the shaft will supply power for the underground and the surface facilities at the Underground Project and associated transmission lines were estimated by Tetra Tech.

18.1.7.1 Power Supply

A new 120 kV transmission line will be constructed from a service point on the existing NV Energy System to the proposed mine site. Nevada Energy Systems has proposed an upgrade through private property, which has been accepted by Nevada Copper.

The new 120 kV transmission line will be routed and terminated at the substation to the north of the current shaft location and will form the basis of the power supply to the underground development.

To develop the site a temporary 25 kV power feed has been provided to the Main Shaft at 6.5 MW for shaft sinking and mine development; this power supply will be cut over to the main distribution network during the execution phase of the Underground Project.

18.1.7.2 Substation

There is a 120 kV switchyard planned for the facility. The main substation will have an incoming 120 kV source serving a 30 MVA power transformer. The voltage will be stepped down to a utilization voltage for distribution at 4.16 kV, with step-up to 13.8 kV as required for the underground distribution requirements.

The facility will be a fenced compound, and will include the transformer, outdoor 120 kV switchgear, neutral earthing resistor, switchboards demountable switch room, substation services distribution transformer and remote operation panels. The meter is proposed to be located in this switchyard.

18.1.7.3 On-Site Distribution

Power will be fed to:

- The surface process plant switchgear to supply the:
  - SAG, ball and regrind mills.
  - Flotation and plant essential services substation including north fan shaft MCC.
  - Concentrate and tailings substation including south fan shaft MCC.
- Standby generation will be provided with supply change-over facility at the south fan shaft to allow emergency operation of underground essential services (ventilation, personnel hoists, communications, and so forth). Generators and fuel storage will be sized to provide minimum six hours' emergency operation to permit safe egress from the underground workings.
- The underground mining substation to supply the:
  - The hoist house.
  - ES and EN ventilation shaft substations.
  - East shaft 1 substation.
  - East shaft 2 loads.
  - Low voltage (LV) services substation.
  - Crushing substation.

The on-site distribution of 13.8 kV will be run underground in cables to feed the substations/loads.

18.1.7.4 Existing Shaft Power Distribution

An existing 25 kV overhead line was brought in from the existing NV Energy grid originally for shaft sinking and development of the Underground Project. This line will serve as the source of construction power. Once the 120 kV overhead line is complete, the 25 kV overhead line and 6.5 MVA 25/4.16 kV transformer will be disconnected and an alternative power supply fed from a 30 MVA 120/4.16 kV transformer will be used to supply the site.

18.1.7.5 Underground Distribution

Power distribution underground will be via 13.8 kV cables to substations located strategically underground. The high voltage distribution system will be used to minimize the required cable size. Two localized systems will be used to step down voltage to 480 V, which is required for most equipment. Where a large number of loads are needed in a small area, a 2,500 KVA/480 V power center will feed distribution boxes for equipment. To provide power to more remote equipment or where mobility is needed, a 1,750 KVA/480 V skid-mounted power center will be used.

18.1.8 Health, Safety & Security

18.1.8.1 Industrial Hygiene

Nevada Copper has overall responsibility for the development of Health, Safety and Environmental (HS&E) and Security policy and its implementation. Nevada Copper's team will produce a Health, Safety and Environmental Plan detailing the requirements for HS&E in full compliance with the laws and regulations of Nevada. All contractors, subcontractors, suppliers, vendors and consultants will comply with these requirements. HS&E costs have been estimated by accounting costs for industrial hygiene, security, first aid and safety training. In addition, appropriate staff will be hired to administer HS&E.

18.1.8.2 Security

Nevada Copper will manage site-wide security. Security services were implemented at the start of construction. The development will have a single guarded entry point at the mine gate adjacent to the proposed administration building. There will be security guards during construction, the mine gate has been located adjacent reception so that both reception and security roles can be undertaken from the one location when the mine goes into production.

18.1.8.3 First Aid

Because of the close proximity of the town of Yerington to the Underground Project, there will be no fire station or trauma room provided on site. Nevada Copper will have sufficient emergency medical technician (EMT) trained personnel and mine rescue team on site to respond in medical emergency situation. To respond to first aid medical situations, Nevada Copper will have first aid stations, rescue equipment and training supplies/equipment readily accessible and strategically located around the mine site.

18.1.8.4 Training

Nevada Copper will use various types of training to meet both safety and performance needs. All employees receive ethics, safety and environmental training following hire and before beginning work. The training meets the requirements of the MSHA. In addition, employees are provided safety task training prior to operating company equipment. This training is performed by experienced trainers, or experienced operators. This training is supplemented by skill or reliability training to enhance the skill and ability of operators and provide for a more skilled and cross-trained workforce.

In the cost estimate, a week of training per year has been included for all staff and personnel on payroll. This includes the cost for training of new hires due to turnover. Other training costs are allowance for training supplies and classes.

18.1.9 General & Administrative

For this Underground PFS, the G&A expenses have been subdivided into:

- Personnel
- Miscellaneous building and facilities supplies
- Sustainable development
- Community and public relations
- Outside consultants/services
- Other owner costs

During construction, G&A personnel and expenses are treated as indirect cost under initial capital expenses. When production of the copper ore commences, the yearly occurring cost will become part of the yearly G&A expenses.

Staff is defined as all positions that will be shared amongst both the process facilities and the mining operation. Salaries have been determined using the Nevada Mining Association's 2016 Annual Wage and Salary Survey, as provided by Nevada Copper. These estimates are still considered reasonable by Nevada Copper. A total of 27 G&A personnel will be hired to manage the mine operations and the process facility. Table 18-3 shows the G&A staff breakdown.

Table 18-3: Administration Manning

Description	No. per Crew	No. of Crews	Total
General Manager	1	1	1
General Manager's Assistant	1	1	1
Mine Manager	1	1	1
Plant Manager	1	1	1
Operational Clerk	1	1	1
IT Services Manager	1	1	1
Safety Manager	1	1	1
H&S Clerk	1	1	1
Human Resources Manager	1	1	1
HR Clerk	1	1	1
Training Manager/Industrial Hygiene	1	1	1
Environmental & Community Relations Manager	1	1	1
Environmental Engineer	1	1	1
Environmental Technician	1	1	1
Site Services General Foreman (Leadman)	1	1	1
Site Yard Laborer	1	1	1
Front Desk & Gate Security	4	1	4
Administrative Manager/ controller	1	1	1
Payroll (Payroll Clerk)	1	1	1
Accounts Payable	1	1	1
Accounting Clerk	1	1	1
Warehouse Manager (Warehouse Supervisor)	1	1	1
Purchasing Agent	1	1	1
Warehouse Shipper/Receiver	1	1	1
Total			27

Overhead costs have been allowed for items such as:

- Health, Safety & Security (Supplies), First Aid, HS&E Training Supplies and Training.
- Building and Facilities: Supplies included in the cost estimate for running the administrative offices and buildings at the site are, amongst others, annual phone and internet cost, courier and postage, copying and printing, software licenses, computer and copier upgrades, sanitary facility supplies.
- Environmental Service & Supplies and Community/Public Relations: Nevada Copper's program is part of its commitment to doing global business with a focus on stewardship regarding sustainability. The efforts in social responsibility are consistent with, and supportive of, Nevada Copper's long-term objectives. The expenses planned to spend on community relations are charitable contributions, supplies, sponsorships, and other contributions.

- Outside Consultants/Services: Consultant services included in the yearly G&A cost are legal services, outside environmental services, internal audits, and other consulting services (tax, accounting, management).

Owners Cost incurred during the construction of the Underground Project is included as indirect initial capital cost. The estimate includes provision for land, site works, management, licenses, land taxes, fees, other government charges, and so forth.

18.1.9.1 Tailings Management

Permitting for the Underground Project is based on DST. This is due to previous work that found reduced limits of disturbance resulting in reduced environmental footprint and increased water efficiency. Conventional slurry deposition and thickened tailings deposition were not considered in this study on that basis.

The filtered tailings management option involves dewatering of the process tailings from the flotation circuit using pressure filters. The moisture content (by dry mass) of the filtered tailings is expected to at or below 15%. The filtered tailings are conveyed to an engineered tailings storage facility via mechanical belt conveyors, a radial stacker and then truck placement.

Ore from the underground mining operations from the East and E2 Deposits will be processed in a dedicated underground mine process facility. The slurry tailings from the flotation circuit will be dewatered in a filtration facility. The filtered tailings will be conveyed and trucked to the DST facility located entirely within the private patented land boundary. The operations are expected to generate tailings at a rate of approximately 4,720 stpd (dry) with a portion of the tailings being used as paste backfill for the stopes. The DST facility is designed to store approximately 12 Mt (dry) of filtered tailings over the LOM.

18.1.9.2 Tailings & Process Water Containment

The engineering containment features for the DST facility include a low permeability synthetic layer designed as a seepage barrier early in the life of the stack. Depending on the results of trial dry stacking at the start of operations, this synthetic layer may or may not be placed under the entire dry stack. No finger drains are proposed in the stack to restrict build-up of hydraulic head within the DST facility. Any entrained pore water within the tailings has been assessed in the stability analysis based on the test work on tailings characteristics. The unsaturated conditions expected within the DST facility from proposed placement methods and self-draining and relatively deep groundwater conditions at the site provide an effective combination to result in a stable stack.

18.1.9.3 Engineering Analysis

The following engineering analyses were conducted as part of the geotechnical design of the DST facility:

- Slope stability
- Liquefaction susceptibility

- Seepage analysis
- Trafficability assessment

The above engineering analyses were based on limited laboratory characterization of tailings samples produced from bench scale flotation tests. The results of the engineering analyses indicate that the proposed DST facility will be resistant to liquefaction due to the unsaturated nature of the tailings and possess adequate factors of safety to ensure overall slope stability under both static and seismic loading conditions.

The results of the seepage indicate minimal flux can be expected at the base of the DST facility. The underground mine tailings will be in an unsaturated condition at the expected placement moisture content of 15% (by dry weight) and therefore should trafficable by trucks after compaction.

18.1.10 Materials Handling Systems & Infrastructure

18.1.10.1 Overview

Being a shaft-accessed underground mine, the Underground Project will use the following processes to handle material underground:

- Material segregation via geological grade control
- Remuck bays for temporary storage of material
- Ore passes for transporting ore to lower levels when feasible. Ore passes will be fitted with an 18-inch by 18-inch grizzly at the top to prevent oversize material from entering and potentially blocking the ore pass system
- Loading and transport of material via LHD scoops and trucks
- COBs for storage of material to be lifted out of the mine. COBs will be fitted with an 18-inch by 18-inch grizzly at the top to prevent oversize material that is directly tipped to the COB by a truck or that passed the ore pass grizzlies, from entering the materials handling system.
- Loadout conveyor and skip loading system to take material from the COBs and place it in the skips
- Two skip hoisting
- Surface handling of material that is discharged from the skips to different stockpiles and the mill for processing

18.1.10.2 Material Segregation

High grade ore, low grade ore and waste materials will be separated based on grade control processes implemented by the technical services staff once the mine is producing. The material will be stockpiled in a designated level storage location, ore pass, directly into a COB (if ore), or a combination. The key criteria for the materials handling will be to minimize dilution and to high grade material, in particular during initial start-up production.

18.1.10.3 Loading & Transportation

Material will be loaded using LHD machines and underground mine trucks, which will be of the dimension and size appropriate for the mine plan (further discussion on LHDs and trucks can be found in Item 16). Ore material will be transported via trucks and/or ore passes to the COBs for storage before being hoisted out of the mine. Waste material will initially be hoisted out of the mine, until there is enough room in secondary stopes to place all waste material as backfill.

18.1.10.4 Ore Passes & Remuck Bays

Where feasible, ore passes will be used to move ore from upper to lower levels for loadout via truck chutes to the COBs. Remuck bays will be used for rehandling and storage of ore and waste material, as required.

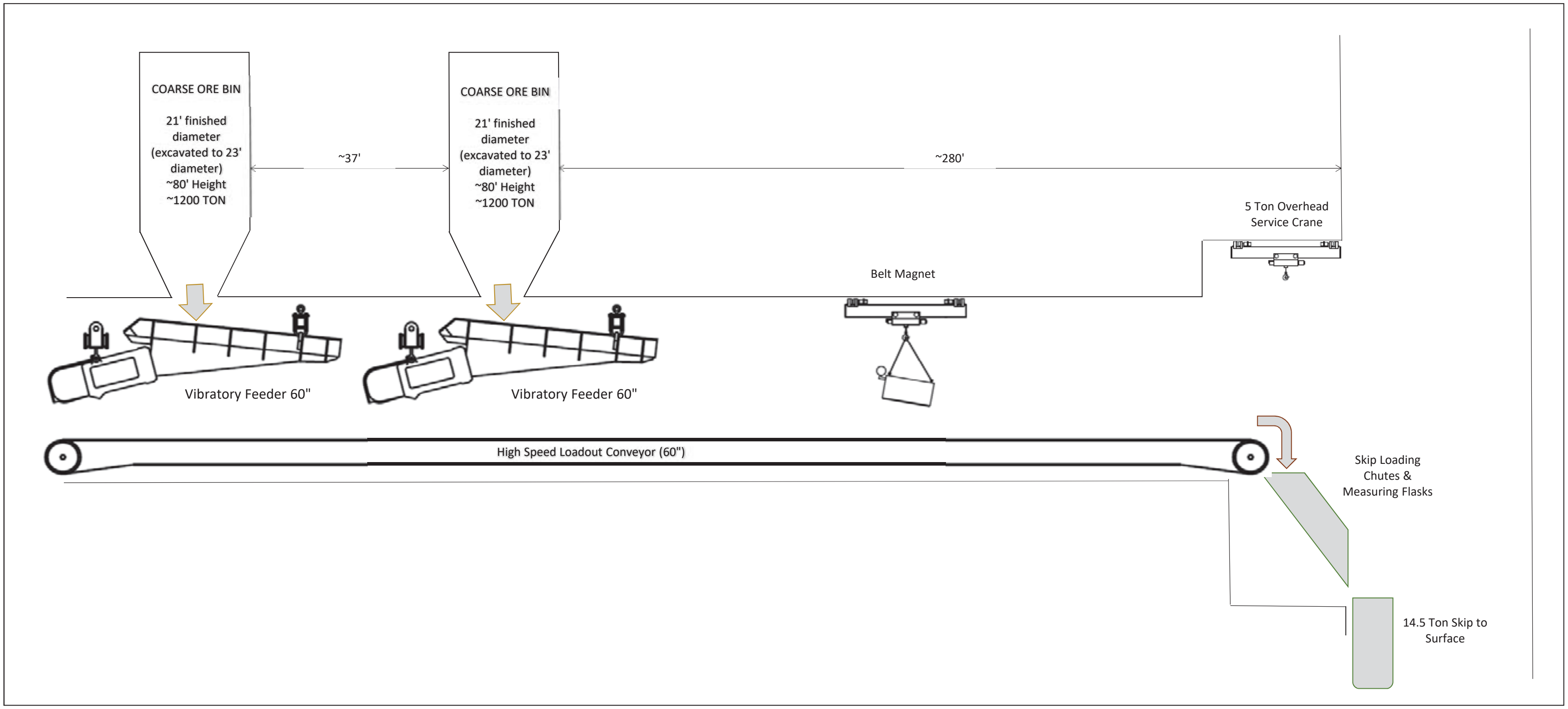
18.1.10.5 Coarse Ore Bins

Two COBs will be constructed in parallel above the loadout conveyor. During initial mine development, one or both of COBs will initially be dedicated to waste material. As ore becomes available, the first COB will become dedicated to ore. The second COB will be predominantly used for ore, and intermittently will be switched to waste on a campaign basis as required. The COBs will have a finished diameter of 21 ft, and an excavation height of 80 ft, for a capacity of approximately 1,200 st in each COB.

18.1.10.6 Loadout Conveyor & Skip Loading System

The COBs will discharge onto vibratory feeders, which will in turn discharge onto a loadout conveyor. The loadout conveyor will feed a movable discharge chute. The chute will allow the conveyor to load one of the two measuring flasks at a time. The measuring flasks will contain the appropriate load for a skip. When a skip arrives, the measuring flask discharge chute will open to fill the skip. Once full, the skip will be hoisted to surface and the discharge chute on the measuring flask will close so that it can be filled again. Figure 18-2 shows the skip loadout system flowsheet.

Figure 18-2: Skip Loadout System Flowsheet (2017 Technical Report)



18.1.10.7 Hoisting

Hoisting will follow specific priorities dependent on the various stages of mine development and production. These stages are listed below:

- Mine development: Initial focus will be to hoist all material out of the mine to make room for required infrastructure. At this stage, it is only expected that waste material will need to be hoisted. The priority for mine development will be to access levels that contain the highest grade ore that can be mined during the initial stope production stage.
- Surface ore stockpiling: Once development begins to pass through mineralized material, hoisting priority will be given to this material, and secondarily to waste material. The goal in this stage will be to stockpile as much high grade and low grade ore material as possible on surface for plant startup.
- Initial stope production: Once high grade stopes have been accessed, the priority will be hoisting as much high grade ore material as possible. Secondary priority will be given to low grade material, and lastly waste material. The majority of waste material generated during this stage will be used to temporarily backfill primary stopes, since the paste fill plant will not be operational at this time.
- Opening of secondary stopes: Once secondary stopes are in production, all waste rock will be directed to these stopes as backfill. Hoisting priority will be given to high grade ore, and secondarily to low grade ore material. At this point, both COBs will be designated for ore.

18.1.10.8 Surface Handling

Once the material has reached surface, it will either be direct fed to the plant, placed on the designated high grade or low grade stockpile, or placed on the existing mine waste rock stockpile.

18.1.11 Other Mine Support Services

18.1.11.1 Mine Dewatering

Mine dewatering is discussed in Item 18.1.14.

18.1.11.2 Power

Electrical power to the underground mine will be delivered from surface via the Main Shaft. Delivery of power to the underground mine will be at 13.8 kV, with surface infrastructure fed at both 4.16 kV and 480 V, as discussed in Item 18.1.7.

18.1.11.3 Compressed Air

Drilling, charging and various work activities for equipment maintenance will require compressed air. Compressed air will be supplied to work areas via compressed air service lines. Compressed air will be sourced from air compressors located on surface.

18.1.11.4 Potable Water

Potable water will be supplied to the mine via pipelines from the on-site treatment plant (as described in Item 18.1.5) to mine shafts.

18.1.11.5 IT & Communications

The mine will have a fiber optic “leaky feeder” system, which will be used to communicate and manage fleet information underground.

18.1.11.6 Escape Ways

As per the Code of Federal Regulations (CFR) Title 30, Mineral Resources, at least two escape ways (means of egress) are required from the lowest level up to surface of the mine. As per these requirements, escape ways are provided via a Mary-Anne hoist installed within each ventilation raise and the Main Shaft, with access available from each level. For levels extending beneath the main ventilation raise (such as in the E2 Zone), a separate escape way raise containing a ladder way system will be installed.

18.1.12 Concurrent Reclamation

The filtered tailings management option adopted for the Underground Project provides the opportunity for concurrent reclamation of the DST facility side slopes. The concurrent reclamation of the DST facility side slopes will:

- Protect the tailings surfaces against erosion
- Reduce the overall contributing watershed for contact process water
- Reduce dust emission

18.1.13 Surface Water Hydrology & Hydraulic Designs

Hydrologic and hydraulic analyses for the Underground Project have been undertaken with further detail outline in Item 20 of this Report. These analyses explain how peak flows and runoff volumes were calculated for management of surface water run-on to the site from adjacent watersheds and on-site stormwater management. Hydrologic rainfall-runoff modeling was developed for the post-development conditions at the site. Peak flows and runoff volumes determined from this modeling were used to design:

- Diversion channels to direct run-on flows away from disturbed areas
- Stormwater management basins
- Secondary containment basins (SCBs)
- Rapid infiltration basins (RIBs)

The mine processing facilities are designed to withstand the effects from a 100-year, 24-hour storm event, and to contain and control the direct precipitation resulting from a 25-year, 24-hour storm event. The water generated during these type of events can be defined as:

- Surface water: water that is generated from an undisturbed watershed upstream of the facilities, which does not come in contact with the facilities or disturbed areas.
- Non-contact water: water that comes into contact with the facilities or disturbed areas, but has not been in contact with the processing facilities or the wet processed ore. Non-contact water also includes runoff generated from small undisturbed, upstream, off-site watershed, which is then transferred to the disturbed facilities area via natural drainage, where impractical to divert as surface water.
- Potential contact water: water that has or may have come into contact with the processing facilities, concrete bunded areas or wet processed ore.

The water generated during a 25-year, 24-hour storm event will be managed using the following methods.

18.1.13.1 Surface Water

Surface water will be diverted using diversion channels, therefore protecting water quality. The existing diversion channel upstream (east) of the facilities is designed to minimize the amount of surface run-on water coming into contact with facilities. A new diversion channel at approximately 4,850 ft will further reduce the surface water run-on by diverting water between the existing mine waste rock stockpile and the process facilities area. A new diversion channel will also be located upstream of the DST pad.

The trapezoidal diversion channels have been sized for the design flows.

18.1.13.2 Non-contact Water

Non-contact water will be drained into and contained in unlined ponds for evaporation and infiltration. These ponds have been designated as RIBs. Local site drainage will direct the water to these basins.

There are currently existing pipelines with three destinations for disposal of non-contact mine dewatering water:

- The first destination consists of four existing RIBs approximately 1.4 miles west of the Main Shaft. These RIBs take piped water from the existing equipped mine dewatering wells. The first two basins operate in parallel to each other, with the remaining two configured in series to the first pair of RIBs.
- The second destination consists of one existing RIB, a lined pond and an irrigated area approximately 2.6 miles west of the Main Shaft. These facilities also take piped water from the existing equipped mine dewatering wells. Water can be diverted to either the RIB or the pond. The pond is equipped with a pump to serve a traveling irrigation spray.

- The third destination consists of four existing ponds approximately 0.6 miles south-southwest of the Main Shaft. All four ponds (called E1 to E4) are in series. The first (Pond E4) is a lined sedimentation pond, the second and third (Ponds E3 and E2) are unlined sedimentation ponds and the fourth (Pond E1) is a RIB. The ponds take piped water from the dewatering of existing mine working.

The following is proposed to manage other non-contact water:

- Two new basins are proposed adjacent to Little Pumpkin Hollow watershed, approximately 2.3 miles west-northwest of the Main Shaft. These basins are sized to dispose of the balance of excess non-contact water. They have been spaced apart to mitigate the height of any mounding under each RIB.
- An additional lined sedimentation pond will be constructed in parallel to Pond E4, to allow each to be dried offline to de-silt by excavator or loader during operations.
- The second and third ponds at the third destination (Ponds E3 and E2) are proposed to be lined.
- Two mine stormwater management basins will be required located adjacent to the processing facilities and the mine waste rock stockpile, respectively.
- Depending on the results of future infiltration investigations, additional irrigation may be constructed if infiltration is limited.

18.1.13.3 Potential Contact Water

A total of two potential contact water ponds or secondary containment ponds will be required:

- One pond will be located next to the processing facilities. This process plant secondary containment pond will have the capacity to hold 110% of the largest process vessel volume within the processing facilities.
- One pond will be located adjacent to the DST pad. The DST drainage channel will surround the DST pad and be designed to contain the volume and flow rate of runoff water produced by the LOM DST surface area.

In each of these locations, synthetic lined drains will capture spills, drainage and/or stormwater runoff, and report the potential contact water into the respective lined secondary containment ponds. The ponds will be double lined with seepage monitoring and control infrastructure provided. Flows into ponds will be managed by either pumping back into the process facilities via sump pumps or left to evaporate.

18.1.13.4 Surface Water Containment Ponds

Table 18-4 depicts the stormwater management basin (SMB) and SCB sizes.

Table 18-4: Water Management Structure Sizes

Pond	Volume (ft3)
Waste Rock SMB	27,174
Process Plant SMB	296,926
Process Plant SCB	43,300
Dry Stack Tailings SCB	443,653

18.1.14 Groundwater Hydrology & Dewatering

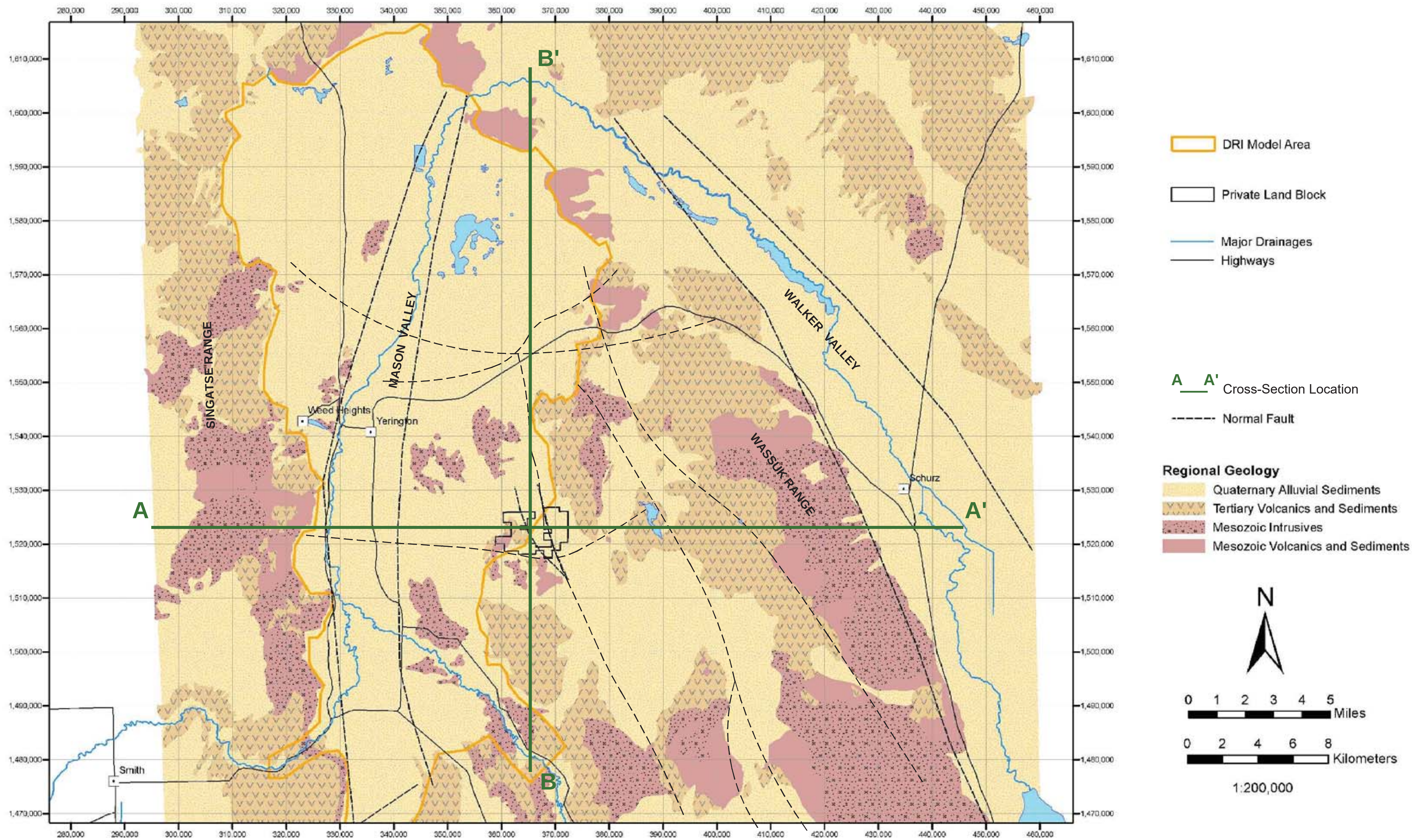
A regional numerical groundwater flow model was constructed in 2012 to estimate inflows to the Underground Project and potential impacts to regional and local water resources. In addition, data and conclusions generated support design of dewatering and water management system and permitting requirements.

The numerical model was based on a compilation of regional, local and site-specific geologic and hydrogeologic data. Primary data sources include USGS, Desert Research Institute (DRI), Nevada Bureau of Mines and Geology (NBMG) and Nevada Copper geologic and mineral resources data. In addition, extensive historical and recent investigations have been conducted to characterize the site hydrogeology.

18.1.14.1 Regional & Site Hydrogeology

Within the Yerington district, the occurrence of groundwater is controlled by the geologic distribution of bedrock and unconsolidated sediments throughout the region (Figure 18-3). The majority (approximately 98%) of the groundwater flow in the region occurs within alluvial aquifers along the Walker River and the adjacent valley floors and alluvial slopes away from the river. Groundwater flow in the alluvium is generally down-valley. Within bedrock, regional flow of groundwater is generally from higher to lower elevations in response to the greater precipitation derived recharge at higher elevations on the ranges. Groundwater within alluvium connects with that in bedrock near the margins of the valleys.

Figure 18-3: Distribution of Bedrock & Unconsolidated Sediments in the Yerington District (2017 Technical Report)



Sources of recharge to the groundwater system include infiltration of a fraction of precipitation and irrigation water. Discharge from the groundwater system occurs primarily through pumping from irrigation wells and evapotranspiration.

Regionally, groundwater flow system boundaries include faults, the Walker River and Walker Lake, and hydrologic divides that separate the hydrographic basins and sub-basins. Within the Underground Project area, a number of subvertical faults and altered clay-rich zones at the margins of some lithologic units compartmentalize flow in the aquifer system.

Groundwater at the site is present mainly in the bedrock; alluvium is saturated only near the west boundary of the site. Groundwater flow is generally toward the north and west in the site, with a vertically downward component of hydraulic gradient occurring at least locally. The bedrock is of generally low hydraulic conductivity except where fractures create secondary permeability and transmit groundwater. Numerous other faults with varying amounts of displacement act as hydrologic boundaries.

Age-dating of groundwater samples from area wells demonstrates that alluvial/Tertiary groundwater is younger in apparent age (11,400 years) than that from Mesozoic bedrock (30,000 years). This, along with permeabilities more than two orders of magnitude greater in alluvium than bedrock suggests that the two groundwater systems act as relatively isolated and separate.

18.1.14.2 Groundwater Model Setup

Groundwater flow models were constructed to simulate steady-state pre-mining conditions, progressive mine development and dewatering, and transient post-mining conditions using the finite-difference model code MODFLOW-SURFACT. The regional model domain consisting of the Mason Valley and the portion of the Walker Valley that lies west of the Walker River (Figure 18-4) was constructed to incorporate the DRI model of Mason Valley to the extent possible.

The model’s hydraulic conductivity distribution was based on a three-dimensional, regional geologic model created from available published data with detailed refinement in the site developed from the site geologic model. The combined geologic model was used to define hydrostratigraphic zones to which hydraulic conductivity values were assigned based on site characterization and model calibration.

Steady-state model calibration included target heads at existing wells at the site and regional locations, and stream flows at gaging stations on the Walker River. Transient targets included drawdown and recovery data from the aquifer pumping test on a dewatering test well sited near the E2 deposit.

Figure 18-4: Model Domain & Model Grid (Tetra Tech, 2019)

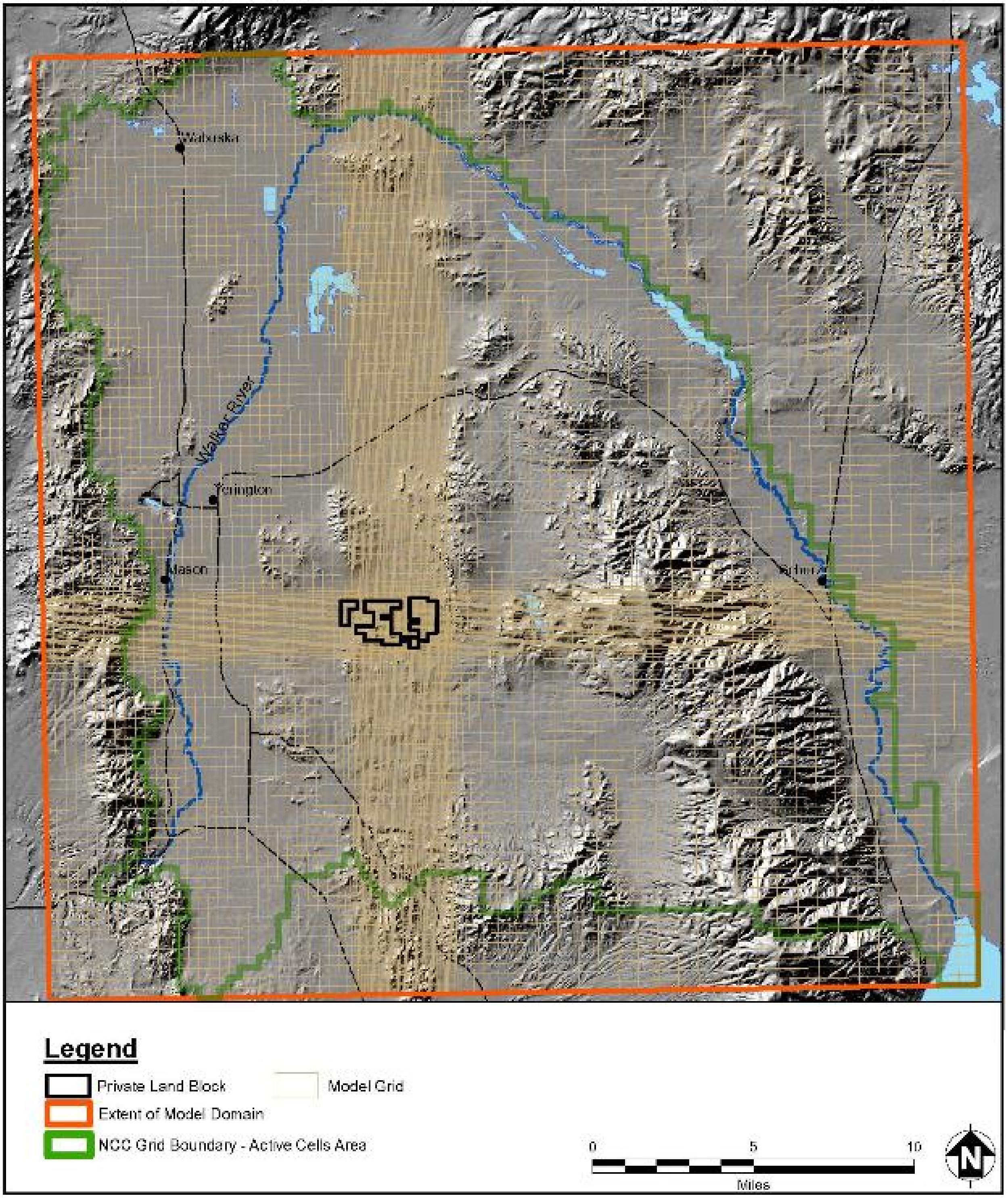
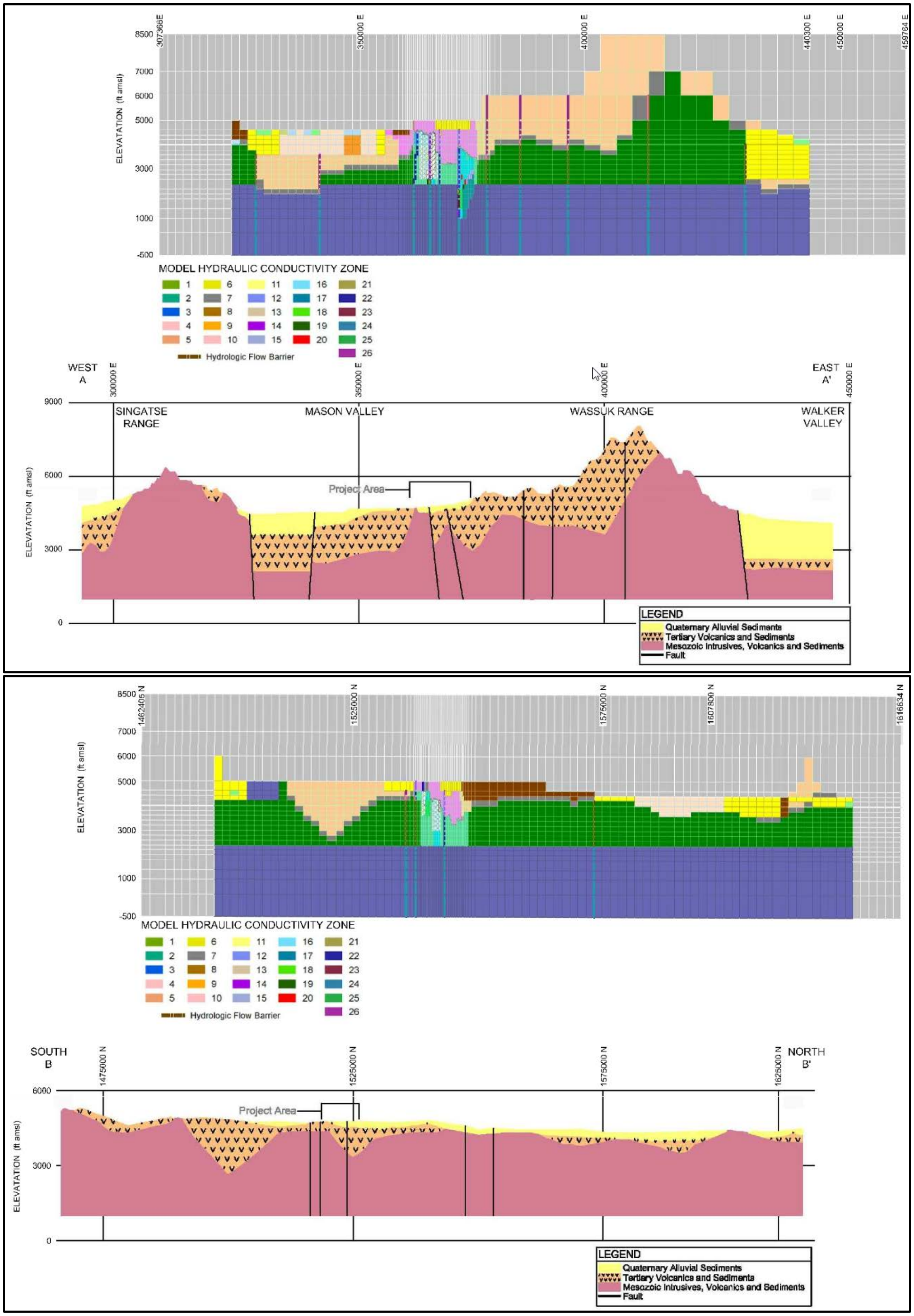


Figure 18-4: Model Domain & Model Grid (Tetra Tech, 2019) (cont.)



18.1.14.3 Mining & Post-mining Simulations

The transient model simulated step-wise mining and dewatering of the Underground Project during mining operations over the LOM. Drain cells for the mine were assigned progressively to simulate mine inflows with the locations and elevations changing as mining progresses, while the mine plan has:

- Lengthened from 12 years (groundwater model assumption in 2012) to 13 years of full production in the current mine plan
- Decreased Mineral Reserve estimates in the East Deposit from approximately 27 Mst in 2012 modeling to approximately 24 Mst currently
- Increased Mineral Reserve estimates in the E2 Deposit from 0 Mst in 2012 modeling to approximately 6.1 Mst currently. This E2 Deposit is currently planned to be mined after two years of mining in the East Deposit
- A higher peak dewatering rate of 3,000 gpm has been assumed for this PFS to address uncertainty with these changes. This groundwater modeling should be updated in the feasibility study
- Post-mining conditions were simulated for the 2012 underground mine plan by modifying the numerical model to accommodate post-closure conditions. A simulation time of 1,000 years was used for the post-mining model, as steady-state is expected to occur within that time.

The groundwater model suggests that dewatering will be necessary for the underground mining operations. Groundwater inflow to the Eastern Area is predicted to reach a maximum of about 2,000 gpm at the start of mining activities. As noted above, the design of infrastructure is based on 3,000 gpm. Inflow to the underground mine will then decrease gradually during the remainder of the mine life. Temporary lowering of the water table will occur in the vicinity of the site as a result of the mine dewatering activities. By the end of the mining period, drawdown of 10 ft or more will extend approximately 0.3 mile beyond the boundary of the patented claims.

Following completion of mining, the water table will begin to recover from the drawdown created by mine dewatering. At the end of the mining period, drawdowns of 10 ft will extend approximately 1,600 ft north of the site but not beyond the westernmost or southernmost boundaries of the site. After mining has ended, groundwater levels will begin to recover toward their pre-mining elevations. Within approximately 20 years, residual drawdown of 10 ft or more will be limited to within 1,200 ft of the site. The alluvial aquifer in the Mason Valley is not predicted to experience drawdowns that would be distinguishable from normal seasonal water table fluctuations. Stream flows in the Walker River are not predicted to decrease as a result of the mining activities.

18.1.14.4 Mine Dewatering

Dewatering of the Underground Project is anticipated to be through passive collection of water in strategically placed sumps within the underground mine with the assistance of pumping from dewatering wells located near the East Deposit. Water collected from the underground mine development and stoping will be lifted to surface in pipelines via the East Main Shaft and the E2 ventilation shaft. Water from dewatering will be directed to the mill for use as make-up. Excess water will be directed by pipelines to either irrigated pastures on lands owned or utilized by Nevada Copper, three east water management basins (WMBs), four north WMBs, one Ranch WMB, or to additional WMBs as needed for re-infiltration.

The mine dewatering system will require modification and refinement as empirical data become available during advanced exploration and initial mine construction and operation. In particular, mine inflow estimates will be refined based on numerical model updates incorporating drawdowns observed during additional aquifer testing planned for the Main Shaft, and from other additional testing. Long-term operation of the pumping wells has been concurrent with shaft sinking and initial mining.

18.1.14.5 Site-Wide Water Balance

The Pumpkin Hollow site-wide water balance model was created with a primary objective of predicting the volume of water that might need to be delivered from the city of Yerington over the entire mine life. A secondary objective is to provide estimates of the WMB designs. The site-wide water balance was modeled using the GoldSim (www.goldsim.com) platform to address the primary and secondary objectives (discussed above) during the operation phase.

Previous modeling shows that a maximum yearly quantity of 23 acre-feet (ac-ft) of water was needed for the underground operation. This amount assumes a typical (mean) condition and is well below the contractually agreed amount of 3,500 ac-ft/year.

While this allocation is available, make-up water to site will be provided from dewatering, using decant water from the east WMBs as process water make-up and water from dewatering and water supply wells on-site for fresh water make-up.

The results of the site-wide water balance model indicate that the pastures and WMBs are more than adequate to sufficiently use or infiltrate the total amount of unused dewatering water for all scenarios. However, there is sufficient land area for additional infiltration basins and all basins will be conservatively sized to infiltrate the maximum dewatering volume, allowing for some of this water to the diverted to average plant make-up demands. This water balance modeling should be updated in the next phase of works.

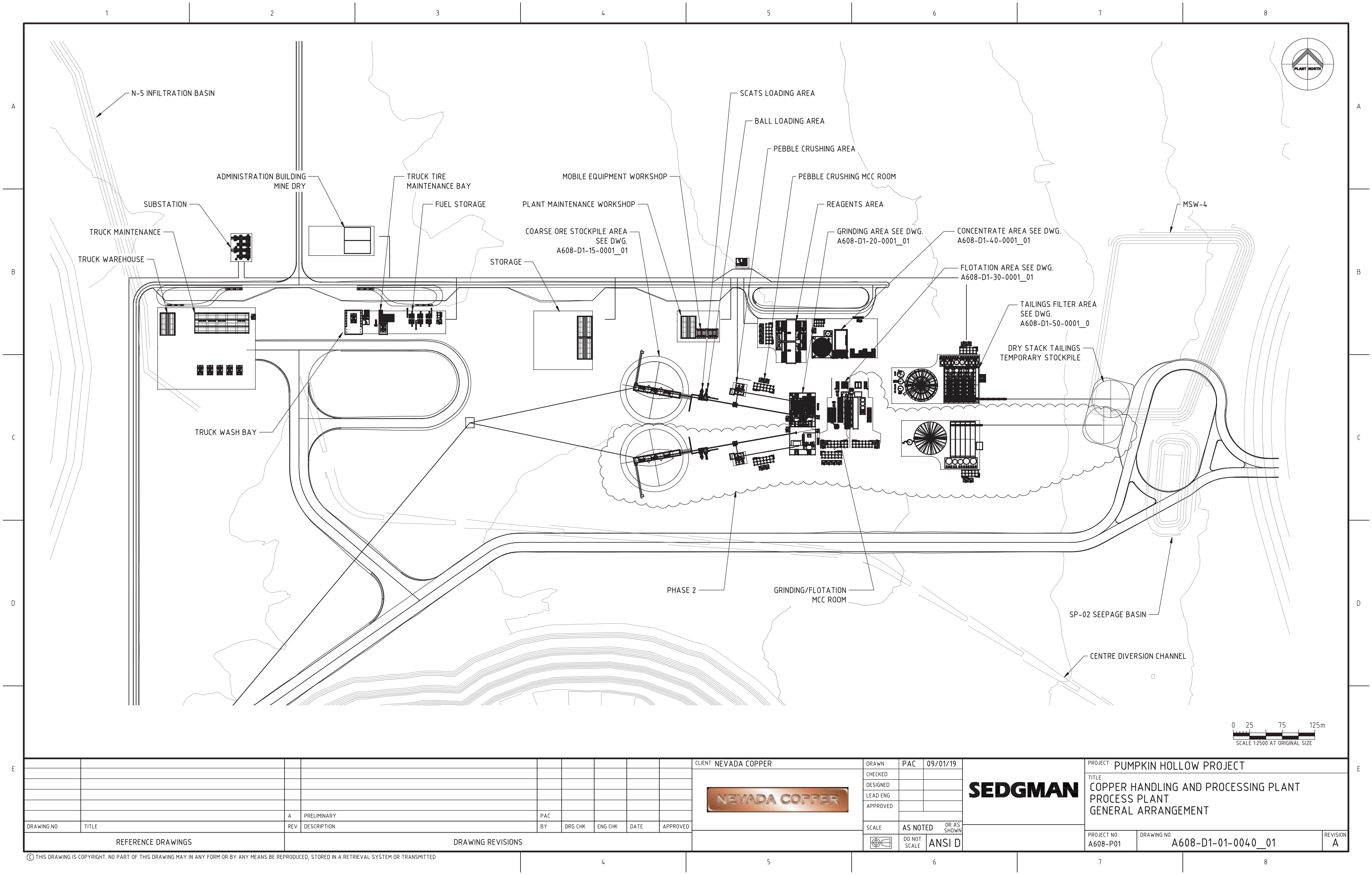
18.2 Open Pit

Infrastructure for the Open Pit Project includes site preparation, site roads, non-process facilities, bulk storage, services and utilities, communications, HS&E and security, G&A and site electrical.

The current layout for the infrastructure for Phase I and II is provided in Figure 4-3.

A more detailed layout for the infrastructure is provided in Figure 18-5, which shows infrastructure nearby to the main processing areas.

Figure 18-5: Phase I and II Detailed Site Infrastructure Layout (Sedgman, 2019)



The majority of the surface infrastructure is located close to the planned open pit processing facility. The following infrastructure facilities are required to operate the Open Pit Project separately from the Underground Project, with the following list of works to be completed during Phase I of the open pit mine development:

- Site earthworks preparations to suit the processing facilities and surface infrastructure facilities.
- Construction of off-site and on-site access roads required to support the mining operation, which will be an extension of the underground access roads currently being developed.
- Installation of stand-alone water infrastructure including a process water tank, a potable water tank and pumps, as well as a package sewage treatment plant to meet the requirements of the open pit stand-alone facilities.
- Installation of an overhead 120 kV line that will tie into and be supplied from the existing Nevada Energy high voltage (HV) infrastructure. This new overhead line will supply power to the Open Pit Project HV substation and distribution systems.
- Installation of site buildings including a site access security hut and gate, a truck shop building, warehouse storage, a mine dry facility and operations offices, noting that the open pit buildings are approximately 2 miles from the underground building; thus, there are limited opportunities to share infrastructure buildings. There will be additions to these buildings during Phase II to suit increased site labor profiles during Phase II.
- Installation of miscellaneous site facilities including parking areas, concentrate loadout, a fuel farm facility (by a fuel supplier), mine truck servicing buildings including a truck wash bay, and general area lighting required to safely travel between the different infrastructure buildings and the processing facilities.

The majority of the open pit infrastructure is planned to be installed during Phase I. The following infrastructure will be modified or added during Phase II, if applicable:

- Minor addition of on-site access roads.
- Addition of water infrastructure including process water tank, raw water tank and related pumps, as well as an upgrade to the sewage treatment plant upgrade based on the required labor for Phase II.
- Electrical substation upgrade, and distribution for Phase II facilities.
- Upgrade to administration building, truck shop and mine dry.
- Additional diesel storage and distribution (by the fuel supplier).

18.2.1 Preparation & Site Layout

18.2.1.1 Site Preparation

The following site preparations are included for the development of the Open Pit Project surface infrastructure works area:

- Clearing and grubbing for disturbed area; soil will be removed and stockpiled for use during site reclamation.
- Grading and capping temporary roads with an all-weather surface.
- Road design and minimal alteration to the existing off-site road to meet Lyon County roadway design and construction standards. Road capping of permanent roads will be supplied by a contractor from a nearby quarry.
- Site grading and road water management.
- Installation of chain link or barbed wire fences at the site's boundary perimeter and around the processing buildings and substations.
- Access gates will be installed at the site entrance and at major facilities, as required.

18.2.1.2 Site Layout

Figure 4-3 and Figure 18-5 show the overall site layout including Phase I and II. The following are the main surface facilities that support the mining and processing operations:

- Power substation.
- Waste rock stockpile.
- Fuel storage tanks, fresh water tanks.
- Concentrate storage and yard.
- Process facilities.
- Process buildings.
- Administration building, parking area, truck shop.
- Tailings filtration plants.
- DST facility.
- Waste water treatment plant.

18.2.2 Internal Roads & Earthworks

18.2.2.1 Internal Access Roads

All roads and earthworks were designed based on a lidar model with due consideration to the constraints of the mine pit shells, the management of site water and the planned vehicle movements for light vehicles and mine vehicles. All internal on-site roads and parking lots will be hardstand and unsealed. The main access road connection to the Underground Project's road (which connects to the county roads) will be sealed.

Signage will be placed to meet the design requirements including regulatory, preventative and informative signage.

Where there is regular traffic, dust will be suppressed by spraying water on the unsealed surfaces with a water truck. During operations, a site maintenance program has been designed to ensure that the roads and other infrastructure remain in optimal condition.

18.2.2.2 Haul Roads

Haul roads will provide access from the North and South pits to the primary crushing station, WRSF and filtered tailings emplacement area. Secondary heavy vehicle roads will be installed for routing mine vehicles to the truck work shop, mine in-pit fuel stations, main infrastructure area fuel and washing facilities and other necessary facilities. These roads will be designed to the required width for heavy mine vehicles on these roads with separate light vehicle roads ways or traffic lanes also provided where dual interaction is required. The haul trucks will drive on the left side of the road.

18.2.2.3 Earthworks

Run-of-Mine Receival Area

A ROM ramp and pad is required to run the trucks up to the dump station, which is a direct dump bin and primary sizing station. The ROM pad will be constructed mainly from waste rock, compacted and capped with structural fill with a road base capping in the vehicle pathways. A stockpile will be located close to the ROM pad in which the mine operations will store lower grade and higher grade materials for blending and management of ore feed to maintain the required production rates. To reduce ore haulage costs, the ROM dump station and run of mine stockpiles are located between the North and South pits, with the primary crushed ore conveyed to the processing facilities approximately 1 mile away.

The required earthworks for the ROM pad are shown in Figure 18-6.

Figure 18-6: Run-of-Mine Pad (Sedgman, 2019)

18.2.2.3.1 Processing & Surface Infrastructure Area

The processing facilities, mine administration building, haul truck workshop, light vehicle workshop, fuel storage and general warehousing is located outside of the nominal blast zones of the outer pit shells on the northern end of the mine lease.

The layout considers the following key vehicle and traffic movements for the facilities:

- Movement of mining equipment to and from the pit for required servicing
- Movement of fuel trucks to and from the mine for in-pit fueling of fleet, as well as the high volume road deliveries of fuel to the site
- The frequency of the road-based semi-trailer vehicles for the haulage of copper concentrate shipping containers
- The routing of the incoming power line and the location of the key power loads such as the processing facilities
- The routing of water around the site, as well as the management of surface water in planned and natural water courses
- The hauling of mine waste and filtered tailings to and from planned stockpiling areas
- Locating facilities outside of the mine blast zones

The surface infrastructure within the processing facilities is covered in Figure 18-7.

Figure 18-7: Processing Area Earthworks (Sedgman, 2019)

18.2.3 Buildings & Facilities

18.2.3.1 Process Facilities

The design, engineering and estimating of the process facilities for the Open Pit Project are discussed in detail in Item 17.2.2.

18.2.3.2 Administration Complex

The Open Pit Project will be serviced by a site administration complex consisting of a main administration pre-engineered building and any additional modules near the open pit site entrance containing:

- Offices, restrooms, change rooms, a lunch room and a conference room
- First aid room
- Site emergency services staging area; the facilities will manage all fire aid, fire monitoring and required dispatch to support the operations
- A parking area to accommodate both visitor and personnel parking spaces as required at the open pit site
- Main site security gatehouse and sign-in area

The administration complex has been located adjacent to the mine entrance gate to avoid the need for a full additional security building or gatehouse.

The administration complex will be expanded during Phase II.

18.2.3.3 Mine Dry

The mine dry will be located near the administration complex. This facility will be designed to serve open pit and plant personnel with a total of 120 workers in each shift (or 240 workers per day). This facility includes the following features within the pre-engineered building:

- Separate male and female showers and toilets
- Fire protection/suppression system as per National Fire Protection Association (NFPA) requirements

The mine dry will be expanded during Phase II.

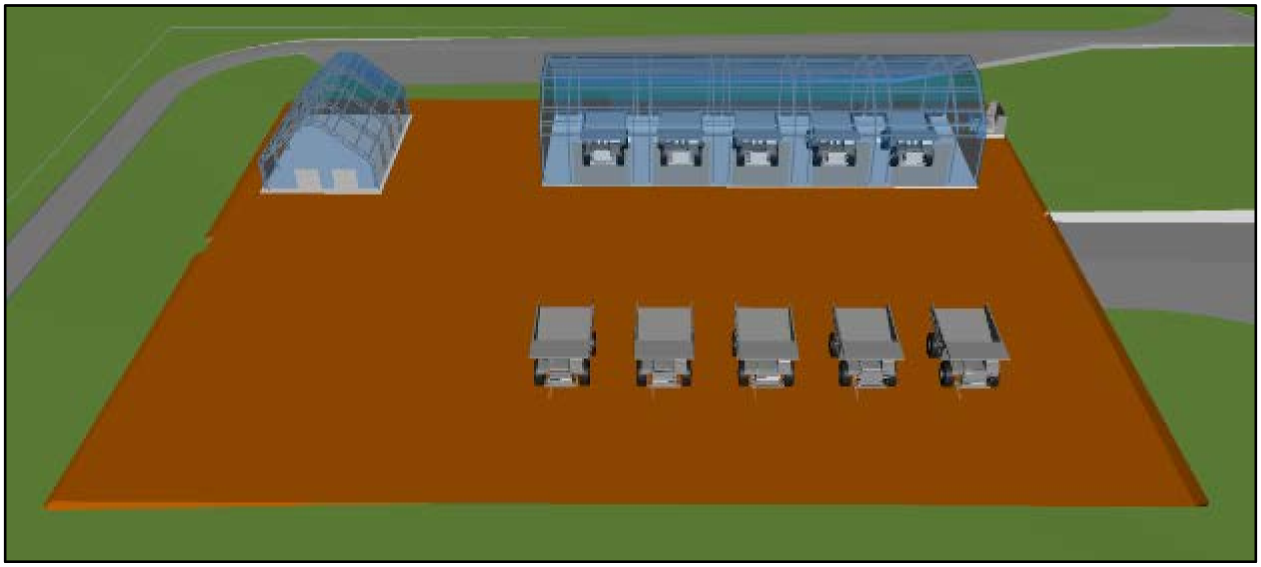
18.2.3.4 Truck Shop

The truck shop (Figure 18-8) will be located north of the North Pit, to the west of the combined infrastructure and processing facilities. The truck shop will be mainly used for maintaining the mobile mining equipment fleet of haul trucks. The truck shop is designed for five bays for haul trucks and is a pre-engineered building including the following:

- Truck maintenance bays
- Roll-up doors for the tire shop
- Warehouse for truck maintenance parts and tools storage
- Offices, restrooms, change rooms, a lunch room and a conference room
- Building overhead crane(s)
- Truck shop maintenance equipment, including:
    - Air compressor, receivers and a dryer
    - Tire repairing tools
    - The truck shop will be fitted out with racking, oil and lubricants distribution with collection and waste management systems
- Fire protection/suppression system as per NFPA requirements
- Eye wash station(s)

The truck shop will be expanded by one bay during Phase II.

Figure 18-8: Truck Workshop Area (Sedgman, 2019)



18.2.3.5 Fuel Farm

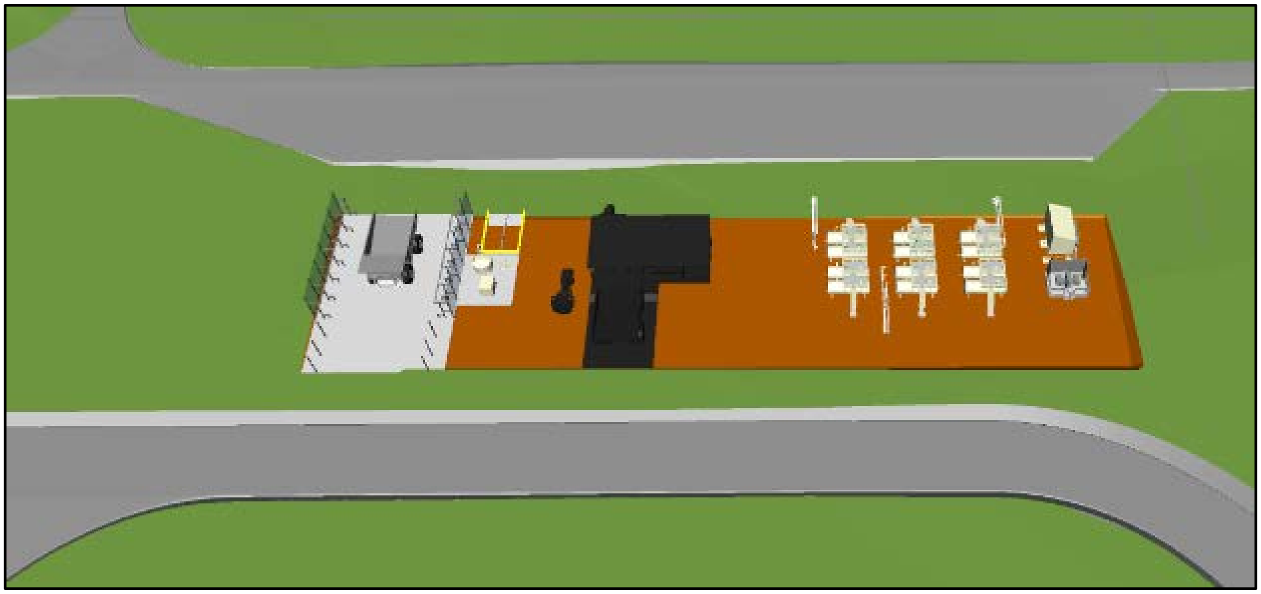
A fuel farm (Figure 18-9) will be located near the truck shop to provide diesel fuel for the mobile mining equipment fleet and gasoline fuel for site use pickup trucks and vehicles (via separate access roads). This equipment for the fuel farm will be provided by the contracted fuel supplier. The fuel farm will include following features and equipment:

- Self-bunded (double walled) diesel tanks with dispensers to meet diesel consumption requirement
- One self-bunded (double walled) gasoline tank with a dispenser
- Fire protection/suppression system as per NFPA requirements
- Bunded area for fuel filling and off-loading
- Card readers to manage inventory controls and management of the site held fuel

Additional diesel tanks with dispensers will be installed during Phase II.

Fuel will be dispensed from these facilities to mobile trucks for in-pit fueling of the fleet.

Figure 18-9: Fuel Farm Area (Sedgman, 2019)



18.2.3.6 Sample Preparations & Analytical Laboratory

On site will be a collection facility to store and house the collected samples for dispatch to off-site contracted testing facilities.

18.2.3.7 Warehouse Storage

The warehouse is an unheated building located near the processing facilities. This will be used to store goods that would not typically be stored outdoors and will contain inventory management controls and warehouse tracking.

An additional smaller warehouse storage has been included within the truck shop building.

18.2.3.8 Parking Areas

Parking areas are part of the site preparation cost and are designed for the:

- Administration building
- Tailings filtration plant
- Process facility
- Truck shop

18.2.3.9 Concentrate Management

Concentrate will be loaded into containers for export that will be weighed during filling of the containers and final sealing, which will help manage inventory control. The containers will be stored temporarily on site in a hard stand area and loaded onto trucks for transport to the transloading location and be designed to suit regional railing requirements.

18.2.3.10 Water Management

The following water services will also be installed as part of the surface infrastructure scope of work:

- Potable water will be pumped to the site from the tie-in point into the water line established with the Underground Project. On-site, the potable water will be distributed to the processing facilities and site offices areas, as required.
- Fire water will be supplied by an on-site raw water storage tank, and radially distributed around the site to the processing facilities, as well as each of the buildings:
  - Each of the buildings will be fitted with sprinklers and hose reels as required by the NFPA code requirements for the given structure.
  - Local fire monitors and hydrants will be located around the process buildings, truck workshop, warehouse and fuel storage buildings as required to protect the facilities.
  - Where required by NFPA requirements, the fuel storage will have sprinklers and fire monitors, as required.
- Raw water will be sought from wells located on site in the agreed nominated areas. The wells will be pumped to the infrastructure area for on-site storage of the raw water for use in the process plant and general mine usage.

Raw water will be distributed from the main header to the following site locations:

- Raw water tank
- Administration complex
- Truck shop
- Waste water treatment plant
- Mine dry
- Process buildings / process water tank
- Tailings filtration plants / tailings thickeners:
  - The process water tank will be located near the process and tailings filtration plant. The process water tank will have a capacity of approximately 245,000 gallons of water (approximately 930 m^3) to provide sufficient process water to both process and tailings filtration plant.

18.2.3.11 Waste Water Treatment

A packaged waste water (domestic sewage) treatment plant meeting local requirements will be constructed on site. The plant will be sized to serve 550 workers.

Sewage and waste water will be collected from a variety of locations throughout the site and fed to the sewage treatment plant by gravity, where possible. Lift-stations will be constructed along the sewage line and provide sufficient head pressure when transport by gravity is not possible.

Treated sewage and waste water will be used as process make-up water when the process water tank is low or needed. Any excess treated water will drain by gravity to a nearby infiltration basin. Sludge generated by the facilities will be collected and disposed of in the local council treatment facilities.

The sewage and waste water treatment plant will be expanded during Phase II.

18.2.3.12 Solid Waste Management

A policy will be established to minimize the use of and provide recycling for domestic wastes such as paper, aluminum, glass, plastics, and so forth. Collection will be undertaken regularly, with separated materials transferred to a secure, central storage facility on site for consolidation and onward transfer to re-processors. In general, dangerous waste will be collected and stored briefly at the point of generation before being transferred to a third-party, licensed disposal facility. Those materials that can be rendered inert will be treated at the site and transported to a suitable nearby landfill.

18.2.4 Fire Protection

The fire systems will be designed within guidelines and criteria that focus on the fire control by automatic fire protection, manual fire suppression, life-safety considerations and control of miscellaneous hazards.

The fire water pressure, flow rate, water supply duration and fire tank required volume will be designed to meet NFPA requirements. It will provide sufficient hydraulic pressure and water flow for the required duration for site buildings.

The fire protection system will include following equipment:

- Fire water storage (as a reserved portion of the raw water tank)
- A fire pump skid, containing:
  - Diesel pump / diesel tank
  - Electric pump
  - Jockey pump
  - Pump skid controllers
- Sprinklers
- Hydrants
- Fire extinguishers
- Call points

Additional fire protection will be installed where applicable for Phase II.

18.2.5 Dry Stack Storage Facility

The filtered DST storage facility design incorporates stable tailings storage, a containment system, a network of drainage pipework for seepage collection, a perimeter dike, surface water diversion and runoff management features. The DST facility will be situated east of the proposed open pits and will be constructed in three stages (Cell 1, Cell 2 and Cell 3) to suit the production schedule and minimize the facility footprint during the mine lifecycle. The ultimate shape of the open pit DST facility has a capacity of approximately 326 million cubic yards. To allow room for independent operation of the underground tailings storage facility, this central area will not be developed for open pit tailings storage until after the underground tailings facility is full and is no longer required.

The filtered tailings management option was selected for the Open Pit Project due to improved water efficiency, reduction in water demand, reduced risks associated with geotechnical stability and environmental impact and for mine closure benefits.

Selected design features and assumptions for the open pit TSF are summarized in Table 18-5.

Table 18-5: Open Pit Filtered Tailings Storage Facility Design Summary

Design Parameter	Value1
Design Tailings Storage Capacity	326 Myd3
Moisture Content (by dry mass) of Filtered Tailings	15%
Average Tailings Dry Density (assumed)	1.48 st/yd3
Tailings Production Rate, Maximum	73 Kstpd
Tailings Facility Operational Life	19 years

Note:

1. Tonnages represent dry short tons of tailings solids.

Filtered tailings will be transported by truck from the tailings filtration plant to the TSF. The material will be stacked, and compacted in staged lifts to manage geotechnical requirements and final stack geometry.

18.2.6 Transportation & Shipping

18.2.6.1 Introduction

Concentrate transportation for the Open Pit Project have been studied from the mine site to, and including, the transload facility. This included assessing the required infrastructure and associated capital and operating costs.

The stand-alone Open Pit Project is expected to be developed in phases with the process plant initially milling 37,000 stpd of ore. The second phase plant expansion will expand the mill production to 70,000 stpd. Concentrate tonnages range from 200,000 stpa to a peak of 500,000 stpa.

To market the independent supply and final logistics of the open pit copper concentrate, the transport and shipping study has considered:

- Trucking from the mine site to an internal market, a transload facility or a West Coast terminal.
- Railing of the concentrate from the transload facility to West Coast terminals.
- The containerizing of the copper concentrate and loading out via trucking to local or remote rail shipping.

18.2.6.2 Trucking from Site

Concentrate is stockpiled under the concentrate filter within the concentrate storage shed. A front-end loader will then load 42 st trucks. Each truck will be parked on a truck scale also located in the concentrate storage shed. Once loaded, the trucks will exit the mine site via a wheel wash.

For transport of concentrate to an internal market and West Coast terminals, the concentrate will then be trucked to a proposed transload facility at an existing railway line.

Trucking from site was evaluated also as an option on its own, as well as with rail. However, trucking to Stockton is the only reasonable option as a port. The route is 250 miles long with an expected one-way travel time requirement of 4.2 hours.

18.2.6.3 Transload Facility

The strategy for the movement of concentrate from the mine site is to use a transload facility that would be located at the UP rail line at Wabuska, just north of Yerington. This route is 20 miles from the mine along existing roads and avoids the most populated areas by accessing the highway just east of the mine site and has a nominal travel time of 26 minutes.

The intent of the transload facility is to act as an off-site storage area for concentrate production at the mine and make use of the nearest existing rail line for movement of the cargo to port. The property is currently owned by UP and is understood to have been previously used for bulk loading of trains with mineral products as the drive-over truck dump station and track sidings are still there.

Nevada Copper has entered negotiations with UP to obtain access to this property, and this would likely be linked to a contract for hauling cargo to the port or other site. Concentrate would be trucked to the Wabuska site from the mine in containers and staged for loading onto trains. Space would be required for a both full and empty containers to allow a quick turn-around on the trains.

18.2.6.4 Rail Transport

Based on the loading criteria for the containers, it is expected that an articulated flat car with an overall length of 90 ft is the most appropriate wagon to handle containers. This would carry a total of six containers with two double-stacked on each end to distribute the load on the axles. The train configuration is expected to consist of between 20 and 48 rail cars. Assuming a nominal length of 24 cars per train, a total trailing length of 2,160 ft behind the locomotives is expected. In this configuration, each train set would carry a total of 144 containers, containing a total of 3,333 st of copper concentrate.

Space would be required for a minimum of 288 containers to manage each train plus another 50% (72 containers) to allow for delays in the logistics chain. Loaded containers will be staged along the length of the tracks for loading onto trains with a reach stack forklift where empties will be returned to the mine site.

The total track distance from the Wabuska transload location to the Port of Stockton along the existing UP railway route is 279 miles, traveling north to Hazen, Nevada, before heading west to Reno and onto Sacramento, California, before the final run south to Stockton. This is a relatively short route by railway standards.

18.2.6.5 Port of Oakland

Another option of moving product in conventional sea containers will require them to be moved through a container terminal within the San Francisco Bay Area, and the closest option will be the Port of Oakland. While this could be done by rail, as the cycle times would be very similar to moving product through the Port of Stockton, it is anticipated that access to the container facilities may be better done using trucks.

18.2.6.6 Port of Stockton

The Port of Stockton is understood to be eager to take more concentrate through its facilities, and it has consistently been keen to take on new cargo volumes. It has expressed early interest in the Pumpkin Hollow cargo. While there are existing bulk facilities in the Port of Stockton that have historically shipped bulk copper concentrate, the Port Authority believes that no State permits would be issuable for any new bulk movement of copper concentrates and therefore the Pumpkin Hollow product must be shipped used using a sealed containerized “rotainer” system. The Port Authority has identified two potential sites for the receipt, storage and discharge of containers at an existing deep berth on what was formerly a U.S. Army/Navy base referred to as “Rough & Ready” Island. The port has two existing mobile harbor cranes that are under-utilized and would be made available for the vessel operations at the dock.

In each of the site options indicated by the Port of Stockton, the only physical improvements needed will be some additional rail track installation to facilitate the placement of railcars for the unloading and storage of containers. In both cases it is likely that the train will need to be broken into multiple sections on arrival at the port. However, preliminary figures provided by the port suggest that they can handle up to 24 rail wagons with six containers each in a single operating shift.

The Port of Stockton is the preferred location for the export of concentrate in bulk containers, but this location is not a port of call for conventional container vessels, and any consideration of using sea containers will require an additional haul distance to either the Port of San Francisco or the Port of Oakland for export through one of their container terminals. Based on the relative proximity of the Wabuska transload facility to the Port of Stockton, the movement of containers between these locations can be handled either by trucking or by rail.

18.2.6.7 Ocean Freight

Bulk freights have shown stability over recent years. Based on advice received from market data analysts, an average of $45.00 per wet metric tonne (wmt) of concentrate has been assumed for the open pit study.

18.2.6.8 Summary

Based on the evaluation, the sealed containerized bulk container option has been selected over the straight sea container option. In addition, the open pit study uses lease costs rather than purchase costs for the containers and other mobile and fixed facilities, while capital purchase costs for items as required.

The capital cost is $2.55 million for items including rail siding, roads, trackwork, turnouts, lighting, power, office, fence, and so forth.

An operating cost on a lease basis of $50.73/wmt Phase I and $48.12/wmt Phase II, for items including hauling, transload facility, rail, containers, loading truck, cars and marine facility.

Ocean freight costs are born by the portion of concentrates shipped overseas (as opposed to the portion sold domestically) on top of the rates reported above.

Costs of $62.0/wmt of concentrate for Phase I of the open pit production and $59.4/wmt for the Phase II of the open pit production are used in the Open Pit PFS, as an average basis for the transport of concentrates to the markets, as described in Item 19.0. Given the concentrate will be in sealed containers, no loss of concentrate is assumed in the freight from mine to market.

18.2.7 Power Supply, Substations & Main Distribution Lines

During the preparation of the Open Pit PFS, a new power study was re-commissioned from NV Energy to review power supply options, connection costs, and energy costs as operating costs. As of the effective date, this power supply report remains outstanding.

During this PFS, detailed electrical load lists and distribution plans were created that were costed to a PFS level of accuracy from the previously assessed connection points as defined by the regional power utility supplier NV Energy.

Given the high electrical load requirements and the need to phase the open pit project into the power needed for each phase of production, the studies assessed a number of 120 kV supply connection options including:

- The routing of a power line from the underground 120 kV switchyard: This line would be approximately 4 miles long and would be suited to the load requirements for Phase I, which is approximately 45 MW of diversified load.
- The routing of a new power line from the primary connection location of the underground 120 kV power source, which is approximately 8 to 9 miles from the open pit: This line would be sized

for both Phase I and Phase II's electrical load of approximately 85 MW (approximately 45 MW of diversified load for Phase I and approximately 40 MW of diversified load for Phase II).

- The routing of a power line from a substation close to the township of Yerington, referred to as Fort Churchill, which is approximately 16 miles from the site: Based on initial feedback from NV Energy prior to completion of the study, this option was considered as the baseline cost to be included in the PFS Capex.

The power supply to the Open Pit Project will come from NV Energy's existing Thorne power line, referred to the new Wassuk spur line. This line runs 4 miles to the Open Pit Project.

18.2.7.1 Open Pit Electrical Substation

The 120 kV power is routed to the site and landed in a stand-alone switchyard and substation planned for the open pit facility.

The main substation will supply two 30 MVA power transformers that will be used to transform the power to a 25 kV distribution voltage for reticulation around the site.

The facility will be a fenced compound, and will include the transformers, outdoor 120 kV switchgear, neutral earthing resistors and power factor correction equipment. The transformed 25 kV voltage will be distributed by switchboards located in a demountable switch room that will also contain remote operation panels and secondary metering equipment as required to meet the connection protection requirements.

Power factor correction equipment and harmonic correction equipment will be connected to the 25 kV switchboard, which will monitor the electrical loads and provide required correction to meet the projected NV Energy connection requirements.

Within each of the power consuming areas such as the processing facilities and the mine infrastructure building areas, there will be a number of secondary substations that will provide loads to additional distribution transformers.

The distribution transformers in each of the substations will further step down the 25 kV power to the following voltages:

- 4.16 kV or 13.8 kV for large motors.
- 600 V for low voltage equipment.
- 600 V and 120 V for the reticulation to distribution systems such lighting and small power.

18.2.7.2 On-Site Distribution

The on-site distribution of 25 kV will be run in a combination of radial overhead and underground cables to feed the substations/loads.

The substation and on-site distribution will be expanded during Phase II.

The general distribution plans are broken down into the following key areas:

- ROM crushing substation
- SAG, ball and regrind mills
- Flotation and plant essential services
- Concentrate and tailings substation
- Tailings filter facilities

The site infrastructure facilities including:

- Mine truck shop and light vehicle shop buildings
- Mine administration complex and mine dry facilities
- Water reticulation utilities and water treatment facilities
- Remote water wells
- The mine area including:
  - Two electric shovels in Phase I
  - A third electric shovel in Phase II
  - General power supplies to pit pumps and area lighting the ROM receival station

18.2.8 Surface Water Hydrology & Hydraulic Designs

The surface water hydrology and infrastructure required to manage those waters was developed to protect waters of the state. The surface water infrastructure was designed to include the management of surface water run-on, non-contact water and potential contact water. The plan resulted in using one existing diversion channel to manage surface water run-on impacting the mine site; utilizing two proposed small diversion channels to protect the DST from being impacted by run-off; utilizing two stormwater basins to collect rainfall that lands on the disturbed portion of the mine; and the construction of a new crossing of Little Pumpkin Hollow and Pursel Lane to ensure safe passage during the minor storm event.

18.2.8.1 Surface Water Hydrology

The Pumpkin Hollow Property is situated within the Little Pumpkin Hollow watershed, which drains an area of approximately 7.5 square miles via the ephemeral Little Pumpkin Hollow, which flows in a westerly direction until its terminus at an irrigation canal. The presence of the canal promotes ponding of water at the west site boundary. The Little Pumpkin Hollow watershed is located to the north of Pumpkin Hollow watershed, which ultimately drains into the Walker River. Steep slopes covered with sparse instances of sage brush and cheat grass characterize the topography of the site.

Double ring infiltrometer tests performed throughout the lowland areas of the site indicate favorable drainage conditions within dominantly sandy soils. Upland areas tend to exhibit lower infiltration capability, resulting in higher surface runoff.

Annual total precipitation is approximately 5 inches, eclipsed by an annual total evaporation of approximately ten times the amount of annual precipitation. The greatest amount of precipitation occurs within the months of December through February and May through June.

Given the configuration of facilities at the site, surface water run-on must be directed away from infrastructure and allowed to infiltrate into the ground. Water contacting ore or processed materials, in addition to stormwater (water not contacting ore or processed materials), must also be collected and managed accordingly. Surface waters are directed, contained and/or infiltrated via a series of diversion channels and basins.

Monitoring of precipitation, surface water flow and water quality should continue at the site to ensure compliance throughout the various stages of the Open Pit Project, including the closure and reclamation phases. Measured values should be compared against modeling parameters and design conditions to ensure they are in agreement.

Surface water runoff from the DST facility will be routed to the lined secondary containment basin MSW-4 located at the northwest corner of the DST facility. The seepage water collected by the overdrains will be routed to a lined seepage containment pond SP-2 located adjacent to MSW-4. Water accumulations within MSW-4 and SP-2 during the active mine life will be pumped back to the mill for re-use in process operations.

The NDEP regulations require that process water ponds or contact stormwater ponds that cannot be dewatered completely within 20 days must be designed as double lined ponds with a leak detection system. Since the pumping rate required to dewater the design storage volume of MSW-4 (121 210 ac-ft) within 20 days is less than the water demand for process operations, the basin is designed as a single lined facility assuming that the stormwater from MSW-4 will be consumed in process operations during operations. The liner system will consist of a 60 mil HDPE geomembrane liner.

The SP-2 seepage collection basin is designed as a double lined pond with a leak detection layer as the seepage flows reporting to the basin are likely to exhibit process signature. The double lined containment system consists of 60 mil HDPE primary and secondary liners with a HDPE geonet leak detection layer between the liners.

18.2.8.2 Regional & Site Hydrogeology

Within the Yerington district, the occurrence of groundwater is controlled by the geologic distribution of bedrock and unconsolidated sediments throughout the region (Figure 18-3). The majority (approximately 98%) of the groundwater flow in the region occurs within alluvial aquifers along the Walker River and the adjacent valley floors and alluvial slopes away from the river. Groundwater flow in the alluvium is generally down-valley. Within bedrock, regional flow of groundwater is generally from higher to lower elevations in response to the greater precipitation-derived recharge at higher elevations on the mountain ranges. Groundwater within alluvium connects with that in bedrock near the margins of the valleys.

Sources of recharge to the groundwater system include infiltration of a fraction of precipitation and irrigation water. Discharge from the groundwater system occurs primarily through pumping from irrigation wells and evapotranspiration.

Regionally, groundwater flow system boundaries include faults, the Walker River and Walker Lake, and hydrologic divides that separate the hydrographic basins and sub-basins. Within the Open Pit Project area, subvertical faults and altered clay-rich zones at the margins of some lithologic units compartmentalize flow in the aquifer system.

Groundwater at the site is present mainly in the bedrock; alluvium is saturated only near the west boundary of the site. Groundwater flow is generally toward the north and west in the site, with a vertically downward component of hydraulic gradient occurring in at least some parts of the Open Pit Project area. The bedrock is of generally low hydraulic conductivity except where fractures create secondary permeability and transmit groundwater. Numerous other faults with varying amounts of displacement act as hydrologic boundaries.

Age-dating of groundwater samples from Open Pit Project area wells demonstrates that alluvial/Tertiary groundwater is younger in apparent age (approximately 11,400 years) than that from Mesozoic bedrock (approximately 30,000 years). This, along with permeabilities more than two orders of magnitude greater in alluvium than in bedrock, suggests that the alluvium and the bedrock act as two relatively isolated and separate subsystems comprising the overall groundwater system.

18.2.8.3 Groundwater Model Setup

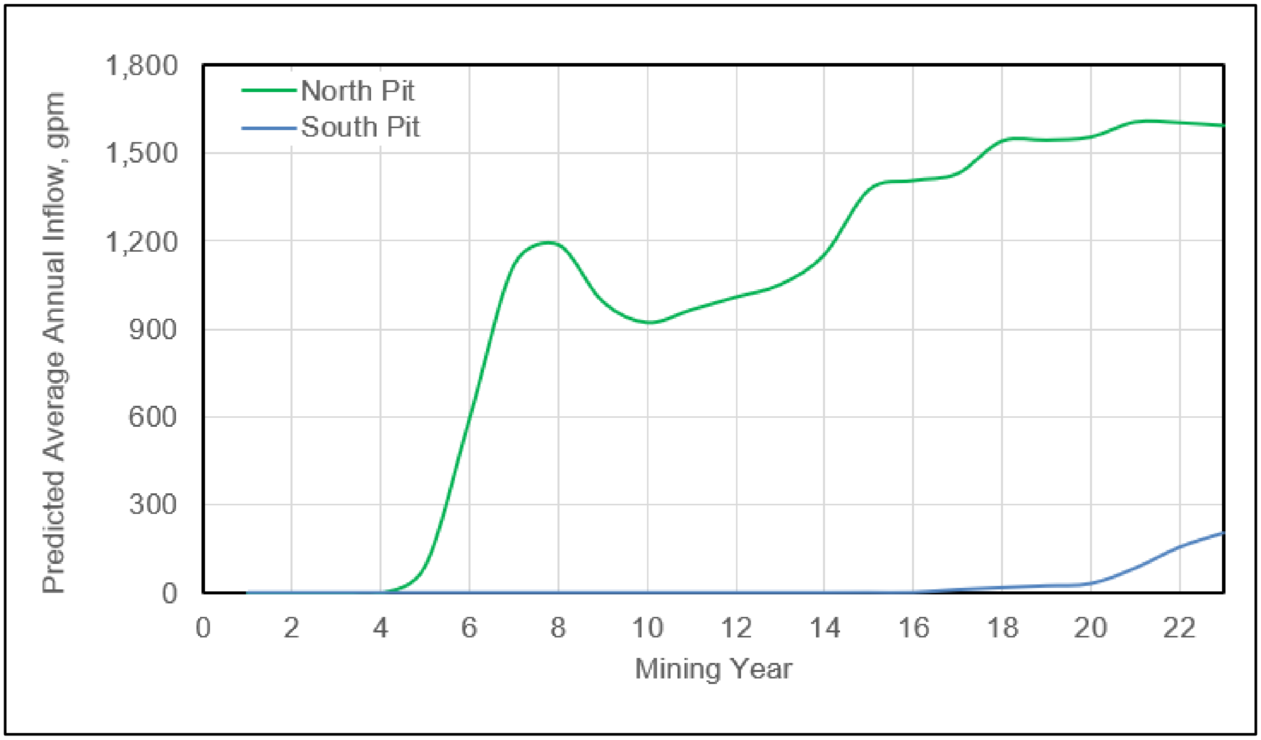
Please refer to Item 18.1.14.2 for an analysis of the groundwater model setup, also used in the Open Pit OFS.

18.2.8.4 Groundwater Model Simulations

The transient model simulated step-wise mining and dewatering of the underground mine and the North and South open pits during mining operations over approximately 23 years. Drain cells were assigned to simulate dewatering of the mine pits and underground mine openings, with the locations and elevations changing progressively in accordance with the mine plan. Post-mining conditions were previously simulated by de-activating the drain cells and representing the open pits as lake features that could receive inflow from groundwater, precipitation, and surface runoff and lose water via evaporation. A simulation time of 1,000 years was used for the post-mining model, as conditions were expected to reach steady state within that time.

The groundwater model indicated that dewatering would be necessary for the mining operations. Groundwater inflow to the open pits would begin when mining intersects the water table and would increase gradually as the pits are deepened. The predicted maximum inflows were about 1,600 gpm for the North Pit and about 210 gpm for the South Pit (Figure 18-10).

Figure 18-10: Predicted Groundwater Inflow during Mining Operations (Tetra Tech, 2019)



The post-mining groundwater model predicted that pit lakes would form in both open pits, with steady-state water surface elevations of approximately 3,799 ft amsl for the North Pit lake and approximately 3,812 ft amsl for the South Pit lake. Groundwater inflow was predicted to stabilize at approximately 885 gpm to the North Pit lake and 499 gpm to the South Pit lake. Although a small amount of flow was predicted to occur from the South Pit lake to the North Pit lake, evaporation would be the only outflow from the overall pit lakes area. Evaporation rates were predicted to stabilize at approximately 967 gpm for the North Pit lake and 552 gpm for the South Pit lake.

Mine dewatering activities and groundwater inflow to the pit lakes would lower the water table at and near the Pumpkin Hollow Property. By the end of the mining period, drawdown of 10 ft or more was predicted to extend approximately 0.1 mile west, 5.4 miles east, 4.6 miles south, 0.6 miles north and 6.2 miles southeast of the Pumpkin Hollow Property. The post-mining model predicted that drawdown of 10 ft or more would extend approximately 7.6 miles northeast and 8.1 miles southeast after 1,000 years. To the southwest, west and northwest toward Mason Valley, the post-mining model predicted drawdown of 10 ft or more extending approximately to the edge of the alluvial aquifer but not into that aquifer except directly west of the Pumpkin Hollow Property, where the thick alluvial aquifer extends about a mile farther east than elsewhere. The rates of groundwater inflow to the mines and the amount of water-level drawdown around the mines as predicted by the model updated for this study were smaller than those predicted by the earlier model. Consequently, the post-mining results predicted by the earlier post-mining model may slightly overestimate the residual, long-term drawdown and changes to streamflow.

Streamflows in the Walker River were predicted to decrease by a maximum of 1.2% during the mining period and a maximum of 0.4 percent in the Mason Valley and 1.2% in the Walker Valley 1,000 years after has mining ended. The maximum predicted changes occurred at the USGS gage Walker River near Mouth at Walker Lake.

The regional scale and equivalent porous media assumptions inherent in the numerical groundwater model, as well as unknowns regarding geologic structures in the vicinity of the Pumpkin Hollow Property, may over- or under-estimate the long-term inflows to the mines and lateral extents of drawdown propagation. Uncertainty in estimation of inflows could be minimized only through compilation of observations made during mine development activities and updating of the groundwater model to reflect the observations.

18.2.8.5 Mine Dewatering

During operation of the open pits, pit walls would be dewatered of groundwater using sub-horizontal inclined drains in the pit walls in areas observed to be generating high groundwater seepage. Discharge from the drains would be collected in bench ditches directing flow to strategically placed sumps. The sumps would also collect surface water pit wall runoff and direct precipitation. Sump water would be removed through pumping and discharge lines to the pit rim. Pumping from the sumps would be progressively increased with booster pumps added in stages with increasing pit depth.

Water collected from the mine development would be directed to the processing facility for raw-water make-up purposes. Excess water would be directed by pipelines to an infiltration basin south of the open pits for re-infiltration. Pipelines would be sized based on maximum estimated flow requirements and placed above ground.

18.2.8.6 Site-Wide Water Balance

The Pumpkin Hollow Open Pit Project site-wide water balance (SWWB) model was created with the objectives of predicting the volume of excess water to convey into the RIBs and to predict the make-up water quantity required by the process plant over the LOM.

The SWWB was modeled using the GoldSim software platform to address the objectives discussed above during the operation phase for the open pit facility. The conceptual SWWB model was developed using inputs from other Pumpkin Hollow Project engineering disciplines including process, mining, hydrogeology, tailings management, and operations.

The modeling approach considers the entire site as consisting of several sub-systems. Each sub-system represents a component of the mine operation and typically centers on a source or sink of water. Water inputs into the system include groundwater from the north and south pits, water entrained in the ore, potable water from the City of Yerington and water extracted from groundwater wells for process make-up water. The RIBs also have a water input from the climate in the form of precipitation; however, the magnitude of precipitation falling on the RIBs is insignificant compared to the amount of water being routed from dewatering to the RIBs for infiltration. Precipitation into the north and south pits is also considered insignificant and is not included in this model.

Water leaves the system through water entrained in the concentrate, water entrained in the tailings, “other” water uses (i.e., dust suppression or truck wash down stations) and infiltration from the RIBs. As with the water inputs, the climate, in the form of evaporation, affects the RIBs. However, the magnitude of evaporation is insignificant compared to the infiltration rate estimated within RIBs. Evaporation from the pit is not modeled as the pit lake surface area is expected to be small.

It was assumed for the water balance that groundwater could be used for the non-potable water uses, which include processing make-up water needs; “fresh” water sources for gland seals, reagent mixing, and cooling within processing; domestic (septic) water needs; and “other water uses,” which includes truck washes and dust suppression uses.

18.2.9 Geochemistry

The geochemical characterization program for the Open Pit PFS included the assessment of geologic materials that will be subjected to long-term storage including uneconomic mine rock (waste rock materials) and DST. The results of this study are being used to assist with the development of the Waste Rock Management Plan, post-closure pit lake chemistry and the design of the mine facilities with the objective of minimizing potential adverse water quality impacts.

In general, the program utilizes mine rock samples selected from each deposit throughout the Open Pit Project area with a focus on geospatially distributed samples from throughout the open pits. These samples represent the primary sources of material that would be stored on the surface. Samples were chosen from the 11 major lithologic units identified throughout the deposits with an emphasis on those units determined to have the highest potential to generate acid rock drainage and metal leaching (ARD/ML). The 11 rock units subjected to geochemical characterization are summarized in Table 18-6 in the coming pages.

Geochemical tests conducted on material deemed to be mine waste rock included:

- Two hundred and twenty mine rock samples underwent acid-base accounting (ABA) to assess the potential to generate and neutralize acid.
- These samples also were subjected to elemental analysis to determine the solid phase elemental concentrations and identify trace metals/metalloids of potential concern. One hundred and twenty-five additional samples also were subjected to whole-rock analysis to determine the content of the major rock forming elements.

- Net acid generation (NAG) testing was conducted on 28 mine rock samples to determine the contact pH and potential to generate acid after complete peroxide oxidation. Metal concentrations after complete oxidation were quantified to assess the upper limit of potential metal release.
- Seventeen mine rock samples underwent meteoric water mobility procedure testing to assess the potential leaching of constituents when subjected to conditions that approximate the effect of natural precipitation.
- Kinetic testing of 11 mine rock samples using humidity cell tests (HCTs) was conducted to provide an indication of the long-term acid production and consumption and concomitant metal leaching.

The following defines the analytical tests conducted on mine tailings material:

- ABA and elemental analyses were conducted on all samples.
- Meteoric water mobility procedure testing to estimate runoff quality of the final composite samples.
- Kinetic testing with HCTs on the final composite samples were used to estimate seepage quality from the DST facility and conditions were bracketed with HCTs of the tailings subsamples (rougher and cleaner scavenger tailings).
- The temporal changes in leachate quality of cemented paste tailings when groundwater recovers after mining ceases were investigated using passive diffusion testing by ASTM C-1308. The diffusion tests were conducted based on a tiered approach, using Open Pit Project area groundwater from Well WW10-01.
- Elemental analysis for Phase II tailings.
- Available water quality data were compared to NDEP Profile I Reference Values.
- NDEP values were used to distinguish potentially acid-generating (PAG) and non-PAG material with the lowest ARD/ML risk being associated with material that has neutralization potential ratio (NPR; NPR = acid neutralization potential / acid generation potential) values greater than 1.2 using the NDEP criteria. The BLM's net neutralization potential criteria of greater than 20 tons calcium carbonate/kilotons of material (t $CaCO_3$/kt) were also used to support the NPR classifications. The findings of the characterization program presented in this Report demonstrate that an NPR of 1.2 is sufficient to characterize material as non-PAG.

The ABA geochemical tests on mine waste rock material can be summarized as follows:

- A majority of the mine rock samples from each lithologic unit are considered non-PAG.
- The unmineralized mine rock designated as cover material (quaternary alluvium, tertiary conglomerate, and tertiary volcanics) are classified as non-PAG and generally contain sulfur below the reporting limit of 0.01 percent by weight (wt. %). On average, marble samples contain the largest excess acid neutralization potential of the rock types. Approximately 66 percent of the limestone samples also have high acid neutralization potential with the remaining limestone samples containing sufficient sulfide mineralization to produce acid in excess of neutralization.
- The dominant form of sulfur in the mine rock samples is sulfide, present as pyrite and chalcopyrite. Of the samples subjected to ABA, only the endoskarn samples contain over 5 wt. % total sulfur (median value of 1.68 wt. %).
- Sensitivity analysis suggests that endoskarn, magnetite skarn, and intrusive are unrepresented.

NAG pH testing on waste rock materials demonstrates that alkaline pH leachate is generated upon complete oxidation of any sample with an NPR >1.2. Further the NAG pH results further demonstrate that sulfur determination alone (total or sulfide) is insufficient to classify mine rock as PAG or non-PAG; however, the results suggest that classification of mine rock using NAG pH to supplement ABA is a viable method for incorporation into the operational characterization program.

Leachate testing of mine waste rock material selected from tertiary conglomerate, tertiary volcanics, marble, hornfels, magnetite skarn and endoskarn lithologic units, which were subjected to meteoric water mobility procedure testing, produced neutral to alkaline pH with some elevated concentrations of constituents that are also observed in groundwater.

In these leaching tests, arsenic is elevated above the NDEP Profile I Reference Values in at least one sample from each rock type except the magnetite skarn. Similarly, uranium above the primary maximum contaminant limits of 0.03 mg/L was observed in leachate from an endoskarn sample; however, samples from the other lithologic units and the other endoskarn sample were not elevated in uranium concentrations.

Selected waste rock materials (eight in total), have been subjected to 138 weeks of humidity cell testing. The samples are selected based upon an ABA rating as being PAG or having uncertain acid generating characteristics with the exception of the tertiary volcanic sample, which contains sulfide sulfur, but is considered non-PAG. Results of this testing show:

- Six of the eight mine rock HCT samples have produced neutral to alkaline pH over the entire test duration. The intrusive sample and magnetite skarn sample identified by ABA (NPR <0.5) and NAG testing (pH <3) as PAG began generating acid after approximately 20 weeks of testing. A hornfels sample continued to produce neutral pH through week 65 but showed increasing acidity, which is consistent with ABA and NAG pH results that predict eventual acid generation.

The sample pH had dropped to pH 5.5 by week 78. The delay in onset of acidic pH under accelerated weathering conditions suggests that sufficient time exists to conduct concurrent reclamation and minimize potential for ARD/ML. The two acid-generating HCTs and the tertiary volcanic HCT were terminated after 78 weeks of testing due to sufficiently stable constituent release rates. These HCTs would be replaced by samples with NPR values in the uncertain range.

- The non-acid generating HCTs continue to produce leachate with constituent concentrations below or similar to groundwater quality. All the mine rock samples exceeded the aluminum and iron guidelines during the first flush, with exceedances being less frequent in leachate from the non-PAG endoskarn hornfels, and silicate skarn samples.

These findings result in the following implications for mine waste rock management:

- Based on net neutralization potential and NPR values, all rock types except for magnetite skarn are considered to be non-PAG with an 85% degree of confidence.
- Sufficient sample numbers exist to define the hornfels and silicate skarn based on statistical distribution.
- Insufficient sample numbers exist to define the endoskarn, magnetite skarn and intrusive based on statistical distribution. It is recommended that additional static samples be collected to further characterize these rock types.

In reviewing the existing chemical character of mine waste materials, it has been determined that waste rock produced from the mining operation will be mixed and placed in the WRSF. A reclamation cover will be placed over the WRSF, which will consist of plant growth medium over the entire surface of the WRSF. The entire WRSF will be revegetated. Surface water conveyance channels will be constructed on each terrace, routing water to down chutes that feed into ponds.

An estimated 20% of the waste material scheduled to be stored at the MSRF will be potentially acid generating. Estimations are based on ABA results and the estimated lithological composition of the waste material. Hornfels rock type is the dominant lithology and accounts for approximately 13% of the potentially deleterious waste; however, as a whole, this rock type is potentially non-acid generating. Of lesser concern are skarn and intrusive rock types that account for 6% of PAG material. Limestone and talc rock types also have a proportion of PAG material. PAG material, regardless of rock type, will likely be blended and comingled with non-PAG rock types. This will effectively decrease any potential concentration buildup of PAG material on the surface of the WRSF.

In addition to waste rock material, tailings material underwent geochemical characterization testing. Based on the results of these tests, the following implications can be made:

- All samples have elevated iron and sulfur concentrations

- Calcium is elevated in the final composite samples

In regard to ABA, it was concluded that:

- The magnetite product, rougher tailings, magnetite tailings, final tailings composites, and cemented paste tailings are non-PAG using the NDEP (1990) criteria.
- The high sulfide sulfur content cleaner scavenger tailings are PAG; however, the mine plan does not include separate storage of scavenger tailings. These represent only 7.0% of the total mass of the mixed composite tailings. They would be fully mixed with rougher tailings prior to placement in the DST storage facility

Leachate testing on tailings samples produced by Hazen in 2007 resulted in constituent concentrations below water quality standards.

Kinetic testing showed for the most part non-PAG behavior for these materials with the resulting leachate having pH values in the neutral to alkaline-range. The only acidic pH leachates are from the PAG cleaner scavenger tailings under the fully oxygenated conditions of the humidity cell testing.

The current above-ground mine plan for the Pumpkin Hollow Project will result in the development of two open pit mines, northwest and west of the underground workings. With an understanding of waste rock material chemistry, it is possible to utilize such data to imply the geochemistry character of similar rocks present on the ultimate pit surfaces of these pits. When used in conjunction with precipitation budgets, evaporation rates, groundwater and surface water chemistry, and the overall geology of the exposed pit wall surfaces, modeling the composition of any resulting pit lakes upon cessation of mining operations is possible.

Hydrogeological studies suggest that upon cessation of mining operations and the termination of dewatering activities in the Open Pit Project area will result in pit lake formation in both pits.

Based on the overall water budget including inflows to the pit lakes (groundwater seepage and precipitation) and annual evaporation from the pit lake surfaces, the pit lakes will be hydraulic sinks. Steady state flux of water flow would be reached approximately 800 years after mining in the North Pit and approximately 600 years after mining in the South Pit. It is expected that periodic mixing and re-oxygenation of the water column will occur and therefore oxidation reactions will predominate, even at depth.

The resulting models for these pit lakes predict that:

- The majority of inflow water entering the pits will be from groundwater sources seeping through the pit walls.
- Water quality is acceptable with respect to NDEP Profile I Reference Values and background groundwater quality data.

In that updated plans for backfilling of the northernmost pit to some extent by materials from the southern pit have changed the volumes and geometries of the two pits, careful assessment of the PAG/non-PAG character of this backfill material as well as recalibration of wall rock composition proportions future models that will need to determine if the current pit lake chemical model is still valid. In general, however, evaporative loss and overall regional groundwater chemistry are felt to be the driving constraints on final pit lake chemistry.

Table 18-6: Representative Lithologies

Lithologic Unit	Abbreviation	Description
Quaternary Alluvium	QAL	Mainly fluvial sands, with pebbles and cobbles, generally located on and near the surface in drainage channels and valleys.
Tertiary Conglomerate	TCG	Pliocene deposits composed of weakly to moderately consolidated conglomerates with a sand and clay matrix, including some volcanic ash beds. Often mixed with clasts (pebbles and cobbles) of older rock types derived from underlying volcanic tuff and older Mesozoic rocks.
Tertiary Volcanics	TV	Igneous rhyolitic tuff breccias (welded volcanic ash).
Limestone	L	Lenses of more pure limestone associated with hornfels. May also be part, larger distinct limestone zones (Mason Valley limestone).
Marble	MA	Limestone that has been thermally metamorphosed due to igneous intrusions.
Hornfels	H	Metamorphosed marine sedimentary rocks with shale, limestone, siltstone and argillite (clay stone).
Magnetite Skarn	Msk	Predominately composed of magnetite ($Fe3O4$) formed by contact metamorphism that may contain sulfide minerals including pyrite ($FeS2$), pyrrhotite ($Fe1-xS$), chalcopyrite ($CuFeS2$), and carbonate minerals including calcite ($CaCO3$).
Silicate Skarn	Ssk	Metamorphosed marine sedimentary rocks with silicate minerals instead of iron oxide minerals that characterize magnetite skarn. Coarser grained than hornfels.
Endoskarn	En	Composed of altered granitic rock where silica has been displaced by calcite and other silicate minerals. This rock type may vary in composition and may contain pyrite ($FeS2$), pyrrhotite ($Fe1-xS$), chalcopyrite ($CuFeS2$) and magnetite ($Fe3O4$) in variable percentages.
Intrusive	In	Fresh to weakly altered granitic rocks that have intruded into marine sedimentary rocks followed by metamorphism of these sediments into hornfels, skarn and marbles.
Talc	Ta	Fine grained magnesium silicate mineral.

Item 19.0 MARKET STUDIES AND CONTRACTS

19.1 Contracts

Generally, the Pumpkin Hollow concentrate is projected to be a very clean material, with few penalty items. The concentrate contains low levels of arsenic, moderate levels of gold, and will be considered a “clean concentrate” for the purposes of marketing. Fluoride and mercury represent the only penalty elements within the concentrate. Fluoride is normally penalized above 300 ppm, and mercury is subject to penalties when in excess of 10 ppm. Neither element prevents marketability only impacting final pricing of the material. The expected levels of impurities will not result in any material penalties.

19.1.1 Underground

An offtake agreement with MF Investments exists for 25.5% of the copper concentrates production derived from the Eastern Area deposits that are from underground mining. This contract is now owned by Transamine, a metals trader.

19.1.2 Open Pit

No offtake agreement or other contracts are in place for the open pit mine concentrates.

19.2 Main Markets

Pumpkin Hollow’s geographical location close to the West Coast shipping ports and North American copper smelters presents three potential marketing outlets for the concentrate: Asia, Europe and North America.

Transportation studies provided by Concord Resources Limited (Concord) and Savage Rail, Industry and Chemicals Logistics Group (Savage) were reviewed and evaluated in the preparation of this Item. Concord is a metals trader and a global metal commodities merchant, and provided information on supply chain related to a concentrates rail load facility to the ultimate market, and copper concentrates marketing advice. Savage is a large well-respected logistics firm headquartered in Midvale Utah and provided information on supply chain advice from the Underground Project location to a concentrates rail loading facility located north of Yerington.

19.2.1 Asia

The impurity levels in the Pumpkin Hollow concentrate are expected to be well below levels allowed for importation into China. China currently has over 8 million metric tonnes of copper smelting capacity, which was being utilized at 75% in 2016. This use is expected to grow, but not to exceed installed capacity over the next five years. With growing smelting capacity and limited mine expansions, the Chinese smelting community is pursuing new sources of raw material supply, which Nevada Copper will target with the Pumpkin Hollow concentrate. Additionally, Nevada Copper is in very preliminary discussions with Indian smelters to explore the acceptability of Pumpkin Hollow to feed their smelter expansions.

Finally, Korean and Japanese smelters could be outlets for sale. Given the moderate gold content in the concentrate, the other east Asian smelters would not necessarily be the priority customers.

Nevada Copper has studied various logistics routes for the movement of concentrate to West Coast terminals and domestic concentrate customers. Currently, routes considered include routes

19.2.2 Europe

Nevada Copper is also considering markets in Europe using swap agreements, whereby Nevada Copper would supply European markets with concentrate from other origins with better freight parities, instead of shipping the Pumpkin Hollow concentrate from U.S. West Coast to Europe.

19.2.3 North America

The Pumpkin Hollow concentrate is also positioned for sale to Rio Tinto’s Kennecott smelter in Magna, Utah, and Grupo Mexico’s Hayden smelter in Tucson, Arizona. Both smelters regularly process domestic concentrate to maximize their smelter utilization rates. Freight rates to Kennecott are significantly cheaper than to Hayden.

Freight to Kennecott is significantly cheaper than to Asian markets. Refer to Item 21.1.11 for details and a summary of freight costs.

19.2.4 Summary

There are a number of possibilities for marketing the concentrates, including Asian, US domestic and European smelters, the latter likely under a concentrates swap arrangement. For cash flow purposes, average concentrate transportation costs are estimated at $67.5/wmt for the stand-alone development of the underground mine and $62.0/wmt for the larger stand-alone open pit mine, based on product moved both via Vancouver for the stand-alone underground near-term concentrates and also domestic consumers, while the stand-alone open pit transport is via Stockton as well as to domestic consumers.

19.3 Copper Price Forecasts

Independent analysts at leading investment banks and research institutes mostly predict a tightening market for copper concentrates into the 2020s, resulting from a lack of investment in greenfield and brownfield copper mines following the bear market in copper prices from 2012. At the same time, world copper demand is forecast to continue increasing at rates similar to those seen recent years (compound annual growth rate of 2.4% from 2007 to 2016) in line with the outlook for increasing world industrial production. This production is the principal driver of demand for copper and other non-ferrous metals. This combination of lower supply and higher demand is expected by many commentators to provide support for copper prices.

Consensus pricing prepared by Consensus Economics Inc. with a survey date of August 14, 2017, was calculated using an arithmetic mean of 22 forecasts. Refer to Table 19-1 for details. This pricing was used in the Underground PFS and is also used in the stand-alone Open Pit PFS. Nevada Copper considers the consensus valid at the effective date of this Report.

Table 19-1: Copper Pricing (nominal terms)

Source	2019	2020	2021	Long Term 2022 onward
Consensus Copper price ($/tonne)	6,250	6,714	6,912	7,049
Consensus Copper price ($/lb)	2.83	3.05	3.14	3.20

19.4 Smelter Charges

Alongside these copper price forecasts, it is important to review forward estimates of treatment charges for copper concentrate, which have an important influence on any copper mining development's profitability. CRU provides treatment charges (TC) and refining charges (RC). Table 19-2 examines these forecasts for 2017 to 2019, which reflect the consensus that a tighter balance in the market will probably emerge as smelters seek to increase metal production to meet rising copper demand. This smelter charges are used in the stand-alone underground mine study and are also used in the stand-alone open pit study. Nevada Copper considers the consensus valid at the effective date of this Report.

Table 19-2: Treatment Charges & Refining Charges Estimate for 2019 and Long-Term

Source	Charge Type	Pricing Units	2019 & LT
CRU smelter charges	TC	$/dmt	75.0
	RC	$/lb	0.075
	TC/RC	$/lb	0.201

The stand-alone Underground Project study and stand-alone Open Pit Project study assume average treatment and refining charges (TCRCs) of $75/dmt and $0.075 per payable pound of copper respectively for the copper concentrates production.

Metal payment assumptions based on market data and used in both the stand-alone Underground PFS and Open Pit PFS are:

- For copper: 96.5% with a minimum 1-unit deduction
- For gold and silver: 90% payable at grades exceeding 1 g/t and 30 g/t respectively and zero for less than those thresholds.

19.5 Metal Pricing

Gold prices have traded in a range between approximately $1,050/t.oz and 1,350/t.oz since the start of 2016 and continue to be caught between positive and negative counter-currents: for example, worries over the outlook for the US economy on the one hand and the Federal Reserve apparently determined to pursue a policy of rising interest rates on the other; and political risks building up in the USA, but fading in Europe (excluding the UK). In this context, it expected that the average of gold price forecasts published in 2017 (and compiled by Bloomberg) follow a similar range out to 2021 (Table 19-3). Nevada Copper considers the consensus valid at the effective date of this Report.

Table 19-3: Gold Price Forecasts (nominal terms)

Source	2019	2020	2021
LBMA gold price ($/t.oz)	1,330	1,408	1,273

After spiking in mid-2016, the trend in silver prices has drifted moderately lower and underperformed gold prices. More recently this is probably explained, in part at least, by renewed worries over the world economic outlook and global geo-political risks, since the former tends to weigh on silver prices, while the latter tends to support gold prices. The average of the silver price forecasts published in 2016 and compiled by Bloomberg shows a moderate rising trend out to 2021 (Table 19-4). Nevada Copper considers the consensus valid at the effective date of this Report.

Table 19-4: Silver Price Forecasts (nominal terms)

Source	2019	2020	2021
Silver price ($/t.oz)	20.23	22.13	19.40

The Underground PFS gold and silver refining charges are assumed to be $5.00/t.oz and $0.40/t.oz of payable metal, respectively.

The Open Pit PFS gold and silver refining charges are assumed to be $4.00/t.oz and $0.35/t.oz of payable metal, respectively.

19.6 Metal Price Assumptions for Cash Flow Projections

The metals prices were used for the cash flow projections were the consensus copper prices shown below (Table 19-5). Sensitivities of the economic results to variations in metals prices, as well as in capital and operating costs, are shown in Item 22.0. These prices, as applicable, are used in both the stand-alone Underground Project and Open Pit Project studies. Nevada Copper considers the consensus valid at the effective date of this Report.

Table 19-5: Metals Prices (used in economic modeling)

Item	Unit	2019	2020	2021	2022 & LT
Consensus Copper Prices	$/lb	$2.83	$3.05	$3.14	$3.20
Consensus Gold Prices	$/oz	$1,276	$1,285	$1,284	$1,325
Consensus Silver Prices	$/oz	$18.77	$19.40	$19.53	$20.01

Source: Consensus Economics Inc. - August 2017.

Item 20.0 ENVIRONMENTAL STUDIES, PERMITTING, & SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The Underground and Open Pit Projects will be completed on 100% privately owned lands as a result of the Yerington Land Conveyance (see Item 20.6). The entire Pumpkin Hollow Property is now under local and Nevada state oversight. There is no other nexus under federal statutes and regulations that requires federal environmental permits or preparation of an environmental impact statement pursuant to NEPA. There are no endangered species located on or near the Property, no surface waters, no jurisdictional waters of the U.S. that require a permit, no designated wilderness near the Property, no Class I air quality designations, no critical habitat areas, no sage grouse (a species of concern in Nevada), and no wildlife migration zones that cause environmental constraints.

The stand-alone Underground Project is being permitted to disturb approximately 445 acres and the stand-alone Open Pit Project is currently permitted to disturb approximately 3,000 acres, for a total of 3,445 acres.

20.2 Overview of Operations & Permitting

Infrastructure for the ore processing facilities, tailings filtration and miscellaneous support buildings will be constructed for the Projects, and will include buildings, process water management basins, light duty and haul roads, utilities, and conveyors for handling ore, mine rock and tailings. Project facilities are listed in Table 20-1. Facility locations are shown on Figure 4-1.

Table 20-1: Pumpkin Hollow Project Facilities

PROJECT FACILITIES
SURFACE WATER FACILITIES
Mining Stormwater Management Basins [per Sedgman General Arrangement]
Diversion Channels [per Sedgman General Arrangement]
Rapid Infiltration Basins [per Sedgman General Arrangement]
PROCESS WATER MANAGEMENT
Secondary Containment Basins [per Sedgman General Arrangement]
DEWATERING INFRASTRUCTURE
Dewatering Pipelines (Underground Mine and Surface Mine)
Stormwater Management Basins
RECLAMATION TEST PLOTS
Revegetation and Erosion Control Test Plots
PLANT GROWTH MEDIUM STOCKPILES
PGM Stockpiles [per Sedgman General Arrangement]
MINE ROCK STORAGE FACILITIES
Underground Mine MRSF
DRY STACK TAILINGS FACILITY
DST Facility
SURFACE INFRASTRUCTURE
Site Entrance Gate, Scale and Security Buildings

Administration Buildings
Sewage Treatment System
Vehicle Shop(s)
Fueling Areas (near Process Plant)
Process Plant
Pebble Crusher
Coarse Ore Stockpile Tunnel
Tailings Filtration Plant
Thickeners (Concentrate Thickener, Tailings thickener)
Process Water Tanks (at Process Plant)
Fresh Water Tanks (at Process Plant)
Explosive Magazine
Electrical Substation(s)
Power Lines
Pipelines (Fresh Water, Sanitary Sewer, Process Water Return, Tailings Slurry)
Conveyors (Tailings, Process Facility)
Fences (Disturbed Area Perimeter, Magazine Area, Plant Area, Water Management Basin Area)
Coarse Ore Stockpile
Temporary Tailings Storage Area
Demolition Debris Disposal Area
Underground Mine Dry
Construction Office
Electrical Substation
Dry Storage
Hoist House
Collar House
East Shaft Headframe and Back legs
Fire Water Tank
Dry & Operations Office
Water Tanks
E-House, Transformers & Generator
Tailings Thickener
Tailings Filtration
Paste Plant
Air Services Facility
Low-Grade Ore Stockpiles
Pipelines
Conveyors
Explosive Magazines
ROADS
Light Duty Roads
MONITORING WELLS
Monitoring Wells (Piezometers, Vadose Wells, Groundwater Monitoring Wells)

Pumpkin Hollow Project mine facilities are contained entirely within the private lands owned and controlled by Nevada Copper. A comprehensive list of permits is included in Table 20-2. Permits required for the project are administered by State of Nevada by the following state agencies:

- Nevada Division of Environmental Protection (NDEP)
- Nevada Division of Water Resources (NDWR)
- Nevada Department of Wildlife (NDOW)

Within NDEP, several bureaus issue key state permits including the BMRR, the BAPC and the BWPC. Within BMRR, the Regulation Branch issues two types of WPCPs: one for mining and one for dewatering. The second one is for management and discharge of water derived from dewatering the mine and issued by BMRR, rather than the BWPC, which consolidates and streamlines the permitting process for mines into BMRR. The Reclamation Branch issues Reclamation Permits, and the Closure Branch reviews and approves mine closure plans. BWPC issues permits for on-site sewage disposal systems (OSDSs).

Nevada Copper has submitted several permit applications and permit amendments to state and local agencies and all have been granted to date. During the development and operation of both the underground and open pit projects, these permits require annual reviews, periodic updates, modifications and revisions as necessary. Nevada Copper has already submitted and obtained approvals for several state permits and permit modifications.

Subsequent to the acquisition of BLM-administered federal land by the City and Nevada Copper, the City of Yerington annexed all of the Pumpkin Hollow Property private lands located in Lyon County in 2016 (approximately 105 acres of the conveyed lands are located in Mineral County). As part of the annexation, the City updated the City master plan to include the Pumpkin Hollow site and zoned all the lands associated with the Pumpkin Hollow Property “M-2 Special Industrial – Mining.” This zoning designation explicitly permits mining and mining-related activities. The only City permits required for construction of the Underground and Open Pit Projects are building permits. There are no special City requirements for buildings and structures other than International Building and Fire Codes.

Refer to Figure 18-1 for the site layout for the Underground Project.

20.3 Environmental Setting

The Pumpkin Hollow Property resides in basin and range topography within the north-south trending Mason Valley situated between the Singatse and Wassuk mountain ranges between 4,550 ft and 4,900 ft amsl.

The climate is arid with hot summers and relatively mild winters. The Pumpkin Hollow Property is located in a dry alluvial valley with low barren hills. Vegetation in the immediate area is sparse low brush with local grasses suitable for limited cattle grazing. The agricultural Mason Valley to the west contains numerous alfalfa and onion fields and grazing lands. These fields are watered by irrigation canals from the nearby east fork of the Walker River and groundwater wells.

Precipitation is sparse on the Property. A 102-plus year precipitation record (nearly continuous daily data spanning from January 1, 1914, through November 1, 2018) exists from the Western Region Climate Center (WRCC, Station Number 269229) station located at Yerington, Nevada. Maximum average monthly precipitation occurs during the month of May, but the majority of annual precipitation falls during the winter months with particularly heavy snowfall in the mountains. Average annual precipitation at the Pumpkin Hollow Property is approximately five inches per year.

The Property area is dominated by salt desert shrub communities, primarily bud sage (*Artemisia spinescens*). Other species found in this plant community include: saltbush (*Atriplex* spp.), rabbitbrush (*Chrysothamnus viscidiflorus*), Bailey's greasewood (*Sarcorbatus baileyi*), spiny hopsage (*Grayia spinosa*) and spiny horsebrush (*Tetradymia spinosa*). Grass cover is sparse and is predominantly Indian ricegrass (*Achnatherum hymenoides*) in the alluvial soils and non-native cheat grass (*Bromus tectorum*). Wildlife likely to inhabit the area includes deer, feral horse, raptors, bats, ground squirrels, mice and rabbits.

20.3.1.1 Federally Listed Species

Nevada Copper has performed surveys and evaluations and determined that none of the federally threatened and endangered species that occur within Lyon and Mineral counties are likely to occur in the Project area based on the flora and vegetation types located on the Property area (reference Bureau of Land Management Final Environmental Assessment, Yerington Land Conveyance, 2015). The only federally threatened or endangered species that occur in Lyon and Mineral counties are fish species, and there are no perennial or fish-bearing streams in the Pumpkin Hollow Property area.

20.3.1.2 Proposed Threatened Species

The Property area does not occur within a Sage Grouse Management Unit (USFWS 2004). Additionally, sage grouse habitat identified by USGS does not occur within the Property area (USGS 2001).

A threatened species is one that is likely to become endangered in the foreseeable future. Two Proposed Threatened species, the greater sage-grouse (*Centrocercus urophasianus*) and the yellow-billed cuckoo (*Coccyzus americanus*), as well as two Threatened species, the Lahontan cutthroat trout (*Oncorhynchus clarkii henshawi*) and the Railroad Valley springfish (*Crenichthys nevadae*), are not known to occur on the Property, based on the absence of suitable habitat.

The Nevada Office of the USFWS has designated Sage Grouse Management Units for the Bi-State area of eastern California and west-central Nevada for the Mono Basin Area population of sage-grouse. As its name suggests, the Bi-State Distinct Population Segment straddles the California-Nevada border, where biologists estimate that between 2,500 and 9,000 of these ground-dwelling birds inhabit about 4.5 million acres of high-desert sagebrush (USFWS 2015). The USFWS declared the greater sage-grouse a Distinct Population Segment in 2010. The BLM posted a notice of cancellation of Withdrawal Application to withdraw sagebrush focal areas in Nevada (and five other states) as recent data show that future mining is not a significant threat to Greater Sage-Grouse Habitat. The BLM has published final environmental impact statements (EISs) and proposed amendments to the current plans. A 30-day period to protest proposed amendments closed on January 15, 2019 (BLM 2019).

Though the Pumpkin Hollow Project is in the vicinity of the Bi-State Distinct Population Segment, year-round habitat, as well as nesting and brooding habitat for the greater sage-grouse does not occur in the project area, and the nearest such habitat is approximately 20 miles to the southwest of the project area. As such, the greater sage-grouse is not likely to be impacted by the Pumpkin Hollow Project.

20.3.1.3 State-Listed Species

The State of Nevada maintains a list of protected species. Removal or destruction of any state-listed plant species requires a special permit issued by the State Forester/Firewarden. The species recorded in Lyon and Mineral counties with the potential to occur on the Pumpkin Hollow Property, based on habitat, include Greater sage-grouse, Churchill Narrows buckwheat, spotted bat, western snowy plover, ferruginous hawk and Swainson's hawk. Consultation with Nevada Department of Wildlife (NDOW) occurs with the approval process for any disturbance-related permit issued by BMRR.

20.3.1.4 Special Status Species within Nevada

NDOW has general management authority over wildlife within the State of Nevada pursuant to the Nevada Revised Statues. NDOW documented a variety of species with the potential to occur in the area based on habitat descriptions and habitat availability (RCI 2015). The Nevada Natural Heritage Program (NNHP) specifically identified the potential for suitable habitat in the Pumpkin Hollow Property for the western small-footed myotis (a BLM special status species). The bat is found in arid upland habitats, and prefers brushy habitats. This species is typically found in close proximity to water including streams, ponds, springs, and stock tanks.

The BLM is also required to protect and manage wild horses and burros on public lands. They must maintain sound thriving populations within herd management areas and remove any wild horses and burros that stray onto private land at the request of the landowner. The Pumpkin Hollow Property, though now private land not subject to BLM management since the 2016 transfer, does not occur within a BLM herd management area (BLM 2001), but feral horses have been noted in the area. A BLM herd management area occurs approximately four miles to the southeast of the Pumpkin Hollow Property.

Publicly available data and existing reports were reviewed to determine the permitting and consultation requirements for plants and wildlife and protected species for the Pumpkin Hollow Property located in parts of Lyon and Mineral counties. Impacts to special status or listed species requiring federal oversight or involvement are not anticipated.

20.4 Heritage Resources

Archaeological surveys were performed on the private lands owned or controlled by Nevada Copper, including the lands underlying the Pumpkin Hollow Property, in 2011–2012. There are currently three prehistoric sites and two historic sites, a total of five sites, within the federal lands that were conveyed to Nevada Copper that are either recommended for eligibility on the national register of historic places (three sites) or require further evaluation (two sites). These sites are now administered by the Nevada State Historical Preservation Office (SHPO) pursuant to a Memorandum of Understanding (MOU) amongst the SHPO, BLM, City and Nevada Copper. Nevada Copper will evaluate these sites per the MOU and mitigate (data recovery, recordation and collection and recovery of artifacts) impacts, if necessary, prior to any disturbance. The Pumpkin Hollow Property does not affect any Native American Reservation Lands or sacred sites.

20.5 Social or Community Impact

The Pumpkin Hollow Property occurs entirely within Lyon County, Nevada, which has historically the highest unemployment rate in the state. The Underground and Open Pit Projects are expected to bring approximately 800 to 900 direct and plus additional indirect jobs to the area.

A major element of the previous stages (advanced exploration and 6,500 stpd underground mine) of the Pumpkin Hollow Project development included obtaining approval of the SUP from the Lyon County Board of Commissioners. On June 11, 2013, the Lyon County Planning Commission recommended approval for the Stage I (6,500 stpd) underground mine by a unanimous vote. Subsequently, on June 20, 2013, the Lyon County Commission unanimously approved the County SUP for the Stage I Underground Project.

Approval of the SUP was a critical milestone of the Pumpkin Hollow Project and are notable in that they confirm that there is strong, local support for the project. An additional SUP is not required for future stages of mine development at the Pumpkin Hollow Project since the Pumpkin Hollow Project is now located entirely within the City of Yerington.

20.6 Yerington Land Conveyance

The Underground and Open Pit Projects will be completed on 100% privately owned lands as a result of the Yerington Land Conveyance. Congress passed “Conveyance of Lands to Yerington, Nevada” in Section 3009(a) of the Carl Levin and Howard P. ‘‘Buck’’ McKeon, National Defense Authorization Act for Fiscal Year 2015 (the “Act” H.R. 3979; Public Law 113-291) on December 19, 2014. The Conveyance transferred 10,050 acres of BLM-administered federal land to the City of Yerington (City). Subsequently, the City re-conveyed 9,145 acres to Nevada Copper. Combined with Nevada Copper's 1,538 acres of leased private land, the Pumpkin Hollow Project includes a total 10,683 acres of private mineral and surface rights for future development, including the mine and related facilities.

The Act allows the City to partner with Nevada Copper to develop approximately 11,600 acres of Pumpkin Hollow Property lands and land surrounding the Pumpkin Hollow Property owned or controlled by Nevada Copper and the City. The lands conveyed by the Act can also be used for industrial, commercial, infrastructure and recreation purposes that will create sorely needed jobs and economic development for Yerington.

The City has already zoned Nevada Copper lands not needed for mining as “M-1, General Industrial.” It has also been awarded a grant from the Nevada Governor’s Office of Economic Development to conduct a preliminary study on potential economic uses on Nevada Copper and City lands outside of mining areas for other economic development including commercial, industrial, a law enforcement training facility and recreational events center. Currently Yerington hosts the largest country music festival in Nevada, “Night in the Country,” which is attended by approximately 7,500 attendees in July every year on temporary facilities. The City envisions creating a larger, permanent facility on the 913 acres it acquired as part of the land conveyance approximately two miles west of mining operations that could host multiple, large festivals. It would include a permanent outdoor venue and stage, a smaller all-season indoor events facility, campgrounds, City water and sewer, permanent electrical infrastructure, a swimming pond, a motorcycle (motocross) course, mountain bike and hiking trails. The study will include preliminary assessment of infrastructure needs, including water, sewer, roads and utilities.

Though these other proposed uses are not part of the Pumpkin Hollow Project, much of the infrastructure for the mine can be utilized to serve these facilities. In particular, Nevada Copper may receive a return on its investment from a sale of land that is not needed for mining but could be disposed of for the commercial and industrial development. Approximately 500 to 1,000 acres of Nevada Copper land, directly adjacent to, and contiguous with, the proposed mining facilities, are not expected to be needed for mine operations and could be available for economic development for the City and County. Nevada Copper continues to work with the local community to explore and augment economic opportunities for the area.

Finally, development of other commercial, recreational or industrial uses on Nevada Copper properties allows mine management to begin site planning upon completion of mining. The post-mining strategy includes conversion of Underground and Open Pit Project facilities to other uses to reduce long-term financial assurance and to minimize economic impact to the Yerington community and closure and reclamation of facilities that cannot be converted. The mine and post-mine development strategy provides a means to sustain local employment, maximize the use of utility assets and recognize economic benefits. This is part of long-term sustainable development goals.

20.7 Permitting Requirements

The need for and status of certain primary state and local permits and approvals are reviewed in the following subsections.

20.7.1 Primary Permits

The permitting strategy for the Pumpkin Hollow Project has been to include both the underground and open pit operations in each permit. As project facilities are designed, these permits are updated and modified to address the location, size and extent of each facility. All permits are required to update or renew at a minimum of five years for Water Pollution Control and Air permits or three years for Reclamation Permits or whenever there are additions to or changes in permitted facilities.

The following primary permits, and status of each, that are required for both underground and open pit mining and ore processing operations include:

- Nevada Division of Environmental Protection (NDEP), BMRR is the primary State agency regulating mining.
- Water Pollution Control Permit (WPCP) 2008103 that includes underground mining, open pit mining and associated surface facilities; approved August 23, 2013; re-approved with modifications January 16, 2015.
- WPCP 2008109 for mine (shaft and underground) dewatering to include all (site-wide) dewatering for mining operations; approved on October 13, 2012; Re-approved with modifications on February 20, 2015.
- NDEP Reclamation Permit #0288 to include reclamation of underground mine and processing facilities; approved on June 7, 2013; re-approved with modifications November 7, 2014; includes approved financial assurance (bond) of $5.3 million for underground facilities; pending approval for ~$6 million for final design of underground facilities; financial assurance for open pit facilities to be determined upon approval of final-designed facilities and modification of the permit to include those facilities.

Nevada Division of Environmental Protection (NDEP), BAPC:

- Class II Air Quality Operating Permit AP1021-3369; approved on September 5, 2013; re-approved with modifications July 30, 2015; to be modified to include final design details for the open pit facilities

Lyon County:

- SUP for underground mining (6,500 stpd); approved on June 20, 2013 by the Lyon County Board of Commissioners; superseded by annexation into the City, no longer required.

City of Yerington:

- Annexation into the City of Yerington and zoned M-2 – Special Industrial Mining. Approved September 28, 2015.

20.7.2 Permit Status

Nevada Copper has received the majority of permits required to mine with an underground or open pit mine and associated process facilities. A 5,000 stpd Underground Project as proposed in this Report is within the throughput limits of those permits. Final layouts, locations and parameters of permitted components, including process solution containment, land disturbances, emission sources and other pollution control facilities have been confirmed for the Underground Project and the permit renewals and modifications have been submitted to the respective state regulatory divisions and bureaus. All of these are classified as modifications to existing permits and can be reviewed and approved within 3 to 6 months. Nevada Copper plans to submit all updates and modifications to permits upon completion of design of the Open Pit Project facilities.

Table 20-2 shows the status of Nevada Copper's mine permitting efforts to date.

Table 20-2: Status of Mine Permitting Activities

Agency / Description, Name and Date of Event	Effective Date	Expiration Date	Explanation and/or Source File
City of Yerington - Zoning and Special Use Permit			
U.S. Government Patent to the City of Yerington (10,059 acres)	August 20, 2015	None	YLC-LyonCountyRecorded_Patent_27-2015-0047_ValidExistingRightLetters.pdf
Deed from City of Yerington to Nevada Copper Inc. (9,145.4 acres)	October 12, 2015	None	YLC-MinCoRecParcel3ConformedCcDeed20151014.pdf
Annexation into City and Master Plan Amendment	October 12, 2015	None	Yerington-5-23-2016 Minutes- Zoning M2-approval.pdf & 'CoY-M-1 M-2 Zone Chapt 7-2015.pdf'; Zoned M-1 Industrial (including mining & solar) & M-2 (commercial & solar)
Yerington Municipal Code, Title 10, Chapter 7 amended Article B: M-2 Special Industrial District for Mining, Mineral Processing & Related activities	May 23, 2016	None	YER-Ord #16-03 M-2 ZoningApproved20160523.pdf
NDEP-BMRR-Regulation Branch - Water Pollution Control Permit (WPCP) Mining Facilities WPCP 2008103; NRS 445A; Water Pollution Control; NAC 445A.350-447; Mining Facilities			
Advanced Exploration (Shaft Development)			
Application submitted to NDEP	June 19, 2010		
NDEP Completeness Review Received	July 9, 2010		
NCI Response to Completeness	July 14, 2010		
Technical Review Q3-4 2010	July 15, 2010		
WPCP 2008103 Permit Approved	March 25, 2011	Superseded	Superseded by Underground Mine (Stage 1) and Open Pit (Stage 2 Open Pit or Integrated Operations Open Pit)
Underground Mine (Stage 1) and Open Pit (Stage 2) or Integrated Operations OP/UG/SPF			
Submit Major Modification for Stage I & Stage 2	July 5, 2012	NA	IP-App-E-WPCP-Narrative-mpd-20130225.pdf
Approval of EDC for five (5) additional dewatering wells	February 26, 2014	NA	WPCP103-DWsApproved20140226pie.pdf
Submitted Application for Two Minor Modifications (Proposed/Existing) for final engineered configuration of process components	March 19, 2014	NA	WPCP103-MinorModApp20140318mpd.pdf WPCP103-ExistingMinorModApp20140318mpd.pdf

Agency / Description, Name and Date of Event	Effective Date	Expiration Date	Explanation and/or Source File
Revised Permit Issued	January 16, 2015	September 17, 2018	WPCP103-Permit20150116js.pdf Expect to prepare and submit application for modification and renewal when UG4K2017 PFS design is complete (July-August 2017)
Mine Rock Management Plan Submitted Schedule of Compliance (SOC) Item	April 22, 2015	Goes with Permit	WPCP103-Mine-Rock-Management-Plan-20150422mpd.pdf
Submitted Application for Modifications for final engineered configuration of underground process components	Pending	Pending	WPCP103-Renewal-Amendment-20180727mpd
NDEP-BMRR-Regulation Branch; Water Pollution Control Permit (WPCP) Infiltration Facilities WPCP 2008109 NRS 445A; Water Pollution Control; NAC 445A.228-263; Discharge Permits and WTS-3 Guidance Document For An Application For Rapid Infiltration Basins.			
All Phases - Advanced Exploration (Shaft Development) Underground Mine (Stage 1) and Open Pit Open Pit (Stand-alone Stage 2 Open Pit or Integrated Operations Open Pit)			
Revised Complete Submission	July 23, 2012	NA	AE-App-FWMP20120723mpd.pdf
Settling Tank Upgrade Approved	April 24, 2014	NA	WPCP109-SettlingTankAs-builtApproved20140612js.pdf
Arsenic Variance Submitted	May 12, 2014	NA	WPCP-Arsenic-Variance-20140512mpd.pdf
Arsenic Variance Approval	In Progress		
Flocculent Pilot Test Requested	November 3, 2014		WPCP109-EDC-FlocTreatment System-20141105mpd.pdf Added to Renewal and Major Modification Application (see WPCP109-2015Application-Renewal-Major-Mod-20151204mpd below)
Flocculent Pilot Test Approved	August 14, 2015		Added to Renewal and Major Modification Application (see WPCP109-2015Application-Renewal-Major-Mod-20151204mpd below) WPCP109-Floc-Blocks-Approved-20150814sg
Acid and Aeration Pilot Test Requested	January 9, 2015		WPCP109-PilotTestingpHAeration20150109mpd.pdf
Acid and Aeration Pilot Test Approval	Not submitted		After suspension of shaft sinking this test program to treat dewatering water was suspended. Preliminary lab tests performed; NDEP application not submitted; pilot test required
Revised Permit Issued	February 20, 2015		WPCP109-Permit-20150220Holmgren.pdf
North Basins Upgrade Approved	February 23, 2015	Goes with Permit	WPCP109-North-Basins-Approved-Asbuilt-20150223mpd.pdf; WPCP109-NorthBasinsRedesignApproved20140624js.pdf
Application - Major Modification	December 1, 2015		WPCP109-2015Application-Renewal-Major-Mod-20151204mpd Technical review underway
Completeness - Major Modification to increase infiltration limits to 1.548M GPD	May 11, 2016		WPCP109-MajorModRenewal-Admin-Review-Completeness-20160510mpd Administratively Complete
Technical Review - Major Modification	Expected July 2017		Under Review by BMRR
Approval - Major Modification	Expected Jan 2018		Pending review by BMMR Expected Completion 2017
Major Modification for additional Rapid Infiltration Basins (RIB's) South Ranch Basins	Pending	Pending	WPCP109-MajorModification-20181207mpd

Agency / Description, Name and Date of Event	Effective Date	Expiration Date	Explanation and/or Source File
NDEP-BMRR Reclamation Branch - Reclamation Plan and Permit #0288			
Advanced Exploration (Shaft Development)			
Initial Submission	June 19, 2010		Superseded by Underground Mine (Stage 1) and Open Pit (Stage 2 Open Pit or Integrated Operations Open Pit)
NDEP Technical Completion	October 1, 2010		Same as above
NOI To Issue Permit	February 9, 2011		Same as above
Reclamation Permit issued	March 25, 2011	Superseded	Same as above
Reclamation Cash Bond for $505,915 posted	June 20, 2011	Superseded	Same as above
Underground Mine (Stage 1)			
Stage I Stand-alone Underground Mine 2013 Configuration Reclamation Plan Submitted	11/10/203		App-I-Rec-PLUM-Standalone20130508.pdf
Modification Submitted	October 8, 2014		REC-S1-Minor-Mod-20141008mpd.pdf
Mod Completed	October 30, 2014		REC-S1RecTechResponseLetter-20141030mpd.pdf
Stage 1 Stand-alone Approved	November 7, 2014	Superseded	REC-Permit-20141107tp.pdf
Revised Financial Assurance of $5,364,055 approved, bond not changed	November 7, 2014	See revised financial assurance below	REC-Bond-Approval-Letter-20141107tp.pdf
Underground (UG) and/or Open Pit (OP) Mine and/or Integrated Project			
Open Pit Project Major Modification Submitted	March 26, 2015		REC0288-Reclamation Plan-20150706mpd.pdf
Financial Assurance of $17,623,768 Approved	June 24, 2015		NDEP-RecBond201506tpmajormodPNbondltr Total bond for the combined open pit & underground operations for the first three years of the project.
Public Notice Period for Major Modification Completed	June 24, 2015		NDEP-RecPermit0288-201506jbNOI_WebPostPumpkinHollow
Integrated UG/OP Permit Issued	July 30, 2015		NDEP-RecPermit0288-201507tp7-30-15FinalPermit
Integrated UG/OP Permit (Revised) Issued	January 21, 2016	Life of Mine Review bond at a minimum every three years	REC0288-Permit-20160121bmrr.pdf Revised permit issued to approve temporary reduction in financial assurance (reclamation bond). Bond must be revised whenever for changes in projected reclamation costs
Revised Financial Assurance Temporary Reduction from $5,364,055 to $1,486,876	January 26, 2016		201601tpsuretyreductionbondltr.pdf; 201601jbBondRiderAccptDecrease.pdf Needs to be revised for all UG4k2017 project designs
Financial Assurance posted	January 28, 2016		SmithManusBondInvoice&Rider20160128.pdf
Submitted Application for modifications for final engineered configuration of underground process components	Pending	Pending	REC0288 Process Bond Minor Modification 20181127mpd
Class II Air Pollution Control Permit & Surface Area Disturbance AP1021-3369 NAC/NRS 445B Air Controls (Class 2 Permit for less than 100 TPY of any regulated pollutant [particulates] or 25 TPY hazardous air pollutants; PSD increment 30 micrograms/m3)			

Agency / Description, Name and Date of Event	Effective Date	Expiration Date	Explanation and/or Source File
Underground Mine (Stage 1)			
Surface Area Disturbance (SAD) application approved	September 13, 2012		AQ-SAD-Pemit-Acceptance-Letter-jd-120913.pdf
Minor Modification for 2013 configuration of UG facilities and increase of SAD Acreage to 500	April 1, 2014		BAPC-ClassII-Revision-Application20140409jps.pdf
Minor Modification Approved	July 21, 2014	Superseded by	AP1021-3369 BAPC NV Copper Permit REV July 17 2014.pdf
Submission of application to BAPC	May 7, 2015		Final NV Copper Air Permit Application 05062015.pdf
Draft Permit Issued	July 6, 2015		Draft AP1021-3369 NV Copper REV July 6 2015.pdf
Permit Issued	July 30, 2015		073015 A0944 - Nevada Copper - AP1021-3369 REV Permit.pdf
Submitted Application for modifications for final engineered configuration of underground process components	Pending	Pending	Final Permit Application Submittal20181101ah
NDWR - State Engineer Dam Permit (if a tailings dam or pond >20 acre ft or >20 ft high) NRS NAC 535 Dams and other Obstructions - All water management basins planned are smaller than the dam height (20 feet) or volume (20 ac-ft) requiring a dam permit.			
Notifications of construction or alteration for all existing basins (as of 20 March 2014) submitted.	March 20, 2014		Additional notifications will be sent as additional RIBs or other small ponds are designed. No approvals required for these small facilities.
NDEP-Bureau of Water Pollution Control - Onsite Sewage Disposal System (OSDS) Permit NRS 445A; Water Pollution Control, NAC 445A.228-63; Discharge Permits			
Submitted	June 1, 2012		OSDS-App-3,000 Gal Onsite Sewage Disposal System GNEVOSDS09S0072-111201-gmf.doc.doc
Approved	August 28, 2012		OSDS-App-Approval to Construct-3,000 Gal Onsite Sewage Disposal System GNEVOSDS090072-111201-al.pdf.pdf
Wastewater treatment facilities anticipated for 5,000 stpd UG mine. New applications will be needed.	Target receipt of permit Jan 2018		Pending design of UG4k2017 project
NDEP- Bureau of Safe Drinking Water - Permit to Operate Privately Owned Water System (Non-Transient, Non-community Public Water System NTNCPWS) NRS 445A; Water Pollution Control NAC 445A.595-6731;		Privately owned public water system; Per NRS 445A.829 a "Non-transient water system" means a non-community water system that regularly serves at least 25 of the same persons for > than 6 months per year.	
Initial Sanitary Survey (Drinking Water)	July 25, 2013		NCI - Sanitary Survey Task List for Shaft Site.msg
Response to Initial Sanitary Survey	November 19, 2013		Pumpkin Hollow Permit App.pdf
Revised Plans submitted	December 4, 2013		NVCOPPER_Plans_12042013.pdf
Response to Technical Comments	February 26, 2014		State Response Letter 1.doc State Response Letter 2.doc State Response Letter 3.pdf
Supplementary Sanitary Survey (Anticipated)	July-September 2017		Subsequent to design of project facilities
Response to Supplemental Sanitary Survey (Anticipated)	July-September 2017		Subsequent to design of project facilities
Revised Engineering Plan (Anticipated)	July-September 2017		Subsequent to design of project facilities
Response to Technical Comments (Anticipated)	July-September 2017		Subsequent to design of project facilities
Permit Issued (Anticipated)	Target receipt of permit Jan 2018		Subsequent to design of project facilities

Agency / Description, Name and Date of Event	Effective Date	Expiration Date	Explanation and/or Source File
Spill Prevention, Control and Countermeasures Plan (SPCC)			
Included in WPCP NEV2008103	Goes with WPCP103	Goes with WPCP103	WPCP103-App-L-SPCC-20141216jps.doc Is revised regularly to accommodate changes in facility design; Must be prepared, kept current and on file at site;
NDEP-BWPC - Stormwater General Permit Stormwater Pollution Prevention Plan (SWPPP) 40 CFR §122.26(b) (14).			
Site SWPPP is approved under Statewide General Permit. Requires Renewal every five years.	Renewed June 13, 2013	February 28, 2018	SWPPP-364-Electronic20130923mpd.pdf SWPPP-364-PermitRenewal20130613mr.pdf NCI implements Best Management Practices (BMP's) for stormwater
Items listed below are no longer applicable and/or not required with change in land status with completion of the Yerington Land Conveyance in October 2015			
BLM Carson City Sierra Front Field Office - Plan of Operations 43 CFR 3809 Surface Management Regulations of public lands by operations authorized by the mining laws			
Kickoff Meeting	October 20, 2014	NA	
Plan of Operations Withdrawn	January 28, 2015	NA	
Not necessary as there is no longer a federal project nexus.	NA	NA	
U.S. Army Corps of Engineers - Clean Water Act 404 33 CFR3 23	NA		Not required. There are no jurisdictional Waters of the U.S. on the project area See CWA-404EvalOpinion20130117LAC.pdf

20.7.3 Other Permits

Since the land conveyance was successful, federal permitting, such as the Plan of Operations through the U.S. BLM and an Environmental Impact Statement (EIS), are no longer required.

20.8 Mine Closure

The State of Nevada has specific reclamation and closure requirements for mining projects, as outlined in the Nevada Administrative Code (NAC) Section 519A and 445A and Nevada Revised Statutes (NRS) 519A and 445A, as well as provisions for dust control in NAC 445B. In accordance with these requirements, reclamation designs at the Underground and Open Pit Projects incorporate local climatic conditions, vegetation communities, and technical and economic practicability of reclamation to generate reclamation plans that include concurrent reclamation, revegetation of disturbed areas, re-contouring and erosion control to achieve features that are stable compared to adjacent areas, meet approved post-mining land use and prevent contaminants from degrading waters of the State of Nevada.

Project-specific disturbance and reclamation areas, reclamation schedule, reclamation approaches and a summary of reclamation costs are presented in the following subsections.

20.8.1 Disturbance and Reclamation Areas

The area within the Pumpkin Hollow Project perimeter fence is approximately 6,700 acres. Of this area, a total of approximately 3,700 acres will be disturbed as part of mining operation. A portion of this area will not be reclaimed, including the North and South pits, permanent water management diversion channels, and selected infrastructure that will be retained for post-mining industrial use. A total area of approximately 3,000 acres will be reclaimed, including the MRSFs, DST facility, reclamation material stockpiles, infrastructure that will be removed at closure, and water management features that will be reclaimed at closure.

20.8.1.1 Reclamation Schedule

Three major planning periods are used to describe mine reclamation and closure activities:

- Pre-production: years prior to commencement of production, Year -2 through Year -1. Reclamation activities to be completed in this period include constructing revegetation and erosion test plots on overburden material stripped from the pit footprints.
- Production: years the mine and processing facilities are active. Production for the Open Pit mine is planned to last 19 years (Year 1 through Year 19). Reclamation activities to be completed in this period include salvaging plant growth medium (PGM) from the footprints of the MRSF and DST, concurrent and final reclamation and revegetation of the MRSF including construction of surface water management channels, concurrent reclamation and revegetation of the DST including construction of surface water management channels, interim reclamation of the PGM stockpiles, and monitoring and maintenance of reclaimed areas.
- Post-production: years between cessation of mine and processing activities and final bond release. The post-production period is divided into periods as follows:
  - Closure: years of major reclamation and closure activities (Years 20 and 21);
  - Post-closure: years of site monitoring and maintenance between the closure period and final bond release (Years 22 through 31). The post-closure period will end when the reclamation performance bond is released.
  - Post-mining: years following final release of the reclamation performance bond.

Reclamation and closure activities to be completed in the post-production period include final reclamation and revegetation of the DST, decontamination, decommissioning and removal of process equipment to facilitate post-closure use of structures, demolition and debris consolidation in an on-site demolition debris disposal area, reclamation and revegetation of haul roads and ancillary areas, sale of mobile equipment fleet retaining value, removal of sediments from basins and channels, monitoring well abandonment, and monitoring and maintenance of reclaimed areas.

20.8.1.2 Reclamation Approaches

Closure Covers and Revegetation

The major reclamation approach used at the Pumpkin Hollow Project includes regrading features (DST and MRSFs) to stabilize slopes, construct surface water management features, place a closure cover, and revegetate features. For infrastructure and process facilities, prior to transfer of facilities to future owners for post-mining industrial use, mining and process equipment will be decommissioned and removed, and hazardous compounds and left-over chemicals and reagents disposed of properly.

Closure covers will be placed over Pumpkin Hollow Project facilities to stabilize erosion and facilitate revegetation of facility surfaces. The type of closure cover used on facilities varies depending on the objectives that must be met by the cover. For the Pumpkin Hollow Project, three types of closure covers are used:

- Revegetated rock cover (with or without underlying suitable PGM)
- Revegetated PGM cover (with or without underlying suitable PGM)
- Direct revegetation cover

Revegetated rock covers will be placed on facilities with steep slope angles, such as the DST and MRSF side slopes, where erosion potential is greater. Revegetated PGM covers will be placed on facilities with shallow slope angles or short slope lengths, where erosion potential is lower, such as DST and MRSF tops, and yard areas. Direct revegetation will be conducted in areas where the surface material at a facility is classified as suitable PGM, such as pipeline corridors and material generated from pit overburden stockpiles.

Closure covers to be placed on Pumpkin Hollow Project facilities are outlined in Table 20-3.

Table 20-3: Facility Closure Covers

Facility	Cover Depth (ft)	Notes
DST	1.5	Revegetated PGM cover, subgrade assumed to be unsuitable PGM
DST Berm / Ditch System, Fuel Farm, Roads	1	Revegetated PGM cover, subgrade assumed to be suitable PGM, HDPE lined ditch portion of DST Berm / Ditch System will not receive a closure cover
Surface Mine MRSF	1	Revegetated rock cover on side slopes and revegetated PGM cover on top, subgrade assumed to be suitable PGM
Ore Stockpiles, On-site Closure Demolition Debris Disposal Area	2	Revegetated PGM cover, subgrade assumed to be unsuitable PGM
Pit Berm, Pipeline Corridors, Magazine	0	Direct revegetation cover

Revegetation will be conducted on all closure covers. Revegetation activities include deep ripping subgrade, placing and grading cover, and seeding. Natural soils in the Pumpkin Hollow Property area are very poorly developed and climatic conditions are such that native or even reclaimed desired plant communities must be adapted to those conditions. Fertilizer application, erosion-control material placement (including straw mulch and erosion-control fabric), and periodic herbicide application to control weeds will be used only where deemed necessary and beneficial. Seeding is conducted to aid in long-term site stabilization, control erosion and off-site sediment transport, control dust resulting from exposed soils, improve visual aesthetics, limit invasive plant species from establishing and proliferating, and contribute to future site land uses. Seeding is conducted using species that are native to the area, and preference is given to species that are present on site prior to construction activities. For closure covers on large facilities such as the DST facility and MRSF, a patchwork of small, shallow dozer basins will be graded into the cover surface. These dozer basins will attenuate peak flows and reduce runoff velocity, partially capturing sediments entrained in surface runoff from the facility. These basins are assumed to enhance soil water storage and uptake by plants, and create micro-climactic variability on re-contoured slopes to enhance vegetation diversity.

Water Management in Reclamation

Throughout production and closure, process water, surface water and groundwater will be managed. At closure, process plant fluids will be managed as part of the decommissioning process. Additional process fluids may occur as seepage from the DST facility foundation drains, which are then collected in the DST seepage basin. These fluids will be recycled to the process plant during the production period. During the closure period, seepage will be routed into the North Pit Lake until the quantity of seepage diminishes to an amount that may be evaporated in the seepage collection basin SP-2. It is estimated that seepage will diminish to an amount that may be evaporated in SP-2 within 10 years following DST reclamation. Seepage management by evaporation will be conducted throughout the post-mining period. Based on the net evaporative environment and surface water control, seepage is not expected to occur from the MRSF.

To control stormwater and limit erosion and sediment transport from disturbed areas during the pre-production, production, and post-production periods, stormwater management, erosion, and sediment control BMPs will be employed as appropriate, including:

- Diversion of stormwater run-on away from mine facilities to prevent stormwater contact with disturbed areas
- Construction of erosion control berms around feature perimeters
- Placing silt fences and straw bales around the perimeters of disturbed areas
- Placing erosion control fabric on slopes during revegetation establishment
- Site grading to route stormwater to constructed channels (i.e., diversion channels, terrace channels and down-chute channels)
- Construction of runoff collection basins
- Protection of natural channels at intersections of constructed stormwater management features and natural drainage channels

During operations, the Open Pit mine will be dewatered. Dewatering water will supply a portion of the water required for the process plant, with the remaining dewatering water managed through infiltration in the WMBs, also known as RIBs. At closure, groundwater will pool in the North and South pits creating two pit lakes, which are anticipated to act as hydrologic sinks not requiring treatment or management.

Feature-Specific Reclamation

Feature-specific reclamation approaches and anticipated schedule for completion are presented in Table 20-4.

Table 20-4: Pumpkin Hollow Reclamation Approaches

Facilities	Reclamation Year(s)	Facility Names	Reclamation Approach
Surface and Process Water Facilities			
Diversion Channels	20	East Diversion, West Diversion, South Diversion, North Diversion, Center Diversion, Existing Diversion	Remove sediments
Diversion Ditches	20, 30	DST Runoff Collection, SP-2: DST Seepage Collection	Remove sediments
DST Runoff Basin	20	MSW-4	Remove sediments, test for PGM suitability, use on site or dispose of at MRSF or off site, regrade to breach embankment, reinforce channels, revegetate
Permanent MRSF Stormwater Basins	15	MSW-5, MSW-6	Remove sediments, test for PGM suitability, use on site or dispose of at MRSF or off site, leave basin in place
MRSF Stormwater Basins	15	MSW-7, MSW-8, MSW-9	Remove sediments, test for PGM suitability, use on site or dispose of at MRSF or off site, cut and fold liner for in-place disposal, regrade, revegetate
Water Management Basins	20	WMBs	Remove sediments, grade perimeter berm to fill basin depression, deep rip subgrade, revegetate
Secondary Containment Facilities	21	SC-3, SC-4	Remove sediments, cut and fold liner for in-place disposal, grade perimeter berm to fill basin depression, deep rip subgrade, revegetate
Process Water Management	20	Process Fluids	Process fluids decanted from the filtration plant will be evaporated in secondary containment basins. Solids (evaporite and sediments) will be tested and disposed of properly as solid waste or hazardous waste as appropriate.
Test Plots			
Test Plots	-1, 1, 5, 10, 15, 20	Revegetation & Erosion Test Plots, Closure Cover Test Fill Plots	Construct test plots to mimic DST and MRSF slopes, monitor to determine efficacy of reclamation approach
Stockpiles			
PGM Stockpiles	2, 11, 14, 15, 20, 21	PGM-2, PGM-3: Concurrent and Final Reclamation	Construct erosion control berm, grade remaining pile to blend with existing topography, rip surface to prepare subgrade, revegetate

Facilities	Reclamation Year(s)	Facility Names	Reclamation Approach
Dry Stack Tailings			
DST Side Slopes and Top	11, 15, 20	DST Concurrent and Final Reclamation	Regrade tailings, place 1.5 ft thick PGM cover, dozer basins to prevent erosion and promote revegetation, construct crest perimeter berm, revegetate
DST Surface Water Management	11, 15, 20	DST Terrace Channels	Excavate channel – 3 ft bottom width, 3 ft depth, 3:1 side slopes, install 16 oz/yd2 nonwoven geotextile, place D50 = 6-inch plus rockfill 1 ft thick
	20	DST Downchutes	Excavate channel - 10 ft bottom width,7 ft depth, 2:1 side slopes, install 16 oz/yd2 nonwoven geotextile, place D50 = 24-inch plus rockfill 3 ft thick
DST Perimeter Berm	21	DST Perimeter Berm	Rip surface to prepare subgrade, place 1 ft thick PGM cover on side slopes and berm road, revegetate
Mine Rock Storage Facility			
MRSF	6, 10, 14	MRSF Slopes	Install erosion and sediment BMPs (silt fence & hay bales), regrade slopes from 1.3H:1V slopes 2.5H:1V with terraces for an overall slope of 3H:1V, rip subgrade, place 1 ft revegetated rock cover on sides and 1 ft revegetated PGM cover on top, dozer basins to prevent erosion and promote revegetation, revegetate
MRSF Surface Water Management	6, 10, 14	MRSF Terrace Channels	Excavate channel – 4 ft bottom width, 3 ft depth, 3:1 side slopes, install 16 oz/ yd2 nonwoven geotextile, place 6 in plus rockfill 1 ft thick
	6, 10, 14	MRSF Downchutes	Excavate channel – 10 ft bottom width, 7 ft depth, 2:1 side slopes, install 16 oz/ yd2 nonwoven geotextile, place D50 = 24 in plus rockfill 3 ft thick
Demolition Debris Disposal Area			
Demolition Debris Disposal Area	21	Demolition Debris Disposal Area	Install erosion and sediment BMPs (silt fence & hay bales), place 2 ft thick PGM cover, blend cover material, dozer basins to prevent erosion and promote revegetation, revegetate
Pits			
North and South Pit	20	Pit Safety Berm	Construct pit safety berm, offset 100 ft from pit perimeter, constructed 10 ft high, 2:1 side slopes, 5 ft top bench
Infrastructure			
Miscellaneous Buildings	20	Magazines	Rubblize reinforced concrete foundations
	20	Main Substation, Fresh Water Standpipe, Fire Water Storage Tank, Security Gate, Administration Building, Wastewater Treatment Plant, Potable Water Treatment	Leave in place for post-mining beneficial land use
Process Facilities	20	Concentrate Thickener, Concentrate Standpipe, Pebble Crusher	Remove fabricated items, load and haul non-uniform demolition debris to on-site demolition debris disposal area, decontaminate and evaporate decontamination fluids, demolish thickener, standpipe, and crusher and haul to on-site demolition debris disposal area, rubblize concrete foundation

Facilities	Reclamation Year(s)	Facility Names	Reclamation Approach
Process Facilities	20	Process Plant, Truck Shop	Remove fabricated items, load and haul non-uniform demolition debris to on-site demolition debris disposal area, decontaminate and evaporate decontamination fluids, leave structure in place for post-mining beneficial land use
	20	Process Water Storage Tank, Substation	Leave in place for post-mining beneficial land use
Tailings Filtration Plant	20	Tailings Thickener, Filtration Plant	Remove fabricated items, load and haul non-uniform demolition debris to on-site demolition debris disposal area, decontaminate and evaporate decontamination fluids, demolish thickener and haul to on-site demolition debris disposal area, filtration plant structure left in place for post-mining beneficial land use
	20	Substation, Oil/Water Separator	Leave in place for post-mining beneficial land use
Fuel Farm Yard	20	Fuel Farm Yard	Deep rip subgrade, place PGM cover 1 ft thick, revegetate
Process Plant Yard	20	Process Plant Yard	Excavate contaminated soils, load into lined/gasketed trucks, haul to appropriate landfill
Fences	24	3-Strand Barbed Wire Fences, Chain Link Fences	Remove fences
Seepage Collection Pumping	20 - 30	Seepage Collection Pumping and Pipeline	Replace seepage control pump, realign process water pipeline to drain into North Pit, operate and maintain pump and pipeline for 10 years after operation until seepage may be managed by passive evaporation in seepage basin SP-2
Pipelines	20, 30	Tailings Slurry and Process Water Pipeline	Remove pipe, grade to blend into existing topography, revegetate
	20	Dewatering Pipeline	Remove pipe, revegetate
Pit Magazines	20	Pit Magazines	Deep rip subgrade, revegetate
Ore Stockpiles	20	Ore Stockpile Footprints	Place 2 ft PGM cover, deep rip subgrade, revegetate
Temporary Tailings Stockpile	20	Temporary Tailings Stockpile	Excavate 2 ft of material underneath facility footprint and place on DST, regrade to drain, deep rip subgrade, revegetate
Conveyors	20	Conveyors	Remove conveyors, rubblize concrete foundations and haul to on-site demolition debris disposal area
Monitoring Wells	-2, 1, 30	Monitoring Wells	Abandon monitoring wells as appropriate by perforating casing, filling wells with bentonite, and capping the tops of wells with concrete
Roads			
Haul Roads	21	Haul Roads	Grade to blend into existing topography, rip surface to prepare subgrade, revegetate
Monitoring and Maintenance			
Monitoring	1 - 30	Concurrent and Post-closure Monitoring	Monitor reclaimed facilities to evaluate and customize reclamation approach. Ensure covers, slopes, and revegetation are stable and meet applicable requirements for bond release
Maintenance	-2 - 27	Cover and Revegetation Maintenance	Conduct maintenance work to repair unsuitable covers, restore unstable slopes, and treat and re-seed unsuccessful revegetation

20.8.2 Reclamation & Closure Cost Estimate

The reclamation cost estimate for the Open Pit Project includes both capital and operating costs. Capital costs over the mine life include $91 million in expenditures (including bond fees and contingency) and $20 million in credits, resulting in a net cost of approximately $71 million. Operating costs include the cost to use mine-owned equipment to regrade the MRSF, and results in a cost of approximately $1 million over the mine life. General and administrative costs were also included in the operating cost estimate.

The cost estimate for reclamation of Open Pit Project facilities was developed from estimates of unit costs for various reclamation activities, multiplied by material take-off quantities obtained from reclamation design drawings. Unit costs were developed from vendor quotes, first principles, best professional judgment, and cost data handbooks. The reclamation bond was estimated using the State of Nevada SRCE workbook (version 1.4 with August 1, 2018, cost data).

20.8.3 Tailings Management

The proposed TSF will encompass an area of approximately 800 acres. The tailings stack geometry includes 25 ft wide benches at 80 ft vertical intervals for stability and erosion control, with inter-bench slopes of 2.75H:1V. The overall slopes of the TSF are 3H: 1V.

A phased construction approach will be adopted wherein the TSF will be constructed in stages to suit tailings storage requirements. The northernmost area of the TSF is closest to the plant site and will be developed first. Cell 1 will be constructed in two stages, west and then east, to optimize capital expenditure. Cell 1 covers approximately 1/3 of the full TSF footprint and has capacity to store up to 10 years of tailings production. To allow room for independent operation of the underground tailings, Cell 2 will be constructed at the southern limit of the TSF footprint, and has capacity for the next three years of operation. Cell 3 joins Cell 1 and Cell 2 together, and will be constructed after the underground TSF is full and is no longer required.

The staged development of the TSF is shown in Figure 20-1, Figure 20-2, and Figure 20-3. Figure 20-4 shows a profile of the tailings stack geometry at the end of mine life.

The phased development of the TSF footprint allows for deferred capital expenditure and a smaller area for environmental and dust management than if the facility were constructed over the full TSF footprint from the outset.

Figure 20-1: Tailings Stacking Plan End – Cell 1 (Tetra Tech, 2019)

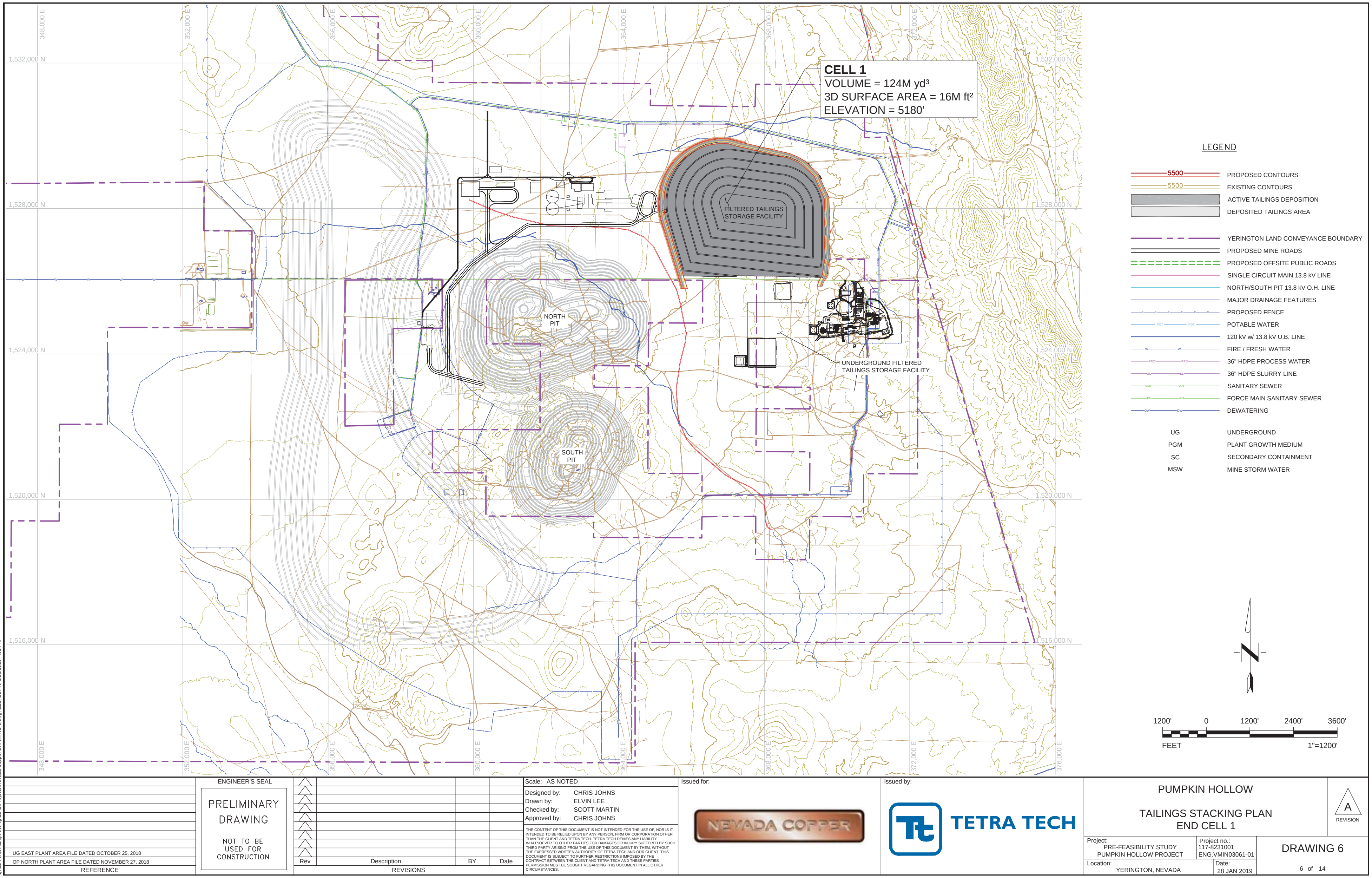
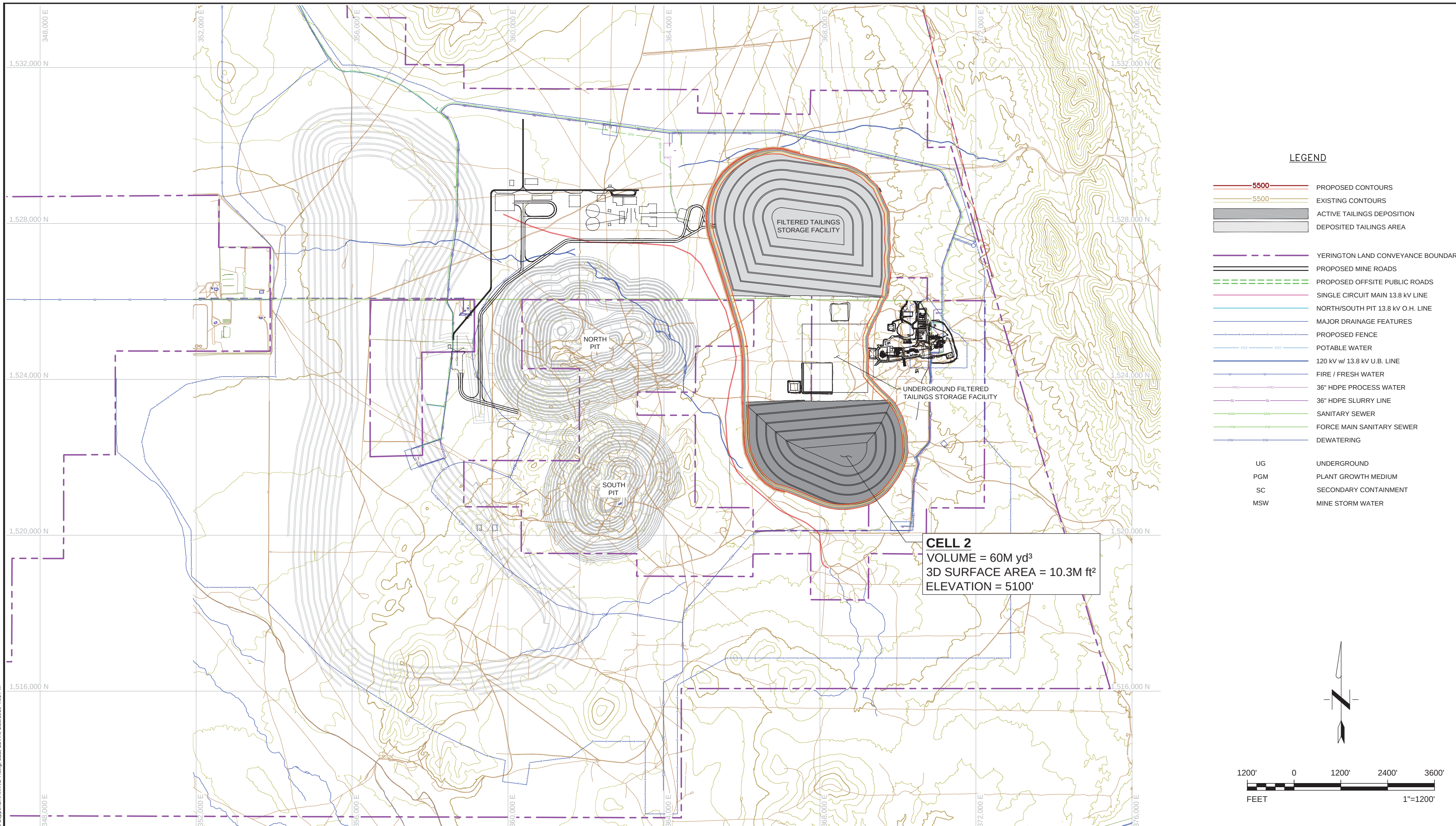


Figure 20-2: Tailings Stacking Plan End – Cell 2 (Tetra Tech, 2019)



Scale:	AS NOTED
Designed by:	CHRIS JOHNS
Drawn by:	ELVIN LEE
Checked by:	SCOTT MARTIN
Approved by:	CHRIS JOHNS

UG EAST PLANT AREA FILE DATED OCTOBER 25, 2018
OP NORTH PLANT AREA FILE DATED NOVEMBER 27, 2018
REFERENCE

ENGINEER'S SEAL
PRELIMINARY DRAWING
NOT TO BE USED FOR CONSTRUCTION

Rev	Description	BY	Date

REVISIONS

Issued for: NEVADA COPPER
Issued by: TETRA TECH

PUMPKIN HOLLOW
TAILINGS STACKING PLAN END CELL 2
REVISION A

Project: PRE-FEASIBILITY STUDY PUMPKIN HOLLOW PROJECT
Project no.: 117-8231001 ENG.VMIN03061-01
Location: YERINGTON, NEVADA
Date: 28 JAN 2019
DRAWING 7
7 of 14

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Figure 20-3: Tailings Stacking Plan End – Cell 3 (Tetra Tech, 2019)

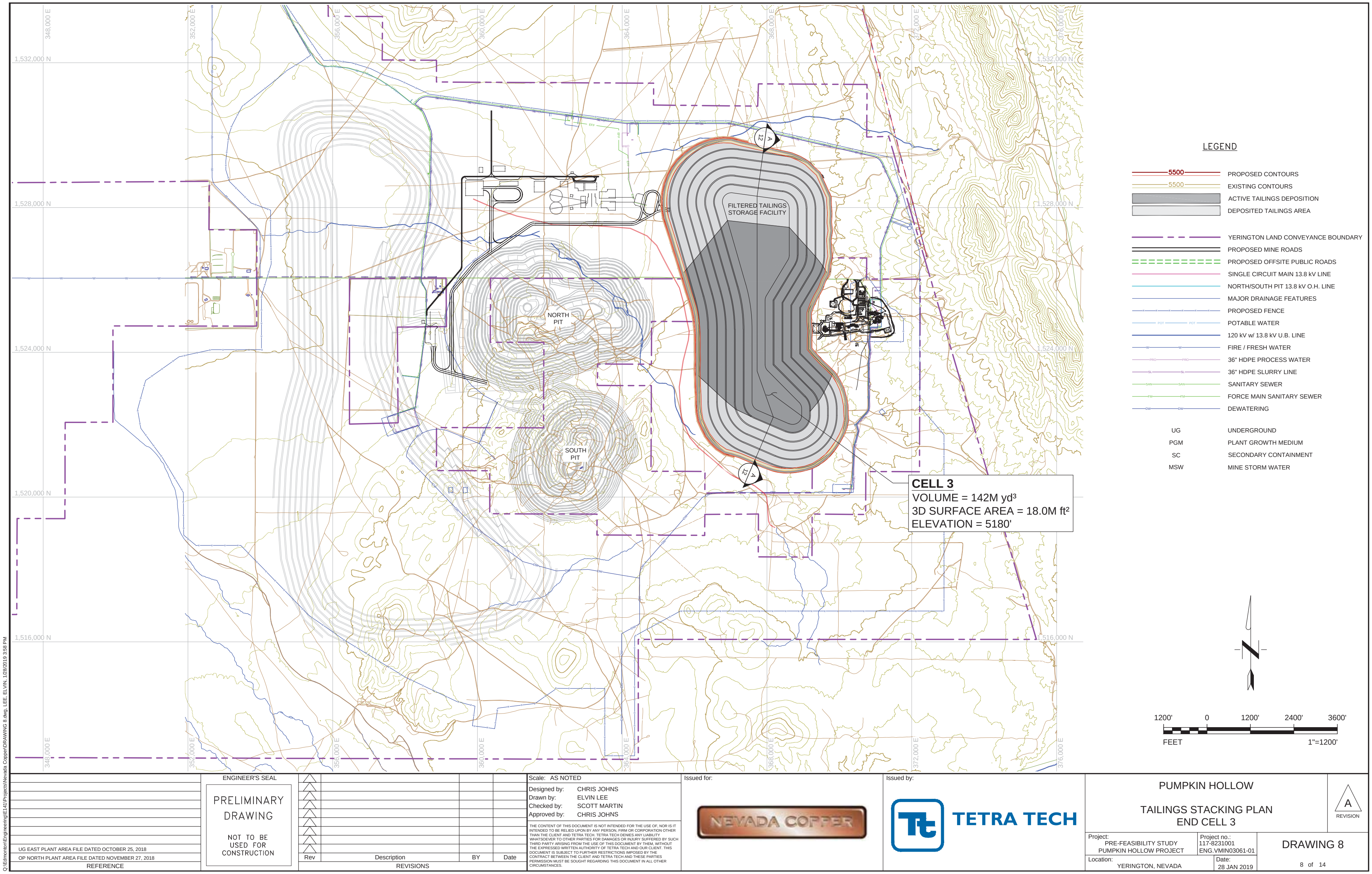
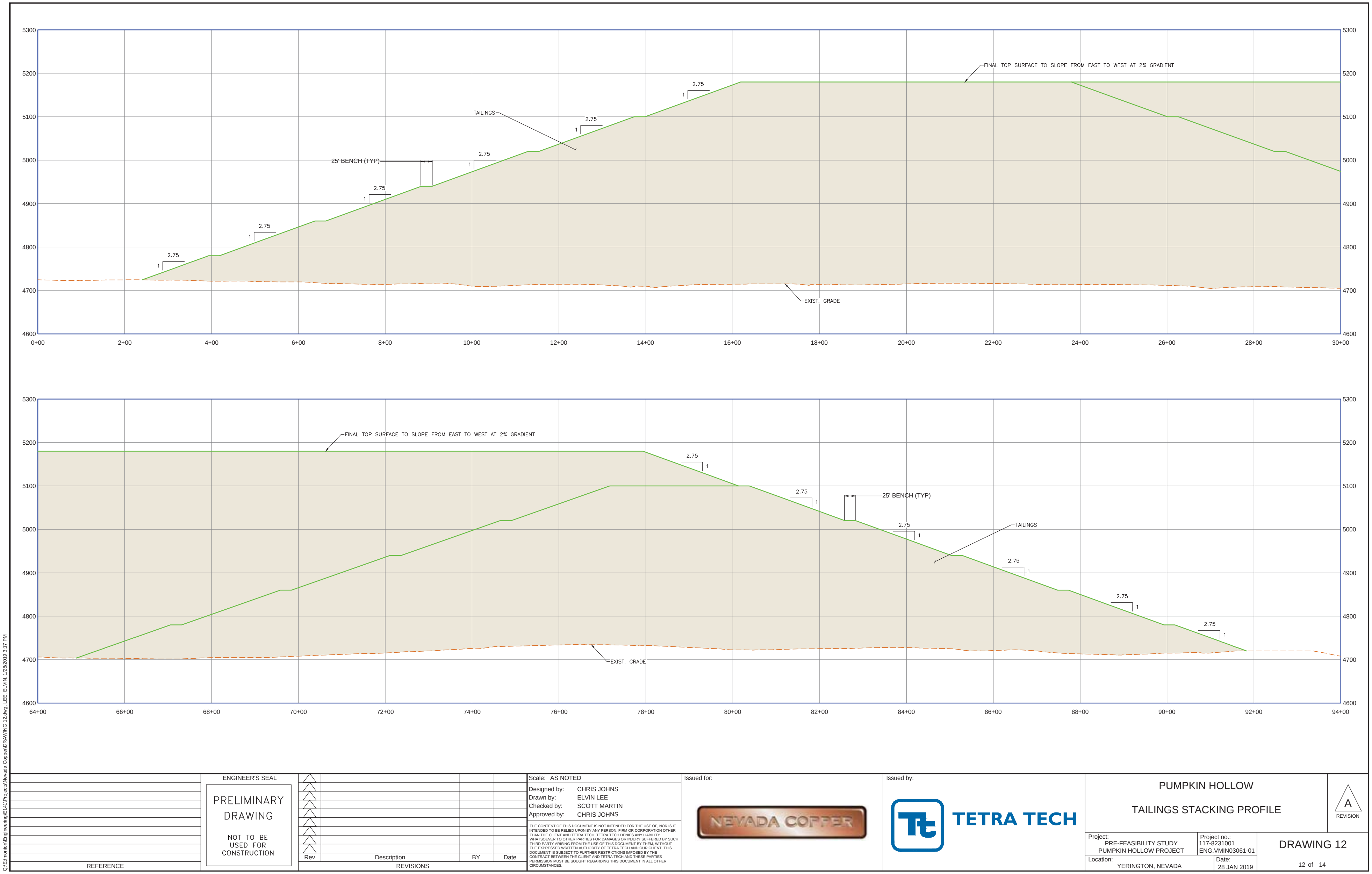


Figure 20-4: Tailings Stacking Profile (Tetra Tech, 2019)



20.8.4 Tailings & Process Water Containment

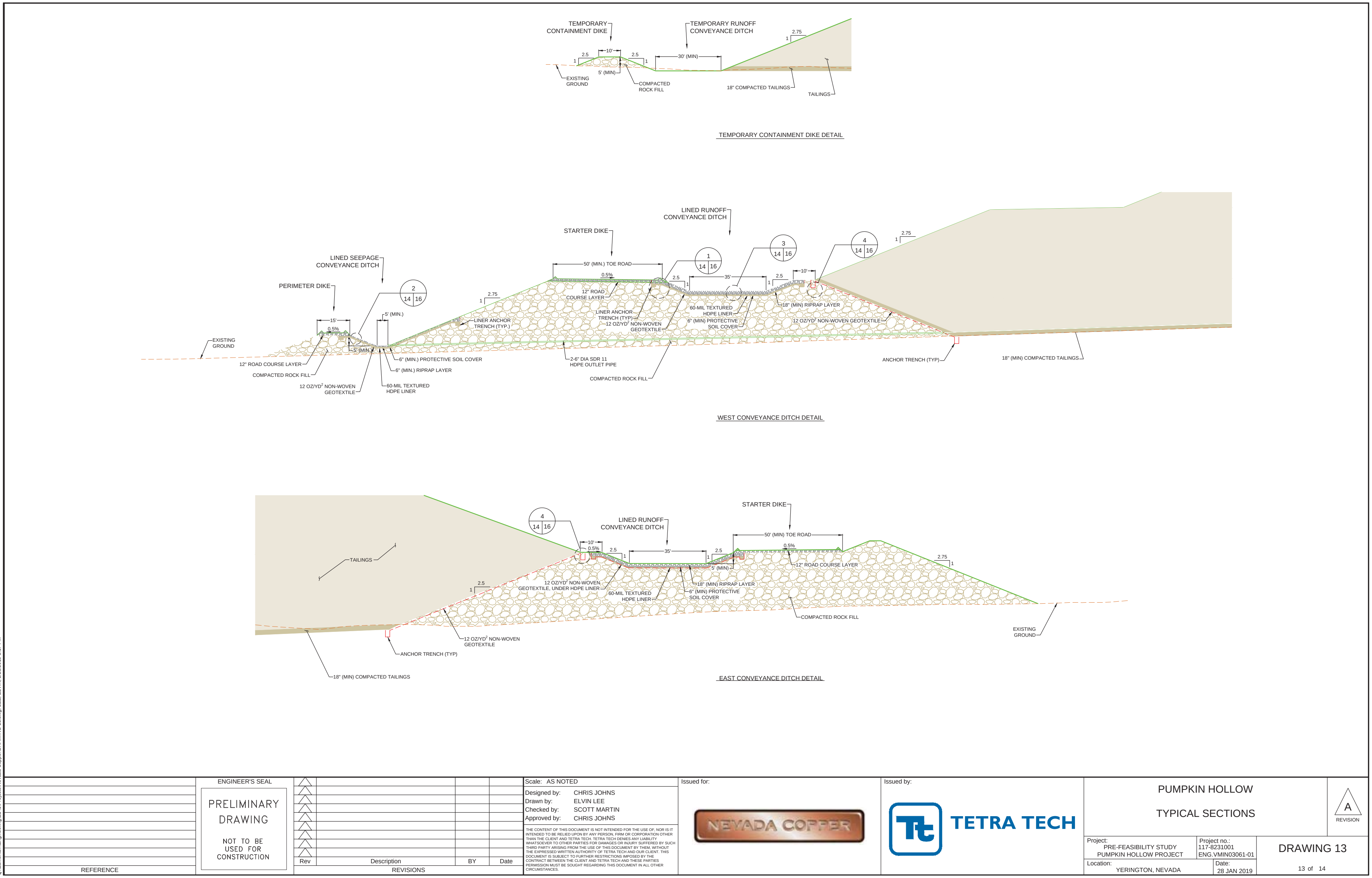
20.8.4.1 Foundation Preparation

The foundation preparation will include clearing and grubbing and salvaging of topsoil for future reclamation use. Several existing natural drainages that are filled with alluvium span the footprint of the proposed TSF. Prior to placement of tailings, the drainages will be leveled or backfilled and compacted to achieve design grades. Unsuitable materials within the foundation, if encountered, will be removed and replaced with suitable fill as required.

20.8.4.2 Perimeter Dike

The initial lift of tailings will be contained within a perimeter dike constructed of mine development rock from the open pit and underground mining operations. The perimeter dike will have a maximum height of 30 ft and a trafficable crest with a minimum width of 50 ft. A 30 ft wide lined runoff conveyance channel will be constructed on the inboard edge of the perimeter dike crest. Typical sections of the perimeter dike are shown in Figure 20-5.

Figure 20-5: Typical Sections (Tetra Tech, 2019)



20.8.4.3 Surface Water Management

The stormwater management features for the TSF were designed to 1) divert stormwater run-on from areas upstream of the TSF and 2) collect contact water drainage and runoff from the TSF. The hydrologic and hydraulic design of the site-wide stormwater management features described in previous work (Tetra Tech 2015) were reviewed and confirmed suitable for the current concept.

Stormwater run-on from areas upstream of the TSF will be diverted to proposed infiltration basins through the north, east, and south diversion channels. The diversion channels were sized to divert flows resulting from the 100-year 24-hour storm event.

Contact water from the TSF will be captured in a lined runoff conveyance ditch at the TSF perimeter. This water will drain to a lined runoff collection basin located near the open pit plant site to the northwest of the TSF.

The contact water runoff conveyance ditch was sized for the 100-year event, and the runoff collection basin was sized to convey or hold the probable maximum precipitation event. The conveyance ditch will drain to the collection basin via six 36-inch diameter HDPE culverts.

Stormwater accumulations in the pond will be pumped back to the process plant for reuse, as required.

20.8.4.4 Seepage Containment

The primary seepage containment layer for the proposed TSF consists of an 18-inch thick (minimum) moisture-conditioned and compacted tailings layer constructed on the prepared foundation and the inner slopes of the perimeter dike. Laboratory tests performed on representative samples of compacted tailings indicate that an average hydraulic conductivity of 3.7×10^{-4} cm/s can be achieved in the compacted tailings layer.

A network of gravel overdrains will be installed atop the liner in a herringbone pattern to provide hydraulic relief. The overdrain system will consist of secondary drains installed at 150 ft on centers connected to primary drains. The secondary drains will consist of a drainage course enclosed within a non-woven geotextile for filtration purposes. The secondary drains will be connected to primary drains consisting of six-inch diameter perforated HDPE collector pipes enclosed within a drainage course and a heavy duty non-woven geotextile. The primary drains will discharge to the geomembrane-lined seepage conveyance channel and the water returned to a seepage collection pond for use in the process plant. The staged construction of the overdrain network layout is shown in Figure 20-6 through Figure 20-8.

Previous TSF designs for the Pumpkin Hollow Project incorporated a HDPE geomembrane liner in the first stage of TSF construction (Cell 1) in order to meet Nevada regulatory requirements for seepage containment. Subsequent cells were to incorporate a compacted tailings liner assuming approval by the Nevada regulator based on field scale measurements of the containment system performance. Nevada Copper will monitor an instrumented test pad of filtered tailings that incorporates a compacted tailings liner as part of the underground TSF project. The test pad construction and monitoring is not part of this PFS design. This test pad is to be completed in 2019, in advance of Cell 1 construction of the open pit TSF in 2022, so there should be adequate time to demonstrate that the compacted tailings liner will provide adequate seepage mitigation.

Figure 20-6: Overdrain Layout – Cell 1 (Tetra Tech, 2019)

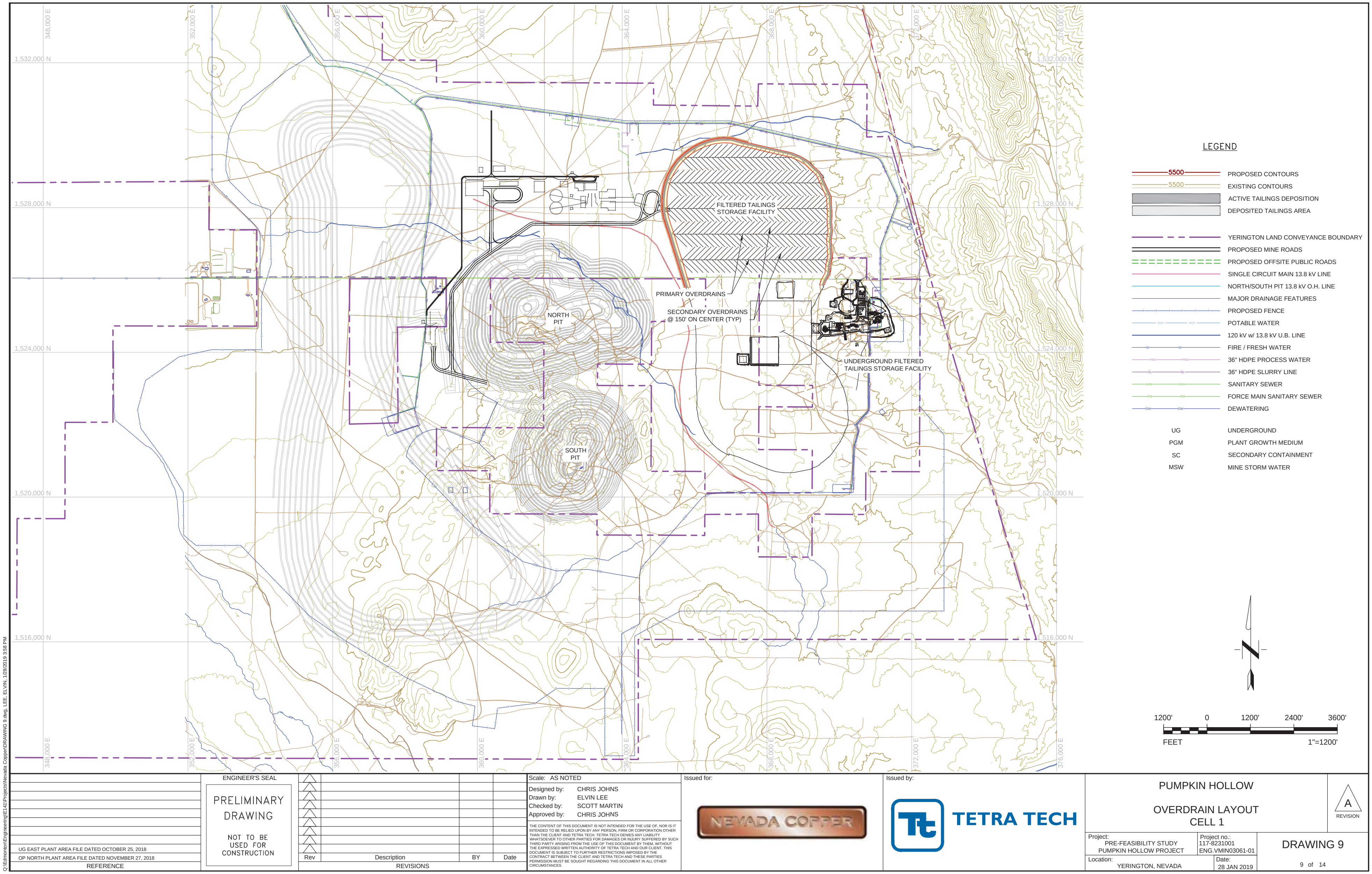


Figure 20-7: Overdrain Layout – Cell 2 (Tetra Tech, 2019)

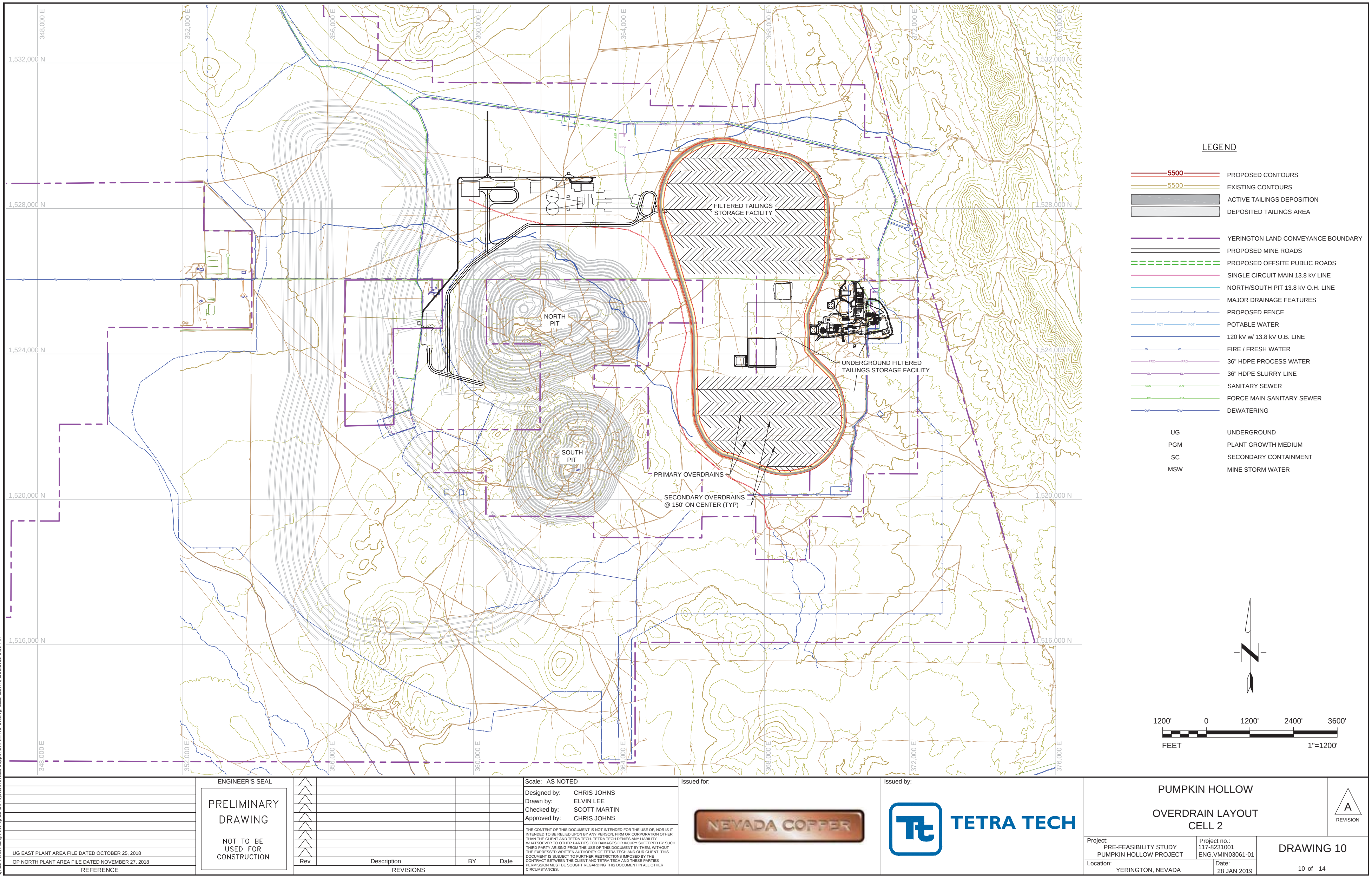
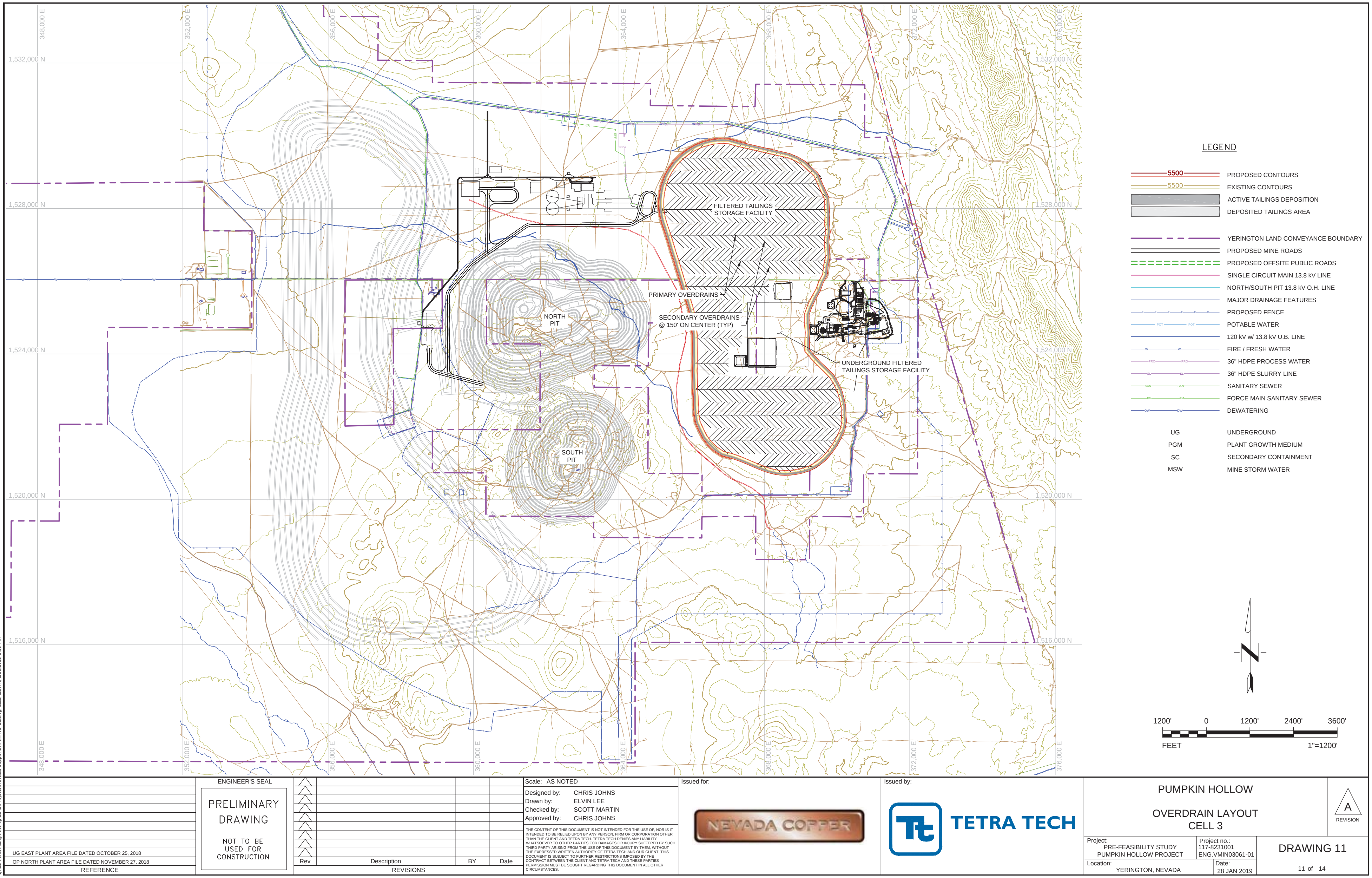


Figure 20-8: Overdrain Layout – Cell 3 (Tetra Tech, 2019)



20.8.5 Engineering Analysis

Engineering analyses undertaken in support of the proposed open pit TSF included:

- Liquefaction potential (from the 2015 feasibility study [Tetra Tech 2015])
- Seepage and fate and transport analyses (from the 2015 feasibility study)
- Stability analyses (updated to reflect current geometry and construction plan)

Risks associated with liquefaction in the filtered tailings will be mitigated through compaction of the perimeter structural zone of the tailings. The geotechnical investigations indicate that the subsurface soil conditions were not susceptible to liquefaction.

Seepage, fate and transport analyses were conducted as part of previous studies (Tetra Tech 2015) to evaluate the rate of infiltration through the filtered tailings stack and estimate the downward flux of pore water into the natural foundation soils. The model results indicated that saturated conditions within the filtered tailings mass were not anticipated to develop during the operation and closure period, and an overall negative (unsaturated) pressure distribution was expected.

Two-dimensional stability analyses were conducted to assess the factor of safety for the maximum height cross section (western slope) of the TSF under static and pseudo-static loading conditions. The geometry for the cross section was determined from the information available for the western slope of the proposed open pit TSF and from geotechnical data (Tetra Tech 2015). The soil material properties were determined from laboratory test results and experience with similar materials. The results indicate the proposed design meets minimum factors of safety for state guidelines developed for heap leach facilities and water impoundment structures.

Item 21.0 CAPITAL & OPERATING COSTS

This Item describes:

- Capital cost estimates (otherwise referred to as Capital Expenditure estimates or Capex)
- Operating cost estimates (otherwise referred to as Operating Expenditure estimates or Opex)

The capital and operating cost estimates for the Underground Project, as detailed in Item 21.1, for the process plant, power supply, surface facilities and all related infrastructure for this Technical Report have been prepared in accordance with:

- **Processing Facilities**: Level of Accuracy reflects an advanced feasibility study definition for the process plant capital costs, as defined in Item 17.1.
- **Surface Infrastructure**: Level of Accuracy reflects a ±20% definition of scope with a clear path to execution completed in this phase of works, for the underground works area layouts of the buildings and infrastructure defined in Item 18.1.
- Mining equipment and pricing was priced using contractors' rates and specific pricing for plant, equipment and construction materials. The level of Accuracy reflects a ±20% definition of scope.

The capital and operating cost estimates for the Open Pit Project, as detailed in Item 21.2, for the process plant, power supply, surface facilities and all related infrastructure for this Report have been prepared in accordance with:

- **Processing Facilities**: Level of Accuracy reflects a +25/-5% definition of scope for the capital costs including equipment selection for the process plant and tailings filters as defined in Item 17.2.
- **Surface Infrastructure**: Level of Accuracy reflects a +25/-5% definition of scope using pricing received for the open pit layouts of the buildings and infrastructure defined in Item 18.2.
- Mining equipment and pricing was based on OEM rated and in house assessments of the required mining methods, balance of plant, equipment and construction materials. The Level of Accuracy reflects a +25/-5% definition of scope.

The estimates were each assembled with supporting datasheets and equipment pricing with installation pricing received from regional contractors specific to the design developed. As part of the price build up, detailed execution plans and supporting methodology have been developed for the next phase of works.

21.1 Underground

21.1.1 Initial Capital Cost Estimate

In deriving construction cost estimates, Sedgman and Mining Plus consulted with contractors, freight forwarders, vendors and service suppliers to establish costs. The Capex includes direct and indirect costs covering all of the traditional items typical of any mine development.

21.1.1.1 Summary

The Capex consists of three components: direct costs, indirect costs and contingency as described below.

The Capex for the Underground Project is approximately $182.4 million, as shown in Table 21-1, subject to qualifications, assumptions and exclusions, all of which are detailed below.

Table 21-1: Initial Capital Costs Summary

Item	**$, millions**
Direct Costs	
Underground mining	42.3
Process Plant (including Concentrate Handling)	59.9
Infrastructure and Tailings	49.9
Indirect Costs	
Infrastructure - EPCM Costs	7.0
Sales & Use Tax on Purchased Equipment	Included in directs
Construction Indirects	4.6
Owner's Costs	8.8
Spares and First Fills	0.6
Commissioning and Start-up	0.4
Total Indirect Costs	**21.7**
Total Direct and Indirect Costs	**173.4**
Contingency	**9.0**
Total Initial Capital	**182.4**

Basis of Estimate

Mining capital costs have been based on the following information:

- Process flow diagrams, preliminary mine design and schedules, engineering drawings for the Main Shaft and headframe, site layout and general arrangement drawings, equipment list, electrical single line diagrams, piping diagrams and drawings from similar designs made specific for the Underground Project
- Vendor quotations for the design/supply of new equipment provided by vendors in accordance with specifications and/or datasheets developed for the prefeasibility study
- Vendor quotations from multiple mining contractors with experience in shaft sinking, production shaft commissioning and underground development operations

- Quantity take-offs from detailed underground development and production designs and a mining schedule
- Quantity take-offs for materials provided by engineering drawings from source projects
- Labor rates provided by local and regional construction contractors
- Productivities for mobile equipment, mining processes and labor based on first principles buildups, and correlation with similar projects in the region, elsewhere in North America and performance by experienced mining contractors who are familiar with the location and conditions

Process plant capital costs have been based on the following information:

- Process flow diagrams, site layout and general arrangement drawings, equipment list, electrical single line diagrams, piping and instrumentation diagrams and drawings from similar projects designs laid out and made specific for this scope
- Vendor quotations for the design/supply of new major and secondary equipment provided by vendors in accordance with specifications and/or datasheets developed by the engineering groups involved
- Prices for permanent materials based on supplier quotations and in-house data and current market conditions
- Quantity take-offs for materials provided by engineers drawing from completed source projects
- Labor rates provided by local and regional construction contractors
- Productivities for installing equipment and materials provided by local and regional construction contractors who are familiar with the location and local conditions
- Supply and installation prices from experienced vendors of pre-engineered and modular buildings
- Freight costs allowances for all process equipment based on vendor and freight forwarder quotations

Direct Costs

Mining direct costs include for new equipment (except some minor equipment supplied by the mining contractor), new materials, excavations and installation for all permanent mine infrastructure associated with:

- Shaft development, ramp development, lateral drift development and underground services development

- Construction of infrastructure including the material handling system, electrical substations, pump station, explosives magazine and maintenance shop, complete with offices and warehouse
- Fitting out of the hoisting shaft with necessary steelwork to suffice the planned shaft hoisting rate of 6,500 stpd
- Material handling activities including fixed and mobile machine operation and hoisting operations to transport material to the surface headframe discharge chutes
- Development drilling and blasting
- Power supply and distribution below the Main Shaft and ventilation raise collars
- Batteries for underground mobile mining equipment
- Personnel transport within the limits
- Maintenance of accesses, travel ways and escape ways to expected standards and in safe condition

Mining direct costs are inclusive of materials, consumables and other items such as:

- Explosives, blasting accessories and drill consumables
- Ground support consumables (rock bolts, plates, mesh, shotcrete etc.)
- Paste fill consumables (anchor pins, timber, mesh, shotcrete etc.)
- Ventilation consumables (vent tube, adaptors, installation accessories)
- Dewatering consumables (pipe, fittings)
- Power supply and distribution below the Main Shaft and ventilation raise collars
- Pre-production mining, warehousing and administration

Process plant direct costs include for all new equipment, new materials and installation for all permanent facilities associated with:

- Crushing, material handling and processing facilities
- Freight costs of vendor equipment and fabricated items to site
- Infrastructure roads and site preparation
- Power supply and distribution
- Concentrate load out
- Pre-production mining

- Stacked tailings storage area
- Warehousing, administration, site maintenance workshop
- Scope services and other utilities including control and communications systems
- Plant mobile equipment

Indirect Costs

Indirect costs include the following:

- Pre-production overhead and administration costs (such as managers, foremen, clerical and technical staff) of the mining contractor, associated with construction of the development
- Temporary construction services including some construction equipment
- Engineering, procurement and construction management services (including travel expenses)
- Owner's costs
- Start-up commissioning allowance and first fills

Working Capital

- An allowance in operating expenses at full production was included as working capital in the economic modeling in Item 22.1

Contingency

- Contingency for the mine pre-production capital costs has been applied and itemized separately, based on assessment of the vendor quotations obtained, including from mining contractors (which were generally inclusive of all materials and consumables) and other vendors for the supply of equivalent materials and consumables
- The contingency for the process plant has been developed using a Monte Carlo risk model contingency for infrastructure and tailings and mining was set at $5.2 million and $3.8 million respectively. This gives an overall contingency allowance of $9.0 million
- The contingency amount is an allowance that has been added to the Capex to cover unforeseeable costs within the scope of the estimate

21.1.2 Qualifications & Assumptions

The following assumptions were made in preparing the estimate:

- Construction work is based on unit and fixed price contracts (no cost plus or time and materials arrangements)
- Fixed price from vendors for equipment and bulk materials

- Where design for some items allowed, the balance of scope used budget quotes from vendors for equipment and materials
- Concrete will be available locally
- Soil conditions will be adequate for foundation bearing pressures
- Construction activities will be carried out in a continuous program
- Labor productivities have been validated with input from experienced contractors and in-house database for current projects
- Bulk materials such as reinforcing bar, structural steel and steel plate, cable, cable tray and piping are all readily available in the scheduled timeframe
- Capital equipment is available in the timeframes shown since availability has been verified by suppliers

21.1.2.1 Pricing

Pricing for processing, infrastructure and mining related equipment is based on a combination of project specific vendor quotations (supported by duty specifications) and budgetary fabricated items quotations relevant to the stated accuracy.

"Budgetary quotation" generally means that indicative pricing has been provided for specified equipment, materials and productivity but no commitment has been made to provide the equipment or materials at this price at a future date.

21.1.2.2 Taxes

The Nevada state sales tax (4.6%) and the Lyon County local sales tax (2.5%) totaling 7.1% has been applied as appropriate. All other taxes are excluded from the Capex.

All capital costs are expressed in U.S. dollars with the following provisions:

- Costs are based on an award date of October 2017 with no provision for escalation beyond this date
- Costs submitted in other currencies have been converted to U.S. Dollars

No provision has been made for any fees applicable to currency charges or currency fluctuations.

21.1.2.3 Accuracy

The Capex accuracy is as follows:

- Process plant: This is based on an intended binding offer for EPC execution
- Mining and infrastructure: This is based on a budgetary estimate with an expected accuracy ranging from 15% to 20% for various components of this scope

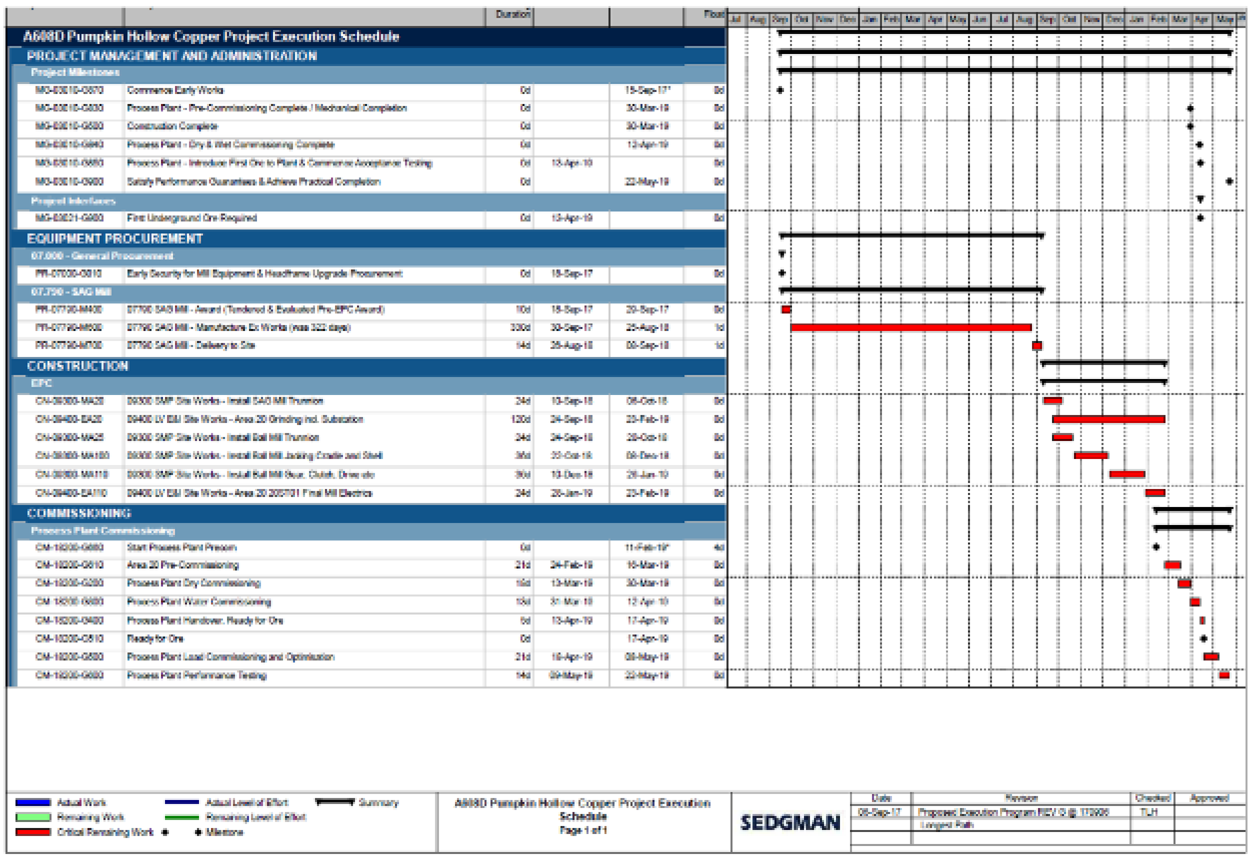
21.1.2.4 Implementation

The Capex is based on an EPC execution model for the process plant, and an Engineering, Procurement and Construction Management (EPCM) execution for infrastructure. It is assumed that Nevada Copper will follow the implementation plan as described. Any deviation from this plan may have a material impact on both the execution schedule and costs.

21.1.2.5 Execution Schedule

A detailed execution schedule and plan, as shown in Figure 21-1, was developed with interaction between the mining, processing and surface infrastructure design teams with direct input from the regional constructors and mine contractors to develop the works forward from engineering to construction and performance testing.

Figure 21-1: Execution Schedule (Sedgman, 2018)



21.1.2.6 Exclusions

The Capex does not include allowances for:

- Escalation due to late commencement
- Scope changes
- Interest during construction
- Schedule delays and associated costs such as those caused by:
    - Scope changes
    - Unidentified ground conditions
    - Extraordinary climatic events
    - Labor disputes
    - Permit applications
    - Receipt of information beyond the control of EPCM contractors
    - Schedule recovery or acceleration
    - Cost of financing
    - Owner’s sunk costs
    - Research and exploration drilling
    - Corporate and mining taxes
    - Sustaining capital
    - Permitting costs
    - Working capital
    - Closure costs and salvage values

21.1.3 Direct Costs

21.1.3.1 Underground Mine

Initial capital costs for the Underground Project are composed of the sinking of the Main Shaft, associated surface infrastructure for shaft sinking, development of the shaft stations and initial drift development, batteries for underground mobile mining equipment, electrical transformers and shop tools. The development equipment will be leased by Nevada Copper, but utilized by a mining contractor during the pre-production development time period. Prime movers and drills will be battery energized, with the batteries being purchased outright by Nevada Copper. Table 21-2 shows the breakdown of total direct capital costs.

Table 21-2: Underground Mine Direct Costs

Area	$, millions
Capitalized Operating	2.8
Shaft sinking – East Main Shaft	12.8
Ventilation Shafts and Lateral Development/Construction	5.6
Mine Lateral Development Capex	10.8
Large Infrastructure Excavations	1.1
Other Mine Capital	9.2
Total	**42.3**

Note: Contingency is not included in above listed items.

Mobile equipment manufacturers have indicated that battery delivery lead times are three months from order. Therefore, battery costs for the battery energized mobile equipment fleet will occur as an initial capital expenditure. These mobile equipment fleet batteries captured as an initial capital expenditure will be for the ES and EN zones.

21.1.3.2 Process Facilities

The equipment capital cost is estimated based on the major process equipment, ancillary equipment, and infrastructure components necessary to process a nominal 5,000 stpd from the Eastern Area Deposits. The purchase price and payment schedules for the major equipment are from vendor quotes requested from vendors and received during study development.

Table 21-3: Process Facility Direct Costs

Area	$, millions
Crushing and Grinding	35.8
Flotation and reagents	13.4
Plant services	4.7
Concentrate handling	6.0
Total	**59.9**

21.1.3.3 Tailings Management

The dewatered tailings from the filter plant will be conveyed to an engineered DST facility for stacking by a mobile fleet. The direct capital costs associated with the conveyance and management of dewatered tailings includes the following items:

- Conveyors and ancillary equipment including installation
- General earthwork and grading
- Geosynthetic liner supply and installation
- Underdrain installation
- Hydraulic control structures

Costs for general earthwork and grading, geosynthetic liner and underdrain installation were based on quotes obtained from construction contractors. Costs for hydraulic control structures including miscellaneous culverts, concrete inlet structures, etc., were based on an estimated lump sum allowance. Material take-offs for all direct capital cost items discussed above were based on a feasibility level layout and design of the DST facility.

The estimated total initial capital costs for tailings management including construction of the DST facility and tailings conveyance are summarized in Table 21-4.

A paste backfill plant has been designed for the required throughput, which will supply CPF to one of two supply points (East North Vent and East South Vent). This cost is based on budgetary quotations from paste plant vendors and installation pricing from the regional constructors.

21.1.3.4 Infrastructure

Infrastructure capital costs include the total cost for construction of the infrastructure buildings and facilities. The direct capital costs include all costs incurred by a construction contractor executing the works, inclusive of non–management resources, plant and equipment, materials, consumables etc. The indirect capital costs include the construction contractor’s overheads inclusive of management and temporary works, the engineering consultancy costs for project management, engineering and supervision. Table 21-4 shows the breakdown of the total infrastructure capital costs. The methodology is consistent with producing an estimate with an accuracy of ±20% for this portion of scope.

21.1.3.5 Quantities & Unit Pricing

Engineering material take-offs have been based on quantities derived by the engineering groups from the detailed mine design and schedule, project drawings, preliminary engineering, sketches and similar projects.

Unit pricing for the underground mine was obtained from quotations from multiple mining contractors with experience in shaft sinking, production shaft commissioning, underground development and production operations.

21.1.3.6 Earthworks

Unit prices were solicited from regional civil contractors who have knowledge of the conditions in the area. Quantities have been based on topographic drawings at 1 ft contour intervals. The earthmoving unit rates were calculated based on data obtained from local contractors. The rates include the rental of earthmoving equipment, operators, fuel and mobilization/demobilization costs. It has been assumed that structural backfill, granular base, road base and sub-base will be supplied from local borrow pits. The unit costs associated with these materials include borrow pit development (crushing and screening) and transport costs.

21.1.3.7 Concrete, Formwork & Reinforcing Steel

Concrete quantities were determined from feasibility-stage drawings and experience from previous projects of a similar nature. The unit rates for concrete placement and finishing have been derived from in-house data from similar projects, and the rates were cross-checked against unit rates provided by regional industrial contractors. A local concrete supplier has priced the supply of concrete including rebar and associated materials.

The unit price includes supply and installation of locally available carbon steel material including sleeves and anchors.

21.1.3.8 Structural Steel

Quantities of structural steel for the mine and process plant were determined from feasibility-stage drawings and experience from previous projects of a similar nature. The unit rates have been provided by regional industrial contractors. The weights shown include allowances for connections and base plates.

The steel unit costs include:

- Material supply, fabrication and surface treatment where required
- Erection at site based on estimated installation man-hours and unit labor costs and including final touch-up of surface coating
- Connection steel, weldments and bolts
- Steel supply and erection rates have been developed based on in-house historical data with supply/erection rates checked against data provided by local contractors

21.1.3.9 Mechanical Equipment

All large capital equipment was itemized and priced in accordance with the duty specifications and data sheets. Firm price and also budget quotations were obtained for all major items in the estimate. Installation hours were estimated from regional constructors working on mining projects in North America. Vendor representatives will be engaged to oversee the installation of the larger equipment.

21.1.3.10 Mechanical (Plate Work & Tanks)

Plate work weights were calculated with allowances made for any necessary stiffeners, weirs, launders, etc. The unit prices include locally available plate purchase, detailing, fabrication and installation.

21.1.3.11 Piping

Piping material quantities and pricing were priced including shop detailing and fabrication. Material pricing and installation man-hours and productivity were reviewed and estimated by the constructors.

21.1.3.12 Electrical

Major electrical equipment for the mine was itemized based on electrical single line diagrams and quoted by vendors.

Major electrical equipment and electrical material prices and installation units for the process plant were provided by Sedgman. Pricing and installation man-hours and productivity were reviewed and estimated by the constructors.

Quantities for all electrical materials were estimated for the scope.

Lengths for overhead lines and high voltage cable were estimated from the overall site plan and priced by regional constructors.

21.1.3.13 Instrumentation

Instrumentation equipment, material prices and installation units were provided by electrical engineers. Pricing and installation man-hours and productivity were reviewed and estimated by Sedgman.

21.1.3.14 Direct Field Labor

Labor rates for the underground mine and construction of the process plant and surface infrastructure were estimated and developed using current pricing, supported by budgetary quotations from local and regional installation contractors.

The labor rates include:

- Base labor wage rate for 40 hours per week
- Overtime premiums for a work schedule of 45 hours per week
- Benefits and burdens
- Workers compensation premiums
- Travel allowances
- Transportation to and from accommodations
- Appropriate crew mixes
- Small tools and consumables allowance
- Field office overheads (included separately in Contractor’s Indirects)
- Home office overheads
- Contractor’s profit

Labor and staff rates for underground construction and development (including the shaft) were sourced from mining contractor estimates, the Nevada Mining Association 2016 Compensation and Benefits Survey, and estimates from equipment suppliers.

21.1.3.15 Off-Site Infrastructure

Off-site infrastructure required for the underground study is very minimal and consists of:

- Upgrading the existing road that runs from the mine to the state road network.
- Provision of the incoming high voltage power line.

21.1.3.16 Surface Water Hydrology

Capital costs for surface water improvements are for the construction of diversion channels and ponds. The costs associated with these items are primarily for earthwork and drainage works. They are based on unit costs developed from vendor quotes or Sedgman in-house estimates. The initial capital required for surface water improvements is included in the Bulk Earthworks costs in the infrastructure section above.

21.1.3.17 Groundwater Hydrology/Dewatering

Groundwater management capital costs are related directly to dewatering of the underground mine. Primary components of the dewatering system entail dewatering wells with associated pumps and appurtenances, and sump pumping systems. The pumping systems will discharge water to pipelines for consumption within the mill or re-infiltration to the aquifer within water management basins.

Initial costs for development of the underground dewatering system are included in mining costs (Table 21-4).

Table 21-4: Infrastructure & Tailings Direct Costs

Area	$, millions
Dry Stack Filter Plant and TSF	14.8
Paste Backfill Plant Package	8.1
Tailings Conveyor System	1.1
Tailings Bulk Earth Work	1.4
Tailings Liners	1.5
Infrastructure Bulk Earthworks	3.6
Infrastructure Electrical (power line)	8.3
Infrastructure Water	2.4
Infrastructure Buildings	1.9
Infrastructure Other	6.6
Total	**49.8**

21.1.4 Indirect Costs

21.1.4.1 Mining Construction Indirects

Mining construction indirect costs include pre-production overhead and administration costs (such as managers, foremen, clerical and technical staff) of the mining contractor. These costs are associated with construction of the scope.

21.1.4.2 Temporary Construction Facilities & Services

Indirect costs include:

- Construction field offices, furnishings, equipment
- Construction equipment not included in unit rates
- Temporary power supply
- Temporary water supply
- Temporary heating and hoarding
- Warehouse and laydown costs
- Temporary toilets
- Temporary communications
- Ongoing and final clean-up
- Yard maintenance
- Janitorial services
- Site safety personnel and training

Construction equipment indirect costs are not applicable to the major earthmoving costs, since unit prices submitted by contractors are “all-in” rates that include contractor’s construction equipment.

21.1.4.3 Construction Equipment

Costs for fully maintained construction equipment have been based on information received from local and experienced contractors.

21.1.4.4 First Fill & Spare Parts

Costs for spares have been based on vendor recommendations and Sedgman and Mining Plus operational experiences. Where vendor information has not been available, an allowance of the equipment purchase value has been included.

Industry standard allowances have been included for first fills for items such as start-up grinding media, reagents and fuel.

Spare parts for commissioning have been included in the Capex. Critical/capital spares have been included in the Opex.

Initial mine development will be completed by a mining contractor who will be responsible for supplying all consumables necessary for mining, including ventilation consumables, drill consumables, explosives, ground support, etc. Costs for the mining contractor will be incurred on a monthly basis (at the end of each month) for the mining work conducted during that month. The lead time to commencement of mining operations from award of the mining contract will be four months. This is to allow the mining contractor time for completion of engineering, fabrication and modifications to shaft sinking equipment, workforce recruitment and procurement of supplies and equipment.

21.1.4.5 Start-up & Commissioning

The requirements for vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up as deemed necessary for equipment performance warranties has been calculated and included in the estimate.

An allowance has been made for vendor representatives to be available at site during start-up, as well as for a team from the contractors' crews and the construction management staff.

21.1.4.6 Freight

Freight costs have been estimated based on vendor freight quotations and freight forwarder estimates. These have been included in the procurement cost estimates above.

21.1.4.7 Engineering, Procurement & Construction Management (EPCM)

Engineering and procurement costs for process plant equipment have been estimated based on the scope equipment list and a detailed engineering estimate.

Engineering and procurement costs for mining are based on manhours buildup from first principles.

Construction management costs have been calculated based on the development of an organization chart, execution schedule, estimated number of personnel required including extended work weeks, transportation, supplies and communications.

Mine development costs have been priced using regional construction contractors in consultation with developed designs for the scope.

21.1.4.8 Owner's Cost

Owner's costs have been developed by Nevada Copper to include items such as:

- Owner's management and operations staff
- Recruitment allowance
- Training programs for operations staff

- Property insurances
- Property taxes
- Corporate office staff
- Sustainability commitments
- Environmental testing and monitoring
- Owners allowances for field operations offices and supplies
- Owner's travel costs during construction
- Housing assistance allowance

Working capital allowance has not been provided.

21.1.5 Contingency

Contingency for the mine pre-production capital costs has been applied and itemized separately, based on assessment of the vendor quotations obtained, including from mining contractors (which were generally inclusive of all materials and consumables) and other vendors for the supply of equivalent materials and consumables. No contingency has been applied to mine sustaining capital costs.

The contingency for process plant direct and indirects covers unforeseeable costs within the scope of the estimate that has been calculated for each discipline and is a reflection of what the quality and the quantities of equipment pricing provided, and knowledge of the actual site conditions. It has been assessed using experience from the senior execution team, which included the constructors, engineers and equipment and fabrication suppliers, the assessment then used @Risk on a Monte Carlo risk model for the scope.

Contingency for the process plant scope is included in the direct costs for that scope, given it is intended to be delivered as an EPC contract. Contingency for infrastructure and tailings and mining is described in Item 21.1.

Contingency is not intended to be used for scope changes or exclusions that would otherwise be added or subtracted from the budget. Nor is it intended to cover such items as labor disputes, currency fluctuations, escalation, force majeure or other uncontrollable risk factors. It should be assumed that the contingency amount will be spent over the engineering and construction period.

21.1.6 Sustaining Capital Costs

Sustaining capital over mine life totals $110.6 million. Table 21-5 outlines a summary of the breakdown of costs. Where applicable, the detail is discussed in the next sections.

Table 21-5: Life of Mine Total Sustaining Capital Expenditures

Area	$, millions
Underground Mine Development	67.7
Process Plant, Infrastructure and Tailings	32.3
Deferred Capital	3.5
Contingency	7.3
Total Sustaining Capital	**110.6**

Reclamation activities include:

- Erosion control measures (e.g., berm construction, BMP implementation)
- Revegetation and herbicide application
- DST reclamation of side batters

These costs exclude closure costs, which are addressed separately in Item 22.

21.1.7 Operating Cost Estimate

Unit operating costs, net of capitalized underground development and other pre-production costs, are $52.55/st milled during the contractor mining phase, and $43.83/st milled during the Owner mining phase, as summarized in the LOM operating costs average $44.52/st milled. The first 1.5 years of costs are higher due to the use of a mining contractor. LOM site unit operating cash costs are as summarized in Table 21-6.

These unit costs have been calculated using fixed and variable costs across a typical production year, and are used as financial model inputs on a production ton basis.

Operating costs apply once production of concentrate has commenced—i.e., ore commissioning and performance testing phase, immediately prior to production ramp-up.

Table 21-6: Life of Mine Unit Operating Cost Summary

Area	LOM Operating Cost $/st Ore Milled (Contractor Miner)	LOM Operating Cost $/st Ore Milled (Owner Miner)
Mining	35.33	27.20
Processing	12.65	12.65
General & Administrative	4.57	3.98
Total	52.55	43.83

Operating costs have been prepared by Mining Plus and Sedgman and are discussed in more detail in the following Items.

21.1.8 Underground Project

21.1.8.1 Unit Operating Costs Estimate Basis

Costs were developed for the time periods set forth by the reporting schedule and calculated in terms of $/st of ore mined. Time periods for costing are monthly and all costs prior to ore production are considered capitalized as initial capital. The operating costs are based on a production rate of 5,000 stpd. General assumptions for the operating schedule used during cost estimating are listed below:

- Effective days per year 365 days
- Total shift length 12 hours
- Travel, inspections, breaks 1.96 hours
- Active working time 10.04 hours
- Effective working hour 50.2 minutes

Total unit operating costs per ton were developed from first principle cost models, which incorporate unit costs from vendor quotations and operating time for the following components:

- Equipment
- Labor
- Consumable supplies (ANFO, drilling supplies, fuel, etc.)
- Electrical power
- Backfill
- Hoisting
- Surface ore handling to plant
- Surface conveyors

Table 21-7 provides the overall estimated operating cost summary for the Underground Project, and is based on 5,000 stpd throughput with 365 days mine production per year at a nominal 5,000 stpd ore production rate.

Table 21-7: Underground Mining Unit Operating Cost Summary

Description	$/st Ore Milled (Contractor Miner)	$/st Ore Milled (Owner Miner)
Operating Development	$7.76	$5.53
Additional Stope Support	$0.02	$0.02
Production Drill & Blast	$11.13	$3.53
Production Load & Haul		$2.99
Equipment Leasing	$2.48	$2.48
Backfilling	$1.66	$1.76

Description	$/st Ore Milled (Contractor Miner)	$/st Ore Milled (Owner Miner)
Power	$3.25	$3.95
Hoisting & Shaft Maintenance		$2.12
Equipment Fleet Maintenance	$4.83	$4.83
Contractors G&A	$4.19	
Total Mining Operating Costs	$35.33	$27.20

21.1.8.2 Basis of Estimate

Mining operating costs have been based on the following information:

- Process flow diagrams, preliminary mine design and schedules, general arrangement drawings, equipment list, piping diagrams and drawings from similar designs made specific for this scope.
- Vendor quotations for the design/supply of new equipment provided by vendors in accordance with specifications and/or datasheets developed by the prefeasibility study engineering team and engineering groups previously involved.
- Vendor quotations from multiple mining contractors with experience in underground development and production operations.
- Quantity take-offs from detailed underground development and production designs and a mining schedule.
- Quantity take-offs for materials provided by engineering drawing from completed source projects.
- Labor rates provided by local and regional construction contractors.
- Productivities for mobile equipment, mining processes and labor based on first principles buildups, and correlation with similar projects in the region, elsewhere in North America and performance by experienced mining contractors who are familiar with the location and local conditions.

Mining operating costs include for new equipment, new materials and excavations associated with:

- Lateral drift development and underground services development.
- Material handling activities including fixed and mobile machine operation and hoisting operations to transport material to the surface headframe discharge chutes.
- Development drilling and blasting.
- Production drilling and blasting.
- Backfill management including waste rock fill and paste fill from the paste fill lines installed at the take-off points from each ventilation raise (excluding paste fill preparation on surface).

- Personnel transport within the area limits.
- Maintenance of accesses, travel ways and escapeways to expected standards and in safe condition.

Mining operating costs are inclusive of materials, consumables and other items such as:

- Explosives and blasting accessories, drill consumables.
- Ground support consumables (rock bolts, plates, mesh, shotcrete, etc.).
- Paste fill consumables (anchor pins, timber, mesh, shotcrete, etc.).
- Ventilation consumables (vent tube, adaptors, installation accessories).
- Dewatering consumables (pipe, fittings).
- Power supply and distribution below the Main Shaft and ventilation raise collars.
- Pre-production mining.
- Warehousing and administration.

21.1.8.3 Equipment Operating Costs

Unit operating costs for mobile equipment were used to generate the equipment costs in line with the mine plan. Unit costs in dollars per hour were developed from supplier quotes and estimated costs for overhaul parts, maintenance, fuel or power, lube and filters, tires and wear parts. Each cost component was obtained from a supplier or based on estimates from benchmarked costs at similar operations. The total costs were then compared against the benchmarked operations to ensure that they were within industry norms.

21.1.8.4 Labor Costs

The Underground Project is planned to begin with contractor labor, followed by a changeover soon after the onset of steady-state production to an Owner operated labor force. During contractor work, oversight by construction management will be done by hired consultants. Owner operated labor is divided into two categories, salaried and hourly. Contractor rates are slightly higher than Owner rates due to the anticipation of having to attract skilled labor to Yerington from the Carlin, Nevada area.

21.1.8.5 Consumable Costs

Consumable costs were derived by quantifying the expected consumption of materials.

21.1.8.6 Life of Mine Operating Costs

Unit operating costs for the underground mine, net of capitalized underground development and other pre-production costs, are $35.33 per ton-milled during the contractor mining phase, and $27.20/st milled during the Owner mining phase, as summarized in Table 21-7 . The LOM operating costs average $44.52/st milled. The first 1.5 years of costs are higher due to the use of a mining contractor. LOM site unit operating cash costs are as summarized in Table 21-6. These unit costs have been calculated using fixed and variable costs across a typical production year, and are used as financial model inputs on a production ton basis. It should be noted that all development drifting (except for ore and waste crosscuts and paste fill digouts) was capitalized and not expensed into the operating cost per ton-milled.

21.1.8.7 Mobile Equipment Lead Times

Mobile equipment manufacturers have indicated that equipment delivery lead times are up to 13 months therefore to meet the planned schedule planning has considered the early placement of orders in advance. Having to place the equipment 13 months in advance causes the production equipment fleet costs to occur as an initial capital expenditure.

21.1.9 Process Plant, Infrastructure & Tailings Facilities

Process operating costs for the 5,000 stpd underground ore concentrator include grinding and flotation circuits to produce a copper concentrate, followed by dewatering and filtration of concentrate prior to shipping and tailings disposal. Primary crusher operating costs are included in the process facilities costs.

Table 21-8 provides the overall estimated operating cost summary for the processing facility, and is based on 5,000 stpd with a mill availability of 92% and 365 operating days per year. The mine schedule also calls for 365 days mine production per year with a nominal 5,000 stpd ore production rate.

Table 21-8: Process Unit Operating Cost Summary

Description	$/st Ore Milled
Electric Power	$1.85
Grinding Media and Liners	$1.18
Reagents	$0.60
Other Process Consumables	$0.62
Labor	$2.90
Maintenance	$1.30
Mobile Equipment	$0.39
Paste Plant	$2.57
Dry stacking	$1.05
Infrastructure	$0.19
Total Process Operating Costs	**$12.65**

The annual operating cost estimate includes the following:

- Paste plant costs cover only the cost of binder addition to paste for backfill. All other costs for paste are included in other Opex line items.
- Dry stacking costs are based on costing the fleet and labor requirements to place and compact dry tailings from a radial stacker.
- Infrastructure costs include maintenance of such items as roads and buildings, and operation of the potable and waste water treatment plants.

21.1.9.1 Electric Power

Power consumption is based on the estimated power drawn by equipment, with the cost of power of $0.056/kWh as provided by Nevada Energy. The estimated power usage was calculated by Nevada Energy from the electrical load list, applying demand, diversification and utilization factors to the installed power.

21.1.9.2 Grinding Media and Liners

Mill liner costs have been based on recent liner pricing received from mill vendors during package pricing, applied at estimated intervals expected for mills in this application.

Grinding media costs based on recent prices received from media suppliers, applied to the following estimated wear rates:

- 0.55 lbs of steel media per short ton of ore for the SAG mill
- 0.79 lbs of steel media per short ton of ore for the ball mill
- 165 lbs per day of ceramic media for the regrind mill

21.1.9.3 Reagents

Reagent operating costs have been priced based on design dosage rates and recent prices received from suppliers.

21.1.9.4 Other Process Consumables

This estimate includes dewatering and laboratory costs.

Filter cloths are based on filter cloth costs applied to expected cloth lives for the proposed filters. Filtering of both concentrate and tailings is included.

Laboratory costs include metallurgical samples consumables, on stream analysis consumables, consignment certification, metallurgical assays and sample transport.

21.1.9.5 Labor

Process manpower used to cost Labor Opex is shown in Table 17-2. Assumptions used in the labor estimate include:

- Manpower is based on the owner operating the process plant on an operator-maintainer model
- Of the 53 personnel:
  - 13 people cover 13 roles on a one panel roster
  - 16 people cover 7 roles on a two panel roster
  - 24 people cover 6 roles on a four panel roster

- Labor rates are based on the Nevada Mining Association's 2016 Annual Wage and Salary Survey, adopting the P50 values and applying a burden calculated to average 30%.

Labor for dry stacking is included in that Opex line item, as described below.

21.1.9.6 Maintenance

Maintenance costs have been calculated by applying various factors to the capital cost of each equipment item. Different factors were used as appropriate for each type of equipment in order to calculate the cost per annum.

21.1.9.7 Mobile Equipment

The mobile equipment assessed includes the fleet for the following key tasks:

- Managing the re-handle of the low grade and high strength ores
- Placement of the dry stack material
- General site vehicles for plant and maintenance purposes

Loading of the concentrate into trucks is costed in the logistics section described in Item 18.1, and is excluded from the process facilities operating cost estimate.

Mobile equipment includes:

- A front end loader (CAT 980 or equivalent)
- A skid steer loader
- Two trucks
- A 50 tonne crane
- An integrated tool handler
- 8 light vehicles

21.1.9.8 Paste Plant

The mix design is based on a 7% cement content to achieve a required unconfined compressive strength (UCS) of 300 psi. Cement costs of $120/st delivered to site have been adopted based on recent pricing. Refer to Item 26 for additional work in this regard.

21.1.9.9 Dry Stacking

The operating costs for tailings management include costs associated with routine stacking operations and maintenance of the DST facility incurred during the active life of the mine after initial ore production. The operating costs include the following items:

- Provision of 40-ton trucks (CAT740 or equivalent), a CAT D6 dozer (or equivalent) and a compactor
- Manpower costs
- Surface water control and dust suppression via a water cart
- Seepage/runoff conveyance ditch maintenance cost

Conveyor costs to stacking are included in other Opex line items. Construction costs associated with the earthworks and liners are included in capital costs.

21.1.9.10 Infrastructure

The operating cost for infrastructure cover the site maintenance costs including:

- Road maintenance and dust suppression
- Fence and cattle guard maintenance
- Drainage and pond maintenance
- Potable water and sewage treatment plant costs
- Power line maintenance
- Building and other infrastructure maintenance supplies including facilities management

21.1.9.11 Other Costs

Operating costs for underground dewatering, both within the underground workings and dewatering wells have been included in the mining costs.

Supervision of the surface dewatering system is planned for environmental staff and will not require full-time personnel. This is costed in G&A operational costs.

Environmental costs are included within the Environmental Program plan described in Item 20 and is costed in G&A operational costs.

21.1.10 General and Administrative

Current G&A costs include:

- The Owner's Team (management and administration) such as managers, assistant, clerks, technician, leadman, laborer, receptionist, security guards, controller.
- Overheads such as health, safety & security (supplies), first aid, HS&E training supplies, training, environmental service & supplies, building and facility, community/public relations and outside consultants/services.
- Property taxes and insurance.

21.1.11 Concentrate Transport and Shipping

The costs of concentrate transportation to smelters are not considered site operating costs, and are therefore excluded from the Opex.

Transportation costs are deducted from gross smelter revenues, along with smelter conversion charges (TCRCs), to yield NSR. They are described in Item 18.1 and applied in Item 22.1 of this Report.

21.2 Open Pit Project

Cost estimates for the open pit process plant and infrastructure are based on a combination of budgetary quotations and pricing from similar previous projects. "Budgetary quotation" generally means that indicative pricing has been provided for specified equipment, materials and productivity but no commitment has been made to provide the equipment or materials at this price at a future date. Similarly pricing for the major mining equipment was based on budgetary quotes provided by equipment manufacturers including Caterpillar and Komatsu. Prices for support equipment such as service trucks and mobile cranes were sourced from Golder's equipment pricing database.

The Nevada state sales tax (4.6%) and the Lyon County local sales tax (2.5%) totaling 7.1% has been applied as appropriate. All other taxes are excluded from the capital cost estimate.

All capital costs are expressed in U.S. dollars with the following provisions:

- Costs are based on Q4 2018 pricing with no provision for escalation beyond this date for either Phase 1 or Phase 2.
- Costs submitted in other currencies have been converted to U.S. dollars.

The Capex accuracy is as follows:

- Process plant and surface infrastructure: This is based on a budgetary estimate with an expected overall accuracy of +25/-5%.
- Mining: This is based on a budgetary estimate with an expected overall accuracy of+25/-5%. for various components of this scope.

No provision has been made for any fees applicable to currency charges, or currency fluctuations.

21.2.1 Open Pit Capital Costs

The Open Pit Project capital cost estimate consists of direct capital costs for Nevada Copper's open pit process facility, DST facility, mining equipment, capitalized stripping and infrastructure. A summary of the capital costs is provided below as Table 21-9.

Table 21-9: Open Pit Capital Costs Summary

Description	Initial $, thousands	Expansion $, thousands	Sustaining $, thousands	Total $, thousands
Mining	128,199	105,753	290,748	524,699
Process (incl tails filters)	426,627	332,612	-	759,239
Infrastructure	89,508	35,130	-	124,638
Dry Stack, Site Water, Env & Reclamation	6,975	-	119,505	126,480
Owner / G&A1	20,272	-	-	20,272
Total	**671,581**	**473,495**	**410,253**	**1,555,328**

Note:

1. Includes concentrate handling offsite and bond for external power construction

21.2.2 Plant and Infrastructure Capital Cost Estimate

The Open Pit Project plant and infrastructure Capex was developed as described below. Estimates do not include allowances for:

- Escalation
- Scope changes
- Interest during construction
- Schedule delays and associated costs such as those caused by:
  - Scope changes
  - Unidentified ground conditions
  - Extraordinary climatic events
  - Labor disputes
  - Permit applications
  - Receipt of information beyond the control of EPCM contractors
  - Schedule recovery or acceleration
  - Cost of financing
  - Owners sunk costs
  - Research and exploration drilling
  - Corporate and mining taxes
  - Sustaining capital
  - Permitting costs
  - Working capital

The Open Pit Project process facilities capital cost estimate as in Table 21-10 was compiled in the Sedgman estimating system.

This estimate was conducted on the basis of a PFS phase estimate to have an accuracy of +25/-5%, and is based on the Sedgman Standards for Capital Cost Estimating.

Sedgman has utilized current equipment pricing along with material quantities taken from drawings and similar projects undertaken in recent years. Installation pricing has also been taken from recent similar projects, as well as the current work on the Underground Project.

Table 21-10: Processing & Surface Infrastructure Capital Cost Summaries

	Area	Phase I Value	Phase II Value	Combined Value
		($, thousands)	($, thousands)	($, thousands)
Concentrator				
	110 Crushing	61,947	27,157	89,104
	120 Grinding	81,925	73,633	155,559
	130 Flotation	33,257	32,668	65,925
	142 Concentrate Thickening	11,826	-	11,826
	152 Concentrate Filtration	10,878	501	11,379
	160 Reagents	3,402	3,402	6,804
	170 Services	8,147	8,086	16,233
	Electrical	28,112	27,552	55,664
	First Fills	1,689	1,220	2,909
	Contingency	19,924	14,392	34,316
	Nevada & Yerington Tax	4,369	3,043	7,411
	Total Concentrator	***265,476***	***191,655***	***457,131***
Tailing Filtration & Dry Stack				
	140 Tailings Thickening	17,431	16,486	33,918
	150 Tailings Filtration	95,824	78,785	174,609
	180 Dry Stack Tailings	14,007	14,032	28,039
	Electrical	17,658	17,695	35,353
	First Fills	1,028	899	1,926
	Contingency	12,057	10,565	22,622
	Nevada & Yerington Tax	3,146	2,495	5,641
	Total Tailing Filtration & Dry Stack	***161,151***	***140,957***	***302,108***
Site Surface Infrastructure				
	Site Power	882	-	882
	Fort Churchill Power	1,935	-	1,935
	Site Stormwater	1,065	137	1,202
	Site Preparation	8,294	6,824	15,118
	Site Roads	8,696	-	8,696
	Mine Buildings	25,754	3,383	29,138
	Concentrate Load Out	Included in "152 Concentrate Filtration"		
	Site Utilities	18,864	13,300	32,165
	Communications	362	-	362
	Temporary Tailings Stacking Area	-	-	-
	Electrical	16,648	8,659	25,308
	Contingency	6,600	2,584	9,185
	Nevada & Yerington Tax	406	241	648
	Total Site Surface Infrastructure	***89,508***	***35,130***	***124,639***
Total Capital Cost Accuracy +25/-5%		**516,135**	**367,742**	**883,877**

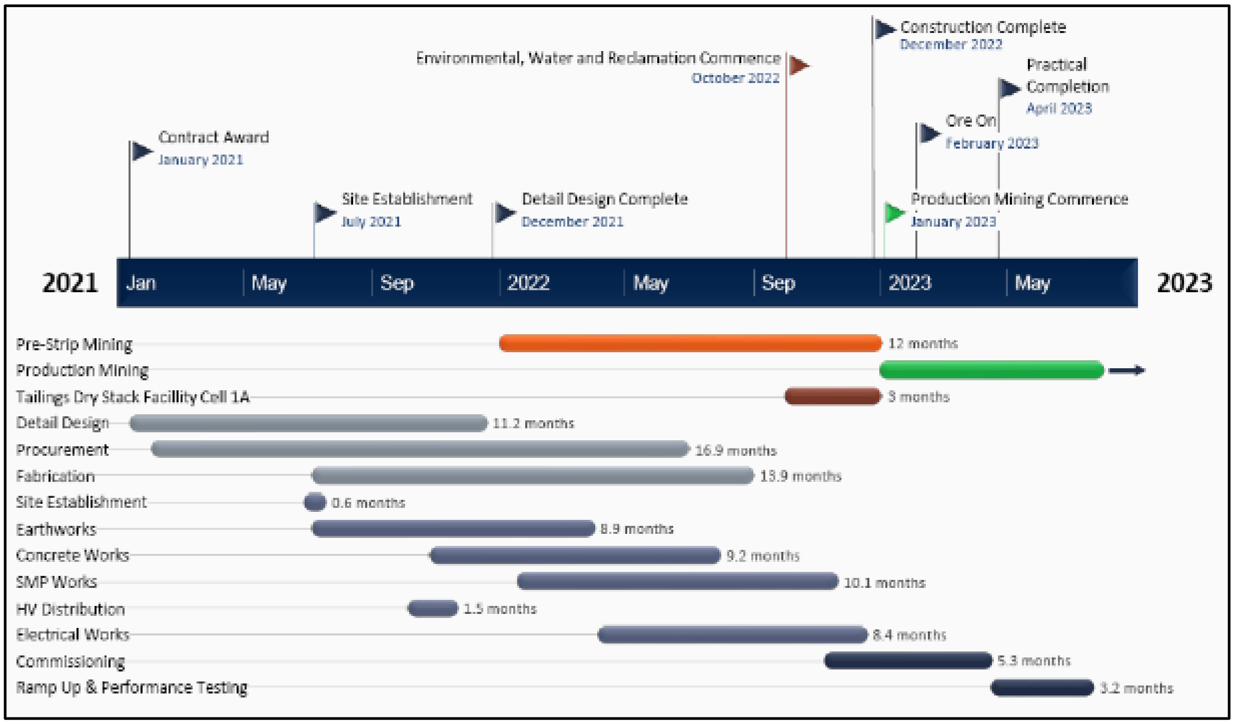
21.2.2.1 Implementation

The Open Pit PFS capital cost estimate is based on a potential future Engineering, Procurement and Construction (EPC) lump sum execution model for the process plant, and an EPCM execution for infrastructure.

21.2.2.2 Execution Schedule

A preliminary execution schedule and plan, as shown in Figure 21-2, was developed with interaction between the mine pit development and the processing and surface infrastructure development with direct input from regional constructors for the scopes of work to develop the works forward from engineering to construction and performance testing.

Figure 21-2:Preliminary Execution Schedule (Sedgman, 2019)



21.2.2.3 Infrastructure

Item 18.2 provides a detailed description of the surface infrastructure scope. Overall infrastructure costs are detailed as “Site Surface Infrastructure” in Table 21-10.

21.2.2.4 Quantities & Unit Pricing

Unit pricing is based on the current construction rates for the Underground Project, and where these were not available, Sedgman has utilized other current source projects.

Quantities of bulk materials are based on multiple sources, for example:

- Earthworks quantities are from high level modeling in Civil 12D.
- Concrete, structural steel, plate work and conveyor quantities are based on layouts and designs from current source projects and in-house calculations.
- Electrical, piping, valving and platework are based on material take-offs and re-use of current source projects designs and in-house calculations.

21.2.2.5 Earthworks

Earthworks unit rates are based on tendered rates recently received for the Underground Project. These rates were solicited from regional civil contractors who have knowledge of the conditions in the area. The rates include the rental of earthmoving equipment, operators, fuel and mobilization/demobilization costs. It has been assumed that where required imported fill will be supplied from local borrow pits. The unit costs for these materials include borrow pit development (crushing and screening) and transport costs.

Quantities were determined using the Civil 12D program for the current open pit layout, and are based on the most recent topography lidar data that Sedgman has received from Nevada Copper.

Earthworks direct costs are in accordance with scope detailed in Item 18.2.2 and Item 18.2.3.

21.2.2.6 Concrete, Formwork, Platework & Structural Steel

Concrete, formwork, platework and structural steel direct costs are in accordance with the scope detailed in Item 18.2 and using rates from the current Underground Project estimates, as described in Item 21.2.2.

21.2.2.7 Mechanical Equipment

Most large capital equipment was itemized and budgetarily priced in accordance with the duty specifications and data sheets. Installation hours were estimated from regional constructors working on mining projects in North America. Vendor representatives will be engaged to oversee the installation of the larger equipment.

21.2.2.8 Piping

Piping material quantities are based on re-use of designs from previously completed projects. Pricing is based on recent rates used for the Underground Project estimates, and includes shop detailing and fabrication.

21.2.2.9 Electrical & Instrumentation

Most major electrical equipment was itemized based on electrical single line diagrams and quoted by vendors. Pricing and Installation man-hours and productivity were based on previous projects, including input from the Underground Project. Quantities for all electrical and instrumentation materials were estimated for the scope. Lengths for overhead lines and high voltage cable were estimated from the overall site plan.

21.2.2.10 Direct Field Labor

Direct field labor rates were determined as a percentage of the direct costs in accordance with Sedgman Standards for Capital Cost Estimating.

21.2.2.11 Site Infrastructure

Site infrastructure required for the Open Pit PFS consists of:

- Re-aligning the existing road and incoming water lines that runs from the mine to the state road and town supply connections
- Provision of an incoming 120 kV high voltage power line for the open pit
- The package pricing of buildings, as described in Item 18.2
- The materials take off of site wide reticulation services for fire water, potable water and raw water
- Internal roads for light and heavy vehicles
- Earthworks from Civil 12D models for material take-off for cut and fill quantities
- Site power reticulation and localized electrical switchyards
- Fit-out of buildings

21.2.2.12 Temporary Construction Facilities and Services

Temporary construction facilities and services indirect costs have been estimated for the scope of the Open Pit Project against the described duration.

21.2.2.13 Construction Equipment

Construction equipment costs were determined as a percentage of the direct costs in accordance with Sedgman Standards for Capital Cost Estimating.

21.2.2.14 First Fill and Spare Parts

First fills and operating spares costs were determined as a percentage of the direct costs in accordance with Sedgman Standards for Capital Cost Estimating.

21.2.2.15 Start-Up and Commissioning

Start-up and commissioning costs were assessed as a percentage of the direct costs in accordance with Sedgman Standards for Capital Cost Estimating.

21.2.2.16 Freight

Freight costs, where not provided by the equipment vendors, have been estimated using the Sedgman estimating system.

21.2.2.17 Engineering, Procurement & Construction Management (EPCM)

Engineering and procurement costs for process plant equipment have been estimated as a percentage of the direct costs in accordance with Sedgman Standards for Capital Cost Estimating.

Engineering and procurement costs for mining are based on manhours buildup from first principles.

21.2.3 Groundwater Hydrology/Dewatering

Total LOM Capex costs for the open pit dewatering system as described are estimated to be $1,589,949 (without contingency). Total LOM Opex costs for the open pit dewatering system are estimated to be $4,284,406 (without contingency). Underground dewatering costs are not included in these estimates.

Capex costs have been developed from direct vendor quotes or Tetra Tech in-house estimates. Opex costs are related to power consumption; no manpower expense is included, as supervision of the dewatering system is planned for existing mine and/or environmental staff and will not require dedicated personnel. Monitoring costs are included within the environmental program plan.

21.2.4 Water Balance

Total LOM Capex costs for the open pit RIB system are estimated to be $1,206,000 (without contingency). Total LOM Opex costs for the open pit RIB and process make-up water pumping systems are estimated to be $729,000 (without contingency). Underground RIB system costs are not included in these estimates.

21.2.5 Tailings Management

Tetra Tech prepared an estimate of open pit DST construction quantities to support Open Pit Project Capex and an estimate of trucking hours required to support project operating cost estimation.

Tailings Storage Facility Construction Quantities

The capital costs associated with the construction of the DST include the following costs:

- Haul trucks and DST facility construction fleet
- General earthwork and grading
- HDPE liner supply and installation and overdrain installation
- Hydraulic control structures
- Temporary construction facilities

Table 21-11 provides a summary of open pit DST construction quantities.

Table 21-11: Open Pit Tailings Storage Facility Construction Quantities

Item	Unit	Quantity
Clear and Grub	acre	690
Grading	yd3, thousands	1,113
Load & Haul Unsuitable Subgrade Material to Waste	yd3, thousands	1,113
Subgrade Prep	yd3, thousands	3,338
Material Handling for Compacted Tailings Layer (Load, Place)	yd3, thousands	1,669
Material Handling for Compacted Tailings Layer (Haul/Transport)	yd3, thousands	1,669
Compaction of Initial 18-inch Tailings Layer	yd3, thousands	1,669
Excavate Seepage Pond and Runoff Pond & Load	yd3, thousands	490
Load & Haul out Seepage Pond Excavated Material	yd3, thousands	490
Load, Haul, Place, & Compact Suitable 2 ft Soil Layer Above Liners in Collection Ponds	yd3, thousands	46
Load & Place Rockfill	yd3, thousands	2,690
Haul Rockfill	yd3, thousands	2,690
Compaction of Rockfill	yd3, thousands	2,690
L&H & Place Riprap for Seepage Ditch (6 inch)	yd3, thousands	6.3
L&H & Place Riprap for Run-off Ditch (18 inch)	yd3, thousands	53
L&H + Place Soil for both Seepage & Run-off Conveyance Ditches	yd3, thousands	31
Screen, L&H, & Place, & Compact Roadbase - (12-inch Layer)	yd3, thousands	51
Primary Overdrain - Gravel	ft3, thousands	491
Primary Overdrain Geotextile	ft2, thousands	740
Primary Overdrain Pipe Material	l.f., thousands	82
Secondary Overdrain - Gravel	ft3, thousands	1,757
Secondary Overdrain Geotextile	ft2, thousands	3,288
Liner Subgrade Preparation in Ditches	ft2, thousands	1,677
80-mil HDPE Liner Supply and Installation Perimeter Ditches	ft2, thousands	1,677
Non-Woven Geotextile Supply and Installation Perimeter Ditches	ft2, thousands	1,677
Liner Subgrade Preparation for Ponds	ft2, thousands	692
60 mil HDPE Geomembrane Supply and Installation Run-Off Collection Pond	ft2, thousands	619
60 mil HDPE Geomembrane Supply and Installation Seepage Collection Pond	ft2, thousands	146

The equipment used to transport, place and maintain the DST facility was selected to be compatible with the mining fleet. The tailings transport and the mining fleet are proposed as integrated fleets to allow excess capacity in the haul of tailings, as a means of optimizing capital purchases of trucks. Hauls trucks for the dry stack will be of the 320 ton class utilized by the mine.

A bulldozer will be required during tailings placement for grading and track compaction, as required, and re-grading the side slopes of the DST in preparation for reclamation. Compactors will be required for compacting the haul ramps and the DST perimeter structural zone. Dozers will be the 300 Hp class and the compactor in the 600 Hp class. Support equipment for maintenance, supervision is assumed to be common with the mining fleet.

General Earthwork & Containment System

General earthwork and containment system construction includes:

- Foundation preparation, grading, and compaction
- Compacted rockfill for perimeter and starter dike construction
- Riprap lining for runoff conveyance ditch
- HDPE liner supply and installation for seepage collection ditch
- Placement and compaction of 18-inch compacted tailings base layer
- Overdrain network supply and installation

Quantities were calculated using Civil 3D® software based on the CAD design and the digital terrain model for the site.

Foundation preparation: Rough grading costs include an allowance for excavation and/or grading of surficial soils following soil salvage operations for future reclamation. Fine grading costs include foundation preparation activities which involve grading and compaction (as needed) to achieve a uniformly graded and firm foundation surface for the DST. Based on information from geotechnical investigations conducted to date, surficial rock outcrops are not expected within the DST footprint.

Soil/tailings/rockfill placement: Costs associated with fill placement include loading, hauling, moisture conditioning, placement and compaction of soil, tailings or rockfill for construction of the TSF perimeter and starter berms, and an 18-inch thick compacted tailings layer over the tailings stack footprint.

Riprap lining: Costs for riprap lining of runoff conveyance ditches include supply and installation of 10-inch riprap on the inside surfaces of the conveyance ditches for erosion protection.

Liner supply and installation: Liner capital costs include supply and installation of 60-mil HDPE geomembrane liner in the perimeter runoff and seepage conveyance ditches and ponds.

Overdrain installation: Costs associated with supply and installation of drains atop the compacted tailings liner include installation of 6-inch diameter perforated HDPE pipe, drainage gravel around the pipe, and a non-woven geotextile separation layer.

Hydraulic control structures will include culverts and inlet structures.

Additional Capital Costs

Additional capital costs associated with the construction of the TSF include costs for home office services and temporary construction facilities.

Home office services include costs associated with Engineering, Procurement, and Construction Management (EPCM). Engineering procurement costs include;

- Civil/geotechnical design
- Geotechnical modeling
- Surveying
- Geotechnical laboratory testing
- Geotechnical field investigations
- Construction quality assurance / quality control
- Instrumentation and monitoring

21.2.6 Mining

21.2.6.1 Mining Capital Cost Estimate

Golder prepared mine capital costs based on original equipment manufacturers' (OEM) quotes for primary mining equipment and relied on Golder's in house database for costs of supporting equipment such as service trucks, mobile cranes and pickup trucks. As directed by Nevada Copper, these costs were used as the basis for developing lease costs for the mining equipment. The capital cost of the equipment from which the leasing costs were derived are provided below in Table 21-14. Leasing costs were developed based on the leasing agreements proposed and provided by OEMs. In general, leasing costs were applied as operating costs except for those leasing costs that were incurred during the initial pre-stripping phase of the mine.

The total required capital costs for the Open Pit Project are estimated at $525 million. This includes $94 million for pre-stripping, $34 million for the initial equipment leasing, and $106 million for expansion related waste stripping and $291 million in sustaining capitalized waste stripping. Capital costs are summarized in Table 21-12.

Table 21-12: Mining Cost Summary

Mining Cost Category	Cost ($, millions)
Pre-Stripping Cost	94
Initial Pre-Production Equipment Lease Cost	34
Total Pre-Production Mining Capital Cost	128
Expansion Waste Stripping Cost	106
Sustaining Capitalized Waste Stripping Cost	291
Total Life of Mine Capital	525

Table 21-12 is in accordance with the Stripping Costs in Production Phase of Surface Mine, published by International Financial Reporting Standards Foundation (IFRIC 20), and other guidance notes there related. As per this standard, in any year where the strip ratio is higher than the average LOM strip ratio, the calculation used, in the PFS cost model, capitalizes the cost associated with this waste.

All related mining costs required for pre-stripping are included as capital costs. Costs related to the leasing of equipment are included as capital costs during the pre-production phase. Once production begins, these lease costs are included as operating costs.

21.2.6.2 Mine Lease Costs

OEMs were contacted for quotes for the selected mining fleet. Based on leasing agreements that were provided by the OEMs, Golder applied the mining capital costs as directed by Nevada Copper. The lease agreement selected was a lease length of 8 years at an interest rate of 5.75%. For this option, there was no down payment, and at the end of the term, a residual payment of 10% and a buyout of $1. For all the assumed mining equipment purchases, this aforementioned leasing structure was used for all assumed mining equipment purchases.

The initial lease costs in pre-preproduction are assumed capital costs. The remaining costs are included as operating costs (Table 21-13 and Table 21-14).

Table 21-13: Equipment Lease Cost LOM Comparison

Cost Category	Cost ($, millions)
Total Equipment Costs (outright purchase)	368
Additional Lease Costs	99
Total Lease Costs	467

Table 21-14: Equipment Lease Cost Summary

Cost Category	Cost ($, millions)
Initial Equipment Lease (Year 1)	34
Operating Equipment Lease	433
Total Life of Mine Equipment Lease	467

21.2.7 Open Pit Operating Cost

21.2.7.1 Open Pit Mine Operating Cost

Mine operating costs were developed using a zero-based approach. Quantities of waste and ore from the mine production schedule were used to estimate explosive consumption, equipment hours and manpower requirements on a quarterly basis for the first five years and annually thereafter, Details of the mining cost estimation follow below.

21.2.8 Open Pit Mining Project Assumptions

Table 21-15 summarizes the general Open Pit Project assumptions used in the cost model.

Table 21-15: Open Pit Project Assumptions

Parameter	Value
Shifts per Day	2
Number of Crews	4 crews
Operating Days per Year	365 days
Delay Time per Shift	85 mins
Lube Price	11.66 $/gallon
Fuel Price	2.00 $/gallon
Electricity Price	0.053 $/kWh
Mining Recovery	98%
Swell Factor	40%
Moisture	4%

Note: Project assumptions were provided by G. French of Nevada Copper Inc., March 2018, fuel price assumption was provided January 2019.

21.2.9 Open Pit Mining Equipment Assumptions

A summary of the required fleet for the Open Pit Project is shown in Table 21-16.

Table 21-16: Equipment Requirements

Equipment	Initial Units	Additional Units
64-yd3 Electric Shovel	2	-
47-yd3 Hydraulic Shovel	0	2
30-yd3 Front-end Loader	1	1
320-t Haul Truck	23	11
Production Drill	3	1
Pre-shear Drill	0	2
700-plus-hp Production Dozer	3	5
500-plus-hp Production Dozer	1	1
300-plus-hp Motor Grader	2	1
Water Truck	1	2
360-plus-hp Excavator	0	1
400-plus-hp Excavator	0	2
480-plus-hp Excavator	1	2
670-plus-hp Wheeled Dozer	1	2
900-plus-hp Wheeled Dozer	0	1
Field Service Truck	1	1
Tire Service Truck	1	2
Fuel and Lube Truck	1	1
Bus	1	1
Light Pickup	22	6
Rough Terrain Crane	1	0
Jaw Crusher	1	0
Light Plant	23	23

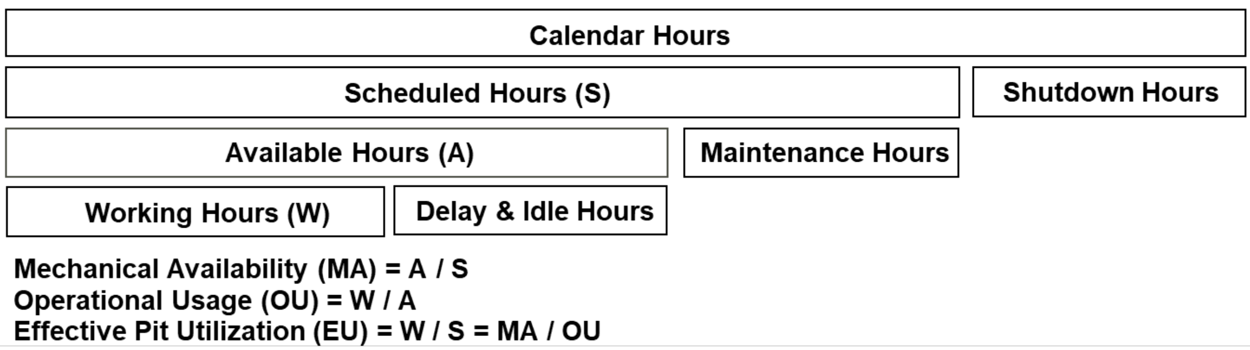
The initial utilization parameters are shown in Table 21-17.

Table 21-17: Initial Equipment Assumptions

Parameter	Electric Shovel	Hydraulic Shovel	Front-end Loader	Haul Truck	Drill	Track Dozer	Wheel Dozer	Motor Grader
Mechanical Availability (%)	92.0	92.0	92.0	92.0	92.0	92.0	92.0	92.0
Operational Usage (%)	72.8	73.3	73.6	74.3	74.3	75.7	56.6	80.2
Effective Pit Utilization (%)	67.0	67.4	67.7	68.4	68.4	69.6	52.1	73.8
Mechanical Availability Degrade (%/yr)	0.5	1.0	1.0	0.0	0.0	0.0	0.0	0.0

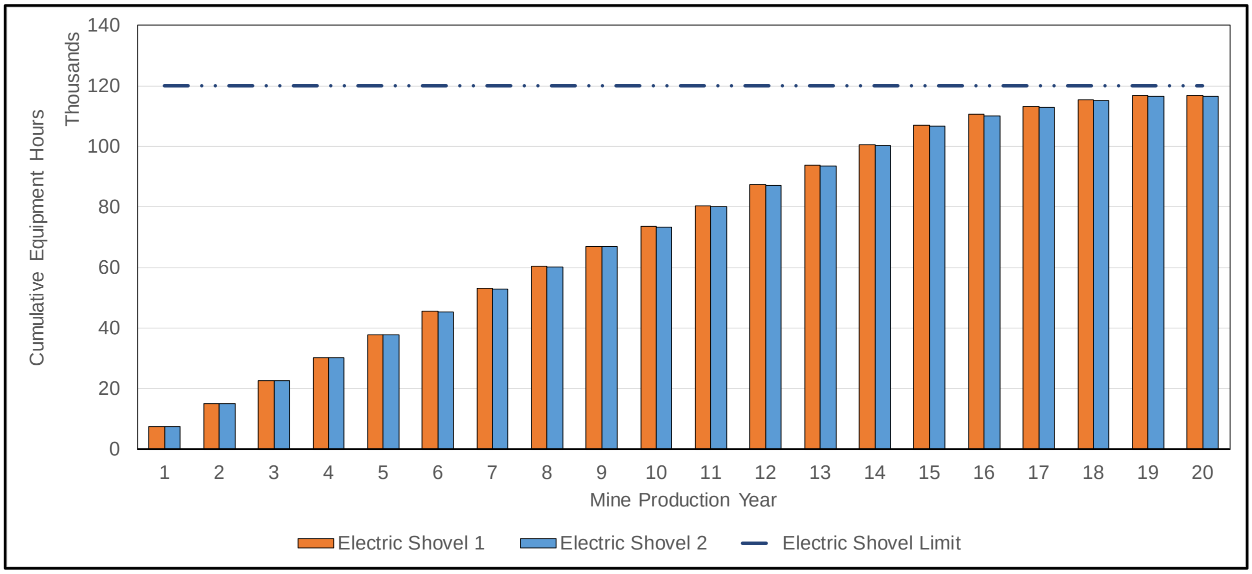
The effective equipment working hours calculation is summarized Figure 21-3.

Figure 21-3: Effective Equipment Working Hours Calculation



For the Open Pit Project, a combination of hydraulic, electric shovels, and front-end loaders will be utilized. The loading equipment was selected to match the 320 st class haul truck fleet. The drills selected were based on the designed bench height, required drill diameter, and drill rod length for single pass. Replacement capital was included for major equipment, such as shovels and trucks. Equipment replacement was based on industry standard lifespans. Equipment lifespans are estimated to be approximately 120,000 hours for the electric shovel, 80,000 hours for haul trucks and production drills, and 60,000 for all other pieces of equipment (Figure 21-4 and Figure 21-5).

Figure 21-4: Comparison of Scheduled Electric Shovel Hours & Industry Standard Lifespan (Golder, 2019)



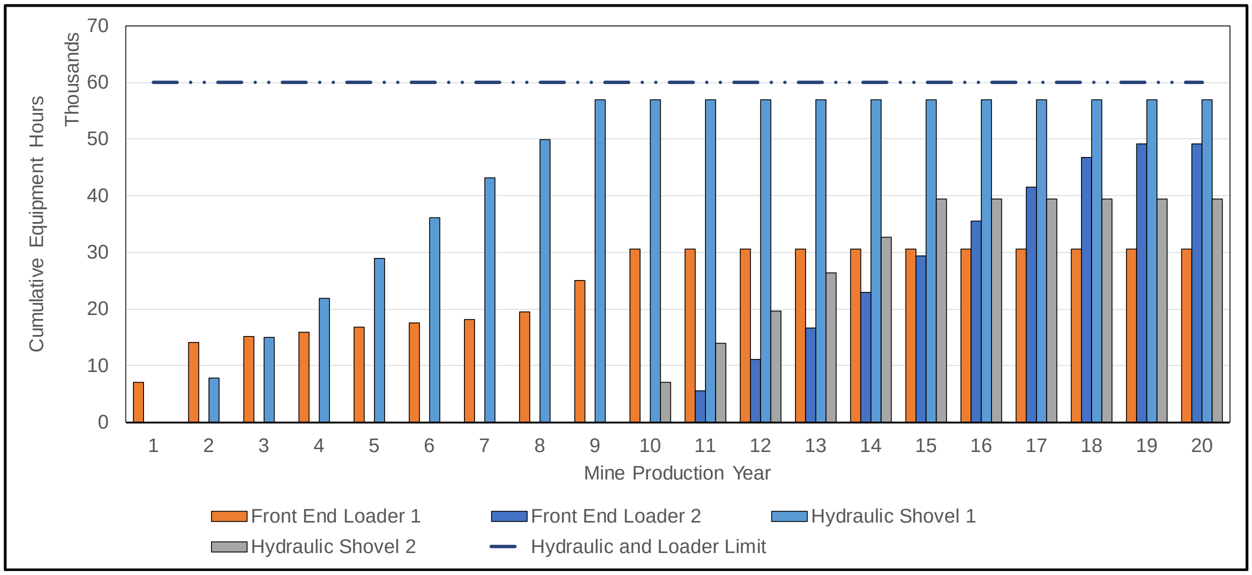
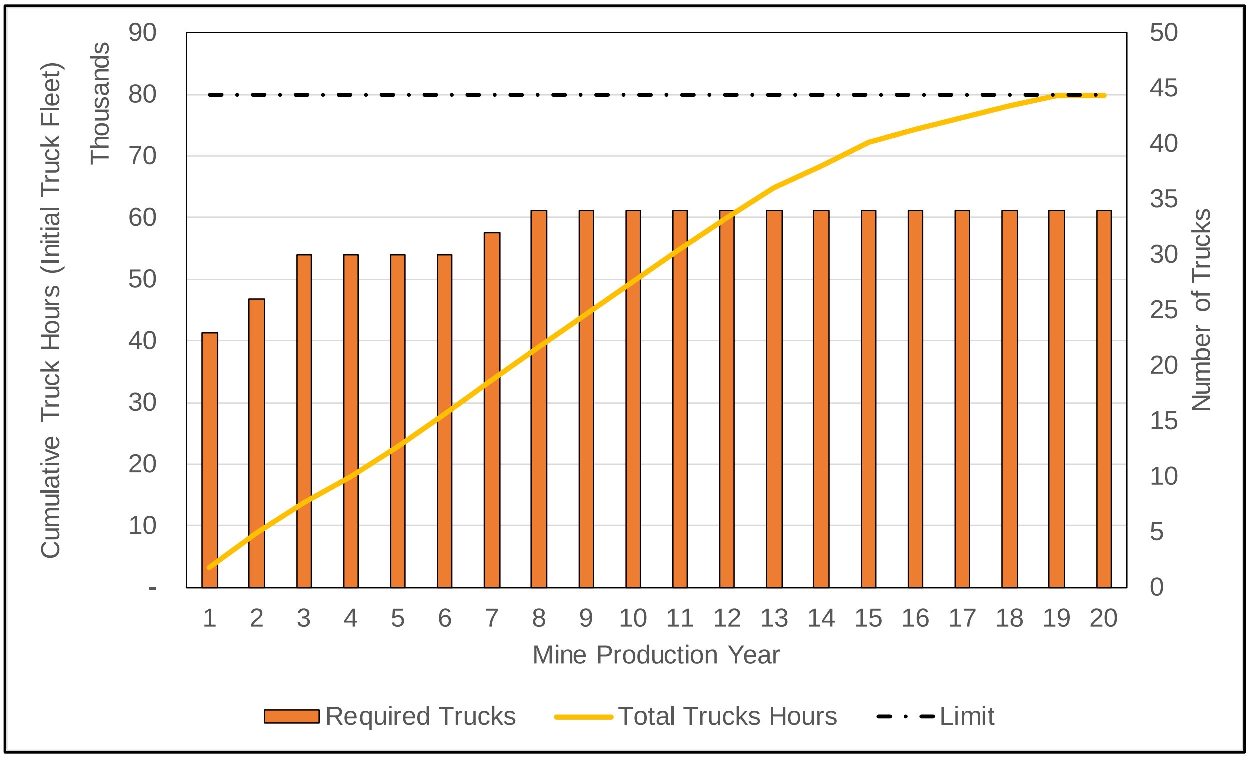
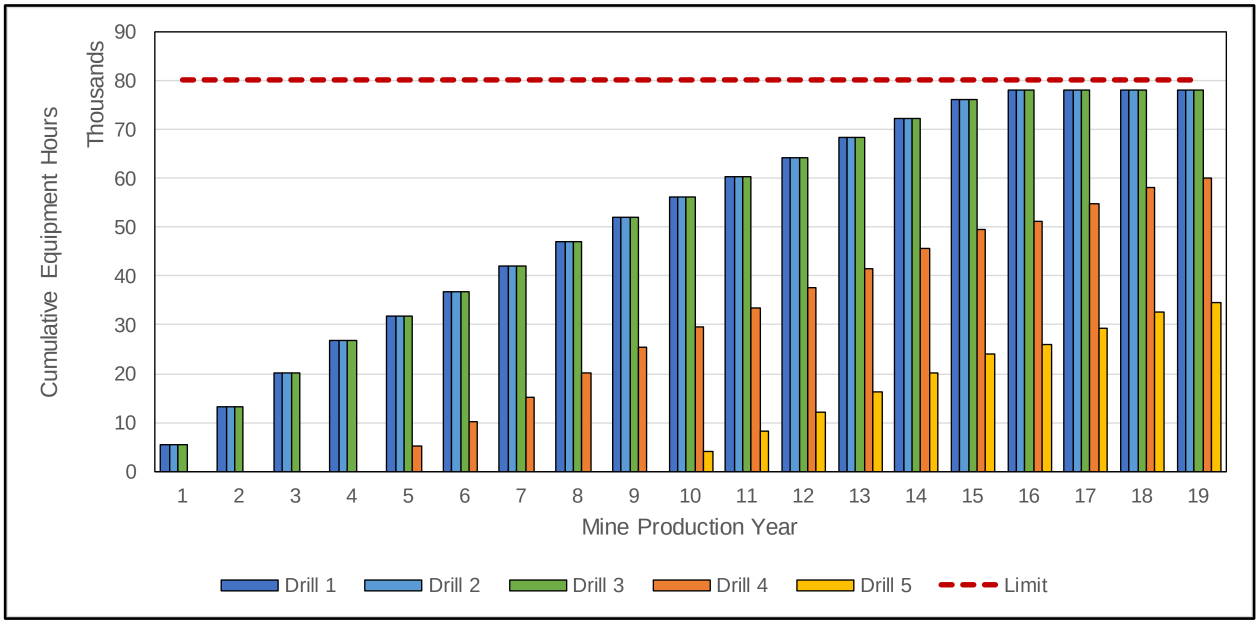


Figure 21-5: Comparison of Cumulative Initial Truck Hours & Industry Standard Lifespan (Golder, 2019)



The comparison in Figure 21-5 shows the initial truck hours do not exceed the industry standard lifespan of 80,000 hours. The scheduled, required truck hours vary by unit and no individual total truck hours exceed the industry standard lifespan.

Figure 21-6: Comparison of Cumulative Production Drill Hours & Industry Standard Lifespan (Golder, 2019)



21.2.10 Open Pit Mining Labor Assumptions

The maximum total non-staff labor requirement is 284 personnel, and the maximum total staff requirement is 37. Summaries of the non-staff and staff requirements by position are provided in Table 21-18 and Table 21-19, respectively. A summary of the maximum number of personnel for the mine is provided in Table 21-20. The total mining labor requirement is reduced to match the decreased production schedule (Figure 21-7). The labor rates include base labor and fringe benefits and are shown in Table 21-18 and Table 21-19. Nevada Copper provide these labor rates, which are fully burdened.

Table 21-18: Non-staff Rates & Personnel Requirements

Non-Staff Employees			
Position	**Full Annual Cost**	**Initial**	**Required**
Shovel Operator	$ 113,898	9	14
Loader Operator	$ 109,745	4	4
Dozer Operator	$ 103,918	20	29
Excavator Operator	$ 103,918	3	4
Drill Operator	$ 103,918	9	15
Grader Operator	$ 103,596	5	6
Haul Truck Operator	$ 97,801	65	135
Mine Utility Operator	$ 91,910	8	8
Service Truck Driver	$ 91,910	1	1
Apprentice	$ 83,782	2	4
Welder	$ 103,902	5	8
Electrician	$ 103,902	10	16
Diesel Mechanic	$ 103,902	10	16
Journeyman	$ 126,936	5	8

Non-Staff Employees			
Position	**Full Annual Cost**	**Initial**	**Required**
Crusher Operator	$ 103,017	3	4
Light Duty Mechanic	$ 98,477	5	8
Dispatch Operator	$ 99,808	4	4
	Non-Staff Total	**168**	**284**

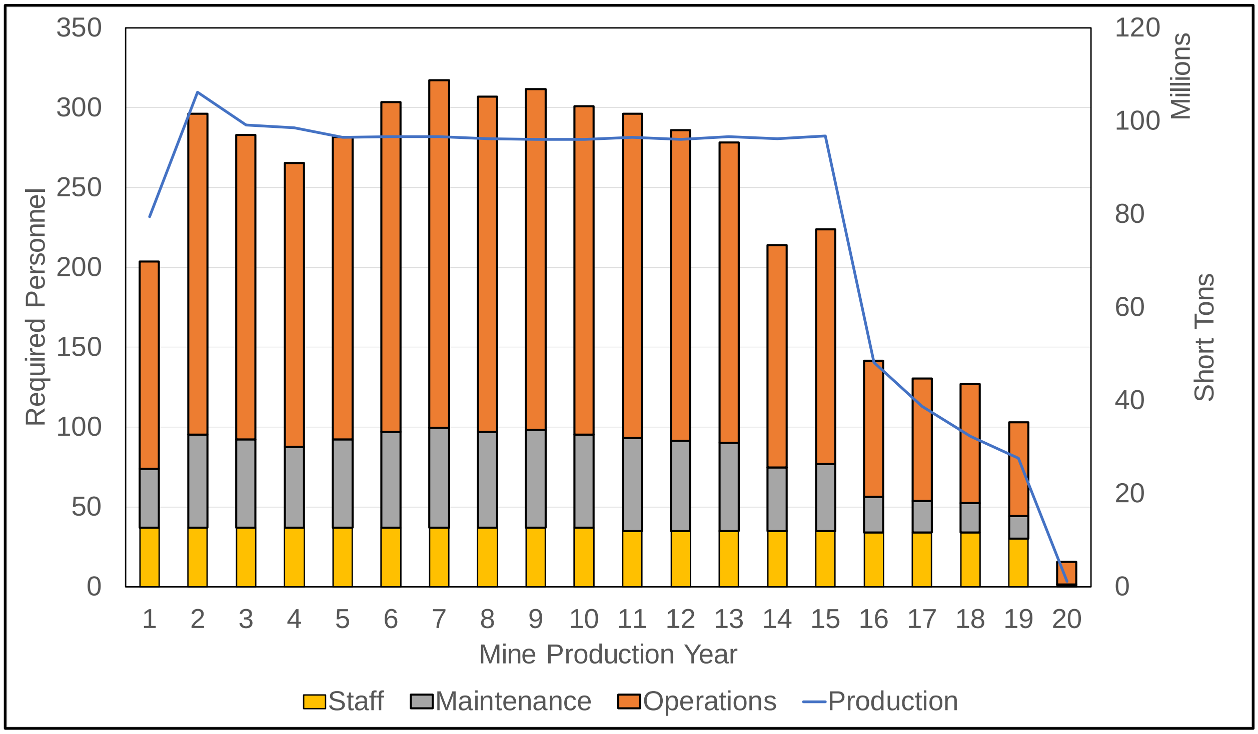
Table 21-19: Staff Rates & Personnel Requirements

Staff Employees		
Position	**Full Annual Cost**	**Required**
Administrative Assistant	$ 73,311	2
Maintenance Superintendent	$ 223,736	1
Maintenance Engineer	$ 132,992	3
Maintenance General Supervisor	$ 177,284	2
Maintenance Supervisor	$ 146,742	4
Maintenance Planner	$ 116,840	2
Mine Superintendent	$ 233,201	1
Operations General Supervisor	$ 177,606	2
Operations Supervisor	$ 146,412	8
Engineering Superintendent	$ 217,989	1
Mine Senior Planning Engineer	$ 155,514	1
Mine Planning Engineer	$ 132,759	2
Surveyor	$ 109,967	2
Survey Assistant	$ 87,277	2
Chief Geologist	$ 222,827	1
Geologist	$ 126,945	1
Geotechnical Engineer	$ 132,759	1
Geology Technician	$ 109,967	1
	Staff Total	**37**

Table 21-20: Total Required Labor Summary

Category	Required
Total Non-staff	284
Total Staff	37
Grand Total	321

Figure 21-7: Comparison of Require Mining Personnel & Production Schedule (Golder, 2019)



As the mining rate decreases, when only the South Pit is being mined, the total personnel is reduced to match the production, as depicted in Figure 21-7.

21.2.11 Open Pit Mining Drilling & Blasting Assumptions

A summary of the drilling and blasting assumptions used in the cost estimate are summarized in Table 21-21. Golder assumes that a contractor will provide loading, stemming and priming services for production and pre-shear blasting.

Table 21-21: Drilling & Blasting Assumptions

Drilling and Blasting Assumptions		
Parameter	**Unit**	**Value**
Drill Productivity	ft/hr	103
Time for Moving	mins	6
Electronic Cap Unit Price	$	37.8
Number of Caps per Hole	#	1
Booster Unit Price	$	4.3
Boosters per Hole	#	1
ANFO Price	$/lb	0.26
Pre-Split Explosive Cost	$/lb	0.26

Parameter	Unit	Ore Value	Waste Value	Pre-split Value
Ore Drill Diameter	inches	10 ⅝	10 ⅝	6
Ore Burden	ft	24	24	n/a
Ore Spacing	ft	30	30	6
Ore Powder Factor	lbs/by3	1.13	1.13	n/a
Ore Stemming	ft	16	16	n/a
Ore Sub drill	ft	6.5	6.5	n/a

Note: by^3 = bank cubic yards.

Golder recommends test blasts and optimization of the drill and blast assumptions during operations.

21.2.12 Open Pit Operating Costs

21.2.12.1 Open Pit Mine Operating Costs

LOM operating costs for the Open Pit Project are summarized in Table 21-22. A detailed bottom-up baseline cost model was constructed to estimate the forecast operating costs for the entire LOM. The production schedule was imported into the cost model and considered aspects such as incremental costs associated with depth and haulage, and so forth. The LOM summary of operating costs (Table 21-22) is reported as unit costs per ton. The total LOM operating costs for the Open Pit Project are estimated to be $1.42/st. The overall average LOM cost is based on the total LOM rock moved to include all pre-stripping.

Table 21-22: Life of Mine Operating Costs

Cost Category	Total LOM ($, millions)	LOM Unit Price ($/st)
General Mine & Engineering	98	0.06
Drilling	129	0.08
Blasting	345	0.22
Loading	295	0.19
Hauling	1,073	0.69
Support	225	0.13
Maintenance	95	0.06
Total1	2,259	1.42

Notes:

1. Includes all LOM operating costs related to pre-stripping.

21.2.12.2 Plant and Infrastructure Operating Cost

The operating costs associated with the DST include operation and maintenance costs for tailings haul trucks and related construction fleet.

The cost of hauling tails is based on the cost of operating the haul trucks and support equipment including fuel, maintenance and labor. Cycle times for haul of tailings were estimated using Runge TALPAC™ software and based on the following:

- The use of Komatsu 930e trucks with a capacity of 320 st of tailings
- Haul strings between the loading facility and the TSF for the period evaluated
- Maximum haul speed of 25 mph
- A fixed time of 4.34 minutes per load

A summary table of the calculated total LOM tailings transport trucking and equipment hours is provided in Table 21-23.

Table 21-23: Total LOM Tailings Transport & Equipment Hours

Equipment Type	Total LOM Hours (thousands)
Haul Truck	399
Compactor	71
Dozer	20

Construction fleet equipment operating costs include fuel and routine maintenance costs.

An allowance in the operating costs for dust suppression will be required to cover application of a dust suppressing agent to the exposed tailings surfaces using tanks mounted on the mobile fleet. TSF closure and reclamation costs are described elsewhere in the Open Pit PFS.

21.2.12.3 Surface Water Hydrology

Water management costs are an average $0.004/st-milled over the LOM. Costs are for a loader and a haul truck, which will perform channel and pond maintenance of sediments.

21.2.12.4 Concentrate Handling

Concentrate handling costs (establishment of loadout-rail handling facility for example) are described in Item 18.0 and applied in Item 22.0 of this Report.

21.2.12.5 Process Facility Operating Costs

Process facility operating costs for Phase I are estimated at $5.33/st-milled (Table 21-24). Process facility operating costs for the combined Phase I and Phase II are $5.04/st-milled as shown in the same table. The estimate does not include capital costs. The estimate was developed relying on first principle bases for labor, mobile equipment, maintenance, electric power, reagents, consumables and G&A.

Table 21-24: Process Facility Operating Costs per Short Ton

Plant Operating Costs	37 Kstpd PFS ($/st-milled)	70 Kstpd PFS ($/st-milled)
Labor	0.55	0.36
Maintenance	0.53	0.47
Mobile Equipment	0.03	0.03
Operating Consumables	2.60	2.60
Power	1.54	1.53
G&A	0.09	0.05
Total (Excluding G&A)	**5.24**	**5.00**

Electric Power

The estimated power usage was calculated from the electrical load list, applying demand, diversification and utilization factors to the installed power. Annual operating costs for the 37 Kstpd and 70 Kstpd production rates are shown in Table 21-25 and uses a rate of $0.0529/kWh provided by Nevada Energy.

Table 21-25: Electric Power Operating Costs

Area Description		37 Kstpd Total ($/a, millions)	70 Kstpd Total ($/a, millions)
110	Crushing	1.2	2.02
120	Grinding	10.44	20.86
130	Flotation	2.84	5.57
140	Thickening	0.89	1.69
150	Filtration	3.38	6.09
160	Reagents	0.03	0.06
170	Services	1.8	2.59
180	Dry Stack Tailing	0.02	0.05
190	Infrastructure	0.15	0.16
Subtotal Power		**20.76**	**39.07**
Subtotal Cost per Feed Short Ton		**$1.54/ROM st**	**$1.53/ROM st**

21.2.12.6 Grinding Media & Liners

Grinding media and liners operating were estimated against available test work. Annual operating costs for the 37 Kstpd and 70 Kstpd production rates are shown in Table 21-26.

Table 21-26: Grinding Media and Liners Operating Costs

Item	Detail	37 Kstpd Total Cost ($/a, millions)	70 Kstpd Total Cost ($/a, millions)
Crushing Primary Mantles	Including Liners	0.39	0.39
Crushing Pebble Crushing Mantles	Including Liners	0.2	0.39
Grinding SAG Mill Liners, Steel	-	3.2	6.39
Grinding Mill Media, Forged Steel Balls	-	8	15.14
Grinding Ball Mill Liners, Steel	-	0.78	1.56
Grinding Mill Media, Cast Steel Balls	-	5.24	9.91

Item	Detail	37 Kstpd Total Cost ($/a, millions)	70 Kstpd Total Cost ($/a, millions)
Grinding Regrind Mill Liners, Steel	-	0.07	0.15
Grinding Mill Media, Ceramic Beads	-	1.56	2.72
Subtotal Media and Liners	-	**19.43**	**36.65**
Subtotal Cost per Feed Short Ton	-	**$1.44/ROM st**	**$1.43/ROM st**

21.2.12.7 Reagents

Reagent usage costs were estimated using the available test work. Annual operating costs for each of the 37 Kstpd and 70 Kstpd production rates are shown in Table 21-27.

Table 21-27: Reagent Operating Costs

Item	Detail	37 Kstpd Total Cost ($/a, millions)	70 Kstpd Total Cost ($/a, millions)
Collectors/Promoters Aerophine 3418A		3.07	5.8
Collectors/Promoters Aero 3477		0.49	0.93
Frothers MIBC (Methyl Isobutyl Carbinol)		1.32	2.5
Modifiers Hydrated Lime (Ca(OH)2)		0.69	1.29
Dewatering Agents Polyacrylamide Flocculant, Powder	Concentrate	0.01	0.02
Dewatering Agents Polyacrylamide Flocculant, Powder	Tailing	1.76	3.33
Subtotal Reagents		**7.34**	**13.88**
Subtotal Cost per Feed Short Ton		**$0.54/ROM st**	**$0.54/ROM st**

21.2.12.8 Other Process Consumables

This estimate includes consumables associated with tailings dewatering and testing of metallurgical samples and external laboratory fees.

Tailings dewatering filter cloths are based costs applied to expected cloth lives for the proposed filters. Filtering of both concentrate and tailings is included.

Laboratory costs include metallurgical samples consumables, on-stream analysis consumables, consignment certification, metallurgical assays and sample transport.

The operating costs associated with other process consumables for the 37 Kstpd and 70 Kstpd production rates are $0.62/ROM st and $0.63/ROM st respectively.

21.2.12.9 Labor

The labor operating cost was determined based on an owner operating the process plant on an operator-maintainer model. The labor force was determined to be 77 personnel of which:

- Twenty-two people cover thirteen roles on a one-panel, 5/2 days only roster.
- Twenty-two people cover six roles on a two-panel, 7/7 days only roster.
- Thirty-two people cover seven roles on a four-panel, 7/7 day/night roster.

Labor rates are based on the Nevada Mining Association's 2018 Annual Wage and Salary Survey adopting the P_{50} values and applying a burden calculated to average 23.5%. The burden has been calculated to include FICA (Medicare and Social Security), FUTA, Nevada Unemployment, Nevada Modified Business Tax, Workers Compensation, 401K Contributions and Medical.

The operating cost associated with the labor for the 37 Kstpd and 70 Kstpd production rates are $0.55/ROM st and $0.36/ROM st, respectively

21.2.12.10 Maintenance

Maintenance operating costs follow the same design methodology as the Underground Project (Item 21.1.9.6). Annual operating costs for the 37 Kstpd and 70 Kstpd production rates are shown in Table 21-28.

Table 21-28: Maintenance Operating Costs

Area Description		37 Kstpd and 70 Kstpd Allowance (% Capex)	37 Kstpd Total Cost ($/a, millions)	70 Kstpd Total Cost ($/a, millions)
110	Crushing	4	0.73	1.02
120	Grinding	4	1.82	3.35
130	Flotation	4	0.7	1.27
140	Thickening	4	0.67	1.06
150	Filtration	4	2.36	3.99
160	Reagents	4	0.07	0.15
170	Services	2	0.61	1
180	Tailings Filtration	2	0.06	0.13
190	Infrastructure	2	0.08	0.16
Subtotal Maintenance			**7.11**	**12.13**
Subtotal Cost per Feed Short Ton			**$0.53/ROM st**	**$0.47/ROM st**

21.2.12.11 Mobile Equipment

The plant mobile equipment operating cost assessed includes the fleet general site vehicles for plant and maintenance purposes. It excludes mobile equipment for managing the rehandle of the low grade and high strength ores and equipment for placement of the dry stack material

Annual operating costs for the 37 Kstpd and 70 Kstpd production rates is $0.03/ROM st.

21.2.12.12 Infrastructure

The operating cost associated with the Open Pit Project infrastructure is captured in the maintenance cost described in Item 21.2.12.10.

21.2.12.13 Other Costs

Environmental and reclamation costs for the Open Pit Project are included within the environmental program plan described in Item 20.

21.2.13 Dry Stack Tailings Facility

DST storage facility costs are shown in Table 21-29. These costs are presented by average LOM cost category and total $0.35/st-milled. Operating cost categories include labor for maintenance of the dry stack ($0.06/st-milled average LOM) and fleet equipment to support maintenance ($0.27/st-milled average LOM).

Table 21-29: Dry Stack Tailings Storage Facility Costs

Cost Category	Average LOM ($/st -milled)
Labor	0.06
Fleet Equipment	0.27
Total	0.35

Water management costs are an average $0.004/st-milled over the LOM. Costs are for a loader and a haul truck, which will perform channel and pond maintenance of sediments.

21.2.14 General and Administrative

21.2.14.1 Owner's Cost

The G&A cost for the pre-production capital cost period is based on a similar basis as for the G&A Opex.

The key differences being this cost is capitalized and the following:

- Operating insurance is replaced by cost of construction insurance during this period.
- Consultants cost includes $1.2 million for basic engineering work.
- The site-specific Real Property Tax is lower as the purchases increase over time during construction.
- Site management and other shared costs are lower during pre-production and increase at a production phase.

The pre-production capital cost per year is summarized in Table 21-30.

Table 21-30: Annual Pre-production Capital General & Administrative Costs

Capital Cost Description	Unit	2021	2022
Health & Safety, First Aid etc.	$, millions	0.0	0.0
Community & Social	$, millions	0.0	0.0
Senior Site Management & Admin	$, millions	0.7	0.7
Shared Facilities, Services, Site Security	$, millions	0.7	0.7
Consultants	$, millions	1.2	0.0
Course of Construction Insurance (0.4% build cost)	$, millions	2.4	2.4

Capital Cost Description	Unit	2021	2022
Real Property Tax	$, millions	0.0	0.7
Total	$, millions	5.1	4.5

Note: In the economic model for the study, the G&A costs are combined with water management and environment etc. to form an overall G&A in the modeling.

G&A expenses have been estimated for areas of the Open Pit Project, including:

- Personnel
- Materials, software and supplies
- Health and safety
- Community relations and social
- Senior site management and administration
- Shared site services
- Mobile vehicles
- Consultants
- Course of construction insurance
- Operating insurance
- Site-specific real property tax

Related costs for environment and water are addressed in separate report sections.

When production of the copper ore starts, the costs will be reported as operating expenses.

Personnel and materials expenses are defined as those that are shared across all open pit operations, facilities and management; they exclude all management and technical functions of specific operational areas, as those costs are included in those separate cost areas.

Annual costs have been determined using the 2018 Nevada Mining Association Annual Wage and Salary Survey as provided by Nevada Copper. A total of 35 personnel is determined for the G&A component of site management. These are presented in Table 21-31. Roles with a 0.5 number indicate these are assumed to be shared with the stand-alone Underground Project providing an overall site function.

Other costs such as utility vehicles, supplies and materials included in the G&A estimate include:

- Specialist software
- Scholarships and community funds
- Light utility vehicles

- Management of shared and administrative facilitates buildings
- External technical consultants
- Insurance during construction and in operation

Table 21-31: General & Administrative Personnel

Personnel	Number
H&S Senior Specialist	1
H&S Administrator	1
Community Relations Manager	0.5
Community & Social Specialist	0.5
Community & Social Administrator	0.5
Site Senior VP	1
General Manager's Assistant (Executive Secretary)	1
Financial Controller	1
Finance Senior Clerk	1
Payroll (Payroll Clerk)	1
Accounts Payable	1
Accounts Receivable	1
Accounting Clerk	1
HR Manager	1
HR Assistant	1
Shared Services General Manager	1
Planner	1
Administrator	1
Site Yard Laborer	1
Site Carpenter	1
Janitorial	4
Gate Security	6
Laborer/Trainee	1
IT Manager	1
Purchasing Supervisor	1
Warehouse	1
Contracts Manager	1
Contract Administrator	1
Total	35

G&A expenses have been estimated for areas of the Open Pit Project. The operating cost is summarized in Table 21-32 on an annual basis and per ton basis at 70 Kstpd production rate.

Table 21-32: General & Administrative Cost and Per Ton ROM (First Year of Full Production)

Operating Cost	$/Year, millions	$/st RoM
Description		
Health & Safety, First Aid etc.	0.3	0.01
Community & Social	0.4	0.02
Senior Site Management & Admin	2.2	0.09
Shared Facilities, Services, Site security	2.1	0.08
Consultants	0.2	0.01
Operating Insurance	1.0	0.04
Real Property Tax	2.6	0.10
Total	8.8	0.35

The operating cost per year of the LOM remains similar to that above, with the exception of the Real Property tax which gets depreciated on an annual basis. The impact being by the last year of the LOM the G&A annual cost has reduced by $0.6 million compared to the first year shown above.

21.2.15 Concentrate Transport & Shipping

The costs of concentrate transportation to smelters are not considered site operating costs and are therefore excluded from the Opex.

Transportation costs are deducted from gross smelter revenues, along with smelter conversion charges (TC and RCs), to yield NSR. They are described in Item 18.2 and applied in Item 22.2 of this Report.

Item 22.0 ECONOMIC ANALYSIS

This Item includes references to non-GAAP and non-IFRS measures such as "EBITDA" and "AISC" that are metrics commonly used in the mining industry and are provided herein for reference purposes. EBITDA refers to "Earnings Before Income Tax Depreciation and Amortization"; AISC refers to "All-In Sustaining Costs." EBITDA is a close proxy to the Pre-tax Cashflow from Operations metric presented on financial statements, while AISC is a unit cost of production per pound of copper metric that includes initial and sustaining capital costs.

22.1 Underground

22.1.1 Key Model Assumptions

All figures in the economic analysis are shown on an unlevered 100% project basis (except where indicated in Item 22.5) and are based upon the projections of ore tons and grades, operating and capital, and other costs as disclosed in earlier sections.

A summary of the projected mill feed tons and grades for the first five years of production from the Underground Project and for the LOM, are shown in Table 22-1.

Table 22-1: Summary of Tons Milled in First 5 Years & Life of Mine

Description	Units	5 Years from Q4 2019	LOM
Ore Milled	Kst	8,970	23,909
Copper in Feed	Kst	162	380
Copper Feed Grade	%	1.80%	1.59%
Copper Grade	% CuEq	1.98%	1.74%
Gold	Koz	68	153
Gold Grade	oz/st	0.0076	0.0064
Gold	g/tonne	0.26	0.22
Silver Grade	oz/Kst	1,459	3,333
Silver	g/tonne	0.16	0.14

22.1.2 Metal Prices

The economic viability of the Underground Project has been evaluated using the metal prices outlined in Table 22-2. The metal prices used in the economic analysis are based on a blend of consensus metal price forecasts.

Table 22-2: Metal Price Assumptions (Consensus Prices)

Description	Units	2019	2020	2021	2022+
Consensus Copper Prices	$/lb	$2.83	$3.05	$3.14	$3.20
Consensus Gold Prices	$/oz	$1,276	$1,285	$1,284	$1,325
Consensus Silver Prices	$/oz	$18.77	$19.40	$19.53	$20.01

Source: Consensus Economics Inc. - August 2017.

The Consensus Economics Inc. copper price forecast of August 2017 is still considered current and relevant for the purpose of this Report.

At the realized metal prices shown above, the revenue breakdown of payable metal recovered to concentrates at Pumpkin Hollow is approximately 93% Cu, 5% Au and 2% Ag.

22.1.3 Working Capital

Working capital is a temporary use of cash and is fully recovered at the end of mine life and these estimated requirements have been allowed for in the Underground Project financial model. Working capital is the net amounts of capital need to finance operations in order to allow for delays in the receipt of revenues, less the normal delay in paying accounts payable.

It is expected that approximately $22 million of working capital will be required to fund operations and expenditures during the ramp-up phase.

22.1.4 Capital and Operating Costs

Operating costs, initial and sustaining capital costs, and closure costs are detailed in earlier sections of this Report and are incorporated in the appropriate areas of the cash flow model.

22.1.5 Economic Analysis Methodology

The Underground Project economics were evaluated using a standard discounted cash flow model. The financial indicators examined for the project included the net present value (NPV), internal rate of return (IRR) and payback period (time in years to recapture the initial capital investment).

In the model, annual projection of net cash flow were estimated for each year over the life of the mine. The annual net cash flow is determined based on:

- Tons of ore mined and processed, with the associated grades of metals
- Metals recovered, yielding gross smelter revenues less:
  - Concentrates transportation costs
  - Treatment and refining charges
  - Royalties
  - Mine operating costs
  - Working capital changes
  - Initial capital expenditures
  - Sustaining capital expenditures
  - Closure costs
  - State and federal taxes

The resulting annual net cash flows were used to calculate a variety of economic indicators and for a number of metal price and costs sensitivity scenarios. All Underground Project costs spent prior to 2017, $210 million, are considered “sunk” and are not included in the cash flow projections, except indirectly. These sunk costs are included in the relevant tax pools and available to reduce future income taxes otherwise payable.

The economic analysis for the Underground Project contained in this Report was conducted without reference to the Stream Agreement. It was based on the sale of gold and silver to offtakers at the metal price assumptions set forth in Table 22-2 and does not reflect the impact on corporate-level cash flows of satisfying the separate delivery obligations under the Stream Agreement. As required under the terms of the Stream Agreement, the Stream Agreement did not impact how Nevada Copper developed its mine plan for the Underground Project or how it intends to operate the Underground Project. As a result of the Stream Agreement, it is likely that the cost of acquiring gold and silver credits for delivery thereunder will exceed the amount of the payments for those credits received from Triple Flag (being 10% of spot price and the amortization of the $70 million Stream Deposit) over the life of the Underground Project. See the description of the Stream Agreement in Item 1.1.

22.1.6 Royalties

Royalties on non-ferrous minerals are payable to RGGS as described in Item 4.

22.1.7 Income & Other Taxes

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalties, property taxes, State mining taxes, reclamation and closure expense, depreciation, tax loss carry forwards and percentage depletion. Depreciation rates for project capital were the seven-year Modified Accelerated Cost Recovery System as shown in Table 22-3. Sunk costs were amortized over 10 years. The basic percentage depletion deduction is determined as the lesser of 15% of gross and 50% of net income as defined. Percentage depletion can give rise to “claw backs” under the Alternative Minimum Tax system that somewhat reduces the benefit of large percentage depletion deductions that may be available.

Table 22-3: Seven-Year Modified Accelerated Cost Recovery System Depreciation Rates

Recovery Year	1	2	3	4	5	6	7	8
Depreciation %	14.29	24.49	17.49	12.49	8.93	8.92	8.93	4.46

As previously mentioned, project costs spent prior to 2017 of $210 million, are considered “sunk” and are included in the relevant tax pools that are available to reduce future income taxes otherwise payable. With the substantial tax pools available due to prior investment in the project, federal income taxes are estimated to be relatively low and not become payable for several years, depending on the level of future metals prices and costs. The sunk costs are applied only to the income tax calculations for the Underground Project alone. Also, the income taxes estimated for the Underground and Open Pit projects are calculated separately on a stand-alone basis.

For Nevada, the nominal corporate tax rates are summarized in Table 22-4.

Table 22-4: Tax Rates

Tax Regime	Tax Rate
Nevada State Corporate income tax:	Not applicable
Nevada Net Proceeds of Mining Tax (NPOMT)	variable but 5% of taxable profit (as defined) for most large mines
Federal Corporate Income Tax	35% (nominal)

The corporate income taxes were estimated on the taxable income described above using the statutory federal rates. There is no Nevada corporate income tax, however there is a state tax on mining operations, the Nevada Net Proceeds of Mining Tax, which is a deduction for federal income tax purposes. Nevada Net Proceeds of Mining Tax were estimated based on the regulations as generally understood. The tax rate is 5% if net proceeds are over $4 million annually. Net proceeds are gross revenues less smelter charges, site operating expenses, and allowances for depreciation of capital expenditures. Otherwise the rate is variable based on the ratio of "net proceeds" to "gross revenues" with a maximum rate of 5% of "net proceeds."

The corporate income taxes calculated are estimates only based on the understanding of the taxation of US mining income, as it is currently defined in legislation, and assume a single project, Pumpkin Hollow, with no other corporate activities.

22.1.8 Results of Economic Analysis

Metal prices employed the mean of analyst's consensus prices for copper gold and silver from 2017 to 2021; thereafter, the prices were held constant. These metals prices are shown in Table 22-5. The metal price forecast of 2017 is considered current and relevant for the purpose of this Report.

Table 22-5: Metal Prices

Type	Unit	2019	2020	2021	2022+
Consensus Copper Prices	$/lb	$2.83	$3.05	$3.14	$3.20
Consensus Gold Prices	$/oz	$1,276	$1,285	$1,284	$1,325
Consensus Silver Prices	$/oz	$18.77	$19.40	$19.53	$20.01

Source: Consensus Economics Inc. - August 2017

The economics were also examined with alternate metals price scenarios with copper prices lower and higher than current spot prices as shown Table 22-6. Gold and silver prices were held constant at the levels show due to their low importance relative to copper. All prices were held constant.

Table 22-6: Alternate Metal Price Scenarios

Metal	Unit	Low	High
Copper	$/lb	$2.60	$3.50
Gold	$/oz	$1,300	$1,300
Silver	$/oz	$17.00	$17.00

The economic analysis of the Underground Project at a copper price of $3.00/lb (Table 22-7), results in an after tax NPV (NPV 5%) of $247 million at a discount rate of 5%, an IRR of 22.9% and a capital payback period of 4.9 years. The LOM is 13.1 years.

Table 22-7: Economic Comparison

Description	Units	Low Price Case	Consensus	High Price Case
Copper Price	$/lb	$2.60	Consensus	$3.50
Gold Price	$/oz	$1,300	Consensus	$1,300
Silver Price	$/oz	$17	Consensus	$17
		$, millions	**$, millions**	**$, millions**
Net Smelter Revenue, after royalty	LOM	1,582	1,941	2,150
Operating Margin (EBITDA)1	LOM	518	876	1,085
Operating Margin (EBITDA)1	Avg./Yr	40	67	83
NPV 0%	Pre-tax	224	582	791
NPV 0%	After-tax	212	496	658
NPV 5%	Pre-tax	108	356	510
NPV 5%	After-tax	100	301	421
IRR	Pre-tax (%)	13.4	27.2	36.8
IRR	After-tax (%)	12.8	25.2	33.6
Payback – years	After-tax (yr)	6.50	4.75	4.00

Note:

1. Net revenues less smelter charges, concentrate transport, site operating costs, royalties and mining taxes.

22.1.9 Sensitivity Analyses

Mining projects have the greatest sensitivity to changes in metals price assumptions. Table 22-8 depicts the sensitivity of economic indicators to copper prices in the table below for various copper prices. Gold and silver prices are only a relatively small component of project revenues and are held constant at $1,300/oz and $17/oz, respectively.

Table 22-8: Economic Indicators vs Copper Price

Description		Copper Price ($/lb)				
		$2.60	**$2.75**	**$3.00**	**$3.25**	**$3.50**
After-Tax NPV5	$, millions	$99.7	$157.6	$246.7	$333.9	$421.2
After-Tax IRR	%	12.8%	16.9%	22.9%	28.4%	33.6%
Payback Period	Years	6.50	5.83	5.00	4.42	4.00
Operating Margin	Avg/Yr	$39.5	$46.8	$58.8	$70.8	$82.9

Note: Assumes $1,300/oz gold and $17/oz silver.

Charts displaying the sensitivity of key economic results to percentage variations from the base case, in capital costs, operating costs and metals are shown below in Figure 22-1 and Figure 22-2. The annual production cashflow projection for the whole property is presented in Table 22-9.

Figure 22-1: Internal Rate of Return Sensitivity (Nevada Copper, 2019)

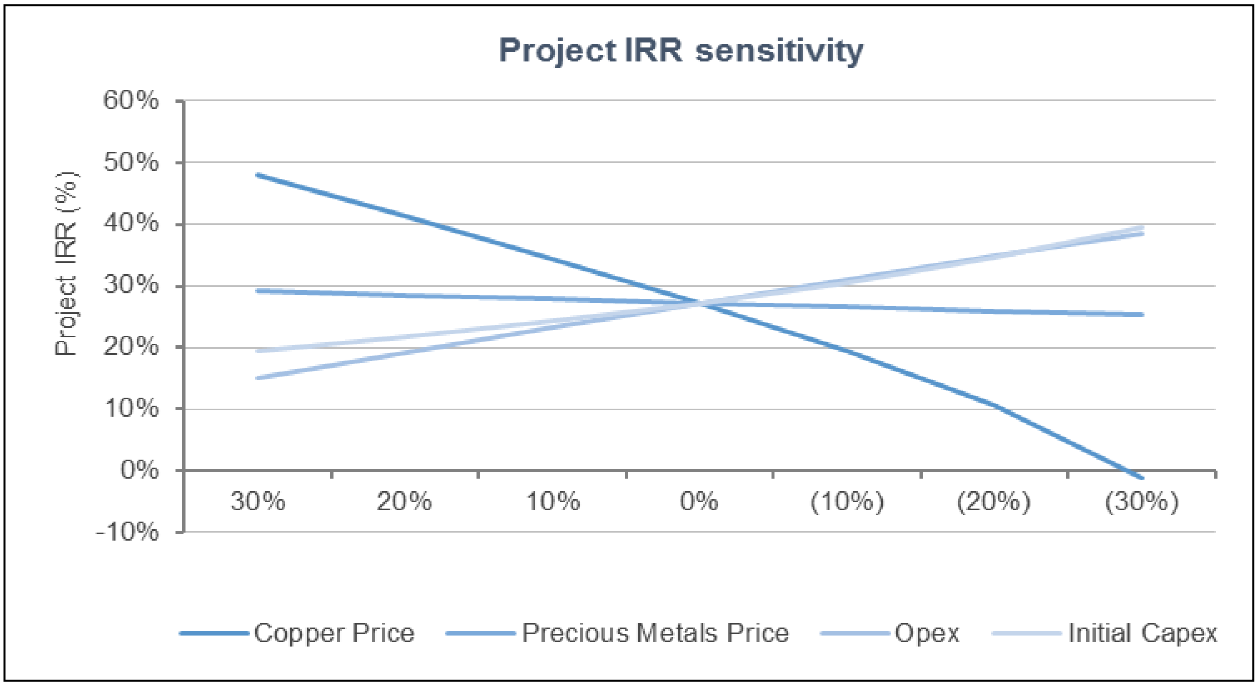


Figure 22-2: Net Present Value 5% Sensitivity (Nevada Copper, 2019)

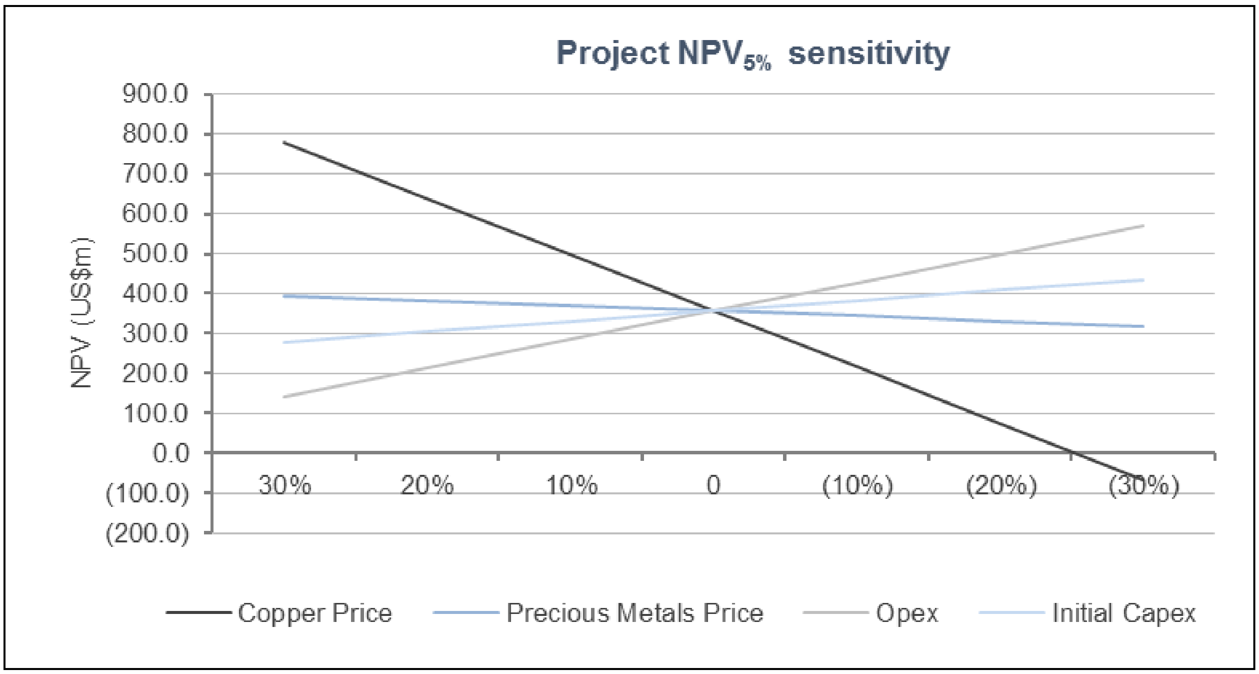


Table 22-9: Underground Mine Annual Cash Flow Projection

Summary - PFS Annual Results

Nevada Copper

Macro Economic Case: Consensus

Operating Case: 5000tpd PFS Mine Plan

Year			2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Period	Units	Total / Avg	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Macro Economics																	
Copper Price	US$ / lb	3.14	2.68	2.83	3.05	3.14	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
Gold Price	US$ / oz	1,316	1,268	1,276	1,285	1,284	1,325	1,325	1,325	1,325	1,325	1,325	1,325	1,325	1,325	1,325	1,325
Silver Price	US$ / oz	19.8	18.21	18.77	19.4	19.53	20.01	20.01	20.01	20.01	20.01	20.01	20.01	20.01	20.01	20.01	20.01
Gold Equivalent Basis	*Ratio*	*66.5*	*69.6*	*68*	*66.2*	*65.7*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*	*66.2*
Production																	
Ore Feed	kst	23,909	0	1,212	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	798
Copper Grade	%	1.59%	0.00%	1.56%	1.80%	1.92%	1.79%	1.72%	1.69%	1.49%	1.54%	1.59%	1.52%	1.44%	1.33%	1.26%	0.68%
Gold Grade	g/st	0.2	0	0.15	0.19	0.24	0.28	0.27	0.27	0.19	0.17	0.17	0.18	0.18	0.16	0.15	0.08
Silver Grade	g/st	4.34	0	3.17	4.09	5.34	5.84	5.72	5.27	3.73	3.64	3.64	3.97	4.21	3.85	3.55	1.99
Contained Copper	klbs	759,388	0	43,731	65,836	69,931	65,187	62,668	61,842	54,439	56,113	58,085	55,388	52,512	48,383	45,961	19,310
Contained Gold	koz	153	0	6	11	14	16	16	16	11	10	10	10	10	9	9	4
Contained Silver	koz	3,333	0	140	240	313	343	336	309	219	214	214	233	247	226	209	91
Recovered Copper	klbs	698,637	0	40,233	60,570	64,337	59,972	57,655	56,895	50,084	51,624	53,438	50,957	48,311	44,512	42,284	17,765
Recovered Gold	koz	120	0	5	9	11	13	12	12	9	8	8	8	8	7	7	3
Recovered Silver	koz	2,333	0	98	168	219	240	235	217	153	150	150	163	173	158	146	64
Payable Copper	klbs	670,691	0	38,623	58,147	61,763	57,573	55,348	54,619	48,081	49,559	51,300	48,919	46,379	42,732	40,593	17,055
Payable Gold	koz	109	0	5	8	10	12	11	11	8	7	7	7	7	7	6	3
Payable Silver	koz	2,100	0	88	151	197	216	212	195	138	135	135	147	156	142	131	58
Copper Reserve Tail	%		100.00%	94.20%	85.60%	76.40%	67.80%	59.50%	51.40%	44.20%	36.80%	29.20%	21.90%	15.00%	8.60%	2.50%	0.00%
Mine Life	Years	13.1															

Table 22-9: Underground Mine Annual Cash Flow Projection (cont.)

Financials																	
Gross Copper Revenue	US$k	2,117,874	0	109,489	177,071	193,655	184,094	176,980	174,648	153,741	158,469	164,036	156,421	148,300	136,638	129,799	54,533
Gross Gold Revenue	US$k	142,819	0	5,755	10,234	12,651	15,022	14,755	14,802	10,341	9,584	9,249	9,847	9,767	8,774	8,317	3,722
Gross Silver Revenue	US$k	41,445	0	1,607	2,838	3,827	4,218	4,233	3,899	2,762	2,696	2,693	2,938	3,111	2,845	2,628	1,152
Total Gross Revenue	US$k	2,302,137	0	116,851	190,142	210,134	203,334	195,968	193,348	166,843	170,748	175,978	169,206	161,177	148,256	140,744	59,407
Treatment Charges	US$k	-100,554	0	-5,791	-8,718	-9,260	-8,632	-8,298	-8,189	-7,208	-7,430	-7,691	-7,334	-6,953	-6,407	-6,086	-2,557
Refining Charges	US$k	-51,687	0	-2,955	-4,462	-4,760	-4,463	-4,291	-4,230	-3,700	-3,807	-3,936	-3,765	-3,577	-3,295	-3,128	-1,316
Transport	US$k	-90,683	0	-5,222	-7,862	-8,351	-7,784	-7,484	-7,385	-6,501	-6,701	-6,936	-6,614	-6,271	-5,778	-5,489	-2,306
Total TC, RC, Shipping	US$k	-242,923	0	-13,968	-21,041	-22,371	-20,879	-20,073	-19,804	-17,410	-17,938	-18,564	-17,713	-16,802	-15,479	-14,703	-6,179
NSR	US$k	2,059,214	0	102,883	169,101	187,763	182,455	175,895	173,544	149,434	152,810	157,414	151,493	144,376	132,777	126,041	53,228
RGGS Royalty	US$k	-118,547	0	-2,936	-10,015	-11,101	-10,755	-10,364	-10,226	-8,835	-9,046	-9,325	-8,962	-8,534	-7,850	-7,453	-3,145
Mining Cost	US$k	-746,364	-9,959	-62,130	-65,932	-56,653	-55,693	-54,475	-51,965	-55,122	-53,136	-52,871	-52,288	-51,373	-50,463	-54,701	-19,604
Milling Cost	US$k	-241,007	0	-12,212	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-18,396	-8,042
G&A Cost	US$k	-77,046	0	-4,555	-6,324	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-5,785	-2,529
Total Opex	US$k	-1,182,964	-9,959	-81,834	-100,667	-91,935	-90,629	-89,020	-86,372	-88,138	-86,363	-86,378	-85,431	-84,088	-82,494	-86,335	-33,321
Pre-Tax Operating Cash Flow	US$k	876,250	-9,959	21,049	68,434	95,828	91,826	86,874	87,173	61,295	66,447	71,036	66,063	60,288	50,283	39,706	19,908
Plant	US$k	-80,200	-71,378	-8,822	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Development Initial	US$k	-42,350	-29,100	-13,250	0	0	0	0	0	0	0	0	0	0	0	0	0
Off-site Infrastructure	US$k	-26,200	-24,104	-2,096	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirects	US$k	-24,709	-22,732	-1,977	0	0	0	0	0	0	0	0	0	0	0	0	0
Contingency	US$k	-8,701	0	-8,701	0	0	0	0	0	0	0	0	0	0	0	0	0
First Equipment Lease	US$k	-1,542	-1,031	-511	0	0	0	0	0	0	0	0	0	0	0	0	0
Initial Capex	US$k	-183,702	-148,346	-35,356	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capex	US$k	-110,602	0	-21,542	-14,293	-17,208	-7,629	-4,859	-4,831	-4,242	-3,858	-5,254	-6,657	-9,545	-6,324	-1,241	-2,340
Sustaining Capex	US$k	-110,602	0	-21,542	-14,293	-17,208	-7,629	-4,859	-4,831	-4,242	-3,858	-5,254	-6,657	-9,545	-6,324	-1,241	-2,340
Total Capex	US$k	-294,304	-148,346	-56,898	-14,293	-17,208	-7,629	-4,859	-4,831	-4,242	-3,858	-5,254	-6,657	-9,545	-6,324	-1,241	-2,340
Taxes (unlevered)	US$k	-85,962	0	-186	-2,204	-3,666	-3,458	-8,315	-10,583	-6,205	-9,050	-10,415	-9,771	-8,702	-6,703	-4,598	-2,106
Change in Working Capital	US$k	0	2,336	688	-1,540	-1,627	-447	1,325	651	689	-1,279	535	-215	533	-337	-326	-988
Pre-Financing Cash Flow	US$k	495,985	-155,969	-35,347	50,398	73,327	80,292	75,026	72,411	51,537	52,260	55,902	49,420	42,574	36,918	33,541	14,474
Cumulative Free Cash Flow	*US$k*		*-155,969*	*-191,316*	*-140,917*	*-67,590*	*12,702*	*87,728*	*160,139*	*211,676*	*263,936*	*319,838*	*369,258*	*411,832*	*448,750*	*482,291*	*496,765*

22.2 Open Pit

22.2.1 Introduction

The economic analysis for the Open Pit Project has been carried out based on mining schedules, cost estimation and related infrastructure plus economic analyses to support a stand-alone open pit mine and process plant.

22.2.2 Model Assumptions

All figures in the economic analysis are shown on an unlevered 100% Open Pit Project and are based upon the projections of ore tons and grades, operating and capital, and other costs as disclosed in earlier sections.

A summary of the projected mill feed tons and grades for the first five years of production, and for the LOM, are shown in Table 22-10.

Table 22-10: Summary of Tons Milled in First 5 Years & Life of Mine

Category	Units	Year 1-5	LOM
Ore Milled	Kst	64,445	385,693
Copper in Feed	Kst	421	1,749
Copper Feed Grade	%	0.65	0.47
Copper Grade	% CuEq	0.69	0.50
Gold	oz	120	617
Gold Grade	oz/st	0.0019	0.0016
Gold	g/tonne	0.064	0.055
Silver Grade	oz/st	0.069	0.055
Silver	g/tonne	2.367	1.890

22.2.3 Metal Prices

The economic viability of the stand-alone Open Pit Project has been evaluated using the metal prices outlined in Table 22-11. The metal prices used in the economic analysis for the Open Pit Project are based on a blend of consensus metal price forecasts. Nevada Copper considers the consensus valid at the effective date of this Report.

Table 22-11: Metal Price Assumptions (Consensus Prices)

Price	Unit	2019	2020	2021	2022+
Consensus Copper Prices	$/lb	2.83	3.05	3.14	3.20
Consensus Gold Prices	$/oz	1,276	1,285	1,284	1,325
Consensus Silver Prices	$/oz	18.77	19.40	19.53	20.01

Source: Consensus Economics Inc. - 2017.

At the realized metal prices shown above, the revenue breakdown of payable metal recovered to concentrates at the Open Pit Project is approximately 93% copper, 5% gold and 2% silver.

22.2.4 Working Capital

Working capital is a temporary use of cash and is fully recovered at the end of mine life, and these estimated requirements have been allowed for in the project financial model. Working capital is the net amounts of capital needed to finance operations in order to allow for delays in the receipt of revenues, less the normal delay in paying accounts payable. It is expected that approximately $26 million of working capital will be required to fund operations and expenditures during the ramp-up phase.

22.2.5 Capital & Operating Costs

Operating costs, initial and sustaining capital costs, and closure costs are detailed in earlier sections of this Report and are incorporated in the appropriate areas of the cash flow model.

22.2.6 Economic Analysis Methodology

The stand-alone Open Pit Project economics were evaluated using a standard discounted cash flow model. The financial indicators examined for the project included the NPV, IRR and payback period (time in years to recapture the initial capital investment).

In the model, annual projections of net cash flow were estimated for each year over the life of the mine. The annual net cash flow is determined based on:

- The tons of ore mined and processed, with the associated grades of metals
- The metals recovered; yielding
- Gross smelter revenues, less:
  - Concentrates transportation costs
  - Treatment and refining charges
  - Royalties
  - Mine operating costs
  - Working capital changes
  - Initial capital expenditures
  - Sustaining capital expenditures
  - Closure costs
  - State Mining Tax
  - Federal Income taxes

The resulting annual net cash flows were used to calculate a variety of economic indicators and for a number of metal price and costs sensitivity scenarios. All Project costs spent prior to the Open Pit study are considered “sunk” and are not included in the cash flow projections. These sunk costs are not included in any taxation considerations in the economic analysis of the Open Pit Project.

22.2.7 Royalties

Royalties on non-ferrous minerals are payable to RGGS Land and Minerals, as described in Item 4.0.

22.2.8 Income & Other Taxes

U.S. federal income taxes were estimated assuming that the Open Pit Project would be developed on a stand-alone basis and using the new measures included for corporations in the recent U.S. tax reform legislation. These measures included:

- A 21% tax rate (previously 35%)
- Elimination of the Alternative Minimum Tax
- An 80% annual income limitation was applied to future tax loss carry forward

Standard depletion deductions were applied: the greater of cost and percentage depletion, and a standard seven-year Accelerated Cost Recovery System rate for all assets.

22.2.9 Results of Economic Analysis

Stand-alone Open Pit Project PFS level cost estimates were prepared by Golder on a quarterly basis for the first five years of mining, with the remainder being annual estimates. Plant and infrastructure costs were provided by Sedgman on a monthly basis for the initial capital and on an annual basis thereafter. Water management, environment, reclamation costs were provided by Tetra Tech on an annual basis. G&A-related costs were estimated by Nevada Copper.

Financial analysis was performed by Nevada Copper, with cost inputs verified by Golder, Sedgman and Tetra Tech. Based upon design summarized in this Report and as completed to a PFS level, the considered accuracy of the estimate is considered (±25%).

A summary of production and economic results is shown in Table 22-12.

Table 22-12: Prefeasibility Study Stand-Alone Open Pit Mine Study Economic Analysis Summary

Category	Unit	LOM	Avg/Year1
Production Summary			
Waste Mined	Ktons	1,174,895	60,842
Tons Processed	Ktons	385,693	20,300
Cu Grade	%	0.47%	
Cu-Equivalent Grade	%	0.50%	
Cu-Equivalent Grade (Yr 1-5)	%	0.65%	
Copper Recovered to Concentrate	Mlbs	3,207	
Payable Cu Production	Mlbs	3,098	163
Payable Cu Production	Ktonnes	1,405	74
Copper Concentrate Production	Kdmt	5,704	316
Financial & Economic Indicators			
NSR (net of royalties)	$, millions	8,986	473
Operating Cost	$, millions	4,440	193
EBITDA	$, millions	4,546	239
C1 Cash Costs	$/lb-pay	1.73	
AISC	$/lb-pay	2.03	
		Pre-tax	Post-tax
NPV 5%	$, millions	1,482	1,203
NPV 7.5%	$, millions	1,042	829
IRR	%	23	21
Payback (from start of project spend)	yrs	4.5	8.1

Note:

1. Based on PFS LOM annual plan.

Details of unit operating costs and C1 cash costs per payable pound of copper are shown in Table 22-13.

Table 22-13: Life of Mine Operating Costs

Description	$/st	$/st ore	$/lb Cu
Mining Cost (incl. capitalized waste)	1.54	4.34	
Mining Cost (excl. capitalized waste)	1.44	3.07	0.38
Mining Ore Cost	n/a	1.52	0.19
Equipment Lease	0.37	1.12	0.14
Processing Cost	n/a	5.05	0.63
Dry Stack Tailings	n/a	0.33	0.04
Subtotal	n/a	11.09	1.38
G&A inc env, water & power rebate	n/a	0.43	0.05
Total Operating Costs	**n/a**	**11.51**	**1.43**
TC/RC Shipping	n/a	n/a	0.35
RGGS Royalty	n/a	n/a	0.18
By Product Credit	n/a	n/a	-0.23
C1 Cash Cost	n/a	n/a	**1.73**

All costs and economic results are presented in U.S. Dollars. Quantities and values are presented using U.S. Customary units unless otherwise specified. No escalation has been applied to capital or operating costs. The annual production cashflow projection for the Open Pit Project is presented in Table 22-14.

Table 22-14: Stand-Alone Open Pit Mine Annual Production Cashflow Projection

	Unit	Total	2021	2022	2023	2024	2025	2026-30	2031-35	2036-40	Remaining
Metals Prices											
Copper Price	$/lb	$0.00	$3.14	$3.20	$3.20	$3.20	$3.20	$3.20	$3.20	$3.20	
Gold Price	$/oz	$0.00	$1,284	$1,325	$1,325	$1,325	$1,325	$1,325	$1,325	$1,325	
Silver Price	$/oz	$0.00	$19.53	$20.01	$20.01	$20.01	$20.01	$20.01	$20.01	$20.01	
Production and Operations											
Production											
Strip Ratio - Tonnage	t:t	3.046	-	-	9.49	4.22	6.85	4.32	2.58	0.86	
Strip Ratio - Volume	yd3:yd3	3.58	-	-	11.14	4.95	8.04	5.07	3.03	1.01	
Ore Mined	k st	385,693	-	-	10,426	18,827	11,851	90,008	132,666	121,915	
Waste Mined	k st	1,174,895	-	79,734	98,979	79,399	81,196	388,601	342,548	104,438	
Waste Mined > Avg LoM SR yd3	k st	243,824	-	-	69,374	41,321	47,547	84,858	725	-	
Total Rock Mined	k st	1,560,588	-	79,734	109,405	98,226	93,047	478,609	475,214	226,353	-
Mill Feed Total	000s tons	385,693	-	-	10,426	13,505	13,505	91,513	127,500	127,500	1,745
Mill Feed Copper	%	0.465%	0.000%	0.000%	0.729%	0.932%	0.639%	0.506%	0.462%	0.354%	0.224%
Mill Feed Gold	oz/ton	0.0016	0.0000	0.0000	0.0022	0.0026	0.0016	0.0015	0.0016	0.0015	0.0008
Mill Feed Silver	oz/ton	0.055	0.0000	0.0000	0.0774	0.0897	0.0647	0.0571	0.0561	0.0466	0.0338
Copper in Mill Feed	k tons	1,795	0.0	0.0	76.0	125.8	86.4	462.7	589.4	451.0	3.9
Gold in Mill Feed	K ozs	617	0	0	23	35	22	140	206	190	1.5
Silver in Mill Feed	K ozs	21,266	0.000	0.000	806.485	1,211.034	873.961	5,224.594	7,155.111	5,935.792	59
Cu Equiv	%	0.50%	0.000%	0.000%	0.778%	0.989%	0.677%	0.540%	0.499%	0.387%	0.244%
Gold Cu Eqv calc	x	15.63	15.430	15.603	15.603	15.482	15.482	15.482	15.517	15.833	
Silver Cu Eqv calc	x	0.20	0.196	0.197	0.197	0.196	0.196	0.196	0.196	0.200	
Mill Recoveries											
Copper	%	89.2%	89.3%	89.3%	89.3%	90.0%	90.0%	90.0%	89.8%	88.0%	88.0%
Gold	%	67.3%	67.3%	67.3%	67.3%	67.3%	67.3%	67.3%	67.3%	67.3%	67.3%
Silver	%	56.3%	56.3%	56.3%	56.3%	56.3%	56.3%	56.3%	56.3%	56.3%	56.3%
Concentrate Grade : Copper	% Cu	0.0	25.5%	25.5%	25.5%	25.5%	25.5%	25.5%	25.5%	25.5%	25.5%
Gold	oz/ton	0.069	0.000	0.000	0.057	0.053	0.049	0.058	0.068	0.082	0.072
Gold	g/tonne	2.356	0.0	0.0	2.0	1.8	1.7	2.0	2.3	2.8	2.5
Silver	oz/ton	1.951	0.0	0.0	1.7	1.5	1.6	1.8	2.0	2.2	2.4
Silver	g/tonne	66.853	0	0	58	53	55	62	67	74	83
CONCENTRATE PRODUCED											
Dry Short Tons	dston (000s)	6,287	-	-	268	444	305	1,633	2,067	1,556	13.7
Dry Tonnes	dtonne (000s)	5,704	-	-	243	403	276	1,481	1,876	1,412	12.4
Wet Short Tons	dwton (000s)	6,986	-	-	298	493	339	1,814	2,297	1,729	15.2
Wet Tonnes	wmt (000s)	6,274	-	-	268	443	304	1,630	2,063	1,553	13.7
Recovered Metals											
Copper	000s lbs	3,206,549	-	-	136,746	226,440	155,434	832,806	1,054,363	793,775	6,984
	stons	1,603,274	-	-	68,373	113,220	77,717	416,403	527,181	396,888	3,492
	tonnes	1,454,468	-	-	62,027	102,712	70,504	377,755	478,251	360,051	3,168
Gold	ozs	415,401	-	-	15,307	23,332	14,816	94,412	138,926	127,625	984
Silver	ozs	11,972,770	-	-	454,051	681,812	492,040	2,941,447	4,028,328	3,341,851	33,241
Recovered Metal Value											
Copper	$000s	10,260,956	-	-	437,586	724,609	497,388	2,664,980	3,373,961	2,540,081	22,350
Gold	$000s	550,413	-	-	20,281	30,916	19,631	125,097	184,079	169,105	1,304
Silver	$000s	239,532	-	-	9,084	13,641	9,844	58,848	80,592	66,858	665
	$000s	11,050,901	-	-	466,951	769,165	526,864	2,848,925	3,638,632	2,776,044	24,320
Payable Metal											
Payable Copper	000s lbs	3,097,526	-	-	132,096	218,741	150,149	804,491	1,018,514	766,787	6,747
Payable Gold	ozs	374,364	-	-	13,776	20,999	13,334	84,971	125,033	115,364	886
Payable Silver	ozs	10,775,493	-	-	408,646	613,631	442,836	2,647,302	3,625,495	3,007,666	29,917
Payable Metal Value											
Copper	$000s	9,912,083	-	-	422,708	699,972	480,477	2,574,371	3,259,246	2,453,718	21,590
Gold	$000s	496,037	-	-	18,253	27,824	17,668	112,587	165,671	152,860	1,174
Silver	$000s	215,578	-	-	8,176	12,277	8,860	52,963	72,533	60,172	599
	$000s	10,623,699	-	-	449,137	740,073	507,005	2,739,921	3,497,450	2,666,750	23,363

Table 22-14: Stand-Alone Open Pit Mine Annual Production Cashflow Projection (cont.)

Revenues and Cashflows		Total	2021	2022	2023	2024	2025	2026-30	2031-35	2036-40	Remaining
Gross Revenue		10,623,699	-	-	449,137	740,073	507,005	2,739,921	3,497,450	2,666,750	23,363
TC/RC Au	$000s	1,497	-	-	55	84	53	340	500	461	4
TC/RC Ag	$000s	3,771	-	-	143	215	155	927	1,269	1,053	10
TC copper	$000s	427,789	-	-	18,243	30,210	20,737	111,105	140,663	105,898	932
RC copper	$000s	232,314	-	-	9,907	16,406	11,261	60,337	76,389	57,509	506
TC/RC	$000s	665,372	-	-	28,349	46,914	32,206	172,709	218,821	164,921	1,452
Penalties	$000s	29,736	-	-	895	1,482	1,017	5,452	9,238	11,550	102
Transport (rail, port OFI)	$000s	376,970	-	-	16,585	27,463	18,851	98,518	122,490	92,216	847
Net Smelter Return (NSR)	**$000s**	**9,551,621**	**-**	**-**	**403,308**	**664,213**	**454,930**	**2,463,243**	**3,146,901**	**2,398,063**	**20,962**
RGGS Copper Royalty	$000s	530,400	-	-	22,613	37,447	25,704	137,862	174,522	131,102	1,151
RGGS Gold/Silver Royalty	$000s	35,581	-	-	1,321	2,005	1,326	8,278	11,910	10,652	89
RGGS Royalty	$000s	565,981	-	-	23,934	39,452	27,031	146,139	186,432	141,753	1,240
NSR after Royalty	$000s	8,985,640	-	-	379,374	624,762	427,899	2,317,104	2,960,469	2,256,309	19,722
Operating Costs		1.66									
Mining Costs	$000s	564,532	-	-	13,800	24,571	14,107	132,517	199,710	179,826	-
Stripping Costs	$000s	1,182,989	-	-	42,800	56,075	49,695	424,194	478,960	131,265	-
Stockpiling Costs	$000s	20,973	-	-	255	1,216	3,613	3,081	3,423	6,634	2,750
Equipment Lease	$000s	432,845	-	-	40,044	43,543	43,543	219,434	58,172	28,110	-
Process Facility	$000s	1,946,447	-	-	54,634	70,766	70,766	467,287	637,499	637,499	7,997
Power - NVE Bond/Rule 9	$000s	-	-	-	-	-	-	-	-	-	-
Tailings Dry Stack Management	$000s	127,824	-	-	2,274	2,925	3,237	25,499	44,591	48,565	734
G&A	$000s	164,483	-	-	9,518	10,146	9,564	39,506	46,329	45,607	3,813
Total Operating Costs	$000s	4,440,093	-	-	163,325	209,241	194,525	1,311,518	1,468,684	1,077,506	15,294
Operating Cashflow	**$000s**	**4,545,547**	**-**	**-**	**216,049**	**415,521**	**233,374**	**1,005,586**	**1,491,785**	**1,178,803**	**4,428**
Capital Costs											
Mine	$000s	$128,199	-	128,199	-	-	-	-	-	-	
Concentrator	$000s	$265,476	60,210	190,532	14,734	-	-	-	-	-	
Tailing Filtration & Dry Stack	$000s	$161,151	36,549	115,658	8,944	-	-	-	-	-	
Site Surface Infrastructure	$000s	$103,497	20,680	77,693	5,124	-	-	-	-	-	
Other	$000s	$13,258	6,179	7,079	-	-	-	-	-	-	
Phase I - Initial Capital Cost	$000s	$671,581	123,618	519,161	28,801	-	-	-	-	-	
Mine	$000s	$105,753	-	-	-	-	-	105,753	-	-	
Concentrator	$000s	$191,655	-	-	-	-	-	191,655	-	-	
Tailing Filtration & Dry Stack	$000s	$140,957	-	-	-	-	-	140,957	-	-	
Site Surface Infrastructure	$000s	$35,130	-	-	-	-	-	35,130	-	-	
Phase II - Expansion Capital Cost	$000s	$473,495	-	-	-	-	-	473,495	-	-	
Concentrator	$000s	$0	-	-	-	-	-	-	-	-	
Tailings, Environment, Reclamation	$000s	$123,559	-	-	21,056	2,289	2,365	19,941	35,650	27,335	10,871
Mining	$000s	$290,748	-	-	90,582	55,646	62,080	82,440	-	-	
Total Sustaining Capex	$000s	$410,253	-	-	111,638	57,934	64,446	102,380	35,650	27,335	10,871
Total Capex	**$000s**	**$1,555,328**	**123,618**	**519,161**	**140,439**	**57,934**	**64,446**	**575,875**	**35,650**	**27,335**	**10,871**
Pre-Tax NPV											
Operating Cash Flow	$000s	$4,545,547	-	-	216,049	415,521	233,374	1,005,586	1,491,785	1,178,803	4,428
Capital Costs	$000s	-$1,555,328	(123,618)	(519,161)	(140,439)	(57,934)	(64,446)	(575,875)	(35,650)	(27,335)	(10,871)
Working Capital	$000s	$0	-	-	1,015	5,607	(6,331)	(504)	(1,363)	3,631	(2,055)
Pre-tax FCF	**$000s**	**$2,990,219**	**(123,618)**	**(519,161)**	**76,625**	**363,193**	**162,598**	**429,207**	**1,454,773**	**1,155,099**	**(8,497)**
		-									
Project Pre-Tax Economics	$0.00	$0									
Project NPV 7.5%	$000s	$1,042,136									
Project IRR	%	22.9%									
		4.5									
After-Tax NPV											
Pre-Tax Cash Flows	$000s	$2,990,219	(123,618)	(519,161)	76,625	363,193	162,598	429,207	1,454,773	1,155,099	(8,497)
Income Tax Est.	$000s	(511,696)	-	-	(8,626)	(25,042)	(16,314)	(46,590)	(209,227)	(205,342)	(556)
After-Tax Free Cash Flow	**$000s**	**$2,478,522**	**(123,618)**	**(519,161)**	**67,999**	**338,151**	**146,284**	**382,617**	**1,245,546**	**949,758**	**(9,053)**
			(122,557)	(122,557)	(122,557)	(122,557)	(122,557)	(612,785)	(612,785)	(612,785)	
Project After-Tax Economics											
Project NPV 7.5%	$000s	$829,190									
Project IRR	%	20.6%									
Payback	year	8.1									

22.2.10 Sensitivity Analyses

A sensitivity analysis was performed to test the impact of changes to prices, operating cost and capital costs included in the economic model for the Open Pit Project, with the following post-tax results at a discount rate 7.5% (Figure 22-3). Figure 22-4 depicts the post-tax Open Pit Project IRR sensitivity.

Figure 22-3: Post-tax Project Net Present Value 7.5% Sensitivity (Nevada Copper, 2019)

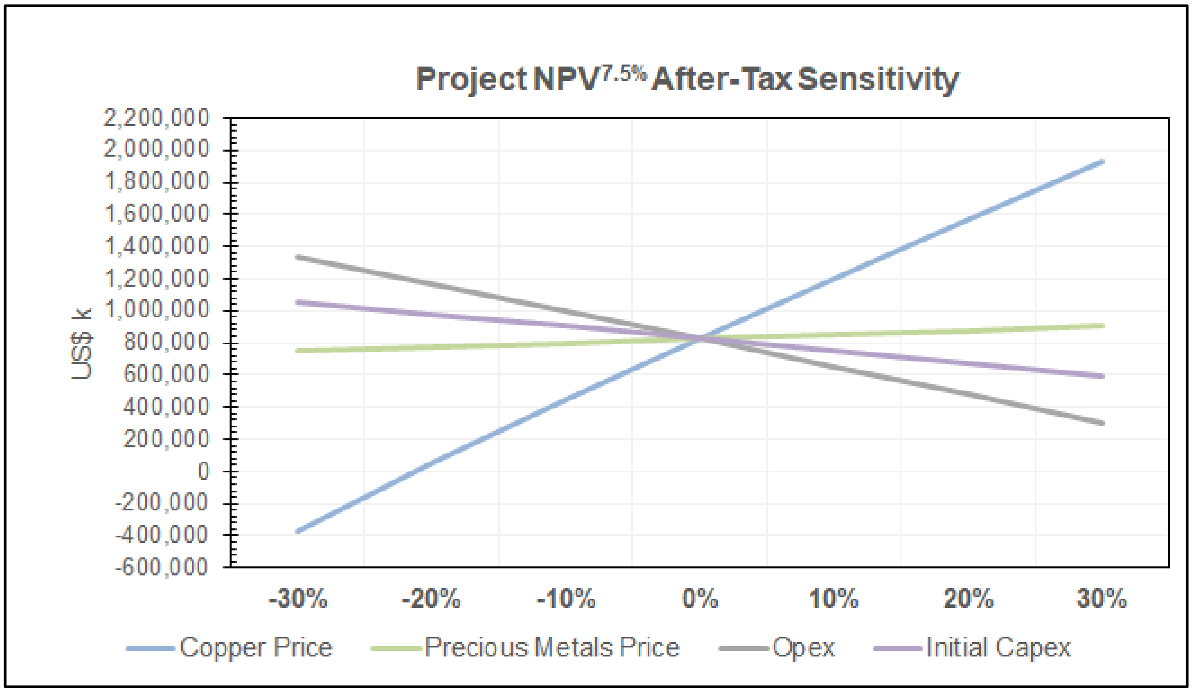
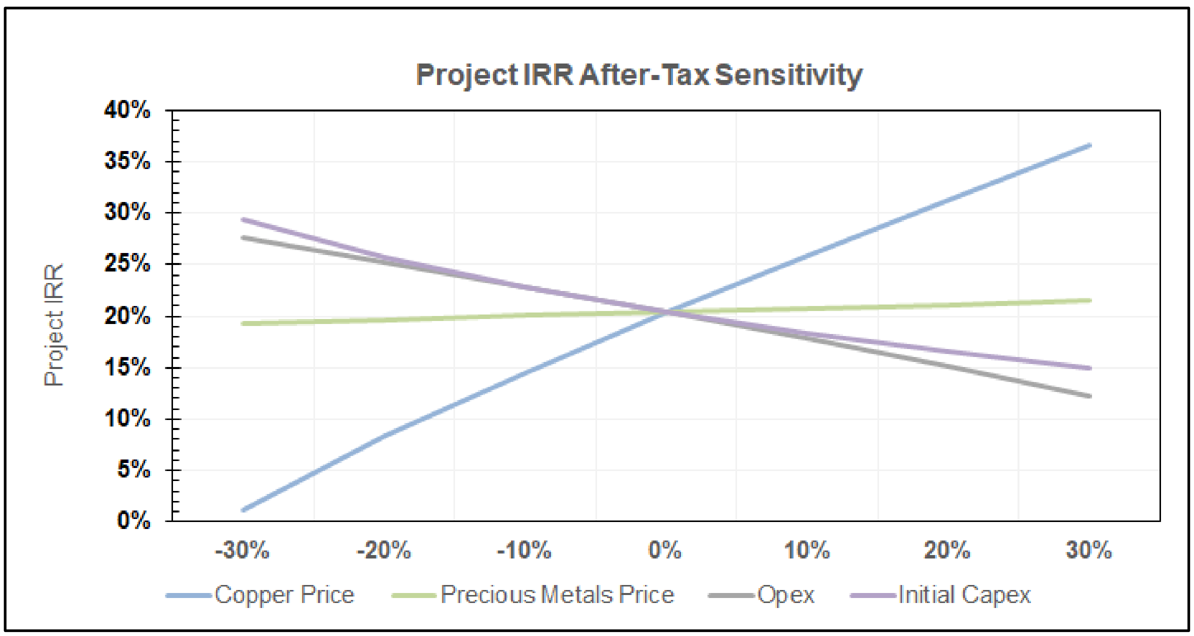


Figure 22-4: Post-tax Project Internal Rate of Return Sensitivity (Nevada Copper, 2019)



22.2.10.1 Discussion

The focus of the stand-alone Open Pit Project PFS study has been to study a scenario of smaller initial production rate with a view to expansion during the LOM. However, the work of the PFS also tested the 37 Kstpd production rate as on its own without expansion as part of the study process. An open pit project which only operates at 37 Kstpd rate for its LOM can generate a 29 year LOM and a post-tax NPV 7.5% of $643 million and a post-tax IRR of 18.7%.

22.3 Whole of Property

22.3.1 Introduction

A combined underground and open pit scenario was prepared to provide an overview of the whole property economic analysis, although decisions to advance the stand-alone Underground Open Pit Projects may be made at different times in a phased development approach.

For the purpose of this combined scenario, the timeline for the Underground Project is set with production commencing in at the end 2019 (noting the mine is in construction) and the Open Pit Project timeline is assumed such that construction starts in 2021, with production ramping up in 2023.

Economic input assumptions draw for the details provided throughout this study for each stand-alone underground and pit component of the property. The results are based from a combination or production, revenue, costs and cashflows as in each stand-alone economic model. The "Combined NPVs" in the table below are the arithmetic sum of the individual case NPVs. However, note that the NPVs have differing start dates and will not match the NPV of the combined annual net cashflows.

22.3.2 Results of Economic Analysis

A summary of production and economic results is shown in Table 22-15.

Table 22-15: Whole of Property Economic Analysis Summary

Category	Units	U/G PFS	O/P PFS	Combined: O/P & U/G	Avg/Year
Production Summary					
Waste Mined	Mst	0	1,175	1,175	-
Tons Processed	Mst	23.9	385.7	409.6	18.6
Production Years	Years	14	20		
Cu Grade	%	1.56%	0.47%	0.53%	-
Cu-Equivalent Grade	%	1.73%	0.65%	0.71%	-
Payable Cu Production	Mlbs	671	3,098	3,768	164
Payable Cu Production	Ktonnes	304	1,405	1,709	74
Financial & Economic Indicators					
NSR (net of royalties)	$, millions	2,060	8,986	$11,046	480
Operating Cost	$, millions	1,183	4,440	$5,623	244
EBITDA	$, millions	877	4,546	$5,423	246
C1 Cash Costs	$/lb-pay	1.81	1.73	$1.75	-
AISC	$/lb-pay	2.26	2.03	$2.07	-
Pre-tax					
NPV 5%1	$, millions	357	1,482	1,839	-
NPV 7.5%1	$, millions	278	1,042	1,320	-
IRR	%	27	23	24	-
Post-tax					
NPV 5%1	$, millions	301	1,203	1,504	-
NPV 7.5%1	$, millions	233	829	1,062	-
IRR	%	25	21	22	-

Note:

1. The "Combined NPVs" are the arithmetic sum of the individual case NPVs. However, note that the NPVs have differing start dates and will not match the NPV of the Combined annual net cashflows.

All costs and economic results are presented in U.S. Dollars. Quantities and values are presented using U.S. Customary units unless otherwise specified. No escalation has been applied to capital, or operating costs. The annual production cashflow projection for the whole property is presented in Table 22-15

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Item 23.0 ADJACENT PROPERTIES

Adjacent properties include a number of small mineral prospects that are within a few miles of the Property and have copper mineralization. They include Quaterra Resources' Wassuk Project and Altan Nevada's' Venus Project.

Within the district and across the Mason Valley, there are several mineral properties. These include Hudbay Minerals (Anne Mason deposit and Blue Hills prospect) and Quaterra Resources (MacArthur deposit and Yerington Pit resource).

Item 24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Underground Project Development

As of the effective date of this Report, there were approximately 250 employees, contractors, and subcontractors on site.

The construction of the Underground Project is progressing on schedule, including:

- Underground Works – Consisting of the production shaft and shaft stations (east main shaft), the ventilation shaft (east north ventilation shaft) and lateral development. The East Main shaft utilities installation has been completed ahead of schedule. Lateral development on the 2850 level and 2770 level have advanced 322 ft and 182 ft respectively (exclusive of shaft station). East North Ventilation Shaft surface infrastructure is complete and shaft sinking has advanced to 150 ft
- Surface Works – Consisting of processing plant, dry stack storage and all other surface facilities. The earthworks are complete for the primary dry stack facilities and concrete foundations for the grinding and cyclone areas are well underway.

Item 25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology & Resources

The Pumpkin Hollow Property is composed of two main deposits, geologically set in a classic copper skarn mineralizing environment. The data used to estimate resources in these areas have been collected using industry standard practices and are sufficient to support the estimation of Measured, Indicated and Inferred Resources.

Updating current reported resource models with new infill and extensional exploration drilling information may upgrade portions of the current Inferred Resources to Indicated Resources, and Indicated Resources to Measured Resources, as well as grow the overall resource base by adding potential new discoveries.

Changes in economic assumptions presented in Table 14-50 pertaining to cutoff grade calculations (e.g., metal price and processing and mining costs), could change the cutoff grade, which would result in a commensurate increase or decrease in the reported resource inventory, as discussed in Item 14.0 and Item 15.0 of this Report.

Golder is unaware of any material effects to the Mineral Resource potentially caused by known environmental, permitting, legal, title, taxation, mining, metallurgical, infrastructure, socio-economic, marketing and political factors. The Open Pit and Underground Projects are fully located on privately owned or leased lands and there are no known legal or title issues affecting the Property. The Open Pit and Underground Projects have all material permits and Nevada Copper is not aware of any known socio-economic factors that could impact the Open Pit or Underground Mineral Resources.

25.2 Underground

The results of the 2019 Underground Study found in this Report are based on the PFS for the Underground Project, as initially presented in the 2017 Technical Report. Since the completion of that study, Nevada Copper has commenced construction of the Underground Project, and has also continued to perform additional studies, which have provided further confirmation of the Underground PFS. These studies are described and included in this Report and have resulted in no material changes to the results presented in the 2017 Technical Report.

25.2.1 Mineral Reserve & Mine Planning (Underground)

Conclusions for the Mineral Reserve and mine planning on the Underground Project are as follows:

- The Mineral Reserve estimate for the Underground Project has been prepared in accordance with industry standard methods. The estimate is based on proven methods, mining practices, and modeling techniques applied to the resource block models for the East and E2 deposits. The cost assumptions and the NSR values assigned to the model are reasonable and support the NSR cutoff developed by Mining Plus for use in defining the Mineral Reserve model and supporting mine plan. Based on this assessment, the Mineral Reserve for the Underground Project will support a 15-year mine life at a planned production rate of 1.8 Mstpa during the steady-state production phase.

- The mine plan and expenditure schedule presented herein is reasonable. The mine plan has been developed based on the currently available Pumpkin Hollow data, established mining practices and supplier quotations. The resource model provided to Mining Plus appears reasonable. Geotechnical data were reviewed, and parameters defined following this review are very similar to geotechnical parameters previously applied to the underground. All data and parameters used are a sound basis for the design of a large-scale and highly mechanized underground mine at a prefeasibility level of confidence.
- The proposed mine plan uses well-established mining technology and techniques. BEVs are currently being used in several mines in North America, and most OEMs are actively developing technology for increasing capacities of BEVs and other further improvements. There is potential to take advantage of these future technology improvements with BEV, as well as other technology improvements. Paste fill has been successfully and widely used in mines worldwide. No unproven equipment or methods are contained in the mine plan.
- The Underground Project is located in a favorable mining jurisdiction with an available skilled workforce.
- In the current design, the East Shaft has a 2.8 Mstpa capacity. This, combined with the reduction in waste hoisting requirements through the mine life, may present an opportunity to achieve a higher ore production rate than has been considered in the prefeasibility mine plan.
- Based on ongoing delineation drilling, continued optimization of the stoping sequence could improve the grade profile, in particular during the early years of production.
- A portion of the East Deposit's Mineral Resource is composed of resources located in the JK-34 deposit. JK-34 is a flat-lying deposit, located deeper and to the south of the East deposit. The current mine plan's Mineral Reserve does not include material from the JK-34 deposit since there is lower geological confidence in this portion of the Mineral Resource and substantial development would be required relative to the potential mineable inventory that could be recovered. The JK-34 area should be viewed as an opportunity for resource upside since the E2 connector drift development passes close to above this deposit.
- The mine production ramp-up is realistic given a competent mining contractor will be utilized for shaft sinking, initial mine development and production operations. The mine schedule utilizes realistic sequencing and rates that are aligned with mining operations by mining contractors and the available local workforce.

- The mine development schedule is predicated on the premise of timely award of the mining contract for shaft sinking and initial mine development and production, procurement of mobile equipment by the Owner, construction of the EN ventilation shaft and equipping with a temporary hoist and commissioning of the Main Shaft.

25.2.2 Metallurgy & Recovery Methods (Underground)

25.2.2.1 Process Plant Accuracy

The design, estimating and execution planning undertaken for the process plant component of this Report has been undertaken to a level well beyond that required to satisfy a PFS. This work has been undertaken by Sedgman, an EPC Contractor, to a standard that would typically support an EPC tender for that work. The works completed for the processing facilities included a basic level of engineering completed. Design documentation prepared included a process criteria document, process flow diagrams, process and instrumentation diagrams, mass balances, a mechanical equipment list and single line diagrams. Documents prepared for major equipment pricing included scopes, datasheets and duty specifications. Construction package scopes of work were also developed to support pricing. The equipment and construction pricing was then tendered and incorporated back into the processing facilities pricing, which was supported by detailed execution planning and implementation scopes being prepared and shared with Nevada Copper.

Based on this level of understanding of the design completed and an identification of some outstanding process test work recommended, conclusions for the metallurgy and recovery methods for the Underground Project are as follows:

- Pumpkin Hollow deposit ores are medium hardness ores that are easily ground through a typical SABC comminution circuit.
- The ore requires a grind size of P_{80} 100 µm for rougher flotation and regrind size of 28 µm to achieve copper recoveries of 92.0%.
- The copper mineralogy in the ore is readily recovered in a three-stage flotation circuit producing a targeted concentrate grade of 26% Cu.
- Process design and capital costs were developed with standard industry practice equipment. Capital costs were typical of industry standards for a flotation plant of this size.
- Operating costs are typical of industry standards for flotation plants of this size. Operating costs are highly dependent on labor and consumable costs and could fluctuate significantly with market conditions.

As described in Item 17.2, the scope of work applying to the process plant includes:

- Primary crushing station based on receival of ore from the head frame skip bin through to sizing of the ore, stacking over a dual zone reclaim tunnel system through to the ore grinding circuit.

The scope of work for the production headframe, skips or skip chutes is within the mining portion of works described in Item 21

- An ore grinding circuit and copper flotation system through to the preparation of concentrate
- A copper concentrate filter system and concentrate shed including loadout weighbridge. The scope for road and services falls into the surface infrastructure scope of works described in Item 21
- A tailings circuit including thickener, deslime cyclones and paste plant feed pumps through to the tailings filters. The scope of the tailings filters and dry tailings transfer materials handling system including conveyor radial stacker for dry stacking tails for truck transfer to the dry stack emplacement area falls into the surface infrastructure scope of works described in Item 21
- The paste plant scope falls into the surface infrastructure scope of works described in Item 21
- Plant services including process water services, plant fire, process consumables including flocculants, and regents within the plant footprint including key services was in the processing facilities scope. The split between the process plant and the surface infrastructure scope was overland pipes to/from the plant. All other water infrastructure remote from the plant (e.g., in ponds), fire system, potable water system and waste water system fall within the surface infrastructure scope of works described in Item 21.0.

25.2.3 Infrastructure (Underground)

Infrastructure applicable to the Underground Project at the Property location is well developed. The City of Yerington, Nevada, is a 15 minute drive away via paved, two-lane access. Rail access is 10 miles from the site. The local airport, Yerington Municipal Airport, is eight miles from the site, while the Reno-Tahoe International Airport is 80 miles from the site.

Off-site access roads include use of an existing road from the mine site to Highway 95 for the hauling of concentrate to Wabuska. The existing E Pursel Lane runs east–west through the site and will be upgraded.

Water supply is ample for the underground mine, and will be supplied from mine dewatering. A package potable water treatment plant will be constructed to treat dewatered water. A package sewage treatment plant will be constructed and effluent will be used for plant process water or discharged to an infiltration basin.

Electrical power will be supplied through the Nevada Energy electrical grid via a 120 kV overhead line.

25.2.4 Tailings Management (Underground)

The DST facility design incorporates stable tailings storage, a containment system, a perimeter dike, surface water diversion and runoff management features, and placement of tails via trucking. The facility is situated west of the proposed production shaft and surface infrastructure and will be constructed in three stages: a trial emplacement, the first year on a liner, and subsequent years which may or may not be on a liner depending on the outcome of the trial emplacement.

The filtered tailings management option was selected for the underground due to reduced risks associated with geotechnical stability and environmental impact, and for mine closure benefits.

The design was developed based on hydrological and geotechnical studies that included review of regional climate data, drilling and testing programs, and laboratory characterization of subsurface and tailings samples. These studies will need to be advanced to detail design level as part of the next phase of work. The design features shall be reviewed as part of this assessment to optimize design elements and ensure performance will meet design criteria and regulatory requirements.

Geotechnical assessments indicate that the design of the dry stack meets regulatory and guideline requirements. The tailings are expected to be trafficable with trucks soon after placement at the design water content, and the risk of foundation and or tailings instability is low for the proposed operation. Based on the results of infiltration and seepage modeling, the storage of tailings in the facility is not expected to impact the regional groundwater system, located approximately 300 ft below ground. Under warm climate conditions, the water balance of the facility is negative, with evaporation being the largest component of the system. Annual evaporation water losses were calculated to be greater than annual precipitation. Under these conditions, seepage is limited to the drainage of moisture that was placed with the tailings material. The flow associated with the drainage of moisture content is anticipated to be minimal throughout operations and into closure. The HDPE liner in the Year 1 footprint and the compacted, low permeability tailings layer in subsequent placement will limit the flow of water into the foundation soils.

The regulations in Nevada require incorporation of a low permeability base layer consisting of compacted native, imported or amended soils, which have an in-place compacted coefficient of permeability of no more than 1×10^{-6} cm/s. Geotechnical laboratory testing must be performed on the process tailings to confirm the feasibility of achieving a hydraulic conductivity less than 1×10^{-6} cm/s with compacted and/or amended tailings prior to the end of Year 1.

Water collected from seepage and surface water runoff from the stack will be retained in a lined pond adjacent to the stack. The water will be reused in mineral processing.

The performance of the DST facility will be monitored, particularly during the initial production years, and the data will be assessed and results incorporated into the detailed design of the future stages of construction and closure.

25.2.5 Environmental, Social & Mine Reclamation (Underground)

The Underground Project requires state and local permitting but does not require federal environmental permits of NEPA compliance:

- A number of environmental studies have been conducted at the Property in support of the underground permitting and approvals. Studies have been conducted to investigate soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology and water quality.
- The Underground Project has received approval for the primary permits (and amendments thereto), including the WPCPs, Reclamation Permit, and Class II Air Quality Operating Permit. Significant modifications to the underground may require resubmittal of permit applications. Models and designs prepared in support of permitting efforts should rely on the best and most current information available.
- There are no federally threatened or endangered species likely to occur on the Property area. The only federally threatened or endangered species that occur in Lyon and Mineral counties are fish species, and there are no perennial or fish-bearing streams on the Property.
- Archaeological surveys have been performed over the full Pumpkin Hollow Project area. The underground development does not affect any Native American Reservation Lands or sacred sites.
- The Underground Project occurs entirely within the City of Yerington in Lyon County, Nevada, which has historically held, and continues to hold, a high unemployment rate. The underground development is estimated to bring approximately 430 direct jobs to the area. There have been no formal objections to the Underground Project from environmental groups or other non-governmental organizations.

25.2.6 Mine Reclamation (Underground)

Underground reclamation is planned to achieve approved post-mining land use and meet the reclamation permit requirements to achieve full bond release based on current understanding and assumptions. Reclamation approaches include regrading features (DST and MRSFs) to stable slopes, constructing surface water management features, placing a closure cover, and revegetating features. For infrastructure and process facilities, prior to transfer of facilities to future owners for post- mining industrial use, mining and process equipment will be decommissioned and removed. The approach will be refined through development and monitoring of test plots and observation of concurrent reclamation success.

Long-term water treatment is not anticipated for the underground, based on the results of seepage modeling from the DST that indicate a nominal seepage flow which will be directed into the pit lake during closure. It is anticipated that seepage flow will decreases to a level which may be managed through passive evaporation in the seepage collection pond. In addition, pit lake water treatment is not anticipated based on modeling results which indicate that the pit lake will be a hydraulic sink. Throughout the mining and post-mining phases, surface water will be managed in diversion channels, runoff collection channels, and basins.

25.2.7 Geochemistry (Underground)

Evaluation of geochemical data related to the Underground Project indicates that with the exception of a few lithologies, acid generation and metals release should not be a major concern.

- Based on net neutralization potential and neutralization potential ratio (NPR) values, all waste rock types except for magnetite skarn are considered to be non-potentially acid generating (non-PAG) with an 85% degree of confidence. The number of samples utilized in the waste rock characterization, however, was minimal at best. Therefore, it is recommended that as mining progresses, additional samples be analyzed and remedial actions be applied if found appropriate.
- Insufficient sample numbers exist to define the endoskarn, magnetite skarn, and intrusive based on statistical distribution. It is recommended that additional static samples be collected to further characterize these rock types to gain greater confidence in waste rock management decisions. Such collection and analysis can be carried out as the skarns and intrusive materials are encountered provided waste from these lithologies can be segregated and kept relatively dry, Otherwise, testing should be conducted sooner rather than later to ascertain their potentially acid generating (PAG) character, When possible encapsulation of PAG materials with neutralizing materials such as marble, limestone and possibly talc, as well as maintaining dry conditions should reduce any acid generating tendencies.
- Based on calculations of waste material proportions, it is estimated that 19% of waste rock material stored at the MRSF will be PAG. Estimations were based on ABA results and the estimated lithological composition of the waste material. Hornfels rock type is the dominant lithology and accounts for approximately 15% of the potentially deleterious waste; however, this rock type is non-PAG. Of lesser concern are the skarn and intrusive rock types which account for six percent of the PAG material. Limestone and talc rock types also have a proportion of PAG. PAG material, regardless of rock type, will likely be blended and comingled with non-acid generating rock types. This will effectively decrease the risk of introducing PAG material on the surface of the MRSF.
- The magnetite product, rougher tailings, magnetite tailings, final tailings composites, and cemented paste are classified as non-PAG using the NDEP (1990) criteria. These materials represent only 7% of the total mass of the mixed composite tailings. They will be fully mixed with rougher tailings prior to placement in the DST storage facility.
- Diffusion testing on cemented paste suggests that most constituents do not pose an environment risk. Arsenic however did exceed NDEP Profile I Reference Values, although observed values do approximate background groundwater conditions. Metal and metalloid release rates appear to be very slow. Such trends suggest that trace constituents are controlled by a combination of dissolution and diffusion rather than pH.

- Based on geochemical analysis of rock present on the final pit walls, in conjunction with local groundwater chemistry, a geochemical model of the final pit lake water quality was developed. The majority of inflow water entering the pits will be from groundwater sources seeping through the pit walls, with predicted water quality being acceptable with respect to NDEP Profile I Reference Values and background groundwater quality data.
- As with any mine, there is the possibility that earlier models could prove to be inaccurate as development progresses and conditions change. Therefore, periodic and systematic monitoring of waste materials produced during the mining phase should be conducted and the appropriate action to minimize environmental risk should be applied.

25.2.8 Groundwater Hydrology & Dewatering (Underground)

- The regional numerical groundwater flow model developed estimated the maximum inflow rates of about 3,000 gpm for the Underground Project. Water pumped from the underground workings will be used to supply the mill, and excess water will be piped to an infiltration basin for re-introduction to the groundwater system.
- The inflow predictions were developed using currently available information on the geologic and hydrogeologic conditions at the site for the 2012 and 2015 studies and applied to this study. Prediction of inflows is inherently subject to uncertainties, and it is possible that as-yet-unidentified conditions that will affect inflow rates could be encountered during mining, resulting in inflow rates higher or lower than those predicted.
- While it is considered unlikely that the predicted inflow rates will be exceeded, if higher inflow rates were encountered, the additional water could be handled with little or no disturbance to the mining operations, via additional infiltration basins.
- If inflow rates prove lower than those predicted to be encountered, supplemental water for process water supply will be available for from groundwater wells or from the City of Yerington municipal utility.

25.3 Open Pit

A study with a prefeasibility level of accuracy for the stand-alone Open Pit Project has been completed, and it is evident that there is a positive business case for further advancement of the Open Pit Project through engineering, geological work, and further study.

25.3.1 Mineral Reserve & Mine Planning (Open Pit)

The extent to which the mineral reserve estimates could be materially affected by mining, metallurgical, infrastructure, permitting, and other relevant factors that are different than the factors used in the PFS and described in this report is shown by the sensitivity analysis in Item 22.0 Except for commodities prices, all other relevant factors including mining, metallurgical, infrastructure, and permitting factors related to the Project and described in this report are factors affecting estimated Project costs and are reflected in the PFS cost estimates that are summarized in this report. If for any reason any of these Project cost factors are changed such that the Project capital or operating cost estimates change materially, then the mineral reserve estimates stated in this report could be materially affected.

Conclusions for the Mineral Reserve and mine planning for the Open Pit Project are as follows:

- The open pit mining schedule has a LOM of 20 years (approx.). The North Pit, which formed the bulk of the LOM production, was notably higher grade (%Cu) than the South Pit (%Cu). In the later years of the expansionary of the Open Pit Project after the North Pit is depleted, ore from the South Pit will commence production. Ore from both pits will be hauled to either a primary crusher, or stockpiles, located between the pits and the WRSFs.
- There are a total of 386 Mst Proven and Probable Mineral Reserves in the Open Pit PFS pit design. Within the plan, there is 107 Mst from the North Pit and 279 Mst from the South Pit. The average diluted copper grade in the North Pit is 0.57% Cu; and in the South Pit, the average diluted copper grade is 0.43% Cu.
- The preparation of a Feasibility Study could improve the reliability or confidence of the Mineral Reserve estimates in this Report. The following criteria are expected to be reviewed during the Feasibility stage of the Project:
  - Geotechnical slope parameters of the North Pit and South Pit
  - Include any results of condemnation drilling
  - Future copper selling price and marketability
  - Processing recovery of copper

25.3.2 Metallurgy & Recovery Methods (Open Pit)

Conclusions for the metallurgy and recovery methods are as follows:

- Pumpkin Hollow open pit deposit ores are moderately hard ores that can be processed in a typical SABC comminution circuit.
- Particle grind size of P80 150 µm for rougher flotation and rougher concentrate regrind particle size of 28 µm are needed in order to achieve estimated copper recoveries for the North Pit of 90% and South Pit of 88% with a LOM average 89.3%.
- The copper mineralogy in the ore is readily recoverable in a three-stage flotation circuit producing a targeted concentrate grade of 25.5% Cu.

- Process design and capital costs were developed with standard industry practice equipment. Capital costs are typical of industry standards for a flotation plant of this size for a two-phased installation approach.
- Operating costs are typical of industry standards for flotation plants of this size.

25.3.3 Infrastructure (Open Pit)

Infrastructure at the Property location is well developed. The City of Yerington, Nevada, is a 15-minute drive away, via paved, two-lane access. Rail access is 10 miles from the site. The local airport, Yerington Municipal Airport, is eight miles from the site, while the Reno-Tahoe International Airport is 80 miles from the site. Off-site access roads include use of an existing road from the north end of the mine site to Highway 95 for the hauling of concentrate and a new road alignment for Pursel Lane. This will build upon the modifications done to the roads during the development of the Underground Project.

The road for hauling the concentrate will connect Highway 95 and the Pumpkin Hollow Property and will run north–south. Also, the existing Pursel Lane runs east–west through the Pumpkin Hollow Property, and will have been re-routed around the north end of the site during the Underground Project's development

Potable water will be supplied from the City of Yerington from an existing pipeline. This water pipeline, which is connected to the City of Yerington water supply, is shared with an existing user and the Underground Project. From the pipeline takeoff point, a new extension will be constructed and water will be distributed within the site through the potable water pipeline. Raw water will be supplied from wells on site or underground and open pit dewatering.

A package sewage treatment plant will be constructed and effluent will be used for plant process water or discharged to an infiltration basin.

Electrical power will be supplied through the Nevada Energy electrical grid via a new 120 kV overhead line.

25.3.4 Tailings Management (Open Pit)

The open pit filtered TSF design incorporates stable tailings storage, a containment system, a network of overdrainage pipework for seepage collection, a perimeter dike, surface water diversion and runoff management features, and trucking for tailings transport. The facility will be constructed in stages to suit the tailings production schedule.

The design was updated from previous feasibility level TSF design based on hydrological and geotechnical studies that included drilling and testing programs and laboratory characterization of subsurface and tailings samples. These studies will need to be advanced as part of the next phase of Open Pit Project. The design features are to be reviewed as part of this assessment to optimize design elements and ensure performance will meet design criteria and regulatory requirements.

Geotechnical assessments indicate that the stability of the TSF will meet regulatory and guideline requirements. Based on the results of previous infiltration and seepage modeling, the storage of tailings in the facility has a very low risk of impacting the regional groundwater system. Under median and warmer climate conditions, the water balance of the facility is negative, with evaporation being the largest component of the system. Under these conditions, seepage is limited to the drainage of moisture that was placed with the tailings material. The flow associated with the drainage of moisture content is anticipated to be minimal throughout operations and into closure.

Regulations in Nevada require incorporation of a low permeability base layer consisting of compacted native, imported or amended soils, which have an in-place compacted coefficient of permeability of no more than 1 × 10-6 cm/s. Geotechnical laboratory testing must be performed on the compacted filtered tailings to confirm the feasibility of achieving the permeability requirement or, alternatively, Nevada Copper must demonstrate the effectiveness of the proposed containment system to the State of Nevada with the proposed instrumented underground tailings test pad.

Water collected in the overdrain seepage collection network and as surface water runoff from the tailings stack will be reused in mineral processing or treated and released as required.

The performance of the TSF will be monitored, particularly during the initial production years, and the data will be assessed and results incorporated into the detailed design of the future stages of construction and closure.

25.3.5 Environmental, Social & Mine Reclamation (Open Pit)

25.3.5.1 <u>Existing Body of Work</u>

A number of environmental studies have been conducted at the site in support of permitting and approvals for the Pumpkin Hollow Project. Studies have been conducted to investigate soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology and water quality.

25.3.5.2 <u>Project Permitting Status</u>

The Pumpkin Hollow Project has received approval for all amendments to primary permits, including the WPCPs, Reclamation Permit, Class II Air Quality Operating Permit, and Lyon County Special Use Permit. Significant modifications to the Pumpkin Hollow Project may require resubmittal of permit applications. Models and designs prepared in support of permitting efforts should rely on the best and most current information available.

25.3.5.3 <u>Environmental Impacts</u>

There are no federally threatened or endangered species likely to occur on the Property. The only federally threatened or endangered species that occur in Lyon and Mineral counties are fish species, and there are no perennial or fish-bearing streams in the Pumpkin Hollow Project area.

25.3.5.4 Heritage Resource Impacts

Archaeological surveys have been performed over the full on the Pumpkin Hollow Project area. The Pumpkin Hollow Project area does not intersect any Native American Reservation Lands or sacred sites.

25.3.5.5 Social or Community Impacts

The Property occurs entirely within Lyon County, Nevada, which has the highest unemployment rate in the state. The Open Pit Project is estimated to bring about 400 direct jobs plus additional indirect jobs to the area. The Lyon County Planning Commission and, subsequently, the Lyon County Commission have unanimously approved the County SUP for the Underground Project. Should an additional SUP be required for the Open Pit Project, County approval is considered highly likely.

25.3.6 Mine Reclamation (Open Pit)

The Open Pit Project reclamation is anticipated to achieve approved post-mining land uses and meet the requirements of the reclamation permit to achieve full bond release based on current project understanding and assumptions. Reclamation approaches, including use of selected infrastructure for post-mining industrial use and recontouring, placing covers and conducting revegetation over remaining site features, will be refined through developing and monitoring test plots and observing levels of success of concurrent reclamation on site features throughout the LOM. Long-term water treatment is not anticipated for the Open Pit Project, based on the results of seepage modeling from the DST that indicates a nominal seepage flow which, during closure, will be directed into the pit lake until the seepage flow decreases to a level which may be managed through passive evaporation in the seepage collection pond. In addition, pit lake water treatment is not anticipated based on modeling results which indicate that the pit lake will be a hydraulic sink. Throughout the mining and post mining phases, surface water will be managed in diversion channels, runoff collection channels, and basins.

25.3.7 Geochemistry (Open Pit)

- Based on ABA, all rock types except for magnetite skarn are considered to be non-PAG with an 85% degree of confidence.
- Based on calculations of waste material proportions, it is estimated that 20% of waste rock material stored at the MSRF will be PAG. Estimations were based on acid-base accounting results and the estimated lithological composition of the waste material. Hornfels rock type is the dominant lithology and accounts for approximately 13% of the potentially deleterious waste; however, as a whole, this rock has little or no acid-generating capacity. Of lesser concern are skarn and intrusive rock types that account for 6% of PAG material. Limestone and talc rock types also have a proportion of PAG material. PAG material, regardless of rock type, will likely be blended and comingled with non-PAG rock types. This will effectively decrease any potential concentration buildup of PAG material on the surface of the MRSF.
- The rougher tailings and final tailings composites are non-PAG using the NDEP (1990) criteria. The high sulfide sulfur content cleaner scavenger tailings are PAG; however, the mine plan does

not include separate storage of scavenger tailings. These represent only 7.0% of the total mass of the mixed composite tailings. They will be fully mixed with rougher tailings prior to placement in the DST storage facility.

- Based on geochemical analysis of rock present on the final pit walls, in conjunction with local groundwater chemistry, a geochemical model of the final pit lake chemistries was developed. Water chemistry is predicted to be slightly modified from local groundwater after simulation periods.

25.3.8 Groundwater Hydrology & Dewatering (Open Pit)

Dewatering of the open pits will be required when pit excavation reaches the water table. The regional numerical groundwater flow model developed for the Open Pit Project estimated maximum inflow rates of about 1,600 gpm for the North Pit and about 210 gpm for the South Pit. Water pumped from the pits would be used to supply the process plant, and excess water would be piped to an infiltration basin for re-introduction to the groundwater system.

The inflow predictions were developed using currently available information on the geologic and hydrogeologic conditions on the Property and in the region. Prediction of inflows is inherently subject to uncertainties, and it is possible that as-yet-unidentified conditions that would affect inflow rates could be encountered during mining, resulting in inflow rates higher or lower than those predicted.

While it is considered unlikely that the predicted inflow rates will be exceeded, if higher inflow rates were encountered, the additional water could be handled by additional RIBs with little or no disturbance to the mining operations.

If inflow rates lower than those predicted were to be encountered, water to make up for that loss of process water supply would be available from dewatering wells associated with the underground operations.

25.3.9 Water Balance (Open Pit)

The site-wide water balance model results indicate that eight RIBs, each with a bottom area of 10,800 ft^2, will be required to infiltrate excess water associated with North and South Pit dewatering operations.

During two periods of the LOM, process make-up water demand exceeds dewatering flowrates from the open pits and underground workings. The first two years of production will require an average of an additional 210 gpm of make-up water, and from Year 12 through the end of LOM, an average of an additional 960 gpm of make-up water. It is assumed that this make-up water will be supplied by the underground operation’s dewatering wells.

The underground mining operations were included in this site-wide water balance model to ensure that all water was being properly accounted for across the site. Model results show that an additional 21 RIBs, each with a bottom area of 10,800 ft^2, would be required to infiltrate excess water from underground dewatering operations. The costs associated with these underground RIBs are not included in the costs for the open pit facility.

Item 26.0 RECOMMENDATIONS

26.1 Underground

The results of the 2019 Underground Study found in this Report are based on the PFS for the Underground Project, as initially presented in the 2017 Technical Report. Since the completion of that study, Nevada Copper has commenced construction of the Underground Project, and has also continued to perform additional studies, which have provided further confirmation of the Underground PFS. These studies are described and included in this Report and have resulted in no material changes to the results presented in the 2017 Technical Report.

Further studies and recommendations for Nevada Copper to still yet perform with respect to the Underground Project are presented below. They include further enhancements to the resource model, refinements of geotechnical parameters and mine plans, further confirmation of metallurgical parameters, evaluation of electric equipment and refinement of costs. All the studies are expected to either confirm, refine, or improve the results of the 2019 Study. There is no known issues under consideration in the studies that would result in a material change that would negatively affect the Underground Project.

26.1.1 Resources (Underground)

The following actions are recommended to improve model certainty and to explore potential remaining upside in the Underground Mineral Resource:

- Commence resource definition drilling within the South and North pods of the Eastern deposit (Underground) when drift access is gained and continually update deposit models;
- Detailed local gold and silver assays are not consistently available due to differences in the manner in which historical drilling programs were conducted:
  - Review and determine where additional assaying of pulps/rejects might be beneficial to increase the data available on gold and silver
- Continue to review lithologic coding
- Review geological and structural controls on the gold and silver mineralization as additional drilling is completed
- Reconcile current estimations as development drilling progresses, alter methods where appropriate.
- Continue to explore resource upside in the following Eastern Area Deposit areas:
  - Down-dip apron of the E2 deposit and its connection with the JK-34 zone.
  - Between the North and South deposit to further connect the two bodies.
  - South of the E2 deposit, following up on results seen in N-48.

- o South of the area Between the Southeast and E2 deposits, following-up on results seen in NC11-05 and FG-58 to determine is another resource body exists.
- o East of the East resource area, to determine the relationship of FG-20 to the main area.

26.1.2 Mineral Reserves & Mining Planning (Underground)

26.1.2.1 Geotechnical

It is recommended that the following works are undertaken:

- In order to complete reliable and detailed McCracken and Stacey (1989) raisebore risk assessments and other geotechnical analysis, pilot holes should be drilled along (or very close to) the axis of all planned vent shafts as part of detailed engineering and prior to making a final decision in regard to raise excavation methods. These pilot holes should be logged by a suitably trained experienced geotechnical professional.
- In order to reliably develop rock mass strength, further additional rock property testing should to be undertaken, including tri-axial testing. This laboratory work could be undertaken on drill core recovered from shaft pilot holes recommended above.
- While the depth below surface of mining is not excessive (<3,000 ft), the assessed moderate rock mass strength and proposed mine plan are likely to result in regular and changing local stress conditions. Therefore, it is recommended that advanced 3D inelastic numerical modeling be considered for future technical studies. This approach can assess excavation interaction and associated stability of excavations which is typically beyond the capability of empirical methods. The 3D numerical modeling study can confirm and/or optimize the extraction sequence, along with stability analysis of LOM excavations, such as large chambers and vertical development. Mining Plus has considerable experience with advanced 3D inelastic numerical modeling and could manage and lead such a study.
- Communicate the raw data referenced in the hydrology report including geothermal gradient and groundwater temperature results.

26.1.2.2 Mine Design

It is recommended that the following works are undertaken:

- Perform peer review of geotechnical information to confirm assumptions and provide feedback on current design.
- Investigate methods to reduce development requirements. Particular focus should be placed on optimizing operational development for East North.

- Prepare detailed designs for BEV battery change-out stations and identify optimal placement of stations to allow for battery regeneration. Maximum benefits will be achieved through a site visit to mines currently operating BEV.
- Decide what types of remote control mucking will be selected (control from surface, line of sight, etc.) as this is needed for digout development and mucking beyond the digouts. Incorporate required design changes.
- Update battery equipment that is available in the market and will be available over the mine life. Perform a trade-off study to decide if larger transport drifts can be justified.
- Review ore pass and connector drift material handling design (including conveyors) to ensure the optimal design.
- Review mining recovery and dilution estimates and add additional detail to stope designs as required. Include consideration for modeling dilution by stope widths for the E2 zone.

Schedule

It is recommended that the following works are undertaken:

- Refine and improve equipment productivity calculations by period to better identify equipment needs by period.
- Discuss with MHSA and perform schedule revision after ventilation has been calculated to account for any restrictions that could be imposed from the study.
- Confirm timeline and performance details on the hoists from manufacturers and suppliers and adjust schedule inputs.
- Optimization for material destinations, i.e., bringing waste from development directly as backfill material into secondary stopes, as part of equipment productivity calculations and overall equipment requirements per period.

Cost

It is recommended that the following works are undertaken:

- Facilitate and expedite cost quotes and communication of required information for final supplier selection.
- Perform detailed cycle time analysis which includes input from the BEV manufacturers/operators in order to calculate BEV energy consumption, number of battery cycles, and maintenance costs.
- Return to BEV OEMs and discuss battery rental and lease options in order to reduce up front and sustaining capital costs (part of BEV OEM selection).

- Perform a trade-off study to determine if remote control of equipment from surface is beneficial and provide the business case for system installation on BEV production equipment and installation of communication system in work areas.
- Identify potential savings with respect to hoists—used hoists, better prices/other manufacturers, etc.
- Review accounting for freight costs.

Battery Vehicles

It is recommended that the following works are undertaken:

- BEV OEM Selection: Prepare formal Request for Quote document to present to OEMs outlining requirements and expectations, in order to shortlist or select a preferred OEM
- Travel to a mine which has been operating battery equipment in a production setting to confirm/quantify performance
- Request examples of BEVs being used remotely from surface. This application will increase equipment utilization, and is particularly beneficial for BEVs as they can work remotely in the presence of blast fumes.

26.1.2.3 Mine Infrastructure

It is recommended that the following works are undertaken:

- Existing buildings proposed to support mining activities in Item 18.1 will be assessed in more detail for suitability of size and condition with a mining contractor.
- Study and optimize the underground power distribution system, as the majority of the machines will be battery powered and require electrical infrastructure.
- Provide loadout system engineering design drawings.
- Provide detailed design of maintenance (mining) workshop and explosives magazine, including equipment to be installed.

26.1.2.4 Ventilation

It is recommended that the following works are undertaken:

- Perform peer review of PFS ventilation design and recommendations from internal/external specialist.
- Perform a detailed study of ventilation requirements for BEVs, incorporating input from regulatory bodies including MSHA. Further refine ventilation design and requirements for initial development and correlate with LOM schedule.

- Complete a trade-off study for a ventilation drift between the East South and East North zones.
- Calculate shock losses in the connector drift under different scenarios and determine if a 16 ft drift is required even if the E2 ventilation raise is completed prior to district development. Investigate the ability to complete E2 Shaft sooner. It should be noted that the 2015 feasibility study (Tetra Tech 2015) utilized 18 ft by 18 ft connector drift to support ventilation requirements for E2.
- Optimize air reuse in the ventilation model.
- Perform schedule alterations to balance the air volume required in individual ventilation raises while still maximizing production. This may not be necessary if equipment air quantity requirements can be reduced or if air reuse is sufficient to lower air volumes. The task will be run in conjunction with mine scheduling.

26.1.2.5 Paste Fill Study

As more underground geotechnical information becomes available, a rock mechanic analysis needs to be conducted to re-evaluate the UCS requirement of the backfill material for mined out stopes to reduce backfill costs. This evaluation also needs to consider various paste fill mixtures of tailings, cement, binder supplements and water in order to arrive at the most cost-effective mixture that will provide adequate support as determined by the rock mechanics analysis.

Binder supplements should focus on locally available, cost-effective binders.

As noted in Item 17, the paste fill requirements will meet the requirements of the mining schedule developed as part of the Underground Project PFS study, apart from three months late in the mine life where the mine paste fill demand exceeds available paste fill. This is planned to be addressed during the feasibility study through either or both of the following:

- Adjustment to the mine schedule
- Re-prioritization of stope voids filling sequencing/timing

26.1.2.6 Ore Sorting of High Grade Material

Substantial portions of the East and E2 Deposits contain high grade ores in vein type structures. Further, conventional drill and blast mining will generate barren overbreak in the back, walls and floor. It may be appropriate to use an ore sorter to reject waste rock, both interburden and overbreak, if these ores are indeed intermingled to any extent with waste rock. A metallurgical test work program should be performed to consider the use of sortation and the potential ramifications on ore grade and ore tonnage. Note that sorting may result in reduced mining costs in that the selected mining method may allow more overbreak.

26.1.3 Concentrate Transport (Underground)

The preferred site for the transload facility at Wabuska is subject to land access confirmation. Access to this site is currently being progressed, and should be confirmed in the next phase of the study.

Nevada Copper should further investigate the use of international containers for transport of concentrate from site to overseas markets.

26.1.4 Metallurgy & Recovery Methods (Underground)

The test work and actions listed here are suggested to evaluate flowsheet optimization and opportunities.

26.1.4.1 Flowsheet Optimization & Opportunities

Opportunities are identified that can optimize the flowsheet and potentially reduce the operating cost such as:

- Potential increase in flotation retention times to improve recoveries.
- Amending the paste plant design to suit mix designs more suited to available cost-effective binders.

26.1.4.2 Additional Test Work

Additional test work listed here is not required for the Underground project, but would add to the better understanding of ore quality and plant performance. This would include:

- Comminution variability test work: samples (allow 10) to include marble lithologies that have not yet been tested.
- Pre-concentration test work: Sighter tests to investigate suitability of pre-concentration techniques for processing the Pumpkin Hollow ores.
- Head assays including ICP scans and copper speciation analysis plus mineralogy: Composite samples—Year 3 to 5 Production Composite, plus 10 variability samples.
- Flotation Test work: Master composite, variability samples (10). flash flotation tests, cleaner tests on variability samples at optimized, aging tests and locked cycle flotation test—include ICP scan on final concentrate.

This additional test work has not been costed given it is optional in the next phase.

26.1.4.3 Surface Blending

The low grade stockpile is used to blend mine production to mill feed. This blending has been modeled in this study. The purpose of this blending is to reduce copper variability to provide a more uniform feed grade. This modeling shows some months where the mill throughput may be slightly reduced due to ore hardness. This issue should be assessed in the next phase, where further minor surface blending optimization is expected to address this matter.

26.1.4.4 Backfilling

A testing program must be developed pursuant to Section B.5 of WPCP103 and must be reviewed and authorized by the BMRR and executed prior to placement of paste backfill to "demonstrate that the backfilling of underground workings with paste will not degrade waters of the State, and propose the mix and physical characteristics of the paste which are to be incorporated into the Permit." This work should be coordinated with other tailings test work to reduce costs. The estimated incremental cost is $50,000.

26.1.4.5 Geochemistry

As mining progresses, lithologies that are encountered that otherwise are under-represented in the initial geochemical assessment should be reviewed. It will be prudent to update the tails geochemistry assessments in such instances. In this way, plans to minimize potential environmental issues can be devised to help mitigate problems before they become an issue. If waste is proposed to not be stored underground, the following work ought to be performed:

- Ongoing NAG pH characterization will further determine the effectiveness of NAG pH as a viable option to segregate PAG and non-PAG material.
- On-site kinetic testing on select rock types and/or blended waste material should be conducted to better simulate storage of waste material at the Project location. Fifty-five-gallon drums, or equivalent sized containments, should be employed to store waste material subjected to natural occurring wetting and drying cycles. Seepage should be collected and analyzed to determine the natural leachate chemical character of waste material on the Property.
- Additional ABA analyses are needed for endoskarn, magnetite skarn, and intrusive rock types based on statistical evaluation.

The estimated cost for the additional geochemical assessment discussed above is $50,000.

26.1.4.6 Surface Water Hydrology

Continuation of the existing compliance program to monitor of precipitation, surface water flow and water quality should be completed to ensure compliance through the development, operational, reclamation and closure plan stages of the Underground Project. Measured values should also be compared to design conditions to ensure that precipitation and runoff factors are substantiated.

The estimated budget for surface water monitoring and analysis and associated consulting services is approximately $10,000 per year.

26.1.4.7 Groundwater Hydrology/Dewatering

It is recommended that mine inflow estimates be updated to reflect any substantive changes to the mine plan, such as development of additional open pits or major changes in the layout or timing of the currently proposed open-pit or underground mining operations. The mine dewatering system will require modification and refinement as empirical data become available during advanced exploration and initial mine construction and operation. It is recommended that mine inflow estimates be regularly updated, refining the numerical groundwater flow model by incorporating observed drawdown of groundwater during the initial periods of mine development and operation. The operation should maintain a database of groundwater data collected from periodic depth-to-groundwater measurements in monitoring and pumping wells and piezometers. Mine discharge flow measurements and infiltration rates should be tracked and included in the database.

The estimated budget for this work is $75,000 per model update and $15,000 per year for database maintenance. Assuming the groundwater model will be updated four times and database maintenance will proceed throughout the mine life, the estimated total cost is $645,000.

26.1.5 Recommendations & Future Work (Underground)

As described above, the following recommendations are made considering the results of the PFS and the Underground Project risks identified. A work program is recommended that includes engineering, studies and investigations in preparation for the detailed engineering phase and feasibility study phase. The costs of these activities are estimated as shown in Table 26-1.

Table 26-1: Underground – Recommended Activities & Costs

Recommended Activity	Cost ($, thousands)
Resource Definition Drilling	Refer to Opex
Exploration & Condemnation Drilling	1,000
Underground Material Handling System Simulation	220
Underground Mining Alternatives	20
Underground Geotechnical	260
Optimized Open Pit Mine Planning	0
Supplemental Geotechnical Investigation	80
Supplemental Mine Planning	230
Additional Metallurgical Testing	180
Tailings, Civil Infrastructure & Geotechnical	300
Data Gathering for Reclamation	85
Geochemical Assessment	50
Water Management	400
Total	**2,825**

Note: These costs are included in Capex, Sustaining Capex, and Opex described in Item 21.0.

26.2 Open Pit

The study of the stand-alone Open Pit Project has been completed to a PFS level of accuracy. The PFS showed that there exists a positive business case to further advance the project through engineering, geological work as well as further study.

There are several areas that could benefit from further examination to enhance the potential of the Open Pit Project, including additional drilling of the open pit deposits, review the year of production expansion (bring forward or delay); and review the impact of higher price market conditions/assumptions on the size of the open pit and its life of mine.

Additional drilling has the potential to improve the economics of the open pit project by adding reserves in areas of open mineralization to the north and north west and by increasing the level of confidence of the Inferred Resources within the pit to Indicated or Measured Resources. These aspects and actions to reduce risk and increase certainty on a range of perspectives are outlined in the following sub-sections

26.2.1 Resources (Open Pit)

With respect to Mineral Resources, Golder offers the following recommendations:

- Undertake drilling to increase the classification of Inferred Resource material, both within the Open Pit Mineral Resource, and on the periphery of where mineralization remains open. Review the larger scale structural regime to determine possible exploration targets (off set mineralization).
    - Re-assay pulps/rejects from historical drilling (that did not include gold and silver) to make a more robust gold and silver estimate.
- Provide additional condemnation drilling to confirm infrastructure locations.
- Evaluate the usage of a sub blocked modelling approach to apply geologic dilution as an alternative to applying all dilution as part of the mining process.

26.2.2 Mineral Reserves & Mining Planning (Open Pit)

Based on the positive results of the PFS study Golder recommends that Nevada Copper consider this Report and advances the Open Pit Project to a Feasibility Study. See also Item 25.3.1 for details on the Feasibility Study.

The scopes for future work should include:

- Evaluate other potentially economic commodities including iron and talc

26.2.3 Metallurgy & Recovery Methods (Open Pit)

The Pumpkin Hollow open pit and underground deposits have been sampled and metallurgically studied for many years. These studies have been carried out intermittently at different research facilities. The ores have been shown to be amenable to conventional crush/grind/float processing and, depending on grind size and various flotation conditions, have consistently produced marketable concentrates.

In the ongoing effort to reduce project risk, additional sampling and continued variability testing will proceed concurrently with future engineering stages of the Open Pit Project. Information collected from this continued effort may potentially influence mine planning and ore blending, as well as mineral liberation and flotation requirements.

26.2.4 Variability Test work (Open Pit)

The following studies on variability samples should be considered in the next stages of development:

- Additional variability samples, complementing the eleven variability samples already studied, would be taken spatially throughout the identified ore zones. These samples would be targeted on the major rock types and correlated to a likely mine life plan which would include the ore hardness and expected variability. The new variability samples would:
  - Undergo Bond grindability testing and SMC testing to characterize the ores with respect to hardness, further defining power and grinding requirements. While prolonged excursions into either extremely soft or hard ores that cannot be handled by ore blending are not expected, the additional information resulting from this test work will be useful for planning purposes as the Open Pit Project progresses.
  - Be subjected to a program consisting of rougher and cleaner flotation testing. These tests would further explore the response of different ore types (rock types) and feed grade variation to standard flotation conditions. Additional composite samples, representing various time periods in the LOM schedule, will also be studied in the same manner.
    - Geometallurgy based characterization of selected samples would be completed in future development phases. This characterization will include samples of both whole ore and individual flotation products. This work, much of which would be founded in automated mineralogy, i.e., QEMScan® or other technologies, will inform on potential grind/liberation size relationships and provide insight to conditions that would be planned for the staged throughout over the production profile of the Open Pit Project and the planned sequences of mining the North and South pits.

26.2.5 Composite Test work (Open Pit)

The following test work on composite samples should be considered in the next stages of development:

- Comminution and flotation test work on the composite sample representing first seven years of mine life to confirm recoveries and establish optimum flotation conditions, particle grind size, pH, reagent regime for the locked cycle flotation test work, and subsequent variability samples flotation test work, which will include:

  - Optimized primary grind and concentrate regrind particle sizing
  - Optimized reagent dosage and flotation parameters such as pH for the roughers and cleaners
  - Conducting flotation test work on the high talc sample (if available)
  - Conducting bulk flotation test work
  - Conducting batch cleaner flotation test work
  - Conducting locked cycle flotation test work
  - Conducting settling and filtration test work on produced concentrate and tailings samples
- Conducting vendor test work using equipment suppliers to increase the accuracy of equipment selection that includes:
  - Comminution testing, including Jar (specific regrind energy) test work
  - Flotation testing
  - Filtration and thickener settling test work

The expected costs to Nevada Copper for the vendor test work are only the supply and delivery of the samples as working with the right selection of vendors in the future phases of work will be critical to the outcome.

The purpose of this proposed test work is to further:

- Improve confidence in the sizing of the milling circuit equipment
- Confirm metal recoveries including regrind particle size and achievable concentrate grades
- Confirm tailings filtration performance for equipment sizing
- Ensure reasonable representation of material that will be processed during first five years of production

The proposed test work would include comminution test work, batch cleaner and locked cycle flotation tests, settling and filtration rates. The estimated timeline to complete the proposed test work and reporting is around 16 to 20 weeks.

The proposed test work and next stages of development work would investigate and conclude the following additional activities:

- The review of potential offtake commercial terms with respect to payable and penalty elements in the concentrate, if the review concludes additional concentrate test work is required for element analysis, then undertake additional work.

- The conclusion of pre-crushing and ore sorting and where this may or not be needed in either the development phase or production plans
- The review of the relevant market conditions to trade-offs if the recovery of Fe and Mo are viable to warrant further consideration

26.2.5.1 Processing Facilities Engineering

As part of the Open Pit PFS test work campaign, metallurgical test work was performed on the variability samples from the North and South deposit in order to determine ore characteristics, most notably hardness, and its effect on the PFS stage grinding circuit design.

During the months of December 2018 and January 2019, SGS performed comminution test work on 11 variability samples from the North and South Deposits.

The results from this test work campaign were compared to results from the historical data package which included the referenced Dawson 2015 metallurgical test work program, which presents a basis for the grinding circuit design in the current Open Pit PFS.

In addition to test work results discussion in Item 13, options for managing ore hardness in the current PFS design will remain as an open task to ensure the SAG Mill and final throughput are optimized without process risk on the net production rates.

During the Open Pit PFS, options for deferral of the pebble crusher circuit or addition of a pre-crushing circuit was investigated. Based on the updated test work from January 2019 and the conclusions outlined in Item 13.0, the pebble crusher was re-instated into the pre-production capex.

Final reviews of the process design will be concluded in the planned FS phase of work, which will include:

- A further detailed review of the ore characteristics and final mine plan in order to better understand variations of the SAG Mill feed over the LOM and the potential for blending of ores of different hardness
- The potential to reduce SAG mill feed size to support throughput, e.g. through blasting optimization or pre-crushing campaigns on a portion of ores
- The conducting of recognized industry peer reviews of the equipment sizing calculations to validate all current design assumptions, specifically focusing on the comminution circuit and flotation recovery circuits
- The conducting of potential additional variability test work data may be required to confirm the milling circuit equipment size.

26.2.6 Environmental & Reclamation (Open Pit)

- It is recommended that a mine rock management plan be developed for placement of mine rock in the MRSF, with sections addressing the placement of PAG and NPAG material (to ensure that PAG mine rock material is not present at the surface of the MRSF at closure), and addressing that a portion of the quaternary alluvium and tertiary conglomerate mine rock material is stockpiled in an area that may be readily used as final reclamation cover material or rock blends at the end of mining.

26.2.6.1 Long-Term Closure Cover Erosional Resistance

- Once the nature of the closure cover materials (surface soil salvage and stockpiled quaternary alluvium and tertiary conglomerate) is further characterized, the long-term erosional resistance of DST and MRSF closure covers should be re-evaluated to:
  - Predict soil loss and head cutting potential from facilities during operations and following closure
  - Develop and evaluate erosion and sediment control options
  - Predict the rate and magnitude of sediment loads to operational and closure stormwater drainage systems (basins, channels, etc.)
- Vegetation monitoring data should be collected from reclamation test plots. These data, and data from the characterization of waste and cover hydraulic properties should be used as inputs to empirical or process-based erosion and sedimentation prediction models (Revised Universal Soil Loss Equation, Water Erosion Prediction Project, Erodibility Index Method, SEDCAD, and others) for the evaluation of facility drainage designs, sediment management plans and erosion and sediment control alternatives.

26.2.6.2 Air Quality

- Finalize the emissions inventories for the open pit mine operations.
- Verify with the NDEP that the PM_{10} background concentrations resulting from sources other than the Pumpkin Hollow Project alternatives are still accurate and appropriate for air modeling inputs.
- Conduct air quality dispersion modeling using the EPA's AERMOD air quality dispersion modeling system for Ambient Air Quality Standard and particle size distribution increment modeling to predict future concentrations at and beyond the project controlled (ambient air) boundary.
- Compare the modeled results to state and federal air quality standards and work with the mine planners to achieve compliance with the applicable Ambient Air Quality Standard and particle size distribution increments.

- Prepare an application for Revision of Class II Air Quality Operating Permit AP1021-3369 for the Pumpkin Hollow Project that will add the open pit mine operations to the underground mine operations that are currently permitted.

26.2.6.3 Geochemistry

- Eighty additional mine rock samples are recommended for characterization to substantiate the feasibility study findings, which suggests that most of the mine rock is non-PAG and non-reactive. The results will be used to provide defensible recommendations on mine rock management and to advance the pit lake water quality modeling. All rock types identified in the block model are included in the Phase IV testing. However, the program focuses on the mine rock with the highest variability in ABA characteristics, including hornfels, magnetite skarn, silicate skarn, endoskarn and intrusive.
- Tailings representative of the process during the first six years of the SPF alternative mine plan are currently being produced at Hazen Laboratories and will be subjected to static testing including ABA, elemental analysis and meteoric water mobility procedure testing. These results will assist with fate and transport modeling associated with the DST. Cemented paste tailings samples will be prepared and subjected to diffusion testing to further assess the water quality associated with the underground mine backfill.
- Kinetic testing of mine rock and tailings samples should continue to further develop source terms for predictive modeling and to support permitting activities.
- Implementation of pilot test piles for long-term monitoring of relatively large quantities of material under ambient field conditions. The larger scale relative to laboratory tests results in field test plots having more representative sample dimensions and particle size, in the case of waste rock, and minimizes impacts from boundary effects, sample heterogeneity, and reduced grain size. A comprehensive characterization of the test charge is required. In combination with a good understanding of the water balance for the test pad (achievable through meteorological monitoring or controlled application of infiltration, or both), reaction rates and loadings can be developed for extrapolation to full-scale mine facilities. Longer monitoring durations may be required because of lower field temperatures, intermittent drying, and lower reactivity of field cell test charges relative to the finer-grained materials commonly included in laboratory tests.

26.2.7 Future Work (Open Pit)

The following areas of future work are based on refining areas of uncertainty and opportunity identified during the Open Pit PFS, as outlined in the Recommendations.

A work program is recommended that includes engineering, studies and investigations in preparation for further project advancement. An indication of costs for such activities are estimated in Table 26-2.

Table 26-2: Recommended Activities & Costs

Recommended Activity	Cost ($, thousands)
Resource Definition Drilling	4,000
Exploration & Condemnation Drilling	2,750
Additional Mining Studies	500
Additional Processing and Infrastructure Studies	450
Additional Metallurgical Testing	80
Tailings, Water, Environment and Reclamation	600
Total	**8,380**

26.3 Common (Underground & Open Pit)

26.3.1 Concentrate Transport

The concentrate handling studies established in the study should be advanced to the next level of detailed assessment to refine design and cost options to advance both the Open Pit and Underground Projects.

26.3.2 Tailings and Waste

The following items are recommended to advance the current design of the DST facility to a detailed engineering level:

- Review existing geotechnical data and if necessary perform a detailed subsurface geotechnical investigation and laboratory testing program within the footprint of the DST facility, the mill, and the filtration plant to assess foundation conditions at the site.
- The tailings samples selected for geotechnical characterization testing as part of this and previous studies should be reviewed to ensure they represent the expected range of materials to be processed over the mine life and the expected process treatment.
- A tailings facility operating manual and a monitoring and surveillance plan should be developed. The monitoring plan should include measurements to confirm the unsaturated condition of the tailings stack, the extent of seepage into the foundation soils (if any) and the performance of the containment system.
- The seepage model should be reviewed and updated as required with consideration of the stacking plan developed as part of detailed design. Based on this update, the design of containment features methods for containing and managing fluid drainage in the tailings stack should be reassessed. Seepage assessment will also be required to determine the timeframe of long-term monitoring and potential treatment requirements.
- The stability model should be reviewed and updated as required with consideration of the final stacking plan and updated information on the material properties of the tailings, construction materials including the HDPE liner, and the foundation. Modeling should include determination of acceptable setback distances for operating at the stack perimeter.

- The liquefaction assessment should be reviewed and updated as required with consideration of updated information on material properties and updated stacking plan.
- The stacking plan and proposed truck haul alignments and ramps should be reviewed with respect to optimizing operations and maintaining adequate work areas.
- A tailings facility operating manual and a monitoring and surveillance plan should be developed. The monitoring plan should include measurements to confirm the unsaturated condition of the tailings stack, the extent of seepage into the foundation soils (if any) and performance of the containment system.
- Dust containment measures should be reviewed and plans to mitigate dust and contain tailings in the stack should be advanced. These measures may include progressive reclamation of the perimeter slopes and incorporation of a perimeter bund to improve the effectiveness of long term tailings containment and erosion control, and optimization of dust suppression application types and amounts.
- The detailed design should be reviewed to ensure consistency and adherence to the anticipated closure and reclamation plan.
- Geotechnical investigations including drilling, field work and lab testing to support civil, structural design of the surface facilities including the dry stack facility final design.
- Geotechnical design work after the geotechnical investigation.

In accordance with regulatory permit requirements, the first cell of the DST will have a single synthetic liner, with subsequent cells to be unlined, other than a compacted tailings base. A separate dry stack trial cell and monitoring program are required to demonstrate the effectiveness of the proposed subsequent unlined DST cells prior to the construction of those cells and deposition of filtered tailings.

26.3.3 Power

Nevada Energy has provided input to the power supply details that were used in the open pit study. Future studies will require additional input from Nevada Energy to complete a suitable design.

26.3.4 Water Balance

It is recommended that multiple infiltration tests be performed in the area selected for RIB construction to determine a design infiltration rate.

For the next site-wide water balance model iteration, RIB size and flow configuration should be optimized to reduce cost while allowing for operational redundancy when regular RIB maintenance is required. The settling basin will need to be sized to allow for an appropriate settling time and storage in case of a process facility shutdown or an increase in dewatering flowrates.

It is also recommended that the site-wide water balance model be periodically updated as empirical hydrologic data become available during mine development and initial production. In particular, actual mine inflows into the open pits should be measured as well as water consumption components. Application rates to the infiltration basins and basin performance should also be monitored.

The estimated budget for the site-wide water balance scope for the open pit operation is approximately $50,000.

26.3.5 Environmental & Reclamation

The following items are recommended to advance the environmental and reclamation items:

- It is recommended that an Order 2 Soil Survey soil sampling and characterization be conducted in and adjacent to the proposed area of disturbance. This information should define the location, volume, properties, uniformity, and retrievability of potential sources of PGM/cover materials on or immediately adjacent to the site. It is also recommended that saturated hydraulic conductivity and soil water characteristic curves (SWCC) of waste rock, tailings and the likely source(s) of PGM/cover materials be determined. Proper characterization of soil material properties will help identify the soil types for later use as PGM and for optimizing closure cover designs. The estimated cost for this work is approximately $15,000.
- Determination of the entire particle size distribution of PGM and the PGM/rock blends is also recommended to better define and predict their water holding capacity and erosion resistance. In addition, optimal in-place density of closure covers and subgrade materials should be determined through a field testing program to provide many of the benefits of compaction without jeopardizing soil cover stability and the viability of vegetation development and growth. The estimated cost for this work is approximately $5,000.
- Revegetation test plots should be installed and monitored. These test plots will allow the methods used to establish the Reclaimed Desired Plant Communities (RDPCs) and control erosion and sedimentation from disturbed areas, to be tested on site prior to full implementation. In addition, the performance of various plant species, methods for controlling erosion, and methods to increase soil moisture and nutrients may be evaluated through testing of different soil amendments, nurse crops, surface roughening approaches, irrigation, soil binders and erosion control fabrics. This work should be integrated into the operating environmental management plan conducted by owner’s team. The estimated costs for materials is approximately $5,000.
- Various aspects of the reclamation approach should be designed and revised based on test plot findings, interim monitoring findings, and concurrent reclamation monitoring findings. The following aspects of the reclamation design should be updated based on monitoring findings:
    - Soil and subgrade compaction criteria

- Ripping specifications
- Soil fertilization and amendment specifications
- Mulching specifications
- Erosion control fabric installation specifications
- Seeding plans and seed mixtures

This work should be integrated into the operating environmental management plan conducted by owner's team:

- Should on-site testing indicate that additional organic matter will need to be added to support a vegetated cover, initial evaluations should be conducted to identify potential soil amendments for use at the site. This evaluation would be performed several years after commencement of operations and should be integrated into the operating environmental management plan conducted by owner's team. The estimated cost for this work is approximately $10,000.

Item 27.0 REFERENCES

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Abbreviations and Acronyms

Abbreviation	Definition
3D	three-dimensional
AAL	American Assay Laboratories
ABA	acid-base accounting
Ag	silver
Ai	Bond abrasion index
Anaconda	Anaconda Corporation
ARD/ML	acid rock drainage and metal leaching
Au	gold
BAPC	Bureau of Air Pollution Control
BBMWi	Bond ball mill work index
BC	607792 B.C. Ltd.
BEV	battery electric vehicle
BLM	Bureau of Land Management
BMRR	Bureau of Mining Regulation and Reclamation
BWPC	Bureau of Water Pollution Control
C1	direct costs (standard economic copper metric)
Capex	capital cost estimate
CFR	Code of Federal Regulations
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
COB	coarse ore bin
CPF	cemented paste fill
CR	County Road
Cu	copper
CuEq	copper equivalence
Cyprus	Cyprus Metals Exploration Corporation
Dawson	Dawson Metallurgical Laboratories
DCS	distributed control system
DRI	Desert Research Institute
DST	dry stack tailings
EDA	exploratory data analysis
EBITDA	earnings before interest, tax, depreciation and amortization
EDCs	engineering design changes
EGL	effective grinding length
EIS	Environmental Impact Statement
EMT	emergency medical technician
EP	equivalent people
EPA	U.S. Environmental Protection Agency
EPC	Engineering, Procurement and Construction
EPCM	Engineering, Procurement and Construction Management
Fe	iron
FOS	factor of safety

Abbreviation	Definition
FS	feasibility study
FW	footwall
G&A	general and administrative
G&T	G&T Metallurgical Services Ltd.
Golder	Golder Associates Ltd.
Hazen	Hazen Research, Inc.
HCT	humidity cell test
HS&E	health, safety & environment
HV	high voltage
HW	hanging wall
I/O	input/output
ICP	inductively coupled plasma
ICP-AES	inductively coupled plasma atomic emission spectroscopy
IDW2	inverse distance weighting to the power of two
IOCG	iron oxide-copper-gold
IRR	internal rate of return
Ja	joint alteration
Jn	joint set number
Jr	joint roughness
Jw	joint water
LHD	load-haul-dump
LOM	Life of Mine
MCC	motor control center
MIBC	methyl isobutyl carbinol
Mining Plus	Mining Plus Canada Consulting Limited
Mo	molybdenum
MOU	Memorandum of Understanding
MRSF	mine rock storage facility
MSHA	Mine Safety and Health Administration
MSO	Mineable Shape Optimizer
n/a	not applicable
NAC	Nevada Administrative Code
NAG	net acid generation
NCI	Nevada Copper Inc. [a subsidiary of Nevada Copper Corp.]
NDEP	Nevada Division of Environmental Protection
NDOW	Nevada Department of Wildlife
NEPA	National Environmental Policy Act
Nevada Copper	Nevada Copper Corp.
NFPA	National Fire Protection Association
NGI	Norwegian Geotechnical Institute
NI 43-101	National Instrument 43-101
NNHP	Nevada Natural Heritage Program
NOI	Notice of Intent

Abbreviation	Definition
non-PAG	non-potentially acid generating
NPDES	National Pollution Discharge Elimination System
NPR	neutralization potential ratio
NPV	net present value
NRS	Nevada Revised Statutes
NSR	net smelter return
NTNCPWS	Non-transient, Non-community Public Water System
NV	Nevada
OEM	original equipment manufacturer
Opex	operating cost estimate
OSDS	on-site sewage disposal system
PAG	potentially acid generating
PC	programmable computer
PEA	preliminary economic assessment
PFD	process flow diagram
PFS	prefeasibility study
PGM	plant growth medium
Plexus	Plexus Resources Inc.
Pocock	Pocock Industrial, Inc.
QA/QC	quality assurance / quality control
QP	Qualified Person
P80	80% passing size
Pb	lead
PGM	plant growth medium
PLC	programmable logic controller
PoF	probability of failure
PPE	personal protective equipment
RC	refining charge
RDPCs	Reclaimed Desired Plant Communities
Re	rhenium
RF	revenue factor
RGGS	RGGS Land & Minerals LTD., L.P.
RIB	rapid inflation basin
RMR76	Rock Mass Rating (after Bieniawski, 1976)
RMR89	Rock Mass Rating (after Bieniawski, 1989)
ROM	run of mine
RQD	Rock Quality Designation
S	sulfur
SABC	semi-autogenous ball mill crusher
SAG	semi-autogenous grinding
SCADA	supervisory control and data acquisition
SCB	secondary containment basin
SEDAR	System for Electronic Document Analysis and Retrieval

Abbreviation	Definition
Sedgman	Sedgman Canada Limited
SHPO	State Historical Preservation Office
SMB	stormwater management basin
SMC	SAG Mill Comminution
SOC	schedule of compliance
SPCC	Spill Prevention, Control and Countermeasure
SPF	single process facility
Sph	Spherical
SRCE	Standard Reclamation Cost Estimator
SRF	stress reduction factor
SUP	Special Use Permit
SWCC	soil water characteristic curve
SWPPP	Stormwater Pollution Prevention Plan
Taurus	International Taurus Resources Inc.
TC	treatment charge
TCRC	treatment and refining charge
Tetra Tech	Tetra Tech Inc.
TML	transportable moisture limit
TSX	Toronto Stock Exchange
TV	Tertiary Volcanics
UCS	unconfined compressive strength
UP	Union Pacific
UPF	uncemented paste fill
URF	unconsolidated rock fill
USACE	U.S. Army Corps of Engineers
USFWS	U.S. Fish and Wildlife Service
USGS	U.S. Geological Survey
USS	United States Steel Corporation
W	tungsten
WMB	water management basin
WPCP	Water Pollution Control Permit
WRSF	waste rock storage facility
Zn	zinc

Units

Abbreviation	Unit
°	degree
°C	degrees Celsius
¢	US cent
$	US dollar
$/t.oz	dollars per metric tonne per ounce
%	percent
ac-ft	acre-feet
amsl	above mean sea level
µm	micron
CaCO3/kt	calcium carbonate per kilotonnes of material
cfm	cubic feet per minute
cm/s	centimeters per second
d	day
dmt	dry metric tonne
ft	foot, feet
ft3	cubic foot
ft3/st	cubic feet per short ton
ft/d	feet per day
ft/hr	feet per hour
ft/min	feet per minute
g	gram
g/st	grams per short ton
g/t	grams per metric tonne
gpm	gallons per minute
hp	horsepower
hr	hour
kcfm	thousands of cubic feet per minute
kg/m2.hr	kilograms per square meter per hour
Kst	thousand short tons
Kstpa	thousand short tons per annum
Kstpd	thousand short tons per day
kV	kilovolt
kVA	kilovolt ampere
kWh/t	kilowatt hours per metric tonne
kWh/st	kilowatt hours per short ton
lb	pound
lbs/ft2-hr	pound per square foot per hour
m3/hr	cubic meters per hour
m3/s	cubic meters per second
m3/s.kW	cubic meters per second per kilowatt
Ma [not My]	mega-annum

Abbreviation	Unit
MPa	megapascal
mi	mile
Mst	million short tons
Mstpa	million short tons per annum
MVA	megavolt ampere
oz	ounce
oz/st	ounces per short ton
oz/yd2	ounces per square yard
ppm	parts per million
sq. km	square kilometer
sq. mile	square mile
st	short ton
st/ft3	short tons per cubic foot
st/yd3	short tons per cubic yard
stpa	short tons per annum
stpd	short tons per day
stph	short tons per hour
t/d	metric tonnes per day
t/y	metric tonnes per year
V	volt
wmt	wet metric tonne
wst	wet short ton
wt. %	percent by weight
yd3/hr	cubic yards per hour